S-K 1300 TECHNICAL REPORT SUMMARY

Mt Todd Gold Project | 15 ktpd Feasibility Study

Northern Territory, Australia

Effective Date: July 29, 2025
Issue Date: September 11, 2025

Project No. 13028

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FORWARD-LOOKING STATEMENTS

This Technical Report Summary 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 Summary that address future activities, events, developments, or outcomes that we or others expect or anticipate will, may, or may not occur in the future, are forward-looking statements and forward-looking information.

Forward-looking statements and forward-looking information include, but are not limited to statements regarding such things as: estimates of Mineral Resources and Mineral Reserves; the gold price and other inputs used to estimate Project output and performance, Project design and availability of required approvals; timing or ability to complete any activity as set forth herein; annual and cumulative gold production at estimated recovery rates over the life of mine; mining methods and procedures; processing methods and procedures; projected Project economics, including but not limited to anticipated production and revenue, cash costs, royalty payments, government royalties and taxes payments, other payments that may or may not have been contemplated, after-tax NPV, IRR, non-U.S. GAAP measures and any other monetarily derived value; and other such matters are forward-looking statements and forward-looking information.

Among the material factors and assumptions used to develop the forward-looking statements and forward-looking information contained in this Technical Report Summary include: the accuracy of test work and interpretation of results used to prepare this Technical Report Summary, Mineral Resources and Mineral Reserves estimates, and exploration findings and assay results; the terms and conditions of the Company's agreements with third-parties; Vista's approved or expressed business plans; the anticipated timing and completeness of approvals and permissions; the potential occurrence of certain threatened species of flora, vegetation, and fauna within the mine site; no change in laws that materially impact mining development or operations of a mining business; the potential occurrence and timing of a formal investment 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 costs, all-in sustaining costs, NPV, IRR, initial and sustaining capital requirements, reclamation and closure costs, and self-funding reclamation proceeds; and Vista's objective to advance the Project to be a producing gold mine.

When used in this Technical Report Summary, the words and derivatives of words such as "optimistic", "potential", "indicate," "expect", "intend", "plan", "believe", "may", "will", "if", "anticipate", and similar words or expressions that reference or imply future conditions are intended to identify forward-looking statements and forward-looking information. Statements that include such words or expressions reflect known and unknown risks, uncertainties and other factors that may cause 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 Resources estimates, estimates of results based on such Mineral Resources estimates and Mineral Reserves estimates; risks relating to cost increases, scope changes, and consumption requirements 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; unforeseen delays; risks relating to fluctuations in the price of gold and foreign exchange rates; the inherently hazardous nature of mining-related activities; potential effects on Vista's operations of applicable and influencing environmental and other regulations; risks due to legal proceedings; risks relating to political, social, and economic instability; as well as those factors discussed under the headings "Note Regarding Forward-Looking Statements" and "Risk Factors" in Vista's Annual Report Form 10-K as filed in February 2025 and other documents filed with 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.

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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
2022 FS	NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Feasibility Study – Northern Territory, Australia, Effective Date December 31, 2021; Issued February 9, 2022.
2024 FS	NI 43-101 Technical Report – Mt Todd Gold Project 50,000 tpd Feasibility Study – Northern Territory, Australia, Effective Date March 12, 2024; Issued April 16, 2024.
A	ampere
a	annum (year)
AA	Atomic adsorption
ABA	acid base accounting
AARL	Anglo American Research Laboratories
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
ANCOLD	Australian National Committee on Large Dams
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
CPT	Cone Penetration Testing
CoA	chart of accounts
CRD	capital recognition deduction
CV	Construction Verification
CWi	Crusher Work index
CWP	Clean Water Pond
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 and Construction Management
EPC	Engineering Procurement and Construction (fixed price with contractors margin)
EQP	equalization pond
F80	80% feed passing size
FIS	Free In Store
FLS	FLSmidth
FS Case	15 ktpd 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 Ag/t	grams silver per tonne
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.

GISTM	Global Industry Standard for Tailings Management
GJ	Gigajoule
gpm	gallons per minute (US)
GPR	Gross Proceeds Royalty
GR	gross realization
GRES	GR Engineering Services Limited
GST	Australian Goods and Services Tax
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
HME	Heavy mechanical equipment – haul trucks, mine shovels .
HNO3	nitric acid
HPGR	High Pressure Grinding Rolls
HQ	88.9 mm drill rod (outer diameter)
hr	hour
HSE	Health, Safety and Environment
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
LPG	Liquefied petroleum gas
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 Lease
MLN	Mineral Lease Number
mm	millimeter
MMP	Mining Management Plan
mo	month
Moz	million ounces
Mpa	megapascal
mPa·s	Centipoise (millipascal second)
MPU	Mobile processing unit
MRA	Mineral Royalties Act
MRT	Mining & Resource Technology Pty Ltd
Mt	million tonnes
Mtpa	million tonnes per annum
MTO	material take-off
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	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
PQ	3.75 in drill rod (outer diameter)
PWC	Power and Water Corporation
PWP	Process Water Pond
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person
Qty	Quantity or number of
R&R	Rest and recreation
RDi	Resource Development Inc.
RESPEC	Mine Development Associates (MDA)
RKD	RKD (Company Name)
RL	Relative Elevation
RO	Reverse Osmosis
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
USD	U.S. dollar
V	volt
VESDA	Very Early Smoke Detection Apparatus
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 (USD) unless otherwise noted. Common units of measure and conversion factors used in this Technical Report Summary 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. EXECUTIVE SUMMARY

1.1 Overview

Vista Gold Corp. and its subsidiaries (collectively Vista or the Company) retained GRES, along with Mining Plus, Resource Development Inc. (RDi), Tetra Tech, Tierra Group and WSP, to prepare this Feasibility Study (FS or Technical Report Summary) for its Mt Todd Gold Project (the Project) in the Northern Territory (NT), Australia. This Technical Report Summary evaluates a development scenario for a 15,000 tonne per day (15 ktpd) processing facility.

Vista and its subsidiary, Vista Gold Australia Pty (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 Project.

The Project contains several 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 unaffiliated with Vista. The Batman deposit has been explored more extensively than the Quigley deposit. Vista has reported the Mineral Resource estimates in accordance with the SEC's Regulation S-K subpart 229.1300 mining disclosure rules.

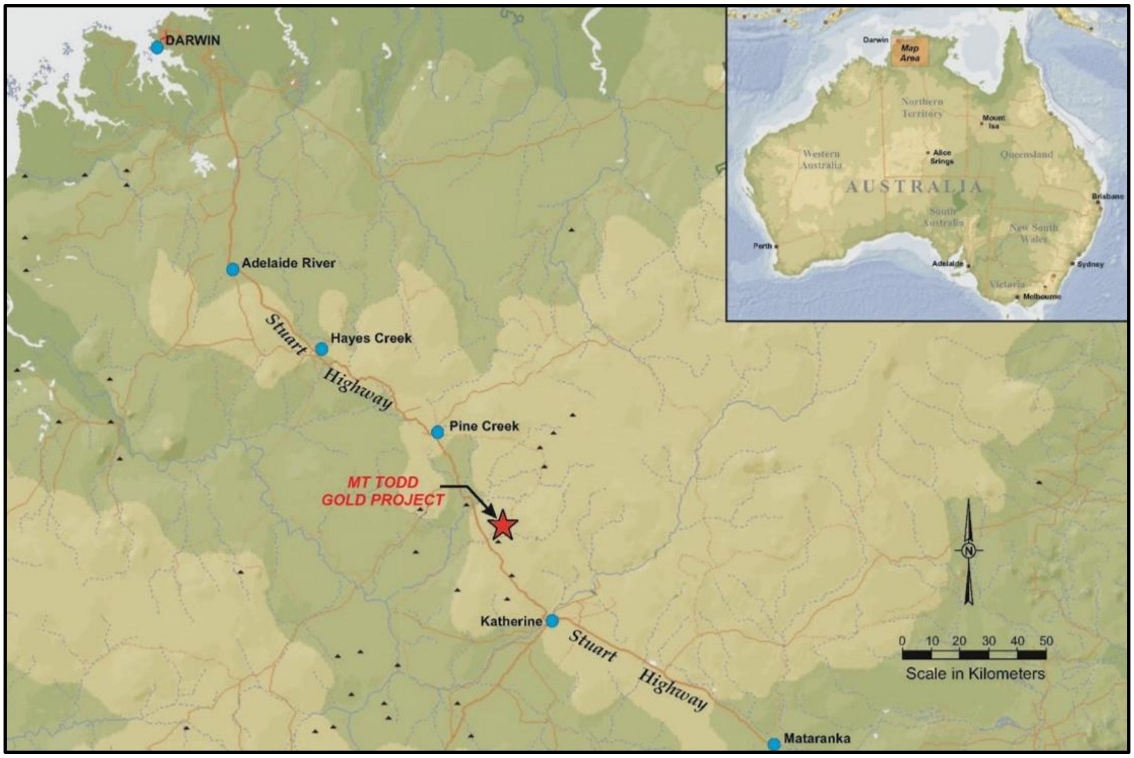
The information presented in this Technical Report Summary is intended to assist stakeholders and other readers of this Technical Report Summary in their understanding of the Project and in forming judgements regarding the quality of the data collected, reported, and used in this Technical Report Summary.

1.2 Property Description and Location

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

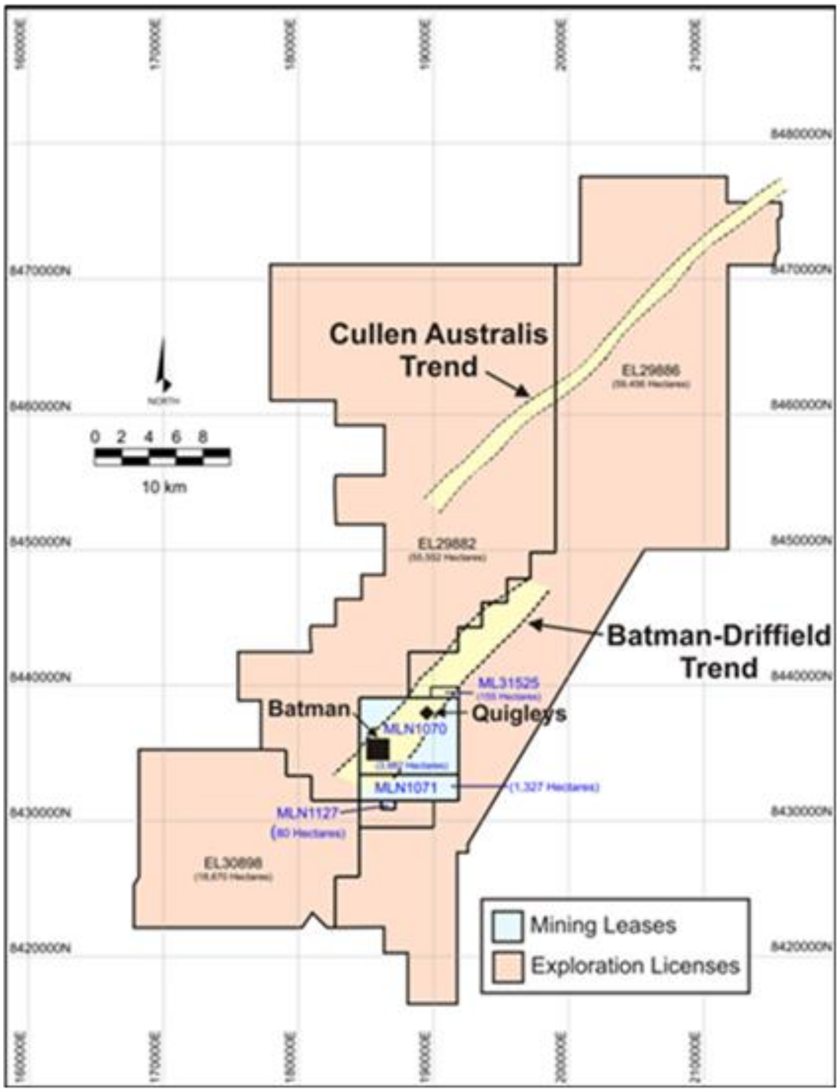
Vista Australia is the holder of four mineral leases (ML) MLN 1070, MLN 1071, MLN 1127, and ML 31525 comprising approximately 55.4 km^2. In addition, Vista Australia controls exploration licenses (EL) EL 29882, EL 29886, EL 30898, comprising approximately 1,337 km^2. In July 2025, Vista surrendered EL 32004, and withdrew its application for EL 32005 as Vista deemed these licenses of no economic benefit for the Company.

Figures 1 to 3 show the Project's location, mining leases and exploration licenses and general overall Project site layout.



Source: Prepared by Vista,2020

Figure 1 Mt Todd Gold Project Location



Source: Prepared by Vista, updated July 2025

Figure 2 Mining Lease and Exploration Licenses, Mt Todd Gold Project, MGA Zone 53 Coordinates

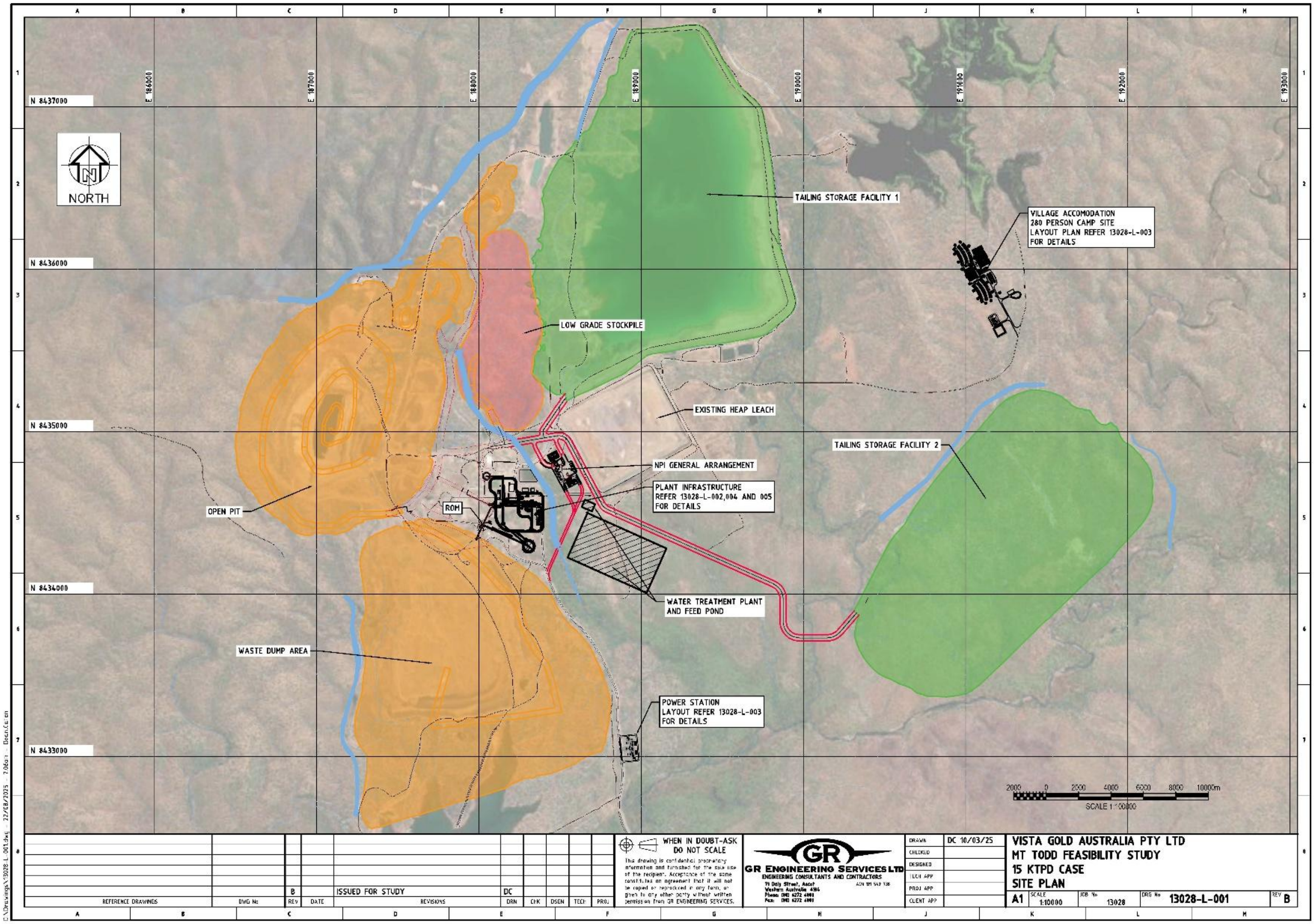


Figure 3 Overall Site Layout for the Project, MGA Zone 53 Coordinates

1.2.1 Topography, Elevation, and Vegetation

The topography of the Project is relatively flat. The mineral leases and exploration licenses 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 m above sea level in the area of the previous and planned site and waste dumps.

1.3 Geology and Mineralization

The Project is situated within the southeastern portion of the Early Proterozoic Pine Creek Geosyncline. 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 which are classified as 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 Batman deposit unique in the PCG. The mineralization within the Batman deposit is directly related to the intensity of the north-south trending quartz sulfide veining. The lithological units impact on the orientation and intensity of mineralization.

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

A variety of mineralization styles occur within the Project area. Of greatest known economic significance are auriferous quartz-sulfide vein systems. These vein systems include the Batman, Jones, Golf, Quigleys and Horseshoe prospects, which occur within a north-northeast trending corridor, and are hosted by the Burrell Creek Formation. Tin occurs in a north-northwest trending corridor. The tin mineralization comprises cassiterite, quartz, tourmaline, kaolin, and hematite bearing assemblages, which occur as bedding parallel to breccia zones and pipes. Polymetallic Au, W, Mo, and Cu mineralization occurs in quartz-greisen veins within the Yinberrie Leucogranite; a late stage highly fractionated phase of the Cullen Batholith. The Batman deposit extends approximately 2,400 m along strike, 600 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.

1.4 Mineral Resource Estimates

The following sections summarize the process, procedures, and results of the Mineral Resource estimates for the Batman and the Quigleys deposits (excludes the Heap Leach Pad, refer note (10)).

The Batman deposit has been the subject of multiple investigations and Mineral Resource estimations throughout the years, with Tetra Tech being involved since 2008 and has been updated in 2025 to include exploration drilling conducted in 2020-2022 and 2024, as well as updated prices, geotechnical parameters, and recoveries. The updated Mineral Resource estimates for the Project are shown in Table 1.

	Batman Deposit			Heap Leach Pad			Quigleys Deposit		
	Tonnes (000s)	Grade (g Au/t)	Contained Ounces (000s)	Tonnes (000s)	Grade (g Au/t)	Contained Ounces (000s)	Tonnes (000s)	Grade (g Au/t)	Contained Ounces (000s)
Measured (M)	47,143	0.61	930	-	-	-	3,702	1.13	134
Indicated (I)	110,644	0.72	2,568	-	-	-	6,965	1.34	299
Measured and Indicated	**157,787**	**0.69**	**3,498**	**-**	**-**	**-**	**10,667**	**1.26**	**433**
Inferred (F)	54,338	0.78	1,369	-	-	-	2,761	0.71	63

Notes:

(1) Measured and Indicated Mineral Resources exclude Proven and Probable Mineral Reserves.

(2) Batman and Quigleys Mineral Resources are quoted at a 0.4 g Au/t cut-off grade. Heap Leach Pad Mineral Resources are the average grade of the Heap Leach Pad, no cut-off grade was applied.

(3) The Point of Reference for the Batman and Quigleys Mineral Resources estimates is in-situ at the property. The Point of Reference for the Heap Leach Pad Mineral Resources estimates is the physical Heap Leach Pad at the property.

(4) Batman and Quigleys: Mineral Resources constrained within a USD1,950/oz gold pit shell. Pit parameters: Mining Cost USD3.00/tonne, Processing Cost USD17.50/tonne processed, General and Administrative Cost USD1.50/tonne processed, Au Recovery 89.7%.

(5) Kira Johnson MMSA of Tetra Tech is the QP responsible for the Statement of Mineral Resources for the Batman deposit, Quigleys deposit, and Heap Leach Pad.

(6) The effective date of the Batman ,Quigleys and Heap Leach Pad Mineral Resource estimates is, July 25th, 2025

(7) Mineral Resources that are not Mineral Reserves have no demonstrated economic viability and do not meet all relevant modifying factors.

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

(9) The Mineral Resources were estimated using in accordance with subpart 229.1300 of Regulation S-K.

(10) The entirety of the Heap Leach Pad Mineral Resource is converted to Mineral Reserves in this Technical Report Summary, therefore, a Mineral Resource exclusive of Mineral Reserves is not reported.

(11) "-" indicates no reported value.

Table 1 Summary of the 2025 Mineral Resource Estimates

1.5 Exploration

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

1.6 Mineral Reserve Estimates

The Project is currently at the FS stage and is based on a conventional open pit, truck, and hydraulic excavator operation, feeding a nominal 15 ktpd processing plant. The Mineral Reserve evaluation within this Technical Report Summary was supported by a Whittle 4X open pit optimization evaluation, excluding Inferred classified material within the Mineral Resources Estimate for the deposits.

The FS level mine design, mine scheduling, mining costing, and overall Project economic model evaluation confirmed positive economic outcomes for the Mineral Reserve. A conservative gold cut-off grade of 0.5 g Au/t was adopted based on economic parameters and recoveries determined as part of this Technical Report Summary. The resulting Mineral Reserve summary includes Proven and Probable Mineral Reserves for the Batman Deposit and the Heap Leach Pad.

The ore body description for the Batman deposit indicates that gold mineralization occurs in sheeted veins within silicified greywackes, shales, and siltstones. The deposit strikes north-northeast and dips steeply to the east, with higher-grade zones plunging to the south. The block model used for mine planning was modified to enhance its suitability for downstream mining processes, including adjustments to the weathering surface boundaries and the addition of new attributes to support mine optimization.

Detailed mine design, production schedule, and the mining costs have been used in the economic model. The final slope zone values reflective of a spiral ramp design were used for optimization, and the final shell selection forms the basis for subsequent phase design and production scheduling.

The resultant Mineral Reserves summary is shown in Table 2.

	Batman Deposit			Heap Leach Pad			Total		
	Ore	Grade	Contained Gold	Ore	Gold Grade	Contained Gold	Ore	Gold Grade	Contained Gold
	Tonnes (000s)	(g Au/t)	Ounces (000s)	Tonnes (000s)	(g Au/t)	Ounces (000s)	Tonnes (000s)	(g Au/t)	Ounces (000s)
Proven	77,359	0.95	2,371	-	-	-	77,359	0.95	2,371
Probable	81,263	0.99	2,588	13,352	0.54	232	94,617	0.93	2,820
Proven & Probable	**158,623**	**0.97**	**4,959**	**13,352**	**0.54**	**232**	**171,975**	**0.94**	**5,190**

Notes:

(1) The Mineral Reserves point of reference is the point where material is fed into the process plant.

(2) Batman deposit Mineral Reserves are reported using a 0.50 g Au/t cut-off grade and USD1,800 per ounce gold price.

(3) Colin McVie and Peter Lock of Mining Plus are the QP's responsible for the Statement of Mineral Reserves for Batman Deposit Proven and Probable Mineral Reserves.

(4) Because all the Heap Leach Pad Mineral Reserves are to be fed through the process plant, these Mineral Reserves are reported without a cut-off grade applied.

(5) Deepak Malhotra is the QP responsible for reporting the Heap Leach Pad Mineral Reserves.

(6) The effective date of the Batman and Heap Leach Mineral Reserves estimate is July 25th, 2025.

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

(8) The Mineral Reserves were estimated in accordance with subpart 229.1300 of Regulation S-K.

(9) "-" indicates no reported value.

Table 2 Project Mineral Reserves Estimate

1.7 Mining Methods

The operations for the Batman deposit will be mined in eight stages (mining phases) using a conventional truck and shovel approach. The mining operations will be conducted by contract mining, utilizing 400 tonne class hydraulic excavators and 190-tonne class rigid frame trucks. The ore material will be classified into High Grade (HG), Medium Grade (MG), and Low Grade (LG), with approximately 15 ktpd being fed to the processing plant. Waste material will be classified as Non-Acid Forming (NAF) or Potentially Acid Forming (PAF) based on its sulfur content.

A geotechnical assessment of the Batman pit slopes, highlighting the need for additional geological, geotechnical, and hydrogeological investigations has been carried out to ensure slope stability. The final pit design and optimal pit shell are based on these geotechnical recommendations, with specific parameters for pit road widths, gradients, and berm-batter configurations.

The mine production scheduling criteria involve multiple scenarios to ensure reliable delivery of mill feed tonnage, with a focus on optimizing Net Present Value (NPV) by prioritizing the processing of higher-grade ore in the initial years. The mining infrastructure, including support facilities, Heavy Mining Equipment (HME) workshops, and administrative offices have been covered in the planning for the mining operations. The workforce planning strategy is structured to align with the operational demands of each project phase, with personnel numbers estimated for Vista, the mining contractor, and the mining infrastructure contractor.

Overall, a comprehensive overview is provided of the mining operations, geotechnical assessments, production scheduling, equipment, waste management, and infrastructure for the Project.

The mining contractor will supply, install, and operate all necessary mine infrastructure during the pre-production period. This includes workshops, maintenance facilities, storage yards, fuel and lubricant farms, explosives storage, water cart filling points, administration facilities, and information and communications technology systems. The infrastructure meets Australian Standards and regulatory requirements, with ownership and maintenance responsibilities remaining with the contractor. The concrete requirements for various facilities are provided, along with an indicative construction schedule for the mine infrastructure. The aim is to ensure the site operates efficiently and retains personnel with comfortable and safe infrastructure.

1.8 Mineral Processing and Metallurgical Testing

Metallurgical test work has been carried out since 1998 for the proposed Project. The historical metallurgical test work programmes completed from 2017 to 2018 and 2018 to 2019 have mostly been used to support this Technical Report Summary. The earlier metallurgical test work and historical production records were however, also considered in the interpretation and process design.

The Batman ore host rock is very hard and competent. Gold is fine grained (<30 µm) and associated with sulfide minerals and quartz. The historical test work has demonstrated that the ore is amenable to gold extraction by conventional cyanidation processes but requires fine grinding to achieve moderately high gold extractions. The ore has moderate to high cyanide consumption due to the presence of iron sulfide and copper minerals. The test work also showed a benefit from pre-aeration and conditioning of the slurry with lime and lead nitrate prior to cyanidation to reduce the hindering effect of iron sulfide minerals present in the ore such as pyrite and pyrrhotite.

The processing plant design is based on the treatment of 5.325 Mtpa (15 ktpd) of hard ore from the Batman open pit. The key process design criteria that the plant was based upon is shown in Table 3. The flowsheet will consist of primary gyratory crushing in open circuit followed by secondary cone crushing in closed circuit and coarse ore storage via a live stockpile. Secondary crushed ore reclaimed from the stockpile will be further crushed by a tertiary stage High Pressure Grinding Roll (HPGR) operating in closed circuit with screens to produce a ball mill feed F_{80} size of 3.25 mm. X-ray transmission (XRT) ore sorting has been included on the HPGR sizing screen top deck oversize-29.5+16 mm fraction. The crushing circuit design utilizing a primary gyratory crusher, secondary cone crusher and HPGR tertiary crushing is a robust, proven technology to generate a grinding circuit feed F_{80} size of 3.25 mm.

The grinding circuit will comprise of a primary overflow ball mill operating in closed circuit with hydro-cyclones and secondary grinding comprising of four vertical stirred media grinding mills. The primary grinding stage will produce a secondary mill feed F_{80} size of 250 µm. The secondary grinding stage will produce a product P_{80} size of 40 µm for subsequent leaching. The secondary grinding stage hydro-cyclone overflow will be thickened to produce a leach feed density of 45% solids. The leach circuit will consist of two pre- conditioning (oxidation) tanks, two leach tanks and six adsorption tanks arranged as a hybrid carbon in leach (CIL) circuit.

Industry standard elution, electrowinning and smelting circuits will be used to produce gold doré. The elution circuit will include carbon regeneration.

Tailings from the process will be detoxified using an Air/SO_2 detoxification system prior to disposal into the tailings storage facility (TSF). The following table provides an overview of the basis of the process design for the average Life of Mine (LOM) parameters.

Description	Unit	15,000 t/d
Annual Ore Feed Rate (ROM feed)	Mtpa	5.325
Operating Days per Year	d/a	355
Daily Ore Feed Rate (ROM feed)	t/d	15,000
Crushing Rate (6,134 hours per year)	tph	868
Ore Sorting Rate (7,838 hours per year)	tph	121
Milling Rate (7,838 hours per year)	tph	624
Gold Head Grade (ROM Feed) – LOM Average	g Au/t	0.97
Design Ore Specific Gravity	t/m^3	2.76
Design Abrasion Index	-	0.23
Design Crushing Work Index	kWh/t	20.0
SMC Drop Weight Index	kWh/m^3	12.95
Design Rod Mill Work Index	kWh/t	22.6
Design Ball Mill Work Index	kWh/t	24.5
Primary Grind P_{80} Size to Secondary Grind	µm	250
Secondary Grind P_{80} Size to Leach	µm	40
Leach System	-	Hybrid CIL
Leach Slurry Density	% solids w/w	45
Total Leach and Adsorption Time - Design	H	30
Elution System	-	Split AARL
Final Tailings Cyanide Destruction Type	-	Air/SO_2
Overall Recovery (LOM Average) – excludes heap leach pad material	% Au	88.5

Table 3 Key Process Design Criteria

1.9 Project Infrastructure

Access to local resources and infrastructure is adequate with a two-lane paved road running past the property and access to Katherine resulting in roughly a 30 minute drive. In addition, Katherine and the surrounding areas offer the necessary support functions that are found in a medium-sized city regarding supplies, accommodations, communications, and hospital with a surrounding population of around 14,000 people.

The property has an existing high-pressure gas line and an electric power line that were used by previous operators. The current mine site area has existing infrastructure such as buildings, temporary camp, raw water dam (RWD), process water ponds (PWP), sediment and run off ponds as well as an existing TSF. The current site is on care and maintenance status conducting operations to manage the water throughout the site with existing pumps and pipes.

Planned infrastructure for the site includes the following:

- Mine Infrastructure will be supplied, installed by a mining contractor and includes – (HME workshop and warehouse, maintenance support facilities, contractor laydown and storage yards, fuel and lubricant farm, explosives storage and facilities, water cart filling point (Turkeys Nest), Mine administration and personnel facilities, information and communication).
- Heap Leach Pad (existing).
- Waste Rock Dump (WRD) (existing and future).
- 250 person Accommodation Camp with all facilities.
- Water Treatment Plant (WTP).
- Waste Rock Water dam.
- Power Supply via existing gas line (supplied by a third-party supplier by contract).
- Pit Dewatering system.
- Communications.
- Gatehouse.
- Emergency Services Building.
- Process Plant and Administration Building.
- Process Plant Workshop and Stores building with offices.
- Reagents Storage Facility.
- Process Plant Control Rooms.
- Sample Preparation and Laboratory.
- Solid and liquid waste disposal facilities, and
- Expanded existing and additional TSF.

1.10 Market Studies and Contracts

The price of gold is the primary factor in determining the Project's profitability and cash flow from operations. The gold price of USD2,500 per gold ounce used in the economic analysis was derived from a combination of sources, including consensus forecasts reflecting a composite of financial institutions, gold prices used in various recent technical reports completed by mining companies, developers and consulting groups, and recent historical price trends.

Vista has no refining or bullion sales contracts in place. Commitments to deliver gold bullion or the equivalent value are presently limited to private royalty agreements in place as of the effective date of this Technical Report Summary. For purposes of the economic analysis, the value of gold associated with these royalties is included in gold sales with an offsetting royalty expense. Vista expects that terms contained within any refining, sales, or other contracts for delivery of gold bullion will be typical of, and consistent with, standard industry practices.

1.11 Environmental Studies, Permitting, Social and Community Impact

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 the "Mining Authorisation" can be issued by the Minister for Mining and Energy, and works can occur. The Mining Authorisation (0331-04) was issued August 2021. After July 1, 2024, MMPs granted under the Mining Management Act (2001) are automatically deemed to hold a Deemed Environmental (Mining) License under the Environment Protection Act (2019), which maintains the prior approval of the MMP and the "Mining Authorisation". The Deemed Environmental (Mining) Licenses must be converted to an Environmental (Mining) License within four years. Vista has commenced the conversion process.

1.11.1 Environmental

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 Project 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. As part of the agreement, the NT Government acknowledged its commitment to rehabilitate the

Project 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.

1.11.2 Human Environment

The Jawoyn Association Aboriginal Corporation (JAAC or Jawoyn Association) has been consulted as part of the planning process for the future of the Project. Vista has a good relationship with the JAAC. Areas of aboriginal significance have been designated, and the mine plan has avoided development in these areas.

Those parts of the JAAC agreement that are within the public domain are presented in this Technical Report Summary; the remaining part of the JAAC agreement, which is confidential, is not presented in this Technical Report Summary.

1.11.3 Potential Emissions, Waste, and Effluents Generated by the Project

Key issues of concern regarding the Project impacts that were addressed in the Environmental Impact Statement (EIS) include:

- Acid rock drainage and metal laden (ARD/ML) seepage and runoff from the WRD, 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 217 ha with an estimated height of 160 m. Final design of the WRD must ensure the structure is safe, stable, not prone to significant erosion, minimizes Acid Mine Drainage (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.

1.12 Capital and Operating Costs Estimates

1.12.1 Capital Cost

Capital costs have been developed from first principles with quotes for all major equipment components. A turnkey engineering, procurement and construction model has been used as the basis for the Project construction. The Technical Report Summary contemplates a 27-month period for engineering, construction and commissioning. Contract mining at an average rate of 32 Mtpa (ore and waste) and a third-party gas-fired generating plant (on a build own operate basis) with an installed capacity of 64 MW are included. Capital costs include a permanent camp facility near the mine site with housing, dining, and recreation facilities.

The closure plan includes re-processing 13 Mt of heap leach pad material from previous operations and then placing that material in the TSF, the revenues from the heap leach pad material has been treated as self-funding reclamation. The heap leach pad material is included in Mineral Reserves.

Summaries of capital costs exclusive of self-funding reclamation proceeds from processing of the heap leach pad material are shown in Table 4.

Capital Expenditure Item	Initial Capital Cost (USD M)	Sustaining Capital (Years 1-30) Cost (USD M)	Heap Leach Pad, Reclamation and Closure Costs[1] [2](USD M)
Mining	$22.03	$28.01	$4.71
Process Plant	$144.80	$46.03	N/A
Project Infrastructure	$83.68	$141.23	$4.41
Site Establishment and Facilities	$36.57	$8.12	N/A
Management, Engineering and EPC Services	$65.22	$8.24	$0.36
Preproduction Costs and Capital Spares	$47.18	N/A	N/A
Reclamation	N/A	$109.57	N/A
Sub-total: Capital Expenditures	$399.48	$341.20	$9.48
Heap Leach Pad, Reclamation and Closure	N/A	N/A	$50.66
Engineering Growth and Contingency (6 - 10%)	$25.03	$35.86	$5.76
Total Capital Costs	**$424.51**	**$377.06**	**$65.90**

Table 4 Capital Expenditures

[1] Excludes cash flows from the reprocessing of Heap Leach Pad ore.
[2] Includes sustaining costs incurred during the reprocessing of the Heap Leach Pad.

1.12.2 Operating Cost

Mining costs have been provided by a well-established Australian contract miner. Power costs are based on a proposal from one of Australia's leading mine site contract power generators on a build own operate basis.

Processing and G&A costs have been developed from first principles with major consumable supply component quotes and competitive Australian labor rates. The operating costs contemplate that approximately 90% of the initial workforce will be contracted on a fly-in-fly-out basis (FIFO) and be housed in a 250-bed permanent camp facility near the mine site.

Summaries of operating costs, before taxes and depreciation, are shown in Table 5.

Operating Cost Description	Units	Years 1- 15	LOM Yr 1-30
Mining Costs	USD/t processed	$18.49	$16.55
Processing Costs[3]	USD/t processed	$17.70	$17.62
G&A Costs	USD/t processed	$2.09	$2.09
JAAC Royalty	USD/t processed	$2.22	$2.08
Wheaton Royalty	USD/t processed	$0.84	$0.73
Refining Costs	USD/t processed	$0.15	$0.14
Total Cash Costs	**USD/t processed**	**$41.49**	**$39.20**

Table 5 Operating Expenditures

1.13 Economic Analysis

Project economics for the 15 ktpd operation are based on inputs developed by GRES, Tetra Tech, Mining Plus, Tierra Group and Vista. Economic results presented in this Technical Report Summary suggest the following conclusions, assuming a 100% equity project, a gold price of USD2,500/oz.

- Mine Life (LOM) — 30 years
- Production Life — 33 years
- Pre-Tax NPV5% — USD1,736 million, IRR: 37.3%
- After-tax NPV5% — USD1,060 million, IRR: 27.8%
- Payback (After-tax) — 2.7 years
- JAAC Royalty Paid — USD342 million
- Wheaton Royalty Paid — USD118 million
- Northern Territory Taxes Paid (Royalties) — USD398 million
- Australian Company Corporate Taxes Paid — USD1,083 million
- Cash costs (including JAAC and Wheaton Royalties) — USD1,438/oz-Au

[3] Incudes water management costs of approximately USD0.78/t processed.

Project cost estimates and economics were prepared on an annual basis. Based upon design criteria presented in this Technical Report Summary, the level of accuracy of the estimate is considered ±10-15%.

Costs and economic results are presented in Q2 2025 U.S. dollars unless otherwise stated. No escalation has been applied to capital or operating costs. The 5% discount rate used is a goldmining industry standard. in North America generally used for comparability purposes among projects; it is not intended to fully reflect consideration of cost of capital, risk adjustments, or other factors.

Technical and 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.

1.14 Conclusion

This Technical Report Summary shows that the mine plan is technically achievable and economically viable taking into consideration all material modifying factors. The resultant Mineral Reserves are also reasonable and achievable.

The mining operations will be executed by a tier 1 Australian contract mining company, selected for its capability to manage large-scale operations and maintain high equipment availability. The contractor will utilize a fleet comprising 400 tonne class hydraulic excavators and 190 tonne class rigid-frame haul trucks, supported by ancillary equipment including loaders, dozers, graders, and water carts. The contractor will also provide site mining infrastructure and all personal to operate and maintain all mining equipment, while ensuring its supervision and operations management.

Reasonable mine designs, mine production schedules, and mine costs have been developed for the Project. Costs for mining and mining infrastructure have been provided based on the analysis of Technical Report Summary mine schedule completed by a tier 1 Australian mining contractor and consider local site and NT requirements, and availability of resources such as equipment and labor. These mining costs have incorporated within the overall Project financial model.

Several opportunities and risk have been identified within this Technical Report Summary, which can be managed as the project progresses its development through to execution and pre-production stages of development towards commencement of the mining operation.

The processing plant has been designed to treat 5.325 Mtpa (15 ktpd). During the initial phases of this Technical Report Summary, GRES reviewed the previous designs and raised several queries in relation to the ore sorting, grind size, recovery method and historical test work. GRES addressed these queries during this study with some revised approaches.

Most of the capital and operating costs are within the front end of the plant. The plant has a restricted front-end layout due to the limited available land and so is restricted in the ability to expand this area of the plant due to the waste dumps, water course and other restricted areas. The post grinding areas can be expanded relatively easily and will require some extensive demolition of the remaining existing facility to utilize this layout space.

When laying out this plant the remanent facilities on site were avoided as much as possible to minimize any major demolition and reduce front end capital costs.

2. INTRODUCTION

Vista operates in the gold mining industry. The Company's flagship asset is its 100% owned Mt Todd Gold Project in the Northern Territory (NT) Australia. All major environmental and operating permits have been approved for the 50 ktpd project which was the basis of past technical studies. Modifications and applications to existing approvals with 15 ktpd project have been initialized.

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 Purpose of the Technical Report Summary

This Technical Report Summary was prepared in accordance with the disclosure requirements of Subpart 229.1300 of Regulation S-K 1300 Technical Report Summary for Vista by GRES, to be attached as an exhibit to support mineral property disclosure, including Mineral Resource estimates and Mineral Reserve estimates for the Mt Todd Gold Project. The quality of information, conclusions, and estimates contained herein are consistent with the level of effort involved in GRES's services, based on:

1. information available at the time of preparation,
2. data supplied by outside sources, and
3. the assumptions, conditions, and qualifications set forth in this Technical Report Summary.

This Technical Report Summary provides Mineral Resource and Mineral Reserves estimates, and a classification of Mineral Resources and Mineral Reserves in accordance with the definitions in subpart 229.1300 – Disclosure by Registrants Engaged in Mining Operations in Regulation S-K 1300 (S-K1300).

This Technical Report Summary is a comprehensive study of a range of options for the technical and economic viability of a mineral 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 Reserves 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.

This Technical Report Summary contains forward-looking statements; refer to the note regarding forward-looking statements at the front of the Technical Report Summary.

2.2 Previous Technical Report Summaries

This Technical Report Summary supersedes the previous Technical Report Summary, "S-K 1300 Technical Report Summary Mt Todd Gold Project - 50,000 tpd Feasibility Study Northern Territory, Australia with effective date March 12, 2024 and issue date of April 16,2024.

2.3 Background Information

Vista retained GRES, to coordinate several consultants under the supervision of Vista to prepare this Technical Report Summary. The FS (Technical Report Summary) evaluates a development scenario of a 15 ktpd processing facility.

The 15 ktpd operation includes:

- Average annual gold production of 153,000 ounces during years 1-15 and 146,000 ounces over the 30-year life of mine.
- Average ore grade of 1.04 grams gold per tonne ("g Au/t") over the first 15 years of operations and 0.97 g Au/t over the life of mine.
- LOM average gold recovery of 88.5% from 3-stage crush, single-stage sort, 2-stage grind, and CIL recovery circuit.
- Contract mining and third-party power generation reduce capital costs and operational risks.
- Initial capital requirements of USD425 million.

2.4 Detailed Personal Inspections

1. A site visit was performed by Tetra Tech professionals, including Kira Johnson, the Qualified Person (QP) for the Geology studies and Mineral Resource estimation of this Technical Report Summary on November 7-8, 2024. During the visit, Tetra Tech found a comprehensive drill hole database comprised of drill core, photographs of the drill core, assay certificates and results, and geologic logs. Areas included in the visit were the office facilities, the Batman and Quigleys deposits, the core logging facility, and the drill rig. Tetra Tech staff also visited the Northern Australian Laboratory (NAL) in Pine Creek during this site inspection.

2. A site visit was performed by Mining Plus Colin McVie who conducted a site inspection from March 11th to 14th, 2025. The visit focused primarily on assessing the current condition of the Batman Pit, with particular attention to pit wall integrity and structural features. Additional inspections were carried out on the existing WRD, site infrastructure, and areas designated for future development.
3. A site visit was conducted by Marthinus Sonnekus, Technical Executive Rock Mechanics from WSP, to the Project on 18 January 2025. The purpose of the site visit was to gain a better understanding of the performance of the existing Batman pit slopes.
4. Justin Knudsen of Tierra Group visited and inspected the Project on March 12–14, 2025. Mr. Knudsen inspected the existing Tailings Storage Facility 1 (TSF 1) and the proposed site for Tailings Storage Facility 2 (TSF 2).
5. Brendan Mulvihill of GRES last visited and inspected the Project on March 12th-14th, 2025. Mr. Mulvihill inspected the existing site infrastructure and process facility.
6. As part of previous issued technical studies, Brad Bijold, of Tetra Tech visited and inspected the existing water reservoirs, water dam and proposed water treatment plant location in 2018.
7. As part of previous issued technical studies, Deepak Malhotra PhD SME RM visited and inspected the existing Heap Leach pad and existing site infrastructure in 2018.
8. As part of previous issued technical studies, Vicki J. Scharnhorst of Tetra Tech visited and inspected the property in 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.

The QPs consider that the site visits conducted prior to 2025 can be regarded as current personal inspections on the basis that the work completed on the Project since that time has been reviewed and the QPs are of the opinion that the limited work carried out on the Project 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 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 reviewed 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.

2.5 Capability and Independence

GRES is an Australian Stock Exchange (ASX) listed engineering and construction company with a global footprint and has previously been involved with feasibility studies and project delivery in Australia and other locations regionally. Mining Plus provides independent mining advisory services to the global mining and finance sectors. Within its core expertise it provides independent technical reviews, resource evaluation, mining engineering and mine evaluation services to the resources and financial services industries. Tetra Tech, Inc. is an American consulting and engineering services firm. Tetra Tech provides consulting, engineering, program management, and construction management services in the areas of water, environment, infrastructure, resource management, energy, and international development. Tierra Group is a multinational engineering design and consulting engineering firm known worldwide for its expertise in civil, geotechnical and water resource engineering, specialized in the mining industry. WSP is one of the world's top professional services firms, bringing together some of the brightest engineers, advisors and scientists from across the globe. WSP's deep understanding of mine geotechnical assessment and risk management, to provide site specific, practical solutions to a broad range of projects in the mining industry.

All opinions, findings and conclusions expressed in this Technical Report Summary are those of the Qualified Persons and their specialist advisors.

2.6 Reliance on Other Experts

The following individuals, by virtue of their education, experience and professional association, are considered QPs as defined in the subpart 229.1300 of Regulation S-K, for this Technical Report Summary, and are members in good standing of appropriate professional institutions.

2.7 Sources of Information and Data

The primary technical documents and files relating to the Project, including previous technical reports, research documents and historical available information on the Project, that were used in the preparation of this Technical Report Summary are listed in Section 24-References.

2.8 Units of Measure

The metric system has been used throughout this Technical Report Summary. 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). The summary capital and operating cost currency is in Q2 2025 U.S. dollars (USD) unless otherwise stated. There are instances where Australian dollar (AUD) has been used, as the Project is in Australia, and the currency exchanged used is detailed within.

3. PROPERTY DESCRIPTION

3.1 Location

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

The native coordinate system of the topography is MGA94 Zone 53, and for the Mineral Resource and Mineral Reserve estimates MGA94 zone 53 is used as the official coordinate system. The surveyed drillhole collar coordinates, once translated to MGA94 zone 53 agree well with the topographic map.

3.2 Land Tenure

Vista Australia is the holder of four mineral leases (ML) MLN 1070, MLN 1071, MLN 1127, and ML 31525 comprising approximately 55.44 km^2. In addition, Vista Australia controls exploration licenses (EL) EL 29882, EL 29886, EL 30898, currently comprising approximately 1,337 km^2. In early July 2025, Vista surrendered EL 32004 and withdrew its application for EL 32005 as Vista deemed these licenses immaterial of no economic value for the Company. Figure 5 illustrates the general location of the tenements and the position of the Batman and Quigleys deposits. A general arrangement of the overall Project layout is provided in Figure 6.

The MLs and ELs were obtained through direct agreements with the Northern Territory Government. Each ML and EL is subject to annual reporting and rent payment; the ELs are also subject to an annual spending covenant. Table 6 summarizes the provisions of the MLs and ELs held by Vista Australia.

Tenement	Km^2	Expiry Date	Annual Rent (AUD)	Annual Spending Covenant	Rent Due
ML 31525	1.6	3-Sep-2042	$4,182	N/A	3-Sep
MLN 1070	39.8	4-Mar-2043	$99,832	N/A	4-Mar
MLN 1071	13.3	4-Mar-2043	$33,457	N/A	4-Mar
MLN 1127	0.8	4-Mar-2043	$2,282	N/A	4-Mar
Total MLs	**55.4**	**N/A**	**$139,753**	**N/A**	**N/A**
EL29882	555.5	15-Sep-2025	$44,632	$106,875	15-Sep
EL29886	594.6	15-Sep-2025	$50,536	$137,428	15-Sep
EL30898	186.7	2-May-2026	$14,620	$14,842	2-May
Total ELs	**1,337**		**$109,788**	**$259,145**	

Table 6 Provision of the MLs and ELs Held by Vista Australia

Vista holds Deemed Environmental (Mining) License valid until 30 June 2028. No violations or fines have been imposed to date. Vista Australia must convert each of its Deemed Environmental (Mining) Licenses to a Environmental (Mining) License prior to expiration. Additionally, Vista Australia has applied for and is working with the Aboriginal Areas Protection Authority (AAPA) to obtain an additional AAPA Authority Certificate to complement the Authority Certificates already held for the MLs and ELs.

3.3 Lease and Royalty Structure

Vista Australia entered into a lease agreement (the Lease Agreement) with the NT government for an initial term of five years commencing January 1, 2006, with an extension of five years at Vista Australia's option and three additional years upon the application of Vista Australia and with the approval of the NT government. Pursuant to the conditions of the first five-year term of the Lease Agreement, Vista Australia undertook a comprehensive technical and environmental review of the Project to evaluate site environmental conditions and developed a program to stabilize the environmental conditions and minimize offsite contamination. Vista also reviewed the water management plan and made recommendations and developed a 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 (USD248,000) 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 and possible, NT-sourced labor and services during the period of the Lease Agreement in connection with the Project, and further, in connection with any proposed mining activities prepare and execute a local Industry Participation Plan.

Pursuant to the JAAC Agreement, Vista was required to JAAC common shares of Vista 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 leases, including the mining lease 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 (USD33,000). Vista also pays the JAAC AUD5,000 (USD3,300) 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 proceeds production royalty that can vary between 1/8% and 2% depending on gold price and the AUD:USD foreign exchange rate. 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 Gold Project. Pursuant to the terms of the Royalty Agreement, Wheaton is entitled to receive 1% of the gross revenue from Mt Todd Gold Project (the “Wheaton 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 Wheaton Royalty shall increase annually at a rate of up to 0.13% to a maximum Wheaton Royalty rate of 2%. Any annual increases beginning April 1, 2028 shall be reduced on a pro rata basis to the extent that the Project has initiated operations but has yet to achieve a completion test at an average daily processing rate of 15 ktpd. 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 Wheaton Royalty associated with Vista Australia delivering 3.47 million gold ounces to a third party. The Wheaton Royalty is payable on production from the Project 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 Reserves at the Project.

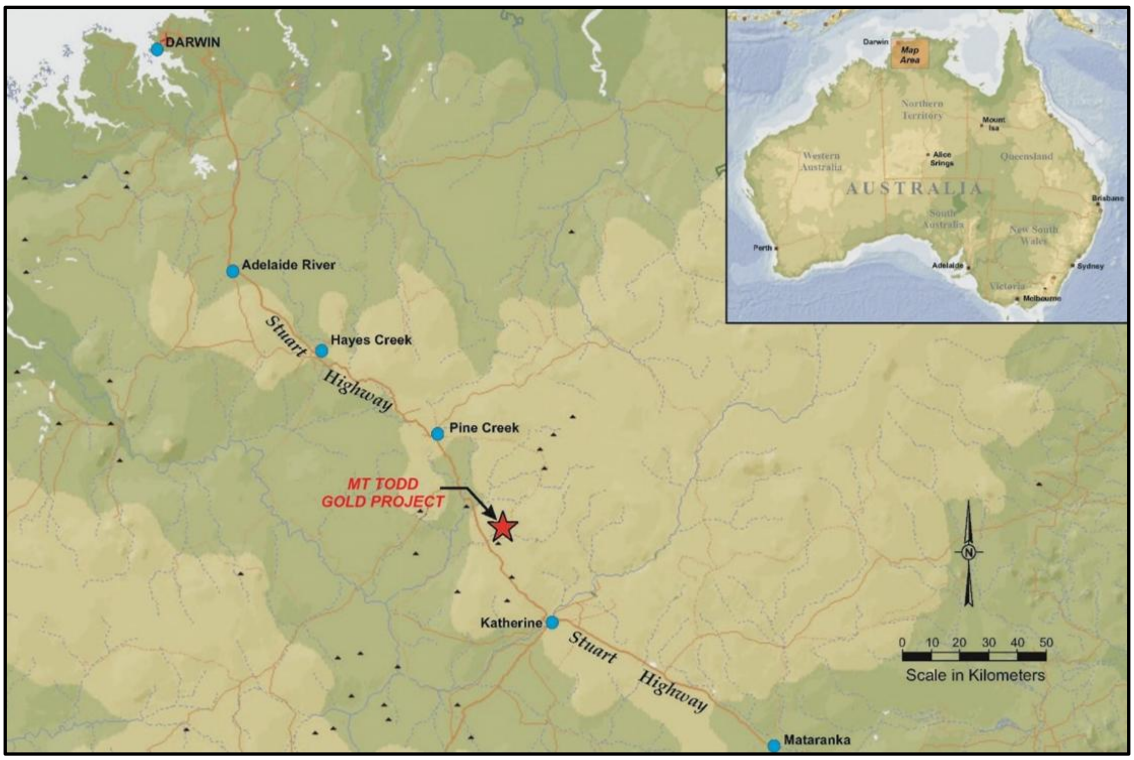
3.4 Ownership

Vista is in sole possession of the title and rights to perform work on the Project. Surface access is granted through Vista’s agreement with the JAAC. Exploration or other similar activities require a MMP to be submitted to the Department of Regional Development, Primary Industry, Fisheries and Resources (DRDPIFR) with approvals typically occurring in thirty days or less. Vista received approval of the Mt Todd Project MMP in June 2021 based on the 2020 PFS and holds all major permits required to start development. Modifications to align with existing approvals with the 2024 FS (automatically granted a Deemed

Environmental (Mining) License under the Environmental Protection Legislation Amendment Act) and this Technical Report Summary have been initiated.

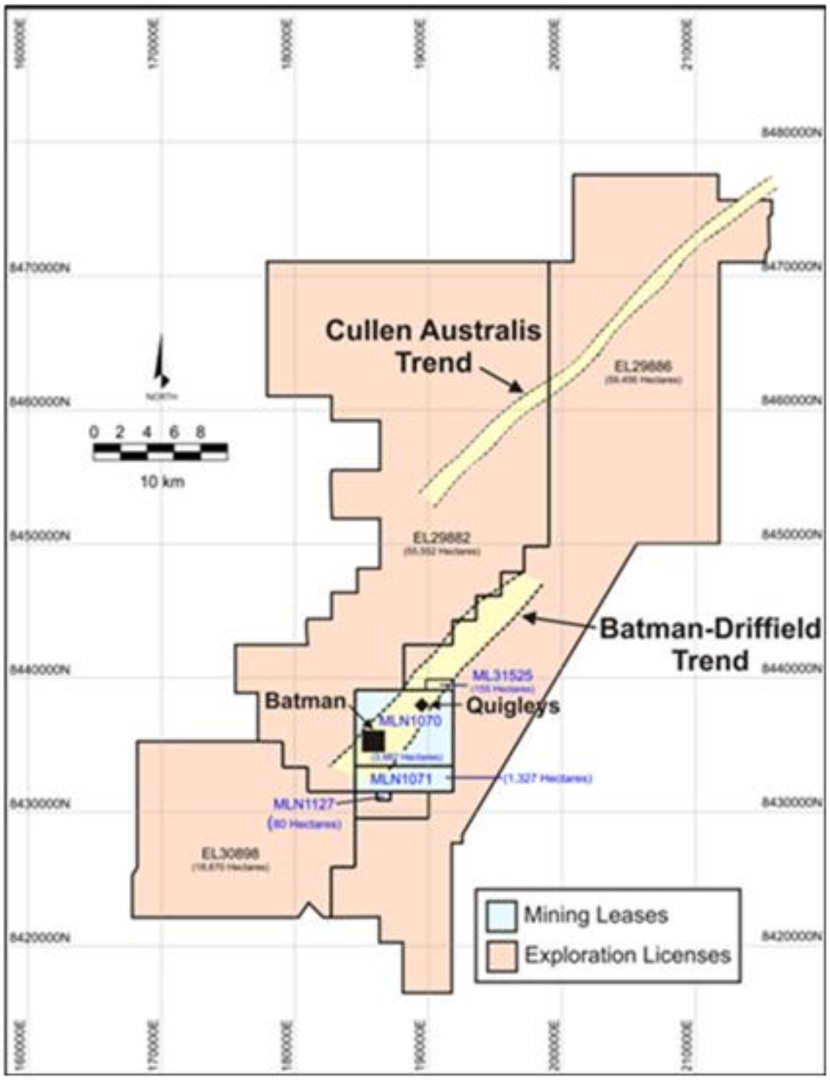
As an approved mining operation Vista was automatically granted a Deemed Environmental (Mining) License under the Environmental Protection Legislation Amendment Act. The Deemed Environmental (Mining) License is valid until June 30, 2028. Prior to expiration Vista must apply for a replacement Environmental (Mining) License.

Figures 4 to 6 show the Project's location , mining and exploration licenses and general Project's layout.



Source: Prepared by Vista,2020

Figure 4 Mt Todd Gold Project Location, MGA94 Zone 53 Coordinates



Source: Prepared by Vista.; updated July 2025

Figure 5 Mining Leases and Exploration Licenses, Mt Todd Gold Project, MGA94 Zone 53 Coordinates

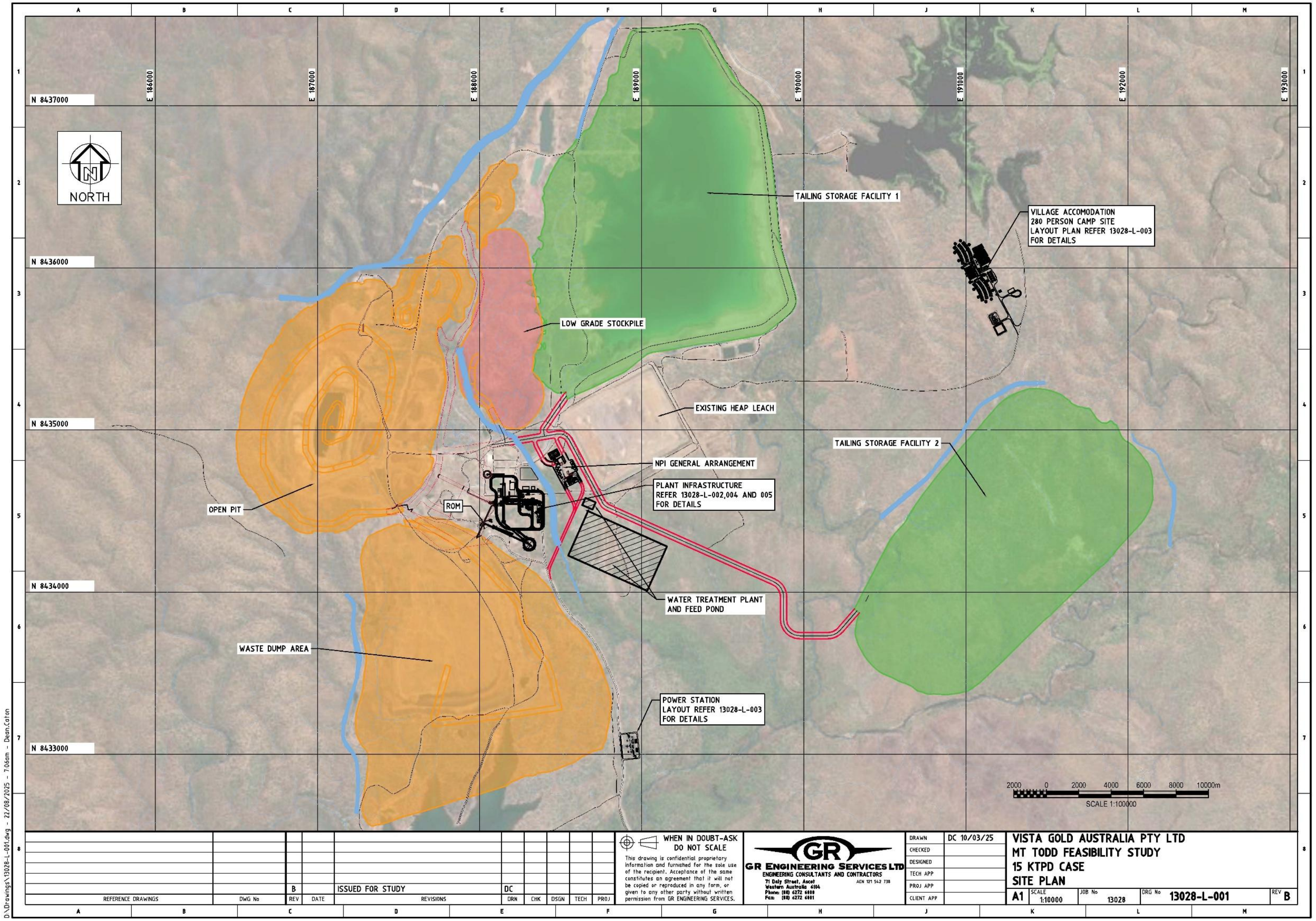


Figure 6 Overall Site Layout for the Project, MGA94 Zone 53 Coordinates

3.5 Risks

Vista is in sole possession of the title and rights to perform work on the Project. Surface access is granted 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. As an approved mining operation Vista was automatically granted a Deemed Environmental (Mining) License under the Environmental Protection Legislation Amendment Act. The Deemed Environmental (Mining) License is valid until June 30, 2028. Prior to expiration Vista must apply for a replacement Environmental (Mining) License.

Risks to access and title are minimal due to secure tenure and strong relationships with local stakeholders. However, delays in obtaining the additional AAPA Authority Certificate or converting the Deemed Environmental (Mining) License to an Environmental (Mining) License in a timely manner could impact Project timelines.

An application for a modified Aboriginal Areas Protection Authority Certificate was submitted on June 19, 2024. Consultation with the aboriginal authorities has been initiated. Based on prior experience with the authorities, an Authority Certificate is expected to be obtained as has been customary in the past.

4. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

4.1 Access

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.

The closest airport to Katherine, is the Katherine Tindal Civilian Airport, which is located approximately 15 km south of the town center. The airport operates as a shared facility with the Royal Australian Air Force Base at Tindal, with a dedicated civilian passenger terminal managed by the Katherine Town Council. Though it is the most convenient option for reaching Katherine directly, it offers limited services, with most commercial flights connecting via Darwin.

4.2 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, there are no limiting weather or accessibility factors, however, intensive dewatering activities will be required after large precipitation events, particularly in the wet season (approximately November to April).

4.3 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, and communications.

The Project is readily accessible (approximately 250 km from Darwin) and conveniently located near well-established population centers. Project 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 property has an existing high-pressure gas line and an electric power line that was used by previous operators. The current mine site area has existing infrastructure such as buildings, temporary camp, raw water dam, process water ponds, sediment and run off ponds as well as an existing Tailing Storage Facility (TSF). The current site is on care and maintenance status conducting operations to manage the water throughout the site with existing pumps and pipes. In addition, wells for potable water and a dam for process water are also located on or adjacent to the 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.

4.4 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 m above sea level in the area of the previous and planned site and waste dump

5. HISTORY

The Project area has several 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 Project 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. Pegasus progressively increased their shareholding until they acquired full ownership of Zapopan in July 1995.

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

Historical production is shown in Table 7.

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 ownership. The QPs and Vista consider the historical estimates to be relevant but not current.*

Table 7 Heap Leach – Historical Actual Production

Phase II involved expanding to 8 Mtpa 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 was AUD232 million.

Design capacity was never achieved due to inadequacies in the crushing circuit. A throughput rate of just under 7 Mtpa 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 USD400 in early 1996 to below USD300 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.

5.1 History of Previous Exploration

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

The Yinberrie Wolfram field, discovered in 1913, is located 5 km west of Mt Todd. Tungsten, molybdenum and bismuth mineralization was discovered in greisenized aplite dykes and quartz veins in a small stock of the Cullen Batholith. Recorded production from numerous shallow shafts is 163 tonnes of tungsten, 130 kg of molybdenite and a small quantity of bismuth. Exploration for uranium began in the 1950s. Small uranium prospects were discovered in sheared or greisenized portions of the Cullen Batholith in the vicinity of the Edith River. The area has been explored previously by Esso for uranium without any economic success. Australian Ores and Minerals Limited (AOM) in joint venture with Wandaroo Mining Corporation and Esso Standard Oil took out a number of mining leases in the Project area during 1975. Initial exploration consisted of stream sediment sampling, rock chip sampling, and geological reconnaissance for a variety of commodities. A number of geochemical anomalies were found primarily in the vicinity of old workings. Follow-up work concentrated on alluvial tin and, later, auriferous reefs. Backhoe trenching, costeaning, and ground follow-up were the favored mode of exploration. Two diamond drill holes were drilled at Quigleys. Despite determining that the gold potential of the reefs in the area was promising, AOM ceased work around Project. The Arafura Mining Corporation, CRA Exploration, and Marriaz Pty Ltd all explored the Project area at different times between 1975 and 1983. In late 1981, CRA Exploration conducted grid surveys, geological mapping and a

14-diamond drill hole 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, not NI 43-101 compliant). Subsequent negotiations between the Mt Todd Joint Venture partners (Billiton and Zapopan) and Pacific Gold Mines resulted in the acquisition of this ground and incorporation into the joint venture. Table 8 presents important historical events in a chronologic order.

1986	
October 1986 – January 1987:	Conceptual Studies, Australia Gold PTY LTD (Billiton); Regional Screening (Higgins); Ground Acquisition, Zapopan N.L.
1987	
February:	Joint Venture finalized between Zapopan and Billiton.
June-July:	Geological Reconnaissance, Regional BCL, stream sediment sampling.
October:	Follow-up BCL stream sediment sampling, rock chip sampling and geological mapping (Geonorth).
1988	
Feb-March:	Data reassessment (Truelove).
March-April:	Gridding, BCL grid soil sampling, grid-based rock chip sampling and geological mapping (Truelove).
May:	Percussion drilling Batman (Truelove) - (BP1-17, 1475m percussion).
May-June:	Follow-up BCL soil and rock chip sampling (Ruxton, Mackay).
July:	Percussion drilling Robin (Truelove, Mackay) – RP 1-14, (1584m percussion).
July-Dec:	Batman diamond, percussion and reverse circulation (RC) drilling (Kenny, Wegmann, Fuccenecco) - BP18-70, (6263m percussion); BD1-71, (8562m Diamond); BP71-100, (3065m R.C.).
1989	
Feb-June:	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.
June:	GP1-8, BP108, TP1-7 (202m diamond, 3090m RC); TR1-159 (501m RAB). Mining lease application (MLA's 1070, 1071) lodged.
July-Dec:	Mineral 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 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 and General Gold.
1999 - 2000	
March - June	Operated by a joint venture comprised of Multiplex Resources and General Gold Operations ceased in July 2000, Pegasus, through the Deed Administrators, regained possession of various parts of the mine assets 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 remain at the site.
2000 - 2006	
	The Deed Administrators, Pegasus, the government of the NT, and the JAAC held the property.
2006	
March	Vista acquired mineral lease rights from the Deed Administrators.
2006-2025	
	Vista completed drilling campaigns, produced environmental, economic, geotechnical, regulatory, and other such studies. Vista undertook remediation of Batman Pit water. A series of S-K 1300 and NI 43-101 technical reports were produced over the period with increasing detail.

Table 8 Property History

5.2 Historical Drilling

The following discussion centers on the historical drill hole databases that were provided to Tetra Tech for use in this Technical Report Summary. Based on the reports by companies, individuals and other consultants, it is the QPs' opinion that the drill hole databases used as the basis of this Technical Report Summary contain all relevant available data. Tetra Tech is unaware of any drill hole data that has been excluded from this Technical Report Summary.

5.2.1 Batman Deposit

There are 730 historical drill holes in the Batman deposit assay database. Figure 7 shows the drill hole locations for the Batman deposit. These drill holes include 225 Diamond Drill hole core (DDH), 435 Reverse Circulation holes (RVC), and 70 Open rotary holes (OP). Nearly all the DDH and RVC holes were inclined 60° to the west. Samples were collected in one-meter intervals. DDH holes included both HQ (88.9 mm drill rod) and NQ (69.9 mm drill rod) 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 drill holes were RVC and OP holes. All drill holes collars were surveyed by the mine surveyor. Down-hole surveys were conducted on most drill holes using an Eastman single shot instrument. All drill holes 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 drill holes from modeling the Batman deposit because the assays from these drill holes 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 rock].* 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.

5.2.2 Drillhole Density and Orientation

Pegasus was aware of the potential problem of drill hole 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 drill holes 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 drill hole database numbered 730 drill holes. It is not known if any drill holes 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 Pegasus's 50 m drill sections through the Batman deposit and saw that there was a marked decrease in drill hole 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 drill hole spacing in the south of 1,000 N on the 954 RL bench plan approached 80 m x 80 m. Pegasus was able resolve this problem by using very long search ranges in its grade estimation. In the main ore zone, Pegasus used maximum search distances in the north and east directions of nearly 300 m.

Another potential problem related to drilling was the preferred orientation of the drill holes. Most of the drill holes 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 drill hole database for the 1994 MRT model because their assay results appeared to be too low compared to other drill hole orientations. If vertical drill hole orientations were actually underestimating the gold content during exploration drilling, the vertical and often wet blast holes, which are used for ore control, might pose a similar problem and will need to be addressed prior to commencing any new mining on the site.

5.2.3 Quigleys Deposit

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

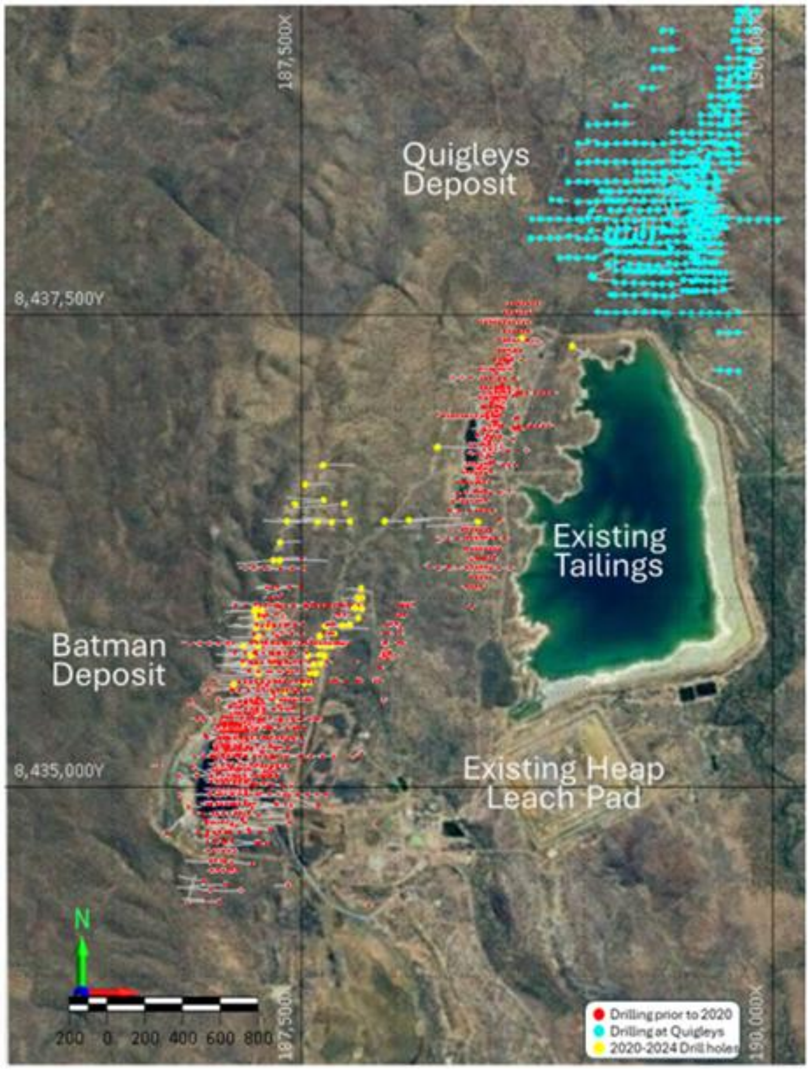


Figure 7 Drillhole Location Map – Batman and Quigleys Deposits, Tetra Tech 2025

5.3 Historical Sampling Method and Approach

NQ core intervals were cut 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 Mineral 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 drill hole database for grade estimation of the central area of the Batman deposit.

Tetra Tech did not witness historical drilling and sampling personally and 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.

5.4 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 accepted 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 drill holes. The procedure involved sawing the NQ core lengthwise in half. HQ core was quartered. RVC samples were riffle split on site and a 3- to 4-kg sample was sent to the laboratory for analyses. Pincock Allen and Holt (PAH) stated that they witnessed the sample preparation process at a number of steps and concurred with the methods in use (PAH, 1995).

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

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

5.4.2 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 Mineral Resource estimation exercise is acceptable and meets industry standards for accuracy and reliability.

5.4.3 Security

The QP responsible for this section is unaware of any 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.

5.5 Historical Process Description

The Batman deposit is a large low-grade gold deposit. The average grade of the gold mineralization is approximately 1 g Au/t. The gold mineralization occurs in a hard, uniform greywacke host and is associated with sulfide and silica mineralization which has resulted from deposition along planes of weakness that had opened in the host rock. Gold is 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 8.

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 2.6 mm. 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 CIL
- 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 9.
- 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.

5.6 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.

5.6.1 Crushing

The four-stage crushing circuit was supposed to produce a product with P_{80} of 2.6 mm. Also, historically the tonnage was projected to be 8 Mtpa 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 Mtpa 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.

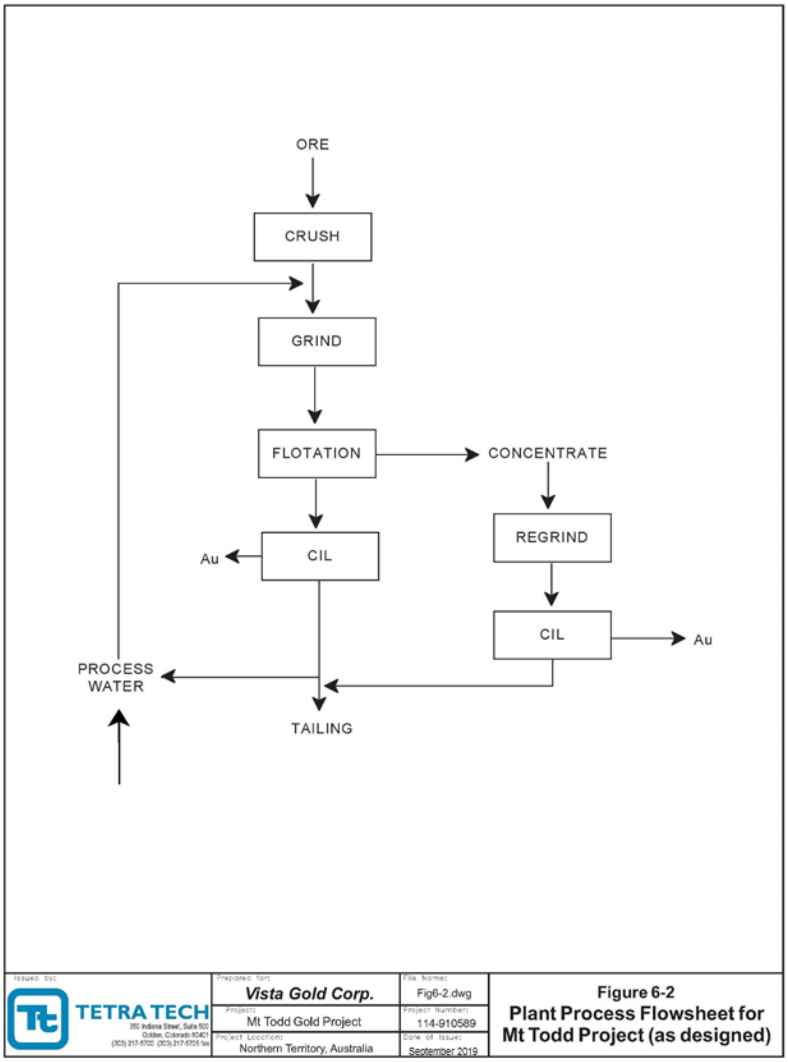


Figure 8 Plant Process Flowsheet for Project as Designed, Prepared by Tetra Tech 2019

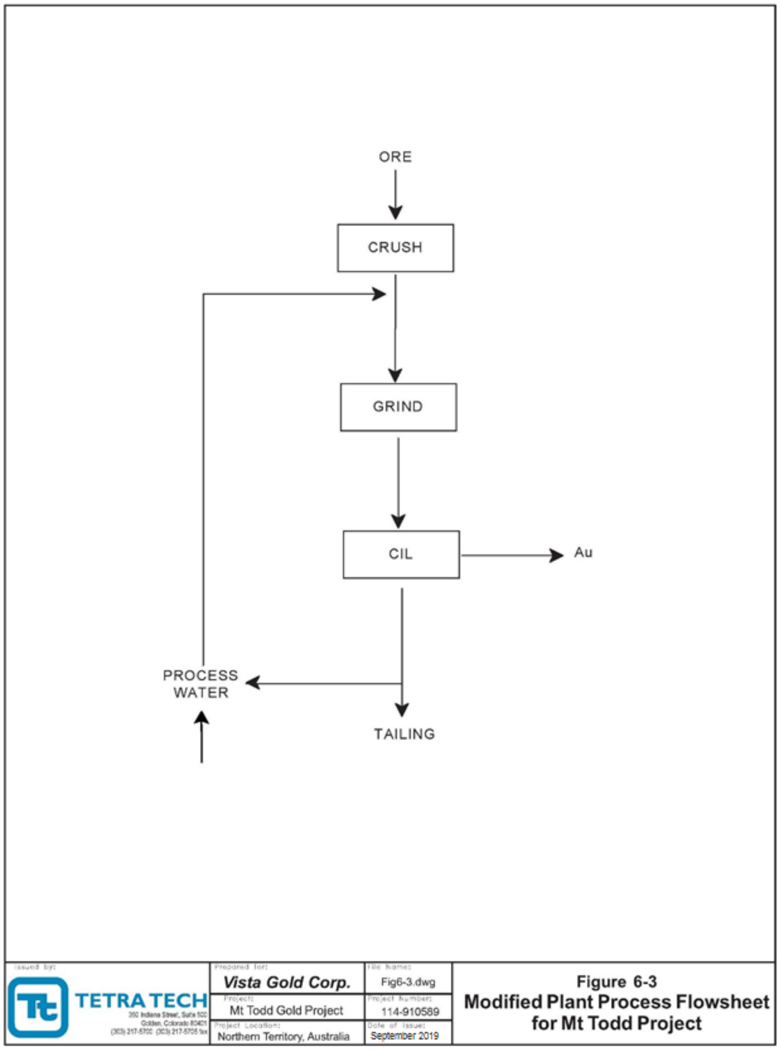


Figure 9 Modified Plant Process Flowsheet for Project Prepared by Tetra Tech 2019

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.

5.6.2 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 test work, 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.
- 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.

5.6.3 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.

6. GEOLOGICAL SETTING, MINERALIZATION AND DEPOSIT

6.1 Geological and Structural Setting

The Project is situated within the southeastern portion of the Early Proterozoic Pine Creek Geosyncline (see Figure 10). Meta-sediments, granitoids, basic intrusives, acid and intermediate volcanic rocks occur within this geological province.

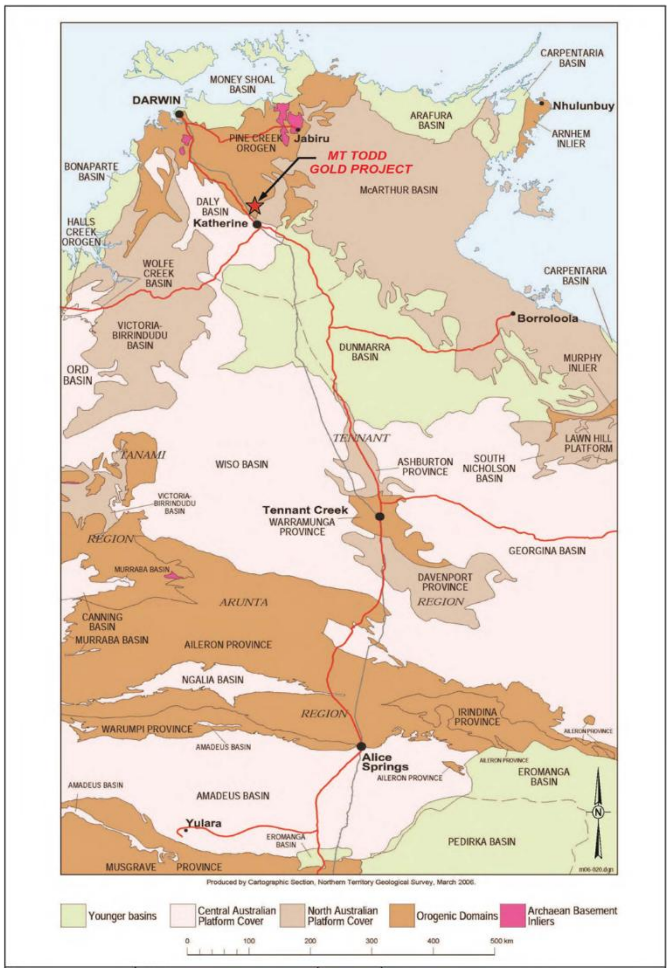


Figure 10 General Geologic Map, Tetra Tech 2024

Within the Project 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 crossbedding. 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, which are folds in bedding. 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, which is associated with the second deformation event in the area. 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. A stratigraphic column of the Project area is shown in Figure 11.

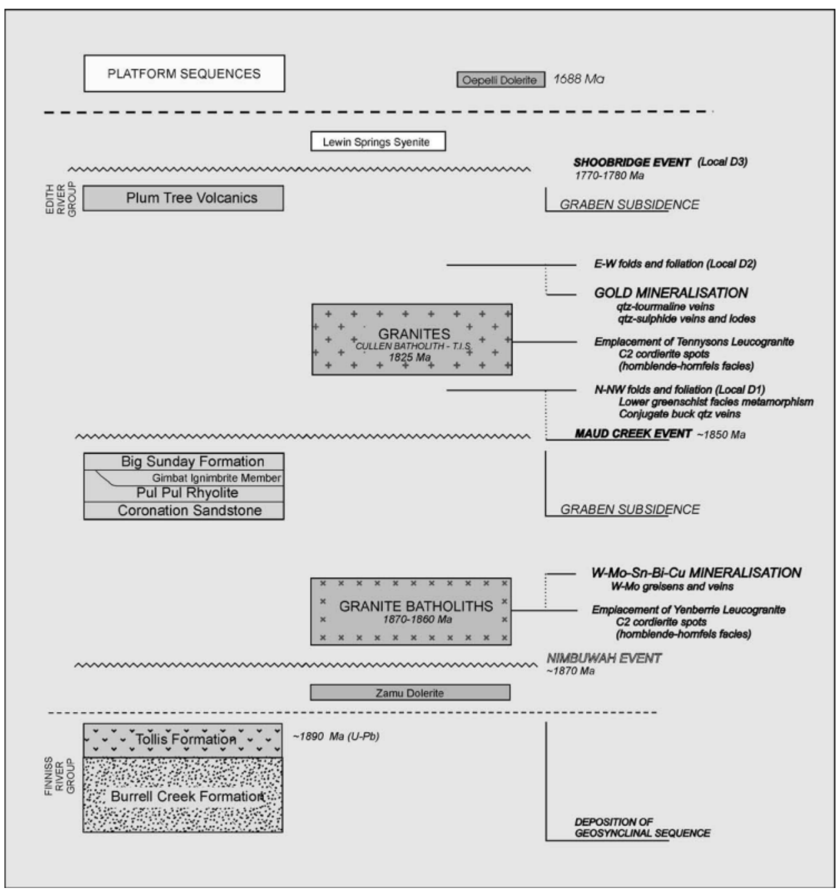


Figure 11 ***Stratigraphic Column of the Mt. Todd Project Area (Hein, 2003)***

6.2 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 9 below from south to north (oldest to youngest).

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

Table 9 Geologic Codes and Lithologic Units

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.

6.3 Mineralization

A variety of mineralization styles occur within the Project area. Of greatest known economic significance are auriferous quartz-sulfide vein systems. These vein systems include the Batman, Jones, Golf, Quigleys and Horseshoe prospects, which occur within a north-northeast trending corridor, and are hosted by the Burrell Creek Formation. Tin occurs in a north-northwest trending corridor. The tin mineralization comprises cassiterite, quartz, tourmaline, kaolin, and hematite bearing assemblages, which occur as bedding 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 2,400 m along strike, 600 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.

6.4 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 (D1 and D2), during cooling and crystallization of the Tennysons Leucogranite and early in D2 (Hein, 2006). It is speculated that pluton cooling resulted in the development of effective tensile stresses that dilated and/or reactivated structures generated during pluton emplacement and/or during D1 (Furlong et al., 1991, as cited in Hein, 2003), or which fractured the country rock carapace as is typical during cooling of shallowly emplaced plutons (Knapp and Norton, 1981, as cited in Hein, 2003). In particular, this model invokes sinistral reactivation of a northeasterly trending channelization basement strike–slip fault, causing brittle failure in the upper crust and/or dilation of existing north–northeasterly trending faults, fractures, and joints in competent rock units such as meta-greywackes and siltstones. The generation of dilatant structures above the basement structure (i.e., along a northeasterly trending corridor overlying the basement fault), coupled with a sudden reduction in pressure, and concomitant to brecciation by hydraulic implosion (Sibson, 1987; Je´brak, 1997; both as cited in Hein, 2003) may have facilitated channelization of predominantly metamorphic fluid in the intermediate contact metamorphic aureole (possibly supra-hydrostatic-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.
- Channelization of predominantly metamorphic fluid in the intermediate contact metamorphic aureole into dilatant structures.

The deposits are like other gold deposits of the Porphyry Copper Gold (PCG) origin 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.

6.4.1 Batman Deposit

6.4.1.1 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.

6.4.1.2 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.

6.4.1.3 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.

6.4.1.4 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.

6.4.1.5 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.

6.4.1.6 Bedding Parallel Mineralization

Bedding parallel mineralization occurs in rock types SH22 to SH20 (as defined in Table 9) 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.

6.4.2 Quigleys Deposit

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

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

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

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

7. EXPLORATION

7.1 Exploration Activities Previous to Vista

Since acquiring the mining leases and exploration licenses for the Project, 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, Golden Eye, 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 10 details soil geochemical samples collected on the exploration licenses (ELs) by year.

Year	Soils	Samples Collected
2008	0	164
2009	1,333	45
2010	3,135	224
2011	1,925	79
2012	2,312	295
2013	572	51
2014	2,601	143
2015	841	53
2016	241	27
2017	1,098	78
2018	341	132
2019	313	170
2020	278	9
2021	0	11
2022	685	71
2023	1,500	44
2024	0	0
Total Samples	**17,175**	**1,596**

Table 10 Exploration Soil sampling

Within the same ELs, Vista obtained 654 soil samples and 222 rock-chip samples in an exploration program between March 2, 2018 and October 7, 2019. Table 11 lists the type, sample count and general location. Table 12 presents information on known exploration prospects.

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 11 Exploration sampling between 2018 and 2019 by target area

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

Year	Drill Hole	Location		Zone	GDA94 Coords		Tasks Completed		
		Prospect	Lease No		Easting	Northing	RL	Depth	Rehab Status
	LL11-005	Limestone Quarry	EL28321	52L	814050	8426300	95	61	Closed
	LL11-006	Limestone Quarry	EL28321	52L	814050	8426250	95	60	Closed
	GE11-001	Goldeneye	EL29886	53L	200300	8455555	177	195	Closed
	GE11-002	Goldeneye	EL29886	53L	200240	8455455	182	351	Closed
	GE11-003	Goldeneye	EL29886	53L	200350	8455455	182	241	Closed
	GE11-004	Goldeneye	EL29886	53L	200400	8455500	186	267	Closed
	GE11-005	Goldeneye	EL29886	53L	200400	8455555	186	240	Closed
2012									
	SD12-001	Snowdrop	EL29882	53L	195169	8457484	171	219	Closed
2015									
	SD15-001	Snowdrop	EL29882	53L	195164	8457302	170	250	Closed
	SD15-002	Snowdrop	EL29882	53L	195142	8457248	170	250	Closed
	SD15-003	Snowdrop	EL29882	53L	195305	8457599	170	250	Closed
	WD15-001	Wandie	EL29882	53L	190947	8455709	169	46	Closed
	WD15-002	Wandie	EL29883	53L	190920	8455696	168	100	Closed
	WD15-003	Wandie	EL29884	53L	190890	8455679	167	135	Closed
2016									
	WD16-001	Wandie	EL29882	53L	190859	8455663	166	204	Closed
								6,445	
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	

Table 12 Exploration prospects

7.1.1 Golden Eye Target

At Golden Eye, an initial 100 m x 100 m soil program identified two 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 veins 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 drill holes on the target. All four drill holes intersected strong sulfide mineralization associated with laminated Fe rich Burrell Creek Formation, with interesting concentrations of copper, lead zinc and anomalous gold mineralization. The best intercept occurred in drill hole GE10-003 and consisted of 1.1 m of 7.7 g Au/t including 0.3 m of 26.7 g Au/t.

Five additional drill holes 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 drill holes or between the drill holes 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 162.07 m to 162.82 m, which assayed 1.86 g Au/t. The remaining drill holes 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 drill hole data has been reviewed and additional drilling may be recommended in the future.

7.1.2 RKD Target

Six drill holes totalling 1,587.4 m were completed on the target known as RKD during 2011. The drill holes intersected a north-northwest trending mineralized shear zone dipping steeply to the west. The best gold intercept was in drill hole 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 the drill holes 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.

7.1.3 Silver Spray Target

Two drill holes totalling 806.8 m were completed at Silver Spray. The drill holes intersected strong chloritic alteration throughout both drill holes. Both drill holes 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).

7.1.4 Snowdrop Target

In 2011, 100 m x 100 m 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.

In 2012, the detailed 20 m by 20 m infill soil sampling program was continued. A total of 3,376 soil samples 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 northeast-southwest and flanks a strong magnetic high. There is a strong correlation with arsenic, bismuth, and iron, with zoned copper and zinc 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 drill hole 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 219.1 m. The drill hole intersected zones of intensely silicified greywackes and shales with minor sheeted quartz veins. The alteration and veining are notably similar to those observed at the Batman deposit in the vicinity of the core zone. The greywacke units are coarser grained than at Batman, but the frequency of lithological changes and alteration types are all very similar. Sulfides are present within the quartz veining and as disseminated blebs within intensely silicified siltstones. Common sulfide minerals include pyrite, pyrrhotite, chalcopyrite, and arsenopyrite with traces of galena, sphalerite, and bornite. Veining has a steep dip to the east, similar to Batman, but appears richer in base metals. Disseminated sulfides are also more abundant, while the vein density is not as intense as Batman. Although the drill hole did not intersect significant ore grade mineralization, assay results were encouraging, and additional drilling is warranted. The highest-grade intercept was 0.9 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 arsenic, bismuth, cobalt, and tellurium.

7.1.5 *Sample Preparation Methods and Quality Control 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. Calico sample bags are purchased pre-numbered, and are combined into parcels of five sample bags into 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.

7.1.6 *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 Quality Assurance and Quality Control (QA/QC) of which it provides the data to Vista. At the time of the work conducted all sample preparation and analytical work was performed at North Australia Assay laboratories, in Pine Creek MLN, 792 Eleanor Rd, Pine Creek NT 0847. The laboratory was 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. At the date of issuance of this Technical Report Summary, North Australia Assay Laboratories, in Pine Creek has ceased operations.

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 100 m x 100 m grid is sampled over a broad target, with 20 m x 20 m 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.

This section describes the exploration drilling at the Mt Todd site. The historical drilling section details additional information for the 730 drill holes from various drilling campaigns before Vista from 1988 to 2007.

7.2 Drilling

This section describes the exploration drilling at the Mt Todd site. The historical drilling section details additional information for the 730 drill holes from various drilling campaigns before Vista from 1988 to 2007.

7.2.1 Batman Deposit

Drill hole data for the Project was provided to Tetra Tech in an Access® database format, maintained by site geology staff. The exploration program at the Batman deposit consisted of diamond core drill holes that targeted both infill definitional drilling and step-out drilling. These drill holes are listed by year and company in Table 13.

Date	Reference	Holes (#)	Percussion (m)	Diamond (m)	RVC (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	170	-	-	22,534
1998-2000	General Gold	105	-	7,436	26,365
2007	Vista	25	-	9,883	-
2008	Vista	16	-	8,938	-
2010	Vista	12	-	6,864	-
2011	Vista	15	-	7,063	-
2012	Vista	27	-	17,439	-
2015	Vista	5	-	3,185	-
2020-2022	Vista	26	-	8,887	-
2024	Vista	34	-	6,776	-
	Batman Total	890	8,239	91,670	67,260

Table 13 Batman Deposit Drilling History

Note that a large percentage of the historical drilling was by RVC of less than 100 meters in depth. RVC drilling was used for ore grade control during the mining operations of Pegasus and General Gold. Vista's drilling discovered a larger Batman Mineral Resource by probing deeper with diamond drilling averaging 550 m in depth.

Most of the drilling has been angled to be approximately perpendicular to the mineralization. This orientation more accurately transects the true thickness of the mineralization. Early drilling sampled the deposit near the surface allowing for shorter drill hole depths. Exploring the deeper portions of the deposit has required drill collars to be offset to the east with longer drill hole lengths to reach the mineralized zone. Vista's drilling has targeted the deeper portions of the Batman deposit requiring the drill hole depths. The positioning of the Vista's drill hole collars has been constrained to be outside of the flooded historical mine pit. Most latter drilling has been oriented to transect the higher-grade mineralized zone.

Eight drill holes were completed for metallurgical testing in 2017-2018. These holes were not included in the Mineral Resource estimation, due to a difference in the data collection and type. The data from the metallurgical drilling was used as a verification check of the Mineral Resource model estimation in section view.

Vista drilled additional holes during 2020, 2021, 2022 and 2024. The 2024 drilling program consisted of infill drilling at the Batman deposit, as well as step out drilling to define the area to the north and north east of the Batman deposit, towards the historical Golf-Tollis pits. This area has been named the South Cross Lode zone. The 2020-2022 and 2024 holes are listed in Table 14 below and were included in the 2025 Mineral Resource update.

Drillhole ID	Northing m (MGA94 z53)	Easting m (MGA94 z53)	Elevation (masl)	Bearing (°)	Dip (°)	Total Depth (m)	Drillhole Type
VB20-001	8435654	187603	148	268	-58	326.82	Diamond
VB20-002	8435933	187288	143	268	-58	280.36	Diamond
VB20-003	8435933	187272	140	268	-55	299.82	Diamond
VB20-004	8435933	187251	144	270	-50	148.04	Diamond
VB20-005	8435896	187263	151	270	-61	197.86	Diamond
VB21-001	8435899	187290	152	270	-61	234.45	Diamond
VB21-002	8436402	187662	164	270	-50	458.6	Diamond
VB21-003	8435849	187322	158.8	272	-62	285.68	Diamond
VB21-004	8436407	187942	148	88	-50	410.8	Diamond
VB21-005	8436404	187586	154	270	-50	445.68	Diamond
VB21-006	8435852	187629	132	93	-50	347.67	Diamond
VB21-007	8436518	187618	148	273	-50	299.85	Diamond
VB21-008	8436406	187758	137	273	-50	477.34	Diamond
VB21-009	8436800	188222	143	90	-50	437.5	Diamond

Drillhole ID	Northing m (MGA94 z53)	Easting m (MGA94 z53)	Elevation (masl)	Bearing (°)	Dip (°)	Total Depth (m)	Drillhole Type
VB21-010	8436413	188071	153	86	-50	417.38	Diamond
VB21-011	8436500	187728	148	265	-50	398.78	Diamond
VB21-012	8436405	188435	155	261	-50	901.16	Diamond
VB21-013	8436407	187424	169	86	-53	311.85	Diamond
VB21-014	8436200	187385	164	88	-50	368.75	Diamond
VB21-015	8436200	187352	164	265	-55	341.69	Diamond
VB21-016	8437334	188936	142	121	-68	449.53	Diamond
VB22-001	8436295	187386	179	87	-56	203.76	Diamond
VB22-002	8437377	188671	151	268	-55	116.67	Diamond
VB22-003	8436703	187614	173	85	-56	281.8	Diamond
VB22-004	8436600	187520	181	86	-55	224.71	Diamond
VB22-005	8436499	187467	184	85	-56	220.94	Diamond
VB24-001	8435849	187323	161	268	-55	362.24	Diamond
VB24-002	8435645	187252	170	267	-54	341.27	Diamond
VB24-003	8435597	187273	171	268	-55	368.25	Diamond
VB24-004	8435697	187235	170	268	-55	233.35	Diamond
VB24-005	8435746	187267	168	267	-58	290.5	Diamond
VB24-006	8435844	187263	150	276	-55	211.96	Diamond
VB24-007	8435794	187266	155	267	-55	220.67	Diamond
VB24-008	8435654	187346	155	262	-55	422.4	Diamond
VB24-009	8435653	187194	171	269	-57	210.28	Diamond
VB24-010	8435542	187141	162	274	-55	144.81	Diamond
VB24-011	8435746	187192	179	269	-53	105.35	Diamond
VB24-012	8435697	187535	139	89	-56	122.3	Diamond
VB24-013	8435944	187821	143	267	-55	174	Diamond
VB24-014	8435896	187802	140	267	-56	139.01	Diamond
VB24-015	8435799	187597	140	87	-55	161.3	Diamond
VB24-016	8435860	187756	135	272	-56	173.69	Diamond
VB24-017	8435601	187576	137	271	-55	151.2	Diamond
VB24-018	8435746	187660	137	270	-56	150.22	Diamond
VB24-019	8435549	187516	142	267	-60	141.6	Diamond
VB24-020	8435647	187568	137	267	-60	141	Diamond
VB24-022	8436003	187797	143	269	-60	151.43	Diamond
VB24-021	8435950	187785	142	267	-60	154.66	Diamond
VB24-023	8436053	187818	144	268	-60	155.39	Diamond
VB24-024	8435748	187596	139	89	-59	89.09	Diamond
VB24-025	8435694	187622	137	269	-60	239.36	Diamond

Drillhole ID	Northing m (MGA94 z53)	Easting m (MGA94 z53)	Elevation (masl)	Bearing (°)	Dip (°)	Total Depth (m)	Drillhole Type
VB24-026	8435502	187415	150	271	-60	119.46	Diamond
VB24-027	8435603	187545	138	268	-60	121.07	Diamond
VB24-028	8435747	187628	138	270	-60	125.45	Diamond
VB24-029	8435554	187545	138	264	-61	212.35	Diamond
VB24-030	8435700	187595	137	270	-60	203.1	Diamond
VB24-031	8435855	187717	135	269	-60	230.34	Diamond
VB24-032	8435812	187675	134	271	-60	150	Diamond
VB24-033	8435450	187390	157	270	-60	320	Diamond

Table 14 Drill Holes Added for Mineral Resource Update

Tetra Tech professionals observed drilling and the removal of the core from the rig for hole VB24-033 as shown in Figure 12.

Figure 12 Drilling at the Batman Deposit , Vista November 2024

Figure 13 is a plan map that details the locations of all exploration drill holes drilled in 2020-2022 and 2024, with the collars, listed in Table 14, shown in yellow.

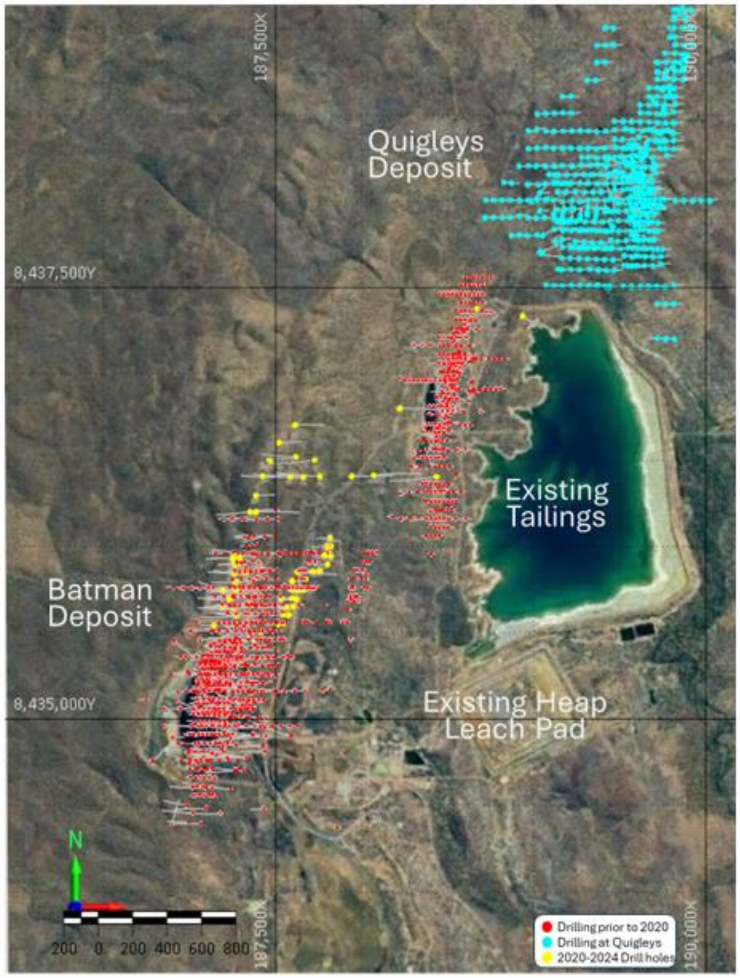


Figure 13 Drillhole Location Map Batman Deposit, Yellow Collars were added for the 2025 Mineral Resource Update, Tetra Tech 2025

Figure 14 shows a map of this historical drilling for the Batman deposit and the following Figure 15, Figure 16, Figure 17, and Figure 18 are the cross sections of this plan view map.

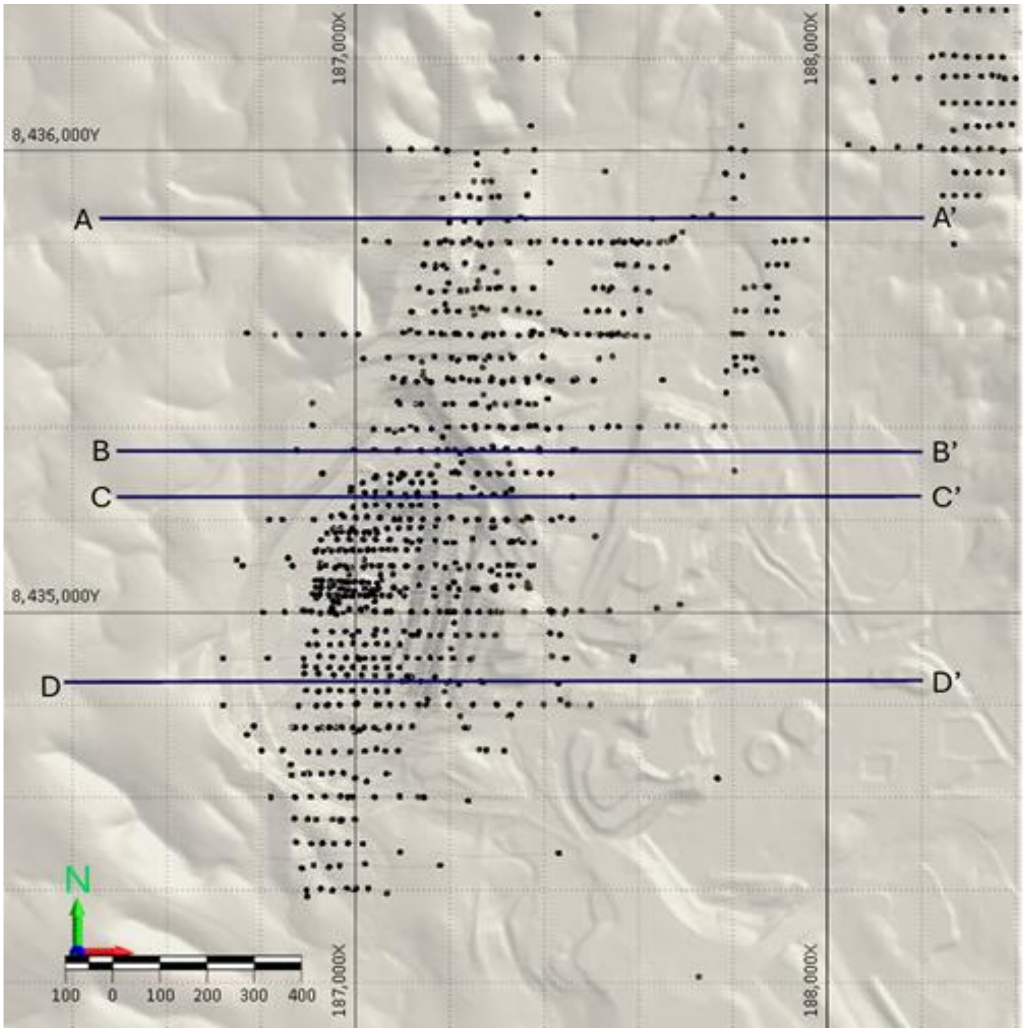


Figure 14 Plan View of the Batman Deposit, with Cross Section Lines, Tetra Tech 2025

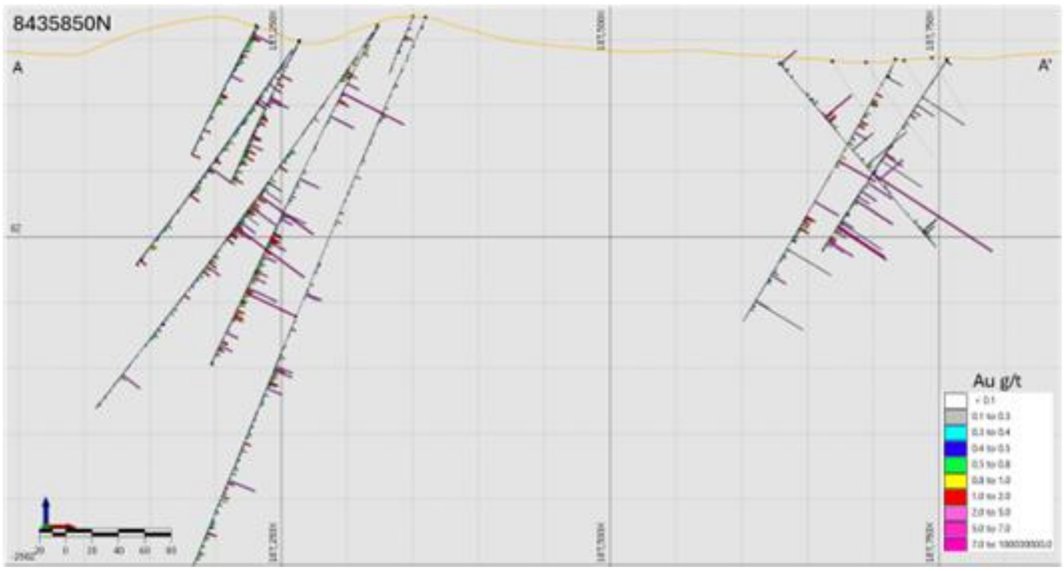


Figure 15 Cross Section A-A' of the Batman deposit Drilling Looking North, Tetra Tech 2025

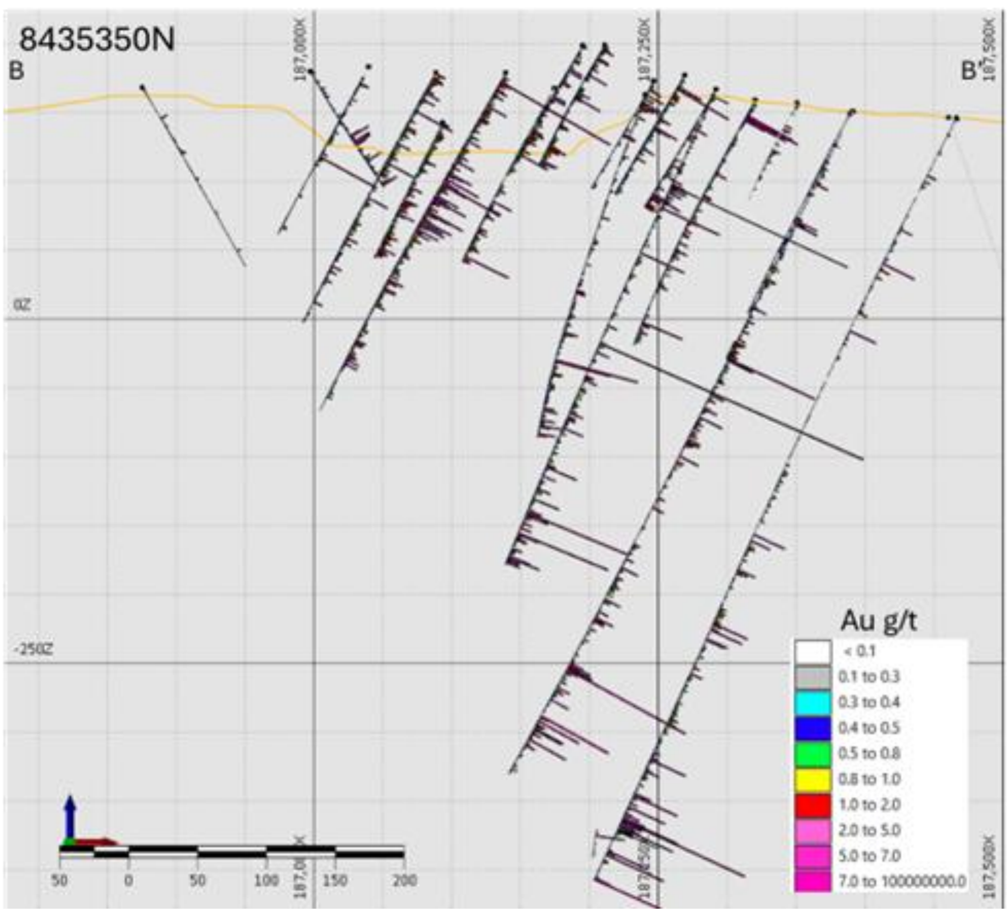


Figure 16 Cross Section B-B' of the Batman deposit Drilling Looking North, Tetra Tech 2025

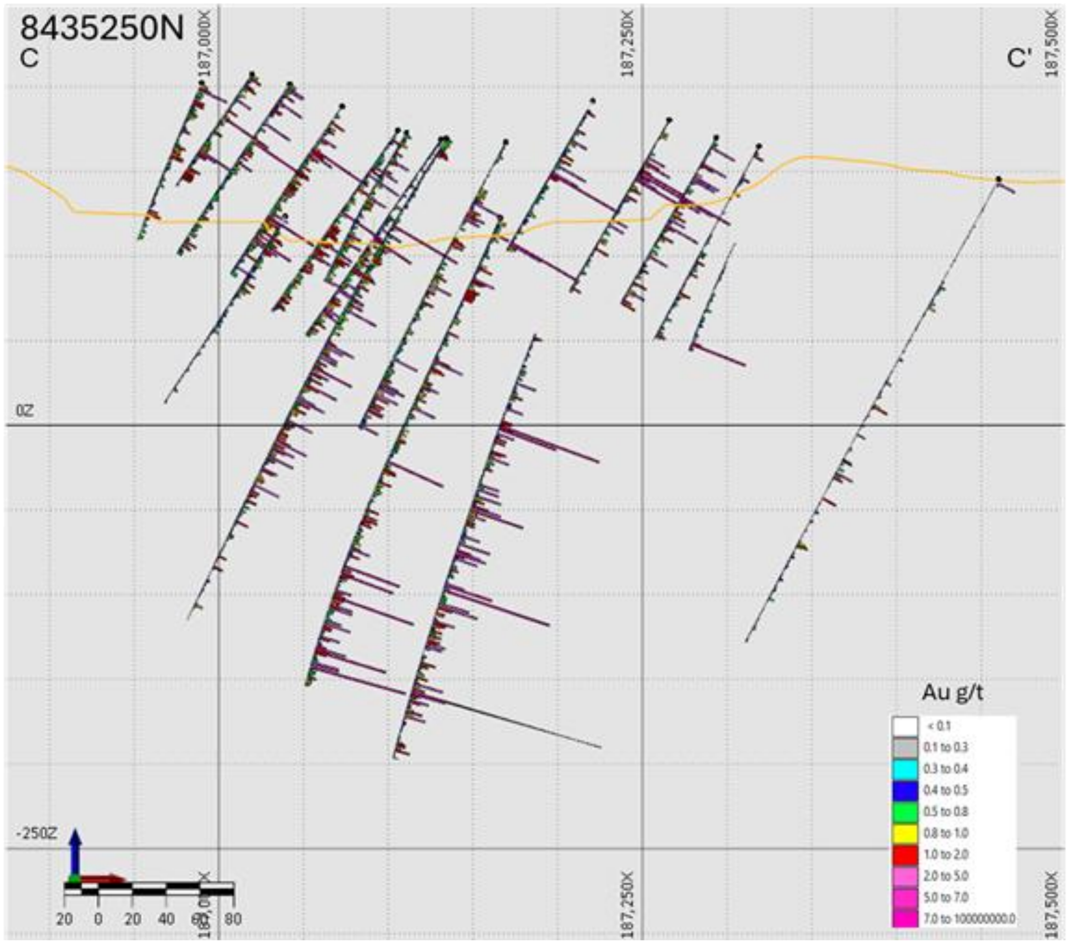


Figure 17 Cross Section C-C' of the Batman deposit Drilling Looking North, Tetra Tech 2025

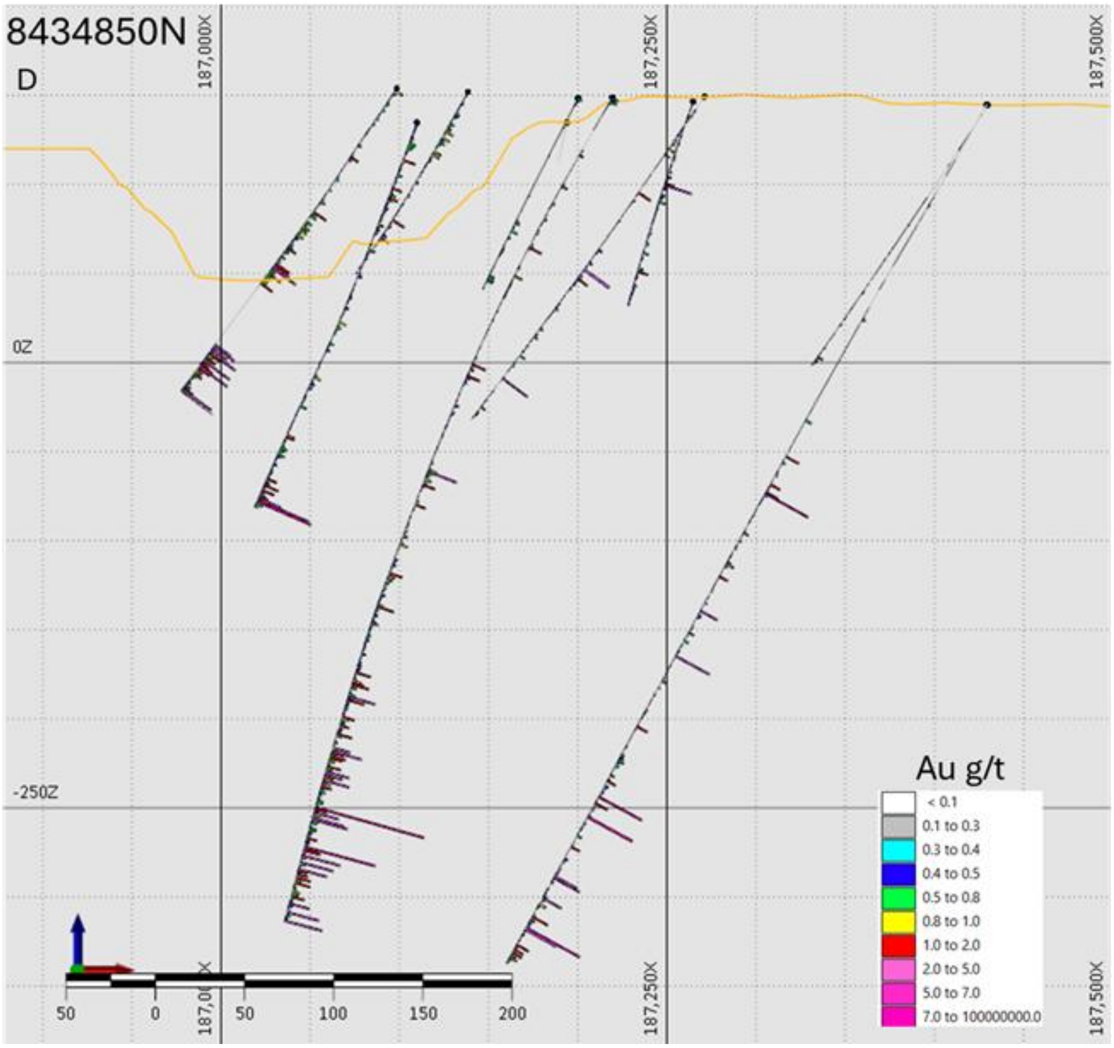


Figure 18 Cross Section D-D' of the Batman deposit Drilling Looking North, Tetra Tech 2025

7.2.2 Quigleys Deposit

The Quigleys deposit drilling history is shown in Table 15 and Table 16. Quigleys was mined from 1982 to 1987 during which the largest amount of drilling was percussion type used for ore grade control.

Relevant intervals of mineralization are contained within blanket-like zones which are modeled with 3-D wireframes for Mineral 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 drill holes, each over 350 meters in depth.

Date	Reference	Holes (#)	Percussion (m)	Diamond (m)	RVC (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	-
	Quigleys Total	**631**	**41,930**	**11,878**	**4,013**

Table 15 Quigleys Deposit Drilling History

Table 16 details the Quigleys exploration database as of the time of this report.

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

Table 16 Summary of Quigleys Exploration Database

Figure 19 shows a plan view of the Quigleys deposit with the drilling cross sections shown in Figure 20 and Figure 21.

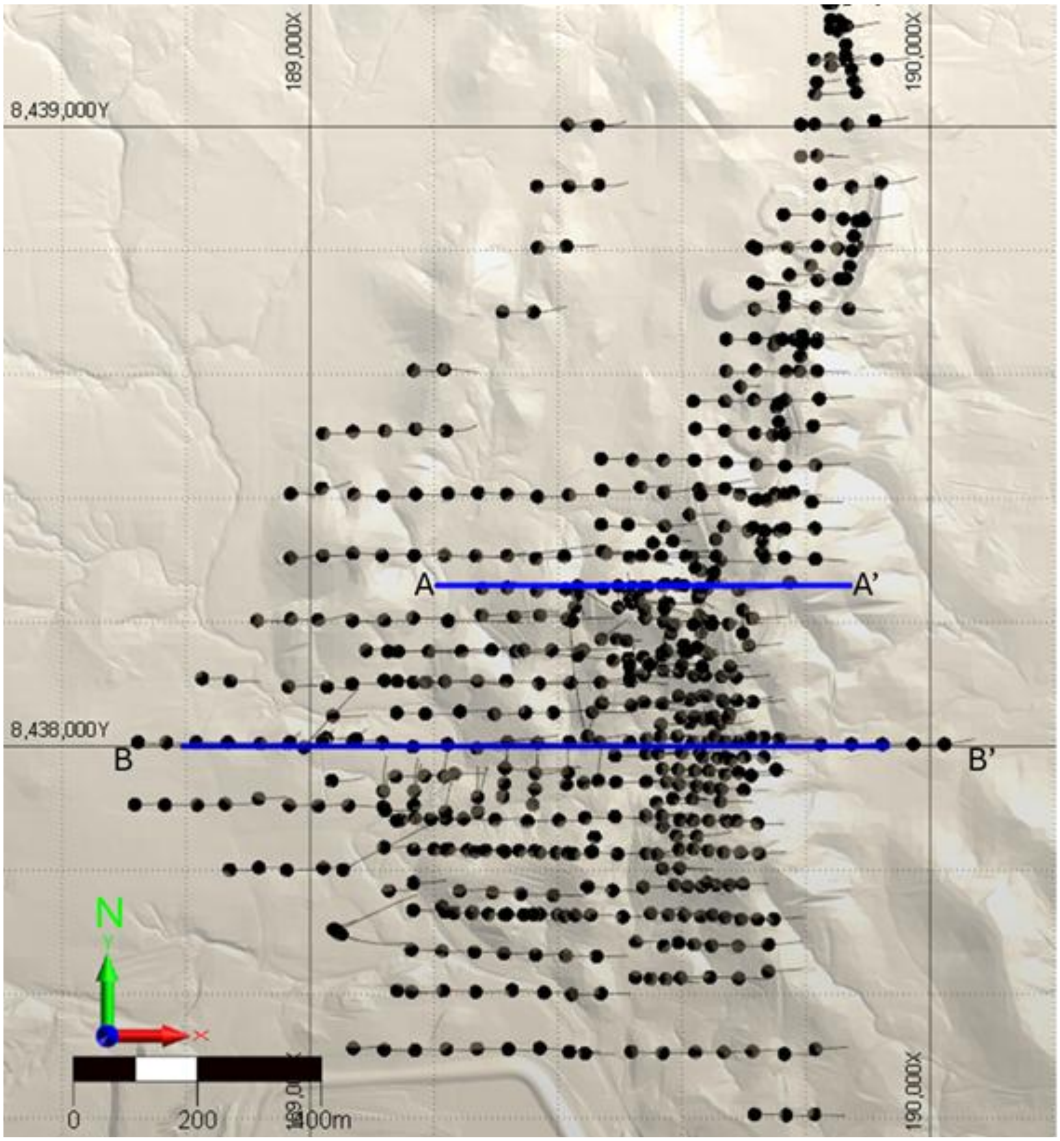


Figure 19 Plan View of the Quigleys Deposit with Section Lines, Looking North, Tetra Tech 2025

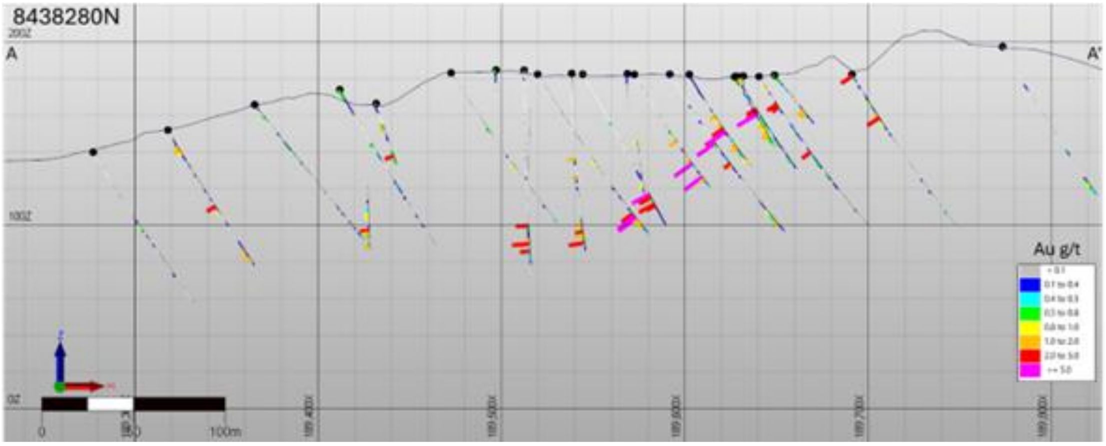


Figure 20 Cross Section A-A' of the Quigleys Deposit Drilling Looking North, Tetra Tech 2025

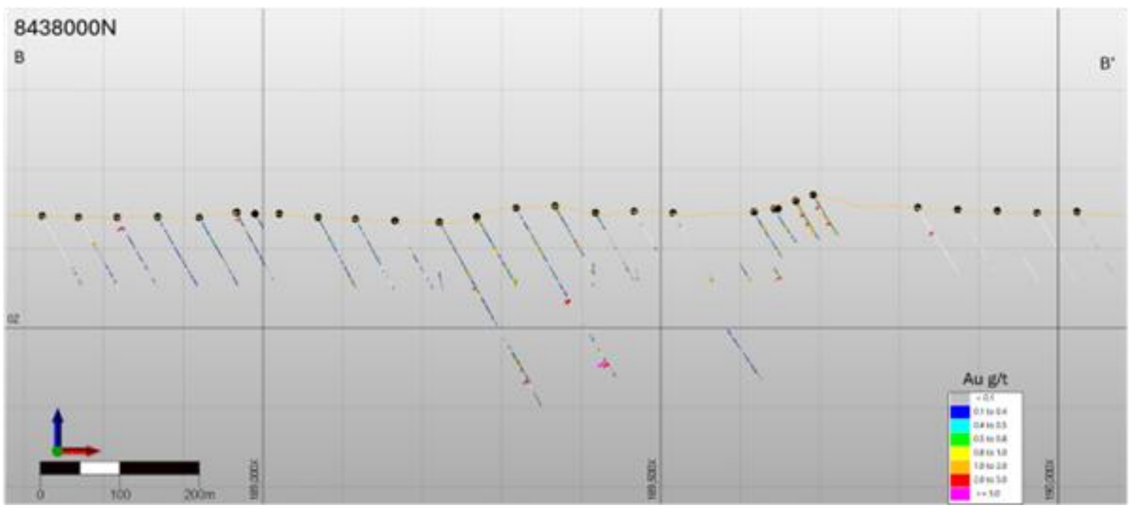


Figure 21 Cross Section B-B' of the Quigleys Deposit Drilling Looking North, Tetra Tech 2025

7.2.3 Drilling Procedures

The drilling procedures followed by the various companies were reviewed 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 drill holes used 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 drill hole at depth is monitored by recording changes in azimuth and dip at depth.
5. Rock material obtained by RC or percussion drilling 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 drill hole. The results along with photographs of the core after this logging exercise are entered into the drill hole database.
7. The core is split with one half sent to an outside laboratory to be assayed (see Section 11 for a generalized description of the laboratory protocols) and the remaining core placed in a secured repository.
8. Assay results are entered into the drill hole database. All the remaining material from the samples prepared by laboratory, referred to as "pulp", are 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.

7.2.4 Sampling

The sampling method and approach for all drill holes 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 core logging 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.

7.2.5 Summary and Interpretation of Relevant Results

Refer to Section 5 of this Technical Report Summary for information on historical drilling sampling. Neither Vista nor the QP are aware of any drilling, sampling, or assaying issues that would materially impact the accuracy or the results presented in this Technical Report Summary.

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.

7.2.6 Qualified Person Opinion

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.

8. SAMPLE PREPARATION, ANALYSIS AND SECURITY

The following section describes the sample preparation, analyses and security undertaken by Vista for the Mt Todd Project.

8.1 Sample Preparation

The diamond drilling program is conducted under the supervision of the geology staff composed of a chief geologist, several experienced geologists, and a core handling/cutting crew.

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

The diamond drill core is boxed and stacked at the rig by the drill crews. Core transported directly into the logging shed by Vista staff.

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

- One meter depth intervals are marked out on the core by a member of the geology staff.
- Core orientation (bottom of core) is marked with a solid line when at least three orientation marks aligned and used for structural measurements. When orientation marks are insufficient, an estimation orientation was Indicated by a dashed line.
- Geologic logging is then done by a member of the Vista geology staff in the core logging facility shown in Figure 22. Assay intervals are selected at that time and a cut line is marked on the core. The standard sample interval is one meter, with a minimum of 0.2 m and a maximum of 1.2 m. The sample is selected to be in the middle of the identified vein, when possible.

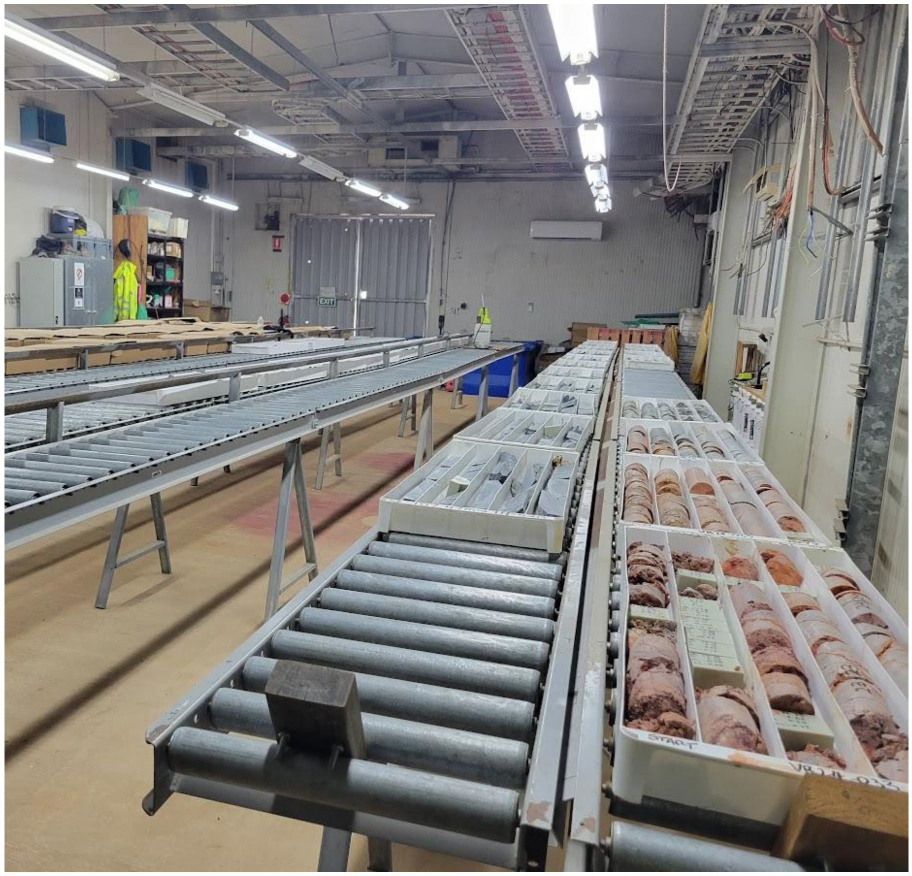


Figure 22 Core Logging Station, Vista November 2024

- Blind sample numbers are then assigned based on pre-labeled sample bags. Sample intervals are then indicated in the core tray at the appropriate locations.
- Each core tray is photographed at a photography station, shown in Figure 23, with a mounted camera, light and water sprayer, and then transported to the cutting area via a roller conveyor.

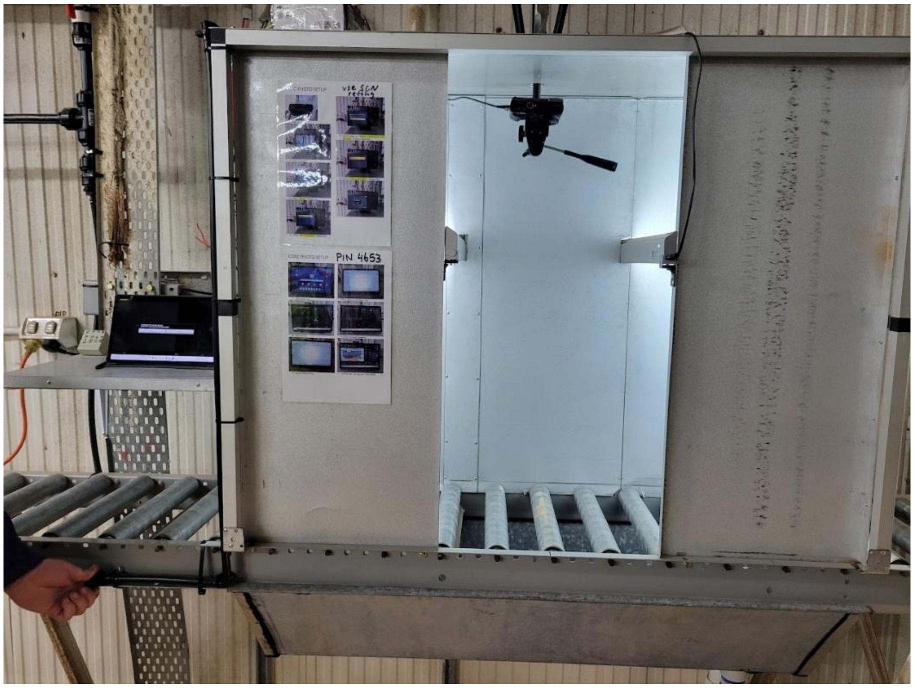


Figure 23 Core Photography Station, Vista November 2024

- The core is then cut in half using diamond saws, Figure 24, with each interval half-core placed in sample bags with a sample tag.

Figure 24 Saw Station for Core Cutting, Vista November 2024

- The standards and blanks are also placed in the plastic bags for inclusion in the shipment to the laboratory, with a reference standard or a blank inserted at a minimum ratio of 1 in 10 and at suspected high grade intervals additional blanks sample are added.
- Standard reference material is sourced from Ore Research & Exploration Pty Ltd and provided in 60 g sealed packets. The standards are selected from the six commercially purchased standards at site, which are stored under refrigeration.
- When a sequence of five samples is completed, they are placed in a shipping bag and closed with a zip tie. All samples are kept in the secure area until crated for shipping.
- Samples are placed in crates for shipping with 100 samples per crate (20 shipping bags). The crates are stacked outside the core shed until transport.
- Samples are delivered to the laboratory by Vista staff.

8.2 Sample Analyses

The following laboratories have been used for lab preparation, analyses, and check assays (Table 17).

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	ISO/IEC 17025 Certified
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 Certified

Table 17 Assay and Preparation Laboratories

Prior to the 2011 drilling campaign, the majority of samples were transported first to ALS in Alice Springs (NT) for sample preparation. After preparation, samples were then forwarded on to ALS in Malaga, Western Australia (WA) for assay analyses. One in every 20 pulp or reject was sent from ALS in Alice Springs to Northern Australian Laboratories (NAL), Vista was notified by email which samples were sent to NAL.

For the 2011-2025 drilling campaign samples for assay were sent to NAL lab in Pine Creek, NT. At the end of each drilling campaign, Vista also submitted samples to Intertek in Darwin and Perth for confirmation of the NAL values. This umpire sampling is standard practice and meets the requirements for QA/QC on a Project at this level.

Professionals from Tetra Tech, along with Vista personnel, visited the NAL laboratory on November 7, 2024. The laboratory was processing Vista samples during the visit. Tetra Tech observed the areas for drying, crushing, pulverizing, and storage of samples, as well as the ICP and fire assay areas.

The crushing equipment is cleaned between samples, and the laboratory processes one job at a time, to ensure that there is no cross contamination. LIMS software is used to track the samples and the results, as well as tracking the QA/QC standards for the facility. Certified standard material from Ore Research & Exploration Pty Ltd (OREAS) is utilized for QA/QC purposes for the laboratory.

The procedure defined by the Vista geologic staff is to duplicate the fire assay of any sample that has an assay value greater than 10 ppm. Any discrepancies in the testing initiates a re-assay of the samples, before reporting data to Vista. Results are transmitted to the Vista geologist by email, including an un-editable PDF file, and an Excel file. Overall, the laboratory meets the requirements for testing and is appropriate for samples of this type. Vista is completely independent of any analytical testing entity presented in this Technical Report Summary.

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. ISO/IEC 17025 certification ensures that laboratories meet international standards for technical competence and quality management in testing and calibration procedure. All laboratories herein operate independently of Vista. The standards specify a recipe and set of quality control steps that the laboratory should follow:

- How the sample should be coded to obscure its relationship to the drilling geometry.
- How the received sample should be prepared.
- The analytical steps be taken.
- Given the required detection level and material analyzed, what instruments should be employed.
- 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.

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

- Core samples from drill holes are split into two with one half archived and the other sent to analytical laboratory.

- At the lab the sample is pulverized into a powder, with a subsample taken for fire assay.
- 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.
- 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.
- 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.
- Due to the relatively low concentration of gold at Mt Todd Gold Project, the lab must choose an analytical method able to detect at least 5 ppb gold. The methods are generally by atomic absorption (AA) or inductively coupled plasma-mass spectrometry (ICP-MS).
- The bead is dissolved in aqua regia or dissolved in hydrochloric acid and then analyzed by the selected instrument.
- 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.
- Vista is completely independent of any analytical testing entity presented in this Technical Report Summary. The QP has determined that there is no apparent conflict of interest between Vista and its analytical laboratories.

The results of the quality control procedures employed by Vista have been reviewed multiple times historically, and for the 2024 drilling program. All analytical procedures employed are consistent with conventional industry practice, and it has been determined that they meet industry standards for use in Mineral Resource estimation.

8.3 Sample Security

NAL was the primary laboratory for the 2020-2022 and 2024 drilling program. The NAL laboratory is located in the town of Pine Creek, approximately 100 km distant by road from Project site. A sample transmittal form is prepared by Vista geologist, chain-of-custody documentation accompanies each shipment and included with each shipment and a copy is filed in the geologist office on site.

When the shipment leaves the site transported by Vista employee in a company owned vehicle, sample transmittals are prepared and e-mailed to NAL. When the shipment arrives at the preparation facility the samples are lined out and a confirmation of sample receipt is e-mailed back to Vista. Following completion of assay results, all pulps and reject material is shipped back to the Project site and stored.

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. Statistical analysis of the various drilling populations and QA/QC samples has not either identified or highlighted any reasons to not accept the data as representative of the tenor and grade.

8.4 Qualified Person Opinion

The QP is satisfied with the adequacy of sample preparation, security and analytical procedures employed by Vista, given the fact that Vista has completed more than 60,000 m of core drilling in the Batman deposit, to verify the approximately 98,000 m of historical drilling and increase the Mineral 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 and Quigleys deposits.

9. DATA VERIFICATION

9.1 Drill Core and Geologic Logs

A site visit was performed by Tetra Tech professionals, including the QP for the Geology and Mineral Resource estimation portion of this Technical Report Summary. During the visit, Tetra Tech found a comprehensive drill hole 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 Mineral 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 the Mineral Resource estimation.

The QP responsible for this section found no significant discrepancies with the existing drill hole 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.

9.2 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 the Mineral Resource estimation and as the Project goes forward GDA94/MGA zone 53 will be the used coordinate system.

Tetra Tech and Vista staff surveyed multiple drill hole collar coordinates during the site visit. The spot checks agreed with the information in the database provided. It is the opinion of the QP for this section that the current topographic map is accurate and accurately represents the topography of the Project area. In addition, it is suitable for the development of the geologic models, Mineral Resource estimates, and Mineral Reserve estimates.

9.3 Verification of Analytical Verification

This Section describes verification procedures from 2007 to 2024.

9.3.1 2007 Verification

As part of the 2007 exploration program, an exercise to both verify the historical assay results and ensure that future analytical work meets current NI 43-101 standards for reporting of Mineral Resources was completed. This program consisted of two components: re-assaying of a portion of the historical drill holes, and assaying of the new core drill holes.

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

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

Figure 25, Figure 26, and Figure 27 detail the results of the analytical check program that was completed on the 2007 exploration drill holes. The program was designed to check both internal laboratory accuracy and inter-laboratory accuracy. NAL was the primary laboratory for completion of the sample analyses. ALS-Chemex in Sydney, Australia performed the inter-laboratory analyses. As can be seen from the plots, the correlation coefficient was 0.997 for the re-splits of original assays, 0.992 for pulp repeats, and 0.986 for inter-laboratory analyses, respectively.

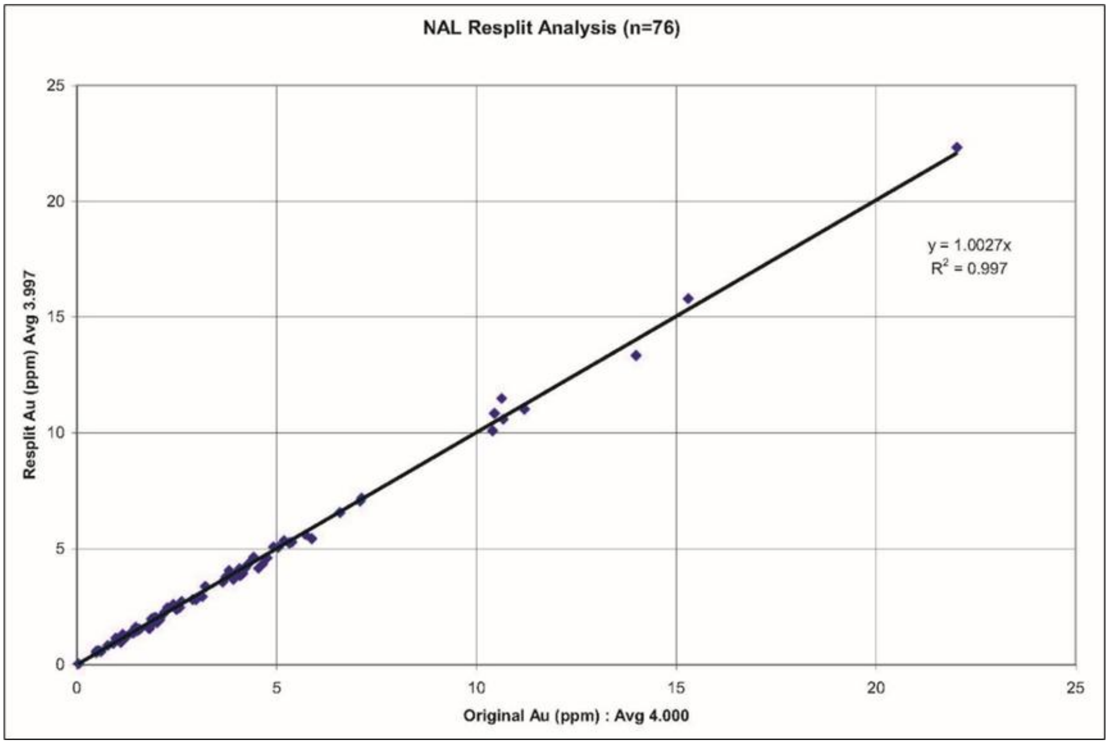


Figure 25 NAL Resplit Analyzes, Tetra Tech 2019

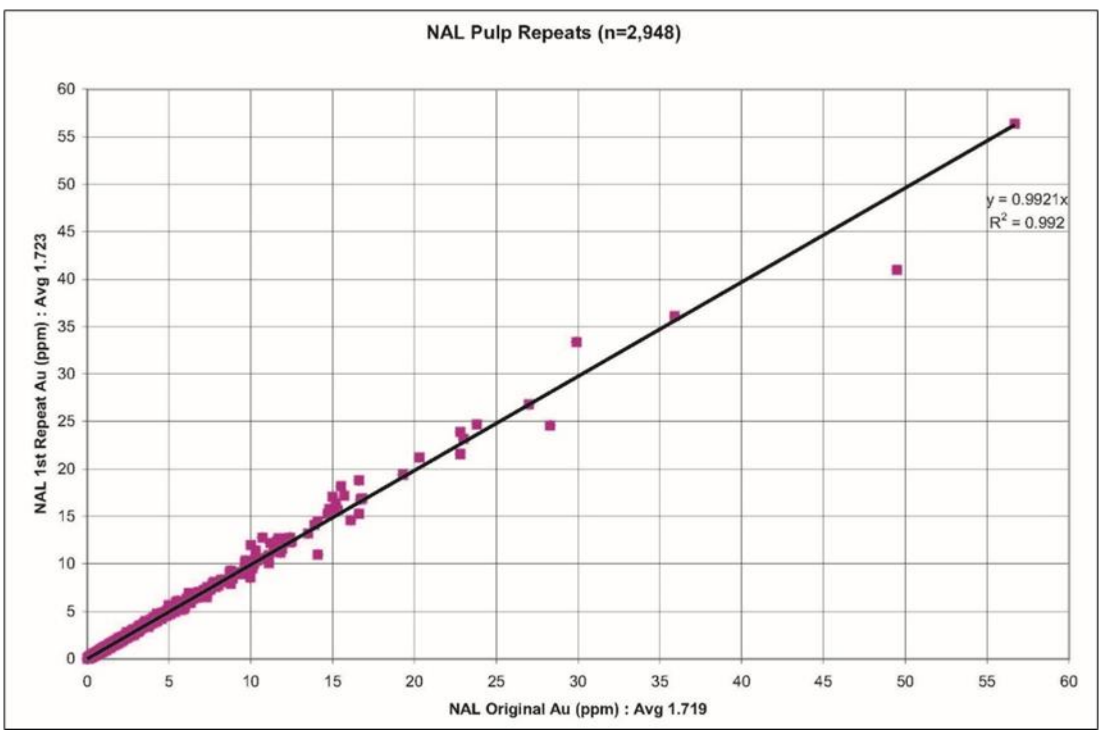


Figure 26 NAL Pulp Repeats, Tetra Tech 2019

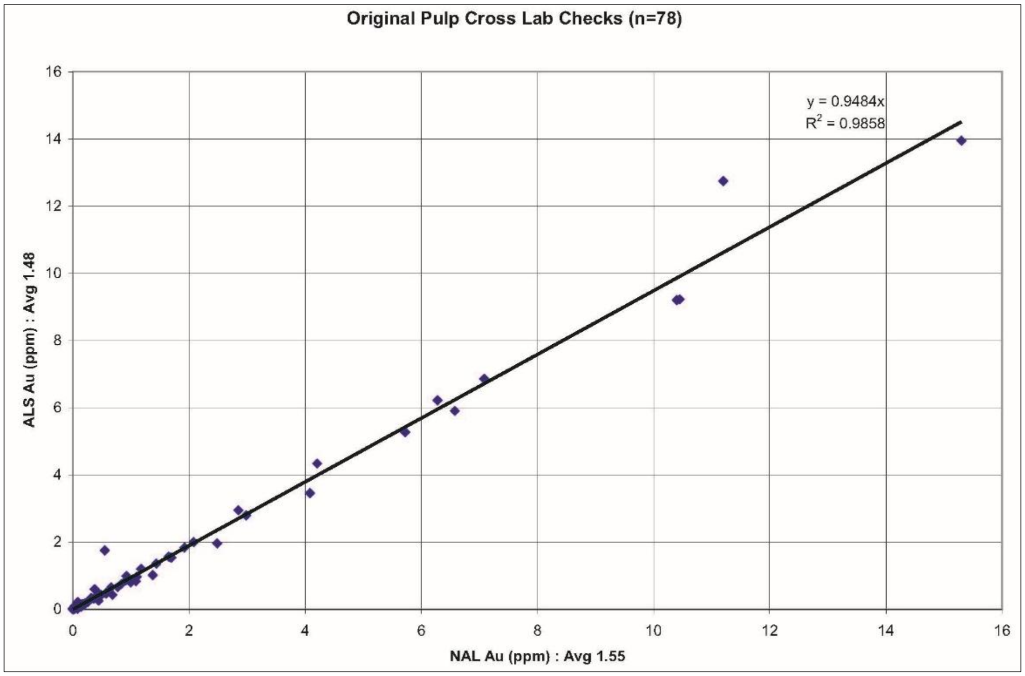


Figure 27 Original Pulp Cross Lab Checks, Tetra Tech 2019

9.3.2 2011 Verification

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 work. For context, Mt Todd assay table as of August of 2011 contained 118,550 records, 26,579 of them originating from Vista 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 18 and Table 19 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 18. Of the 20 differences noted, 4 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 drill hole VB08-036. This audit shows that discrepancies within the database on the global Mineral Resource estimate are not material.

Based on this review, the QP has determined that the historical and Vista assays in the Mt Todd database are useable for Mineral Resource modelling.

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 18 Summary of Comparisons of Historical 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

Table 19 Summary of Comparisons of Vista Assays (MDA, 2011)

9.3.3 2018 Verification

For the March 2018 Mineral Resource estimates, a detailed data verification procedure was undertaken by Tetra Tech which focusing on the 2012-2017 exploration programs. This verification was accomplished by reviewing the assay database received from Vista and comparing results with laboratory certificates received directly from the laboratory and reviewing results of the field QA/QC samples.

For the 13 drill holes from the 2012 exploration program, there were 7,601 intervals assayed. For the nine drill holes from the combined 2015-2017 exploration programs, there were 1,770 intervals. In addition to Au and other precious metals, most intervals had multi-element and environmental test results as well. Similar to previous work, the assay interval averaged one meter with a minimum interval of 0.4 m and a maximum interval of 1.4 m. No errors were noted in the assay data received other than selenium results for one drill hole 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 drill hole 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 28. 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 mislabelled sample and when plotted with the assumed correct standard fell within the 10% range. Figure 28 demonstrates the variance is greatest at lower Au concentrations and this is normally seen with most Au analytical data.

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

Because this drilling campaign produced core, a regular program of field duplicates was 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 acceptable for the drill holes. Equally good correlation is seen for the second replicates against the original and against the first replicate value.

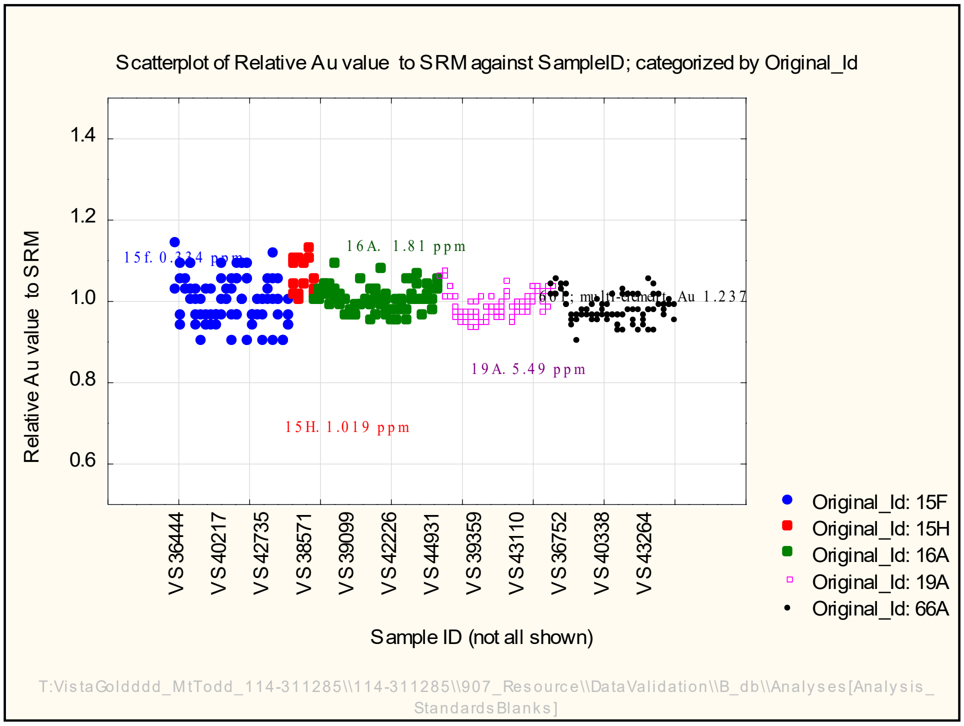


Figure 28 Scatterplot of Relative Au Value to Certified Standard Reference Material Value, Tetra Tech 2017

9.3.4 2020-2022 Verification

Every sample interval from the VB20 drilling program was assayed at North Australian Laboratories (NAL) for a total of 9349 samples with an additional 520 blanks and 524 certified reference materials (CRM). Each sample was assayed for gold using fire assay (50 g), and if it contained gold, it was repeated up to 5 times. A selected suite of multi-elements was also assayed using 4-acid digestion, 12 elements using ICP-OES and another 12 elements using ICP-MS.

Five hundred forty pulps were sent to Intertek Darwin, Northern Territory Environmental Laboratories (NTEL) for 4-acid ICPOES/ICPMS analysis, then forwarded onto Intertek Genalysis Perth for 50 g fire assay Au analysis.

9.3.5 2024 Verification

Tetra Tech professionals, including the Geology and Mineral Resource estimation QP, visited the NAL facility in Pine Creek November 7, 2024. The laboratory was processing samples from the Vista 2024 drill program during the visit. Tetra Tech observed the areas for drying, crushing, pulverizing, and storage of samples, as well as the ICP and fire assay areas.

As part of the QAQC requirement for following the standards in accordance with the requirements and guidelines of subpart 229.1300 of the Regulation S-K 1300.

Every sample interval from the VB24 drilling program was assayed at NAL for 7,910 samples, which include 391 blanks and 409 certified reference materials (CRM). Each sample was assayed for gold using a fire assay (50 g), and if it contained gold, the assay was repeated. A selected multi- element suite was also assayed using a 4-acid digestion method, with 12 elements analyzed using ICP-OES and another 12 elements analyzed using ICP-MS.

An assay certificate is electronically sent to the Mt Todd Gold Project staff, and the assay results are entered into the drilling database. After Vista's senior geologist checks the assays, the pulps and coarse residue material are shipped back to Mt Todd and stored.

Finally, 244 pulps, 243 coarse residues, and 28 certified reference materials (CRM) were sent to an independent laboratory, Intertek Genalysis Perth (INT), for fire assay (50g) gold and a 33-element 4-acid ICP-MS analysis.

9.3.5.1 Quality Control Samples

Certified reference materials and blanks are inserted into the sample sequence at a minimum ratio of 1 in 20 for each type. At suspected high-grade intervals, additional blank samples are added. All the certified reference materials are sourced from Ore Research & Exploration Pty Ltd (OREAS) and have gold ranges from 0.542 to 9.834 ppm. The blank material is sourced from Rowland Quarries in Katherine, NT. A gravel blank is used to monitor cross-contamination from the crushing and pulverizing stages during sample preparation. Before each batch of gravel is used as a control blank, three samples are assayed to determine whether they contain gold.

After each batch of samples has been reported, Vista site geologists check every control sample for errors or trends in variability. If an error is found, the laboratory (NAL) is contacted to investigate the problem. The assays are only entered into the final Technical Report Summary and the drilling database after all errors have been explained and verified by Vista senior geologist.

9.3.5.1.1 Certified Reference Materials (CRM)

A total of 409 CRM's were used in the 2024 (VB24) drilling program. Overall, the assays compared well with the CRM's, as shown in Figure 29. On average, NAL CRM gold assays are lower than the CRM-certified values. Multi-element assays of the CRM material are also compared to ensure that any error reported was not caused by mislabeling prior to laboratory submission.

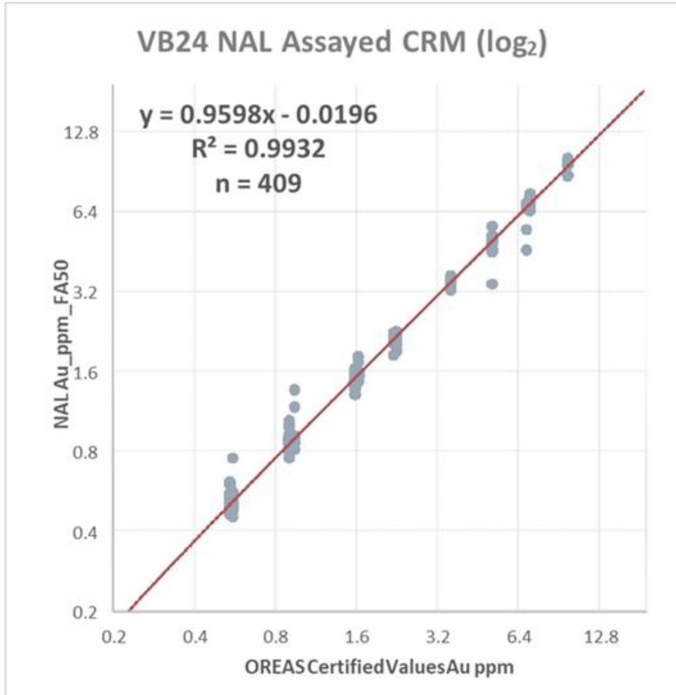


Figure 29 NAL Assayed CRM versus OREAS Certified Values Au ppm. Tetra Tech 2019

9.3.5.1.2 Blanks

Three hundred ninety-one blanks, shown in Table 20, were used in the 2024 drilling program, and forty-nine samples (12.5%) were assayed above the detection limit of 0.1 ppm Au, which was not accepted. After working with the laboratory (NAL), the problem was identified to be the low-level calibration drift of the AAS instrument and was not caused by contamination, as reported in McIlwain (2024). After identifying the AAS drift problem, no further blank assays were reported over 0.01 ppm Au.

Au ppm	G2021	G2024
-0.01	9	239
0.01	6	26
0.02	8	12
0.03	13	6
0.04	1	3
0.05	3	0
0.06	3	2
Total	43	348

Table 20 Blanks Data Summary

9.3.5.2 *Independent laboratory checks*

At a ratio of one in forty, 244 pulps, 243 coarse residues, and 28 certified reference materials (CRM) were sent to an independent laboratory, Intertek Genalysis Perth (INT), for fire assay (50 g) gold and a 33-element 4-acid ICP-MS analysis (Table 21). A full range of gold concentrations was selected to ensure that low values or values near the detection limit were not the dominant sample type. Extra blank pulps and residues were also chosen to ensure that there was no gold contamination.

Range of Gold ppm	Number
Blanks <0.02	15
<0.04	22
0.04 to 0.08	38
0.08 to 1.3	43
1.3 to 5.0	95
5.0 to 30.0	31
Total	244

Table 21 Number of Samples Selected per Gold Grade

9.3.5.2.1 Gold Checks

Gold assays from NAL are plotted against INT for both pulp checks and coarse residue checks in Figure 30 and Figure 31. Both plots show that, on average, INT reported slightly higher gold grades. The INT Pulps vs INT Residues plot is shown in Figure 32. The results show that the pulps reported on average higher grades than the coarse residues, due to the difference in pulverizers.

Overall, the gold assays match very well with the independent laboratory checks and are acceptable for use in the Mineral Resource estimation at the Project.

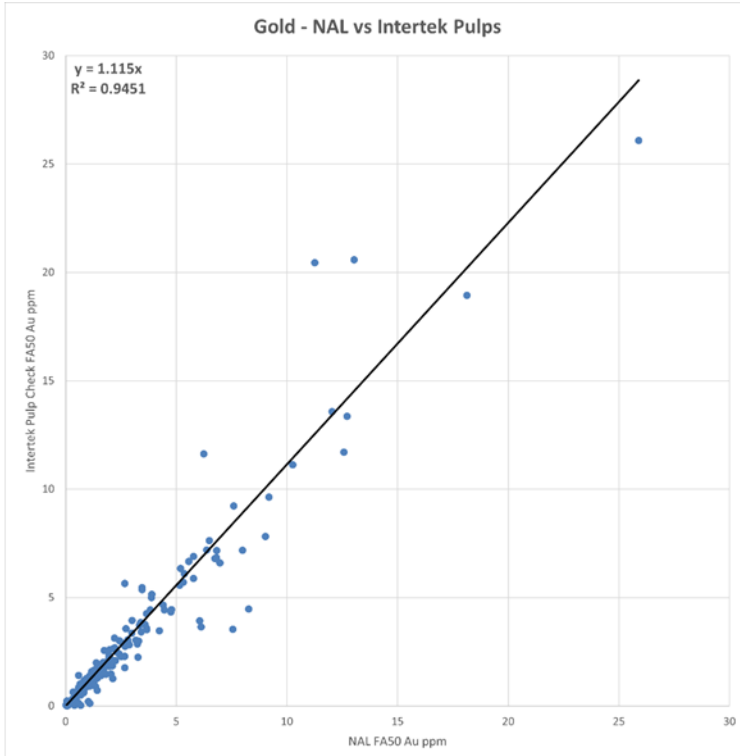


Figure 30 Gold – NAL vs Intertek Pulps, QA/QC 2024

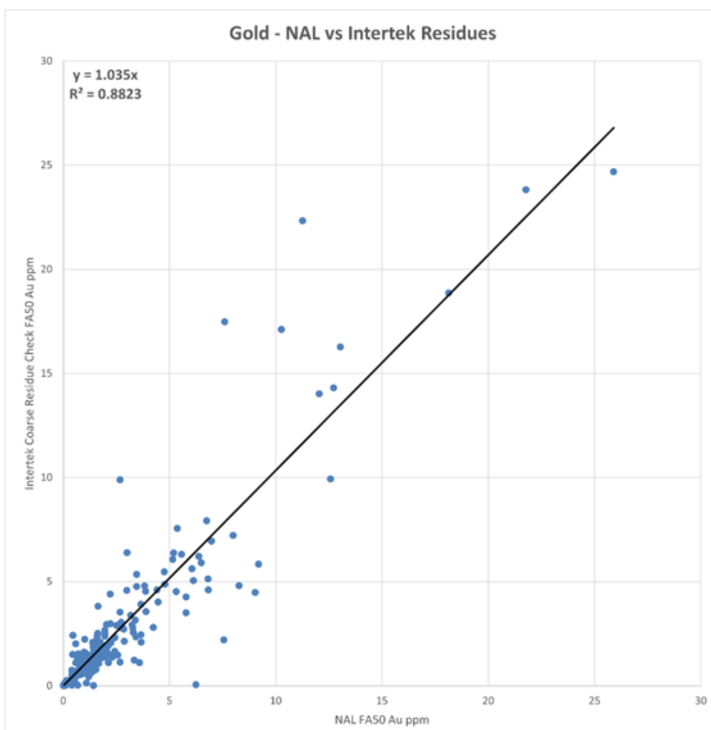


Figure 31 Gold – NAL vs Intertek Residues QA/QC 2024

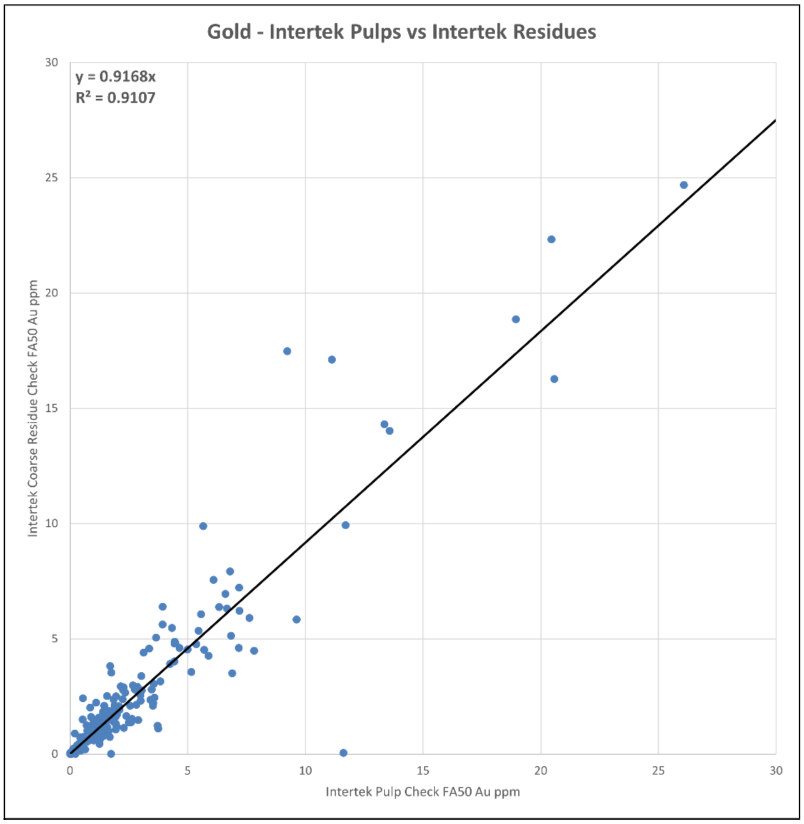


Figure 32 Gold – Intertek Pulps vs Intertek Residues QA/QC 2024

9.3.5.2.2 Multi-element Checks

The NAL versus INT verification also included copper, silver, sulfur, and bismuth. These values showed an overall good correlation between the laboratories. Scatter plots of these elements are shown in Figure 33.

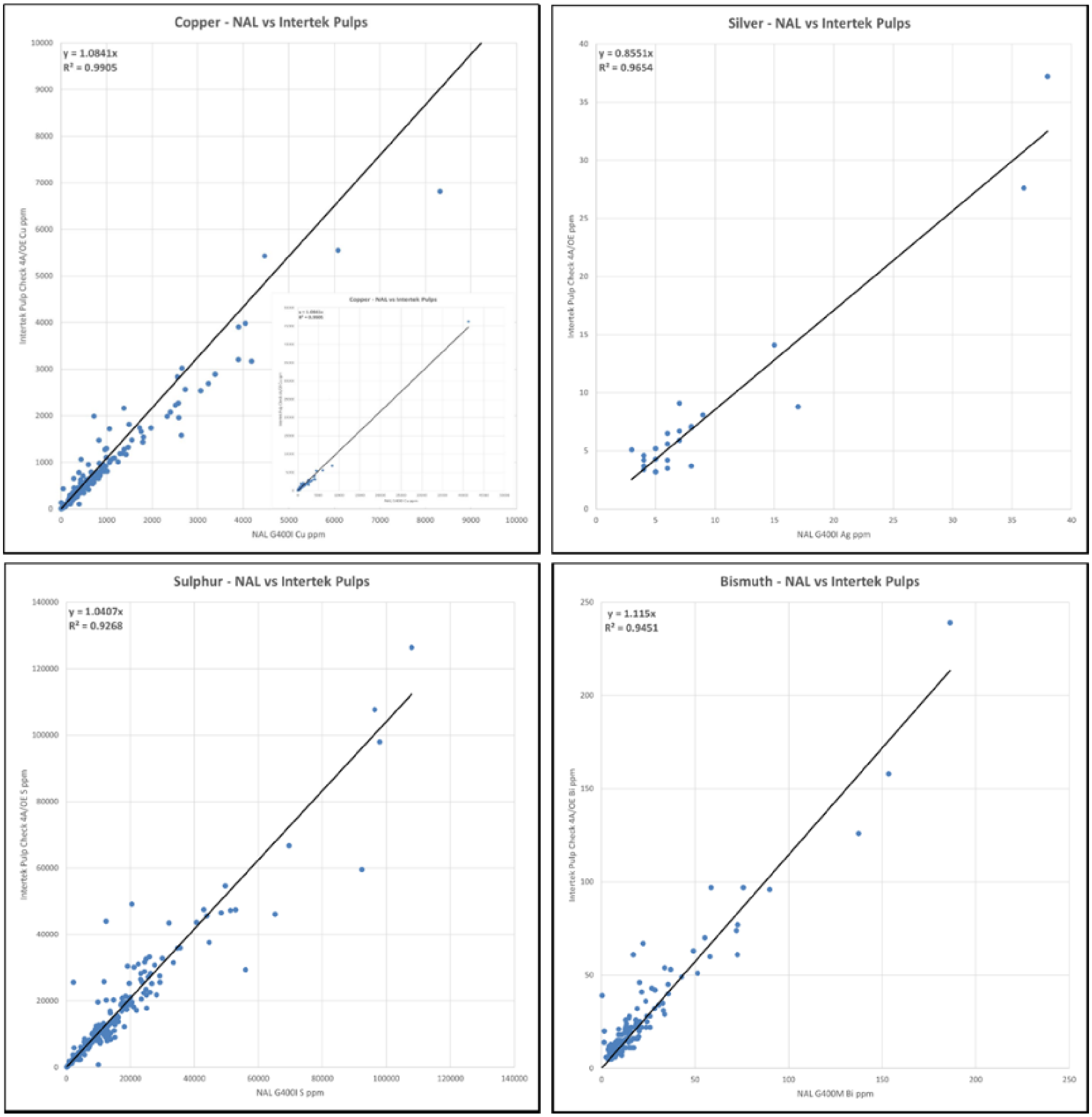


Figure 33 Multi-element comparison of Intertek vs NAL

9.3.5.2.3 Molybdenum Contamination from NAL's pulverizer mills.

Molybdenum is not of economic interest for the Project and it has never been modeled as part of the Mineral Resources estimation for the Project. However, as part of the rigorous review including multielement check, it was found that the returned interlaboratory assays from INT, the pulp checks closely matched the NAL assays for molybdenum; however, the coarse residues were low or below the detection limit, as shown in Table 22. Gravel blanks and a large proportion of oxide samples did not exhibit significant differences. The fresh rock samples show elevated molybdenum in the NAL/INT Pulps, but the coarse residues are low.

Vista has worked with NAL to determine the source of potential molybdenum contamination is the pulverizer mills at NAL, which were produced with up to 3% molybdenum. The high hardness of fresh rock at Mt. Todd caused abrasion to the pulverizer plates, which lead to contamination in pulverized samples. INT uses low chrome steel ring pulverizer mills, which do not contain molybdenum.

Previously, this was not identified due to the multi-elements that were not analyzed in the coarse residues. Examining the past molybdenum assays, it appears that the contamination is not limited to the 2024 drill program, but rather affects all samples analyzed by NAL. It is recommended that molybdenum not be used and be recorded in the database as unreliable.

Sample ID	Depth (m)	NAL Mo (ppm)	Intertek Mo (ppm)	
			Pulps	Residues
VS72247	Gravel Blank	0.78	-2	-2
VS72387	Gravel Blank	1.21	-2	-2
VS76926	6	1.25	-2	-2
VS74497	11	0.33	-2	-2
VS72183	16	1.8	2	-2
VS74025	25	2.29	2	-2
VS72219	48	4.38	4	-2
VS72293	115	7.48	9	-2
VS72340	154	26.83	30	-2
VS72378	187	26.47	28	-2
VS73518	100	46.76	58	3

Table 22 Selected Range of Samples Showing Molybdenum Contamination.

9.3.5.2.4 QA/QC Conclusion

The QA/QC procedures and checks were reviewed and found to be within industry standards under international guidelines. The program uses standards, blanks, duplicates, and umpire sampling at an independent laboratory. Checks at the independent laboratory correlate at an acceptable rate with the NAL tests, except for molybdenum, which has been investigated and documented.

9.4 Qualified Person Opinion

The QA/QC procedures and checks were reviewed by the QP, and found to be within industry standards under international guidelines. The program uses standards, blanks, duplicates, and umpire sampling at an independent laboratory. Checks at the independent laboratory correlate at an acceptable rate with the NAL tests, except for molybdenum, which has been investigated and documented.

10. MINERAL PROCESSING AND METALLURGICAL TESTING

This section reports on the historical work completed 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. No additional metallurgical test work has been conducted for the current study.

10.1 Summary

Key conclusions drawn from the historical metallurgy studies to date are:

- Mt Todd Gold Project (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 single primary ball mill grinding and secondary grinding utilizing four vertical grinding mills.
- The ore is free milling, is not preg robbing, and is amenable to gold extraction by conventional cyanidation processes. The ore contains pyrrhotite and will require an oxygen plant to assist with leach extraction. A hybrid leach/CIL circuit has been adopted for the process. Pre-leach conditioning tanks have been included in the design along with lead nitrate to assist with oxidation prior to the leach circuit.
- 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 the lack of cyanide recycling from leach residue prior to cyanide destruction.
- A final grind size P_{80} of 40 µm was selected to improve liberation and leach gold extraction. A total of 30-hours leaching and adsorption residence time was determined based on batch leach test work to maximize gold extraction.

10.2 Historical Metallurgical Test Programs

The Batman deposit is a large, low-grade gold deposit. The average grade of the gold mineralization is less than 1.0 g Au/t. The gold mineralization occurs in a hard, uniform greywacke host and is associated with sulfide and silica mineralization, which 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 hard and competent, with Bond Ball Mill Work Index (BWi) measurements in the range of 23 to 30 kWh/t, while JK Technologies Axb values ranging from 19 to 30, with an 85th percentile figure of 26.9 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, from recent sampling of the remaining ore body. The historical metallurgical test work programs completed from 2017 to 2018 and 2018 to 2019 have been mostly used to support the feasibility study and only these data have been discussed in detail in this section. The earlier metallurgical test work and historical production records were however, also considered in the interpretation and process design. The reader is directed to consult the historical test work reports listed in the previous studies for additional information and the full historical description of metallurgical test work.

Observations from the historical test work conducted prior to 2017 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, the 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).
- 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 metallurgical 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, including HPGR technology, 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 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.

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 test work conducted on 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 that 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 Mineral 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 74 μm 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 were conducted on the remaining Batman Mineral 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 test work 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 Summary pit shell.

All samples from drillholes labelled VB11 were drilled in 2011, logged, packaged then shipped directly to the laboratory for processing. Drillholes labelled 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 test work 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 > 15.8 mm 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.

10.3 2017-2018 Metallurgical Test Work

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 five (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 were spatially located to provide variability both horizontally and vertically.

The primary objective of this phase of the test program was to perform sufficient metallurgical test work to confirm the preferred process flowsheet developed during the previous two years and associated reagent consumption rates.

Four composites were defined from the drilling program, as shown in Table 23.

Metallurgical Justification	Composite	Length	Volume	Tonnage	Grade (g Au/t)- Estimated
High Grade	1	291	1.6	4.57	1.54
Above Average Grade	2	342.6	1.9	5.39	0.89
Average Grade	3	308	1.7	4.84	0.74
Low Grade Edge	4	284	1.6	4.46	0.56

Table 23 Composite Drill Hole Makeup

The composites have been selected to represent both near-term and a longer-term mining and were spatially located to provide variability both horizontally and vertically. Estimated grades were determined by assigning the grade from the nearest block of the Mineral Resource model to 2.0 m intervals down each drill hole.

However, it was later found that the drill hole angles used resulted in greater dilution of waste, while the assayed grades are reflected in Table 25. The drill intercept angles are detailed in Table 24.

BHD	Average Intercept Angle (degrees)
VB17-001	10
VB17-003	20
VB17-002	40
VB16-002	40

Table 24 2016/17 Drill Hole Angles

Hole ID	From	To	Length (m)	Block Model – Nearest Hole (g Au/t)	Comp No.	Tonnes (t)	Model Comp Grade (g Au/t)	Assayed Head Grade (g Au/t)	Calc/ Model (%)
VB17-002 VB17-003 VB17-003	372 268 362	485 328 480	113 60 118	1.40 1.50 1.71	1	4.57	1.54	0.70	46
VB16-002 VB17-003 VB17-003 VB17-003 VB17-003	392 142 230 328 480	485.7 230 268 362 568.9	93.7 88 38 34 88.9	1.12 1.14 0.57 0.88 0.92	2	5.39	0.74	0.46	62
VB17-001 VB17-003	0 0	166 142	166 142	0.73 0.75	3	4.84	0.56	0.28	50
VB16-002 VB17-002	238 242	392 372	154 130	0.57 0.55	4	4.46	0.99	0.70	71

Table 25 Adjusted Composite Head Grade

The composite with a calculated head grade deviating most severely from the model grades was composed of holes VB17-001 and VB17-003. These drill holes had the lowest vein intercept angles.

10.3.1 HPGR Testing at Thyssen-Krupp

The four composites were sent to Thyssen-Krupp Industries (TKI) for an HPGR crushing test program. All the material was crushed in a 1 m diameter HPGR unit. The material was subjected to a single pass through the HPGR and screened at (5/8 inch) 16 mm. The fines fractions were weighed and retained while the coarse fractions were sent to Tomra/Outotec for ore sorting test work. A diagram of the HPGR crushing circuit is shown in Figure 34. The material balance is reported in Table 26.

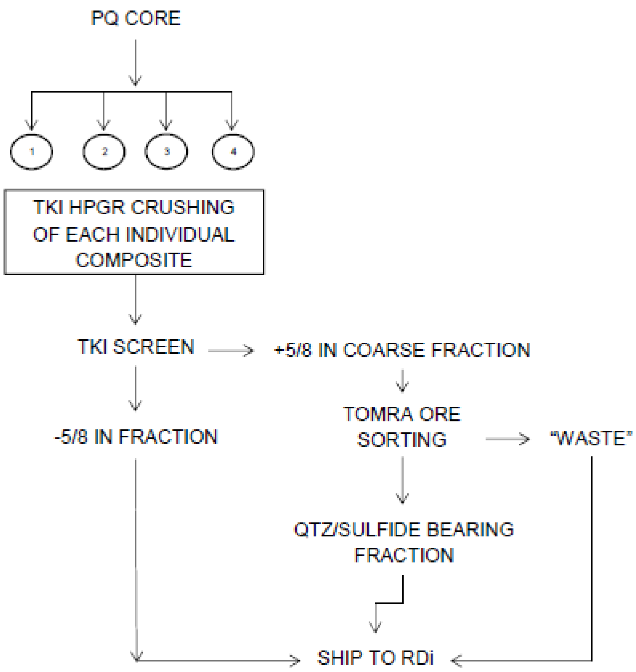


Figure 34 TKI HPGR Flowsheet, RDi September 2019

Material Balance for HPGR Tests			
Composite No	Sample Weight, kg	HPGR Products %	
		-32 mm+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

Table 26 Material Balance for HPGR Testing

10.3.2 Tomra-Outotec Ore Sorting Test Work

The >16 mm composite fractions were sent to Tomra-Outotec for ore sorting assessment. The fractions were weighed on arrival and washed to remove fines which were accounted for within the mass balance. The composites were split in equal fractions with each fraction subjected to two-step sorting design to separate out gold bearing sulfide mineralization and quartz from non-gold bearing waste material. The first step used XRT to separate material based on density, targeting the gold bearing sulfide material with a higher specific gravity. Different sensitivities were tested (1%, 2% and 5%) to assess the best gold recovery. The second stage used laser sorting to separate quartz (gold bearing) mineralization from the gangue non-gold bearing waste.

The calculated effects are shown for gold in Table 27, indicating an 8% to 9% mass reject, with 1.5% gold deportment for Composite C2. (from Table 26) A ~ 1.0 g Au/t head grade was improved to 1.2 g Au/t. This corresponds to 98.5% of the gold reporting to the mill post ore sort reject gold deportment. Despite a loss of 1.5% of the gold, 8% to 10% of the feed tonnage is rejected to waste which further reduces the power required in the grinding stage. Waste rock has the highest breakage energy requirement.

	Units	Comp 1			Comp 2			Comp 3			Comp 4		
Senstivity	%	C1 5%	C1 2%	C1 1%	C2 5%	C2 2%	C2 1%	C3 5%	C3 2%	C3 1%	C4 5%	C4 2%	C4 1%
TK HPGR Feed	kg	4399.9			4977.7			4370.7			4317.3		
Feed Grade	g/t Au	**1.54**			**0.99**			**0.74**			**0.55**		
Gold In Feed Stock	g Au	6.78			4.93			3.23			2.37		
HPGR Screen Undersize -16mm	kg	3629.9			4091.7			3645.2			3510.0		
Gold in -16mm by Mass Balance	g Au	6.36545	6.5379	6.521	4.6541	4.6674	4.6063	3.1103	2.9593	3.1356	1.8748	1.9579	1.987
Recovery as % of Total Feed	% Au	**93.94%**	**96.49%**	**96.24%**	**94.44%**	**94.71%**	**93.47%**	**96.16%**	**91.50%**	**96.95%**	**78.95%**	**82.46%**	**83.68%**
HPGR Screen Oversize +16mm	kg	770.0			886.0			725.5			807.3		
Tomra +16mm Head Grade	g/t Au	**0.53**	**0.31**	**0.33**	**0.31**	**0.29**	**0.36**	**0.17**	**0.38**	**0.14**	**0.62**	**0.52**	**0.48**
Gold in +16mm Sample Head Grade	g Au	0.41	0.24	0.25	0.27	0.26	0.32	0.12	0.27	0.10	0.50	0.42	0.39
Mass Split to XRT Product	%	57.0%	42.1%	33.2%	53.7%	47.1%	37.3%	45.5%	37.0%	26.5%	58.6%	45.2%	38.3%
Spilt Matreial to Product	kg	439.0	323.9	255.5	476.2	416.9	330.2	330.3	268.5	192.2	473.2	365.0	308.9
XRT Product Gold Assay	g/t Au	**0.83**	**0.53**	**0.79**	**0.40**	**0.46**	**0.59**	**0.25**	**0.63**	**0.30**	**0.94**	**0.97**	**1.05**
Gold in +16mm XRT Product	g Au	0.36	0.17	0.20	0.19	0.19	0.20	0.08	0.17	0.06	0.44	0.35	0.32
Stage Au Rec	% Au	88.79%	72.29%	78.69%	69.23%	73.46%	60.89%	67.62%	61.91%	58.05%	88.92%	84.99%	83.86%
Recovery as % of Total Feed	% Au	**5.38%**	**2.54%**	**2.96%**	**3.85%**	**3.88%**	**3.97%**	**2.59%**	**5.26%**	**1.77%**	**18.71%**	**14.91%**	**13.69%**
Split to XRT Reject	kg	331.0	446.1	514.5	409.8	469.1	555.8	395.3	457.0	533.3	334.1	442.3	498.4
Gold in +16mm XRT Reject	g Au	0.046	0.066	0.054	0.084	0.069	0.126	0.040	0.105	0.041	0.055	0.063	0.063
Wash Material	kg	6.0	9.9	5.5	7.3	7.8	10.4	13.3	12.5	9.6	5.9	6.6	4.9
Gold in Wash	g Au	0.003	0.003	0.001	0.001	0.001	0.005	0.003	0.004	0.001	0.002	0.002	0.001
Recovery as % of Total Feed	% Au	**0.05%**	**0.04%**	**0.01%**	**0.02%**	**0.02%**	**0.11%**	**0.10%**	**0.11%**	**0.02%**	**0.07%**	**0.07%**	**0.06%**
Laser Feed	kg	325.0	436.2	509.0	402.5	461.3	545.4	381.9	444.6	523.8	328.3	435.8	493.5
Gold in Laser Feed	g Au	0.043	0.063	0.053	0.083	0.068	0.120	0.037	0.101	0.041	0.054	0.061	0.061
Laser Product Mass Split	%	1.4%	2.0%	2.2%	6.7%	6.0%	7.6%	5.1%	3.5%	7.7%	1.4%	1.6%	1.6%
Laser Product Weight	kg	11.0	15.8	16.9	59.0	53.3	67.3	36.9	25.7	55.7	11.1	13.2	13.2
Gold in Laser Product	g Au	0.007	0.016	0.011	0.010	0.024	0.035	0.010	0.003	0.014	0.001	0.007	0.006
	g/t Au	0.63	1.00	0.64	0.17	0.46	0.52	0.27	0.11	0.25	0.08	0.50	0.48
Stage Au Rec	% Au	16.19%	25.02%	20.26%	11.69%	35.64%	28.94%	27.14%	2.87%	34.61%	1.63%	10.87%	10.39%
Recovery as % of Total Feed	% Au	**0.10%**	**0.23%**	**0.16%**	**0.20%**	**0.49%**	**0.71%**	**0.31%**	**0.09%**	**0.44%**	**0.04%**	**0.28%**	**0.27%**
Laser Reject Weight	kg	314.0	420.4	492.1	343.5	408.1	478.1	345.0	418.9	468.1	317.1	422.6	480.3
Gold in Laser Reject	g Au	0.036	0.047	0.043	0.074	0.044	0.086	0.027	0.098	0.027	0.053	0.054	0.055
	g/t Au	**0.11**	**0.11**	**0.09**	**0.21**	**0.11**	**0.18**	**0.08**	**0.23**	**0.06**	**0.17**	**0.13**	**0.11**
Overall Recovery of Gold in Feed	%	**99.47%**	**99.30%**	**99.37%**	**98.51%**	**99.11%**	**98.26%**	**99.17%**	**96.96%**	**99.17%**	**97.77%**	**97.72%**	**97.69%**
Deportment of Gold in Feed	%	0.53%	0.70%	0.63%	1.49%	0.89%	1.74%	0.83%	3.04%	0.83%	2.23%	2.28%	2.31%
Mass % to Product	%	92.9%	90.4%	88.8%	91.9%	91.8%	90.4%	92.1%	90.4%	89.3%	92.7%	90.2%	88.9%
Mass % to Reject	%	**7.1%**	**9.6%**	**11.2%**	**8.1%**	**8.2%**	**9.6%**	**7.9%**	**9.6%**	**10.7%**	**7.3%**	**9.8%**	**11.1%**
Adjusted Feed Grade	g/t	**1.65**	**1.69**	**1.72**	**1.20**	**1.07**	**1.08**	**0.80**	**0.79**	**0.82**	**0.58**	**0.60**	**0.60**
% Inc in Grade	%	7.12%	9.79%	11.88%	21.25%	7.96%	8.71%	7.67%	7.24%	11.07%	5.52%	8.32%	9.92%

Table 27 Tomra Test Work Yields

10.3.3 Preparation of Composites for Metallurgical Test Work

The minus 16 mm fines generated during the TKI HPGR test work were recombined with the Tomra ore sorting product stream (XRT+Laser+Wash fines) for the combined composite in each case, on a weighted basis, to generate an adjusted feed grade composite samples (Composites 1 to 4).

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.

10.3.4 Head Assays

The composite samples were submitted for head analysis. The test results are summarized in Table 28.

Element	Composite			
	1	2	3	4
Au , g/t	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
Cu, Total ppm	467	285	241	384

Table 28 2017 Composite Head Grade

- The samples assayed from 0.348 g Au/t to 0.760 g Au/t.
- The total sulfur content ranged from 0.43% to 1.26%.
- The copper values ranged from 241 ppm to 467 ppm.

The samples contained significantly lower gold values than projected from the drilling data as shown in Table 23. Vista investigated the drill holes in question used for the 2017 program and found that the drill holes were completed at an oblique angle, where too few ore zones were intersected to provide a representative sample, resulting in a biased sample for the test work program. Additional drilling was implemented to address the geological block model with projected (adjusted) grades detailed in Table 29.

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

Table 29 Projected – Corrected g Au/t

10.3.5 Mineralogy

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 key findings are listed as:

- Quartz was the primary phase in all samples and accounts for over 60% of the volume.
- Quartz occurred as very fine mosaic grains (5 to 10 µm) or as angular to rounded grains in sizes from 5 to 125 µm. Some very coarse 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 anhedral 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 a monoclinic structure.
- Most of the gold grains identified were associated with pyrite and ranged in size from 3 to 28 µm.
- No discrete silver minerals were identified in any of the composite samples.

10.3.6 Abrasion Indices

The samples were submitted for bond abrasion index determination. The test results are summarized in Table 30.The results indicate that the material is low to moderately abrasive.

Sample	Ai, g
Composite CC ¾ x ½ inch	0.1603
Composite 1 – 16 mm	0.2278
Composite 2 – 16 mm	0.1616
Composite 3 – 16 mm	0.2006
Composite 4 – 16 mm	0.2250

Table 30 Abrasion Indices Ai, g

10.3.7 Bond Ball Mill Work Indices

BWi were determined, Table 31, at a closing screen size of P_{80} of 100 mesh (150 µm) for the various products, namely HPGR-crushed, conventionally crushed, ore-sorting “XRT” product, ore-sort “Laser” waste.

	Sample	BWi (kWh/t)
1	Comp 1 - HPGR	23.1
2	Comp 2 - HPGR	24.41
3	Comp 3 - HPGR	23.79
4	Comp 4 - HPGR	24.48
5	Composite Crushed	25.01
6	1.1 XRT Product	23.88
7	2.1 XRT Product	25.15
8	3.1 XRT Product	25.98
9	4.1 XRT Product	26.55
10	5.1 XRT Product	26.91
11	6.1 XRT Product	26.44
12	7.1 XRT Product	24.54
13	8.1 XRT Product	24.63
14	9.1 XRT Product	25.44
15	10.1 XRT Product	25.37
16	11.1 XRT Product	25.89
17	12.1 XRT Product	25.61
18	2.2 Laser Waste	25.78
19	4.2 Laser Waste	26.34
20	8.8 Laser Waste	24.04
21	10.2 Laser Waste	23.89

Table 31 Bond Mill Work Indices

The test results indicate the following:

- The BWi for the > 16 mm 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 BWi for the products ranged from 23.10 to 24.28 for Composites 1 to 4. A BWi of 24.50 kWh/t was selected for the design of the primary ball mill circuit.

10.3.8 Leach Test Work

Several series of leach tests were conducted to examine the effects of various grind sizes, variation in feed slurry density, variations in cyanide concentration and a two-stage grinding approach (Table 32 and Table 33).

The test procedure consisted of grinding the ore to the desired particle size in a single stage or two stages, to replicate the proposed plant configurations, and transferring the ground pulp 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 50 ppm lead nitrate. Sodium cyanide was then added to a calculated 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 concentration of 20 g/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.

- Gold extraction increased with a finer grind size as expected with further liberation based on a single stage initial grind. Composites 1, 3 and 4 all showed improving trends, while the trend for Composite 2 was flat and showed little to no improvement with a finer initial grind.
- Gold extractions for the two-stage grind conducted at 53 µm, improved significantly for composites 2, 3 and 4, while composite 1 was in line with the grind size trend for improving extraction.
- The gold extraction for average grade composites 1 and 4 ranged from 82.8% to 87.6% at a P_{80} of 46 µm in a single-stage grind. However, for a two-stage grind to P_{80} of 53 µm, gold extraction improved from 86.4% to 89.7%.
- The sodium cyanide consumption in the two-stage grind tests was approximately 20% lower compared to the single-stage grind.
- The preliminary optimization study indicated that the leach circuit could potentially operate at higher pulp density (approximately 50% solids) and lower cyanide concentration (750 ppm initial concentration) without impacting gold extraction.

Test	Comp	Target grind size	%Ext Au	Head g Au/t	% Ext Ag	Head g Ag/t	NaCN Consumption kg/mt	Lime Consumption kg/mt
BR1	1	74	84.4	0.75	26.9	1.4	0.515	3.782
BR2	1	74	84.9	0.68	31.3	0.9	0.512	3
BR3	1	63	85.1	0.65	4.5	1.0	0.471	3.351
BR4	1	63	85.4	0.66	50.1	2.0	0.514	2.987
BR25†	*1*	*53*	*86.6*	*0.67*	*55.8*	*0.9*	*0.393*	*4.972*
BR26†	*1*	*53*	*86.2*	*0.67*	*16.6*	*0.5*	*0.336*	*4.866*
BR5	1	44	85.1	0.69	4.0	1.2	0.516	3.578
BR6	1	44	87.6	0.77	20.5	0.2	0.515	3.446
BR7	4	74	80	0.65	20.9	0.2	0.551	3.237
BR8	4	74	79.7	0.71	11.1	0.4	0.516	2.992
BR9	4	63	81.8	0.75	59.9	0.5	0.576	2.98
BR10	4	63	82.9	0.72	20.3	0.2	0.513	3.008

Test	Comp	Target grind size	%Ext Au	Head g Au/t	% Ext Ag	Head g Ag/t	NaCN Consumption kg/mt	Lime Consumption kg/mt
BR31†	*4*	*53*	*86.1*	*0.69*	*40.6*	*0.7*	*0.392*	*4.521*
BR32†	*4*	*53*	*86.4*	*0.68*	*25.7*	*0.5*	*0.397*	*4.501*
BR11	4	44	82.8	0.72	20.0	0.2	0.575	3.458
BR12	4	44	84.1	0.65	51.5	0.4	0.576	2.939
BR13	2	74	77	0.42	25.1	0.8	0.336	3.743
BR14	2	74	76.4	0.44	29.7	0.9	0.393	3.46
BR15	2	63	77.3	0.45	7.9	0.7	0.393	3.533
BR16	2	63	75.1	0.44	13.8	0.7	0.394	3.493
BR27†	*2*	*53*	*85.8*	*0.44*	*39.9*	*0.7*	*0.398*	*4.446*
BR28†	*2*	*53*	*85.2*	*0.44*	*49.3*	*0.8*	*0.458*	*4.529*
BR17	2	44	68.3	0.5	7.9	0.7	0.453	3.631
BR18	2	44	75.5	0.48	11.3	0.5	0.453	3.678
BR19	3	74	65.2	0.3	7.6	0.7	0.456	4.554
BR20	3	74	64	0.27	7.6	0.7	0.397	4.545
BR21	3	63	66.5	0.29	11.0	0.5	0.454	4.555
BR22	3	63	69.8	0.29	7.7	0.7	0.396	4.678
BR29†	*3*	*53*	*80.1*	*0.31*	*28.2*	*0.6*	*0.514*	*4.773*
BR30†	*3*	*53*	*80.5*	*0.32*	*11.5*	*0.5*	*0.513*	*4.93*
BR23	3	44	69.8	0.27	7.7	0.7	0.454	4.7
BR24	3	44	70	0.27	18.2	0.7	0.454	4.632

† Two-stage grinding tests completed at 53 µm target grind size.

Table 32 Grind size variations – Gold Extractions

Test	Comp	Target Grind Size	% Ext Au	Head, g Au/t	% Ext Ag	Head g Ag/t	NaCN Consumption kg/mt	Lime Consumption kg/mt	Conditions
BR62	1	53	87.8	0.65	36.1	0.6	0.399	3.01	40% Solids, 1 g/l NaCN
BR63	1	53	88.8	0.67	49.8	0.8	0.399	3.003	40% Solids, 1 g/l NaCN
BR64	1	53	89.1	0.66	58.4	1	0.273	3.008	45% Solids, 1 g/l NaCN
BR65	1	53	88.7	0.64	43.2	0.7	0.271	3.011	45% Solids, 1 g/l NaCN
BR66	1	53	87.5	0.63	44.2	0.7	0.27	3.028	45% Solids, 0.75 g/l NaCN
BR67	1	53	88.4	0.62	39.9	0.7	0.221	3.024	45% Solids, 0.75 g/l NaCN
BR68	1	53	88.8	0.64	41.6	0.7	0.21	3.007	45% Solids, 0.50 g/l NaCN
BR69	1	53	88.4	0.65	46.7	0.8	0.212	3.007	45% Solids, 0.50 g/l NaCN
BR70	1	53	89.5	0.66	45.7	1.1	0.305	3.021	50% Solids, 1 g/l NaCN
BR71	1	53	89.7	0.63	37.7	1	0.344	3.015	50% Solids, 1 g/l NaCN

Test	Comp	Target Grind Size	% Ext Au	Head, g Au/t	% Ext Ag	Head g Ag/t	NaCN Consumption kg/mt	Lime Consumption kg/mt	Conditions
BR72	4	53	86.7	0.62	39.8	1	0.337	3.014	40% Solids, 1 g/l NaCN
BR73	4	53	86.8	0.62	53	0.9	0.275	3.012	40% Solids, 1 g/l NaCN
BR74	4	53	85.9	0.61	53.4	0.9	0.315	3.012	45% Solids, 1 g/l NaCN
BR75	4	53	86.8	0.62	49.1	0.8	0.27	3.017	45% Solids, 1 g/l NaCN
BR76	4	53	86.4	0.6	41.3	0.7	0.222	3.013	45% Solids, 0.75 g/l NaCN
BR77	4	53	86	0.62	42.9	1.1	0.27	3.018	45% Solids, 0.75 g/l NaCN
BR78	4	53	86.5	0.64	35.5	0.9	0.21	3.015	45% Solids, 0.50 g/l NaCN
BR79	4	53	86.1	0.62	50.5	0.8	0.21	3.022	45% Solids, 0.50 g/l NaCN
BR80	4	53	86.4	0.63	43	0.7	0.264	3.014	50% Solids, 1 g/l NaCN
BR81	4	53	86	0.64	46.9	0.8	0.262	3.023	50% Solids, 1 g/l NaCN
BR82	3	53	84.7	0.25	10.9	0.5	0.46	3.011	40% Solids, 1 g/l NaCN
BR83	3	53	94.9	0.25	22.6	0.5	0.272	3.011	40% Solids, 1 g/l NaCN
BR84	3	53	84.7	0.25	9.8	0.4	0.271	3.010	45% Solids, 1 g/l NaCN
BR85	3	53	84.8	0.25	9.8	0.4	0.372	3.010	45% Solids, 1 g/l NaCN
BR86	3	53	83.2	0.24	9.9	0.4	0.269	3.017	45% Solids, 0.75 g/l NaCN
BR87	3	53	86.3	0.25	27.1	0.6	0.322	3.01	45% Solids, 0.75 g/l NaCN
BR88	3	53	83.8	0.25	9.9	0.4	0.211	3.011	45% Solids, 0.50 g/l NaCN
BR89	3	53	84.4	0.24	9.8	0.4	0.211	3.016	45% Solids, 0.50 g/l NaCN
BR90	3	53	85	0.25	9.3	0.4	0.347	3.011	50% Solids, 1 g/l NaCN
BR91	3	53	84.9	0.25	9.2	0.4	0.346	3.011	50% Solids, 1 g/l NaCN

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

Table 33 Variation in Density and Cyanide additions @ 53 µm two-stage grind

10.3.9 Cyanide Destruction

The cyanide leach residue for composites No. 1 and No. 4 was subjected to cyanide destruction tests using the INCO air/SO_2 method. Approximately 1.5 liters of leach residue at 50% solids was agitated with sodium metabisulfite (SMBS) three times the stoichiometric amount of free cyanide and copper sulfate. Samples were taken every hour, and free cyanide was determined. Although 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 34. The test results indicate the following:

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

- One hour of detox residence time is sufficient for the process.

Forms of Cyanide ppm	Composite 1		Composite 4	
	Start of Detox	End Of Detox	Start of Detox	End of Detox
Free	600	6.3	590	4.0
Total	587	3.6	615	2.2
WAD	590	5.0	560	2.6

Table 34 Cyanide Destruction Test Results

10.3.10 Thickening Tests

Static cylinder thickening tests on leach residue with 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 35, 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 requirement increases significantly if the desired underflow solids is 50%.

Composite	Flocculant	Unit Area Required m^2/mt/day			
		40%	45%	50%	55%
1	8 g/t DAF-10	0.031	0.044	0.164	2.41
2	8 g/t DAF-10	0.050	0.069	0.150	2.448
3	8 g/t DAF-10	0.042	0.081	0.191	2.436
4	8 g/t DAF-10	0.083	0.182	0.650	2.425

Note: *All tests conducted at 25% feed solids, pH of 11 and two stage grind to 53 µm.*

Table 35 Thickener Unit Area Requirements

10.4 2018-2019 Metallurgical Test Work

The metallurgical test program was developed to assess higher feed gold grades using the process flowsheet developed in the previous pre-feasibility study. Vista conducted a drilling program consisting of four PQ drill holes, totaling 497.5 m, to obtain six tonnes of sample. The sample was used to generate six composites for the metallurgical study.

The following is a brief description of the composites:

- Big Yellow, totaling 2.5 tonnes, with a head grade of 1.7 g Au/t.
- Big Blue, totaling 2.5 tonnes, with a head grade of 1.4 g Au/t.
- Weir, totaling 1.0 tonnes, with a head grade of 1.0 g Au/t for HPGR testing.
- Small Yellow, totaling, 40 kg, with a head grade of 1.27 g Au/t.

- Small Blue, totaling 40 kg, with a head grade of 0.84 g Au/t.
- Small Red, totaling 40 kg, with a head grade of 1.02 g Au/t.

10.4.1 2017-18 Metallurgical Drill Hole Program

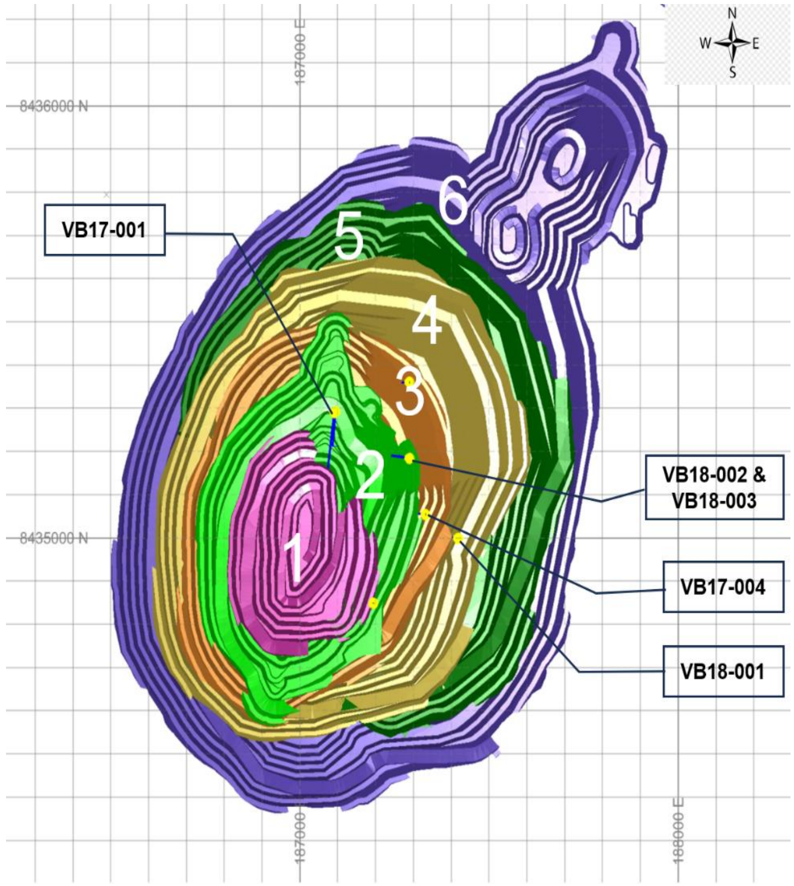
An additional metallurgical drill hole program was started in December 2017 and completed in 2018. For the program it was decided to drill holes approximately perpendicular to the mineralization. Locations were chosen to limit waste drilling as much as possible, to "split" known existing drilling and to avoid "twinning," while remaining within 15 m to 60 m of existing holes.

The results of the 2017-18 PQ metallurgical drill holes VB17-004, VB18-001, -002 and -003 are shown in Table 36 and Figure 35.

Drill Hole ID	HG Core Length (m)	Composite (g Au/t)	Block Model (g Au/t)
VB17-004	113.5	1.461	1.45
VB18-001	132	1.13	1.52
VB18-002	110.7	1.499	1.56
VB18-003	141	1.1	1.13

Table 36 2017-2018 Drill Hole Data

Figure 35 as prepared by Vista illustrates the location of the 2017-18 drill holes in relation to the FS mining phases.



Source: prepared by Vista 2025

Figure 35 Drill Hole locations, Vista 2025

10.4.2 HPGR Testing Thyssen Krupp Industries (TKI)

The Big Yellow and Big Blue composites were dispatched to TKI and underwent the same HPGR crushing program used in the 2018 tests (Table 37). Material was pre-crushed with a jaw crusher before undergoing a single pass through the 1 m diameter HPGR. The product was screened at 16 mm (5/8 inch). Both coarse and undersize fractions were weighted and drummed. The > 16 mm coarse fraction was dispatched to Tomra-Outotec for ore sorting tests.

Composite	Sample Weight, kg	HPGR Product +16 mm %	HPGR Product - 16 mm %	M-dot (ts/hm^3)	Specific Grinding Pressure (N/mm^2)	Specific Power (kWh/t)
Big Yellow	2400	18.6	81.4	296	3.82	1.98
Big Blue	2370	17.8	82.2	301	3.74	1.89

Table 37 TKI – HPGR Results

10.4.3 HPGR testing at Weir Minerals

Approximately one tonne of drill core was also sent to Weir Minerals for evaluating the Weir Enduron HPGR for Mt Todd ore treatment. 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 tph·m³. The average mass oversize at 16 mm screen was 17.3%. The results in Table 38 are similar to the previous HPGR testing at TKI.

Composite	Sample Weight, kg	HPGR Product +16 mm %	HPGR Product -16 mm %	M-dot (ts/hm^3)	Specific Grinding Pressure (N/mm^2)	Specific Power (kWh/t)
Weir (A1)	294.5	16.8	83.2	250	3.6	1.99
Weir (A2)	286	18.0	82.0	255	3.5	1.92
Weir (A3)	283.5	17.1	82.9	258	3.6	1.91

Table 38 Weir HPGR testing

10.4.4 TOMRA/Outotec Ore Sorting Test Work

The > 16 mm Big Yellow and Big Blue composite fractions were received at Tomra-Outotec for ore sorting assessment (Table 39). The fractions were weighed on arrival and washed to remove fines which were accounted for within the mass balance. The composites were split in equal fractions with each fraction subjected to two-step sorting design to separate out gold bearing sulfide mineralization and quartz from non-fold bearing waste material. The first step used XRT to separate material based on density to target the gold bearing sulfide material with a higher specific gravity. Two different sensitivities were tested (2% and 5%). The second stage used laser to separate out the quartz (gold bearing) mineralization from the gangue non-gold bearing waste.

The key findings from the tests indicated:

- Open circuit HPGR produced approximately 18% of the feed fraction, where the ore sorting of the plus 16 mm fraction rejected 44.5% of Big Yellow and 63.1% of Big Blue.
- The calculated head grade of the plus 16 mm for Big Yellow was 0.73 g Au/t and Big Blue was 0.74 g Au/t. The vein material is softer than the greywacke host rock and preferentially crushes leaving the coarser fraction with a lower head grade than the original composite.
- Ore sorting of the Big Yellow composite rejected 8.7% mass fraction of the plus 16 mm feed.
- Ore sorting of the Big Blue composite rejected 7.9% mass fraction of the plus 16 mm feed.
- Removal of waste resulting in 7% improvement in mill feed grade.

Sample (XRT Sensitivity)	% Distribution						Grade (g Au/t)			
	XRT Product		Laser Product		Reject		Feed	XRT Product	Laser Product	Reject
	Wt	Au	Wt	Au	Wt	Au				
Big Yellow (5%)	53.1	90.4	2.4	3.0	44.5	6.6	0.74	1.26	0.90	0.11
Big Yellow (2%)	45.6	52.4	2.9	8.2	51.5	39.5	0.79	0.91	2.26	0.60
Big Blue (5%)	50.6	87.4	2.4	2.4	47.0	10.2	0.73	1.26	0.73	0.16
Big Blue (2%)	33.5	78.1	3.4	4.4	63.1	17.5	1.12	2.60	1.45	0.31

Table 39 2019 Tomra Test Results

10.4.5 Steinert Ore Sorting Test Work

Recombined 2017 test composites were dispatched to Steinert for ore sorting evaluation. Three of the composites were testes (composite 1, 3 and 4). The results are summarized in Table 40 and were similar to those obtained from the 2017 Tomra test runs. Steinert tested five sensitivity settings for the -60 mm-12 size range using 3d XRT-T system.

	Sensitivity Setting	Total Sample (kg)	Product Mass	Cum Mass Rec %	g Au/t	g Au/t Cum	Au Rec %	Au Cum Rec %	Waste Rejection (%)	Waste g Au/t
Comp 1	1	120.4	3.2%	3.2%	3.71	3.71	45.5%	45.5%	96.8%	
	2	116.6	3.9%	6.9%	0.82	2.15	11.9%	57.4%	96.1%	
	3	112.1	9.9%	16.1%	0.32	1.10	11.5%	68.9%	90.1%	
	4	101	22.8%	35.2%	0.15	0.59	11.2%	80.1%	77.2%	
	5	78	41.0%	61.8%	0.08	**0.37**	7.7%	87.9%	58.8%	0.08
Comp 2	1	119.6	1.8%	1.8%	3.99	3.99	51.9%	51.9%	98.2%	
	2	117.5	2.4%	4.1%	0.91	2.23	15.8%	67.8%	97.6%	
	3	114.7	7.5%	11.3%	0.19	0.93	9.9%	77.6%	92.5%	
	4	106.1	19.0%	28.2%	0.03	0.39	4.3%	81.9%	81.0%	
	5	85.9	35.7%	53.8%	0.03	**0.22**	6.5%	88.4%	64.3%	0.03
Comp 2	1	126.7	3.2%	3.2%	3.99	3.99	35.5%	35.5%	96.8%	
	2	122.6	4.2%	7.3%	0.86	2.25	9.5%	44.9%	95.8%	
	3	117.5	11.2%	17.7%	0.49	1.21	13.9%	**58.8%**	88.8%	
	4	104.3	22.9%	36.5%	0.32	0.75	16.7%	75.5%	77.1%	
	5	80.4	42.7%	63.6%	0.06	**0.46**	4.6%	80.1%	57.3%	0.20

Table 40 Steinert Ore Sorting Results

10.4.6 Preparation of Composites for Metallurgical Test Work

The HPGR fines product (-16 mm) and the ores sorting products (+16 mm product and wash fines) were recombined on a weight % basis per composite. The recombined samples were used for the leach stage tests.

10.4.7 Head Assays

A representative sample from each composite was pulverized and submitted for head analysis. Samples were submitted for gold, silver and sulfur using Inductively Coupled Plasma (ICP) analysis and Fire Assay/AA. Results are detailed in Table 41 and Table 42 and Table 43.

Composite	Original Head Grade Estimate (g Au/t)	<16 mm HPGR Fines (g Au/t) -Fire Assay	>16 mm XRT Product – Fire Assay	Combined Composite Sample- Florin Head – Triplicate 5 Assay (g Au/t) – Fire Assay	Combined Composite Sample – Hazen Head Assay Split for Forin Sample (g Au/t) – Fire Assay
Big Yellow	1.70	1.64	1.26	0.83	1.68
Big Blue	1.39	1.65	1.26	0.91	1,18
Weir	1.00	1.05	N/A	N/A	N/A

Table 41 Head Assays – Large Composites

Composite	Original Head Grade Estimate (g Au/t)	Initial Florin Head Assay – (g Au/t) Fire Assay	Combined Composite Sample- Florin Head – Triplicate 5 Assay (g Au/t) – Fire Assay	Combined Composite Sample – Hazen Head Assay Split for Forin Sample (g Au/t) – Fire Assay
Small Yellow	1.27	1.48	0.67	0.72
Small Red	1.02	0.44	0.51	0.65
Small Blue	0.84	2.60	2.62	2.95

Table 42 Head Assays – Small Composites

Element	Big Yellow	Big Blue	Weir	Small Yellow	Small Blue	Small Red
Au Assay 1 g/t	0.912	0.840	1.049	0.668	3.420	0.532
Au Assay 2 g/t	0.902	0.819		0.684	2.253	0.517
Au Assay 3 g/t	0.098	0.806		0.651	2.186	0.480
Au – Resample g/t	1.68	1.13		0.720	2.95	0.652
g Ag/t	<1	<1	<1	<1	<1	<1
S_{Total}, %	1.90	1.49	1.89	1.19	0.57	0.81
$S_{Sulfide}$, %	1.53	1.14	1.45	0.88	0.38	0.65
$S_{Sulphatel}$, %	0.37	0.35	0.44	0.31	0.20	0.16

Table 43 Head Assays -Detailed

10.4.8 Mineralogical Study

The Big Yellow and Weir composite samples were submitted for optical mineralogical analysis to examine gold occurrence and required liberation size. Each sample was prepared as a standard polished thin section for the study by reflected light (RL) and transmitted polarized light (PL) microscopy.

10.4.8.1 Silicate Mineralogy

Quartz is the dominant mineral phase and accounts for 56% of the samples' overall volume for both composites. Grain sizes vary from 5 to 300 µm and occur as mosaic aggregates. Fragments of coarse vein quartz are also present with grain sizes up to 1 mm. Mixed with the mosaic aggregates are significant amounts of mica. Green-colored chlorite is the main type and occurs as interstitial mats between the quartz grains. Dark-colored brown biotite and clear muscovite are also present. Mica grain sizes do not exceed 100 µm.

10.4.8.2 Carbonate Mineralogy

Carbonates are present in low amounts as calcite/dolomite in the two composites. It occurs as small grains and mats mixed with mica and fills some of the fractures in the quartz. Grain sizes are very fine from 2 to 100 µm, with the odd larger liberated fragments measuring up to 600 µm.

10.4.8.3 Oxide Mineralogy

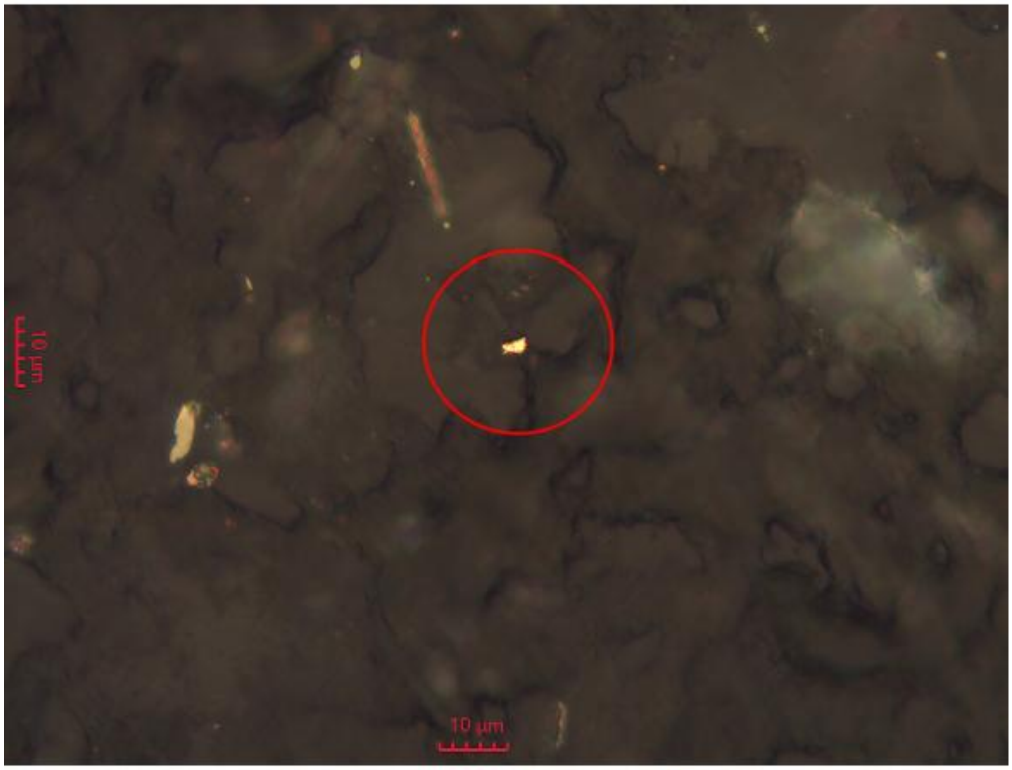
Oxides are present in trace amounts in the two composites. Hematite/goethite are noted but are rare. Iron oxide is seen staining some silicates and as thin, rare rinds along pyrite. Rutile occurs as small, rounded honey-colored grains and thin, needle-shaped prisms captured in quartz. Minor ilmenite and magnetite are noted.

10.4.8.4 Sulfide Mineralogy

Sulfides are present in low amounts with several noted species, see Table 44. Pyrite occurs as anhedral to subhedral grains and thin strings along quartz margins, with measurements ranging from 2 to 250 µm. Pyrrhotite tends to be coarser and occurs as anhedral grains that vary from 2 µm to 1000 µm. A small population of pyrrhotite shows replacement by fine grained marcasite. Chalcopyrite is present as trace and measure up to 125 µm. Traces of galena and sphalerite were noted in the Big Yellow composite with grain sizes ranging from 5 to 25 µm.

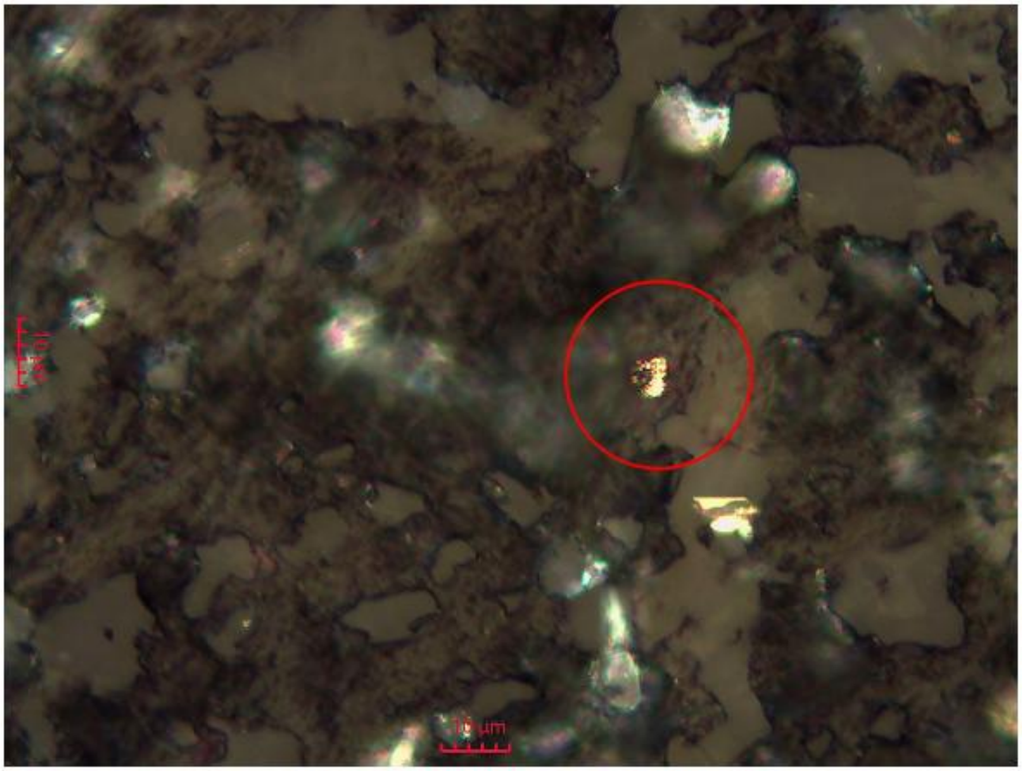
10.4.8.5 Gold Mineralogy

An extensive search using reflected light at high magnification identified one grain of Au in both samples. Gold in both composite samples is extremely fine grained with measurements ranging from 3 to 6 µm, see Figure 36 and Figure 37. In both cases the gold is associated with quartz in multi-mineral rock fragments.



Client Sample No.: **Mt. Todd BYellow**
Small 3µm grain of Au sits between quartz grains – 500X RL

Figure 36 Gold Mineralization – Mt Todd Yellow composite



Client Sample No.: **Mt. Todd Weir**
Small 6µm grain of Au sits next to quartz – 500X RL

Figure 37 Gold Mineralization – Mt Todd Weir composite

Major Mineral	Big Yellow	Weir	Minor Minerals (Trace Amounts Noted)	Big Yellow	Weir
Quartz – SiO_2	56%	56%	Chalcopyrite- $CuFeS_2$	Y	Y
Chlorite-$(X,Y)_{4\text{-}6}(Si,Al)_4O_{10}(OH,O)_8$	24%	25%	Galena – PbS	Y	Y
Muscovite/Biotite- $KAl_2(AlSi_3O_{10})(F,OH)_2$	17%	15%	Sphalerite – ZnS	Y	Y
Dolomite/Calcite -$CaMg(CO_3)_2$	1%	2%	Rutile- TiO_2 Ilmenite - $FeTiO_3$	Y	Y
Pyrite – FeS_2	1%	1%	Marcasite – FeS_2 Magnetite – Fe_3O_4 Hematite -Fe_2O_3	Y	Y
Pyrrhotite - FeS	1%	1%	Gold – Au	Y	Y

Table 44 Mineral Makeup – Big Yellow/ Weir – Major/Minor Mineralization

10.4.9 Bond Ball Mill Work Index Tests and Abrasion Index

A series of BWi tests were conducted on the recombined ore sorted HPGR products for Big Yellow, Big Blue and Weir composites, where the product stream was recombined with the product fines to generate the new mill feed equivalent sample. The tests were conducted with a closing screen of 150 µm, see Table 45.

- The BWi values for Big Yellow and Big Blue samples, following rejection of ore sorting waste, were lower than the 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 comparable to the value selected for mill design from the 2017 test composites of 24.4 kWh/t.
- The abrasion index generated from the Weir –(ROM) sample was 0.2187 g.

Composite	BWi (kWh/t)	Ai, g
Big Yellow	25.08	
Big Blue	23.41	
Weir	25.81	0.2187

Table 45 BWi and Ai for Big Yellow , Big Blue and Weir Composites

10.4.10 Secondary Grinding Test Work and Technology Selection

The previous metallurgical study indicated that gold extraction improves with finer liberation size due to the nature of the gold size in the quartz veins, which was highlighted in the mineralogical study. A two-stage grind option was developed in the laboratory to reduce the primary grind transfer from 250 µm in the plant design to a secondary grind size of 40 µm to improve the gold extraction kinetics.

Alternative secondary grinding methods were applied including:

- RDi – Lab Mill.
- IsaMill.
- FLS VXP Mill.

The use of IsaMills and the VXP mill was also investigated in 2017 along with the standard lab rod mill for secondary size reduction. Grind results for each mill type are detailed in Table 46 - Table 48 and Figure 38 and Figure 39.

Test	Comp	Primary Grind (µm)	Secondary Grind (µm)	Type	Measured Grind size µm	Extraction % Au	Head Grade g Au/t	Residue Grade g Au/t
BR104	4	250	63	IsaMill	70	85.3	0.63	0.09
BR105	4	250	63	IsaMill	70	84.9	0.61	0.09
BR106	4	250	63	IsaMill	70	84.1	0.61	0.1
BR143	**Big Blue**	250	59	IsaMIll	59	84.8	1.17	0.18
BR144	**Big Blue**	250	59	IsaMIll	59	84.9	1.21	0.18
BR147	**Weir**	250	69	IsaMIll	69	85.5	0.95	0.14
BR148	**Weir**	250	69	IsaMIll	69	85	0.91	0.14
BR153	**Small Blue**	250	53	IsaMIll	53	93.6	1.96	0.12
BR154	**Small Blue**	250	53	IsaMIll	53	93.6	1.9	0.12
BR157	**Small Yellow**	250	59	IsaMIll	59	88.5	0.77	0.09
BR158	**Small Yellow**	250	59	IsaMIll	59	87.4	0.93	0.12
BR161	**Small Red**	250	52	IsaMIll	52	91.4	0.72	0.06
BR162	**Small Red**	250	52	IsaMIll	52	92.3	0.73	0.06
BR165	2	250	49	IsaMIll	56	83.6	0.52	0.08
BR166	2	250	49	IsaMIll	56	85	0.52	0.08
BR167	3	250	49	IsaMIll	49	85.8	0.32	0.05
BR168	3	250	49	IsaMIll	49	78.5	0.21	0.04
BR173	**Big Blue**	250	59	IsaMIll	59	84.2	1.19	0.19
BR174	**Big Blue**	250	59	IsaMIll	59	83.1	1.2	0.2
BR175	**Big Blue**	250	59	IsaMIll	59	79.2	1.06	0.22
BR204	**Big Yellow**	250	22	IsaMIll	22	93.1	1.7	0.12
BR205	**Big Yellow**	250	22	IsaMill	22	93	1.63	0.11
BR206	**Big Blue**	250	20	IsaMill	20	92.7	1.1	0.08
BR207	**Big Blue**	250	20	IsaMill	20	92.7	1.09	0.08
BR208	**Weir**	250	22	IsaMill	22	91.9	0.84	0.07
BR209	**Weir**	250	22	IsaMill	22	91.8	0.88	0.07
BR210	2	250	22	IsaMill	22	88.5	0.42	0.05
BR211	2	250	22	IsaMill	22	89	0.41	0.05
BR212	3	250	21	IsaMill	21	87.5	0.26	0.03
BR213	3	250	21	IsaMill	21	86.9	0.26	0.03

Table 46 IsaMill Regrind Results

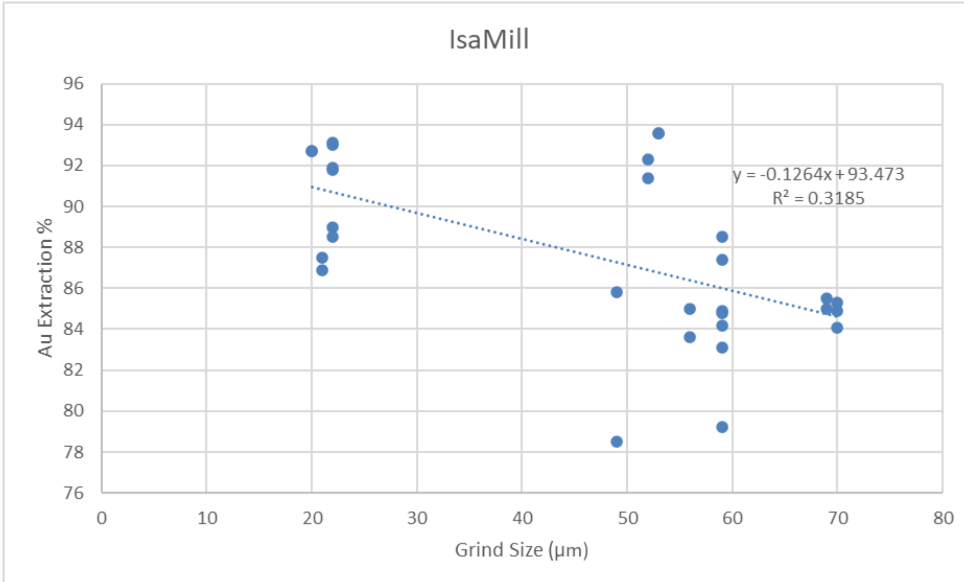


Figure 38 IsaMill Regrind – Grind Sensitivity, GRES 2025 from 2017 Test Data

Test	Comp	Primary Grind (µm)	Secondary Grind (µm)	Type	Measured Grind size µm	Extraction % Au	Head Grade g Au/t	Residue Grade g Au/t
BR113	Big Yellow	250	60	VXP	101	86.1	1.77	0.25
BR114	Big Yellow	250	60	VXP	101	85.4	1.77	0.26
BR115	Big Yelllow	250	45	VXP	74	86.4	1.67	0.23
BR116	Big Yelllow	250	45	VXP	74	87	1.7	0.22
BR117	Big Yelllow	250	38	VXP	76	87.3	1.74	0.22
BR118	Big Yelllow	250	38	VXP	76	87	1.7	0.22
BR119	Big Blue	250	60	VXP	91	87.6	1.82	0.23
BR120	Big Blue	250	60	VXP	91	88.9	1.74	0.19
BR121	Big Blue	250	45	VXP	97	86.6	1.2	0.16
BR122	Big Blue	250	45	VXP	97	84.6	1.24	0.19
BR123	Big Blue	250	38	VXP	74	89.1	1.26	0.14
BR124	Big Blue	250	38	VXP	74	87.5	1.21	0.15
BR125	Weir	250	60	VXP	87	86.5	0.87	0.12
BR126	Weir	250	60	VXP	87	85.5	0.88	0.13
BR127	Weir	250	45	VXP	79	87.4	0.86	0.11
BR128	Weir	250	45	VXP	79	88.4	0.89	0.1
BR129	Weir	250	38	VXP	69	89.1	0.83	0.09
BR130	Weir	250	38	VXP	69	86.9	0.86	0.11
BR131	2	250	38	VXP	59	84.8	0.46	0.07
BR132	2	250	38	VXP	59	86.2	0.46	0.06

Test	Comp	Primary Grind (µm)	Secondary Grind (µm)	Type	Measured Grind size µm	Extraction % Au	Head Grade g Au/t	Residue Grade g Au/t
BR133	3	250	38	VXP	60	81.6	0.18	0.03
BR134	3	250	38	VXP	60	80.6	0.18	0.03
BR182	Big Blue	250	44	VXP	44	87.8	1.22	0.15
BR195	Big Yellow	250	28	VXP	31	90.4	1.69	0.16
BR196	Big Yellow	250	28	VXP	31	90.3	1.73	0.17
BR197	Big Blue	250	28	VXP	29	90.4	1.21	0.12
BR198	Big Blue	250	28	VXP	29	90.1	1.16	0.11
BR199	Weir	250	28	VXP	35	89.5	0.9	0.09
BR200	Weir	250	28	VXP	35	89.6	0.85	0.09
BR215	Big Yellow	250	31	VXP	31	87.8	1.75	0.21
BR216	Big Yellow	250	31	VXP	31	89.2	1.7	0.18
BR217	Big Yellow	250	31	VXP	31	85.6	1.68	0.24
BR218	Big Yellow	250	31	VXP	31	84.9	1.73	0.26
BR219	Big Blue	250	29	VXP	29	88.8	1.2	0.13
BR220	Big Blue	250	29	VXP	29	60.7	1.7	0.67
BR221	Big Blue	250	29	VXP	29	86.9	1.19	0.16
BR222	Big Blue	250	29	VXP	29	86.1	1.18	0.16
BR223	Weir	250	35	VXP	35	87	0.79	0.1
BR224	Weir	250	35	VXP	35	88.1	0.84	0.1
BR225	Weir	250	35	VXP	35	87.8	0.79	0.1
BR226	Weir	250	35	VXP	35	88.2	0.79	0.09

Table 47 VXP Mill Regrind Results

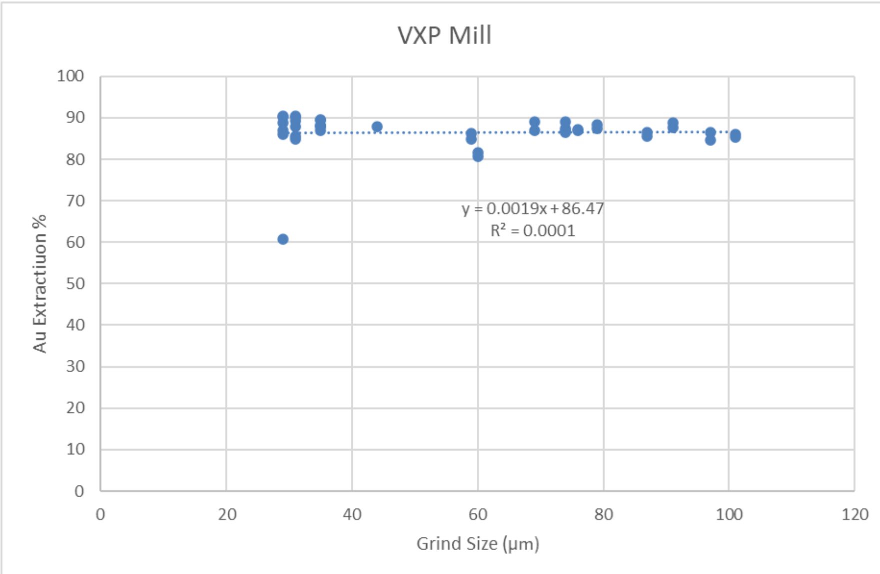


Figure 39 VXP Mill Regrind Results, FLS January 2018

Test	Comp	Primary Grind (µm)	Secondary Grind (µm)	Type	Measured Grind size (µm)	Extraction % Au	Head Grade g Au/t	Residue Grade g Au/t
BR95	1	250	63	RDI	49	89.6	0.66	0.07
BR96	1	250	63	RDI	49	89.9	0.68	0.07
BR97	1	250	63	RDI	49	89.5	0.66	0.07
BR98	4	250	63	RDI	36	88.3	0.7	0.08
BR99	4	250	63	RDI	36	89.7	0.73	0.08
BR100	4	250	63	RDI	36	92.1	0.79	0.06
BR101	1	250	63	RDI	39	90.5	0.65	0.06
BR102	1	250	63	RDI	39	90.9	0.64	0.06
BR103	1	250	63	RDI	39	90.4	0.64	0.06
BR107	1	250	15	RDI	18	89.4	0.68	0.04
BR108	1	250	15	RDI	18	93.8	0.66	0.04
BR109	1	250	15	RDI	18	94	0.69	0.04
BR110	4	250	15	RDI	15	92	0.6	0.05
BR111	4	250	15	RDI	15	91	0.61	0.06
BR112	4	250	15	RDI	15	90.9	0.6	0.06
BR135	Big Yellow	250	58	RDI	58	88.4	1.69	0.2
BR136	Big Yellow	250	58	RDI	58	88.5	1.75	0.2
BR137	Big Yellow	250	45	RDI	45	86.8	1.64	0.22
BR138	Big Yellow	250	45	RDI	45	87.5	1.67	0.21
BR139	Big Yellow	250	38	RDI	38	89.1	1.71	0.19

Test	Comp	Primary Grind (µm)	Secondary Grind (µm)	Type	Measured Grind size (µm)	Extraction % Au	Head Grade g Au/t	Residue Grade g Au/t
BR140	Big Yellow	250	38	RDI	38	88.3	1.91	0.22
BR141	Big Blue	250	58	RDI	58	88.1	1.42	0.17
BR142	Big Blue	250	58	RDI	58	86.3	1.24	0.17
BR145	Big Blue	250	38	RDI	38	88	1.2	0.14
BR146	Big Blue	250	38	RDI	38	89.2	1.18	0.13
BR149	Weir	250	45	RDI	45	87.4	0.85	0.11
BR150	Weir	250	45	RDI	45	87.8	0.87	0.11
BR151	Weir	250	38	RDI	38	90.6	0.98	0.09
BR152	Weir	250	38	RDI	38	90.9	0.99	0.09
BR155	Small Blue	250	38	RDI	38	95.5	2.13	0.1
BR156	Small Blue	250	38	RDI	38	95.7	2.02	0.09
BR159	Small Yellow	250	38	RDI	38	92.2	0.79	0.06
BR160	Small Yellow	250	38	RDI	38	91.5	0.82	0.07
BR163	Small Red	250	38	RDI	38	90.8	0.53	0.05
BR164	Small Red	250	38	RDI	38	91.5	0.57	0.05
BR176	Big Yellow	250	44	RDI	44	85.7	1.59	0.23
BR177	Big Yellow	250	44	RDI	44	88.1	1.68	0.2
BR178	Big Yellow	250	38	RDI	38	87.4	1.7	0.21
BR179	Big Yellow	250	38	RDI	38	89	1.65	0.18
BR180	Big Yellow	250	28	RDI	28	90.9	1.61	0.15
BR181	Big Yellow	250	28	RDI	28	90.6	1.64	0.15
BR183	Big Blue	250	44	RDI	44	87.9	1.37	0.17
BR184	Big Blue	250	38	RDI	38	88.8	1.36	0.15
BR185	Big Blue	250	38	RDI	38	88.1	1.26	0.15
BR186	Big Blue	250	31	RDI	31	86.5	1.24	0.17
BR187	Big Blue	250	31	RDI	31	89.3	1.28	0.14
BR188	Weir	250	43	RDI	43	86.8	0.88	0.12
BR189	Weir	250	43	RDI	43	88.3	0.87	0.1
BR190	Weir	250	38	RDI	38	85.9	0.86	0.12
BR191	Weir	250	38	RDI	39	89.1	0.88	0.1
BR192	Weir	250	30	RDI	30	91	0.87	0.08
BR193	Weir	250	30	RDI	30	89.2	0.89	0.1
BR201	Big Yellow	250	25	RDI	19	91.8	1.56	0.13

Table 48 RDI Lab Mill

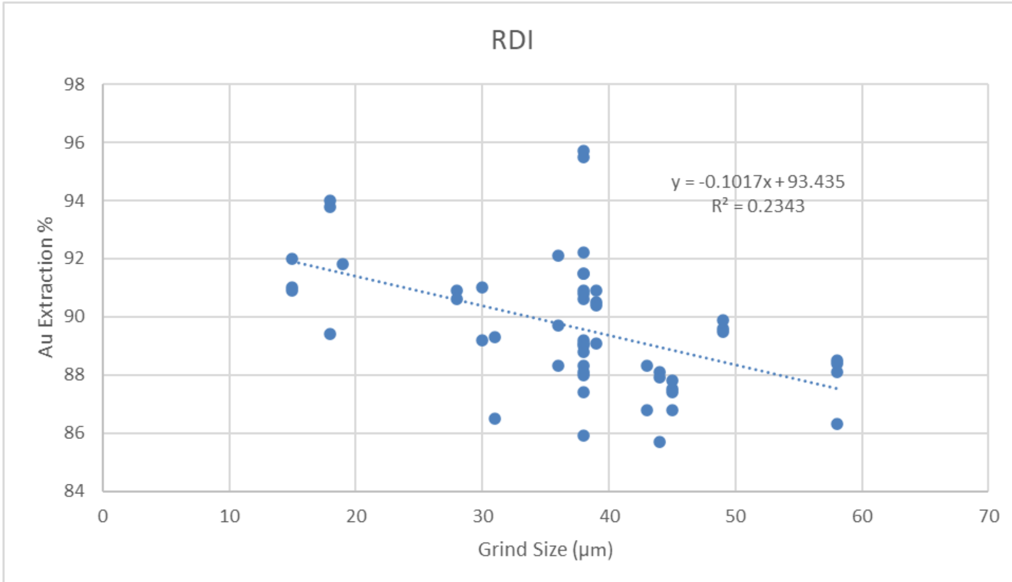


Figure 40 *RDi Regrind Mill Results, 2019*

Samples milled in both the IsaMill and the RDi lab rod mill show the expected increase in extraction with decreasing grind size. The FLS VXP data is somewhat skewed with a flat line across the grind size and recovery curve. The FLS sizing was done by Malvern laser sizer during the test work, which appears to have been affected by the particle shape/aspect ratio effects from silicate minerals in the sample tested. Resizing by sieve screening gave a much higher P_{80} size for the initial test runs BR113 to BR134.The secondary grinding specific energy was also assessed in this test work.

- Isamill signature plots indicated specific energy requirements between 26 and 34 kWh/t would be required for the size reduction from an F_{80} size of 250 µm to the required P_{80} size of 40 µm.
- Initial studies at FLS indicated a much lower specific energy of 16.7 kWh/t to 17.4 kWh/t to achieve the P_{80} of 40 µm.
- Corrected specific energy for the VXP tests was calculated to be 21.7 kWh/t to 24.7 kWh/t after the particle size measurements were repeated by screening.

Investigations into the FLS data indicated that the Malvern laser particle size analyzer did not provide a suitable method for analysis of the particle size distribution for the ground products for this application. Products were subsequently sieve screened to obtain a more accurate energy requirement that was comparable with the other grinding test work results. Further inconsistencies were found in the FLS test work, including discrepancies in flowrates, lab times per pass and recorded densities. A corrected weight basis was used with the recorded power, which impacted the specific power requirements, as noted above and reported in Table 49 for cumulative passes up to pass eight in the test procedure.

Sample	Pass (8) P_{80} µm	Cum SE (kWh/t)	Adjusted SE (kWh/t)
Weir	37	15.61	22.13
Big Yellow	33	15.95	21.77
Big Blue	52	16.20	24.72

Table 49 Corrected FLS SE basis

Due to the inconsistencies in the historical test work used to support the selection of the VXP mills in the previous studies, an alternative Vertimill option was investigated for this Technical Report Summary and ultimately was incorporated into the plant design for the secondary grinding stage. The Vertimill option follows well-proven technology for the secondary grinding application and the duty is well within the size reduction capabilities of the particular technology. The grinding media for the Vertimills will be hi-chrome steel grinding balls to minimize the effects of iron dissolution on cyanide consumption and leach kinetics.

10.4.11 Leach Testing

A series of batch leach tests were performed on the six composites to evaluate the effect of feed grade against gold extraction at various grind sizes. A total of 123 tests were conducted including duplicate tests. 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 transferring the ground pulp 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 50 ppm 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 concentration of 20 g/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.

Alternative milling types were also applied to the test procedure, utilizing an FLS VXP mill, IsaMill and a Lab steel ball mill.

The tests demonstrated that gold extraction is size dependent. Increased gold extraction was recorded with finer grind size, as noted in Section 10.4.10. An average gold extraction of 90% was achieved with feed gold grades greater than 0.6 g Au/t and particle sizes less than 53 µm. In general higher gold grades provided higher gold extractions.

Cyanide consumption averaged 0.647 kg NaCN/t based on the usage for samples that have been ground to less than 59 µm. Assuming an excess of 200 ppm of cyanide and 45% feed solids the estimated cyanide consumption was 0.891 kg/mt. Cyanide consumption was notably higher for the tests conducted with the RDi laboratory steel ball mill compared to the VXP and IsaMill, which used ceramic media. Historically, General Gold operation at Mt Todd used 1.37 kg NaCN/t.

Lime consumption in the batch testing was high with the average 4.76 kg/t being consumed. Expected lime consumption in the operating plant will be lower due to recirculated process water recovery from the leach feed thickener and tailings decant return water and was estimated to be 2.86 kg/t lime (60% reduction). It should be noted that historical operations data reviewed from the previous General Gold operations (oxide/sulfide) treatment of Batman pit ore reported an actual consumption 1.22 kg/t of lime.

Leach test results are summarized in Table 50.

Test	Comp	Mill Type	Target Grind Size (µm)	Ext % Au	Head g Au/t	Ext % Ag	Head g Ag/t	NaCN kg/mt	Lime kg/mt $Ca(OH)_2$
BR197	Big Blue	VXP	29	90.4	1.21	50.6	0.9	0.677	6.956
BR198	Big Blue	VXP	29	90.1	1.16	33.8	1.2	0.638	6.958
BR219	Big Blue	VXP	29	88.8	1.2	61.2	1.4	0.625	7.982
BR221	Big Blue	VXP	29	86.9	1.19	54.4	1.1	0.463	7.995
BR222	Big Blue	VXP	29	86.1	1.18	56.4	0.9	0.43	8.005
BR192	Weir	RDI	30	91	0.87	29.1	1	1.072	5.636
BR193	Weir	RDI	30	89.2	0.89	36.8	0.8	1.28	5.317
BR186	Big Blue	RDI	31	86.5	1.24	67	1.2	1.571	4.728
BR187	Big Blue	RDI	31	89.3	1.28	57.1	0.9	1.656	4.787
BR215	Big Yellow	VXP	31	87.8	1.75	51.8	1.7	0.63	7.959
BR216	Big Yellow	VXP	31	89.2	1.7	5.2	1.7	0.67	12.537
BR217	Big Yellow	VXP	31	85.6	1.68	53	0.7	0.46	12.334
BR218	Big Yellow	VXP	31	84.9	1.73	46.6	1.4	0.46	12.605
BR195	Big Yellow	VXP	31	90.4	1.69	48.9	1.6	0.875	6.978
BR196	Big Yellow	VXP	31	90.3	1.73	42.6	1.5	0.924	7.011
BR199	Weir	VXP	35	89.5	0.9	30.9	0.9	0.499	6.969
BR200	Weir	VXP	35	89.6	0.85	29	0.9	0.545	6.969
BR223	Weir	VXP	35	87	0.79	33.1	1.7	0.527	7.982
BR224	Weir	VXP	35	88.1	0.84	55	1.1	0.528	7.972
BR225	Weir	VXP	35	87.8	0.79	2.1	1.6	0.365	7.975
BR226	Weir	VXP	35	88.2	0.79	36.3	1	0.414	7.974
BR98	4	Vertimill	36	88.3	0.7	9	0.4	0.514	3.002
BR99	4	Vertimill	36	89.7	0.73	33.3	0.6	0.574	3.003
BR100	4	Vertimill	36	92.1	0.79	12.1	0.5	0.577	3.003
BR139	Big Yellow	IsaMIll (RDi)	38	89.1	1.71	52.4	1.4	1.172	5.188
BR140	Big Yellow	IsaMIll (RDi)	38	88.3	1.91	50.2	1.2	1.177	5.063
BR145	Big Blue	IsaMIll	38	88	1.2	40.7	0.8	0.975	5.488
BR146	Big Blue	IsaMIll	38	89.2	1.18	37.5	0.7	1.028	5.951
BR151	Weir	IsaMIll	38	90.6	0.98	29.9	0.8	0.975	5.168

Test	Comp	Mill Type	Target Grind Size (µm)	Ext % Au	Head g Au/t	Ext % Ag	Head g Ag/t	NaCN kg/mt	Lime kg/mt $Ca(OH)_2$
BR152	Weir	IsaMIll	38	90.9	0.99	12.7	0.6	1.022	4.928
BR155	Small Blue	IsaMIll	38	95.5	2.13	70.3	1.5	0.635	7.959
BR156	Small Blue	IsaMIll	38	95.7	2.02	60	1.8	0.632	4.341
BR159	Small Yellow	IsaMIll	38	92.2	0.79	13.2	2	0.875	6.157
BR160	Small Yellow	IsaMIll	38	91.5	0.82	15.1	2.1	0.875	5.103
BR163	Small Red	IsaMIll	38	90.8	0.53	54.7	2.1	1.215	4.276
BR164	Small Red	IsaMIll	38	91.5	0.57	52.9	2.1	1.213	3.746
BR178	Big Yellow	VXP	38	87.4	1.7	74.2	1.7	1.809	4.558
BR179	Big Yellow	VXP	38	89	1.65	63.3	1.5	1.751	5.968
BR184	Big Blue	VXP	38	88.8	1.36	48	0.8	1.371	4.376
BR185	Big Blue	VXP	38	88.1	1.26	59	0.9	1.566	4.576
BR190	Weir	VXP	38	85.9	0.86	44.4	1	1.514	4.728
BR191	Weir	VXP	39	89.1	0.88	47.1	1	1.514	4.728
BR101	1	IsaMill	39	90.5	0.65	62.1	1.1	0.573	3.849
BR102	1	IsaMill	39	90.9	0.64	57	0.9	0.457	3.279
BR103	1	IsaMill	39	90.4	0.64	62.9	1.1	0.515	3.281
BR188	Weir	VXP	43	86.8	0.88	44.1	1.2	1.425	3.924
BR189	Weir	VXP	43	88.3	0.87	63.5	1.1	1.358	4.312
BR176	Big Yellow	VXP	44	85.7	1.59	48.5	1.6	1.572	4.69
BR177	Big Yellow	VXP	44	88.1	1.68	62.7	1.3	1.513	5.698
BR182	Big Blue	VXP	44	87.8	1.22	46.8	1.2	1.32	4.002
BR183	Big Blue	VXP	44	87.9	1.37	66.3	1.4	1.175	3.846
BR137	Big Yellow	IsaMIll (RDi)	45	86.8	1.64	34.6	1.4	1.222	4.981
BR138	Big Yellow	IsaMIll (RDi)	45	87.5	1.67	40.7	1.5	1.218	5.997
BR149	Weir	IsaMIll	45	87.4	0.85	29.1	0.5	0.92	4.315
BR150	Weir	IsaMIll	45	87.8	0.87	12	0.6	0.923	4.473
BR95	1	RDI	49	89.6	0.66	12.3	0.5	0.636	3.004
BR96	1	RDI	49	89.9	0.68	42.5	0.7	0.635	3.003
BR97	1	RDI	49	89.5	0.66	21.3	0.5	0.636	3.003

Table 50 2019 Leach Batch Test Results

The mineralogy done in the historical test work indicated the presence of iron sulfide minerals which consume cyanide and dissolved oxygen thereby hindering the dissolution of gold with cyanide solution. However, only limited oxygen uptake test work was conducted by ALS in 2012 and did not show a high oxygen requirement on the sample tested. These tests were completed on coarser ground material (assumed P_{80} of 90 µm) and lower density of 40% solids (45% solids design). Dissolved oxygen levels were generally also not well reported in the historical leaching test work programs. The anticipated higher temperature at the Mt Todd site is likely to provide a lower dissolved oxygen level compared to the test work done under laboratory conditions at about 20ºC.

The monthly operating reports from the previous general gold operations also indicated a high oxygen demand resulting in lower gold extractions being achieved in the historical operations. General Gold installed 12 tpd of oxygen generation capacity via a PSA plant supplemented by a liquid oxygen system and used high shear oxygen reactor units to maximize oxygen transfer efficiency in the previous operation. Notes from the monthly operations indicated maximum oxygen demand of 17 tpd. This has been considered in the current design and flowsheet with addition of oxygen instead of air added to the pre-oxidation stages and to the leach and adsorption tanks.

10.4.12 Recovery Model

A correlation relating mill feed gold grade to gold extraction was derived from the historical leach test work for predicting gold recoveries in the plant according to the production schedule. The test work results from both the 2018 and 2019 leach tests were used to derive the gold recovery model, based on mill feed gold grade. Tests done on samples that were overground with reported grind sizes well below the design P_{80} size of 40 µm have been excluded from the assessment. The test work samples selected for the analysis gold head grade range was 0.6 g Au/t to 2.1 g Au/t with which to assess the effect of gold head on gold recovery. The results obtained are illustrated in Figure 41.

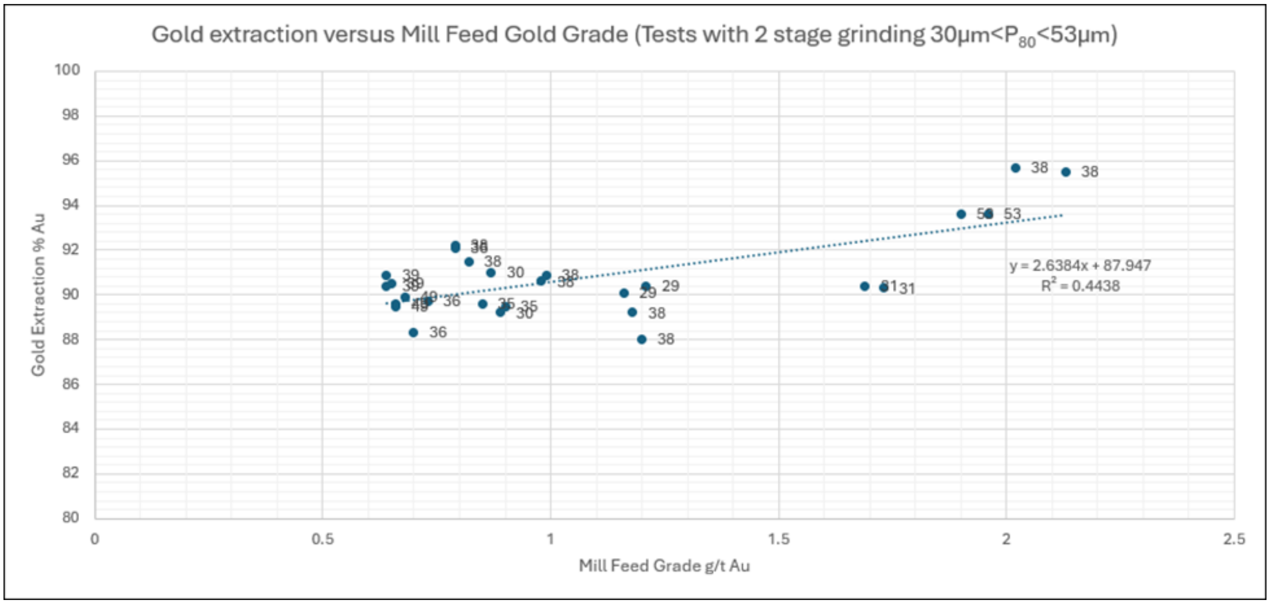


Figure 41 Gold Extraction vs Mill Feed Au Grade, GRES 2025

The resulting gold recovery equation used for predicting gold recoveries according to the production schedule is outlined in Table 51.

Mt Todd	Gold Equation
Head Grade 0.5 g/t Au – 2.0 g/t Au	Au Recovery = ((2.6384 x (Head Grade g/t Au) + 87.947) * 0.991)

Table 51 Gold Process Recovery Equation

A carbon adsorption efficiency of 99.1% for gold been factored into equation to account for expected solution losses. It should be noted the model does not use the ROM gold head grade, as the ore sorting prior to milling increases the mill feed gold grade due to the gold loss into the ore sorter reject stream.

The ROM gold head grade requires adjustment to account for 91.9% of mass recovered after ore sorting or 8.1% mass rejection from the ore sorter of coarse material. The ore sorter reject stream has been calculated to carry 1.69% Au in feed, resulting in a recovery of gold to the mill/leach feed section of 98.31%.

Example Calculation

From a ROM daily feed of 15,000 dry tonnes at a head grade of 0.97 g/t Au, the mill feed will be 13,786 dry tonnes. Gold contained in ROM feed is 14,550 g Au, based on a mass a recovery to the mill feed stream of 98.31% this equates to 14,304.1 g Au contained in the mill feed. The adjusted mill feed gold grade is now 14,304.1 g Au/13786 t = 1.037 g/t Au.

A gold recovery of 89.9% (net of solution loss) was calculated based on mill feed from the LOM average ROM gold head grade of 0.97 g/t Au. This equates to 88.3% based on the ROM feed.

10.4.13 Thickening Tests

Static and dynamic thickening tests were performed at RDi and Pocock Industrial Inc. in 2019, using leach residue samples. The samples used had related grind sizes between 37 µm to 44 µm in order to represent the plant design figure of P_{80} 40 µm. Results are tabulated in Table 52 and Table 53 for static and dynamic thickener results.

Material P_{80}	Flocculant		Unit Area (m^2/tpd)			Max Underflow Solids Conc (%)
	Type	Dose g/t	10% Feed Solids	15% Feed Solids	20% Feed Solids	
44 µm (325 mesh)	AN913 SH	25 g/t @ 0.1 g/l	0.243	0.321	0.382	57%
37 µm (400 mesh)	AN913 SH	25 g/t @ 0.1 g/l	0.249	0.330	0.480	55%

Table 52 Pocock Static Tests

Material P_{80}	Flocculant		Tested Feed Solids (%)	Design Basis Net Feed Loading (m^3/m^2h)	Predicted Overflow TSS Conc. (mg/l)	Predicted Underflow Solids Conc (%)
	Type	Dose g/t				
44 µm	AN913 SH	35 – 40 g/t	15.38	0.321	150-250	57%
37 µm	AN913 SH	40 – 45 g/t	14.71	0.330	150-263	55%

Table 53 Pocock Dynamic Thickener Testing

The test work on this sample has indicated the material can be thickened to achieve underflow densities ranging from 55% to 57% solids w/w at specific settling rates of between 0.32 $m^3/m^2/h$ and 0.33 $m^3/m^2/h$ with a reasonable overflow clarity for recycling as process water. The dynamic thickener test work completed by Pocock Industrial is reported as a specific slurry flow $m^3/m^2/h$ rather than as solids settling flux rate t/m^2h.

10.4.14 Pulp Rheology

Viscosity tests were performed in 2019 by RDi to examine the rheological behavior of the thickened pulps at grind sizes between 37 µm to 44 µm across a specific shear rate range. Results from the viscosity tests is summarized in Table 54.

Material	Solids Conc (%)	Coefficient of Rigidity (Pa)	Yield Strength Value (N/m^2) or (Pa)	Apparent Viscosity, (Pa.sec) @ the following Shear Rates:								
				5 s^{-1}	25 s^{-1}	50 s^{-1}	100 s^{-1}	200 s^{-1}	400 s^{-1}	600 s^{-1}	800 s^{-1}	1000 s^{-1}
Thickened 325 mesh	59.4	0.091	54.9	6.336	2.152	1.352	0.849	0.533	0.335	0.255	0.211	0.181
	58.4	0.059	36.3	4.392	1.464	0.912	0.568	0.354	0.220	0.167	0.137	0.118
	56.7	0.037	22.3	2.924	0.918	0.557	0.338	0.205	0.125	0.093	0.076	0.064
	53.8	0.020	11.5	1.799	0.502	0.290	0.167	0.096	0.056	0.040	0.032	0.027

Material	Solids Conc (%)	Coefficient of Rigidity (Pa)	Yield Strength Value (N/m^2) or (Pa)	Apparent Viscosity, (Pa.sec) @ the following Shear Rates:								
				5 s^{-1}	25 s^{-1}	50 s^{-1}	100 s^{-1}	200 s^{-1}	400 s^{-1}	600 s^{-1}	800 s^{-1}	1000 s^{-1}
Thickened 400 mesh	60.8	0.110	80.4	8.661	2.874	1.787	1.111	0.691	0.430	0.325	0.267	0.229
	59.0	0.061	55.4	5.829	1.929	1.198	0.744	0.462	0.287	0.217	0.178	0.153
	54.9	0.022	26.8	2.925	0.917	0.557	0.338	0.205	0.124	0.093	0.075	0.064
	49.2	0.014	10.2	1.382	0.401	0.235	0.138	0.081	0.047	0.035	0.028	0.023

Table 54 Rheology Results

The decreasing apparent viscosity with increasing shear rate or “shear thinning” behavior of the sample pulps examined is characteristic of pseudoplastic class of non-Newtonian fluids. It demonstrates the need to maintain a specific velocity gradient or shear rate in order to initiate and maintain flow. The yield strength values determined for each sample indicated increasing yield values with increasing density, classifying the pulps as Bingham plastics up to the yield value noted for flow, after which the pulps behave as near-Newtonian fluids. In summary, the rheology test work indicated that a leach feed solids density of 45% by weight would be appropriate for the leaching and adsorption circuit design.

10.4.15 Cyanide Destruction

The cyanide leach residue having a P_{80} of 45 µm and free cyanide of 200 ppm was subjected to cyanide destruction using the INCO air/SO_2 method. Leach test BR175 formed the basis for detoxification testing, using 1.22 liters of solution with 2.7 g of SMBS and 0.12 g of copper sulfate (2.19 g/l SMBS, 0.097 g/l Cu SO_4).

The cyanide speciation before and after destruction for the test is given in Table 55. The test results indicate that the air/SO_2 process will reduce the cyanide in the tailings to low range Weak Acid Dissociable (WAD) cyanide levels.

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

Table 55 Detoxification Results

The cyanide detoxification circuit for Mt Todd is based on the test work carried out previously. The air/SO_2 process will reduce the tailings slurry cyanide concentration down to nominally 20 ppm WAD cyanide prior to deposition in the tailings storage facility to minimize the cyanide levels and prevent the accumulation of copper in the process water, recycled from the TSF.

10.5 Qualified Person Opinion

The QP of this SK-1300 Technical Report Summary is of the opinion that the metallurgical testing data disclosed in this section is adequate for the purposes used in this Technical Report Summary. The following notes are relevant to the mineral processing design;

- The process design has been based on a grind target P_{80} size of 40 µm to maximize gold extraction. To reach this target four VTM4500 Metso (VertiMills) were included in the process design however no vendor specific "Jar" tests recommended for sizing the Vertimills have been conducted. The specific energy requirements for the secondary grinding stage were therefore calculated based on vendor's experience using the design BWi derived from the test work and the estimated circuit efficiency factor for the secondary grinding application. The specific energy used for design including 10% contingency was 17.9 kWh/t for the secondary grinding based on grinding from an F_{80} size of 250 µm to a P_{80} size of 40 µm. Metso "Jar" tests are recommended to confirm secondary grinding energy requirements. The total specific energy available from the four VTM4500 selected is approximately 19.4 kWh/t.
- Only limited oxygen uptake test work was conducted in the historical test work and the tests were completed on coarser ground material and lower density of 40% solids compared to 45% solids used in the design. The mineralogical test work which indicated the presence of iron sulfide minerals and historical operating data from general gold operations indicated a high oxygen demand in contrast to the previous laboratory test work. The design has incorporated oxygen instead of air to be added in to pre-conditioning/oxidation tanks and the leach and adsorption circuit. Additional oxygen uptake testing is recommended on fresh samples to confirm oxygen requirements.
- Cyanide soluble copper minerals were reported to occur in high enough concentration to impact on the historical processing operations. The test work has indicated copper is present mainly as chalcopyrite in the ore scheduled to be treated which is the least cyanide soluble of the copper minerals and dissolves to a very limited extent under typical conditions applied for cyanide leaching of gold ores. The mine inventory for the Project is almost entirely made up of sulfide ore and comprises very little oxide and transition ores, which typically contained higher cyanide soluble copper minerals. Nevertheless, to mitigate the risk of copper impacting on the future processing operations, a cold cyanide wash prior to elution of gold has been allowed for in the proposed flowsheet to remove copper from the loaded carbon. Ore blending will also need to be applied to the production plan where possible to avoid high spikes in plant feed.

11. MINERAL RESOURCE ESTIMATES

The Project contains several 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 Quigleys deposit. The Mineral Resource estimates have been classified in accordance with the SEC's Regulation S-K subpart 229.1300 mining disclosure rules.

This section provides updated material, scientific, and technical information based on additional data obtained from drilling conducted from 2020 to 2024, as well as updated test work, geotechnical evaluation, and updated economic parameters.

11.1 Introduction

The following sections summarize the process, procedures, and results of the Mineral Resource estimates for the Batman and the Quigleys deposits (excludes the Heap Leach Pad, refer note (9) in Table 56).

The Mineral Resource estimates for the Batman and Quigleys deposits and the Heap Leach Pad Mineral Resource are classified in accordance with subpart 229.1300 of Regulation S-K. The Mineral Resources for the Batman and Quigleys deposits have been reported within a shell generated using the MicroMine® pit optimizer module. Mineral Resources within such a shell are not Mineral Reserves and do not demonstrate economic viability.

It is the opinion of the QP for this section that the reported Mineral Resource classifications comply with current S-K 1300 definitions for each mineral class.

Mineral Resource estimations and 3-D visualization were completed in MicroMine® software. Additional statistical analysis was performed in Access® and Excel®.

Table 56 shows the summary of the 2025 Mineral Resource estimates for Batman, Heap Leach Pad and Quigleys deposits, Figure 42 shows the relative locations of the three Mineral 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.

Cautionary statements regarding Mineral Resource estimates:

Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources will be converted into Mineral Reserves. Inferred Mineral Resources are that part of a Mineral Resource for which quantity and grade, or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

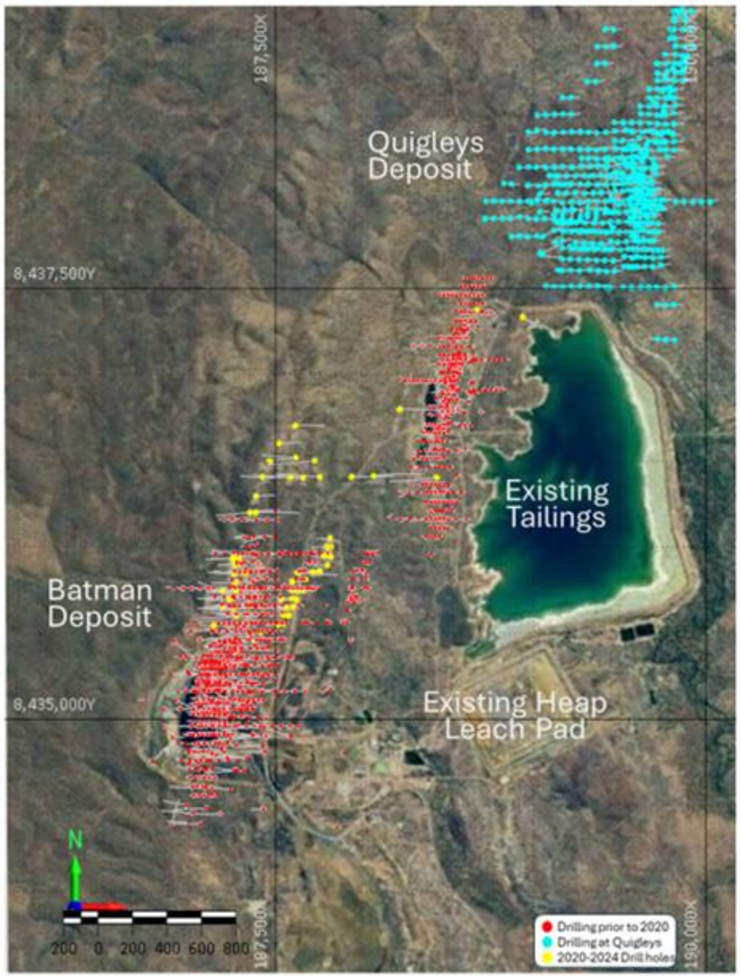


Figure 42 Drillhole location map Batman and Quigleys deposits and Heap Leach Pad (Yellow Collars were added for the 2025 Mineral Resource Update)

	Batman Deposit			Heap Leach Pad			Quigleys Deposit		
	Tonnes (000s)	Grade (g Au/t)	Contained Ounces (000s)	Tonnes (000s)	Grade (g Au/t)	Contained Ounces (000s)	Tonnes (000s)	Grade (g Au/t)	Contained Ounces (000s)
Measured (M)	47,143	0.61	930	-	-	-	3,702	1.13	134
Indicated (I)	110,644	0.72	2,568	-	-	-	6,965	1.34	299
Measured and Indicated	**157,787**	**0.69**	**3,498**	**-**	**-**	**-**	**10,667**	**1.26**	**433**
Inferred (F)	54,338	0.78	1,369	-	-	-	2,761	0.71	63

Notes:

(1) Measured and Indicated Mineral Resources exclude Proven and Probable Mineral Reserves.

(2) Batman and Quigleys Mineral Resources are quoted at a 0.4 g Au/t cut-off grade. Heap Leach Pad Mineral Resources are the average grade of the heap, no cut-off applied.

(3) Batman and Quigleys: Mineral Resources constrained within a USD1,950/oz gold pit shell. Pit parameters: Mining Cost USD3.00/tonne, Processing Cost USD17.50/tonne processed, General and Administrative Cost USD1.50/tonne processed, Au Recovery 89.7%.

(4) Kira Johnson MMSA of Tetra Tech is the QP responsible for the Statement of Mineral Resources for the Batman deposit, Quigleys deposit, and Heap Leach Pad.

(5) The effective date of the Batman , Quigleys and Heap Leach Pad Mineral Resource estimates is July 25th, 2025.

(6) Mineral Resources that are not Mineral Reserves have no demonstrated economic viability and do not meet all relevant modifying factors.

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

(8) The Mineral Resources were estimated using in accordance with subpart 229.1300 of Regulation S-K.

(9) The entirety of the Heap Leach Pad Mineral Resource is converted to Mineral Reserves in this Technical Report Summary, therefore, a Mineral Resource exclusive of Mineral Reserves is not reported.

(10) "-" indicates no reported value.

Table 56 Summary of the 2025 Mineral Resource Estimates for Batman and Quigleys deposits

11.2 Batman Deposit Mineral Resource Model

The Batman deposit has been the subject of multiple investigations and Mineral Resource estimations throughout the years, with Tetra Tech being involved since 2008. This section details the estimation procedures and input parameters for the 2025 update.

11.2.1 Input Data

This Mineral Resource update included an additional 59 holes from the previous update, as detailed in Table 14. The additional drilling explored the northern portion of the known Batman deposit, and an area extending northeast from the historical pit. The drilling totals 15,424 meters of core drilling, which includes 12,182 total assays. Table 57 shows the breakdown of the drilling included in the Mineral Resource estimation for this Technical Report Summary.

Year	Number of Drill holes	Total length (m)	Drillhole Type
Pre-2007	730	98,134	Mixed
2007	25	9,983	Diamond
2008	16	8,938	Diamond
2010	12	6,864	Diamond
2011	15	7,063	Diamond
2012	27	17,439	Diamond
2015	5	3,185	Diamond
2020-2022	26	8,887	Diamond
2024	34	6,776	Diamond

Table 57 Drill Holes Included in the Batman Mineral Resource Estimation

11.2.2 Grade Capping

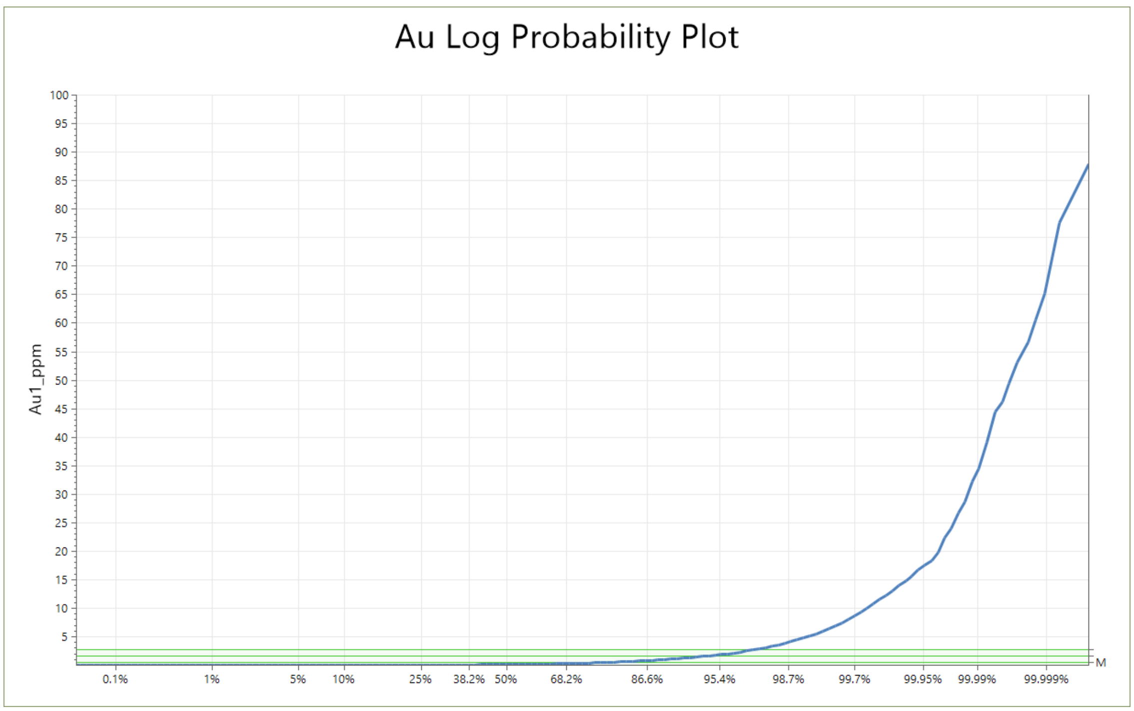


Figure 43 Log Probability Plot for Batman Deposit

Figure 43 shows the log probability plot for the Batman deposit. A 50 g Au/t cap was selected upon review of the log probability plot for the composited gold grades for the Batman deposit. Capped values were reviewed in 3D to ensure the samples were not clustered and represent true outliers. All gold composites were capped at this value.

11.2.3 Compositing

Down hole assays for the drill holes were composited to a length of 4 meters. Compositing was preformed after grade capping.

11.2.4 Wireframe Modeling 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. Most 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.

11.2.4.1 Mineralization Modeling

For the 2025 update, Tetra Tech investigated the further simplification of wireframes used for estimation. This was done to determine if the wedge-shaped gaps between the previous versions of the wireframes could be included in the estimation parameters for a main mineralized zone. The goal was also to include a mineralized zone methodology for the mineralization to the north and northeast of the historical pit area, named the South Cross Lode zone, which appears to contain more thin veins and stockwork veins.

For this exercise, a model was first run with no wireframe constraints, utilizing the search parameters defined by variography for the deposit. As expected, the model without wireframe constraints showed signs of grade smearing around the peripheral of the mineralized zone. Using the unconstrained model as a guide, incremental grade shell wireframes were then created for the deposit. The incremental grade shells were created for the 0.1 g Au/t, 0.2 g Au/t, 0.3 g Au/t, and 0.4 g Au/t. A block model estimation was created for each of these grade shells, using the grade shell to constrain the blocks and the composites for the estimation. Statistical analysis showed the shell most closely following the inputs was the 0.1 g Au/t model.

Based on this, a mineralized domain was created for the Batman deposit at a 0.1 g Au/t cut-off to define the mineralized envelope for estimation, shown in Figure 44 in plan view, and cross section in Figure 45 and Figure 46.

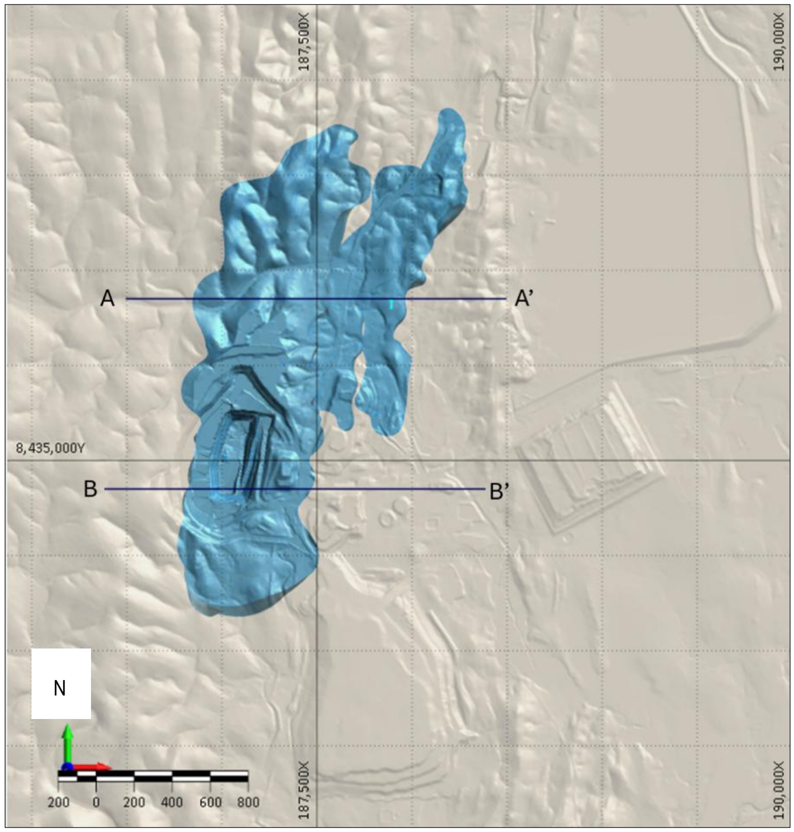


Figure 44 Plan View of the Batman Mineral Resource 2025 Wireframe Model and Section Lines, Tetra Tech 2025

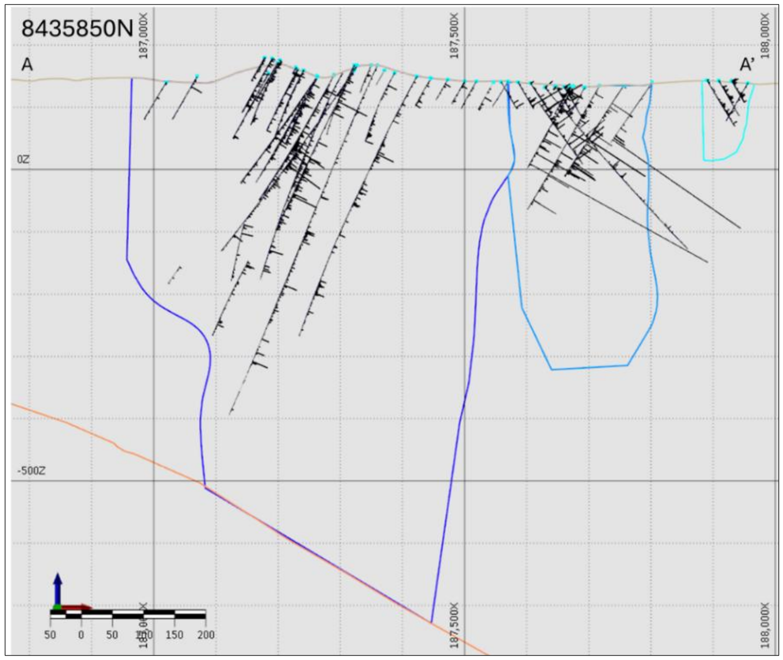


Figure 45 Cross Section A-A' of the Batman Deposit Wireframes, Looking North. Granite Surface is Shown in Orange, Tetra Tech 2025

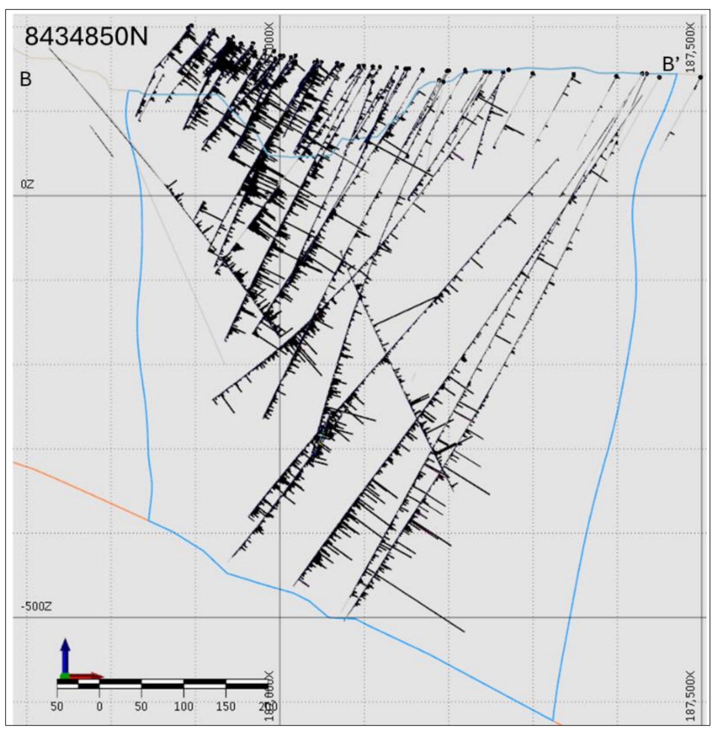


Figure 46 Cross Section B-B' of the Batman deposit Wireframes, Looking North. Granite Surface is Shown in Orange, Tetra Tech 2025

11.2.4.2 Granite Surface

For the previous update, deeper step-out drilling by Vista indicated 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 data was used to 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. The granite surface was extended to cover the larger extent of the 2025 block model.

11.2.4.3 Oxidation Surfaces

Oxidation surfaces were created using the data in the drill hole database. For the 2025 update, the previously created wireframe surfaces were examined and updated as necessary. The surfaces were also extended to cover the full extent of the 2025 model. The modeled surfaces included the top of the sulfides and the top of the transition zones. The block model was coded with OX for oxides, TR for transition material, and FR for sulfide material.

11.2.5 Specific Gravity Data

For the 2025 Mineral Resource model update of the Batman deposit, specific gravity data was estimated into the block model using ordinary kriging methods, where specific gravity data was available. Geostatistics determined the search distance to be 250 meters.

Models of the oxide, transition, and sulfide surfaces were created from the drill hole data. These surfaces were used to restrict the blocks and composites used for the estimation of the specific gravity data. Areas outside of the 250-meter search radius were assigned a bulk specific gravity based on the oxidation state assigned, based on test work from the Vista specific gravity testing on their exploration programs. These specific gravity tests were carried out on a 10 to 15 cm piece of core from a 1-m sample. Based on this work, the bulk densities applied to the Mineral Resource model outside of the estimation search radius are presented in Table 58.

Oxidation	Mean
Oxide	2.45
Transitional	2.64
Primary (Fresh)	2.77

Table 58 Summary of Batman Bulk Specific Gravity Data by Oxidation State for Vista Drilling

11.2.6 Batman Deposit Estimation Methods and Parameters

The 2025 Mineral Resources were estimated for the Batman deposit using a 3D block model in MicroMine software. The block model is a non-rotated model, with a block size of 12x12x6 meters. Block estimations were completed using ordinary kriging. The limits of the block model were extended to the north and to the east for the 2025 block model estimation. Table 59 details the physical limits of the Batman deposit block model.

Direction (dir)	Minimum (m) MGA94 z53	Maximum (m) MGA94 z53	Block Size	#Blocks
y-dir	8,433,807 mE	8,437,275 mE	12 m	227
x-dir	186,005 mN	188,717 mN	12 m	290
z-dir	-991 m	224 m	6 m	204

Table 59 2025 Block Model Physical Parameters – Batman Deposit

11.2.6.1 Variography and Search

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.

Omnidirectional explored the best continuity of mineralization given the combination of control by bedding and sulfide veining. Figure 47 shows the omnidirectional variogram for the Batman deposit.

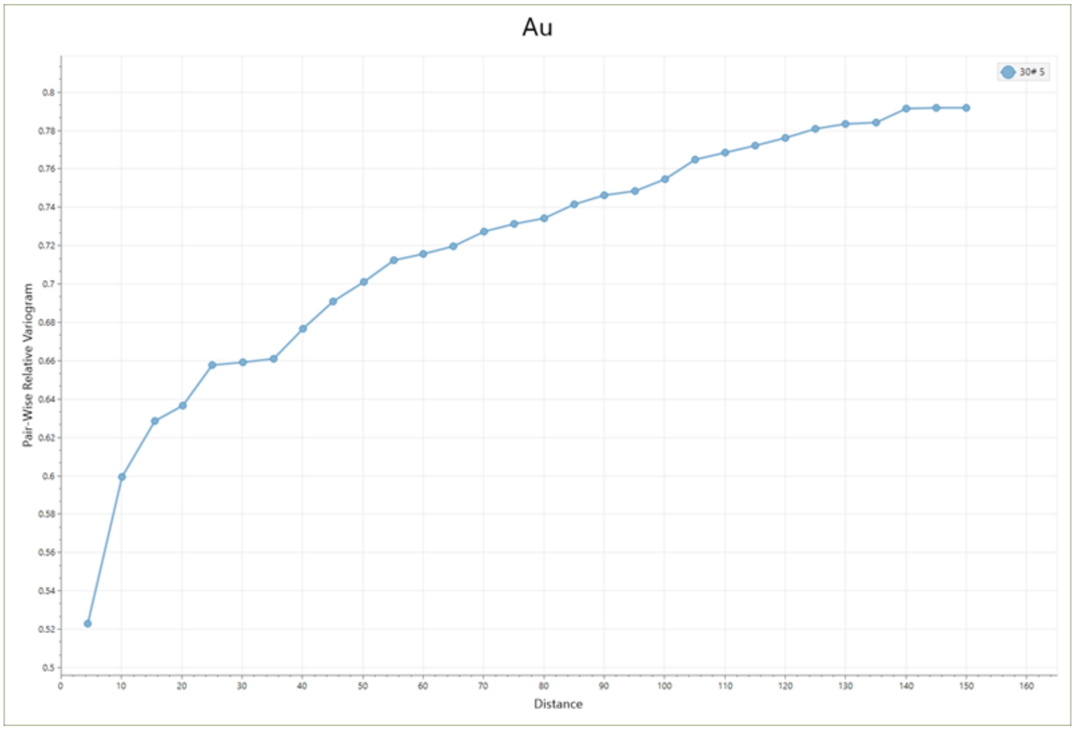


Figure 47 Variogram for the Batman Deposit

The Mineral Resource estimation was performed in two ordinary kriging passes within the modeled mineralized zone. Pass 1 used a search radius of 150 meters and Pass 2 used a search radius of 50 meters. The pass search parameters are listed in Table 60.

Pass	Search Range (m)	No. of Sectors/Max Points per DH	Search Anisotropy	Min Points
Pass 1	150	4/2	(1.0:0.7:0.4) [110:80:0]	2
Pass 2	50	4/3	(1.0:0.7:0.4) [110:80:0]	4

Table 60 Batman Mineral Resource Estimation Criteria

The wireframed mineralized domain was used to assign a code to the blocks and the composites within the shape. This code was used to constrain the block model estimation.

Blocks and composites above the topography were considered in the estimation and excluded from the Mineral Resource as mined out material. Blocks below the modeled granite surface were excluded from the reported Mineral Resources.

Figure 48 shows a plan view of the Batman resource block model and the resource shell with section lines for the cross sections below.

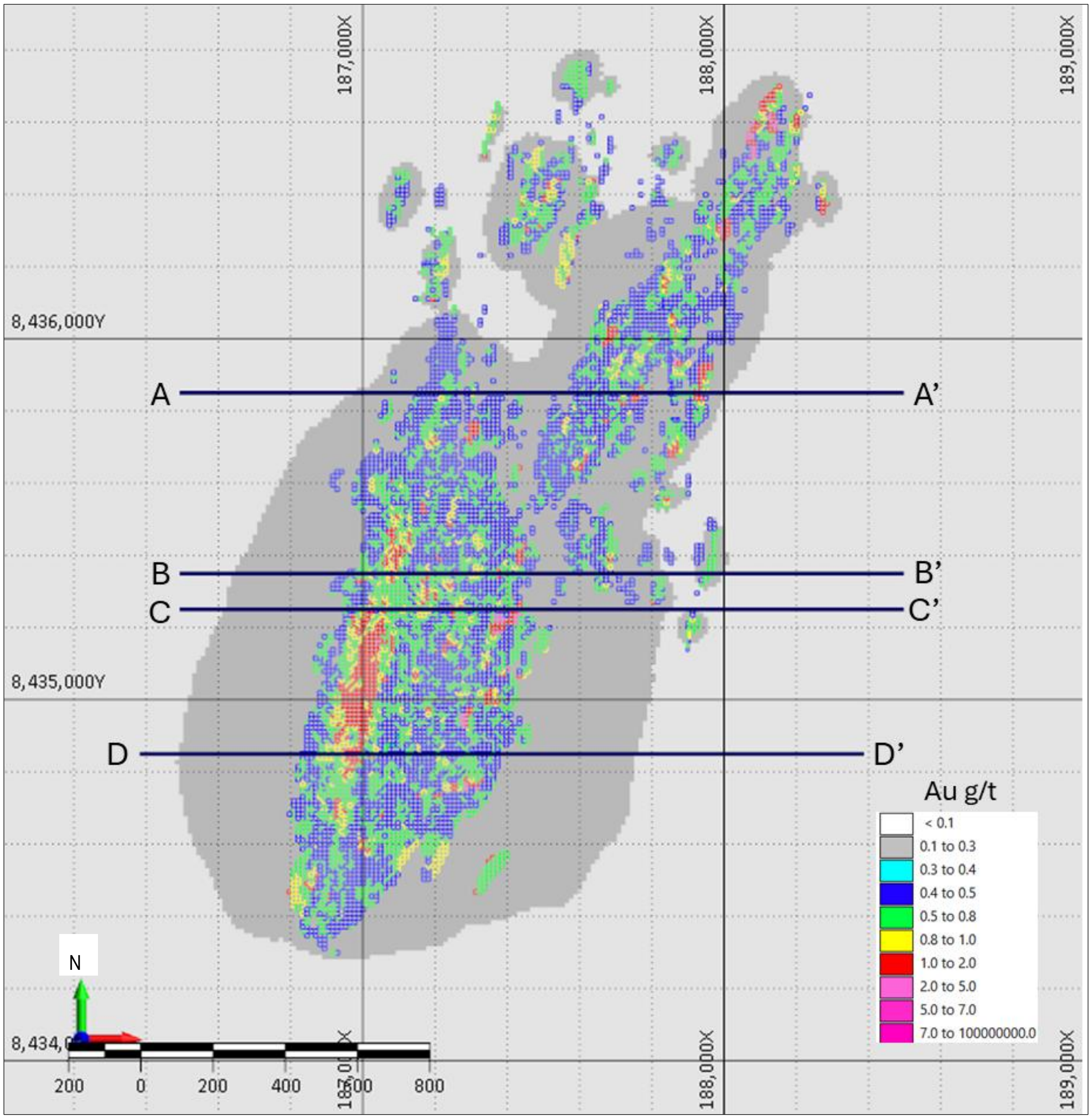


Figure 48 Plan View of the Batman Mineral Resource Model with Section Lines, Tetra Tech 2025

Figure 49, Figure 50, Figure 51, Figure 52 show the estimated blocks in the updated Mineral Resource block model colored by gold grade. The sections also show the current topography, drill holes, gold assays, and the 2025 Mineral Resource pit shell, created with a gold price of USD1,950/oz.

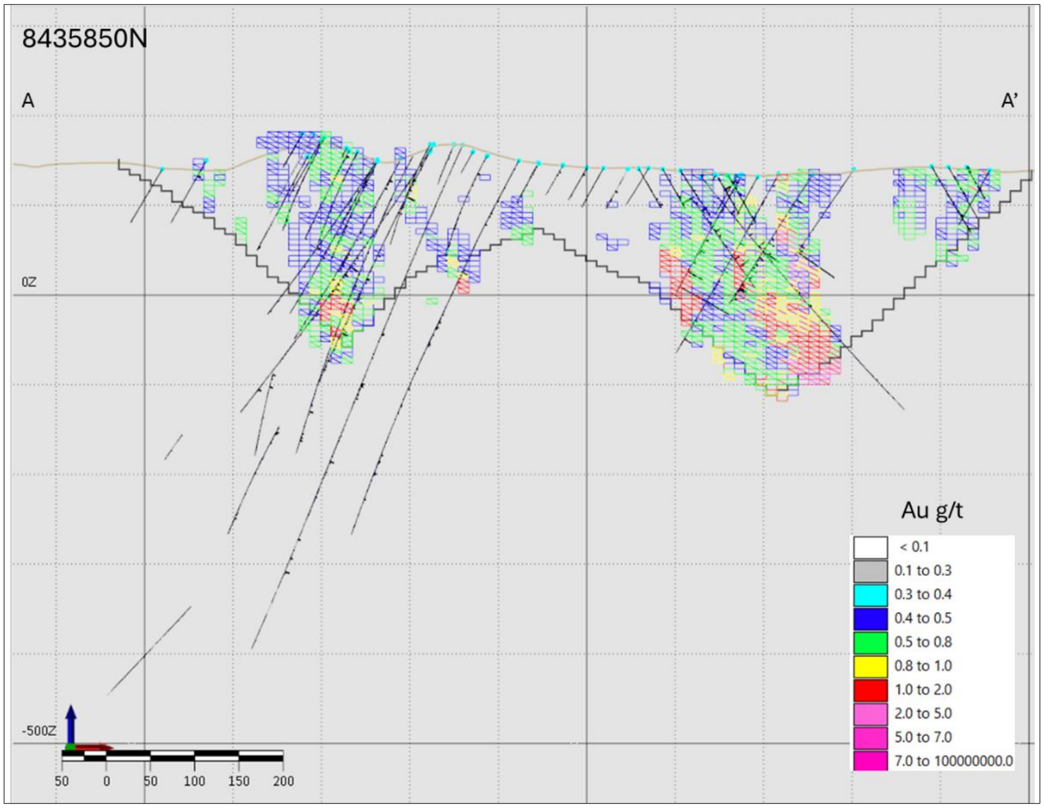


Figure 49 Cross Section A-A' of the Batman Deposit, Looking North, Tetra Tech 2025

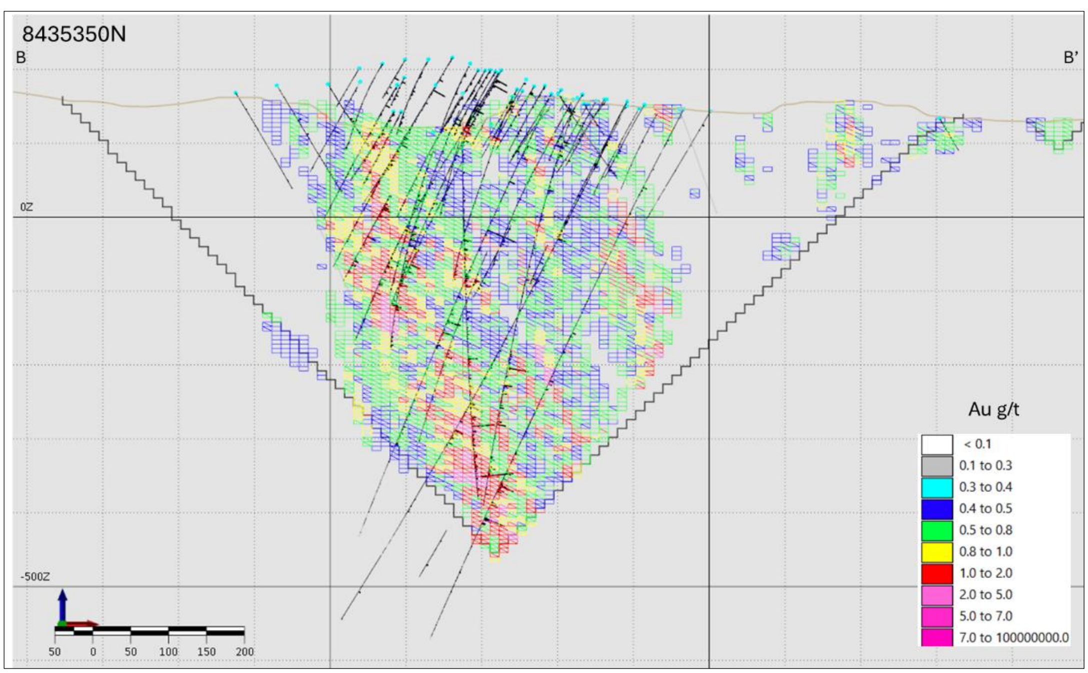


Figure 50 Cross Section B-B' of the Batman Deposit, Looking North, Tetra Tech 2025

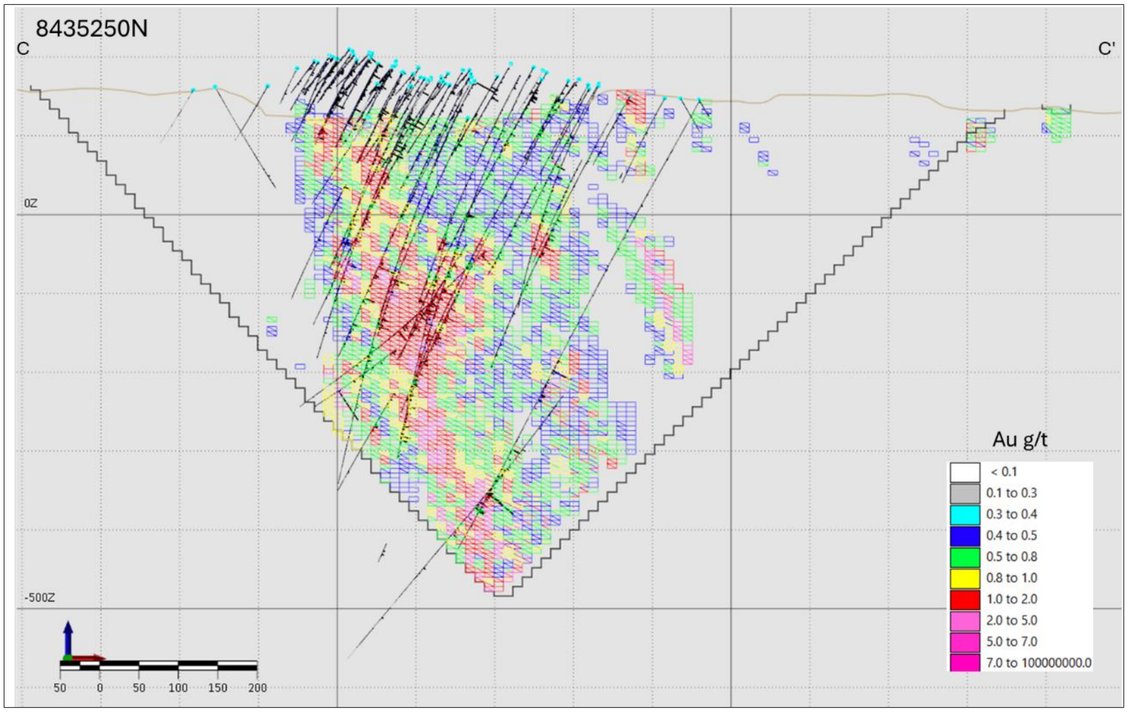


Figure 51 Cross Section C-C' of the Batman Deposit, Looking North, Tetra Tech 2025

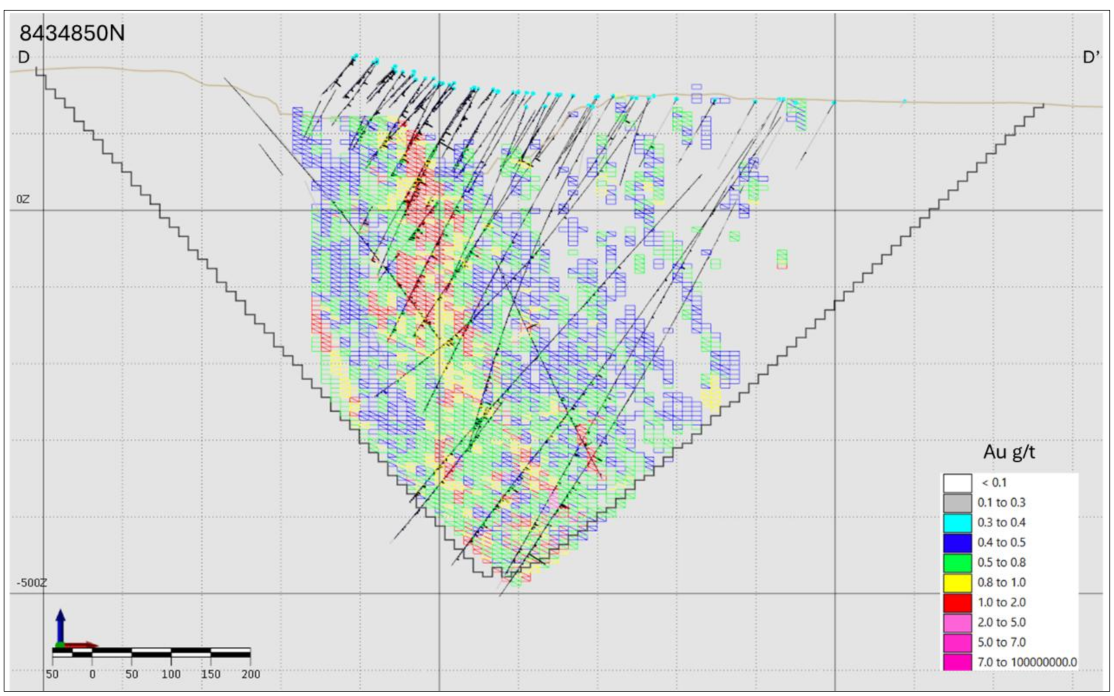


Figure 52 Cross Section D-D' of the Batman Deposit, Looking North, Tetra Tech 2025

11.2.6.2 Mineral Resource Classification

Mineral Resource classification of the estimated block model was assessed by the estimation pass, number of points, and kriging variance. Table 61 shows the Mineral Resource classification criteria for the Batman Mineral Resource model.

Category	Pass	Average Distance	Kriging Variance
Measured	Pass 1	<30 meters	<0.5
Indicated	Pass 1 or Pass 2	<150 meters	>0.5
Inferred	Pass 2	>150 meters	>0.5

Table 61 Batman Mineral Resource classification criteria

11.2.6.3 Cut-off Grade

The Mineral Resource cut-off grade for the Batman deposit was calculated to be 0.4 g Au/t. This cutoff grade was calculated using economic inputs provided by Vista, Vista and the consultants responsible for this Technical Report Summary, see Table 62. Upon comprehensive review, the Geology and Mineral Resource Qualified QP confirmed that the techno-economic inputs utilized for the Mineral Resource Estimate are appropriate. The applied assumptions represent a conservative set of inputs, thereby satisfying the requirements and professional judgment of the QP.

11.2.7 Batman Statement of Mineral Resources

Mineral Resources have been constrained by a pit shell, created in the MicroMine pit optimizer module. Table 62 shows the parameters used to create the Mineral Resource shell for the Batman deposit.

The USD1,950/Au oz was selected based on benchmarking of global projects reported during 2024 by Vista's consultants and provided to Tetra Tech. The number selected by Vista based on this Technical Report Summary is lower than the three-year trailing average and was considered to be conservative and was accepted for use in reporting this Mineral Resource.

Item	Input
Gold Price	USD1,950 per troy ounce
Gold Recovery	89.7%
Overall Mining Cost	USD3.00 per tonne
Processing Cost	USD17.50 per tonne processed
General and Administrative Cost	USD1.50 per tonne processed
Slopes	Overall angles determined by geotechnical parameters

Table 62 Pit shell parameters

Table 63 shows the summary of the Mineral Resource Estimate for the Batman deposit.

Classification	Tonnes (000s)	Grade (g Au/t)	Contained Gold Ounces (000s)
Measured (M)	47,143	0.61	930
Indicated (I)	110,644	0.72	2,568
Measured and Indicated	**157,787**	**0.69**	**3,498**
Inferred (F)	54,338	0.78	1,369

(1) Measured and Indicated Mineral Resources exclude Proven and Probable Mineral Reserves.
(2) Batman Mineral Resources are quoted at a 0.4 g Au/t cut-off grade.
(3) The Point of Reference for the Batman Mineral Resource estimates are in-situ at the property.
(4) Batman Mineral Resources constrained within a USD$1,950/oz gold pit shell. Pit parameters: Mining Cost USD3.00/tonne, Processing Cost USD17.50/tonne processed, General and Administrative Cost USD1.50/tonne processed, Au Recovery 89.7%.
(5) Kira Johnson MMSA of Tetra Tech is the QP responsible for the Statement of Mineral Resources for the Batman Deposit.
(6) The effective date of the Batman Mineral Resource estimates is July 25th, 2025.
(7) Differences in the table due to rounding are not considered material.
(8) The Mineral Resources were estimated in accordance with subpart 229.1300 of Regulation S-K.

Table 63 Summary of the Mineral Resource Estimate for the Batman deposit reported at a 0.4 g Au/t cutoff grade

Mineral Resources that are not Mineral Reserves have no demonstrated economic viability and do not meet all relevant modifying factors

Figure 53 shows the grade-tonnage for Measured and Indicated Mineral Resources for the Batman deposit.

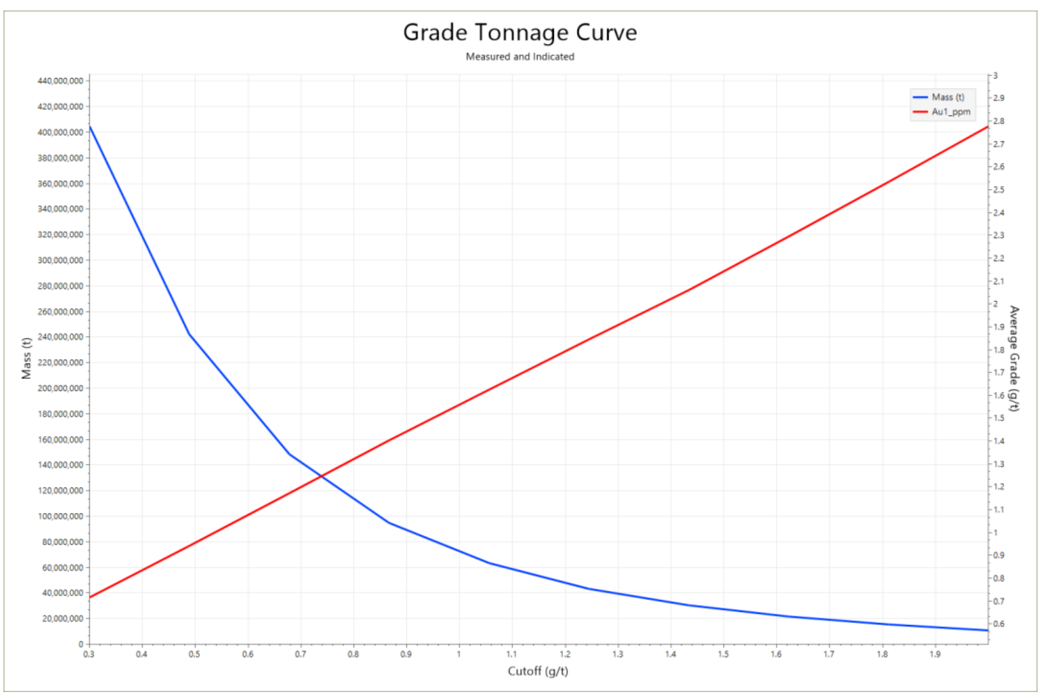


Figure 53 Grade Tonnage Curve of Measured and Indicated Mineral Resources for the Batman Deposit

Figure 54 graphically shows the grade-tonnage for the Inferred classified Batman deposit Mineral Resources.

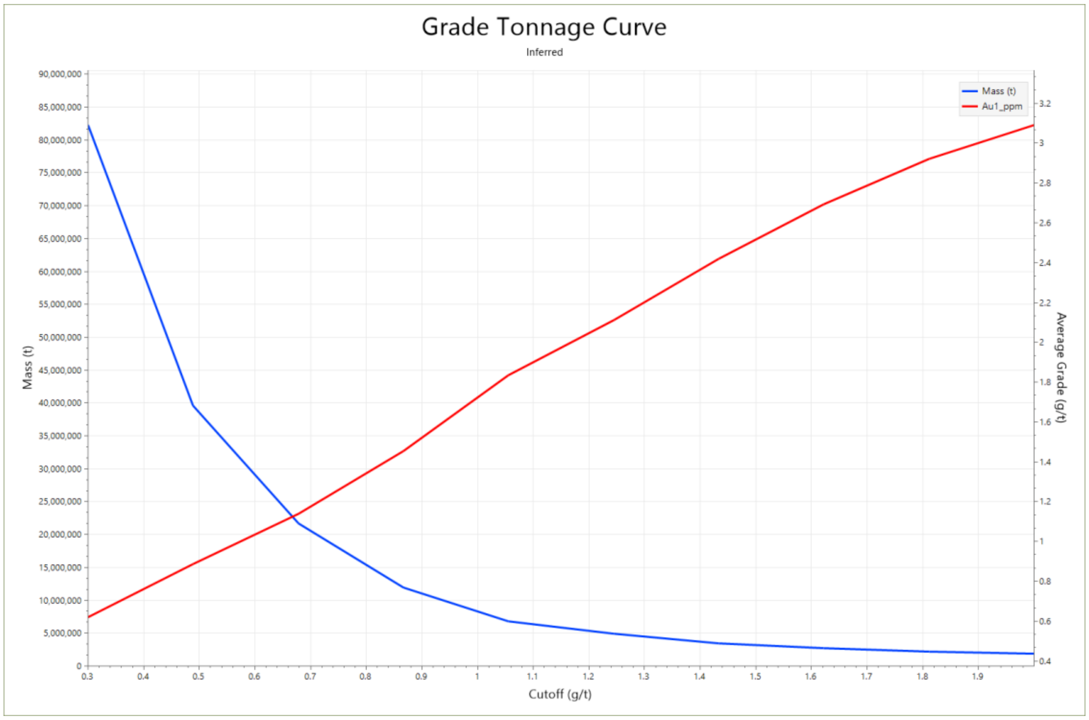


Figure 54 Grade tonnage curve of Inferred Mineral Resources for the Batman deposit

11.2.8 Multi-Element Estimation

The Batman Mineral Resource model was primarily estimated for gold. The database contains additional information for other elements. The multi-element assay data was composited to 4 meters. After compositing, the additional elements were estimated using ordinary kriging methods with a 1-pass, 250-meter search. These assays included iron, sulfur, lead, and copper. Drill hole coverage for these elements was not the same amount of drill holes as the Au assays. The QA/QC through the years has primarily focused on gold assays. Due to these factors, the multi-element estimation should be considered as Inferred in nature, excluded of economic value for the Project.

11.2.9 Batman Mineral Resource Model Verification

Several methods were used to validate the block model to determine the adequacy of the Batman deposit Mineral Resource.

The use of visual inspection of the kriged blocks models in section along with the assay and composite values were conducted by multiple professionals. Model comparisons were run by inverse distance and near neighbor methods as a verification. The inverse distance squared model was within 1% for grade, 1.4% for tonnage, and 0.2% for Au content in ounces.

Swath plots were also analyzed for the block estimation process. Figure 55 shows a north south swath plot of the composites, the ordinary kriging estimation, the inverse distance estimation, and the nearest neighbor estimation.

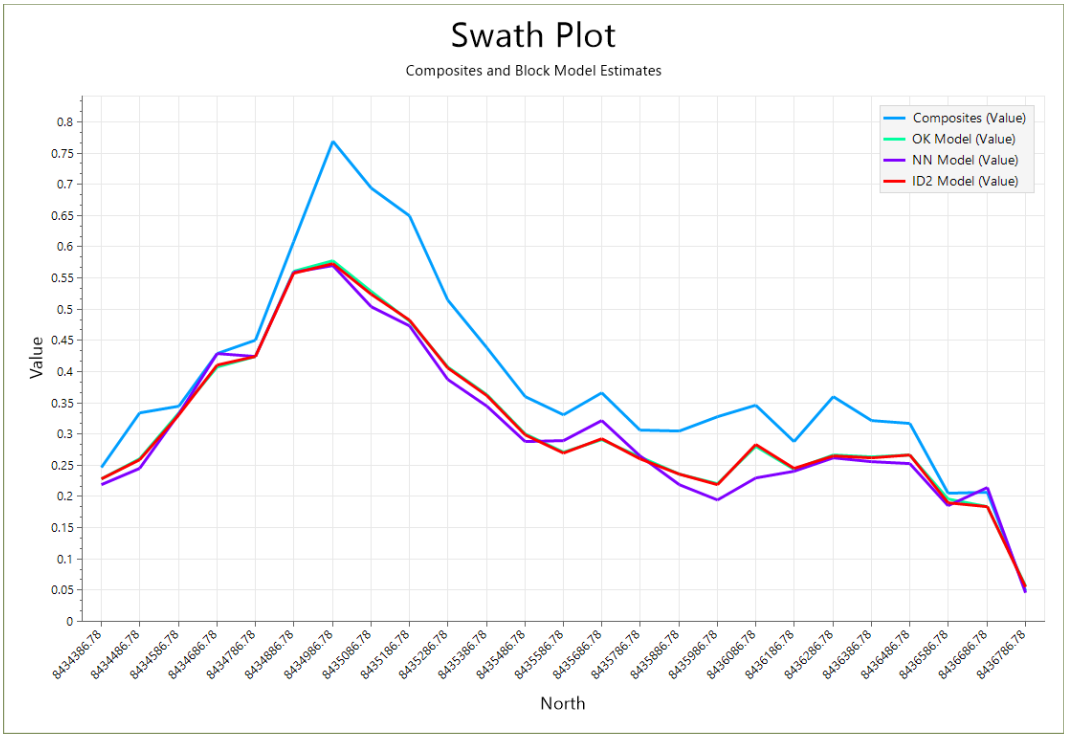


Figure 55 North-south swath plot for the composites and estimated block models for the Batman deposit

11.3 Quigleys Mineral Resource Model

The location of the Quigleys Deposit is approximately 3.5 km northeast of the Batman Deposit and has been the site of previous mining operations. The deposit is not as deep as the Batman Deposit; it reaches a maximum depth of approximately 200 m based on current information. The deposit has been sampled with 57,600 m of drilling by 631 drill holes, with the majority reaching a depth of 100 m at a 60° dip, oriented 83° azimuth.

11.3.1 Input Data

Table 64 summarizes the 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

Table 64 Summary of Quigleys deposit exploration database

11.3.2 Grade Capping

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, shown in Figure 56, shows that there is a distinct break in the distribution at 12 g Au/t. All gold composites were capped at this value.

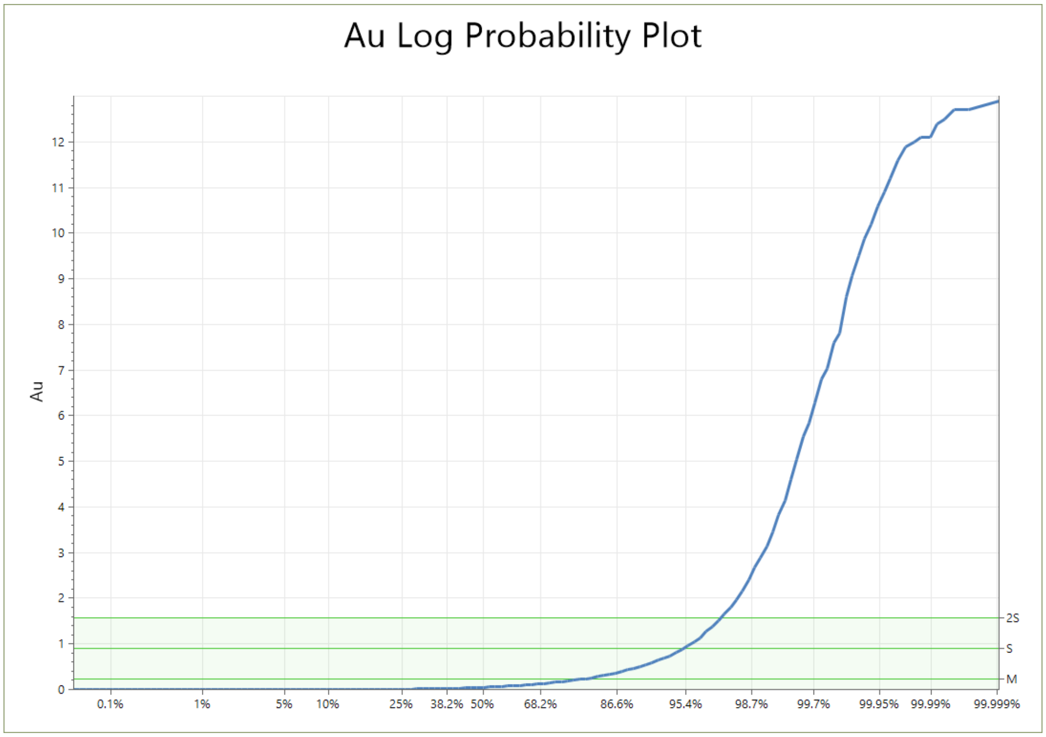


Figure 56 Log probability for Quigleys deposit

11.3.3 Compositing

Down hole assays for the drill holes were composited to a length of 2 meters. Compositing was performed after grade capping.

11.3.4 Wireframe Modeling of the Quigleys Deposit

The Quigleys Deposit is located approximately 3.5 km northeast of the Batman Deposit. The deposit has been sampled with 57,600 m of drilling by 631 drill holes, with the majority reaching a depth of 100 m 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 0.

Figure 57 shows wireframes and rock codes used for the Quigleys estimation.

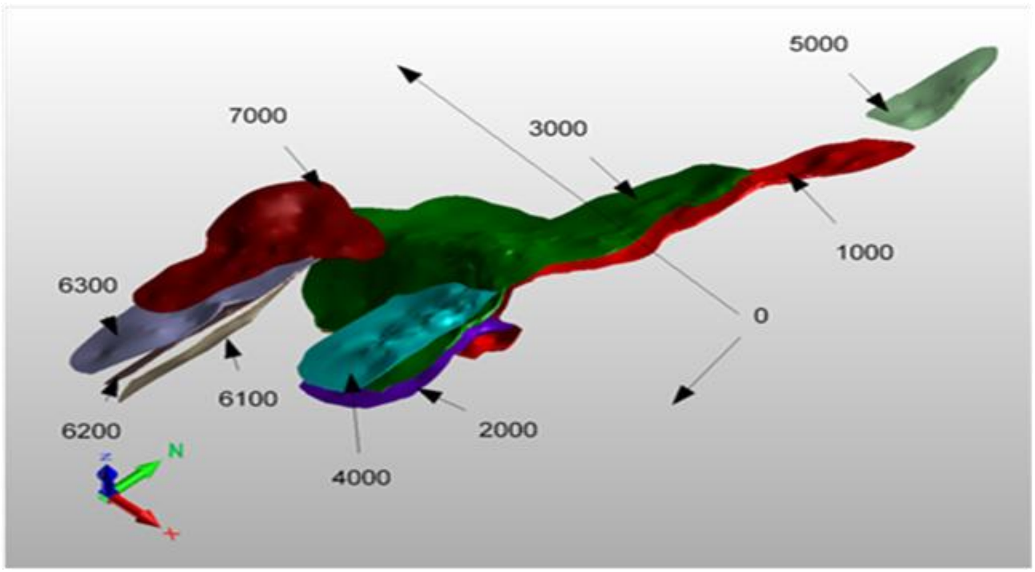


Figure 57 3-D Visualization of the Quigleys Deposit Mineralized Zone Positions with Wireframe Codes, Tetra Tech 2025

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).

11.3.5 Specific Gravity Data

Bulk specific gravity 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 specific gravity 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 specific gravity was therefore estimated by assuming the final envelope contained 15% shear and 85% stockwork and weighting the specific gravity values accordingly. Table 65 shows the specific gravity data assigned to the Quigleys area according to oxidation state.

Material Type	Specific Gravity
Oxide within modeled shear (t/cm)	2.60
Oxide Waste (t/cm)	2.62
Transition within modeled shear (t/cm)	2.65
Transition waste (t/cm)	2.58
Primary (Fresh) within modeled shear (t/cm)	2.70
Primary (Fresh) waste (t/cm)	2.61

Table 65 Quigleys deposit specific gravity data

11.3.6 Quigleys Estimation Methods and Parameters

The Quigleys block model was estimated in MicroMine software. The model was constructed as a non-rotated model with a block size of 5x5x1 m, as shown in Table 66. Gold grades were estimated into the block model using ordinary kriging.

Direction	Minimum (m) AMG84 z53	Maximum (m) AMG84 z53	Block Size (m)	# Blocks
x-dir	188,702 mE	190,000 mE	5 m	261
y-dir	8,437,400 mN	8,439,400 mN	5 m	401
z-dir	-178 m	220 m	1 m	220

Table 66 Block model physical parameters – Quigleys deposit

11.3.7 Variography and Search

A number of absolute, log and indicator directional variograms were calculated and modeled. From these, Figure 58 shows the log (Au) variogram.

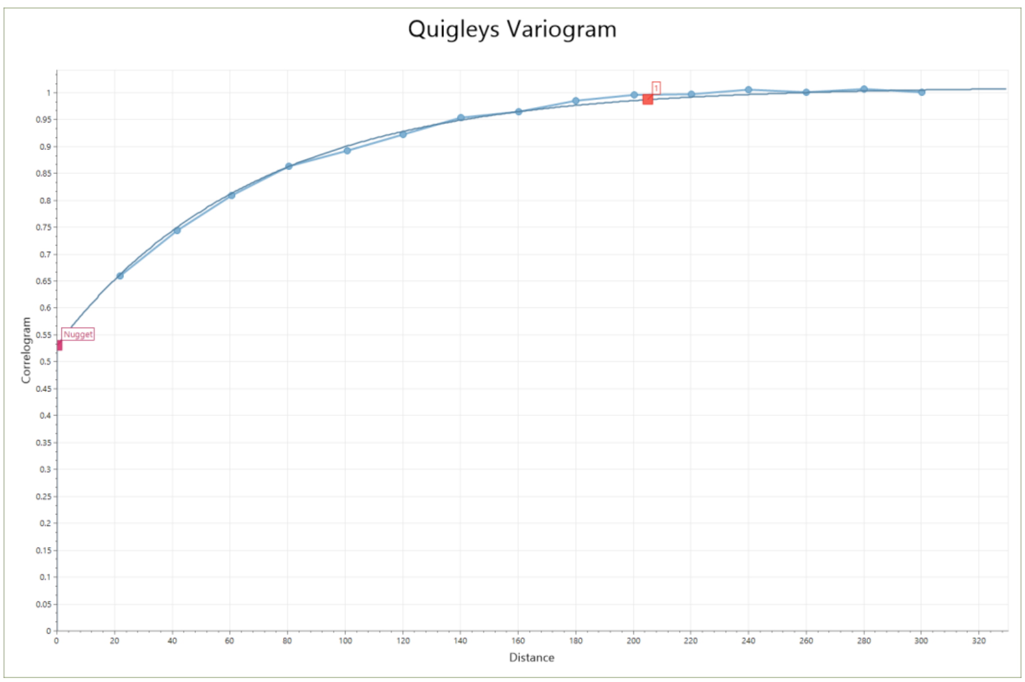


Figure 58 Quigleys Variogram

The variogram has a nugget of 0.77, with an ultimate sill of 2.74. The range is 90 meters along strike Table 67 shows the search parameters selected 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 67 Search Parameters for Each Domain, Quigleys Deposit

Figure 59 shows a plan view of the Quigleys resource block model and the resource shell with section lines for the cross sections below.

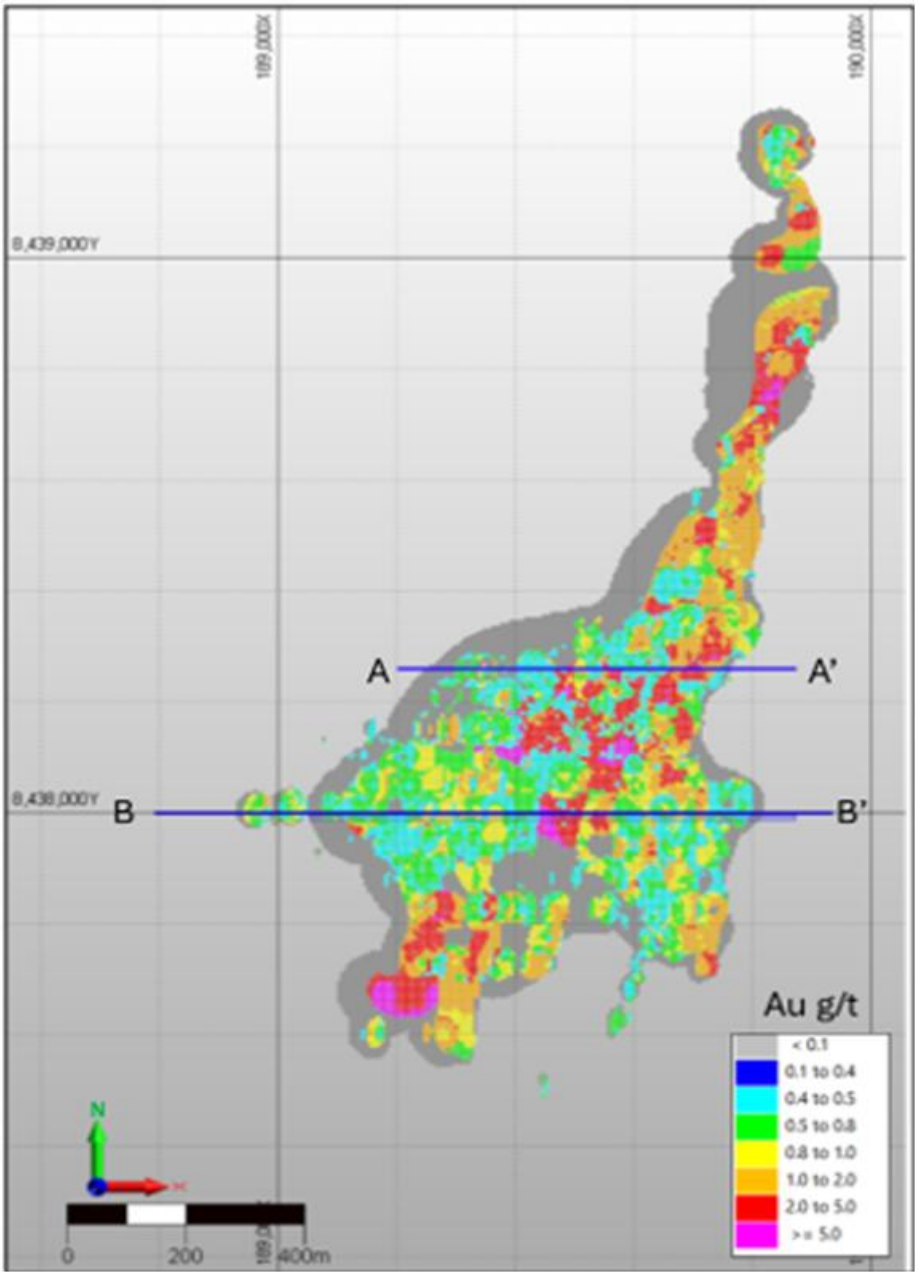


Figure 59 Plan View of the Quigleys Mineral Resource Model and 2025 Quigleys Mineral Resource Shell with Section Lines, Tetra Tech 2025

Figure 60 and Figure 61 show the estimated blocks in the updated Quigleys resource block model colored by gold grade. The sections also show the current topography, drill holes, gold assays, and the 2025 Mineral Resource pit shell, created with a gold price of USD1,950/oz.

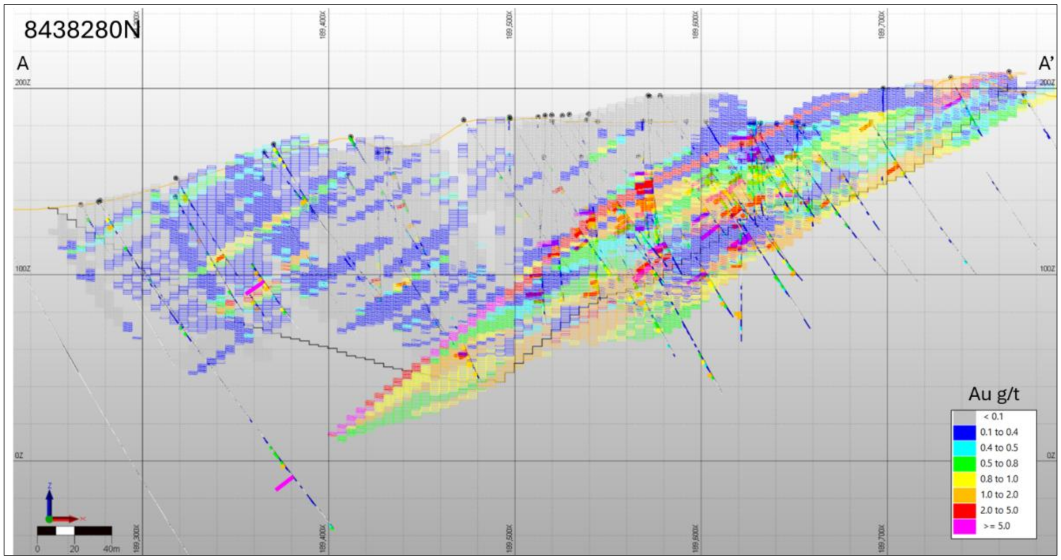


Figure 60 Cross Section A-A' of the Quigleys Deposit, 2025 Mineral Resource Shell, Looking North, Tetra Tech 2025

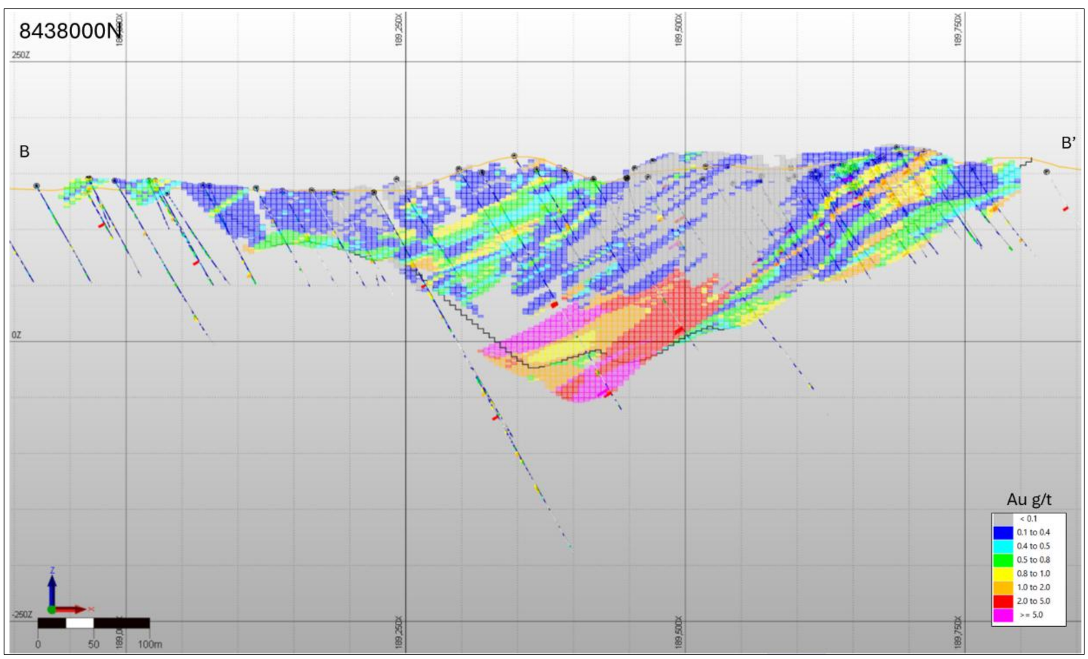


Figure 61 Cross Section B-B' of the Quigleys Deposit, 2025 Mineral Resource Shell, Looking North, Tetra Tech 2025

11.3.8 Mineral Resource Classification

Table 68 lists the Mineral 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 drill hole. 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 Mineral 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, and Inferred (MIF) codes were then adjusted to a more restricted class when a blocks kriging error exceeded this value.

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

Table 68 Search parameters and sample restrictions, Quigley’s deposit

For the outside zone, a two-stage kriging for MIF class 3 (see Table 128) was done inside and outside of modeled 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. Blocks and composites above the topography were considered in the estimation and excluded from the Mineral Resource as mined out material.

11.3.8.1 Cut-off Grade

The Mineral Resource cut-off grade for the Quigleys deposit was calculated to be 0.4 g Au/t. This cutoff grade was calculated using economic inputs provided by Vista and the consultants responsible for this Technical Report Summary. Upon comprehensive review, the Geology and Mineral Resource Qualified QP confirmed that the techno-economic inputs utilized for the Mineral Resource Estimate are appropriate. The applied assumptions represent a conservative set of inputs, thereby satisfying the requirements and professional judgment of the QP.

11.3.9 *Quigleys Statement of Mineral Resources*

Mineral Resources have been constrained by a pit shell, created in the MicroMine pit optimizer module. Table 69 lists the parameters used to generate a pit shell for reporting the Mineral Resources.

Item	Input
Gold Price	USD1,950 per troy ounce
Gold Recovery	89.7%
Overall Mining Cost	USD3.90 per tonne
Processing Cost	USD17.50 per tonne processed
General and Administrative Cost	USD1.50
Slopes	Degrees

Table 69 Quigley's Pit shell parameters

Table 70 shows the summary of the Mineral Resource Estimate for the Quigleys deposit.

Classification	Tonnes (000s)	Grade (g Au/t)	Contained Gold Ounces (000s)
Measured (M)	3,702	1.13	134
Indicated (I)	6,965	1.34	299
Measured and Indicated	**10,667**	**1.26**	**433**
Inferred (F)	2,761	0.71	63

(1) Measured and Indicated Mineral Resources exclude Proven and Probable Mineral Reserves.

(2) Quigleys Mineral Resources are quoted at 0.4 g Au/t cut -off grade.

(3) The Point of Reference for the Quigleys Mineral Resource estimate is in-situ at the property

(4) Quigleys: Mineral Resources constrained within a USD1,950/oz gold pit shell. Pit parameters: Mining Cost USD3.00/tonne, Processing Cost USD17.50/tonne processed, General and Administrative Cost USD1.50/tonne processed, Au Recovery 89.7%.

(5) Kira Johnson MMSA of Tetra Tech is the QP responsible for the Statement of Mineral Resources for the Quigleys Deposit.

(6) The effective date of the Quigleys Mineral Resource estimates is July 25th, 2025.

(7) Mineral Resources that are not Mineral Reserves have no demonstrated economic viability and do not meet all relevant modifying factors.

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

(9) The Mineral Resources were estimated in accordance with subpart 229.1300 of Regulation S-K.

Table 70 Summary of the Mineral Resource Estimate for the Quigleys deposit reported at a 0.4 g Au/t cutoff grade

Mineral Resources that are not Mineral Reserves have no demonstrated economic viability and do not meet all relevant modifying factors.

Figure 62 shows the grade tonnage relations for Measured and Indicated classified Mineral Resources.

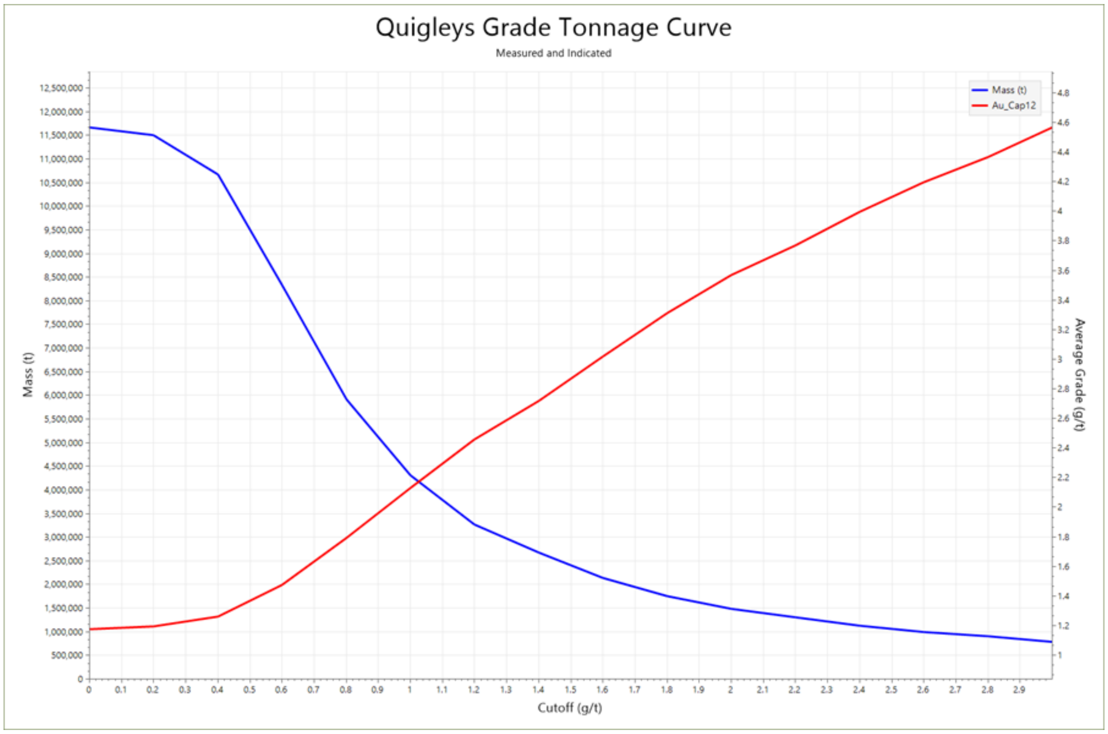


Figure 62 Measured and Indicated grade tonnage curve for the Quigleys deposit

Mineral Resources that are not Mineral Reserves have no demonstrated economic viability and do not meet all relevant modifying factors.

Figure 63 shows the grade tonnage relations for Inferred classified Mineral Resources.

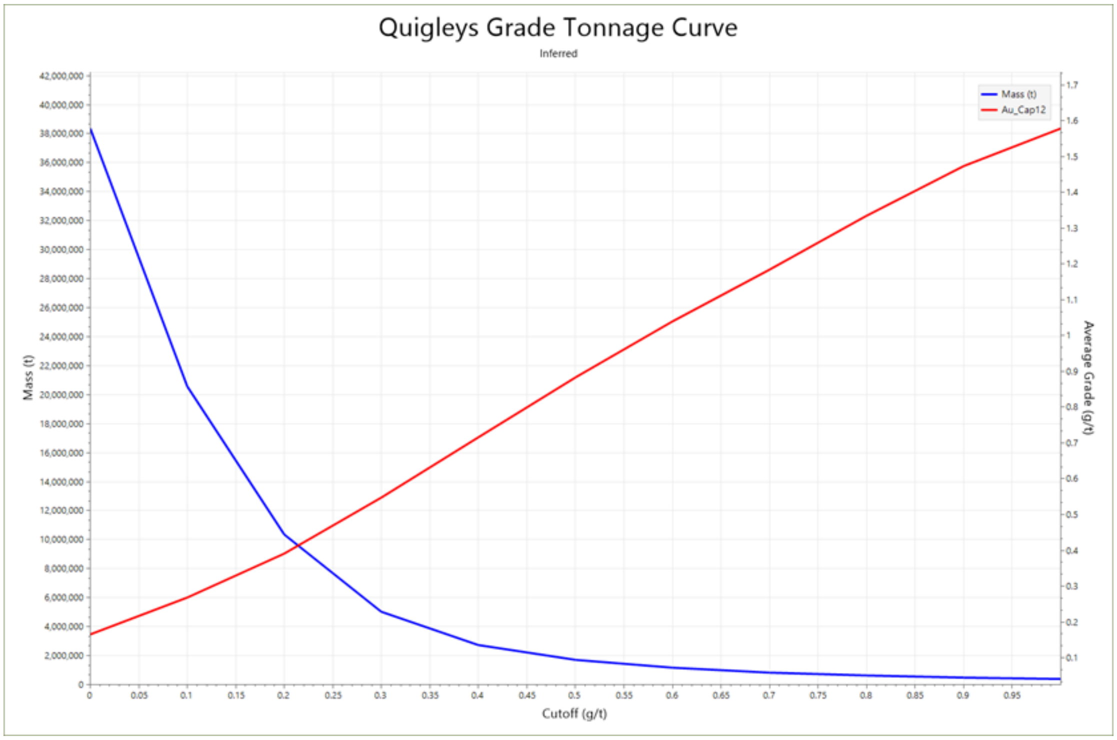


Figure 63 Inferred grade tonnage curve for the Quigleys deposit

11.3.10 Quigleys Mineral Resource Model Verification

Several methods used to validate the block model were used to determine the adequacy of the Quigleys Mineral 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.

The use of visual inspection of the kriged blocks models in section along with the assay and composite values were conducted by multiple professionals. Model comparisons were run by inverse distance and near neighbor methods as a verification.

Swath plots were also analyzed for the block estimation process the inspection of gold histograms of assays, composites and blocks. Figure 64 shows a north south swath plot of the ordinary kriging, inverse distance squared, and nearest neighbor block models.

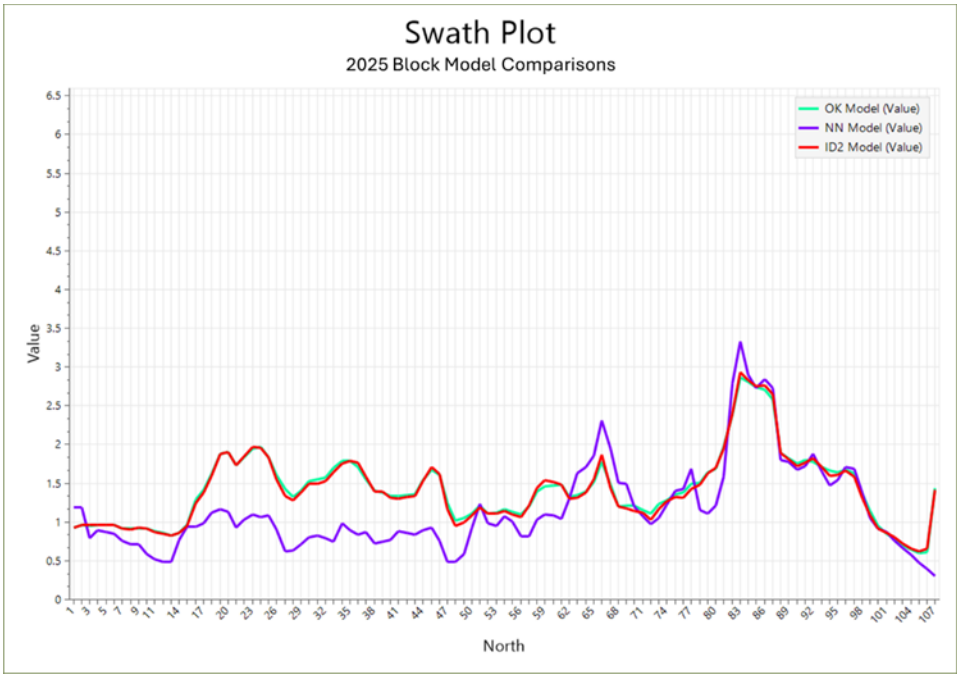


Figure 64 North-south swath plot for the estimated block models for the Quigleys deposit

11.4 Existing Heap Leach Pad Gold Mineral Resource

In addition to the in-situ gold Mineral Resource for the Batman deposit and Quigleys deposit, a historical heap leach pad (HLP) adjacent to the current Batman pit was analyzed for gold. The HLP is a remnant of the Pegasus operation, pre-2006. The HLP's geometry was analyzed using historical maps to determine the pile bottom and current surveys of the present day surface. This work produced two surfaces which were used to calculate the volume of the pile. The concentration of gold was analyzed with 24 vertical drillholes separated by 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 was measured using a dual density sidewall gamma probe. The probe uses a gamma source and a scintillation detector to estimate density via the Compton Effect. A total of 1,162 samples were collected from 11 drill holes, and the density was estimated to be 2.01 for the heap leach pile.

A nearest neighbor 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 232,000 ounces of gold within 13.4 Mt at an average grade of 0.54 g Au/t. It is the opinion of the QP that the Heap Leach Pad Mineral Resource can be classified as an Indicated Mineral Resources as the surveyed volume, the tonnage derived from density measurements, and grade assays from drillhole sampling reconciles with Pegasus' original reported values.

Table 71 lists the estimated Mineral Resources for the existing Heap Leach Pad inclusive of Mineral Reserves. The entirety of the Mineral Resource is converted to Mineral Reserves in this Technical Report Summary, therefore, a Mineral Resource exclusive of Mineral Reserves are not reported. A gold cut-off grade was not applied to the Heap Leach Pad Mineral Resource as all material will be processed as part of the site rehabilitation process. Tetra Tech performed a confirmatory estimate of the Heap Leach Pad Mineral Resource in 2025.

Category	Tonnes (000s)	Grade (g Au/t)	Contained Gold Ounces (000s)
Indicated	13,352	0.54	232

Note:

(1) No cutoff grade is technically applied as all Heap Leach Pad material will be re-processed.

(2) Mineral Resources are reported at 0.4 g/t cut-off gold grade to be consistent with the reported Batman and Quigleys Mineral Resource.

(3) The Point of Reference for the Heap Leach Pad Mineral Resources estimates is the physical Heap Leach Pad at the property.

(4) Mineral Resource is defined by the geometry of the existing Heap Leach Pad.

(5) Mineral Resource & Reserve estimates for the Heap Leach Pad materials are the same because 100% of the heap leach material is processed at the conclusion of mining the Batman Pit.

(6) Kira Johnson MMSA of Tetra Tech is the QP responsible for the Statement of Mineral Resources for Heap Leach Pad.

(7) The effective date of the Quigleys Mineral Resource estimates is July 25th, 2025,

(8) Mineral Resources that are not Mineral Reserves have no demonstrated economic viability and do not meet all relevant modifying factors.

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

(10) The Mineral Resources were estimated in accordance with subpart 229.1300 of Regulation S-K.

Table 71 Existing Heap Leach Pad Mineral Resource Estimate

11.5 Mineral Resource Uncertainty Discussion

Due to the nature of Mineral Resource estimation and modeling, there are many factors that can affect the reported Mineral Resource in this Technical Report Summary, which include:

- The model inputs for estimation are current through the effective date of the Technical Report Summary and additional data may materially alter the reported Mineral Resource
- Modeling of the mineralized veins and their orientation may be different than the actual size and location due to the nature of the data inputs and the mineralization at the site
- Changes in the costs of the Project could affect the cut-off grade for the reported Mineral Resource
- Changes in metal prices and market conditions could affect the cut-off grade, which can alter the reported Mineral Resource
- Metallurgical recovery can have an influence on the cut-off grade, which can alter the reported Mineral Resources
- Additional drilling at the site could provide further confidence in the reported Mineral Resource; however, additional drilling may provide information to restrict or reduce the modelled shapes and estimated Mineral Resource model

The relevant technical and economic parameters applied to calculate the cut-off grade for the Mineral Resource were compiled by Vista and the consultants responsible for this Technical Report Summary.

11.6 Relevant Factors Affecting Mineral Resource Estimates

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

11.7 Qualified Person Opinion

Kira Jonhson, MMSA of Tetra Tech, the QP for this section of the Technical Report Summary, is of the opinion that the geologic and drill hole data are adequate and suitable for use in estimating Mineral Resources for this Technical Report Summary.

It is the opinion of the QP for this section that the reported Mineral Resource classifications comply with current S-K 1300 definitions for each mineral class. Due to the nature of the veins, there will be an inherent degree of uncertainty over their location and size. Additional drilling can increase the confidence of the shape and grade of the mineral deposit. Changes in metal prices, metallurgical recovery, market conditions, and cut-off grade should continue to be analyzed as the project progresses, as they could result in material changes in the reported Mineral Resource.

12. MINERAL RESERVES ESTIMATES

12.1 Introduction

The Project is at FS stage conceived as a conventional open pit, truck and hydraulic excavator operation feeding a nominal 15 ktpd processing plant. The Mineral Reserve estimation is supported by this Technical Report Summary. To support the Mineral Reserve evaluation within this Technical Report Summary a Whittle 4X open pit optimization evaluation was completed with no value given to the Inferred classified material within the Mineral Resource estimate for the deposits. The Measured and Indicated Mineral Resource estimates for the Batman Deposit presented in Section 11 were used to estimate Mineral Reserves.

The Whittle run completed for this Technical Report Summary, included the resultant Mineral Reserve Pit shell as the last phase in the optimization, with interim shells also selected allowing for minimum mining widths and practical stage integration with the preceding stages. The resulting set of shells was then used for guidance in pit design, following the Geotech slope design parameters provided by the geotechnical consultant and detailed in this Technical Report Summary.

Following this a FS level mine design, mine scheduling, mining costing and overall Project economic model evaluation was completed to confirm positive economic outcomes for the Mineral Reserve. A conservative gold cut-off grade of 0.5 g Au/t was adopted based on economic parameters and recoveries determined as part of this Technical Report Summary. Mining costs and overall Project economic models completed included engagement with Tier 1 Australian based mining contractors to provide current industry pricing based on recent experience including within the nearby region.

The following section includes definition of the Mineral Reserves based on strategic mine planning including block model review and analysis and optimization work. Later sections detail the mine design, production schedule and the mining costs used in the economic model.

12.2 Mineral Reserves Estimations

The resultant Mineral Reserve summary is shown in Table 72.

	Batman Deposit			Heap Leach Pad			Total		
	Ore	Grade	Contained Gold	Ore	Grade	Contained Gold	Ore	Grade	Contained Gold
	Tonnes (000s)	(g Au/t)	Ounces (000s)	Tonnes (000s)	(g Au/t)	Ounces (000s)	Tonnes (000s)	(g Au/t)	Ounces (000s)
Proven	77,359	0.95	2,371				77,359	0.95	2,371
Probable	81,263	0.99	2,588	13,352	0.54	232	94,617	0.93	2,820
Proven & Probable	**158,623**	**0.97**	**4,959**	**13,352**	**0.54**	**232**	**171,975**	**0.94**	**5,190**

Notes:

(1) The Mineral Reserves point of reference is the point where material is fed into the process plant.

(2) Batman deposit Mineral Reserves are reported using a 0.50 g Au/t cut-off grade and USD1,800 per ounce gold price.

(3) Colin McVie and Peter Lock of Mining Plus are the QP's responsible for the Statement of Mineral Reserves for Batman Deposit Proven and Probable Mineral Reserves.

(4) Because all the Heap Leach Pad Mineral Reserves are to be fed through the process plant, these Mineral Reserves are reported without a cut-off grade applied.

(5) Deepak Malhotra is the QP responsible for reporting the Heap Leach Pad Mineral Reserves.

(6) The effective date of the Batman and Heap Leach Mineral Reserves estimate is July 25th, 2025.

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

(8) The Mineral Reserves were estimated in accordance with subpart 229.1300 of Regulation S-K.

Table 72 Project Mineral Reserves Estimate

12.3 Ore Body Description & Block Model

12.3.1 Ore Body Description

The full ore body description for the Batman deposit used as the basis for this Mineral Reserve is outlined in the Mineral Resource - Section 11 of this Technical Report Summary.

The 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 Batman deposit extends approximately 2,400 m along strike, 600 m across dip and drill tested to a depth of 800 m Mineralization is open at depth as well as along strike, although the intensity of mineralization weakens to the north and south along strike.

Further discussions of the block model used and its interpretation for mining analysis are outlined in the following sections.

12.3.2 Block Model

The Mineral Resource block model used for this Technical Report Summary Mineral Resource Estimate (MRE) for the Batman deposit was created and issued by Tetra Tech, defined here as 2025 MRE model.

12.3.3 Modifications to the 2025 MRE Model

Several modifications were applied to the 2025 MRE block model prior to mine optimization and design to ensure its suitability for downstream mining processes. These adjustments included:

- Extension of the weathering surface boundaries to fully encompass the ultimate pit shell, improving geometallurgical accuracy.
- Removal of non-essential attributes in the 2025 MRE block model, retaining only those relevant to this Technical Report Summary.
- Addition of new attributes to support mine optimization, pit design, and scheduling workflows.

12.3.4 Additional Model Fields

A number of additional block model fields were added to prepare the block model for use in the mining evaluation.

au1_mi – Gold Grades for Measured and Indicated Mineral Resource category only
average processing recovery – Recovery Model provided by GRES based on the following formula:

$$Au\ Leach\ Recovery = [(2.6384\ \times au1_{ppm}) + 87.947] \times 0.991$$

mcaf – Mining Cost adjusted by bench elevation using the following regressions based on contractor pricing:

Ore mining cost for z bench; $mcaf = (-0.005\ \times\ Bench\ RL) + 5.1497$
Waste mining cost for z bench; $mcaf = (-0.0057\ \times\ Bench\ RL) + 5.2655$

oreflag – Binary code for ore or waste
rocktype – A single digit integer representing the oxidation code
rockcode – A four-digit identifier combining the resource classification and the oxidation state of each block
topobelo – The percentage of the block contained below the topography
sg2 - A function of the original density and the topobelo attribute to return a true tonnage of the block
slopezon – Geotechnical slope domains extracted from WSP geotechnical study

12.4 Dilution and Recovery Estimates

An assessment of potential mining dilution and recovery estimates on the 2025 MRE model was completed multiple ways to identify assumptions to be used in this Technical Report Summary. This was completed considering different aspects of the Mineral Resource and its geology, with consideration how it will be mined and modeled. The following components formed the dilution and recovery assessment.

1. Selective Mining Unit (SMU) Assessment.
2. Deswik Stope Optimizer (Deswik.SO).
3. Contact Edge Investigation.
4. Skin Dilution Analysis.

Following the completion of the dilution and ore loss study, an updated Mineral Resource model was released. The revision was primarily driven by additional infill and extensional drilling conducted in the northeastern sector of the deposit, which successfully delineated further mineralization. The updated model was provided regularized with a block size of 12x12x6 m, as shown in Figure 65. Comparative analysis from the dilution and ore loss study indicated negligible variance in model performance between the 12x12x6 m and 6x12x6 m block configurations. Consequently, it was considered technically appropriate to adopt the updated model in its native block resolution, including its inherent dilution and ore loss, for subsequent mine planning and evaluation workflows.

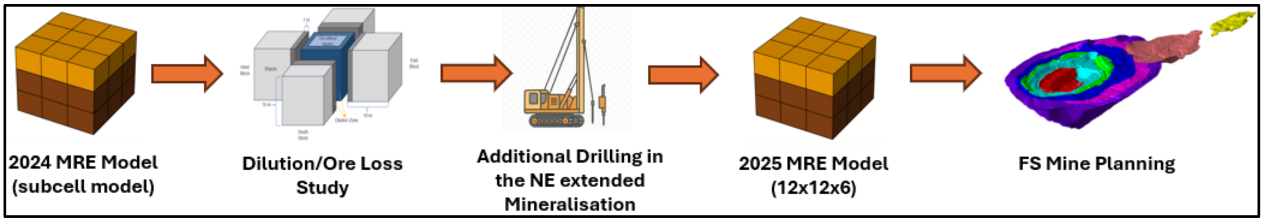


Figure 65 Mineral Resource Model Evolution, Mining Plus 2025

12.5 Cut-off Grade

Cut-off grade determination was conducted using first-principles economic analysis, informed by input parameters provided by Vista and further validated with GRES and other contributors of this Technical Report Summary. The summary of these key parameters is outlined in Table 73. This process involved calculating the break-even grade at which the revenue generated from gold recovery, including gold price assumptions, metallurgical recovery rates, operating costs, and selling costs, offsets the costs of processing and general and administrative expenses. Since the lower grade material will be stockpiled and rehandled for processing, an allowance was used for rehandle costs.

A detailed analysis of gold prices was used in Ore Mineral Reserve calculations of global projects reported during 2024; the range used during this time was USD1,400/oz to USD2,145/oz. Hence a conservative gold price of USD 1800/oz was selected to calculate the cut-off grade for the declaration of Mineral Reserves.

The resultant economic marginal cut-off grade is 0.41 g Au/t. Taking into account operational factors and to assist in improving the economics of the Project, Vista decided to apply a conservative cut-off grade of 0.5 g Au/t, which was deemed economically viable under the defined Project conditions. This value served as the basis for Mineral Reserve classification and subsequent mine planning activities.

Parameter	Unit	Value
Mineral Resource Classification		Measured + Indicated
Production		
Processing Throughput	tpd	15,000
Economic Factors		
Gold Price	USD/oz	$1,800
Gold Price	AUD/oz	$2,647
Exchange Rate	USD/AUD	0.68
Processing and Refining		
Average Processing Recovery	%	Recovery Model (Section 10.4.12)
Average Feed Grade	g Au/t	0.97

Parameter	Unit	Value
Selling Costs		
Payability	%	99.90%
Royalty Costs	USD/oz	$49.50
Refining Costs	USD/oz	$3.00
Mining Cost		
Reference Mining Cost	USD/t mined	$3.00
Stockpile Rehandle Cost	USD/t mined	$1.62
Processing Cost		
Total Processing Cost	USD/t processed	$17.50
G&A Cost		
G&A	USD/t processed	$1.50
Marginal Cut-Off Grade	**g Au/t**	**0.41**

Table 73 Cut-Off Grade Estimation

12.6 Pit Optimization & Mining Design Summary

Optimizations were undertaken in GEOVIA Whittle software using the engineering block model discussed in section 12.3.3. Table 74 provides the techno-economic parameters used in the Whittle optimizations.

Parameter	Unit	Input (AUD)	Input USD
Economic Factors			
Gold Price	$/oz	$2,647	$1,800
Exchange Rate	USD:AUD	0.68	
Processing and Refining			
Processing Throughput	tpd	15,000	
Processing Recovery	%	Recovery Model (Section 10.4.12)	
Average Gold Feed Grade	g/t	0.97	
Selling Cost			
Payability	%	99.9	
Royalty Costs	$/oz	$79.41	$53.99
Refining Costs	$/oz	$4.41	$3.00
Total Selling Cost	**$/oz**	**$83.82**	**$56.99**
Mining Cost			
Contractor Mining Cost	$/t mined	*Mcaf* Mining Cost adjusted by bench elevation	
Processing Cost			

Parameter	Unit	Input (AUD)	Input USD
Total Processing Cost	$/t processed	$25.74	$17.50
G&A	$/t processed	$2.21	$1.50
Processing Cost & G&A	**$/t** processed	**$27.94**	**$19.00**

Table 74 Whittle Optimization Input Parameters

A processing recovery model formula that is outlined in Section 10.4.12 was supplied by GRES and used for calculating the attribute value, repeated below:

Gold Extraction (Leach) = (2.6384 x Mill Head Grade (Au) + 87.947) x 0.991(Ads Eff).

This formula was only applied to Measured and Indicated ore.

12.6.1 Initial Optimization Derivation

A preliminary assessment was carried out to determine the appropriate slope angles to use for the Technical Report Summary in accordance with the 2025 WSP Geotechnical parameters. An initial review of the previous slope design parameters used in previous studies and the current recommendations for this Technical Report Summary was completed with an objective to understand any variances and how it would impact overall pit optimization and pit design.

Table 75 shows the variance in Inter-Ramp Angle (IRA) between the previous mining study and this Technical Report Summary.

Zone	Slope Zone	Weathering	2024 FS IRA (deg)	2025 DFS IRA (deg)	Var (deg)
1	North-East	Fresh	44	45	1
2	East	Fresh	50	45	-5
3	South	Fresh	55.1	52	-3.1
4	South-West	Fresh	55.1	49	-6.1
5	North-West	Fresh	51.4	49	-2.4
6	North-East	Oxide/Trans	27.4	31	3.6
7	East	Oxide/Trans	27.4	37	9.6
8	South/South-West	Oxide/Trans	42.9	37	-5.9
9	North-West	Oxide/Trans	42.9	37	-5.9

Table 75 IRA Comparison between 2024 FS Study and this 2025 Technical Report Summary

Decreases to IRA and the incorporation of the geotechnical safety berm were highlighted as a risk to overall slope angle and pit optimization results, prompting further detail on ramp placement and opportunities to minimize the impact of the changes.

A ramp configuration analysis was undertaken to validate the Overall Slope Angle (OSA) calculations with a result to provide a recommendation for the ideal ramp design configuration. This required creating two high-level pit designs:

- Spiral Ramp, for Overall Slope Angle OSA 1
- Switchback ramps on the East, for OSA 2

The comparative analysis of slope configurations returned minimal variance, with only a 1 to 2-degree change in OSA between the two slope parameter sets. Given the negligible difference and operational practicality, the OSA1 slope set was selected for further design and optimization work. From an operational point of view the spiral ramp configuration eliminates switchbacks which causes increased wear and tear on the truck fleet and increased road maintenance requirements as trucks will cause rutting in the road surface when doing tight cornering.

This decision was particularly influenced by the use of the ramp geometry in lieu of the 25 m geotechnical safety berms, which allowed for more efficient spatial utilization. While the West wall slope was slightly reduced under the OSA1 configuration, the compounding flat sections introduced by the switchback ramp design constrained pit depth development in the East. In contrast, the spiral ramp configuration provided greater flexibility for potential pit expansion in the eastern domain.

Pit optimization analysis conducted under both slope configurations yielded comparable pit extents in the West, excluding sensitive zones. This further reinforced the suitability of the OSA1 slope set for practical implementation.

Final slope zone values reflective of a spiral ramp design, OSA1, carried through in Whittle are shown in Table 76.

Zone	Slope Zone	Weathering	OSA (deg)
1	North-East	Fresh	38.0
2	East	Fresh	38.0
3	South	Fresh	45.5
4	South-West	Fresh	42.0
5	North-West	Fresh	42.0
6	North-East	Oxide/Trans	30.5
7	Remaining	Oxide/Trans	36.0

Table 76 Final Slope Zone Angles used for Optimization

12.6.2 *Final Whittle Results*

Final whittle optimizations were undertaken between Revenue Factors (RF) values 0.3 to 1.2. Results from the final optimization are shown in Figure 66 with chosen stage shells demarcated with black arrows. Figure 67, shows RF 1.0 shell with block model by grade bin for Measured and Indicated category only.

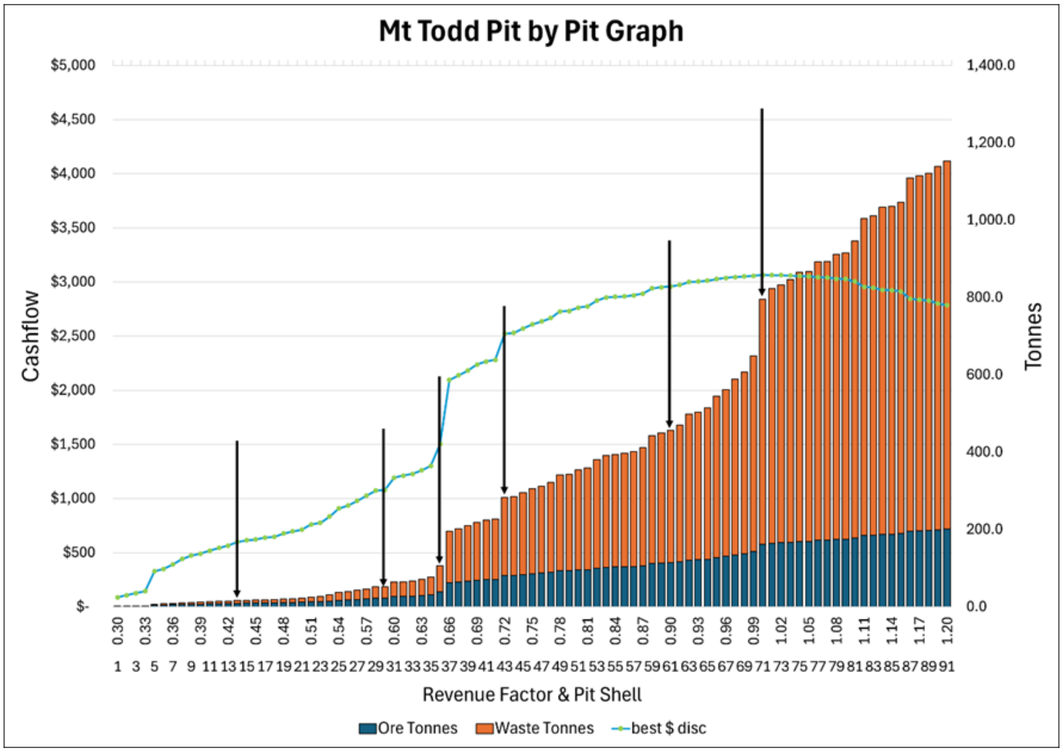


Figure 66 Oblique View of the RF1 Optimization Shell with the Block Model Colored by Ore Grade Bin, Mining Plus 2025

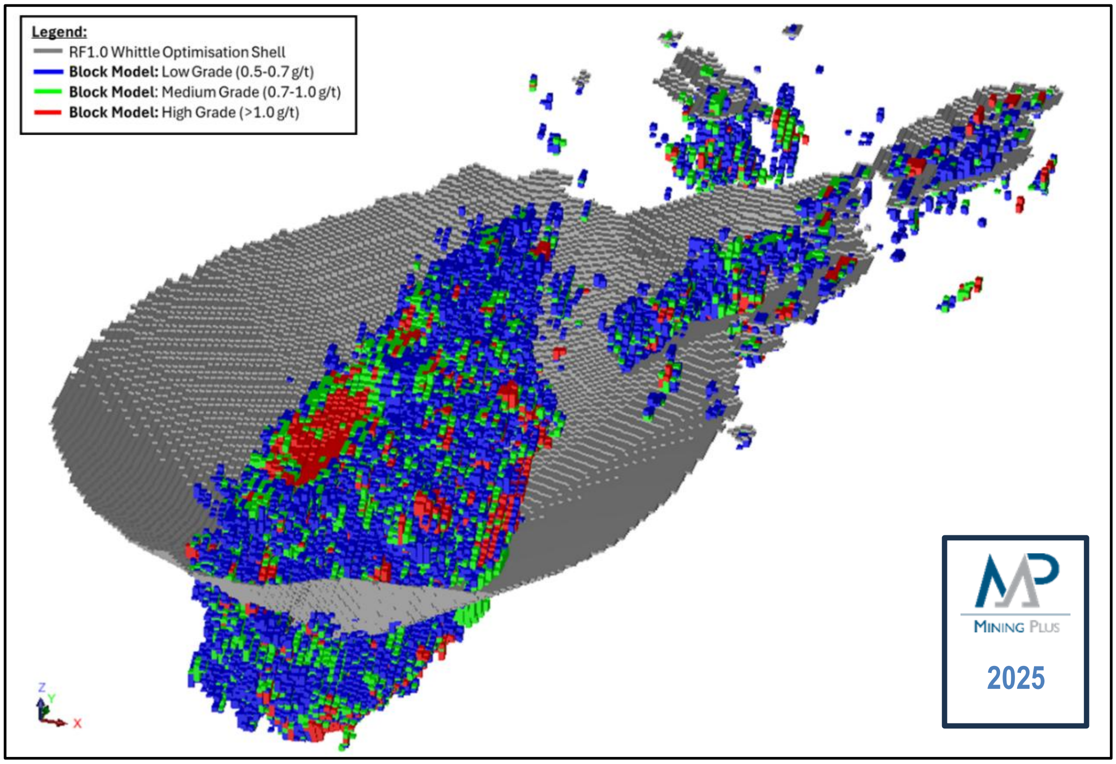


Figure 67 3D View RF1.0 Optimization Shell with Block Model by Grade Bin, Constrained to Measure and Indicated Only

The selection of pit shells was guided by a multi-criteria evaluation incorporating contained gold ounces, incremental economic value, and geometric suitability for staged pit development (based on visual inspection in 3D within the mine planning software). Emphasis was placed on identifying shells that provided a strategic balance between maximizing economic return and maintaining operational efficiency.

Key considerations included ensuring a sufficient number of mining stages to support multiple active working fronts, while also advancing waste stripping to expose adequate ore for consistent mill feed delivery. Each selected shell was evaluated to confirm that it provided adequate working width to support safe and efficient mining operations, particularly in deeper sections of the pit. Where the minimum required mining width of 30 meters was not maintained between pit shells, adjustments were incorporated to accommodate operational constraints during the final design process.

The final shell selection, summarized in Table 77, reflects the outcome of this optimization process and forms the basis for subsequent phase design and production scheduling.

Stage	RF Factor	Shell
1	0.43	14
2	0.58	29
3	0.65	36
4	0.72	43
5	0.89	60
6 - Ultimate	0.99	70

Table 77 Whittle Shells Selected for Mt Todd Pit Staging

Figure 68 illustrates the selected stage shells.

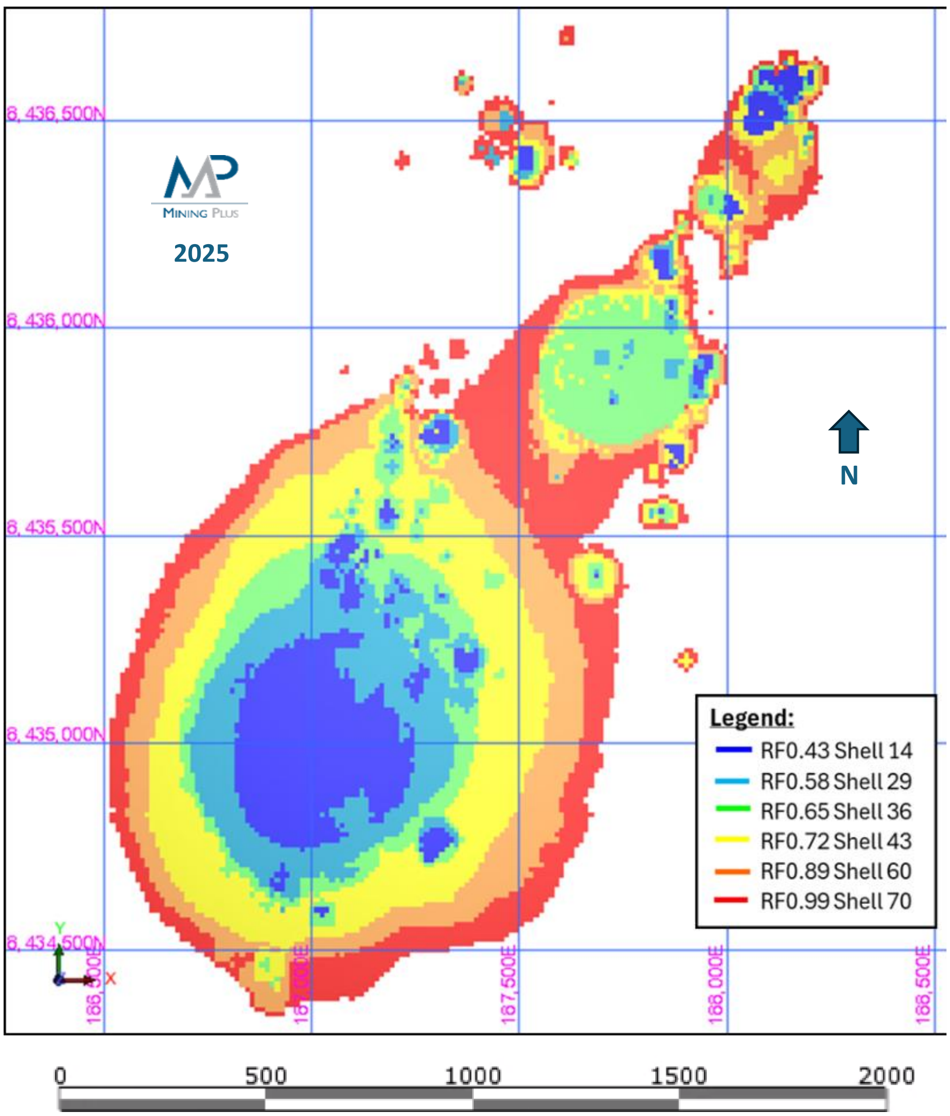


Figure 68 Plan View of Selected Whittle Optimization Shells for Staging

Figure 69 provides a cross section of the Whittle shells at the 8,435,000N overlaid with the topographical surface. This highlights the challenges of maintaining minimum mining widths between stages to ensure mining efficiency. This will need further consideration and review during the mine design process with the use of the shells as a guide for pit designs with it difficult to be followed in some areas while maintaining efficient mining areas and ramp access for each stage.

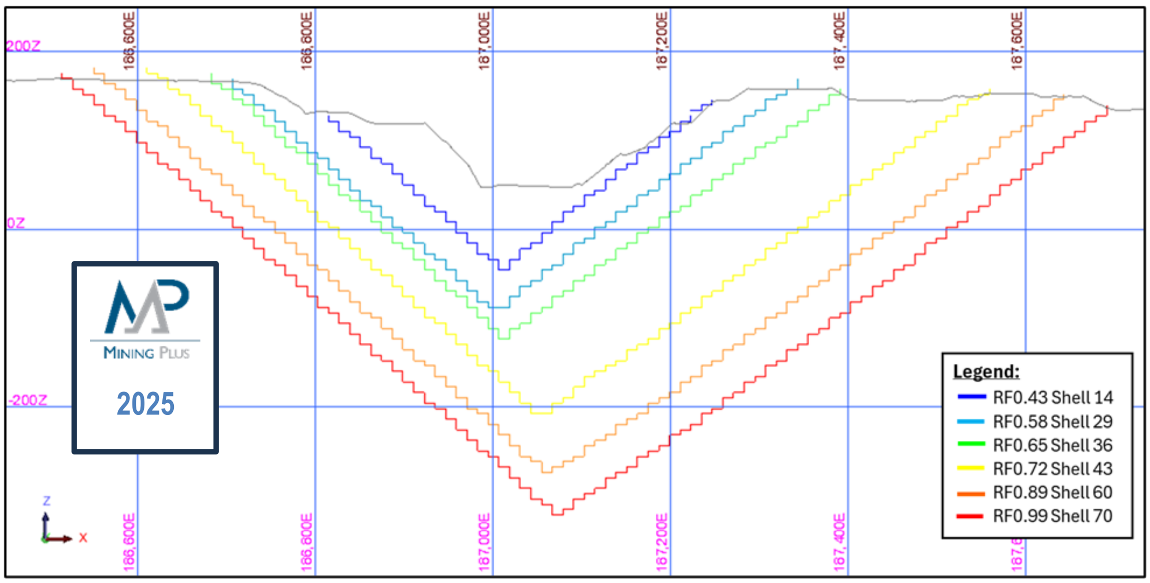


Figure 69 Cross-Section @ 8,435,000N of Selected Whittle Optimization Shells Looking North for Staging

12.6.3 Mine Designs

The following mining designs were created for the Batman deposit:

- Pit Designs, including Pit Stages and final design.
- Low Grade Stockpile (this a longer-term stockpile comprised of mainly low-grade, but early in the mine life some HG & MG may be stockpiled here as part of ore feed blending requirements), and
- Surface Haul Roads.

Additional infrastructure designs were provided by GRES , Tierra Group and other contractors and consultants including:

- ROM Pad location and size.
- Processing Infrastructure.
- Workshop/Office Area including mining contractor designs for mining infrastructure area comprising mobile maintenance workshops, fuel and fluid storage, wash bay, LV workshop and other mining operations requirements.
- Waste Dump.
- Tailings Storage Facilities.

- Figure 70 shows the ultimate pit design and mine site layout.

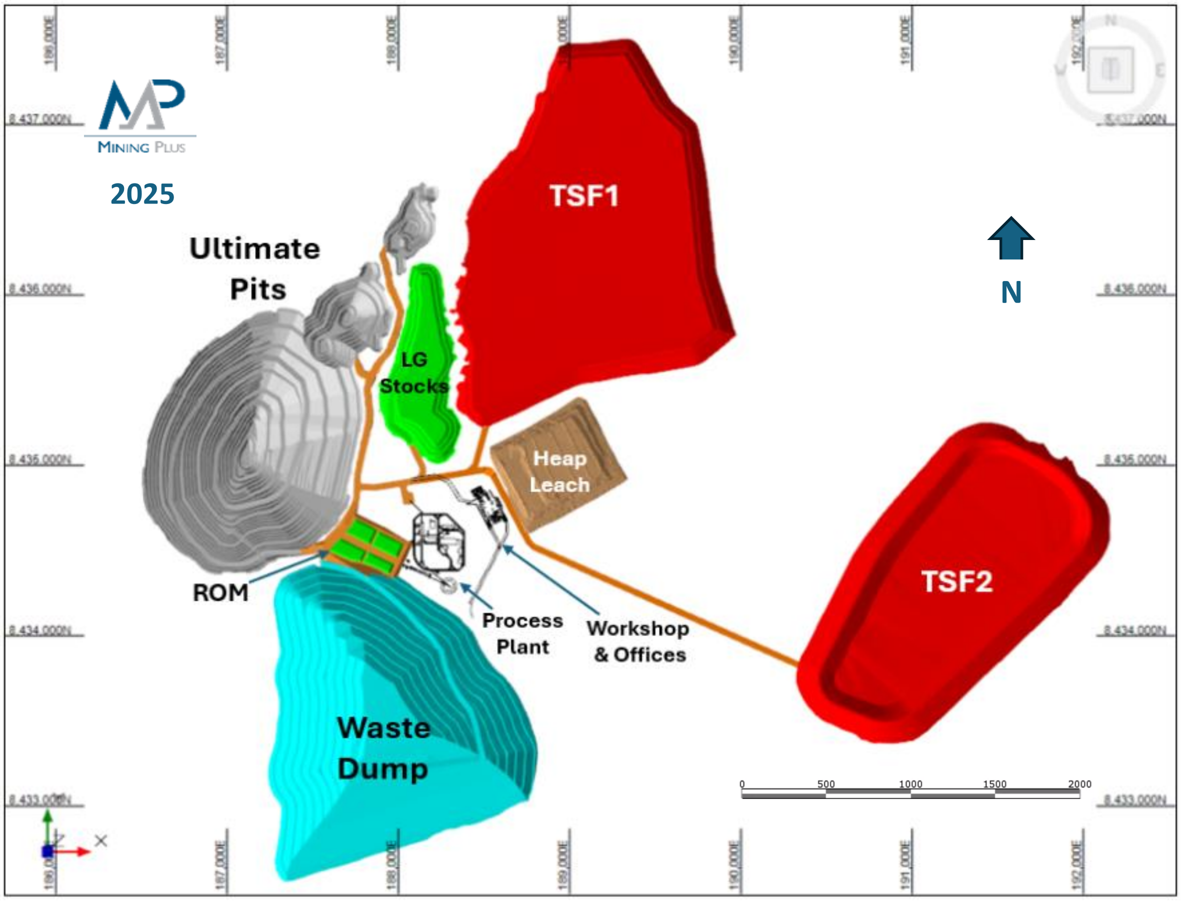


Figure 70 Mt Todd Ultimate Pit Design and Mine Layout

12.7 Heap Leach Pad Mineral Reserve Estimate

Heap Leach Pad Mineral Reserves are provided in Table 78. 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 process plant 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 test work 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 test work indicates that it is economically feasible to process and recover gold from the heap leach pad material. It is the opinion of the QP that the heap leach pad can be considered as Mineral Reserves categorized as Probable since limited drilling and assaying was undertaken to estimate the gold content of the heap leach pad residues. The Mineral Reserves included in the Mineral Reserve tabulation based on the following:

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

12.8 Mineral Reserves

Mineral Reserves for the Project were developed by applying relevant economic criteria (modifying factors) to define the economically extractable portions of the estimated Mineral Resources. Mining Plus developed the Mineral Reserves to be in accordance with subpart 229.1300 of Regulation S-K. S-K 1300 Mining Rules. S-K 1300 Mining Rules define Mineral Reserves as:

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.

Proven Mineral Reserve

A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve and can only result from the conversion of a Measured Mineral Resource.

Proven and Probable Mineral Reserves are stated by Mining Plus based on Mineral Resources in the pit designs for the Batman pit.

Deepak Malhotra is responsible for reporting of the Heap Leach Pad Mineral Reserves. This is based on the tonnage and grade of heap leach pad 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. The Heap Leach Pad Mineral Reserves are shown with the Batman Mineral Reserves.

The resultant Mineral Reserve summary for the Project summary is shown in Table 78.

	Batman Deposit			Heap Leach Pad			Total		
	Ore	Grade	Contained Gold	Ore	Grade	Contained Gold	Ore	Grade	Contained Gold
	Tonnes (000s)	(g Au/t)	Ounces (000s)	Tonnes (000s)	(g Au/t)	Ounces (000s)	Tonnes (000s)	(g Au/t)	Ounces (000s)
Proven	77,359	0.95	2,371				77,359	0.95	2,371
Probable	81,263	0.99	2,588	13,352	0.54	232	94,617	0.93	2,820
Proven & Probable	**158,623**	**0.97**	**4,959**	**13,352**	**0.54**	**232**	**171,975**	**0.94**	**5,190**

Notes:

(1) The Mineral Reserves point of reference is the point where material is fed into the processing plant.

(2) Batman deposit Mineral Reserves are reported using a 0.50 g Au/t cut-off grade and USD1,800 per ounce gold price.

(3) Colin McVie and Peter Lock of Mining Plus are the QP's responsible for the Statement of Mineral Reserves for Batman Deposit Proven and Probable Mineral Reserves.

(4) Because all the Heap Leach Pad Mineral Reserves are to be fed through the processing plant, these Mineral Reserves are reported without a cut-off grade applied.

(5) Deepak Malhotra is the QP responsible for reporting the Heap Leach Pad Mineral Reserves.

(6) The effective date of the Batman and Heap Leach Pad Mineral Reserves estimate is July 25th, 2025.

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

(8) The Mineral Reserves were estimated in accordance subpart 229.1300 of Regulation S-K.

(9) "-" indicates no reported value.

Table 78 Mt Todd Mineral Reserves Estimates

12.9 Qualified Persons Opinion

The Qualified Person (QP) is of the opinion that the technical data presented in this Technical Report Summary—including the geological model, Mineral Resource estimates, and the modifying factors applied—have been prepared in accordance with Regulation S-K 1300 and reflect industry best practices.

The economic parameters and assumptions used in the evaluation provide a reasonable basis for assessing the potential for economic extraction of the reported Mineral Reserves.

The QP has thoroughly assessed all relevant technical and economic factors that may influence the Project's viability. Based on this assessment, the QP concludes that the Project has a reasonable prospect of eventual economic extraction.

In accordance with Regulation S-K 1300, the QP has relied on certain information provided by the registrant regarding macroeconomic conditions, legal frameworks, environmental considerations, governmental factors, local accommodations and the Mineral Resource model. While this information has not been independently verified by the QP, such reliance is disclosed as required by regulatory standards.

13. MINING METHODS

13.1 Introduction

The Batman deposit will be mined in eight stages using a conventional truck and shovel approach with all material requiring drilling and blasting for fragmentation prior to excavation. Mining operations will be undertaken via contract mining with the contractor utilizing 400 tonne class excavators and 190 tonne class rigid frame trucks.

Multiple stages will be developed at any one time to manage mill feed tonnage and grade requirements, waste movement and stockpile balances.

Ore material will be designated as HG, MG or LG with approximately 15 ktpd being fed to the Processing Plant. HG and MG material not being fed directly to the Processing Plant will be stockpiled on a Run of Mine (ROM) pad adjacent to the plant with excess LG material (and a small amount of MG material particularly during initial years of mining operations) being placed on a dedicated LG stockpile.

Based on its sulfur content waste material will be classified as NAF or PAF with management plans and dumping strategies in place to ensure containment of the PAF material on a single waste rock landform. For waste material planned to be mined with no sulfur assays, as part of this Technical Report Summary, this unclassified material will be deemed as low level PAF. Recommendations for further work at the next stage of development include improved laboratory work and rock sampling for improved delineation of NAF and PAF waste rock.

Figure 71 illustrates the proposed layout of the operation highlighting the final pit geometry as well as mine roads, waste dump, low grade stockpile, processing plant, workshop and TSF locations.

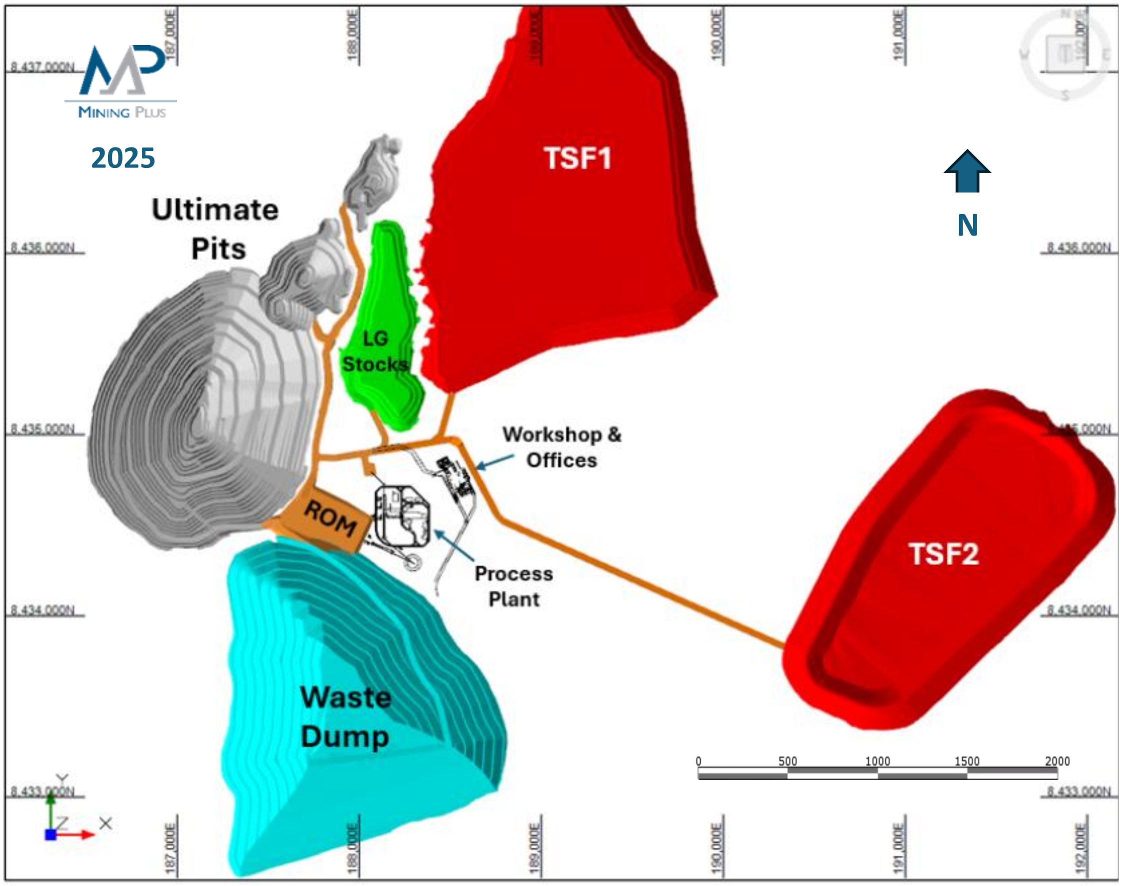


Figure 71 Mt Todd Final Pit Design and Mine Layout Pit Geotechnical Analysis

13.2 Pit Geotechnical Analysis

Vista engaged WSP Australia Pty Limited (WSP) to conduct the geotechnical assessment of the Batman pit slopes which forms part of this Technical Report Summary.

WSP considers that the slope angles and geometries adopted in the previous LOM pit design are optimistic for a 600 m vertical slope height. WSP derived more suitable slope angles and geometries that satisfy the pit slope design criteria as set in Read and Stacey (2009), Guidelines for Open Pit Slope Design. However, the recommended slope angles in this Technical Report Summary should still be validated by adequate geotechnical investigation and stability assessment before the final design stage.

There is a major deficiency in the geological structure, geotechnical and hydrogeological information available for this Technical Report Summary. To account for this uncertainty assumptions had to be made during the slope stability assessment to derive pit slope angles and geometries. This high level of uncertainty can impact the safety of personnel, loss of mining equipment and potentially adversely impact on the mining of ore Mineral Reserves.

Additional geological, geotechnical and hydrogeological investigations are recommended in this Technical Report Summary to ensure that adequate information is available prior to the next stages of the final detailed design for the geotechnical assessment of the Batman pit slopes. It is recommended to reassess the slope angles and geometries derived by WSP in this Technical Report Summary when this new information becomes available. This reassessment needs to be conducted before the final detailed design stage for the Batman pit.

The dominant slope failure mode for Batman pit slopes includes failure along geological structures. This failure mode is evident in the current pit slopes and confirmed in stability analysis. Adequate geological structure data and groundwater information was not available for this Technical Report Summary. The regular high rainfall during the wet season has the potential to increase pore-water pressures in pit slopes that can result in slope failure and requires further groundwater assessment and monitoring.

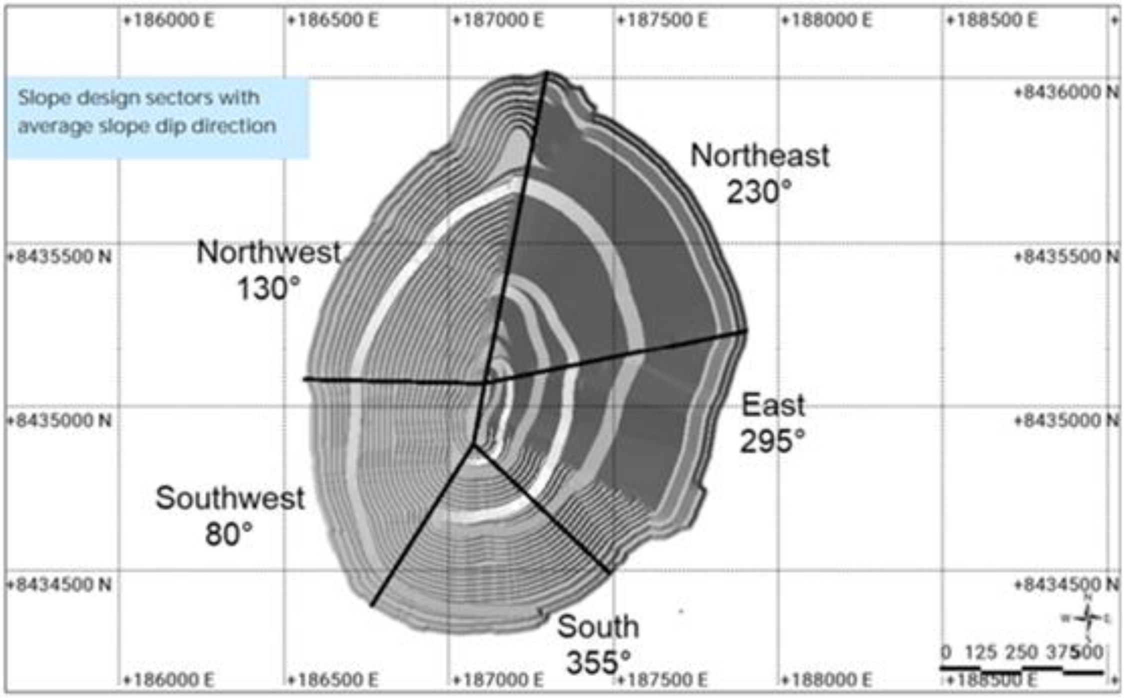
Mining last occurred at the Batman pit in early 2000 resulting in slopes that have been exposed for more than 25 years. The current southwest and northwest slopes are characterized by a significant number of small rockfalls and bench scale wedge and planar failures. There is also evidence of blast damage along the west wall. The slopes along the northeast, east and south slopes appear to perform better from a stability perspective. For access ramp placement the northeast, east and south highwalls will be more favorable.

Implementation of smooth wall blasting practices along final pit slopes is recommended. This is to reduce blast damage that can contribute to slope instability and rock falls. The use of pre-split and adequately designed trim blasts will be required. Berm crests should be protected to ensure berms remain effective in containing rock falls and failed material. Incorporate geotechnical information such as rock strength and geological structures in blast designs.

A budget allowance is recommended for the installation of ground support along pit slopes where geotechnical conditions require it. The requirements for ground support should be determined during mining. Ground support includes rock bolts, cables, mesh and rockfall catch fences to help stabilize local areas and manage the risk of rock falls. The northeast slope may require the installation of systematic ground support due to persistence of the bedding.

Ongoing geotechnical risk management of the Batman pit during mining is recommended. This includes the establishment of a site geotechnical department that support daily mining operations. Development of a ground control management plan (GCMP) for Batman pit is recommended.

The five slope design sectors are shown in Figure 72. The slope angles and geometries for weathered/oxide greywacke and fresh greywacke are provided in Table 79 and Table 80.



Source: WSP 2025

Figure 72 Slope Design Sectors for the Batman Pit, WSP 2025

Pit Design Sector	Maximum Inter-Ramp angle (°)	Maximum Bench Height (m)	Minimum Berm Width (°)	Bench Face Angle (°)	Geotechnical Berm
Northeast	31	12	8	40	25 wide, every 96 m vertically from pit crest. Ramp can be used in lieu of geotechnical/safety berm.
		24	12		
East, South, Southwest, Northwest	37	12	8	50	
		24	12		

Table 79 Weathered/Oxide Greywacke Slope Angles and Geometries

Pit Design Sector	Maximum Inter-ramp angle (°)	Maximum bench height (m)	Minimum berm width (°)	Bench face angle (°)	Geotechnical / safety berm	Comments
Northeast	45	N/A	N/A	N/A	25m wide, every 96m vertically from pit crest. Ramp can be used in lieu of geotechnical / safety berm.	The inter-ramp design angle is to follow the bedding dip.
East	45	12	7.5	65		-
		24	10	60		-
South	52	12	7.5	75		-
		24	10	70		-
Southwest	49	12	7.5	65		The bench design angle is to follow the veins dip.
		24	10	65		
Northwest	49	12	7.5	65		
		24	10	65		

Table 80 Fresh Greywacke Slope Angles and Geometries

13.3 Pit Hydrogeological and Hydrological Considerations

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 6—Geological Setting and Mineralization, Section 7; the geologic information in this section is presented from a hydrogeologic perspective as it relates to groundwater flow and pit dewatering.

13.3.1 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.

13.3.2 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 as the model has not yet been finalized or verified by comparison to measured groundwater inflows to the pit and measured changes in groundwater levels due to the nature of the status of the Project. Thus, the estimates of groundwater inflow to the expanded Batman Pit and post-mining groundwater system recovery should be considered preliminary, although sufficient for this level of Technical Report Summary. 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 and related hydrologic features such as streams.

For this Technical Report Summary, 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.

13.3.2.1 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 m^3 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.

13.3.3 Pit 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 81.

Mining Year	Nov-Jan (Wet Season) Mean Monthly Inflow Volume (m^3)	Jun-Aug (Dry Season) Mean Monthly Inflow Volume (m^3)
1	57,736	37
5	97,549	4,115
20	144,092	8,066
30	230,745	6,674

Table 81 Seasonal Inflow Volumes and for Mine Dewatering Design in FS pit design

13.3.4 Mine Dewatering

Dewatering of the proposed Batman Pit, for this Technical Report Summary 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. The 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 WTP Feed Pond. Depending on the depth of the pit and routing of the pipeline to the crest of the pit, waterpumped from the pit floor would first go through a pair of pumps mounted in the pit sump and then through skid mounted booster pumps. Lifts with booster pumps will be added in stages with increasing pit depth. Once at the surface, the water would be piped to the WTP Feed Pond. 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.

13.4 Final Pit Design and Optimal Pit Shell

Designs were based on the geotechnical recommendations provided by WSP and used in the optimizations. Road widths and gradients were calculated based on 190 tonne haul trucks.

Table 82 outlines the pit road parameters used for the pit designs.

Item	Units	Value
Pit Haul Ramp Width (dual-lane)	m	32
Pit Haul Ramp Width (single-lane)	m	18
Haul Ramp Gradient	1:X	10

Table 82 Pit Design Haul Road Parameters

Single lane ramps were used for the lower 100 m vertical development of each stage design. A 12 m "goodbye cut" was also incorporated into the final floor for all designs. A goodbye cut is the final material moved from an open pit where the backhoe excavator top loads trucks (trucks are parked next to the excavator, or the same level, instead of on the bench below) and selectively picks the ore from the pit floor, allowing final bench with no ramp access and minimalized waste.

Table 83 outlines the pit berm-batter configuration and bench heights for the final design.

Zone	Slope Zone	Weathering	Bench Height (m)	Berm Width (m)	Batter Angle (deg)	IRA (deg)	OSA (deg)
1	North-East	Fresh	24	0	45.0	45.0	38.0
2	East	Fresh	24	10	60.0	45.2	38.1
3	South	Fresh	24	10	70.0	52.0	45.7
4	South-West	Fresh	24	10	65.0	48.6	41.8
5	North-West	Fresh	24	10	65.0	48.6	41.8
6	North-East	Oxide/Trans	24	12	40.0	30.6	30.6
7	Remaining	Oxide/Trans	24	12	50.0	36.8	36.8

Table 83 Final Pit Design Berm-Batter Configuration

An alternative pit berm-batter configuration was applied to the interim stages based on their exposure time and development relative to the succeeding stage. Preliminary schedules identified that interim walls, particularly in the early stages, were exposed for 12 to 18 months before the succeeding stage would be developed and new pit wall established. Under this condition, alternative berm widths were applied to interim stages whilst maintaining batter angles and bench heights. This was an increase of approximately a 1.2 to 1.6 degrees to the inter-ramp angle for the fresh rock whilst a 1.3 to 1.8 degree change was applied to oxide and transitional.

Table 84 provides the berm-batter configuration and bench heights for the interim stages.

Zone	Slope Zone	Weathering	Bench Height (m)	Berm Width (m)	Batter Angle (deg)	IRA (deg)	OSA (deg)
1	North-East	Fresh	24	0	45.0	45.0	38.0
2	East	Fresh	24	9	60.0	46.4	39.1
3	South	Fresh	24	9	70.0	53.5	47.0
4	South-West	Fresh	24	9	65.0	49.9	42.8
5	North-West	Fresh	24	9	65.0	49.9	42.8
6	North-East	Oxide/Trans	24	10	40.0	31.9	31.9
7	Remaining	Oxide/Trans	24	10	50.0	38.5	38.5

Table 84 Interim Pit Design Berm-Batter Configuration

As recommended by WSP, an additional 25 m geotechnical safety berm was also required every 96 m depth with pit ramps able to be used in lieu of this safety berm. An equivalent safety catch berm, determined by matching the OSA, was applied to the design relative to vertical distance between pit ramps or full vertical 96 m sections.

Additional pit design parameters were also applied to the North-Eastern extension of the Batman Pit. By default, this area was assigned North-East slope zone parameters. Observations from the Project site visit completed by the mining QP on March 11-13, 2025, displayed an alternative behavior to the default assignment within the slope zone recommended, so it was decided for a simplified replica of the Batman pit slope zones specific to the North-East extension based on similar observed rock structure and conditions.

Figure 73 shows the simplified Batman pit parameters that were applied to the North-East and Table 85 details the berm-batter configuration applied to the design.

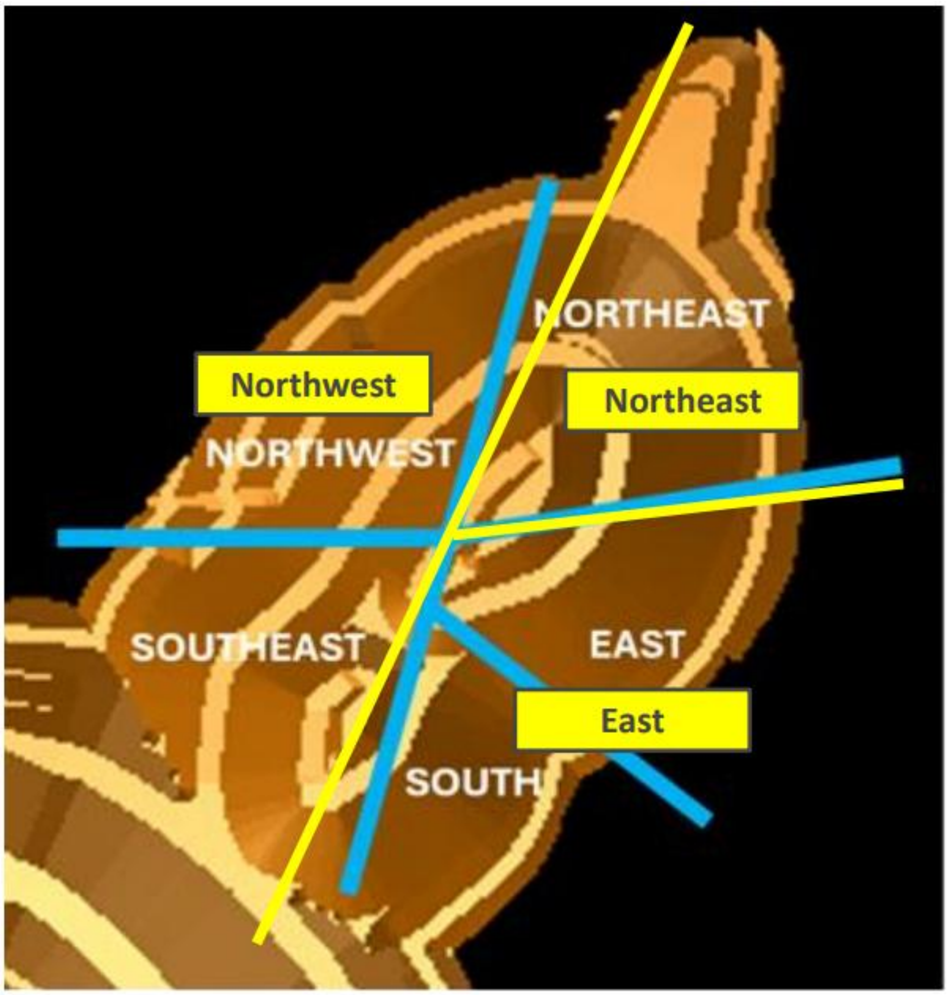


Figure 73 Simplified Batman Pit Parameters Applied to North-East Extension, Mining Plus 2025

Zone	Slope Zone	Weathering	Bench Height (m)	Berm Width (m)	Batter Angle (deg)	IRA (deg)	OSA (deg)
1	North-East	Fresh	24	0	45.0	45.0	38.0
2	East	Fresh	24	10	60.0	46.4	39.1
3	West	Fresh	24	10	65.0	49.9	42.8
4	North-East	Oxide/Trans	24	12	40.0	31.9	31.9
5	Remaining	Oxide/Trans	24	12	50.0	38.5	38.5

Table 85 Northeast Pit Design Berm-Batter

Figure 74 illustrates application of the slope zone sectors to the block model, as used with the North-East pit extensions to the Batman deposit.

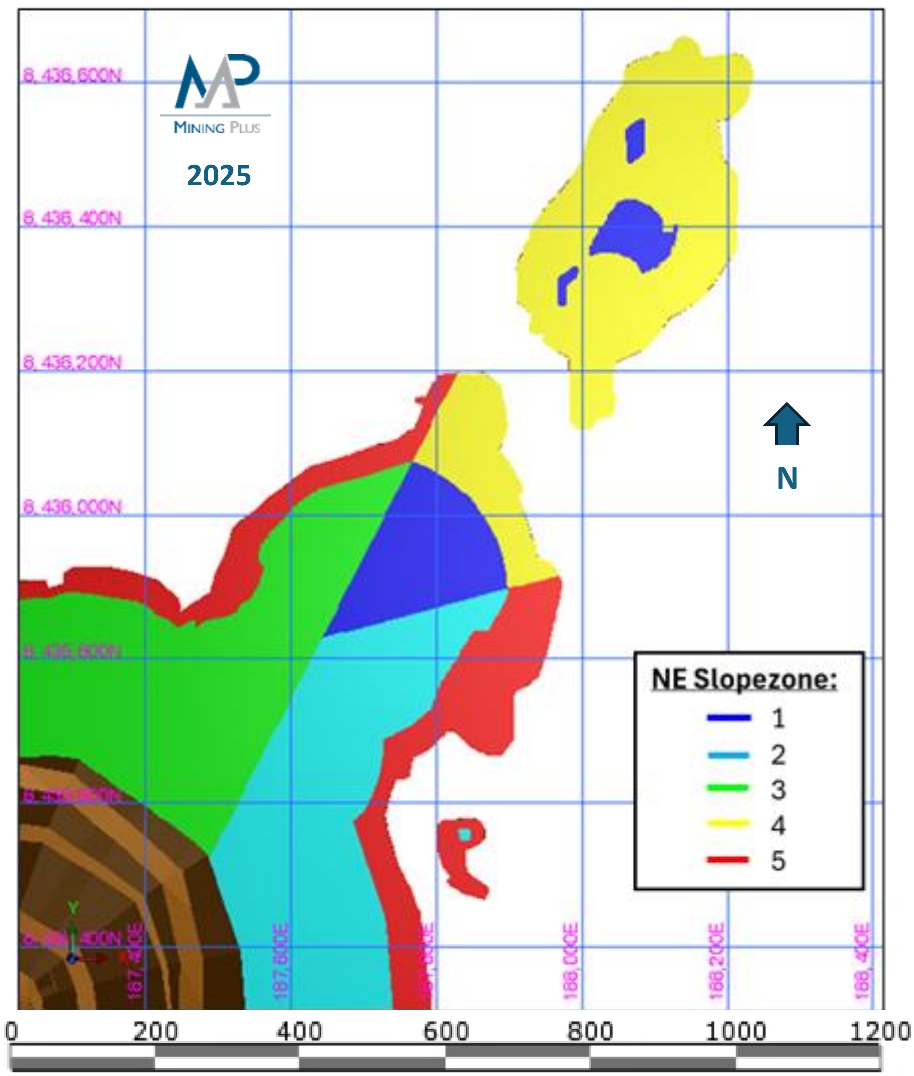


Figure 74 Slope Zones Applied Block Model for Design to the Northeast Extension

Stage designs were developed to closely conform to the selected Whittle pit shells derived from economic pit optimization analyses. The objective was to preserve the economic integrity of the Whittle outputs while translating them into practical, mineable geometries, with close consideration of mining widths to facilitate efficient mining operations by the mining contractor. It should be noted positive feedback was provided by the Tier 1 mining contractor regarding the stage designs, including the final pit, and the interaction of pushbacks to promote efficient mining.

This design process involved aligning stage boundaries with the nested shell contours to the greatest extent possible, while incorporating necessary modifications to accommodate geotechnical constraints, operational access, and equipment maneuverability. The resulting stage designs maintain the strategic intent of the Whittle optimization, ensuring that material movement and phase sequencing remain economically and operationally viable throughout the life of mine. Images of the created stage designs are shown Figure 75 to Figure 80.

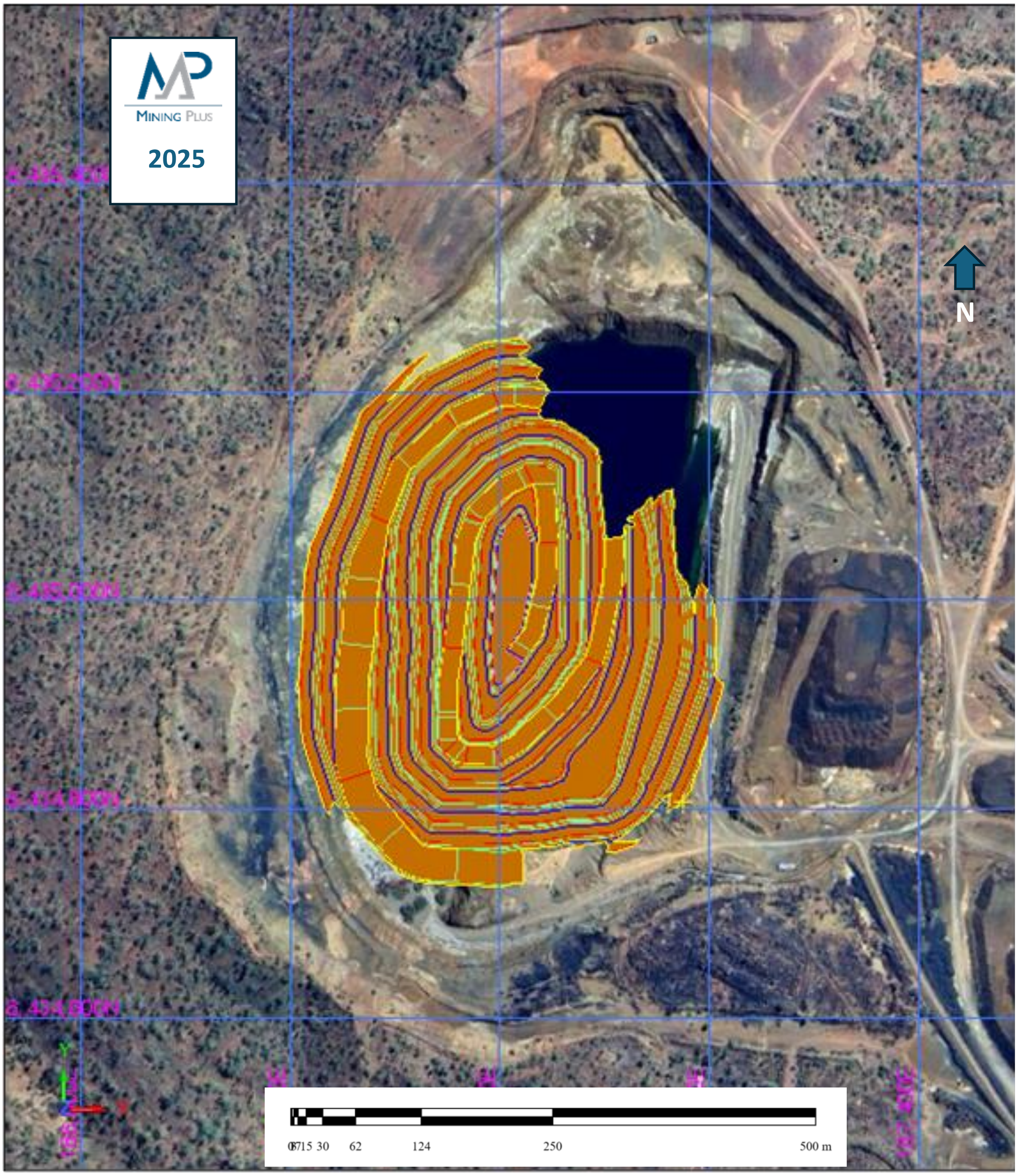


Figure 75 Stage 1 Design

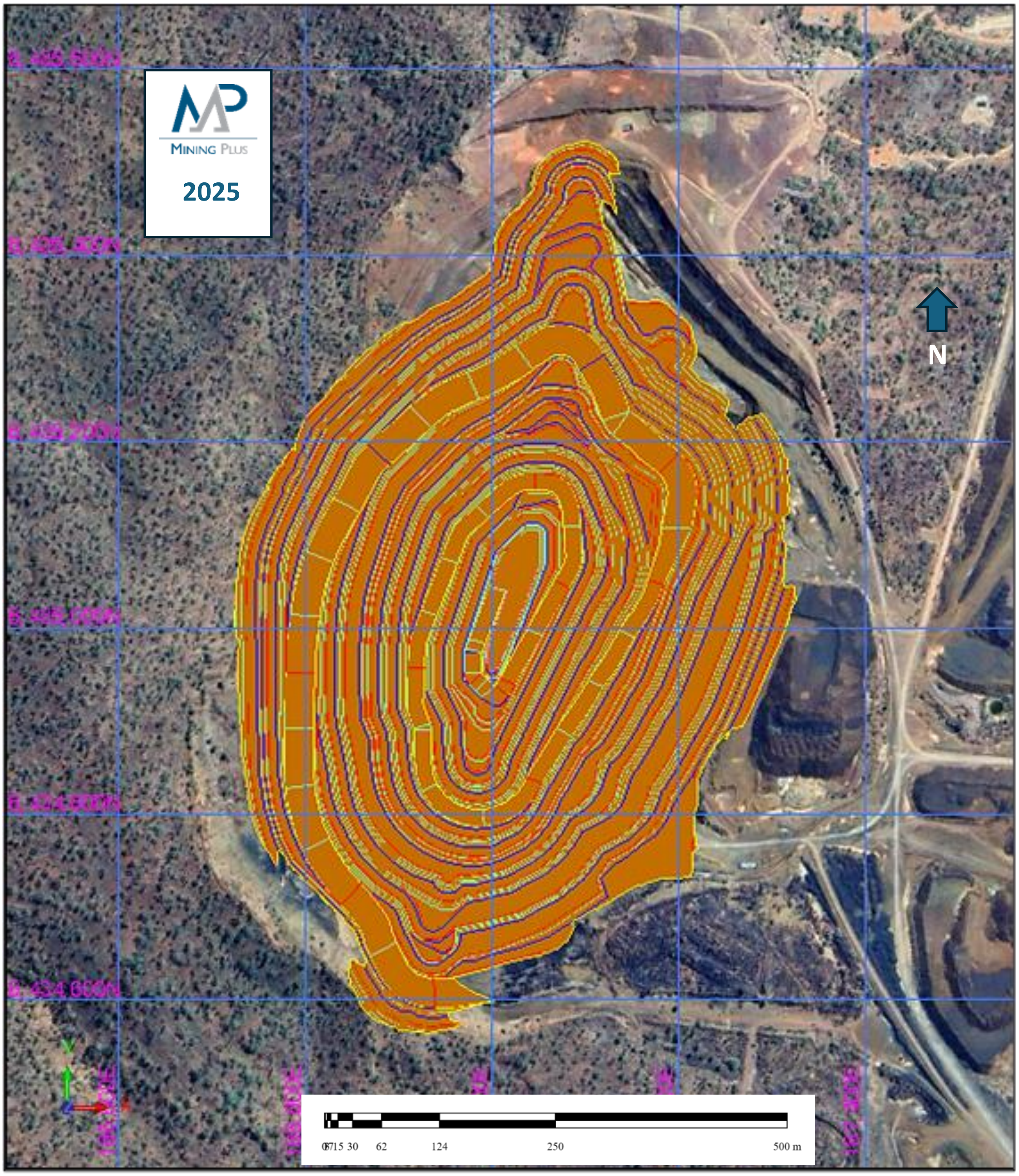


Figure 76 Stage 2 Design

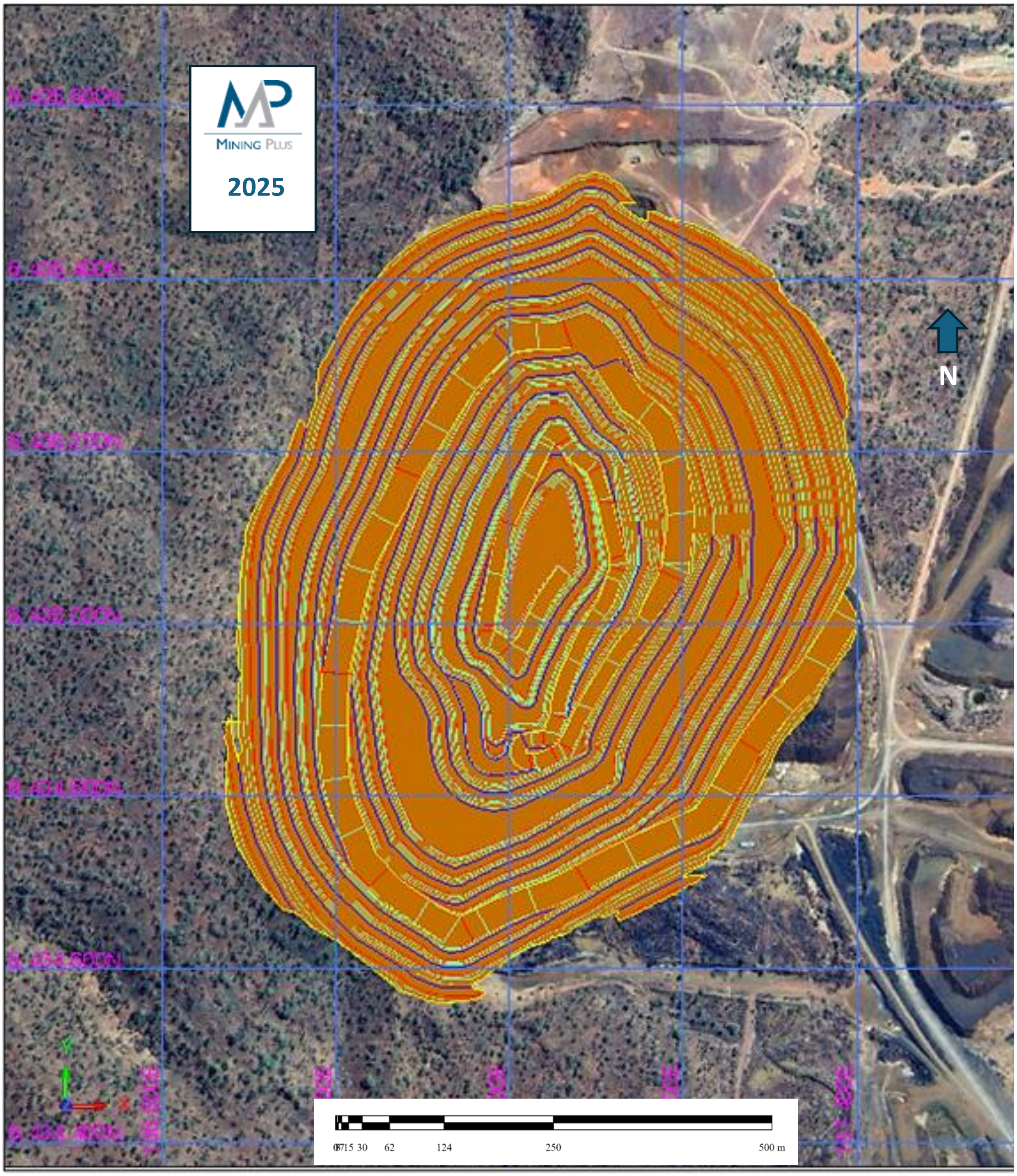


Figure 77 Stage 3 Design

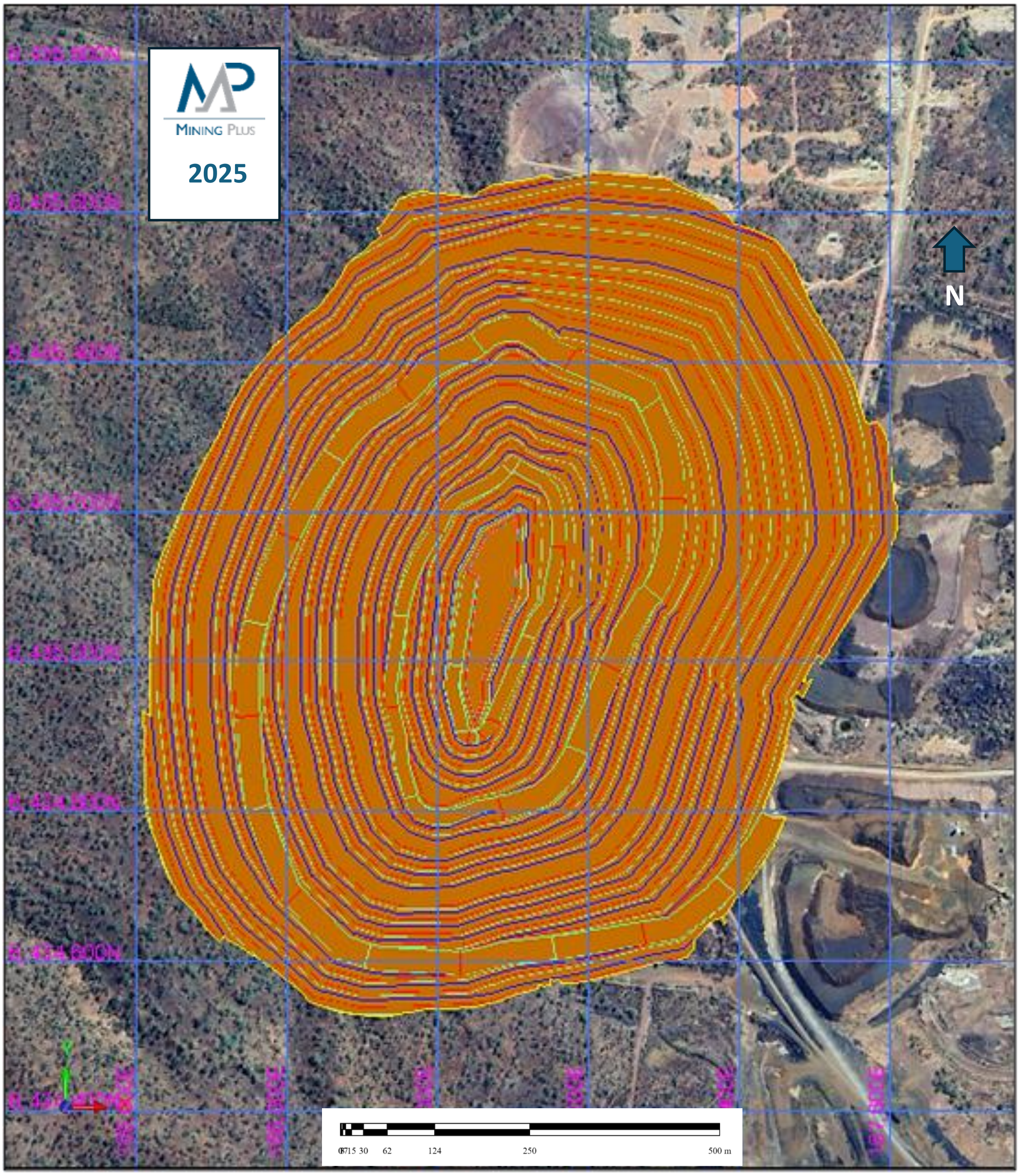


Figure 78 Stage 4 Design

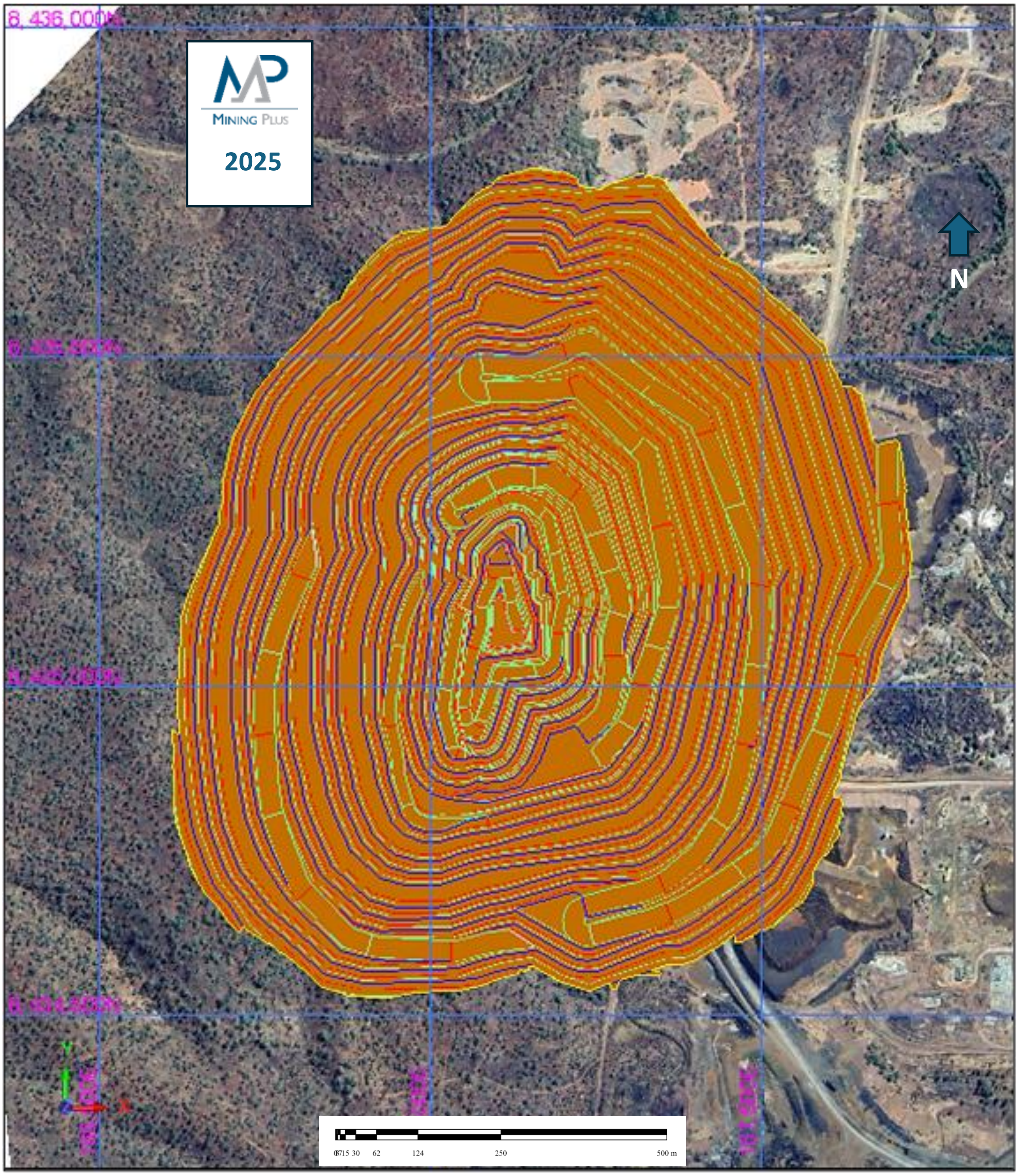


Figure 79 Stage 5 Design

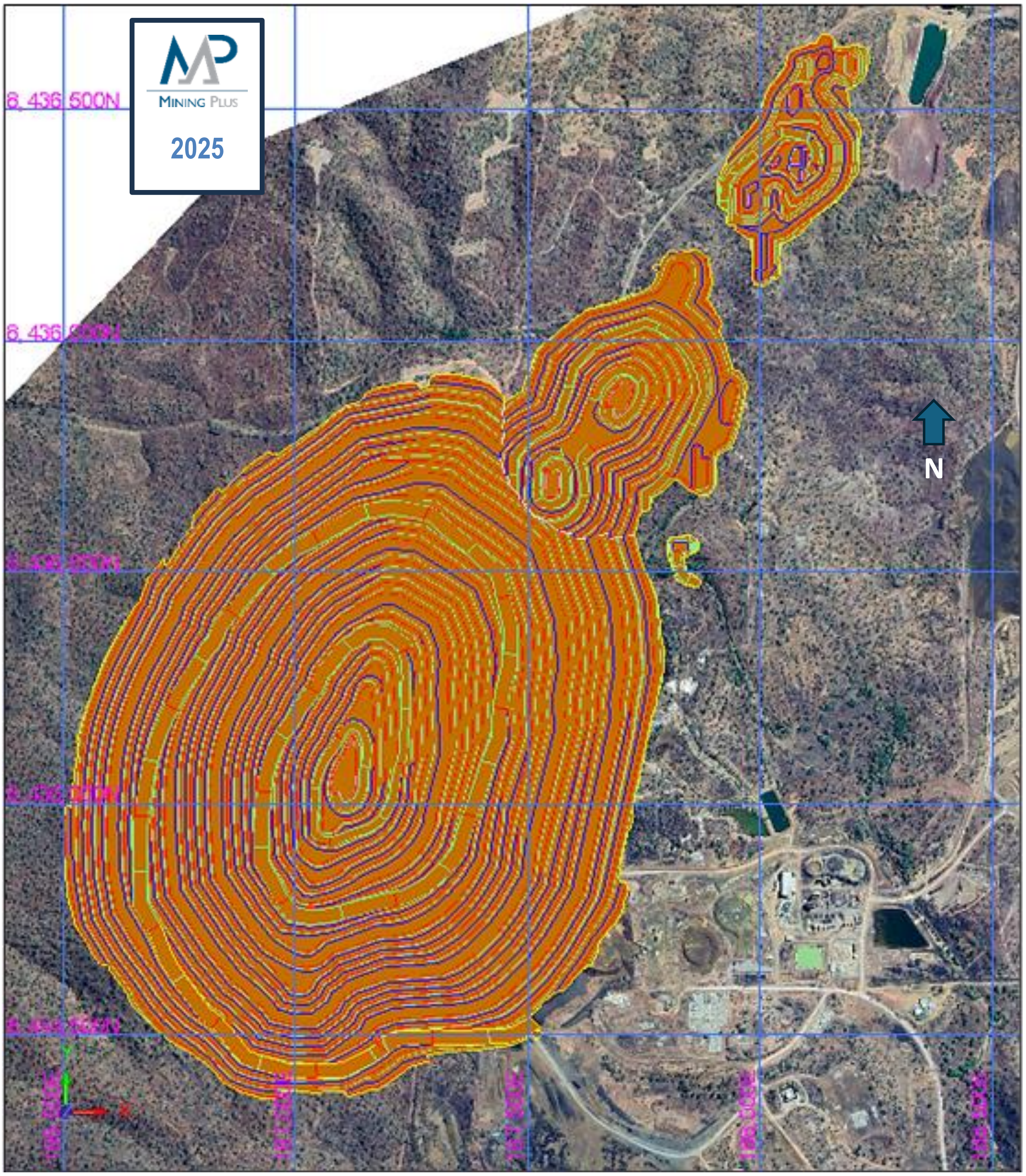


Figure 80 Final Pit Design (Includes Stages 6, 7 and 8)

Table 86 lists the inventory of the individual stage designs with a comparison against their associated Whittle shells shown in Table 85. Please note due to geometric approximations in different mine software and rounding these numbers may varying to other inventories provided in this Technical Report Summary including those of the stated Mineral Reserve.

Stage	Waste (t)	Ore (t)	Total Material (t)	Ore Grade (g Au /t)	Contained Au (oz)	Strip Ratio	Ore Feed (years)
1	5,959,133	8,143,989	14,103,122	1.32	346,122	0.73	1.5
2	26,215,280	21,335,229	47,550,509	1.14	784,258	1.23	4.0
3	67,819,405	36,282,216	104,101,621	1.05	1,225,777	1.87	6.8
4	199,132,597	78,599,450	277,732,047	1.02	2,574,182	2.53	14.8
5	310,935,272	101,062,466	411,997,738	0.99	3,201,343	3.08	19.0
Final Pit	631,539,188	158,738,593	790,277,781	0.97	4,964,596	3.98	29.8

Table 86 Stage Design Inventories

Stage	Stg1	Stg2	Stg3	Stg4	Stg5	Final Pit (6,7,8)
Whittle Shell	14	29	36	43	60	70
Ore Tonnes (Δ%)	-5%	-12%	-8%	0%	-7%	1%
Waste Tonnes (Δ%)	-14%	26%	24%	4%	-4%	4%
Total Tonnes (Δ%)	-9%	5%	10%	3%	-4%	4%
Strip Ratio (Δ)	-0.07	0.37	0.49	0.09	0.10	0.12
Au grade (Δ)	-0.01	0.13	0.09	0.00	-0.01	0.00
Au grade (Δ%)	0%	13%	9%	0%	-1%	0%
Au Oz (Δ)	-20,376	-4,417	3,194	17,093	-259,397	63,481
Au Oz (Δ%)	-6%	-1%	0%	1%	-7%	1%

Table 87 Stage Design vs Whittle Shell Comparison

Table 86 demonstrates the consistent cumulative increase in size of the pushbacks up to the final pit outline, expanding the mining faces as the pit progresses. Table 87 shows conformity between the optimized interim pit shells and the designed interim pit outlines; any variances within the interim outlines is compensated in the next incremental pit outline. The variances between the final optimized shell and design are within the industry standard of 5%. The increase of total mining of 4% results in minimal impact on the gold grade and strip ratio, and a 63 koz increase in contained gold mined (1%).

13.5 Mining Phase Design & Selection

Staged pit designs were converted into three-dimensional solids to facilitate detailed mine scheduling. Each stage solid was sequentially clipped against the preceding stages to isolate only the remaining schedulable inventory, thereby eliminating overlap and ensuring accurate volumetric representation. These refined scheduling solids were then spatially encoded into the geological block model using stage and bench-level identifiers. This coding enabled the scheduling software to distinguish material by stage and elevation on a block-by-block basis, allowing for precise control over phase sequencing and production targeting.

The final pit design delineated four geologically and spatially distinct mining zones, each of which could be developed and mined independently. To enhance operational flexibility and optimize sequencing during mine scheduling, these zones were individually coded as discrete mining stages—Stages 6, 7and 8.

Figure 81 illustrates the spatial configuration of the defined pit phases. Notably, Stage 7 intersects with Stage 6, resulting in concurrent mining of the upper benches of both stages in the final schedule to optimize equipment utilization and access.

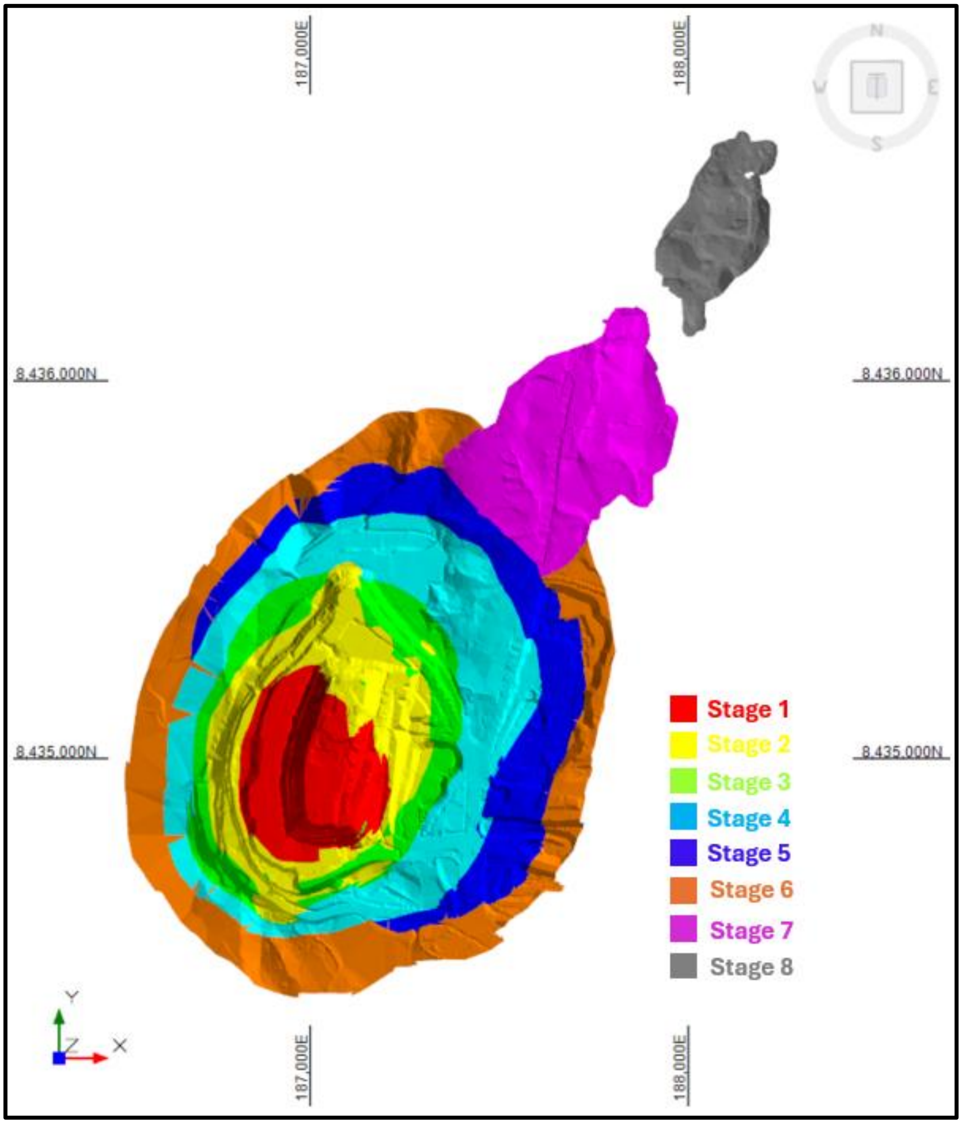


Figure 81 Pit Phases Applied to Block Model for Scheduling, Mining Plus 2025

13.6 Mine Production Scheduling Criteria

Multiple schedule scenarios were undertaken in Geovia's MineSched software.

MineSched uses a target-driven, constraint-based scheduling logic to create practical, executable schedules. MineSched applies heuristic algorithms to generate schedules that meet specific targets while respecting the defined constraints.

Key targets and constraints used for scheduling were:

- Spatial Relationships.
- Production Rates.
- Mill Feed Tonnes.
- Mill Feed Grades.
- Stockpile Capacities.

To accurately schedule discrete parcels, each block was allocated a material code based on characteristics highlighted in Table 88 and Table 89. Table 90 provides an indication of the different classes of waste to be dumped on the waste dumps; waste material with gold grades below the established cut-off threshold of 0.50 g Au/t yet exceeding a lower established bound of 0.37 g Au/t, has been classified as Mineralized Waste. Although this material contains domains that have been categorized within the Mineral Resource framework as Measured and Indicated, it has not been converted into Mineral Reserves due to current economic constraints and lack of demonstrated feasibility for profitable extraction under prevailing market conditions.

This sub-economic material has been scheduled for deposition in the designated WRD. However, in alignment with best practices for Mineral Resource stewardship and future value optimization, it will be systematically segregated from non-mineralized waste. This strategic separation facilitates potential re-evaluation and reprocessing should future changes in commodity pricing, metallurgical recovery, Company's strategy or operational cost structures render the material economically viable. The segregation protocol will include appropriate geospatial tracking and material characterization to ensure traceability and facilitate future Mineral Resource reconciliation.

It is estimated that approximately 62.7 Mt of Mineralized Waste falls in the Measured and Indicated category with an average grade of 0.43 g Au/t for a total of 868 koz gold contained. This material is deemed uneconomic for this Technical Report Summary and has not been included in the Mineral Reserve estimates.

13.6.1 Mining Dilution and Recovery

The assessment of potential mining dilution and recovery are addressed in Section 12.4 of this Technical Report Summary.

Material	Grade Bin		Weathering	Mineral Resource Classification
Ore	Mineralized Waste	(g Au /t) 0.37 - 0.5	Oxide	Measured
	Low Grade	(g Au /t) 0.5 - 0.7	Transitional	Indicated
	Medium Grade	(g Au /t) 0.7 - 1.0	Fresh	Inferred
	High Grade	(g Au /t) 1.0+		

Table 88 Ore Material Code Designations

For scheduling purposes all mineralized waste and Inferred material was sent to the waste dump.

Material	Classification	
Waste	Non-Acid Forming (NAF)	S (%) < 0.25
	Potentially Acid Forming 1 (PAF1)	S (%) 0.25-0.4
	Potentially Acid Forming 2 (PAF2)	S (%) > 0.4
	Unclassified	S (%) -

Table 89 Waste Material Code Designations

Class	Grade Band g Au/t	AMD Class	Weathering	Tonnes Mt	Grade g Au /t
Waste	0-0.37	NAF	-	211.8	0.08
	-	PAF1	-	79.1	0.15
	-	PAF2	-	165.8	0.20
	-	UNC	-	89.8	0.03
Measured-Indicated	0.37-0.5	NAF	Oxide	1.5	0.43
(Min Waste)	-	-	Transitional	0.6	0.42
	-	-	Fresh	3.5	0.42
	-	PAF1	Oxide	0.4	0.43
	-	-	Transitional	0.2	0.42
	-	-	Fresh	7.2	0.43
	-	PAF2	Oxide	0.8	0.43
	-	-	Transitional	0.8	0.43
	-	-	Fresh	47.1	0.43
	-	Unclassified	Oxide	0.3	0.42
	-	-	Transitional	0.0	0.42
	-	-	Fresh	0.2	0.42
Inferred	>0.37	NAF	Oxide	0.3	0.65
(HG, MG, LG, MW)	-	-	Transitional	0.1	0.71
	-	-	Fresh	1.5	0.63
	-	PAF1	Oxide	0.1	0.57
			Transitional	0.0	0.74

Class	Grade Band g Au/t	AMD Class	Weathering	Tonnes Mt	Grade g Au /t
			Fresh	4.0	0.59
		PAF2	Oxide	0.1	0.53
			Transitional	0.1	0.53
			Fresh	15.4	0.71
		Unclassified	Oxide	0.3	0.58
			Transitional	0.0	0.52
			Fresh	0.4	0.57
Total				**631.5**	**0.17**

Table 90 Indicative Inventory of Waste

A large number of blocks within the provided resource block model had blank or negative sulfur grades. In the engineering block model these values were set to zero and the material categorized as Unclassified Waste (UNC).

Material Movement was modeled by first creating source and destination nodes (Stages, Stockpiles, Waste Dumps and Mill) and then allocating which material types could be moved between the different combinations of sources and destinations. The complete material flow diagram is shown in Figure 82.

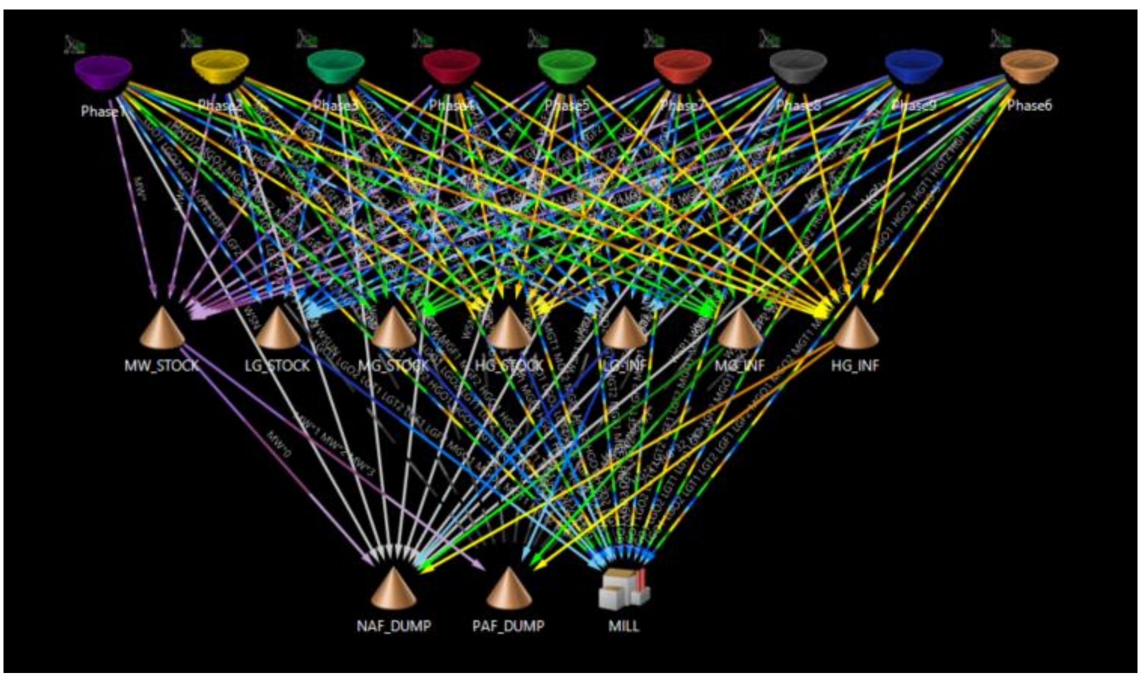


Figure 82 Mt Todd Movement Allocation by Material Code, Mining Plus 2025

Stage sequencing of the main pit was guided by spatial relationships which ensured no undermining of material. In practical terms this meant at any point in time Stage 2 could not mine below Stage 1, Stage 3 could not mine below Stage 2, and so forth.

The schedule was set to begin mining operations in Year 1. The starting month varied for different scenarios tested. Table 91 shows the period durations used for the schedule.

Year	Scheduling Periods
Year 1- Year 2	Monthly
–Year 3- Year 5	Quarterly
Year 5 to end LOM	Annually

Table 91 Scheduling Periods

Ex-pit productive movement capacity was determined based on excavator productivity rates provided by the contractor. The contractor estimated two 400 t class excavators each able to move 16 Mtp for a maximum ex-pit material movement of 32 Mtpa, based on average dig rates of around 1040 bcm/60 min hour (2955 tphr) in waste and 940 bcm/60 min hr (2650 tphr) in ore.

The truck fleet haul speeds were restricted by the following conditions:

- 20 km/hr downhill when loaded, 30 km/hr downhill when empty.
- Dump max speed 40 km/hr.
- Bench max speed 30 km/hr.
- Dig face max speed 20 km/hr.
- Corners 20 km/hr.
- Switchbacks 15 km/hr.
- Rimpull truck capability restrictions on all other gradients and rolling resistances.

To find the most optimal mine schedule, a set of several mine schedule scenarios were run at a targeted a mill feed rate of 15 ktpd. The start date of processing varied by scenario however each scenario adhered to a ramp up production profile as indicated by GRES as shown in Table 92.

Month	1	2	3	4	5	6	7	8	9
tph (% of 15 kt/day)	24%	50%	73%	88%	95%	97%	98%	99%	100%

Table 92 Processing Ramp Up Profile

Gold was the only element in the schedule with targeted grades. Mill feed gold grade targets varied by scenario, however, the logic generally aimed to achieve the higher grades at the start of the schedule, slowly decreasing over time. This led to lower grade material being stockpiled and only reclaimed towards the end of the schedule. Silver (Ag ppm), Copper (Cu%) and Sulfur (S%) grades were reported as outputs to the schedules but are not considered of economic value for the Project.

There was no fixed stockpile capacity for the scenarios tested. Some scenarios were run fully unconstrained whereas other scenarios aimed to minimize total stockpile balances.

13.6.2 Mine Production Scheduling

From the series of iterative mine scheduling scenarios were developed and evaluated prior to selecting the final optimized mine schedule. Each successive iteration incorporated incremental refinements, allowing for controlled assessment of individual parameter impacts while maintaining continuity across scenarios.

Initial scheduling scenarios assumed simultaneous commencement of mining and processing operations in Month 1, year 1. However, subsequent iterations decoupled these activities, with mining initiating in January and processing commencing in Month 4, year 1 to accommodate infrastructure construction sequencing and plant commissioning timelines.

Gold grade targeting served as the primary lever for scenario differentiation. Some scenarios employed narrow grade bands to maintain consistent mill feed grades, while others adopted more flexible grade constraints to maximize early-period gold recovery. Across all scenarios, the overarching objective was to optimize Net Present Value (NPV) by prioritizing the processing of higher-grade ore in the initial years, followed by a gradual decline in grade over the life of mine.

Various mining ramp-up profiles were analyzed to ensure reliable delivery of mill feed tonnage. Early iterations focused solely on achieving throughput targets, while later scenarios incorporated additional constraints to ensure sufficient availability of clean Non-Acid Forming (NAF) waste rock for use in civil construction activities. The final scenario assumed a staggered equipment deployment strategy, with one primary excavator mobilized ahead of the second, resulting in a ramp-up profile at 50% of peak mining capacity. Two months at 50% of peak mining capacity are assumed to allow the site assembly and commissioning of the 2nd primary excavator and its associated trucks and support equipment over two months.

Ramp-down strategies were similarly evaluated, with the primary objective of managing stockpile inventories during the latter stages of the schedule. Initial models featured a gradual reduction in mining rates, whereas later iterations assumed the demobilization of one excavator, effectively halving production capacity during the final years leading to pit depletion.

Both the ramp-up and ramp down strategies are aligned to the proposed planned production rates of the primary excavators provided by the mining contractor, which can individually achieve 50% of the in-pit production.

The final mine schedule and cost estimate for this Technical Report Summary assumes the following ramp-up at the commencement of mining:

- Pre-production period in year -1 to allow construction of mine infrastructure to be ready for commencement of mining in Jan of year 1.
- First two months (Month 1 and 2, Year 1) planned production at 50% rate which equates to mobilization of one large hydraulic excavator and required trucks and support equipment.
- From the third month (Month 3, year 1), planned production lifts to full 100% rate which equates to the mobilization of the second large hydraulic excavator and required trucks and support equipment.
- During month 4 (Month 4, year 1) feed commences to the processing plant. This allows three months of mining to be completed to ensure adequate feed is available and construction of ROM pad also.

Stockpile balances were primarily influenced by production rates and grade targeting strategies. Across the various scenarios, peak stockpile volumes ranged from 15 Mt to 26 Mt. Final iterations focused on minimizing total stockpile accumulation to reduce rehandling requirements and optimize material flow. The maximum total stockpile capacity required in the final mine schedule is 17 Mt.

The following results present the outputs associated with the selected scheduling scenario, which was prioritized based on its optimized grade distribution, consistent mill throughput, practical ramp-up and ramp-down execution, and effective management of stockpile inventories.

Figure 83 illustrates the ex-pit movement by stage highlighting a consistent maximum production rate of 32 Mtpa with a ramp down at half capacity.

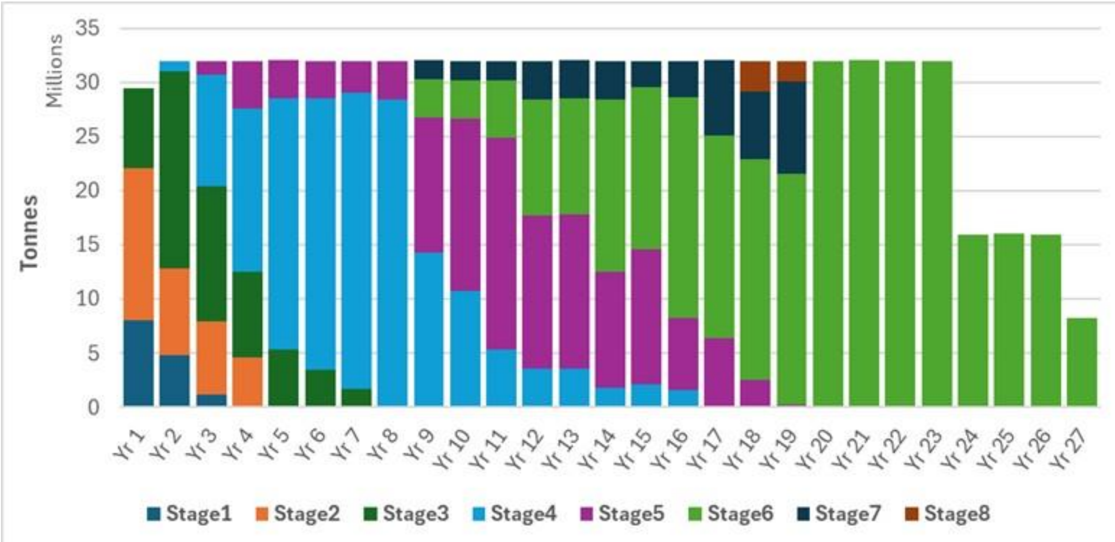


Figure 83 Material Movement by Stage, Mining Plus 2025

Sink rates were evaluated to ensure the feasibility of the proposed mining schedule. Figure 84 illustrates the number of 12 m benches initiated annually by mining stage. The data indicates that Year 1 represents the most intensive development period, with both Stage 1 and Stage 2 initiating eight benches each. This elevated activity level aligns with expectations, as initial cutback benches typically involve lower material volumes. The scheduled sink rates are considered operationally viable when benchmarked against industry standards, which generally assume a development rate of one bench per month. In subsequent years, the sink rates decrease substantially, remaining well within the capacity of the allocated mining fleet.

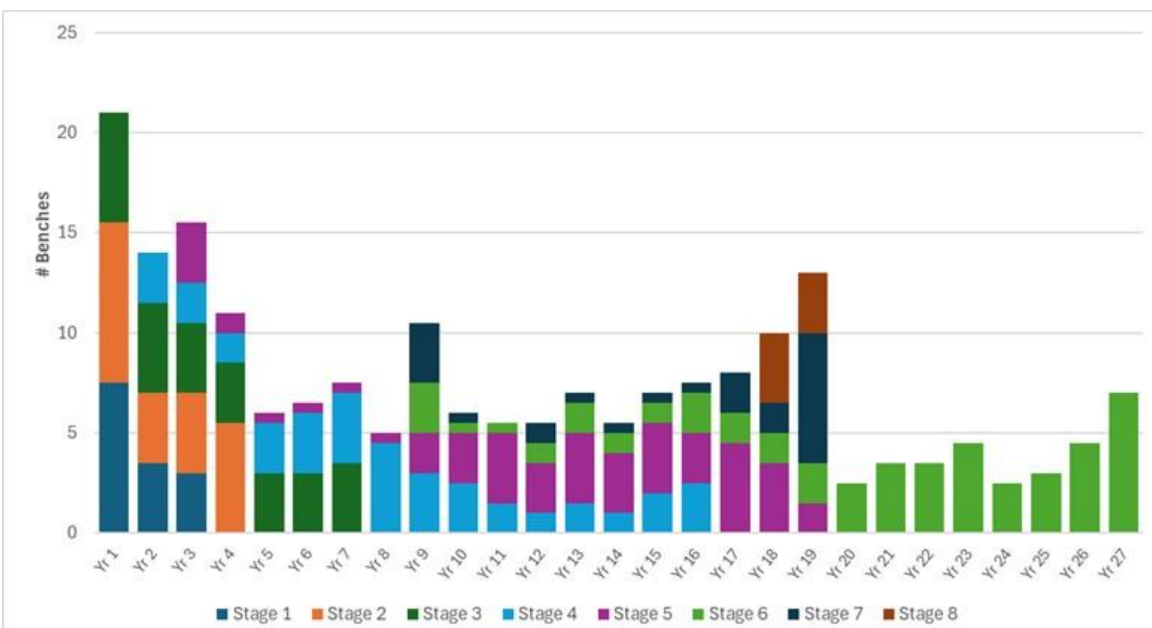


Figure 84 Sink Rates, Mining Plus 2025

Figure 85 shows the ore and waste movement by period highlighting a relatively consistent stripping ratio for a majority of the Project. This consistency is impacted only towards the end of the schedule when Stage 6 is the only area being mined.

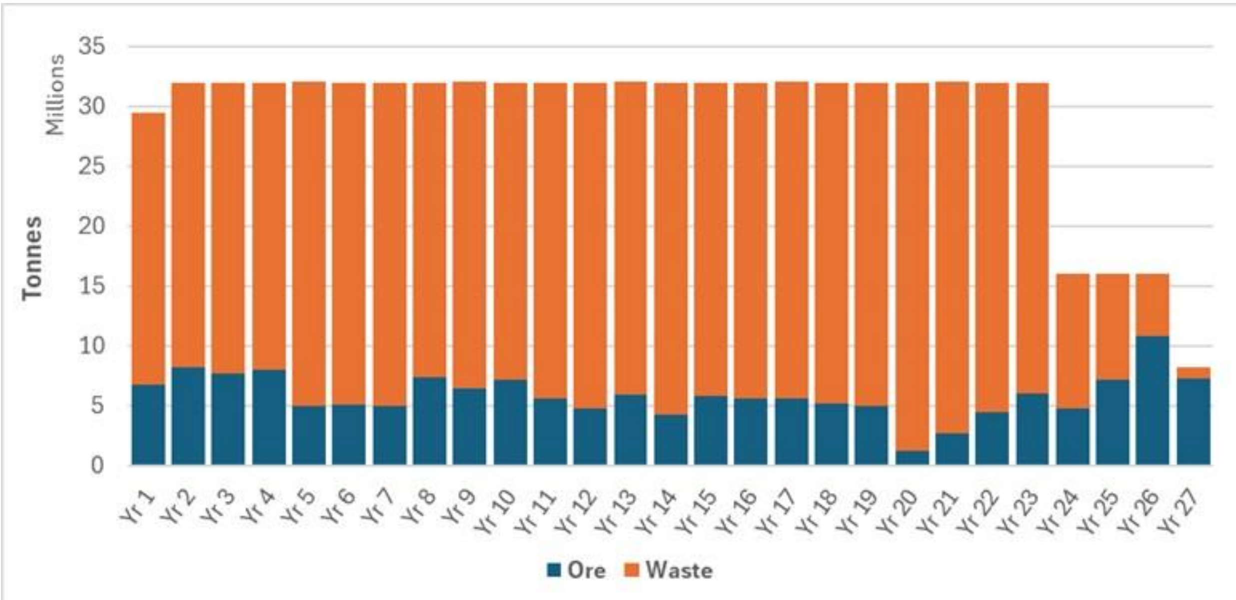


Figure 85 Ore and Waste Movement, Mining Plus 2025

The mining ramp-up profile commenced in Month 1, year 1 and consisted of the first two months production operating at 50% capacity, the equivalent to one excavator, with the second excavator being commissioned in Month 3, year 1. The mining production profile by material type for the first year of operations is shown in Figure 86.

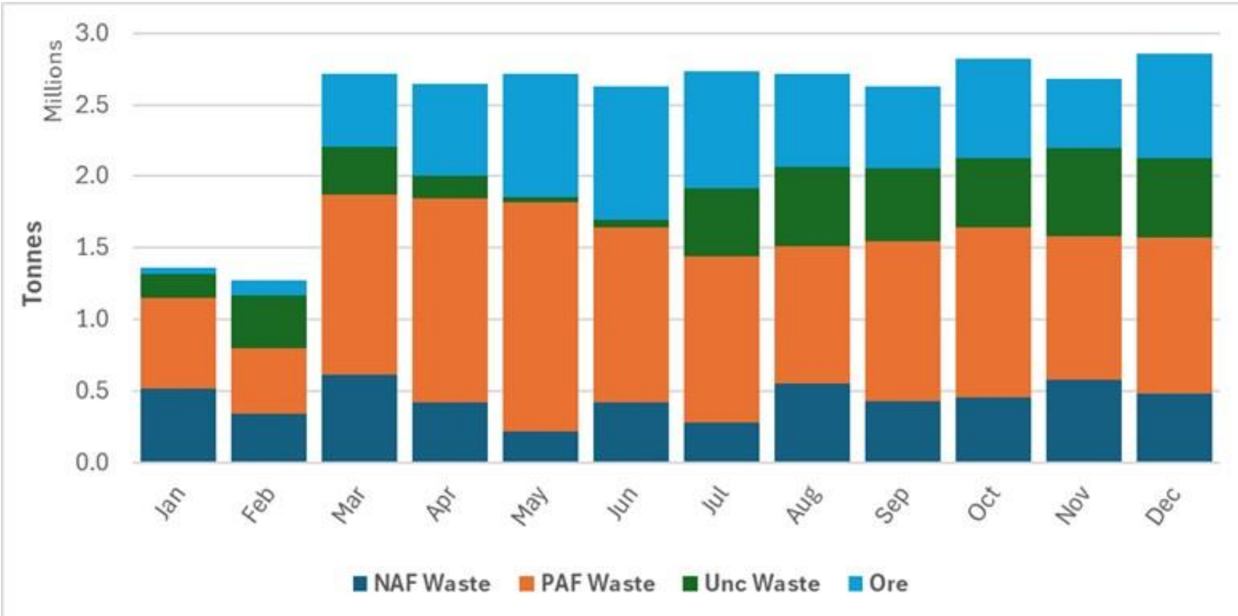


Figure 86 Mining Production Profile by Material Type, Year 1, Mining Plus 2025

In addition to ensuring sufficient ore feed material at grade was produced, a secondary key parameter in scheduling the first year of production was ensuring that enough clean waste (NAF waste) was produced to meet civil construction needs. Civil construction works requiring clean waste in the first year were Stage 2 of TSF1 construction, the base of the low-grade stockpile, the base of the ROM pad and a proportion of the remaining ROM pad (in addition to LG material used in ROM construction). Table 93 shows the production and usage of NAF waste material for the first year of the Project. NAF waste produced in proceeding years will be used for further TSF construction requirements and on the waste dump for encapsulating PAF waste material.

Date	NAF Produced (Kt)	NAF Used for Construction (Kt)			
		TSF1 (Stg2)	ROM Pad Base	ROM Pad Top	LG Stock Base
Month 1, Y1	515	-	515	-	-
Month 2, Y1	342	-	273	-	69
Month 3, Y1	616	365	-	-	251
Month 4, Y1	414	-	-	-	414
Month 5, Y1	215	-	-	-	215
Month 6, Y1	419	-	-	-	419
Month 7, Y1	280	-	-	138	142
Month 8, Y1	552	-	-	552	-
Month 9, Y1	423	142	-	281	-
Month 10, Y1	455	455	-	-	-

Date	NAF Produced (Kt)	NAF Used for Construction (Kt)			
		TSF1 (Stg2)	ROM Pad Base	ROM Pad Top	LG Stock Base
Month 11, Y1	579	579	-	-	-
Month 12, Y1	481	334	-	-	-
Total	5,291	1,875	788	971	1,510

Table 93 NAF Production and Usage, Year 1

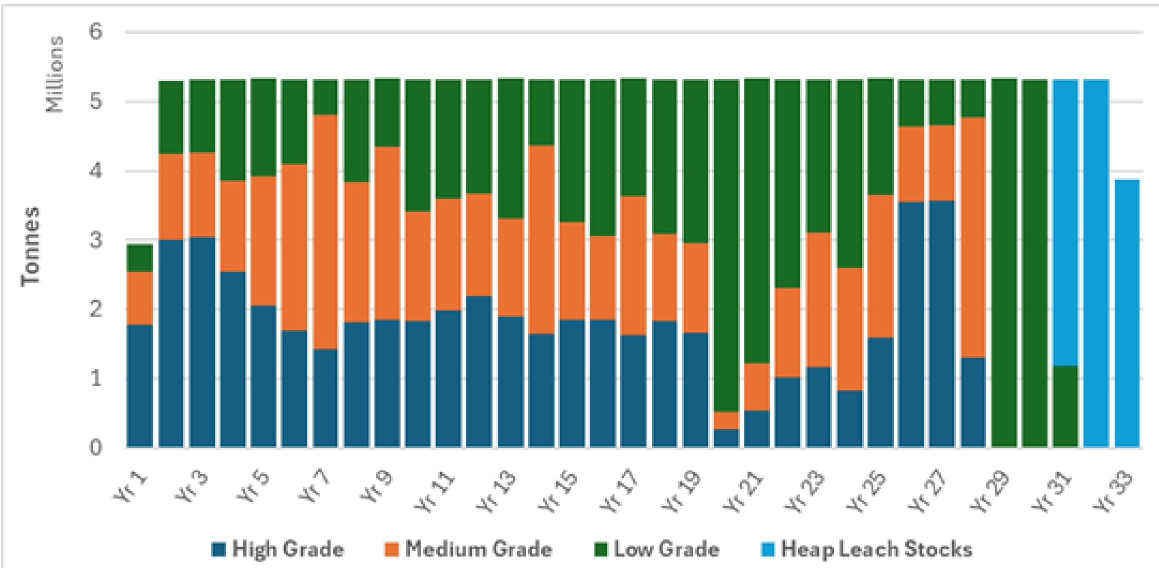


Figure 87 Ore Feed Tonnes, Mining Plus 2025

Processing was scheduled to commence in month four of Year 1, of the schedule. The processing ramp-up followed the profile previously stated in Table 92. Figure 87 shows the ore feed tonnes by year clearly highlighting the consistent feed rate equivalent to 15 ktpd (5.3 Mtpa) is achieved for the duration of the Project. The chart also shows the inclusion of legacy Heap Leach Pad material at the end of the Project which was added after the MineSched works. Details of the Heap Leach Pad material and its Mineral Reserve are provided in Section 12.

Ore feed gold grade (g Au/t) served as the primary scheduling parameter, with the objective of maintaining grade consistency while front-loading higher-grade material to enhance early Project economics. The scheduling strategy was designed to deliver a sustained elevated feed grade during the initial production years, followed by a stable long-term average.

As illustrated in Figure 88, the mine schedule achieves an average feed grade of approximately 1.2 g Au/t over the first four years, transitioning to a consistent grade of around 1.0 g Au/t for the subsequent 14-year period. Grade variability emerges in Year 18, coinciding with mining activities being confined to Stage 6, which lacks sufficient high-grade ore to meet mill requirements. This necessitates the rehandling and processing of previously stockpiled low-grade (LG) material to supplement mill feed.

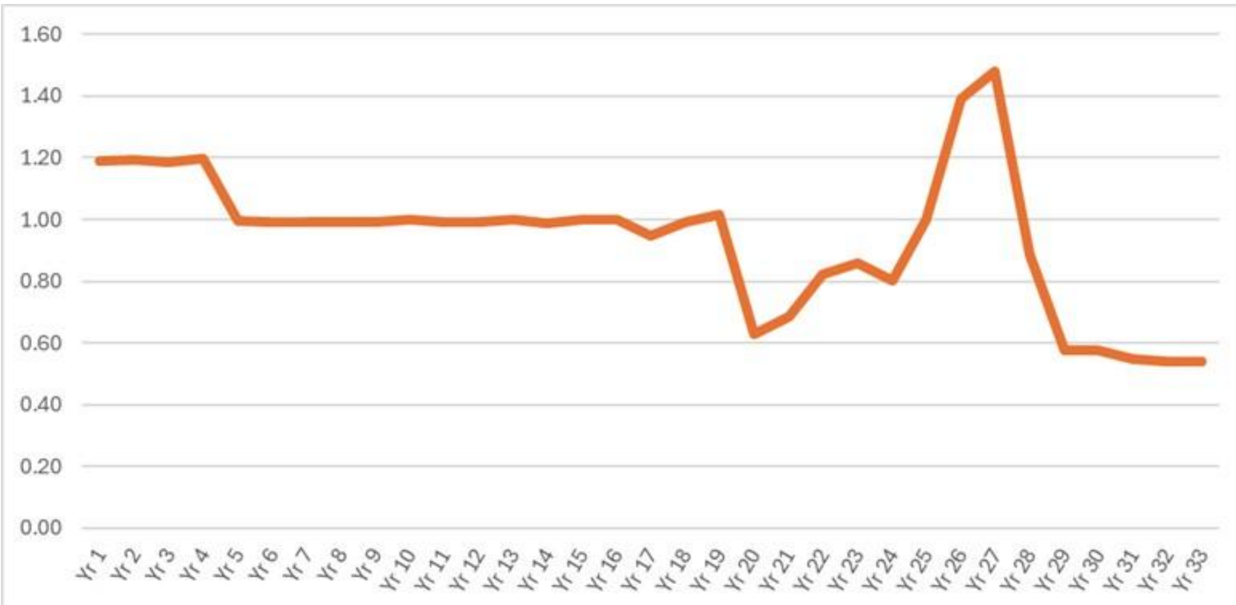


Figure 88 Gold Feed Grade (g Au/t), Mining Plus 2025

Although silver (g Ag/t), copper (Cu %) and sulfur (S %) grades were not targeted during scheduling they were considered, tracked and reported. Figure 89 shows the scheduled mill feed grades for silver, copper and sulfur. The reporting of the grades ends when the mining stockpiles are depleted and do not include grades from the legacy Heap Leach Pad as only gold grade data was available for this material.

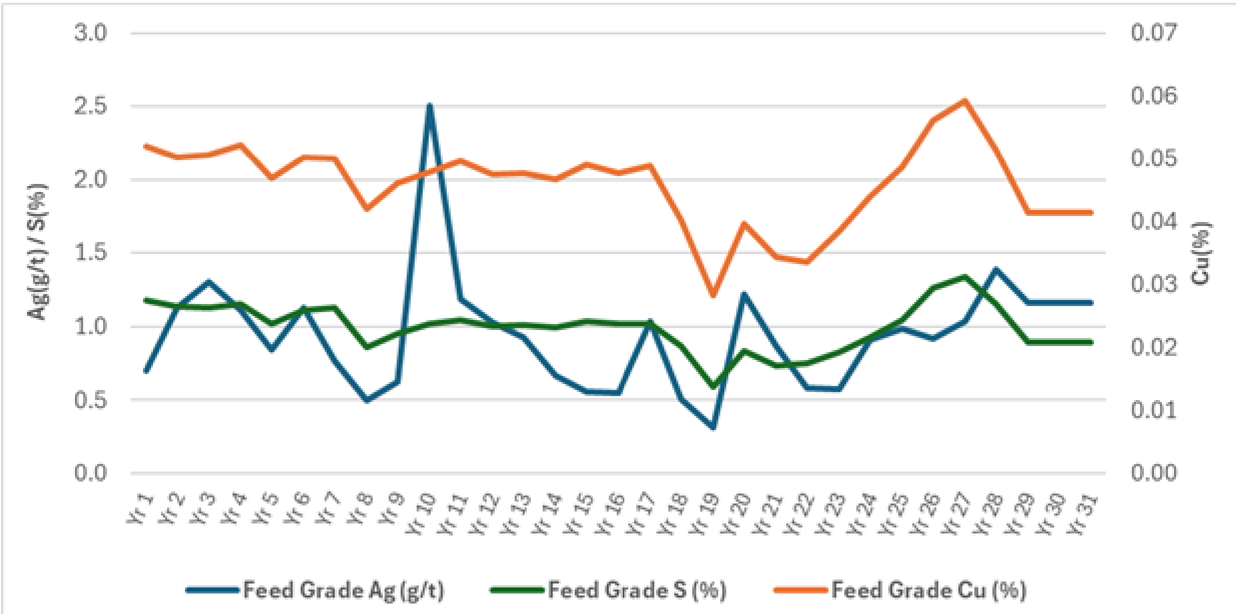


Figure 89 Other Feed Grades, Mining Plus 2025

Stockpile management throughout the Project was governed by maintaining a consistent overall stripping ratio, ensuring a balanced progression of ore and waste movement. Strategic front-loading of higher-grade mill feed in the early phases of the schedule resulted in the accumulation of low-grade (LG) ore, which was deferred for processing until later in the mine life.

A notable increase in total stockpile inventory toward the latter part of the schedule reflects surplus ore generation, even with the mining ramp down (operating at 50% of production capacity). This surplus included high-grade (HG) material exceeding the plant's processing capacity, necessitating the stockpiling of ore across all grade bins (HG, MG and LG).

Throughout operations, LG ore will be systematically placed on a dedicated LG stockpile. MG and HG material will primarily be stored on the Run-of-Mine (ROM) pad adjacent to the processing facility. However, the LG stockpile will also serve as a contingency storage location for MG and HG ore during periods of ROM pad capacity constraints particularly during initial years of mining operations.

Cumulative stockpile balances over the life of mine are illustrated in Figure 90 below. The annual stockpile balance and the amounts of LG, MG, and HG expected each year are shown.

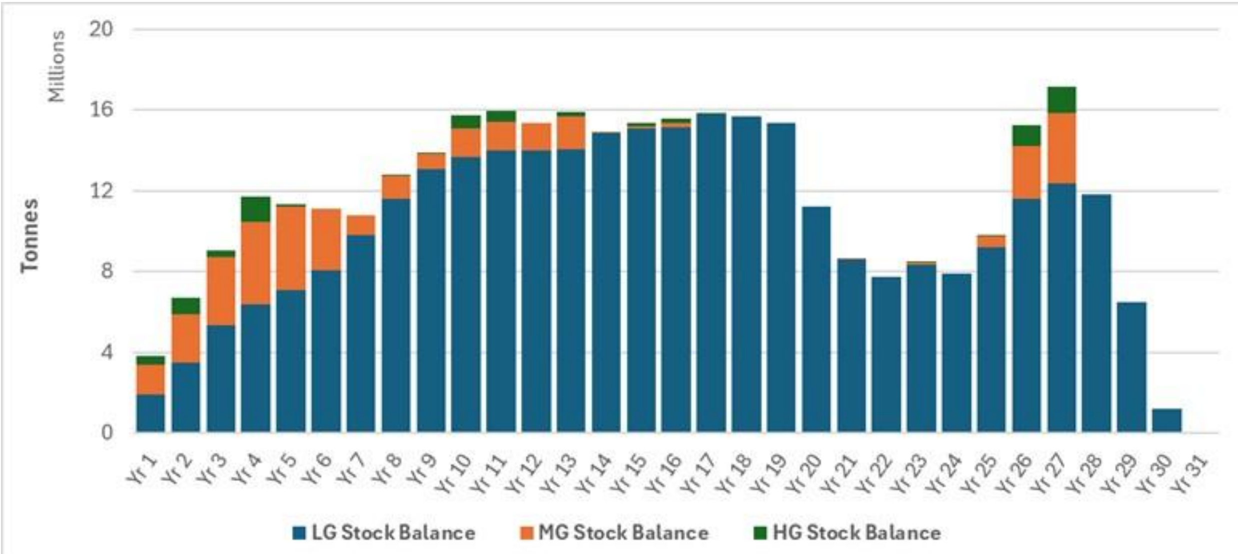


Figure 90 Stockpile Balances, Mining Plus 2025

The summary of the Mine Production Schedule is detailed in Table 94.

VISTA GOLD

GR

Material Movement Summary

MINING PLUS

Feasibility Study - Vista Gold
-

Date: 08-July-2025
Mine Plan: MF12D

Mine Schedule

		1/01/2028	1/02/2028	1/03/2028	1/04/2028	1/05/2028	1/06/2028	1/07/2028	1/08/2028	1/09/2028	1/10/2028	1/11/2028	1/12/2028
EXPIT ORE	HG ORE (t)	18,480	12,780	89,066	165,387	306,829	273,691	258,963	273,016	246,300	183,466	177,738	178,492
	MG ORE (t)	19,495	54,548	199,152	212,690	264,872	362,808	277,720	212,793	162,473	210,075	109,618	198,089
	LG ORE (t)	4,678	33,689	220,909	250,545	295,131	294,814	263,583	170,233	163,829	202,682	143,394	219,422
EXPIT WASTE	NAF (t)	514,961	341,533	615,577	414,477	215,227	419,443	279,631	551,944	423,131	454,856	579,242	480,614
	PAF (t)	497,350	363,495	762,920	1,045,441	1,113,716	718,597	715,813	554,212	721,575	773,131	689,792	730,084
	UNCLASSIFIED (t)	167,217	374,998	334,998	154,629	28,761	55,125	477,292	555,417	516,381	481,060	615,776	551,249
	MINERALISED WASTE (t)	86,151	68,555	299,339	262,284	340,979	324,594	296,367	242,449	262,877	263,954	206,370	282,787
	INFERRED MATERIAL (t)	50,554	21,616	195,808	124,645	152,254	181,027	148,402	157,706	133,532	148,544	108,170	77,031
EXPIT TOTALS	ORE TOTAL (t)	42,652	101,017	509,127	628,622	866,832	931,314	800,265	656,042	572,603	596,223	430,750	596,004
	WASTE TOTAL (t)	1,316,232	1,170,198	2,208,642	2,001,476	1,850,937	1,698,785	1,917,504	2,061,727	2,057,496	2,121,546	2,199,350	2,121,765
	TOTAL MINED (t)	1,358,884	1,271,214	2,717,769	2,630,099	2,717,769	2,630,099	2,717,769	2,717,769	2,630,099	2,717,769	2,630,099	2,717,769
MILL FEED	EXPIT to MILL (t)				200,155	222,060	214,897	334,552	350,954	339,633	314,135	355,819	276,388
	STOCKS to MILL (t)				14,741			16,402			103,754	48,590	141,500
	HEAP LEACH to MILL (t)												
	TOTAL FEED (t)				**214,897**	**222,060**	**214,897**	**350,954**	**350,954**	**339,633**	**417,888**	**404,408**	**417,888**
	GRADE (Au g/t)				**1.18**	**1.20**	**1.20**	**1.18**	**1.20**	**1.20**	**1.18**	**1.19**	**1.18**
	GRADE (Ag g/t)				**0.56**	**0.61**	**0.76**	**0.67**	**0.76**	**0.70**	**0.63**	**0.83**	**0.74**
	GRADE (Cu%)				**0.05**	**0.05**	**0.05**	**0.06**	**0.05**	**0.05**	**0.05**	**0.06**	**0.05**
	GRADE (S%)				**1.04**	**1.06**	**1.20**	**1.31**	**1.22**	**1.20**	**1.09**	**1.22**	**1.19**
MILL FEED by RESCAT	MEASURED FEED (t)				210,691	211,256	207,227	323,992	338,930	286,642	359,870	324,330	349,482
	MEASURED GRADE (Au g/t)				1.19	1.20	1.21	1.18	1.21	1.16	1.15	1.15	1.15
	MEASURED GRADE (Ag g/t)				0.56	0.61	0.77	0.68	0.77	0.68	0.63	0.82	0.72
	MEASURED GRADE (Cu%)				0.05	0.05	0.05	0.06	0.05	0.05	0.05	0.06	0.05
	MEASURED GRADE (S%)				1.05	1.08	1.22	1.32	1.23	1.18	1.09	1.22	1.17
	INDICATED FEED (t)				4,205	10,803	7,670	26,962	12,024	52,990	58,019	80,078	68,407
	INDICATED GRADE (Au g/t)				0.68	1.28	0.96	1.22	0.97	1.40	1.38	1.35	1.35
	INDICATED GRADE (Ag g/t)				0.35	0.43	0.44	0.47	0.53	0.81	0.63	0.85	0.89
	INDICATED GRADE (Cu%)				0.03	0.03	0.03	0.05	0.05	0.06	0.05	0.06	0.06
	INDICATED GRADE (S%)				0.66	0.62	0.63	1.18	1.07	1.31	1.09	1.23	1.28
INFERRED MINED	HG INFERRED (T)	1,511		23,147	23,211	46,869	66,684	60,090	71,379	63,836	47,583	63,637	22,888
	HG INFERRED GRADE (Au g/t)	1.27		1.17	1.30	1.34	1.39	1.48	1.85	1.56	1.72	1.91	1.25
	MG INFERRED (T)	20,803	6,771	52,733	52,036	43,677	61,515	31,200	43,178	19,174	40,974	18,160	34,930
	MG INFERRED GRADE (Au g/t)	0.85	0.80	0.81	0.83	0.86	0.87	0.84	0.81	0.88	0.87	0.89	0.83
	LG INFERRED (T)	28,240	14,845	119,929	49,398	61,708	52,827	57,112	43,148	50,522	59,987	26,374	19,213
	LG INFERRED GRADE (Au g/t)	0.59	0.60	0.59	0.61	0.60	0.61	0.58	0.59	0.59	0.57	0.62	0.58
	INFERRED TOTAL (T)	**50,554**	**21,616**	**195,808**	**124,645**	**152,254**	**181,027**	**148,402**	**157,706**	**133,532**	**148,544**	**108,170**	**77,031**
	INFERRED TOTAL GRADE (Au g/t)	**0.72**	**0.67**	**0.72**	**0.83**	**0.90**	**0.99**	**1.00**	**1.22**	**1.09**	**1.02**	**1.42**	**0.89**

		1/01/2029	1/02/2029	1/03/2029	1/04/2029	1/05/2029	1/06/2029	1/07/2029	1/08/2029	1/09/2029	1/10/2029	1/11/2029	1/12/2029
EXPIT ORE	HG ORE (t)	229,980	202,009	204,876	149,411	222,409	219,965	298,298	366,633	382,555	420,711	445,313	268,836
	MG ORE (t)	224,490	103,565	180,800	249,731	116,697	195,800	208,782	234,452	212,048	180,131	148,740	96,383
	LG ORE (t)	237,879	157,900	163,785	323,671	164,808	226,578	197,987	263,346	298,621	270,315	235,142	111,776
EXPIT WASTE	NAF (t)	303,128	476,271	417,400	286,927	250,936	277,937	184,408	155,216	208,445	26,164	261,256	357,796
	PAF (t)	982,693	706,342	940,850	830,266	1,170,691	1,019,625	1,264,592	1,034,399	861,669	1,333,179	973,666	1,004,946
	UNCLASSIFIED (t)	252,911	456,433	491,331	421,411	300,007	182,152	148,598	203,723	251,839	66,927	115,043	681,367
	MINERALISED WASTE (t)	372,050	217,833	237,036	296,281	308,388	355,373	297,263	330,389	294,919	341,168	302,068	162,454
	INFERRED MATERIAL (t)	114,639	134,407	81,691	72,400	183,834	152,670	117,840	129,610	120,002	79,174	148,871	34,212
EXPIT TOTALS	ORE TOTAL (t)	692,348	463,473	549,462	722,813	503,914	642,343	705,068	864,432	893,224	871,157	829,196	476,994
	WASTE TOTAL (t)	2,025,421	1,991,286	2,168,308	1,907,286	2,213,855	1,987,757	2,012,701	1,853,337	1,736,875	1,846,613	1,800,904	2,240,775
	TOTAL MINED (t)	2,717,769	2,454,759	2,717,769	2,630,099	2,717,769	2,630,099	2,717,769	2,717,769	2,630,099	2,717,769	2,630,099	2,717,769
MILL FEED	EXPIT to MILL (t)	401,498	370,599	348,691	278,414	441,642	386,316	452,260	452,260	437,671	452,260	437,671	452,260
	STOCKS to MILL (t)	45,335	32,992	98,142	159,257	10,619	51,356						
	HEAP LEACH to MILL (t)												
	TOTAL FEED (t)	**446,833**	**403,591**	**446,833**	**437,671**	**452,260**	**437,671**	**452,260**	**452,260**	**437,671**	**452,260**	**437,671**	**452,260**
	GRADE (Au g/t)	**1.18**	**1.18**	**1.18**	**1.19**	**1.18**	**1.18**	**1.20**	**1.20**	**1.20**	**1.20**	**1.20**	**1.20**
	GRADE (Ag g/t)	**0.91**	**0.72**	**0.76**	**0.96**	**0.68**	**0.79**	**2.18**	**1.14**	**1.79**	**1.12**	**1.20**	**1.32**
	GRADE (Cu%)	**0.05**	**0.05**	**0.05**	**0.05**	**0.05**	**0.05**	**0.05**	**0.05**	**0.05**	**0.05**	**0.05**	**0.05**
	GRADE (S%)	**1.12**	**1.19**	**1.12**	**1.08**	**1.09**	**1.06**	**1.09**	**1.19**	**1.19**	**1.14**	**1.22**	**1.19**
MILL FEED by RESCAT	MEASURED FEED (t)	368,154	321,825	369,975	360,604	359,205	319,310	308,048	308,484	296,284	292,447	284,342	294,911
	MEASURED GRADE (Au g/t)	1.16	1.13	1.16	1.18	1.14	1.16	1.17	1.17	1.22	1.16	1.15	1.07
	MEASURED GRADE (Ag g/t)	0.74	0.65	0.73	1.02	0.64	0.77	2.05	1.23	1.47	0.80	0.84	0.99
	MEASURED GRADE (Cu%)	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05
	MEASURED GRADE (S%)	1.13	1.11	1.13	1.11	1.02	1.06	1.06	1.18	1.23	1.07	1.22	1.16
	INDICATED FEED (t)	78,680	81,767	76,858	77,068	93,056	118,362	144,212	143,776	141,387	159,813	153,329	157,349
	INDICATED GRADE (Au g/t)	1.27	1.39	1.28	1.22	1.34	1.24	1.26	1.26	1.15	1.28	1.30	1.44
	INDICATED GRADE (Ag g/t)	1.66	0.93	0.85	0.64	0.87	0.84	2.42	0.91	2.47	1.72	1.87	1.92
	INDICATED GRADE (Cu%)	0.05	0.06	0.05	0.04	0.06	0.05	0.05	0.05	0.05	0.06	0.06	0.06
	INDICATED GRADE (S%)	1.09	1.48	1.09	0.93	1.34	1.09	1.14	1.21	1.11	1.28	1.23	1.23
INFERRED MINED	HG INFERRED (T)	26,429	50,417	16,874	21,687	58,648	37,515	36,182	31,132	23,934	26,374	38,428	9,692
	HG INFERRED GRADE (Au g/t)	1.24	2.03	2.15	1.50	1.92	1.71	1.43	1.63	1.45	1.32	1.20	1.24
	MG INFERRED (T)	31,155	38,346	14,456	21,700	60,128	50,463	40,788	38,501	38,447	23,979	52,769	9,866
	MG INFERRED GRADE (Au g/t)	0.82	0.81	0.76	0.79	0.80	0.78	0.83	0.83	0.80	0.84	0.85	0.87
	LG INFERRED (T)	57,055	45,644	50,362	29,014	65,058	64,692	40,870	59,976	57,622	28,821	57,674	14,654
	LG INFERRED GRADE (Au g/t)	0.60	0.59	0.60	0.60	0.59	0.59	0.59	0.59	0.61	0.60	0.59	0.58
	INFERRED TOTAL (T)	**114,639**	**134,407**	**81,691**	**72,400**	**183,834**	**152,670**	**117,840**	**129,610**	**120,002**	**79,174**	**148,871**	**34,212**
	INFERRED TOTAL GRADE (Au g/t)	**0.81**	**1.19**	**0.95**	**0.93**	**1.08**	**0.93**	**0.93**	**0.91**	**0.84**	**0.91**	**0.84**	**0.85**

		1/01/2030	1/04/2030	1/07/2030	1/10/2030	1/01/2031	1/04/2031	1/07/2031
EXPIT ORE	HG ORE (t)	875,582	558,089	387,194	759,045	1,033,224	818,736	1,149,758
	MG ORE (t)	348,268	408,662	757,563	683,490	483,575	491,013	696,417
	LG ORE (t)	508,213	631,914	800,436	956,103	728,581	587,757	767,593
EXPIT WASTE	NAF (t)	1,012,360	1,338,115	1,340,440	1,005,435	2,636,556	1,951,373	2,109,908
	PAF (t)	2,696,105	2,972,539	2,658,176	2,394,729	1,502,227	1,882,650	1,709,110
	UNCLASSIFIED (t)	1,332,661	762,210	770,570	918,776	649,673	1,211,432	785,126
	MINERALISED WASTE (t)	842,885	1,011,425	993,999	1,087,179	649,051	774,887	751,646
	INFERRED MATERIAL (t)	274,222	295,012	357,259	260,882	207,411	260,119	96,079
EXPIT TOTALS	ORE TOTAL (t)	1,732,064	1,598,666	1,945,194	2,398,637	2,245,380	1,897,506	2,613,768
	WASTE TOTAL (t)	6,158,234	6,379,301	6,120,444	5,667,001	5,644,918	6,080,462	5,451,870
	TOTAL MINED (t)	7,890,297	7,977,967	8,065,637	8,065,638	7,890,298	7,977,968	8,065,638
MILL FEED	EXPIT to MILL (t)	1,313,014	1,272,092	696,297	1,168,156	1,313,014	1,327,603	1,342,192
	STOCKS to MILL (t)		55,511	645,895	174,036			
	HEAP LEACH to MILL (t)							
	TOTAL FEED (t)	**1,313,014**	**1,327,603**	**1,342,192**	**1,342,192**	**1,313,014**	**1,327,603**	**1,342,192**
	GRADE (Au g/t)	**1.20**	**1.18**	**1.18**	**1.18**	**1.20**	**1.20**	**1.20**
	GRADE (Ag g/t)	**1.09**	**0.76**	**1.29**	**2.08**	**1.33**	**1.37**	**0.93**
	GRADE (Cu%)	**0.05**	**0.05**	**0.05**	**0.05**	**0.05**	**0.05**	**0.06**
	GRADE (S%)	**1.12**	**1.05**	**1.24**	**1.12**	**1.14**	**1.09**	**1.20**
MILL FEED by RESCAT	MEASURED FEED (t)	893,770	857,673	698,830	664,170	733,267	736,965	888,146
	MEASURED GRADE (Au g/t)	1.04	1.14	1.18	1.18	1.22	1.10	1.10
	MEASURED GRADE (Ag g/t)	0.80	0.70	1.15	1.80	1.42	1.27	0.74
	MEASURED GRADE (Cu%)	0.05	0.05	0.06	0.05	0.05	0.05	0.06
	MEASURED GRADE (S%)	1.06	1.03	1.27	1.09	1.11	1.04	1.21
	INDICATED FEED (t)	419,244	469,930	643,362	678,021	579,747	590,637	454,045
	INDICATED GRADE (Au g/t)	1.54	1.25	1.19	1.18	1.18	1.33	1.40
	INDICATED GRADE (Ag g/t)	1.72	0.84	1.43	2.35	1.21	1.49	1.28
	INDICATED GRADE (Cu%)	0.06	0.05	0.05	0.05	0.05	0.05	0.05
	INDICATED GRADE (S%)	1.24	1.08	1.21	1.14	1.18	1.15	1.18
INFERRED MINED	HG INFERRED (T)	50,555	53,081	72,550	31,426	34,429	27,668	4,841
	HG INFERRED GRADE (Au g/t)	1.61	1.15	1.35	1.50	1.18	1.19	1.23
	MG INFERRED (T)	82,523	79,194	116,005	83,016	83,263	81,618	43,250
	MG INFERRED GRADE (Au g/t)	0.83	0.83	0.81	0.82	0.85	0.90	0.80
	LG INFERRED (T)	141,144	162,737	168,704	146,440	89,719	150,833	47,988
	LG INFERRED GRADE (Au g/t)	0.56	0.58	0.58	0.58	0.57	0.58	0.60
	INFERRED TOTAL (T)	**274,222**	**295,012**	**357,259**	**260,882**	**207,411**	**260,119**	**96,079**
	INFERRED TOTAL GRADE (Au g/t)	**0.84**	**0.75**	**0.81**	**0.77**	**0.78**	**0.74**	**0.72**

1/10/2031	1/01/2032	1/04/2032	1/07/2032	1/10/2032	1/01/2033	1/01/2034	1/01/2035	1/01/2036	1/01/2037	1/01/2038	1/01/2039	1/01/2040	1/01/2041	1/01/2042	1/01/2043	1/01/2044
	160,912	237,125	236,305	303,438	1,586,826	1,430,710	1,848,180	1,855,355	2,449,473	1,883,643	1,655,582	2,135,279	1,406,790	1,961,107	1,974,355	1,381,055
356,802	514,006	391,709	524,019	470,703	1,280,757	1,293,341	2,233,609	2,137,873	2,199,368	1,633,589	1,434,248	1,660,387	1,121,476	1,534,675	1,314,405	1,811,090
418,085	504,177	388,806	631,672	629,814	2,217,114	2,255,395	3,266,782	2,427,223	2,536,789	2,040,091	1,649,801	2,076,663	1,747,684	2,299,802	2,292,535	2,374,076
2,799,745	3,258,794	4,368,491	3,319,623	3,357,537	11,134,984	6,437,640	3,004,383	9,902,134	11,797,779	13,889,979	11,023,974	7,586,847	8,262,135	7,224,669	9,513,646	12,947,814
1,790,535	1,770,061	1,095,728	1,677,506	1,606,021	9,614,867	15,008,156	15,878,788	8,214,694	6,413,377	5,741,768	7,731,178	11,061,355	9,144,933	10,197,784	8,140,938	5,075,734
1,424,306	1,004,129	1,015,168	978,475	995,030	3,498,223	1,870,423	898,948	4,867,481	4,071,741	4,638,636	6,414,652	4,526,378	7,717,360	5,705,376	5,785,042	5,972,295
721,465	659,016	419,168	640,928	653,380	2,379,992	2,966,485	3,730,532	2,518,034	2,433,805	2,061,482	1,891,320	2,595,967	2,203,754	2,809,220	2,718,739	2,333,993
110,926	106,873	61,771	57,109	49,715	286,774	737,384	1,138,312	164,420	97,210	110,353	198,788	444,338	395,410	266,908	259,882	191,154
1,218,660	1,179,095	1,017,640	1,391,996	1,403,954	5,084,696	4,979,446	7,348,572	6,420,451	7,185,630	5,557,323	4,739,631	5,872,329	4,275,950	5,795,584	5,581,295	5,566,220
6,846,976	6,798,872	6,960,327	6,673,641	6,661,683	26,914,840	27,020,088	24,650,962	25,666,762	24,813,912	26,442,219	27,259,912	26,214,885	27,723,592	26,203,957	26,418,245	26,520,990
8,065,637	7,977,967	7,977,967	8,065,637	8,065,637	31,999,536	31,999,534	31,999,534	32,087,213	31,999,542	31,999,541	31,999,543	32,087,214	31,999,542	31,999,541	31,999,540	32,087,211
1,047,513	1,024,517	1,017,640	1,060,236	1,118,365	4,085,509	3,094,513	5,037,183	4,614,430	5,325,000	5,216,976	4,739,631	5,339,589	3,478,815	5,325,000	5,325,000	4,872,034
294,679	303,086	309,962	281,956	223,827	1,239,491	2,230,487	287,817	725,159		108,024	585,369		1,846,185			467,555
1,342,192	**1,327,603**	**1,327,603**	**1,342,192**	**1,342,192**	**5,325,000**	**5,325,000**	**5,325,000**	**5,339,589**	**5,325,000**	**5,325,000**	**5,325,000**	**5,339,589**	**5,325,000**	**5,325,000**	**5,325,000**	**5,339,589**
1.18	**0.99**	**1.00**	**1.00**	**0.99**	**0.99**	**0.99**	**0.99**	**0.99**	**1.00**	**0.99**	**0.99**	**1.00**	**0.99**	**1.00**	**1.00**	**0.95**
0.82	**0.73**	**0.72**	**0.83**	**1.13**	**1.15**	**0.79**	**0.51**	**0.63**	**2.52**	**1.20**	**1.06**	**0.95**	**0.68**	**0.57**	**0.57**	**1.05**
0.05	**0.05**	**0.04**	**0.05**	**0.05**	**0.05**	**0.05**	**0.04**	**0.05**	**0.05**	**0.05**	**0.05**	**0.05**	**0.05**	**0.05**	**0.05**	**0.05**
1.19	**1.03**	**0.94**	**1.10**	**1.03**	**1.11**	**1.13**	**0.86**	**0.95**	**1.02**	**1.05**	**1.01**	**1.01**	**0.99**	**1.03**	**1.02**	**1.02**
852,395	881,848	799,141	795,975	754,940	3,196,144	3,555,808	3,165,553	3,193,210	2,814,664	2,810,231	2,921,497	2,786,820	2,698,398	2,153,339	2,302,747	2,327,871
1.20	0.96	0.95	0.95	0.95	0.90	0.96	0.96	0.98	0.97	0.97	0.95	0.98	0.97	1.05	0.95	0.90
0.86	0.76	0.58	0.73	1.06	1.10	0.80	0.54	0.70	1.50	1.18	1.27	0.81	0.62	0.49	0.53	1.08
0.06	0.05	0.04	0.05	0.04	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05
1.22	1.04	0.90	1.05	0.94	1.01	1.12	0.93	0.97	1.02	1.03	0.97	1.01	0.98	1.08	0.97	1.01
489,797	445,754	528,462	546,216	587,251	2,128,856	1,769,192	2,159,447	2,146,379	2,510,336	2,514,769	2,403,503	2,552,769	2,626,602	3,171,661	3,022,253	3,011,718
1.16	1.05	1.07	1.06	1.04	1.12	1.06	1.03	1.00	1.03	1.01	1.04	1.02	1.01	0.97	1.04	0.98
0.75	0.66	0.87	0.95	1.19	1.17	0.71	0.45	0.51	3.63	1.20	0.74	1.06	0.71	0.60	0.57	1.01
0.05	0.05	0.05	0.05	0.05	0.06	0.05	0.04	0.04	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05
1.14	1.01	1.00	1.19	1.13	1.26	1.15	0.76	0.92	1.02	1.07	1.05	1.02	1.00	1.00	1.05	1.02
34,242	16,235	7,261	2,436	2,351	14,393	122,891	187,974	43,501	7,261	2,116	30,399	93,064	26,474	23,966	33,291	16,742
1.23	1.24	1.27	3.49	1.10	1.16	1.50	1.31	1.60	1.33	1.14	1.27	1.41	1.33	1.42	1.70	2.47
31,067	40,626	26,047	16,670	14,375	64,992	202,211	295,023	30,569	51,122	53,403	94,145	180,111	175,523	67,691	78,459	35,885
0.81	0.80	0.73	0.78	0.85	0.80	0.81	0.82	0.83	0.80	0.80	0.80	0.80	0.82	0.79	0.80	0.85
45,617	50,012	28,463	38,004	32,989	207,389	412,283	655,316	90,350	38,826	54,834	74,244	171,164	193,413	175,251	148,132	138,527
0.58	0.59	0.63	0.62	0.63	0.58	0.58	0.58	0.61	0.58	0.57	0.58	0.59	0.59	0.59	0.60	0.59
110,926	**106,873**	**61,771**	**57,109**	**49,715**	**286,774**	**737,384**	**1,138,312**	**164,420**	**97,210**	**110,353**	**198,788**	**444,338**	**395,410**	**266,908**	**259,882**	**191,154**
0.85	**0.77**	**0.75**	**0.79**	**0.71**	**0.66**	**0.80**	**0.76**	**0.91**	**0.76**	**0.69**	**0.79**	**0.85**	**0.74**	**0.71**	**0.80**	**0.80**

1/01/2045	1/01/2046	1/01/2047	1/01/2048	1/01/2049	1/01/2050	1/01/2051	1/01/2052	1/01/2053	1/01/2054	1/01/2055	1/01/2056	1/01/2057	1/01/2058	1/01/2059	1/01/2060	Grand Total
1,824,821	1,661,116	262,674	541,813	1,020,386	1,179,918	829,288	1,621,449	4,544,685	3,863,840							**51,476,733**
1,234,334	1,297,840	260,038	686,928	1,283,377	2,007,547	1,684,642	2,581,802	3,216,622	1,910,308							**46,380,434**
2,158,307	2,027,002	644,202	1,485,144	2,177,517	2,832,442	2,284,439	2,985,918	3,043,946	1,474,863							**60,765,601**
12,153,749	12,784,196	17,702,160	12,314,492	5,770,473	1,349,200	28,752	9,646	2,403	0							**211,835,953**
8,243,170	8,434,140	7,346,869	11,400,419	15,524,884	18,789,243	8,538,839	6,228,922	3,285,621	374,531							**244,954,637**
3,533,536	3,069,791	4,127,115	2,272,106	915,553	148,598	0	0	0	0							**89,765,452**
2,562,542	2,483,223	1,411,860	2,624,330	3,770,285	4,287,241	2,432,154	2,567,808	1,904,080	623,667							**70,967,472**
289,081	182,031	244,615	761,970	1,537,055	1,405,343	201,657	48,059	2,412	0							**13,969,174**
5,217,462	4,985,958	1,166,914	2,713,885	4,481,281	6,019,907	4,798,370	7,189,170	10,805,254	7,249,010							**158,622,768**
26,782,078	26,953,381	30,832,618	29,373,317	27,518,251	25,979,624	11,201,401	8,854,436	5,194,516	998,198							**631,492,689**
31,999,540	31,939,339	31,999,532	32,087,201	31,999,531	31,999,531	15,999,770	16,043,605	15,999,770	8,247,208							**790,115,457**
5,217,462	4,985,958	929,823	2,207,014	3,806,350	4,788,123	4,233,062	5,339,589	5,325,000	5,325,000							**119,831,834**
107,538	339,042	4,395,177	3,132,575	1,518,650	536,877	1,091,938				5,325,000	5,339,589	5,325,000	1,177,823			**38,790,935**
													4,147,177	5,325,000	3,881,823	**13,354,000**
5,325,000	**5,325,000**	**5,325,000**	**5,339,589**	**5,325,000**	**5,325,000**	**5,325,000**	**5,339,589**	**5,325,000**	**5,325,000**	**5,325,000**	**5,339,589**	**5,325,000**	**5,325,000**	**5,325,000**	**3,881,823**	**171,976,768**
0.99	**1.02**	**0.63**	**0.69**	**0.82**	**0.86**	**0.80**	**1.00**	**1.39**	**1.48**	**0.89**	**0.58**	**0.58**	**0.55**	**0.54**	**0.54**	**0.94**
0.62	**0.46**	**1.23**	**0.89**	**0.61**	**0.59**	**0.92**	**0.99**	**0.92**	**1.04**	**1.39**	**1.16**	**1.16**	**-**	**-**	**-**	**0.96**
0.04	**0.03**	**0.04**	**0.03**	**0.03**	**0.04**	**0.04**	**0.05**	**0.06**	**0.06**	**0.05**	**0.04**	**0.04**	**-**	**-**	**-**	**0.05**
0.87	**0.59**	**0.84**	**0.74**	**0.75**	**0.83**	**0.93**	**1.05**	**1.27**	**1.34**	**1.15**	**0.90**	**0.90**	**-**	**-**	**-**	**1.00**
2,805,377	2,688,850	3,485,055	2,725,013	1,841,715	1,291,568	1,850,516	1,191,360	1,670,553	1,540,494	1,595,477	2,116,494	2,110,711	466,863			**77,359,458**
0.94	0.94	0.63	0.65	0.77	0.85	0.80	0.99	1.44	1.58	0.88	0.58	0.58	0.58			**0.95**
0.41	0.18	1.07	0.93	0.81	0.55	0.85	1.13	0.84	0.95	1.86	1.12	1.12	1			**0.90**
0.04	0.03	0.04	0.04	0.04	0.04	0.04	0.05	0.06	0.06	0.05	0.04	0.04	0			**0.05**
0.78	0.63	0.88	0.83	0.86	0.87	0.92	1.04	1.32	1.37	1.16	0.90	0.90	0.90			**1.00**
2,519,623	2,636,150	1,839,945	2,614,576	3,483,285	4,033,432	3,474,484	4,148,229	3,654,447	3,784,506	3,729,523	3,223,095	3,214,289	4,858,137	5,325,000	3,881,823	**94,617,310**
1.06	1.09	0.64	0.73	0.85	0.86	0.80	1.01	1.37	1.44	0.89	0.57	0.57	0.54	0.54	0.54	**0.93**
0.62	0.45	1.51	0.81	0.47	0.59	0.95	0.95	0.95	1.07	1.19	1.19	1.19				**0.99**
0.04	0.03	0.04	0.03	0.03	0.04	0.04	0.05	0.06	0.06	0.05	0.04	0.04				**0.05**
0.97	0.54	0.76	0.64	0.69	0.82	0.93	1.05	1.24	1.33	1.15	0.89	0.89				**0.99**
25,911	57,443	52,682	134,222	326,541	213,785	7,207										**2,625,087**
1.30	1.37	1.28	1.24	1.47	1.65	1.14										**1.48**
60,224	18,850	55,169	143,684													**3,150,463**
0.78	0.79	0.83	0.77													**0.82**
202,946	105,738	136,763	484,065													**5,546,631**
0.58	0.58	0.58	0.58													**0.59**
289,081	**182,031**	**244,615**	**761,970**	**326,541**	**213,785**	**7,207**										**11,322,181**
0.69	**0.85**	**0.79**	**0.73**	**1.47**	**1.65**	**1.14**										**0.86**

Table 94 Mine Production Schedule

13.6.3 *Open Pit Mining Operation*

The Batman gold deposit will be developed in eight distinct mining stages using a conventional open-pit truck and shovel method. All in-situ material will require pre-treatment via controlled drilling and blasting to achieve optimal fragmentation for efficient excavation and downstream processing. The mining operations will be executed by an Australian contract mining company, selected for its capability to manage large-scale operations and maintain high equipment availability. The contractor will utilize a fleet comprising 400-tonne class hydraulic excavators and 190-tonne class rigid-frame haul trucks, supported by ancillary equipment including loaders, dozers, graders, and water carts.

The contractor scope of this Project includes the drilling and blasting of all material as required, followed by the loading and hauling of material to ROM pads, waste dumps, and stockpiles as directed. It also encompasses the rehandling of low-grade material from stockpiles to the ROM. The contractor shall be responsible for the maintenance of all equipment utilized throughout the Project, as well as the provision of all consumables and coverage of operational costs. Comprehensive management, supervision, short-term planning (3 months), maintenance, and operational labor will be supplied by the mining contractor to ensure the successful execution of all activities.

Mining support facilities, including a Heavy Mobile Equipment (HME) workshop comprising six domes, each capable of accommodating one 190 tonne truck with its tray raised. Additional infrastructure includes a stores warehouse, drills workshop, crib rooms, ablutions, meeting rooms, wash pad, clean and dirty water ponds, and fuel and oil storage, with the latter supplied as a rate-only estimate. The scope also includes explosive storage and associated facilities, as well as a fully equipped administration office and related amenities to support Project operations.

Vista will retain responsibility for medium to long-term mine planning, geotechnical engineering, geology, grade control, and overall technical services. This includes the development of long-term mine plans, reconciliation, and continuous monitoring of pit wall stability and orebody performance. The owner's technical team will work closely with the contractor to ensure adherence to design parameters, safety standards, and production targets.

Mining will proceed in a phased approach, with multiple stages active concurrently to optimize ore blending, maintain consistent mill feed tonnage (targeting 15 ktpd), and manage waste movement. Ore will be classified based on grade cut-offs into High Grade (HG), Medium Grade (MG), and Low Grade (LG) categories. HG and MG ore not immediately processed will be stockpiled on a ROM pad adjacent to the processing plant, with LG ore directed to a dedicated long-term LG stockpile for potential future processing.

Material handling and stockpile management will be governed by a dynamic ore control system, integrating blast hole sampling, real-time fleet management systems (FMS), and GPS-based grade control to minimize dilution and ore loss.

Waste rock will be geochemically characterized based on total sulfur content to determine its classification as Non-Acid Forming (NAF) or Potentially Acid Forming (PAF). PAF material will be encapsulated within a purpose-engineered waste rock landform (WRL), designed with basal liners, internal drainage, and progressive rehabilitation to minimize long-term environmental impact. NAF material will be used for construction, haul road maintenance, and as encapsulation cover for PAF zones.

Comprehensive environmental and geotechnical monitoring programs will be implemented throughout the life of mine, including air and water quality monitoring, slope stability radar, and drone-based volumetric surveys to ensure compliance with regulatory requirements and best practice mine stewardship.

13.7 Mining Equipment

13.7.1 Mining Fleet

All mining equipment operations and maintenance activities will be executed by the appointed contractor, utilizing a fleet of machinery and support systems of appropriate capacity and specification to meet the full scope of operational demands and performance criteria.

The selection and configuration of the primary mining fleet and ancillary support equipment were established through a collaborative process involving the mining contractor, Vista and Mining Plus. This process was guided by a detailed assessment of several key parameters, including:

Production Capacity: Evaluation of equipment throughput relative to the mine plan and targeted production rates.

Operational Suitability: Alignment of fleet capabilities with site-specific conditions, such as pit geometry, material characteristics, haul profiles and working hours.

Availability and Logistics: Consideration of equipment availability within the regional market, lead times and logistical constraints impacting mobilization and maintenance.

Mining Plus undertook a technical review of the proposed fleet configuration, including equipment specifications, utilization rates and productivity assumptions. This review incorporated benchmarking against industry standards and similar operations, as well as validation through first-principles calculations.

The outcome of this review confirmed that the selected fleet composition and associated productivity metrics are appropriate and sufficient to meet the operational and production objectives of the Project. The configuration is considered technically robust and aligned with the overall mine design and scheduling strategy.

The mining contractor selected the 400-tonne hydraulic excavators and 190-tonne trucks as the preferred primary load and haul fleet. This fleet fits the schedule with two fully utilized excavators to achieve mine plan production requirements. The excavators will be in backhoe set up, to support wall control requirements recommended by geotechnical consultants, and minimize ore dilution on boundaries of the ore body.

13.7.2 Drilling

The primary drilling fleet has been selected considering the Project site rock mass 's material properties and previous expertise at the site and in the nearby region. Furthermore, drilling fleet selection considers blast patterns developed during this Technical Report Summary, based on the patterns provided by this Technical Report Summary team in-conjunction with a site fragmentation study completed by Orica. The contractor has proposed the deployment of drill rigs with 30,000 kg pull down capacity for production drilling activities. These units will be utilized for blast hole drilling operations, inclusive of grade control, and pre-split hole drilling to facilitate controlled wall stability and minimize overbreak in final pit walls.

Drilling will be performed on 12 m benches with 171 mm diameter holes with an estimated penetration rate of 25 m/hr based on the following blasting outcomes tailored for the Project site:

- Ore patterns – high powder factor to assist with processing plant production rate and energy use.
- Waste rock patterns – lower powder factor for adequate fragmentation of waste rock for efficient loading and dumping in the waste dump.
- Waste rock for civil use patterns – higher powder factor (similar to ore blast) for adequate fragmentation of waste rock which is planned to be used in civil construction on site such as road construction and TSF wall construction.
- Modified wall blasts or trim shots, with lower powder factors &/or lower density explosives, etc. based on Geotech recommendations for pit wall outcomes.

Scheduling requires the deployment of three drill rigs simultaneously to maintain continuous production rates in accordance with the mine plan.

Additional drilling support for secondary breakage activities, contour blasting, and wall control blasting will be provided by two smaller drill rigs.

13.7.3 Blasting

Blasting supplies and services were based on a company like Orica or similar on a sub-contract basis. Ammonium Nitrate (AN) would be trucked from an Orica depot in Yarwun Queensland (near Gladstone) and stored at the dedicated AN storage facility on site. Explosives will be stored in an appropriately located and designed magazine.

Based on expertise, it was assumed that 50% of material will be wet and require emulsion products for blasting. Four patterns were designated by Orica based on the material type and the dry/wet state of the holes. As part of this Technical Report Summary, Orica was engaged to provide a fragmentation study based on the Batman deposit rock properties, and required blast outcomes, as summarized in Table 97. Higher powder factor blasts for ore will assist with overall size distribution and consistency provided to the processing plant, reducing crushing power requirements. Also, high powder factors will be used for waste rock which is planned to be used for site construction activities such as planned TSF lifts. It is recommended to revisit the drill and blast study before commencement of operations with more knowledge of the hydrology, hydrogeology and meteorology characteristics to better define the relationship between wet and dry holes.

Material	Dry/Wet Holes	Hole Diameter	Explosive	Burden m	Spacing m	Bench m	PF	Sub Drill	BCM/hole	Charge kg
Waste	Dry	171 mm	Fortan	5.9	6.8	12	0.57	1.4	481	274
Waste	Wet	171 mm	Fortis	5.8	6.7	12	0.58	1.4	466	270
Ore/Construction Waste	Dry	171 mm	Fortan	4.9	5.6	12	0.84	1.4	329	277
Ore/Construction Waste	Wet	171 mm	Fortis	4.9	5.6	12	0.84	1.4	329	277

Table 95 Blasting Patterns Prepared by Orica

13.7.4 Loading

Material loading operations will be executed using 400 tonne class hydraulic excavators configured in a backhoe arrangement to optimize dig efficiency and cycle times. During production ramp-up and ramp-down phases one excavator will be deployed at full utilization. For steady-state operations, the production schedule necessitates the concurrent full utilization of two excavators to meet production targets.

Further loading and haulage as required will be provided by a secondary load and haul fleet costed by the mining contractor. The rehandle of low-grade stockpile material to the ROM pad over the LOM and after mining has finished requires two 140 tonne dump trucks loaded by 130 tonne Front End Loaders. The front-end loaders will also be used for ROM crusher feed when trucks direct tipping is not occurring, or further ROM blending is required. Also note this secondary load and haul fleet is also assumed to be available for other small tasks on site such as the planned TSF wall lifts costed and discussed in other parts of this Technical Report Summary.

13.7.5 Hauling

Production haulage operations will be executed using 190 tonne class rigid-frame dump trucks, selected for their compatibility with 400 tonne excavators. This truck-excavator pairing is engineered to achieve optimal pass-matching, typically in 3 to 4 passes, thereby minimizing load cycle times and maximizing material movement efficiency. The selected truck class offers a practical balance between payload capacity, maneuverability, and cycle time performance, ensuring high productivity across varying haul profiles and pit geometries.

During the initial production ramp-up phase, a fleet of eight 190 tonne class rigid frame trucks will be deployed to support early-stage material movement requirements. As mining progresses and transitions to deeper operational zones, specifically the lower benches of Stage 6, the haulage demand increases due to extended cycle times and greater vertical lift. Accordingly, the truck fleet will be incrementally scaled up to a total of 24 units to maintain target production rates.

13.7.6 Ancillary and Support Equipment

Ancillary equipment required to support primary mining operations will include a fleet of track dozers, motor graders, and water trucks. Up to three heavy-duty track dozers will be deployed to assist excavator operations, facilitate efficient material handling, and manage the shaping and stability of waste dumps and stockpile areas. These units will also be critical for maintaining safe working surfaces and optimizing bench geometry.

To ensure haul road integrity and operational efficiency, up to two motor graders will be utilized for continuous surface conditioning, profile correction, and removal of rutting and fines. Additionally, up to two water trucks will be mobilized as needed to suppress dust emissions, maintain road surface cohesion, and comply with environmental and safety standards. The deployment of ancillary equipment will be dynamically adjusted based on operational demands, weather conditions, and production sequencing to maintain optimal working conditions and equipment productivity.

The rehandling of low-grade ore from the stockpile to the ROM pad will be conducted throughout the mining operations and continue post-mining to support final processing campaigns. This operation will utilize two 140 tonne rigid frame trucks, selected for their payload capacity and maneuverability in stockpile environments. Material will be loaded using a 130 tonne front-end loader, which offers high breakout force and bucket capacity suitable for efficient bulk rehandling. The loader-truck configuration is optimized for short-cycle haulage, ensuring consistent feed rates to the ROM pad while minimizing idle time and fuel consumption. Equipment deployment will be scheduled to align with mill feed requirements and stockpile drawdown strategies, ensuring seamless integration with downstream processing operations.

13.7.1 Summary Mining Equipment Profile

Equipment fleet breakdown by period can be seen in Table 96.

Equipment #	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31
400 t excavator	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	0	0	0	0
190 t Dump Trucks	0	10	10	12	12	14	12	15	16	16	14	15	14	14	14	16	16	18	15	16	19	19	21	22	24	24	24	4	0	0	0	0
130 t FEL	1	1	1	1	1	1	1	1	1	1	1	0	1	0	0	1	1	1	1	0	1	1	1	1	1	1	0	0	0	0	1	0
140 t Dump Trucks	2	2	2	2	2	2	2	2	2	2	2	0	2	0	0	2	2	2	2	0	2	2	2	2	2	2	0	0	0	0	2	0
Dozers	1	2	2	2	3	3	3	3	3	3	3	3	3	3	2	2	2	2	2	2	2	2	2	2	2	2	2	1	0	0	0	0
Graders	1	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	3	1	0	0	0	0
Watercarts	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	3	1	0	0	0	0
Drills	0	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	1	0	0	0	0

Table 96 Mining Equipment Profile by Period

13.8 Waste Storage Area

13.8.1 Waste Material Type Characterization

The WRD is intended to store waste material in perpetuity. The waste materials include PAF and NAF materials. These materials were identified based on ICP analyses of samples from drill holes as determined by Tetra Tech. PAF material is classified based on the percent of sulfur that is calculated from the ICP analyzes. The sulfide percent was estimated into the Mineral Resource block model by Tetra Tech geologists. As summarized in Table 97, 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 Mineral 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

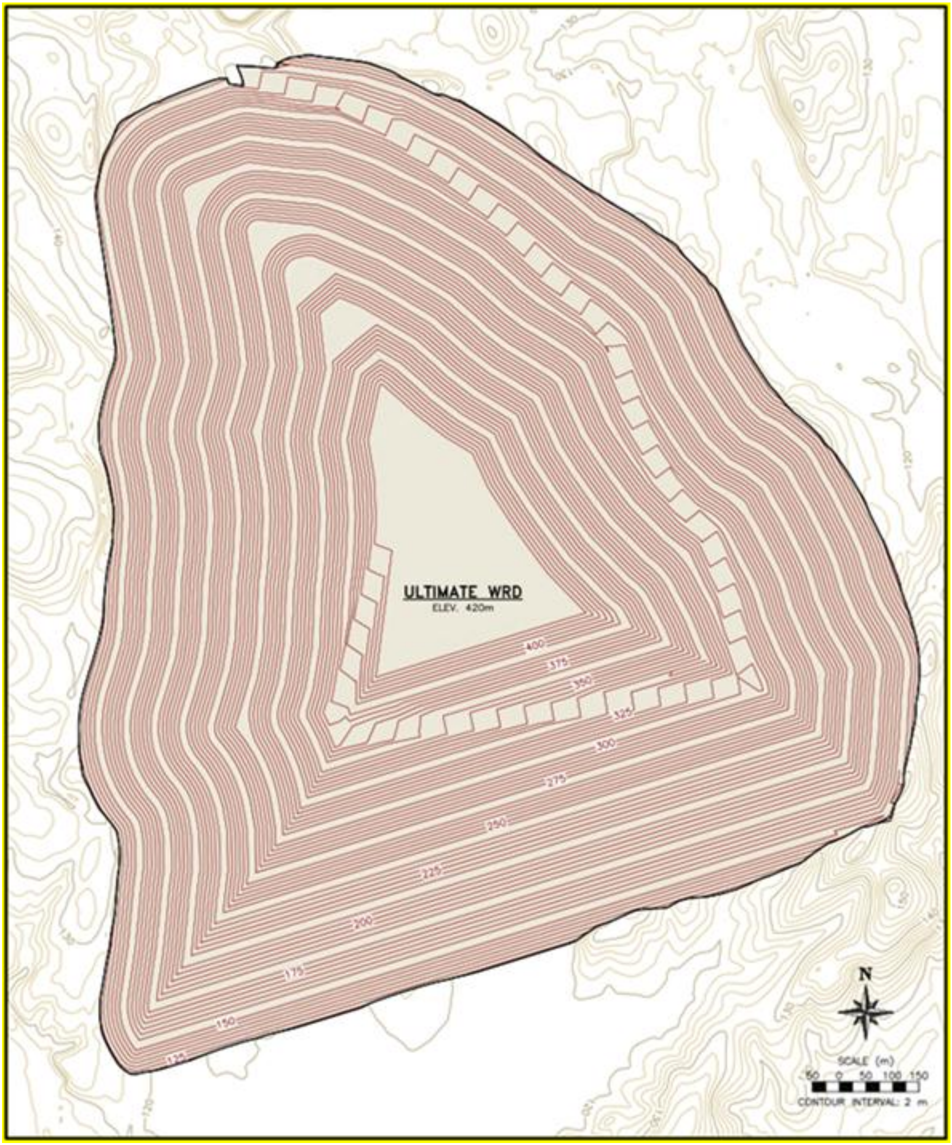
Material	Classification	
Waste	Non-Acid Forming (NAF)	S (%) < 0.25
	Potentially Acid Forming 1 (PAF1)	S (%) 0.25-0.4
	Potentially Acid Forming 2 (PAF2)	S (%) > 0.4
	Unclassified	S (%) -

Table 97 Waste Material Code Designations

13.9 Waste Rock Construction

13.9.1 Waste Rock Construction

ROM waste will be stored in the existing WRD throughout the LOM. The WRD will consist of waste rock classified as PAF or undefined, and encapsulated by a layer of NAF material, referred to as a NAF rind. Approximately 465 million tons (Mt) of waste rock will be stored to a maximum elevation of 420 m (Figure 91). The 15 ktpd mine plan was used to develop the WRD design and a proposed stacking plan based on the expected delivery schedule of the waste rock.



***Note** : Prepared by Tierra Group; updated August 2025*

Figure 91 **Final WRD Plan**

The WRD will be constructed in 10 m high lifts following side slopes at the angle of repose (approximately 1.5H:1V (Horizontal: Vertical)). An 8 m wide bench will be placed at 30 m elevation intervals to provide an overall slope of 1.75H:1V. The WRD will have a maximum height of 300 m, with the final lift having a 1% grade to the south to prevent surface water ponding. Each lift will commence with the construction of the NAF rind, followed by the filling of PAF waste rock. A minimum width of 10 m for the NAF rind was established through a seepage analysis and geochemical modelling. The NAF rind will be constructed to a width of 42 m to accommodate waste rock placement and facilitate traffic flow.

The primary goal of the WRD design is to prevent long-term infiltration of surface water into the dump. During operational phases, water runoff must be captured and managed. For long-term closure, the WRD will require a cover to achieve the design objective. A bituminous geomembrane (BGM) liner will be installed on each bench to establish an intermittent, impermeable barrier over the waste material. This BGM liner will be placed at the outer portion of the bench and extend into the dump a minimum of 52 m, fully covering the NAF rind from the previous lift. A layer of ore sorting rejects will be placed below and above the BGM liner to reduce the risk of liner damage from waste rock. Figure 92 presents the typical NAF interface section.

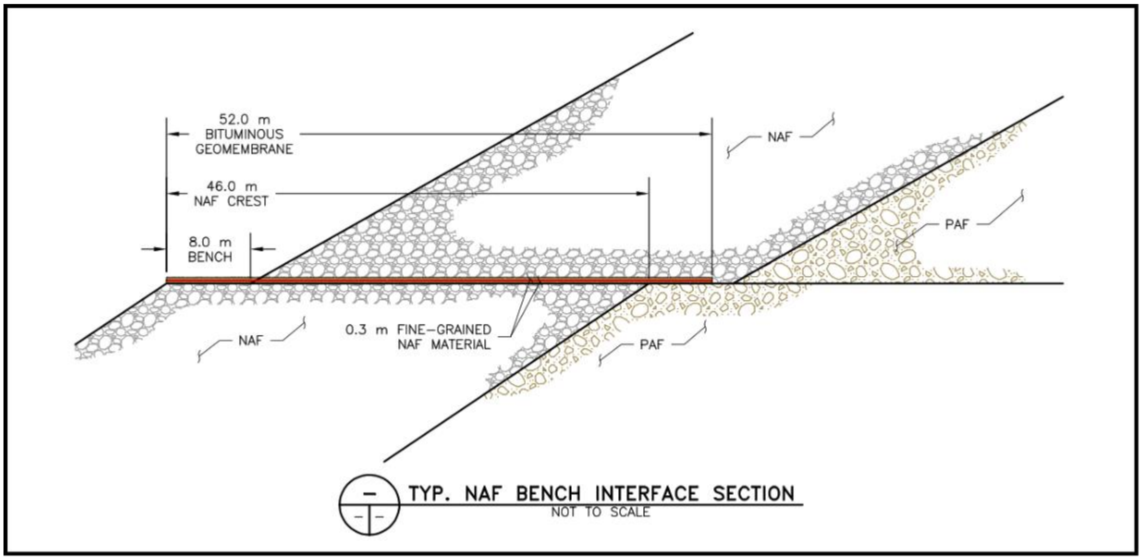


Figure 92 Typical NAF Bench Interface Section, Tierra Group 2025

Construction of the WRD will start with placing material along the north side of the existing WRD, then extending southward until reaching the Lift 2 elevation of 180 m. Lifts will be built by top-dumping waste rock from the full height of each lift, gradually progressing until each lift is complete. This construction method enhances the stability and compaction of the waste rock by having loaded haul trucks drive over the placed material. Figure 93, Figure 94, Figure 95, illustrate the dumping plan for the first five years of construction.

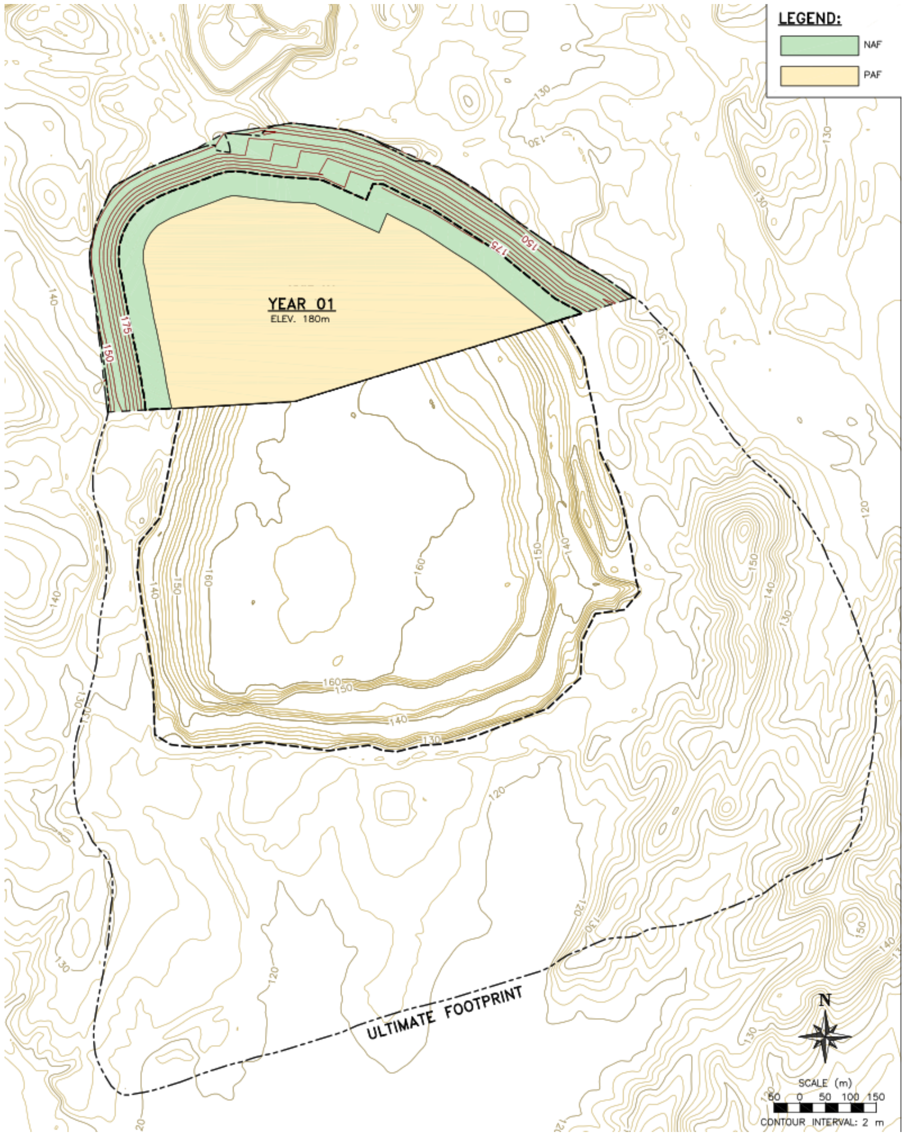


Figure 93 Year 1 of WRD Construction, Tierra Group August 2025

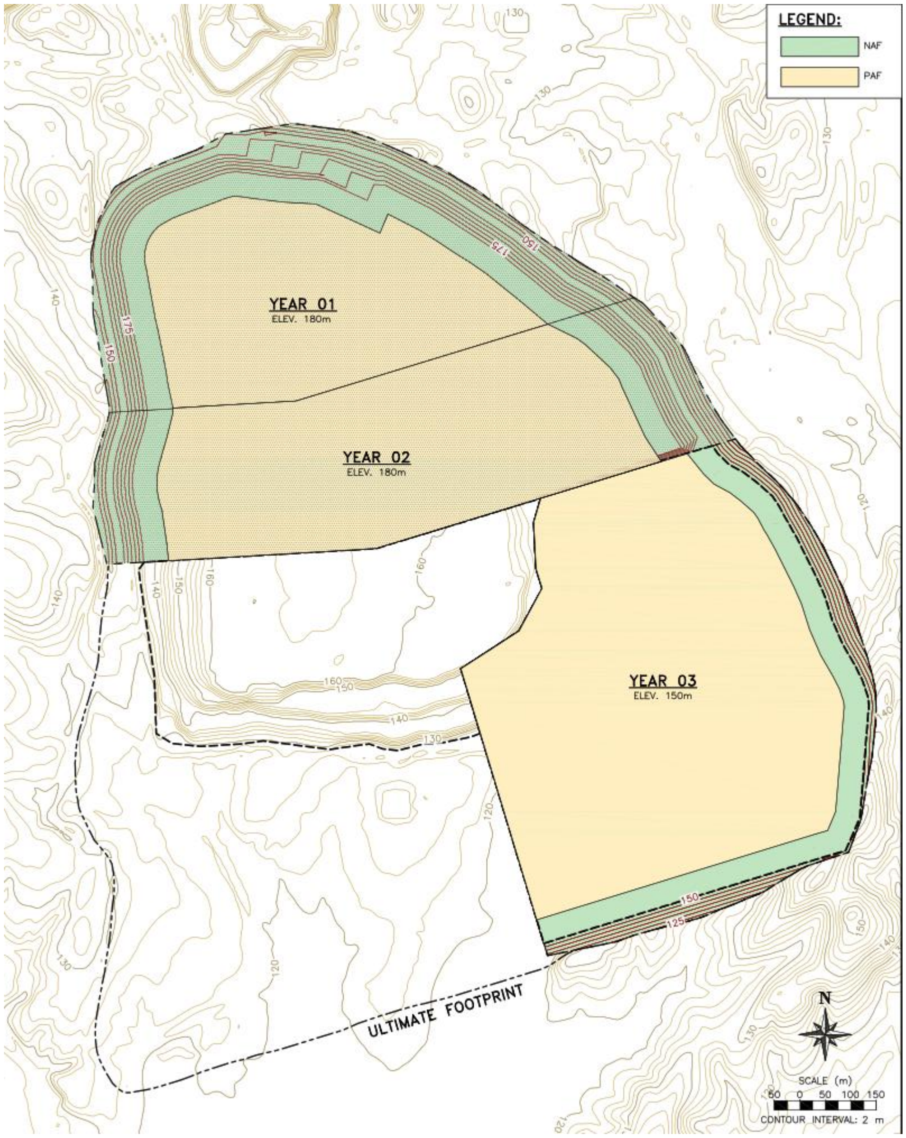


Figure 94 Year 3 of WRD Construction, Tierra Group August 2025

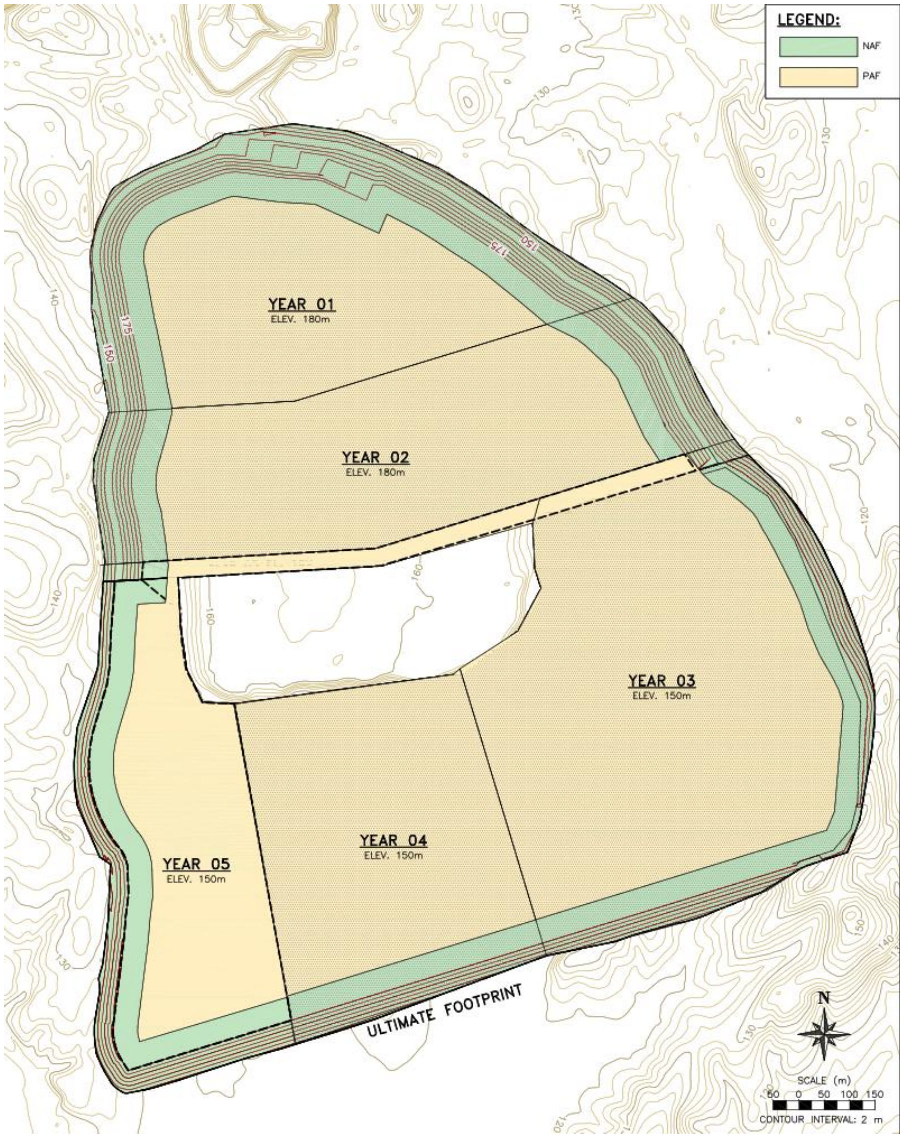


Figure 95 Year 5 of WRD Construction, Tierra Group August 2025

13.9.2 Mineralized Waste Management

Mineralized Waste as described in Section 13.8.1, will be placed in the northern sector of the WRD to ensure accessibility for potential future re-mining and processing. This material will be mined incrementally throughout the operational life of the pit and will therefore be stored in successive lifts. Strategic placement of this material in defined zones within the northern WRD will maintain proximity to the primary crusher, thereby facilitating potential rehandling if required.

Subsequent technical studies will further delineate the extent and characteristics of the mineralized waste to support development of detailed stacking plans and integration with mine scheduling. At closure, the current plan assumes the placement of a cover system over the mineralized waste; however, this approach will be reassessed as future studies advance and in consideration of ongoing economic, environmental, and operational factors.

13.9.3 Waste Rock Geotechnical Analysis

Stability criteria are carried over from the previous feasibility studies. WRD design during the previous geotechnical analysis used Leps (1970) strength function to model the strength properties of the waste rock. Updated strength properties used in this Technical Report Summary incorporated the Hawley and Cunning (2017) strength function. This update was made to enhance the modelling of the WRD based on the proposed construction method. Satisfactory slope stability factors of safety were achieved for the maximum WRD height.

13.10 Mining Stockpile Areas & Pit access

13.10.1 Surface Road Designs/Haul Road Layout

Surface haul roads for the Batman pit have been designed to incorporate vehicle access from the pit exit to the various landforms. A minimum 10 m pit crest offset has been applied to the internal surface road whilst a minimum 10 m offset has also been applied to the external edge, adjacent to the Low-grade stockpile. This standoff has been included to allow for any pit perimeter or catch bunds as required by site operations.

Table 98 provides the parameters used for the surface haul road designs.

Item	Units	Value
Surface Haul Road Width (dual-lane)	m	39
Surface Haul Road Width (single-lane)	m	25
Haul Ramp Gradient	1:X	10

Table 98 Surface Haul Road Design Parameters

13.10.2 Low Grade Stockpile

A low-grade stockpile was designed between the pits and the TSF1 based on parameters in Table 99 and as shown in Figure 96.

Figure 96 Low Grade Stockpile Design, Mining Plus 2025

Operational Face Angle (degrees)	Berm Width (m)	Bench Height (m)
34	15	10

Table 99 Low Grade Stockpile Design Parameters

The LG stockpile will be constructed with a 2 m base of NAF waste material. The LG stockpile was designed to a capacity exceeding the maximum scheduled stock balance (Table 100) as a contingency for operational adjustments. In addition to the excess stockpile capacity, it is planned that a component of the low-grade stocks will be used during ROM pad construction thus adding further contingency to the low-grade stockpile design.

Item	Volume (Mm^3)
Maximum LG Stock Balance @ 30% swell	7.46
LG Stockpile Design Capacity	7.78

Table 100 Low-Grade Stockpile Design Capacity

13.10.3 Run of Mine Pad

A ROM pad has been designed for access to the crusher pocket and for temporary storage of MG and HG material that is not fed directly into the crusher. The access to the ROM pad has been designed at the final pit exit and shares an access to the waste dump. The location of the ROM pad in relation to the pit, waste dump and processing plant can be seen in Figure 97. The ROM pad will be constructed with a 2 m base of NAF waste material sourced from the pit with an upper layer and constructed from both NAF waste and low-grade ore material. Provisions have been made in the mining schedule for the presentation of construction materials with the full balance of required material available by month nine from mining commencement. Any MG or HG material not directly fed to the crusher during this period will be stored on temporary stockpiles at appropriate locations adjacent to the crusher.

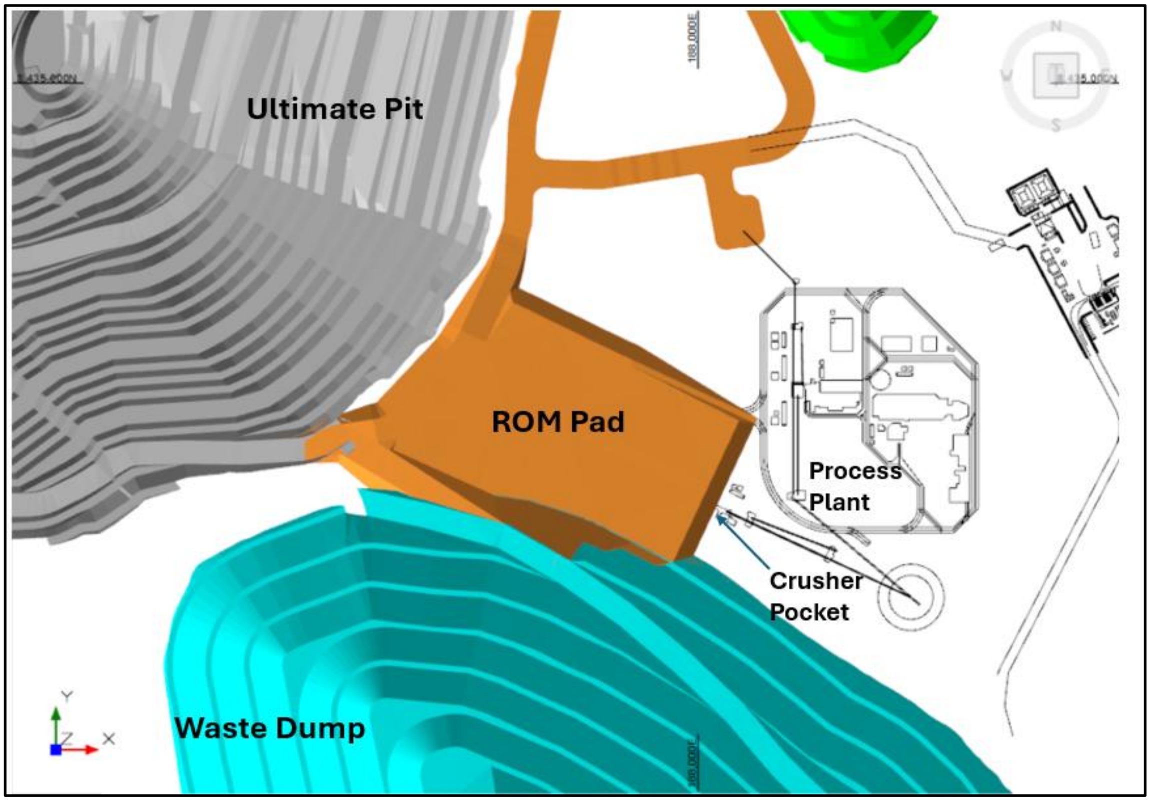


Figure 97 ROM Pad Location, Mining Plus 2025

Utilizing four temporary stockpiles for blending capability, a dual lane running track and light vehicle access around each temporary stockpile, the ROM pad has the capacity to store approximately 1 Mt of ore at any one time. Ore stock balances above this capacity will be stored temporarily within the LG stockpile footprint for rehandle to the ROM as required. Figure 98 illustrates the potential layout of MG/HG temporary stockpiles on the ROM pad surface.

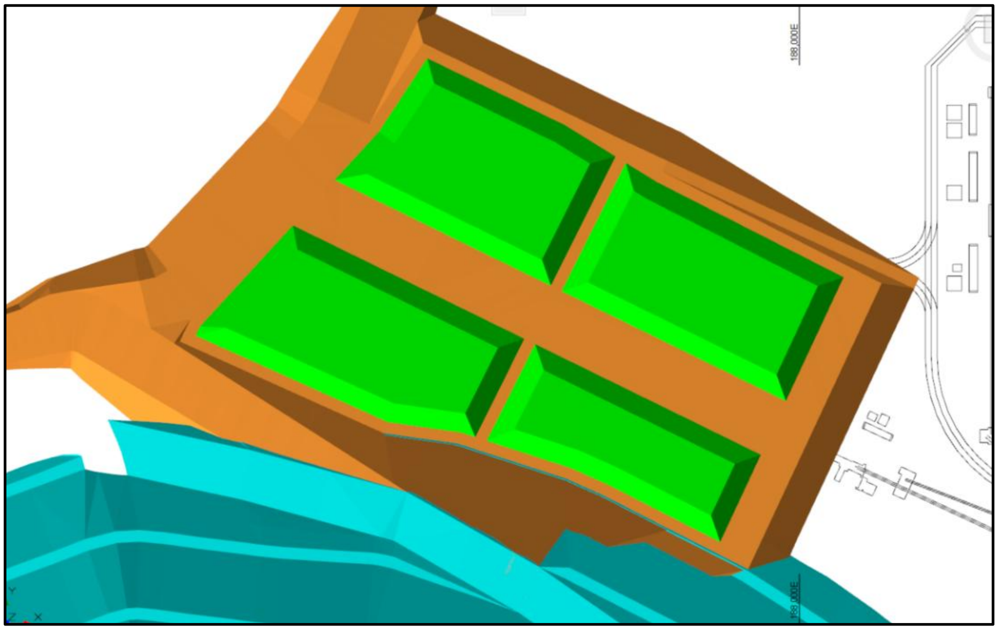


Figure 98 ROM Stockpile Design, Mining Plus 2025

13.10.4 Mining Infrastructure

The mining infrastructure for the Project has been engineered and designed by the mining contractor to support the operational, logistical, and administrative requirements of the mining operation. Key components of the mining infrastructure include a fully equipped heavy vehicle workshop, designed to accommodate scheduled and unscheduled maintenance of the mining fleet, with provisions for overhead cranes, service bays, and parts storage. A dedicated go-line and fuel farm have been incorporated to facilitate efficient dispatching of mobile equipment and secure bulk fuel storage and distribution, ensuring compliance with environmental and safety standards. Administrative functions are supported by purpose-built office facilities, which include operational control rooms, meeting spaces, and amenities for site personnel. Additionally, a secure explosives magazine and associated access road have been designed in accordance with regulatory guidelines for the safe storage and handling of blasting materials, with controlled access and separation distances maintained to mitigate risk

13.11 Personnel

Vista's workforce planning strategy is structured to align with the operational demands of each Project phase. Mining operations will be supported by approximately 90% personnel on rotating rosters, ensuring 24/7 coverage where required. Personnel numbers have been estimated in three categories:

- Vista personnel.
- Mining Contractor personnel.
- Mining Infrastructure Contractor (Construction Crew) personnel.

During the construction phase, personnel requirements are minimal, reflecting the limited scope of early-stage construction activities. As the Project transitions into full-scale mining operations, staffing levels are stabilized to support continuous production and technical services. In the final stockpile reclamation phase personnel numbers are reduced in accordance with the declining operational intensity. The Vista mining team comprising mining management and technical services, as per the positions and assumed rosters outlined below. Positions are assumed to be FIFO also for this Technical Report Summary, with a focus to place personnel locally, if possible, particularly for roles such as Pit Geotech and Geology Technician, and Surveyors.

- Chief Mining Engineer (roster 8/6).
- Senior Mine Planning Engineer (roster 8/6).
- Mining Engineer (roster 8/6).
- Mine surveyor (roster 14/7).
- Surveying Helper (roster 14/7).
- Geology Superintendent (roster 8/6).
- Grade Control Geologist (roster 14/7).
- Senior Geotech Eng (roster 8/6).
- Pit Geotech Technician (roster 8/6).
- Geology Field Technician (roster 8/6).
- Contract Management Superintendent (roster 5/2).
- Project Engineer (roster 8/6).

Table 101 shows the Vista positions required for each phase of the Project.

Vista Mining Personnel	Roster	Construction	Mining Operations	Stockpile Reclamation
Chief Mining Engineer	8/6	1	1	1
Senior Mine Planning Engineer	8/6	0	1	0
Mining Engineer	8/6	0	1	1
Mine Surveyor	14/7	0	4	4
Surveying Helper	14/7	0	4	4
Geology Superintendent	8/6	1	1	0
Grade Control Geologist	14/7	0	2	2
Senior Geotech Engineer	8/6	1	1	0
Pit Geotech Technician	8/6	1	1	0
Geology Field Technician	8/6	0	2	2
Contract Management Superintendent	5/2	1	1	1
Project Engineer	8/6	0	1	1

Table 101 Vista Mining Personnel Numbers

Similar to Vista internal workforce strategy, the mining contractor's personnel deployment is phase-dependent and aligned with operational requirements. During the construction phase, a reduced contractor workforce is maintained, primarily to support civil works and infrastructure development activities. As the Project transitions into active mining operations, personnel numbers stabilize, with fluctuations occurring only within maintenance and operator roles in response to variations in fleet size and equipment utilization. In the final stockpile reclamation phase contractor staffing is scaled down to a minimal level, reflecting the reduced operational complexity and lower equipment demand.

Mining contractor personnel numbers required for each phase of the Project are shown in Table 102.

The mining contractor operations team is assumed to be on FIFO rosters from other parts of Australia. This is based on the contractor's recent experience at other projects in the Northern Territory and across Australia. It was assumed recruitment and procurement of personnel would likely be achieved with 50% from Perth and 50% from Brisbane. This has been assumed for this Technical Report Summary and the costing of flights etc. required to support this assumption.

The mining contractor operations (direct) personnel is detailed by position and by year within the six areas outlined below, and also showing the corresponding roster, construction labor for mining infrastructure is found in Table 102:

- Staff/Supervision (roster 8/6).
- Maintenance (roster 14/7).
- Operators (roster 14/7).
- Dewatering Labor (roster 14/7).
- Shotfirers (roster 14/7).
- Blast Crew Labor (roster 14/7).

Mining Contractor Direct Personnel	Roster	Construction	Mining Operations	Stockpile Reclamation
Staff/Supervision	8/6	15	30	3
Maintenance	14/7	12	30 (max)	6
Labor/operators	14/7	0	120 (max)	9
Dewatering Labor	14:/7	0	3	0
Shotfirers	14/7	0	2	0
Blast Crew Labor	14/7	0	3	0

Table 102 Mining Contractor Personnel Numbers

Mining infrastructure contractor personnel are required exclusively during the construction phase of the Project, as detailed in Table 103. Their deployment is aligned with the execution of civil, structural, and utility installation activities associated with site establishment. This includes the development of support facilities such as workshops, warehouses, administration buildings, water management systems, and fuel storage infrastructure. Once construction is complete and the site transitions into operational readiness, the requirement for mining infrastructure contractor personnel ceases, with ongoing facility maintenance and operational support absorbed by the permanent site workforce.

Mining Infrastructure Construction Contractor Personnel	Construction
Project Manger	1
Supervisor	1
Engineer	1
HSE (Health, Safety and Environment)	1
Internal Labor	5
Batch Plant	4
Concrete Works	10
Office Building Works	4
Structural Steel Works	8
Pond Lining Works	4
Fencing Works	4
Lube Farm	5
Wash Bay/OWS Works	5
Electrical Works	6
Miscellaneous	1

Table 103 Mining Infrastructure Contractor Personnel Numbers

Work Area	Roster	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33
Owner's Personnel																																				
Chief Mining Engineer	8/6	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Senior Mine Planning Engineer	8/6	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0	0
Mining Engineer	8/6	0	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Surveyor	14/7	0	0	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2
Surveying Helper	14/7	0	0	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2
Geology Superintendent	8/6	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0	0
Grade Control Geologist	14/7	0	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Senior Geotech Eng	8/6	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0	0
Pit Geotech Technician	8/6	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0	0
Geology Field Technician	8/6	0	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Contract Management Superintendent	5/2	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Project Engineer	8/6	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mining Contractor Personnel																																				
Staff / Supervision	8/6	15	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	30	15	3	3	3	3	3	3	3
Maintenance	14/7	12	14	21	22	22	27	24	27	30	30	27	27	27	27	27	27	30	30	27	27	30	30	30	30	30	30	12	6	4	4	6	6	6	6	3
Operators	14/7	0	60	63	78	78	86	78	88	91	94	87	89	86	86	86	91	93	101	87	91	103	104	107	112	120	123	33	14	9	9	9	9	9	9	6
Dewatering Labor	14/7	0	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	0	0	0	0	0	0	0
Shotfirers	14/7	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	0	0	0	0	0	0	0
Blastcrew Labor	14/7	0	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	0	0	0	0	0	0	0
Mining Infrastructure Construction Personnel																																				
Project Manager		1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Supervisor		1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Engineer		1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Survey		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
HSE (Health, Safety, Environment)		1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Internal Labor		5	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Batch Plant		4	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Concrete Works		10	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Office Building Works		4	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Structural Steel Works		8	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Pond Lining Works		4	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Fencing Works		4	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Fuel Farm		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Lube Farm		5	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Wash Bay/OWS Works		5	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Electrical Works		6	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Miscellaneous		1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0

14. PROCESS AND RECOVERY METHODS

14.1 Introduction

The Project processing facility has been designed to process 15 ktpd of hard ore from the Batman open pit, equating to an annualized throughput rate of 5.325 Mtpa .

The processing plant has been designed to operate at a nominal treatment rate of 679 tph based on 7,838 operating hours per year or a grinding circuit utilization rate of 92% (based on 355 available operating days per year) The crushing circuit has been designed at a higher nominal treatment rate of 868 tph based on 6,134 operating hours per year or an annual utilization rate of 72% (based on 355 available operating days per year). Utilization is defined as the percentage of total time that the process plant operates with feed, while availability refers to the percentage of total time that the process plant is mechanically and electrically ready to operate.

The processing facility unit processes are based on well proven technologies for gold recovery and treatment of hard ore, following a processing route consisting of:

- Primary crushing in open circuit using a gyratory crusher to produce a nominal product P_{80} size of 120 mm.
- Secondary crushing incorporating a cone crusher operating in closed circuit with a double deck vibrating screen to produce a nominal circuit product P_{80} size of 30 mm.
- Coarse ore stockpile (COS) with reclaim via apron feeders.
- Tertiary crushing circuit incorporating a HPGR in closed circuit with two double deck wet vibrating screens to produce a nominal circuit product P_{80} size of 3.25 mm.
- X-ray transmission ore sorting on the HPGR product screen top deck oversize -29.5+16 mm fraction.
- Primary grinding in an overflow ball mill operating in closed circuit with hydro-cyclones to a produce P_{80} of 250 µm.
- Secondary grinding utilizing four Vertimill mills operating in parallel to produce P_{80} of 40 µm.
- Pre-leach thickening to produce an underflow of increased pulp density suitable for leaching.
- Conditioning/pre-oxidation of the leach feed slurry with lead nitrate and oxygen in two agitated pre-conditioning/oxidation tanks.
- Direct cyanide leaching and adsorption in a hybrid CIL circuit comprising two leach tanks followed by six adsorption tanks.
- Acid washing and elution of the loaded carbon in a split AARL elution circuit, and thermal regeneration of the barren carbon prior to its return to the CIL circuit.
- Smelting of cathode sludge from electrowinning to produce the final gold doré product.
- Detoxification of leach tailings via the Air/SO_2 process.

- Transfer of the final tailings to the TSF for deposition with the supernatant water recovered from the surface of the TSF for re-use in the process plant.

An overall schematic flowsheet depicting the unit operations incorporated in the selected process flowsheet is presented in Figure 99.

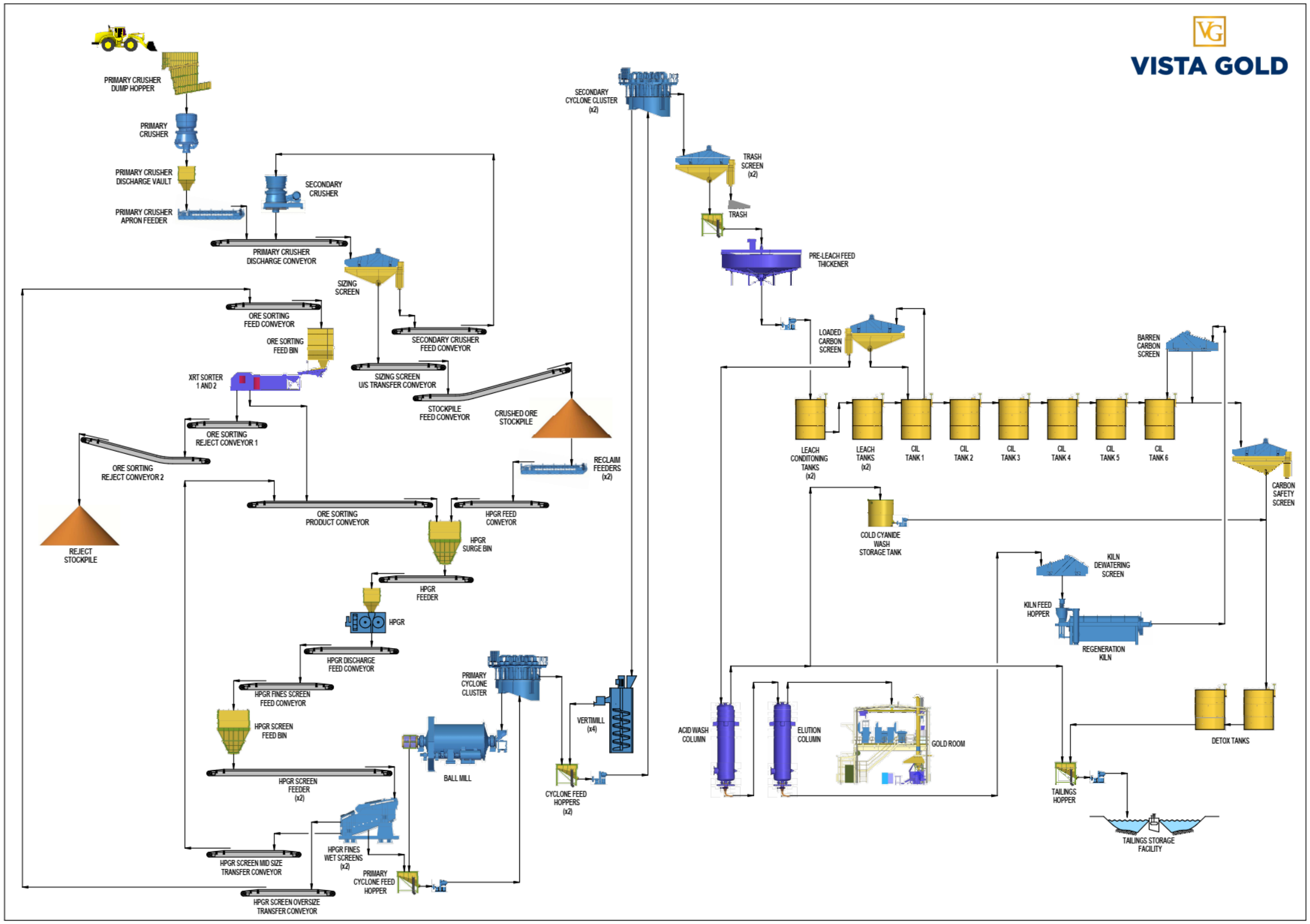


Figure 99 Process Flow Schematic Diagram, GRES 2025

14.2 Process Design Criteria

A detailed design criteria has been developed for the 15 ktpd throughput configuration. The process plant has used the following key design criteria listed in Table 104

Description	Unit	15 ktpd
Annual Ore Feed Rate (ROM feed)	Mtpa	5.325
Operating Days per Year	d/a	355
Daily Ore Feed Rate (ROM feed)	tpd	15,000
ROM Feed F_{100} Size to Primary Crushing	mm	1,000
ROM Feed F_{80} Size to Primary Crushing	mm	711
Crushing Rate (6,134 hours per year)	tph	868
HPGR New Feed Rate (7,838 hours per year)	tph	679
HPGR Screen Recircuit Circulating Load	%	100
Ore Sorting Rate (7,838 hours per year)	tph	121
Milling Rate (7,838 hours per year)	tph	624
Primary Milling Circuit Circulating Load	%	220
Gold Head Grade (ROM Feed) – Design	g Au/t	1.29
Gold Head Grade (ROM Feed) – LOM Average	g Au/t	0.97
Gold Head Grade (Mill Feed) – Design	g Au/t	1.38
Silver Head Grade	g Au/t	1.58
Copper Head Grade	ppm	550
Cyanide Soluble Copper	ppm	65
Design Ore Specific Gravity	t/m^3	2.76
Design Abrasion Index	-	0.23
Design Crushing Work Index	kWh/t	20.0
SMC Drop Weight Index	kWh/m^3	12.95
Design Rod Mill Work Index	kWh/t	22.6
Design Ball Mill Work Index	kWh/t	24.5
Primary Grind P_{80} Size to Secondary Grind	µm	250
Secondary Grind P_{80} Size to Leach	µm	40
Leach System	-	Hybrid CIL
Leach Slurry Density	% solids w/w	45
Total Leach and Adsorption Time - Design	H	30
Elution System	-	Split AARL
Final Tailings Cyanide Destruction Type	-	Air/SO_2
Overall Recovery (LOM Average)	% Au	88.5
Maximum Gold Extraction – Design	%	91.6

Table 104 Key Process Design Criteria

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

14.3 Crushing

The crushing circuit will be based on a primary gyratory crusher in combination with a secondary cone crusher. Product from the crushing circuit will be conveyed to the coarse ore stockpile (COS). The circuit will crush 868 dry tph of ore to a nominal product size P_{80} of 30 mm.

Run-of-mine (ROM) ore will be trucked using 190 tonne class rigid-frame haul trucks from the mine and either directly tipped into the ROM Bin or stockpiled on an earthen ROM. Stockpiled ROM ore will be reclaimed using 130 tonne 12 m^3 Front End Loader which will then tip directly into the dual tip primary ROM bin.

Any oversize material fed into the dump hopper will be broken using a fixed rock breaker. The primary crusher will be a 50-65 gyratory crusher equipped with a 375 kW motor. The primary gyratory crusher will have a nominal top size capacity of 1,270 mm and will be operated with an open-side setting (OSS) of 140 mm and a 32 mm stroke.

The primary crusher will discharge into the crusher vault. Ore will be reclaimed via the discharge apron feeder onto the primary crusher discharge conveyor. A self-cleaning magnet located at the discharge conveyor head chute will remove magnetic tramp metal from the ore stream and discharge it into a tramp-metal bunker. The crusher discharge conveyor will feed onto the sizing screen feed conveyor, discharging into the sizing screen feed bin.

Ore will be reclaimed from the feed bin via a vibratory sizing screen feeder to maintain a steady feed rate to the primary sizing screen. Oversize from the sizing screen will be conveyed via the secondary crusher feed conveyor to the secondary crusher feed bin. Level control on the bin and cavity control for the secondary crusher are maintained via a vibratory feeder to ensure a steady throughput. The secondary crusher will reduce the sizing screen oversize F80 of 137 mm to the required product P_{80} of 47.8 mm. The crusher will operate with a closed-size setting (CSS) of 40 mm.

The secondary crusher is located above the sizing screen feed conveyor and will discharge directly onto the primary crushed ore stream heading to the sizing screen. The sizing screen undersize will be the final crushed product and will be conveyed by the stockpile feed conveyor, which will discharge ore to the crushed ore stockpile (COS).

This stockpile will have a nominal live storage capacity of 32,600 tonnes, equivalent to 48 hours of milling time at the 5.325 Mtpa reclaim rate.

To minimize dust emissions, a primary crusher dust collector will be installed to service the primary crusher discharge chamber, the secondary crusher discharge and the sizing screen discharge. In addition, the primary crusher dump hopper, the secondary crusher feed conveyor and the sizing screen feed conveyor will be serviced by dust suppression water sprays.

14.4 Coarse Ore Storage - HPGR Circuit and Ore Sorter

The crushed ore will be reclaimed from the crushed ore stockpile using a combination of two apron feeders under the stockpile, which discharge ore onto the HPGR feed conveyor which runs within the tunnel beneath the stockpile. The HPGR feed conveyor will feed the HPGR circuit with the final product being the wet screen undersize reporting to the mill discharge sump.

The reclaim apron feeder will be 1.2 m wide by 8 m long, equipped with a variable speed drive. The reclaim rate to the HPGR circuit will be controlled via a weight meter located on the HPGR feed conveyor. Conveyed reclaimed ore will discharge from the HPGR feed conveyor into the HPGR surge bin, prior to entering the HPGR circuit.

New ore, along with oversize from HPGR fines screen, will be combined on the HPGR feed conveyor. Ore feed to the HPGR will be controlled via the variable speed HPGR feeder. A self-cleaning magnet will be located upstream of the surge bin to remove metal. In addition, a metal detector and bypass flop-gate system will be located on the HPGR feeder to divert ore containing possible metal to a bunker for metal removal.

The HPGR will have a 2.0 m roll diameter with a 1.5 m face width, running at 1.3 m/s. Feed top size of 31.5 mm, will be reduced to a P_{80} 3.25 mm. All HPGR product will discharge onto the HPGR fine screen feed conveyor. The conveyor will deliver the HPGR crushed product to the HPGR fine screen feed conveyor. The bin will be equipped with two reclaim-belt HPGR screen feeders. Each feeder will provide a constant steady flow rate to the HPGR screen pulping box prior to the HPGR fine screens.

The HPGR screens will be double-deck, 4.2 m wide by 8.5 m long with upper deck aperture of 16 mm and lower deck aperture of 4.5 mm. A pulping box will slurry the crushed ore to approximately 67% solids prior to delivery to the screen feed box. Screen sprays on both decks will assist with separation and washing of fines from the HPGR product. The screen undersize, at a nominal density of 50% solids and P_{80} of 3.25 mm, will report to the primary cyclone feed hopper. The HPGR fine screens are located adjacent to the primary ball mill to utilize a common product discharge hopper.

HPGR screen bottom-deck oversize, at a nominal moisture content of 87%, will be conveyed by HPGR screen midsize transfer conveyor to the ore sorting product conveyor for return to the HPGR surge bin.

HPGR screen top-deck oversize, at a nominal moisture content of 87%, will be conveyed by HPGR screen oversize transfer conveyor. Ore will be transferring to the ore sorting feed conveyor prior to entering the ore sorting feed bin. The bin will have a bifurcated trouser-leg arrangement to feed two primary XRT ore sorters. Each XRT unit will be equipped with a vibratory pan feeder to control the reclaim rate from the ore sorting feed bin.

Rejects from the ore sorting system will be conveyed to a rejects stockpile via the ore sorting reject conveyors. The product stream will gravitate from the ore sorter units to the ore sorting product conveyor and will be mixed with the return stream from the lower deck of the HPGR screens, prior to transfer back to the HPGR surge bin.

Rejects will discard approximately 55 tph, reducing the mill circuit feed rate to 624 tph.

14.5 Grinding and Classification

The HPGR-ball milling circuit configuration is conventional, with the primary ball mill fed by the underflow from the primary cyclones. HPGR fine screen undersize slurry will be combined with ball mill discharge slurry in a common primary cyclone feed hopper.

The ball mill will be a 7.16 m diameter (IS) by 9.75 m long (EGL) overflow discharge mill with a dual-pinion variable-speed drives, each with 5,500 kW installed power, totalling 11,000 kW. The ball mill will be charged with 75 mm grinding balls and will be designed to operate with a ball charge of 28-29%. The ball charge will be adjusted to suit the ore type and power draw requirements. The mill will be fitted with discharge trommel screen (3.5 m Ø x 5 m) with 8 mm square-aperture rubber panels for oversize ball scat control.

Ball mill trommel undersize will flow by gravity into the primary cyclone feed hopper and be combined with HPGR fine screen undersize. One of two centrifugal slurry pumps, arranged in a duty/standby configuration, will pump mill discharge slurry to a cyclone cluster for classification. The cyclone cluster will consist of six 800 mm diameter cyclones (three operating and three standby) with an operating pressure of 55 kPa. The primary grinding circuit will operate with a cyclone overflow target P_{80} size of 250 µm. Cyclone underflow will gravitate to the primary ball mill.

Cyclone overflow will gravitate to the regrind feed box, allowing slurry flow to two secondary cyclone feed hoppers. Each hopper will be equipped with two pumps in a duty/standby arrangement to feed the secondary cyclone cluster.

The first secondary cyclone feed hopper will be equipped with two secondary cyclone feed pumps. The cyclone feed pump will supply slurry to the secondary cyclone cluster no 1. The cyclone cluster underflow launder will be equipped with two valves to control the feed to Vertimill.

The second secondary cyclone feed hopper will be equipped with two secondary cyclone feed pumps. The cyclone feed pump will supply slurry to the secondary cyclone cluster no 2. The cyclone cluster underflow launder will be equipped with two valves to control the feed to Vertimill.

Each cluster will be fitted with Cavex 250 mm diameter cyclones, with 22 operating and 4 spares per cluster. The cyclones will operate at 130 kPa with a target P_{80} of 40 µm for the leaching circuit.

Cyclone underflow will gravitate to the associated secondary grinding mills. The Vertimill model will be a VTM4500, equipped with a 3,350 kW motor. There will be a total of four mills for the required duty specification. The mills will be charged with 19 mm balls. The overall circuit has an installed power of 13,422 kW and will draw nominally 11,164 kW at the 5.325 Mtpa throughput requirement.

Cyclone overflows from the two secondary cyclone clusters will gravitate to two trash screens operating in parallel, one per cluster. The screens will be 3 m wide by 7.3 m long horizontal wet vibrating screens. The screen aperture will be 0.83 mm x 12 mm. Trash screen undersize will be pumped to the pre-leach thickener. Trash screen oversize will discharge into the trash bunker and be periodically removed for disposal.

The HPGR and primary grinding areas will be serviced by two vertical spindle centrifugal slurry pumps for clean-up. The secondary grinding area will also be serviced by two vertical spindle centrifugal slurry pumps for clean-up.

Grinding media will be delivered in bulk and stored in dedicated ball bunkers. Grinding balls will be added to the primary ball mill and secondary grinding mills by automated ball charging systems.

14.6 Leaching and Adsorption

Trash screen undersize will gravitate to the pre-leach thickener transfer hopper. One of two centrifugal slurry pumps, arranged in a duty/standby configuration, will pump the trash screen undersize slurry to the feed box of the pre-leach thickener. The thickener is designed to increase solids concentration from 34% on trash screen undersize to a design underflow of 62% solids, suitable for feed to the leach circuit.

The Project flowsheet includes two agitated pre-oxidation/conditioning tanks ahead of the leach circuit. Test work included four hours of pre-conditioning with lead nitrate at 50 g/t. To minimize the influence of the reactive iron sulfide minerals, the slurry is pre-treated by agitating with lime and oxygen prior to cyanide addition. An insoluble layer of ferric hydroxide is formed on the iron sulfide particle surfaces preventing further reaction. Addition of lead salts is also used to overcome the detrimental effects of sulfur species.

The two tanks will be the same size as the leach and adsorption circuit tanks, with a 3,750 m^3 nominal capacity. This will allow for 7.5 hours of pre-conditioning/oxidation prior to cyanide addition into the first leach tank.

The leaching and adsorption circuit will consist of two 3,750 m^3 leaching tanks and six 3,750 m^3 CIL tanks, with a nominal residence time of 30 hours at the 5.325 Mtpa treatment rate. The design will include the ability to bypass any tank in the train, should this be required.

Cyanide will be stage-dosed into the two leach tanks and the first three CIL tanks as required. A high-shear pump and reactor will be installed on both leach tanks to improve dissolved oxygen ($D0_2$) levels, with oxygen supplied to the leach reactor/mixer prior to leach tank 1. The remaining oxygen will be injected down the agitator shaft into the first and second CIL tanks as required to maintain the dissolved oxygen levels in the leach circuit prior to the CIL. Provision to supply oxygen to CIL 3 was included for cases where CIL 1 or 2 is bypassed and not in the leach/adsorption circuit.

Each CIL tank will have two 8.5 m^2 mechanically wiped, pumped intertank screens with 1 mm-aperture stainless steel wedge wire to retain carbon. The design carbon concentration will be 8 g/L. Carbon will be advanced through the CIL circuit counter-current to the pulp, on a batch basis, using recessed impeller pumps. Loaded carbon from the first stage of the CIL will be pumped to the loaded carbon screen. The loaded carbon screen will be a 1.5 m wide by 2.4 m long, horizontal wet vibrating screen. Loaded carbon from the loaded carbon screen will gravitate into the elution column. The design advance rate for the circuit is 12 tonnes per day based on the expected loaded carbon grade of 1,927 g Au/t, 1,410 g/t Ag and 195 g/t Cu. Barren carbon from the kiln (or directly from the elution column) will be returned to the circuit via the barren carbon screen. The barren carbon screen will be a 0.9 m wide by 2.40 m long, horizontal wet vibrating screen with a screen panel aperture of 1 mm x 12 mm.

The leaching and adsorption area will be serviced by three vertical spindle centrifugal slurry pumps for clean-up.

14.7 Carbon Handling and Gold Recovery

The carbon handling and gold recovery system will comprise the following:

- A 12 t mild steel, rubber lined acid wash and cold cyanide wash column.
- A 12 t stainless steel elution column.
- A 5 kW LPG-fired elution heater.
- A split AARL elution system with one 120 m^3 recycle elute and one 140 m^3 pregnant solution tanks.
- A cold cyanide rinse tank with 150 m^3 capacity and bleed pumps to transfer cold cyanide wash residue to the detoxification tank at a rate of 5.5 m^3/h.
- A 0.75 tonne per hour regeneration kiln and its associated quench tank.
- An eductor water system for carbon transfer.
- An electrowinning circuit with three 800 mm x 800 mm electrowinning cells with each cell fitted with 9 stainless steel cathodes and 10 anodes and supplied by an 1,800 A rectifier.

- A cathode station and filter to recover precious metal precipitate.
- A drying oven for the steel cathode wool.
- An A200 smelting furnace and crucible to produce gold doré.
- A secure gold room with a safe for the storage of bullion.

The elution circuit will utilize two columns, one for cold cyanide strip/acid washing and one for elution. The elution system has been designed for a nominal six elution cycles per week, however the elution frequency can be increased if necessary. The cold cyanide strip was required due to the possibility of high copper in solution in the leach circuit, resulting in high copper loading on the carbon. The ore source contains secondary copper minerals in some of the upper zone, with more chalcopyrite in the lower zone. The initial years of mining may be subject to higher cyanide-soluble copper in the feed.

The cold cyanide pre-strip solution of copper will be transferred to a dedicated cold cyanide rinse tank (3500-TK-007), allowing the cyanide bleed pump to transfer a steady state lower flow rate of high- concentration cyanide residue solution to the detoxification circuit.

A split AARL circuit was selected to reduce the requirement for fresh water.

The elution sequence will comprise the following process steps:

- Loaded carbon recovery over a period of approximately eight hours.
- Cold cyanide stripping with 2% cyanide and 2% caustic for 30 minutes, followed by a 120-minute water rinse step. Total cold cyanide strip/wash of 133 m^3 per elution cycle.
- Acid washing with 3% hydrochloric acid solution over 25 minutes.
- Rinsing of the carbon with raw water over 120 minutes, including preheating to 90°C during the final 25 minutes of the rinse stage.
- Transfer of the carbon from the acid wash column to the elution column over 60 minutes, requiring 120 m^3 of water.
- Pre-treatment of the carbon with hot (120°C) 3% cyanide and 3% caustic solution over 25 minutes.
- Elution of the loaded carbon with hot (120°C) recycle eluate over 125 minutes.
- Elution of the carbon with hot (120°C) potable water for 100 minutes.
- Cooling of the elution column with one bed volume of cold potable water over 25 minutes.

At the end of the elution sequence, the barren carbon will be transferred with raw water to the regeneration kiln dewatering screen. This screen will be a 0.9 m wide x 2.4 m long, horizontal wet vibrating screen. Carbon from the screen will gravitate into the feed hopper for the kiln. Carbon will be regenerated through the carbon regeneration kiln over a period of approximately 15 hours and will be collected in the carbon quench tank prior to being transferred back to the barren carbon screen in the CIL circuit.

The elution area and the gold room will each be serviced by a dedicated vertical spindle centrifugal slurry pump for clean-up.

14.8 Tailings Detoxification and Disposal

Final tailings from the leaching and adsorption circuit will be screened to recover any lost carbon, with the slurry passing the carbon safety screen via gravity to the first stage of the detoxification system.

The cyanide detoxification process will utilize the INCO Air/SO_2 detoxification process. The Air/SO_2 process is based on conversion of WAD cyanides to cyanate (OCN-) using a mixture of SO_2 and oxygen in the presence of a soluble copper catalyst at a controlled pH.

$$CN^- + SO_2 + O_2 + H_2O \rightarrow CNO^- + H_2SO_4$$

Dissolved copper (II) is required as a catalyst, typically at concentrations of 5 to 50 mg Cu/L. Sufficient soluble copper, in the form of a copper cyanide complex, is available in the leach tail solution based on leach test work of the ore source. This eliminates the need to add copper sulfate and associated mixing and reagent distribution for detoxification.

Slurry will gravitate from the carbon safety screen underpan to the first cyanide detoxification tank. Sodium metabisulfite (SMBS) and caustic will be added to the first tank. Control of the SMBS will be by a ratio controller and CN WAD analyzer. A pH probe in the first tank will control the lime addition to ensure that the formation of HCN is negated as the net reaction develops sulfuric acid. Air will be provided from the plant oxygen supply and will be directed through a high-shear reactor to improve the DO_2 level for detoxification.

Flow from the first tank will move to the second tank via an up-flow launder for further treatment before being discharged to a final tailings hopper.

The tailings slurry will be pumped to the TSF by one of two pump trains arranged in a duty/standby configuration. Return from the TSF, including decant return and seepage collection, will be returned to the PWP.

The tailings area will be serviced by one vertical spindle centrifugal slurry pump for clean-up.

14.9 Reagent Mixing, Storage and Distribution

The following process reagents and consumables will be required for the processing facilities:

- Steel grinding balls.
- Quicklime.
- Sodium Cyanide.
- Oxygen.

- Carbon.
- Lead Nitrate.
- Sodium Hydroxide.
- Hydrochloric Acid.
- Sodium Metabisulphite (SMBS).
- Flocculant.
- Liquefied petroleum gas (LPG).
- Smelting Fluxes.

These materials will be received on site, mixed and dosed, as outlined in Table 105.

Process Additive	Packaging	Mixing	Storage	Addition
Quicklime	Bulk	200 t silo, dust filter, feeder, slaking Mill	30 m^3 storage tank	Circulating pump, ring main, dosing valves
Sodium Cyanide	Sparge Isotainer System	Vendor dissolving tank	145 m^3 tank	Circulating pump, ring main, and dosing valves
Oxygen	Liquid Storage	Not required	18 tpd system	Reticulated from header and control panel to process equipment
Carbon	500 kg bags	Not required	-	By crane to CIL
Lead Nitrate	Bulk Bag	Mix system to 20%. 5 m^3 mixing tank	10 m^3 storage tank	Dosing pumps
Sodium Hydroxide	Bulk liquid (50% solution)	Not required	55 m^3 storage tank	Dosing pumps
Hydrochloric acid	Bulk liquid (32% solution)	Not required	30 m^3 storage tank	Dosing pumps
SMBS	1,200 kg Bulk Bag	9,600 kg per mix to 55 m^3 tank	100 m^3 storage tank	Dosing pumps
Flocculant	750 kg Bulk Bag	Automated mixing system to 0.25% solution 20 m^3 mixing tank	80 m^3 storage tank	Dosing pumps
LPG	Bulk	Not required	Vendor Supplied Bullet	Reticulated from main pipeline to process equipment
Smelting fluxes	25 kg bags	Not required	Warehouse	Manual
Steel balls (75 mm)	Bulk	Not required	Bunker	Automated charging system ball feeder, high lift conveyor, loading conveyor

Process Additive	Packaging	Mixing	Storage	Addition
Vertimill Hi-Chrome balls (19 mm)	Bulk	Not required	Bunker	Automated charging system ball feeder, high lift conveyor, loading conveyor, diversion gates

Table 105 Details of Grinding Media and Reagent Systems

14.10 Water Services

14.10.1 Raw Water Supply

The Raw Water Main Facility (RWMF) dam is supplied by rainwater catchment in the area and ground water catchment from seasonal inflows. The RWMF has a design storage capacity of 4.7 GL. Raw water currently flows by gravity from the RWMF to the existing plant infrastructure area. Provision for the supply and installation of two raw water supply pumps was included in the design as the new tank supply system may require additional head.

Raw Water will be supplied to a new raw/fire water tank located near the process plant.

14.10.2 Process Water

Plant process water will be sourced from the pre-leach thickener overflow, and the TSF decant return line and stored in a process water pond. If more process water is required, raw water will be supplied from the raw water main facility (RWMF) via a new Raw/Fire Water Tank. Two dedicated raw water pumps will supply the reagents system, with a bleed to the process water dam as required for make-up water requirements.

Additional inflow streams will include:

- WTP2 – RO Reject (plant system for elution and potable water requirement).
- WTP1 – Clean Water Pond bleed to the process water pond.

Process water will be distributed from the process water pond to various end-use points within the process plant by the process water pumps, which will be arranged in a duty/standby arrangement.

14.10.3 Raw Water (Plant Tank System)

Raw water pumps will be located at the RWMF with two duty/standby pumps to supply raw water to the raw/fire water tank: the raw water tank, which has a capacity of 400 m^3, and the Reverse Osmosis (RO) plant feed tank, which has a capacity of 30 m^3. The raw water tank will be located adjacent to the administration building and is the feed to the fire water pumps. The fire water will include a ring main around the plant site and infrastructure buildings, including the crushing circuit, administration office and mining contractor areas. The fire water ring main will be fed by an electric and diesel fire water pump and includes a jockey pump to maintain the main fire ring system pressure.

14.10.4 Potable Water – Water Treatment Plant 2

Potable water will be sourced from a raw water dam via the RO plant feed tank. A dedicated RO feed pump will supply the RO treatment plant (WTP2). The RO plant will supply the fresh water for elution (no chlorination) and the potable water. A chlorination unit will be required to ensure compliance with final potable water standards. After treatment, potable water will be held in the potable water tank, which has a capacity of 400 m^3. The potable water is fed into a ring main by two potable water pumps in a duty/standby arrangement.

There will be no potable service points or direct connection of potable water to process equipment to prevent contamination of the potable (drinking) water supply.

One of two potable water electric pumps, arranged in a duty/standby configuration, will supply the potable water services around the processing facility and non-process infrastructure.

Note: provision to supply RO water from WTP1 to the RO Tank is included. There is no provision to supply this water to the potable water system. This is to ensure that water sources from dirty water systems via WTP1 cannot be cross contaminated and used for human consumption.

14.10.5 Safety Shower Water

The potable water ring main will supply water to the safety shower tank. One of two safety shower water electric pumps, arranged in a duty/standby configuration, will supply the safety shower water services around the processing facility. Safety showers and eyewash stations are located throughout the plant site, including the crushing circuit, administration office, mining contractor areas and infrastructure buildings. All safety shower supply lines located a distance from the ring main will be thermally insulated to maintain safe water temperature during summer and heat-traced to prevent freezing in winter. Additionally, the safety showers will be fitted with a thermal relief valve to maintain safe temperatures.

If pressure within the ring main drops due to consumption from multiple service points, a pressurized accumulator vessel will be installed at each shower to ensure sufficient water volume is available for operation at the required pressure.

14.11 Compressed Air Services

A set of two wet screw air compressors with 110 kW motors will generate plant air, which will be stored in a plant air receiver with 3 m^3 capacity for distribution around the process facility. A separate air receiver will be located at the crusher area.

Plant air will be filtered and dried in a refrigerated instrument air dryer before being directed to a 1 m³ capacity instrument air receiver. Instrument air will be distributed to instruments throughout the plant from this air receiver.

14.12 Projected Energy, Water and Process Material Requirements

The projected energy, water and process material requirements are itemized Table 106.

Reagent	Consumption	Annual Usage
Ball Mill Steel Media	1.23 kg/t	6,015 t
Vertimill 19 mm Hi-Chrome	0.813 kg/t	3,976 t
Quicklime	3.1 kg/t	16,954 t
Sodium Cyanide	1 kg/t	5,328 t
Oxygen	995 g/t	6,443 t
Carbon	30 g/y	147 t
Lead Nitrate	50 g/t	245 t
Sodium Hydroxide	111 g/t	593 t
Hydrochloric Acid	842 kg/elution	264 t
SMBS	670 g/t	3,582 t
Flocculant	45 g/t	239 t
LPG	5,947 l/d	1408 t
Smelting Fluxes	0.001 kg/t	8 t
Water	0.55 m³/t	2,664,920 m³
Power	45 kWh/t	237,875,855 kWh

Table 106 Reagent and Media Estimated Annual Usage

14.13 Process Control System

The process plant control system will be a programmable logic controller (PLC) based system. The human machine interface (HMI) will utilize standard personal computers running Citect SCADA software to facilitate control. The process facility will be controlled from the centrally located main control room in the plant area.

4-20 mA analogue I/O signals will predominantly be associated with process instrumentation and control, including flow, pressure, density and the control of modulating valves and actuators, and variable speed drives.

Digital I/O will generally be based on 24 Visual Display Cabinets (VDC) hardwired signals, typically associated with the status and control of drives, valves and actuators and mechanical plant.

In each area, the I/O associated with the Motor Control Centers (MCC) will be installed in one or more tiers of the MCC and will be hard wired to the starter modules within the MCC. The digital and analogue I/O associated with the process instrumentation will be wired to Process Control Cubicles (PCCs).

Two Visual Display Units (VDUs) will be installed within the control room to provide operators with HMI. These units will present graphical process information in the form of trends, mimic pages, alarm summaries, logs and reports. This HMI will also enable the operator to start and stop equipment remotely, control variable speed drives and alter process set-points.

Controller parameters will be adjusted from the controller faceplate, and this adjustment can be password protected to prevent unauthorized changes. Display screens will be configured for the trending of individual or related parameters, and alarm pages will be developed to facilitate the setting of alarm points specific to various parameters. All analogue input signals, including outputs from flow, pressure, temperature and weighing instruments will be displayed appropriately on mimic pages. A short-term trend plot for each input and output from the system can be provided where required on the mimic pages.

The analogue and digital I/O associated with the plant instrumentation will be cabled to one or more PCC within the plant areas. These units will be located within the area switchrooms and house the PLC racks, instrumentation power supplies and communication hardware. Communication between these units and the control system HMI will be via Ethernet, using fiber optic or copper cable as appropriate.

External and emergency communication will be available in the control room.

The processing plant number of personnel required is detailed in Section 18.13.

15. PROJECT INFRASTRUCTURE

15.1 Mine Infrastructure

This Technical Report Summary assumes that all associated mine support infrastructure will be supplied, installed and operated by the mining contractor as part of their contractual mobilization and operating framework for the Project mining operations. This infrastructure is planned for construction during the pre-production period in year -1. Detailed design sizing, and layout will be finalized by the contractor during mobilization. Capital costs for this infrastructure are included in the contractor's mobilization and infrastructure establishment fees; no additional direct owner capital is required for this scope. A summary of the infrastructure is outlined below:

- HME workshop & warehouse – Main heavy equipment workshop with service bays, drills workshop, parts warehouse, tool crib and inventory management systems. Main maintenance areas are provided by six domes to fit 190 tonnes trucks with trays raised. The domes are planned to be replaced every ten years.
- Maintenance support facilities – Tire change pad, wash down area with oily water separator, light vehicle bay and lubricants store.
- Contractor laydown & storage yards – Secure fenced areas for consumables, spares, and bulk items, plus covered storage for weather sensitive materials.
- Fuel & lubricant farm – Double lined diesel and lube tanks with automated dispensing, spill containment and fire suppression; diesel supply will be secured under a separate fuel supply agreement. The existing site fuel facility has been deemed sufficient to be utilized until the contractor's facility is established.
- Explosives storage & facilities – Licensed explosives magazines (ANFO, boosters, detonators) and a Mobile Processing Unit (MPU) for on-site emulsion manufacture and loading.
- Water cart filling point ("Turkey's Nest") – Pump or gravity fed standpipe connected to the raw water pond, enabling rapid water cart loading for dust suppression and other water management duties.
- Mine administration facility and personnel amenities – Administration office, meeting rooms, crib rooms, ablutions, change rooms and first aid facilities sized for the peak workforce.
- Information & communications technology – Site communications network (Wi Fi/4G/LTE mesh), fleet management and high precision GPS systems, fatigue monitoring and associated IT hardware/software for mining equipment.

All facilities will meet applicable Australian Standards, regulatory requirements and the Owner's HSE specifications. Ownership and maintenance responsibilities remain with the contractor for the life of the mining services agreement.

Where applicable, interfaces and tie-ins with permanent site infrastructure (power supply, communications backbone, water supply and site roads) will be coordinated between Vista team and the contractor during the early works phase.

Given the site's long operational life, personnel retention and performance will benefit from comfortable, safe and long-term infrastructure. A well-designed layout with concrete pads for workshops and other infrastructure based on a worst-case scenario rating for cyclonic zone B has been provided. The cost for a steel workshop structure was considerable and included utilizing an overhead crane and six bays high enough for 190 tonnes trucks to raise trays under cover. This workshop design was revised to use six domes mounted on double-stacked sea containers as workshop bays, allowing raised truck trays and resulting in considerable cost savings. The expected lifespan is ten years; replacement costs for the dome structures were allowed for every decade and are included in the overall Technical Report Summary capital cost estimation.

The concrete requirement and construction year in the establishment phase is shown in Table 107.

Description	Concrete Volume (cubic meters)	Year -1	Years X
Workshop including 10-year replacement	1540.0	x	x
24 m × 12 m Operations Office & 9 m × 12 m Maintenance Office	63.1	x	
24 m × 12 m Operations Shift Change (PSI/Crib/Ice/Locker)	70.7	x	
12 m × 12 m Maintenance Shift Change (PSI/Crib/Ice/Locker)			
2 m × 12 m × 3 m Ablutions – Operations & Maintenance			
Drill workshop	11.2	x	
Stores Facility – 60 m × 40 m Fenced Area with 40 ft Dome Tyre Bay – Uncovered Jack Slab with 40 ft Dome	274.8	x	
Wash bay	322.1	x	
Clean and dirty water ponds	2.9	x	
Comms, water, power and lighting reticulation		x	
Lubricant facility	71.0	x	
Magazine yard and mags	10.2	x	
AN Dome store and facilities	1016.0	x	

Table 107 Establishment Phase

Figure 100 shows an illustration of an example shelter proposed to be used for equipment maintenance and proposed by the mining contractor.

Figure 100 Equipment Maintenance by Mining Contractor, Mining Plus 2025

Regarding other site infrastructure, such as main haul roads: Initial pre-production will utilize existing haul roads and pit access, which are well established. Some maintenance, such as grading, will be required to reinstate roads to operational standards. Once mining commences, rock excavated will be used for the construction of longer-term roads, and the ROM pad and ongoing road construction and maintenance throughout the mine life. Main haul roads comprise access between the pit exits points, the ROM pad, LG stockpile, mine infrastructure area, and haulage to the TSFs will be constructed as required by the mine contractor utilizing their support equipment as provided for in their mining contractor costs submission.

Pit dewatering flow rates have been estimated as part of the hydrological studies for the Batman deposit as outlined elsewhere in this Technical Report Summary. These incremental flow rates and the staged pit designs were provided to the mine contractor, who then determined pumping requirements and costing. These costs are detailed in the overall site dewatering estimate, with the staged dewatering pumps based on the following increments:

- Dewatering Pumps Year 1 to 3.
- Dewatering Pumps Year 3 to 7.
- Dewatering Pumps Year 7 to 15.
- Dewatering Pumps Year 15 to 27.

An indicative schedule for mine infrastructure construction is shown in Table 108. This schedule will be refined as part of further Project development, with the main objective of ensuring the mine infrastructure is ready to support mobile equipment mobilization to site and the commencement of mining.

Activity Description	Start Date	End Date
Workshop (replaced every 10 years, except concrete pad)	July-27	Jun-27
Offices	Sept-27	Dec-27
PSI, Crib and ablutions	Jan-27	Dec-27
Drill workshop	Jan-27	Dec-27
Stores area and tire bay	Jan-27	Dec-27
Wash bay	Nov-27	Dec-27
Clean and dirty water ponds	Jan-27	Dec-27
Comms, water, power and lighting reticulation	Jan-27	Dec-27
Lubricant facility	Oct-27	Dec-27
Magazine yard and mags	Oct-27	Dec-27
AN Dome store and facilities	Jan-27	Dec-27

Table 108 Mining Infrastructure Construction Schedule

15.2 Process Plant Layout and Infrastructure

The plant site location is primarily determined by the location of the open pit, waste dumps and ROM pad, and by the use of site topography to minimize the impact of plant operations on surrounding areas outside the planned mining activities.

The processing plant layout will reflect the sequential nature of the processing operations, with ROM receival at one end of the facility that feeds to the crushing area, followed by the HPGR and grinding areas and finally the pre-leach thickening and leaching area. The gold room and reagents area are alongside the leaching area.

Water management ponds are located around the site to manage all of the site's water requirements. Design of the water management system and construction of these ponds is by others. The equipping of the ponds is included in the plant site scope.

Preliminary layout drawings for the site and process plant have been prepared and are shown in Figure 101 to Figure 103.

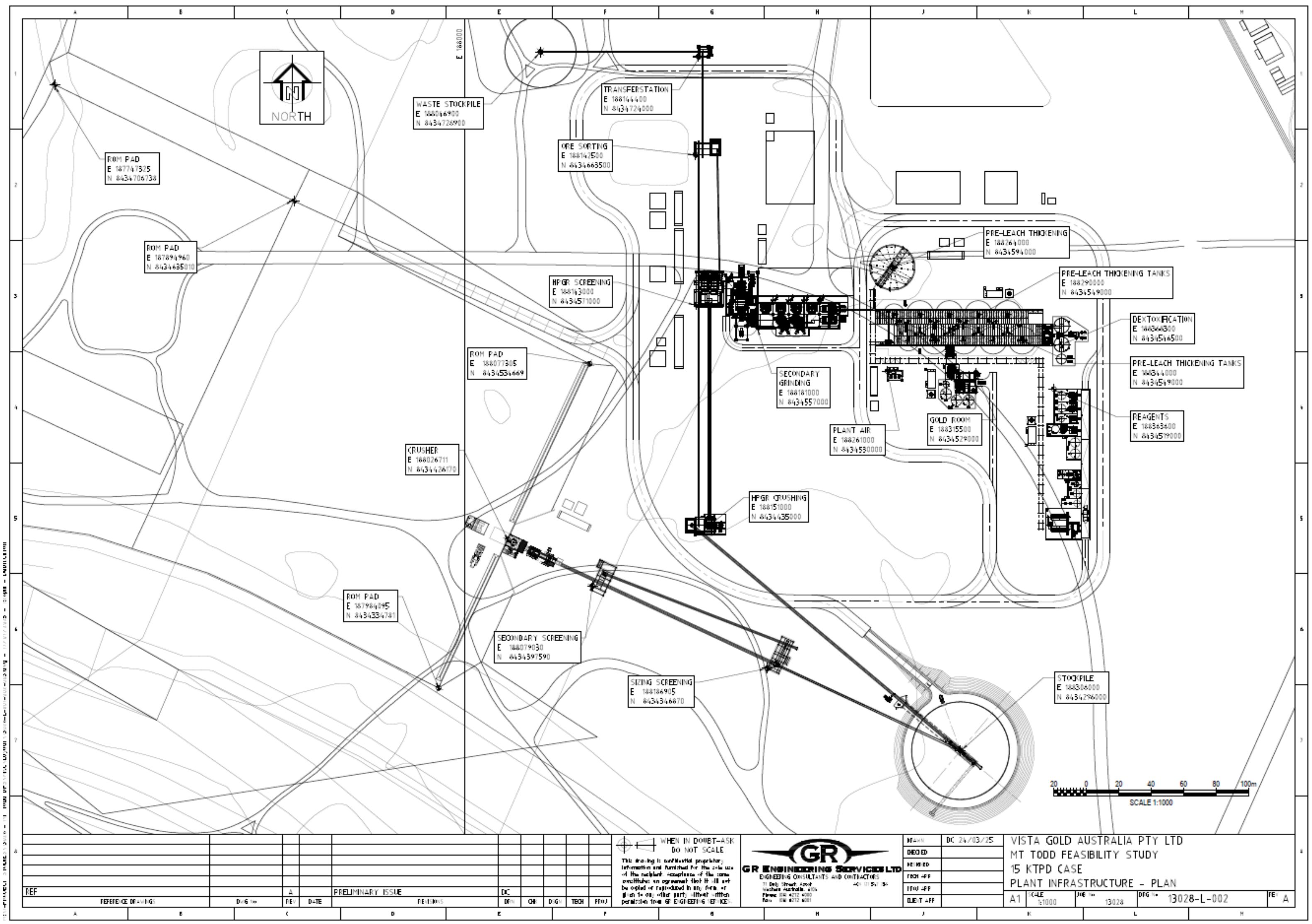


Figure 101 Overall Plant Layout

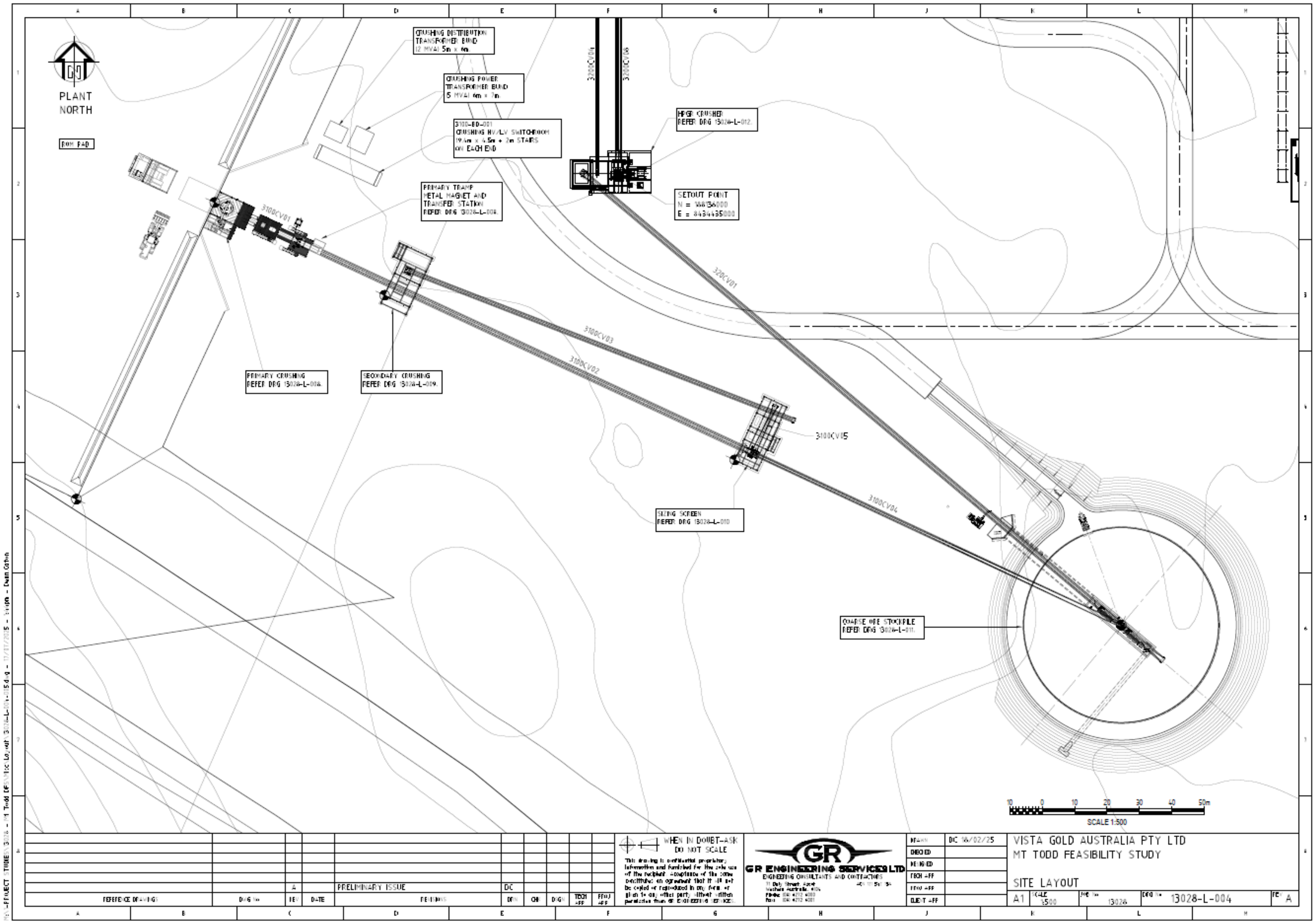


Figure 102 Location of Key Process Equipment for Crushing

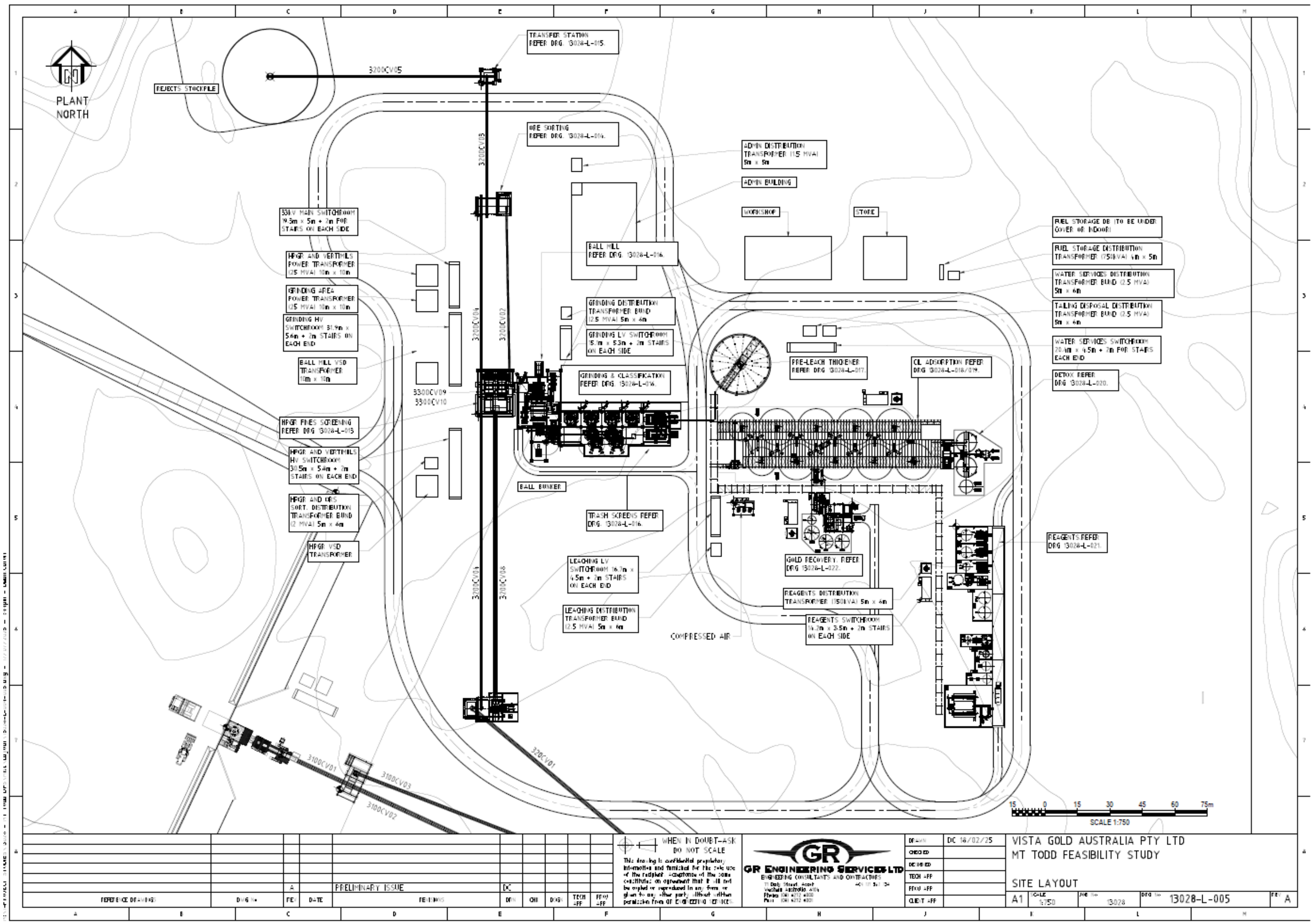


Figure 103 Location of Key Process Equipment and Infrastructure

15.2.1 Process Plant Engineering Design

The processing plant has been located and laid out to minimize the overall footprint, to facilitate the flow of process streams and to provide ease of operator and maintenance access.

Design complies with current Australian regulatory requirements and generally accepted good practice within the industry.

It is intended that the principles of good design incorporated in the preliminary engineering undertaken for this Technical Report Summary be continued and further developed during detailed engineering for the Project.

15.2.1.1 Earthworks and Drainage

The processing plant is in an area that slopes to the west, providing the necessary drainage falls across the site. The plant areas have been designed with six separate benched areas at different RLs to house the different areas of the plant. Each benched area is graded as required to optimize the topography. The benched areas are as follows:

- Crushing.
- HPGR.
- Wet screen and ball milling.
- Vertimill, CIL, gold room and reagents.
- Pre-leach thickening.
- Services.

Runoff from areas not subject to possible contamination will be collected and pumped around the plant area to rejoin the natural watercourse. Runoff from disturbed areas will report to sediment control dams to reduce sediment loads before being pumped to the main water management facility for use in the process plant.

15.2.1.2 Civil and Structural

Civil and structural work will comply with relevant Australian Standards.

All bunded areas will be suitably sloped toward area sumps, which will be located to facilitate easy access.

15.2.1.3 Mechanical

All plant and equipment have been designed for a 15-year life and been laid out with appropriate clearances and accessibility. Allowances in sustaining capital costs have been included to allow for the expected 33 year life span.

The mechanical design of the dry plant facilities has considered industry standard practice relating to material flow characteristics for conveyor, chute and bin design. All conveyors have been specified with due consideration for limiting spillage and conveyor drive motors have been sized for abnormal operating conditions such as blocked chute starts. Chute and bin liners of standard specification and thickness have been included in all areas that contact flowing material.

The mechanical design of the wet plant facilities has considered industry standard practice relating to slurry flow to ensure suspended flow and reduce excessive wear by using appropriate pipe sizing, slopes for gravity flow launders and pipes and allowing appropriate residence time in feed chutes. All slurry and solution pumps have been sized for an additional 15% flow to the process requirement.

Due consideration has been given to the nature of the various types of process and reagent slurries and solutions being present in the plant in terms of materials of construction, wear, corrosion and scaling.

All mechanical equipment has been selected through an evaluation process to ensure robust equipment suitable for the duty, sourced from proven equipment suppliers.

Where possible, equipment selection has been rationalized and standardized to reduce the spares inventory requirement.

15.2.2 Fuel Services

The diesel fuel services will be provided by Vista.

LPG will be supplied to site via road tanker to provide fuel for the elution heater, the regeneration kiln and the smelting furnace in the gold room. LPG storage will be supplied and installed at site. The vendor has indicated a 30,000 kL tank would be sufficient for the required plant operations.

15.2.3 Electrical

Power generation for the Project will be provided by natural gas reciprocating engines located in the power station around 2.2 km southeast of the process plant. This will be provided by a third-party operator as an over the fence supply carrying both a fixed and variable costs.

Power will be generated at 11 kV, then step-up transformers will convert the voltage to 33 kV. Dedicated 33 kV overhead power lines will transfer power to the mine site.

Eight major MCC’s will be installed to service major processing areas:

- Crushing.
- HPGR and ore sorting.
- Grinding.

- Leaching.
- Gold recovery.
- Reagents.
- Water services.
- Tailings disposal.

The switchrooms will be steel-constructed and clad buildings, insulated, air-conditioned and fully fitted. Switchroom linings will generally be made of non-combustible materials. VESDA smoke detection and hand-held fire extinguishers will be installed in major switchrooms.

The switchrooms will house the 415 V MCCs, variable speed drives, lighting and small power distribution boards and PLC cubicles.

All drives will be provided with full current isolation either in the MCC or the substation.

The PLC equipment associated with the motor control modules will be built into one or more tiers of the MCC and the PLC inputs and outputs (I/O) will be hard wired between drive modules and the PLC racks.

15.2.4 Process Plant Mobile Equipment

The plant mobile equipment to be purchased for the process plant is outlined in Table 109:

Equipment Description	Quantity
Light Vehicles	Quantity
Plant Manager Vehicle	1
Plant Utility Ops (Dual Cab)	2
Plant Utility Ops (Single Cab)	2
Plant Utility Maint (Single Cab)	2
Stores Utility (Single Cab)	2
Subtotal	**9**
Loader	Allowed for in mining
10 tonne all terrain truck with Hiab crane	1
3 tonne long tray truck	1
Integrated tool carrier	1
Forklift (3 t capacity)	2
Skid steer loader	2
Tele-handler 3 tonne	1
Container reach stacker – 30 t	1
25 t mobile crane (Franna or equivalent)	2
50 t mobile crane	1

Equipment Description	Quantity
Articulated boom lift	1
Subtotal	**15**

Table 109 Mobile Equipment for Process Plant

15.2.5 Process Plant Buildings

The process plant support buildings comprise of the building infrastructure for the process plant, see Table 110. The support building sizes and number of operations personnel have been developed in coordination.

Main Process Plant Buildings	Size
Administration/plant office, inclusive of a tropical roof	45 m x 30 m
Plant operations crib room (and breezeway to toilet block)	N/A
Maintenance workshop, include office	40 m x 20 m – 800 m^2
Stores building, including office	20 m x 20 m – 400 m^2
Reagents storage shed	20 m x 20 m – 400 m^2
Laboratory (sample prep)	N/A
Laboratory office (wet lab)	N/A

Table 110 Main Support Buildings

15.2.6 Administration Offices

The administration offices will consist of multiple transportable buildings and be used by mine management, plant management and administration personnel. The building system furniture and provide cellular and open-plan offices along with conference and meeting spaces.

The footprint of the administration offices is approximately 1,350 m^2 and will be sized to accommodate around 100 people.

15.2.7 Workshop/Warehouse

The workshop/warehouse will have a footprint of approximately 40 m x 20 m. Within this footprint, it will include offices, tool store, light vehicle (LV) workshop with engine repair area, vehicle service bays with an overhead crane, 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 adjacent to the external gas storage.

The building will include services such as overhead travelling crane, power, lighting, communications, compressed air, water, specialist equipment and other utilities necessary for the maintenance of process plant equipment and the LV fleet.

15.2.8 Sample Preparation and Laboratory

Laboratory buildings will be provided. It is envisaged that the laboratory will conduct analyses for both mining grade control and metallurgical samples. Laboratories typically include a dry sample preparation area with a drying oven and a dedicated dust collector, a weighing room and an analytical laboratory based on using atomic absorption spectrophotometer (AAS) analysis for gold, with fume hoods and a dedicated gas scrubber. A separate metallurgical testing area will be set up to include pulp sample filters, size analysis screens, bottle rolls etc. Additional space will be allocated for offices and storage areas

15.2.9 Control Building – Crushing

The crushing control room in the previous 2024 FS was a single transportable building located at the primary crusher. This single-person 3 m x 3 m control room has been removed in this Technical Report Summary and replaced by a workstation in the main control room.

15.2.10 Control Building – Main Control Building

The main control room will be located within the main administration building and include the necessary system furniture for one supervisor and three operators

15.3 Project Services

This section details the supply of all services outside the process plant.

15.3.1 Water Supply

This section covers the water supply to the process plant and between facilities (mine infrastructure, buildings, camp and TSF).

15.3.2 Water Treatment Plant (WTP)

A WTP will be fed with a combination of decant return, runoff pond water and pit dewatering. The WTP is designed for a nominal throughput of 300 m^3/hr and discharge to the Clean Water Pond (CWP). From the CWP, the water can either be used in the process plant, for dust suppression, or discharged to the Edith River during the wet season based on a rate that is determined by water chemistry and flow in the Edith River.

15.3.3 Raw Water

The raw water requirement will be approximately 340 m^3/hr for the process plant, fluctuating due to current operations and weather. A pump will be installed on the existing line from the Raw Water Dam (RWD) and will be used to convey the water to the operations

Raw water will be supplied to the mine support facilities via a 3 km supply line to the CWP or to the fire water tank.

15.3.4 Potable Water

Raw water from the RWD will be piped to the permanent camp location. A potable water treatment plant at the camp will produce potable water for the camp. A potable water treatment plant is located at the process plant and will supply potable water to the process plant and associated infrastructure.

15.3.5 Power Requirement and Distribution

A summary of the nominal predicted power requirements and distribution is provided in Table 111.

WBS Area	Description	15 ktpd Average Consumed Power (MW per annum)
2000	Mine including: ANFO, HV Workshop, Washdown, Mining Offices, Mines Support Services, Core shed.	0.5
All areas	Processing Plant	33.5
4000	Project Services including: ▪ Pit Dewatering: - Wastewater Treatment, Tailings Return, Diesel, Heap Leach, Bores.	2.5
6000	Permanent Accommodation	1
Total		**37.5**

Table 111 Predicted Power Requirements

15.3.6 Power Distribution

Power generation for the Project will be by natural gas reciprocating engines located in the power station around 2.2 km southeast of the process plant. This will be provided by a third-party operator as an over the fence supply carrying both fixed and variable costs, refer Section 15.7 for detail.

The power will be generated at 11 kV, then step-up transformers will convert the voltage to 33 kV. Dedicated 33 kV overhead power lines will transfer power to the mine site.

At the mine site, 33 kV will be used as the main HV reticulation voltage to feed different areas. As all the large motors on site, such as mills, will be operated at 11 kV, there will be step-down transformer on site to provide the necessary voltage level. Some distribution transformers will be selected with 11 kV primary.

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

The camp will have its own dedicated 33 kV overhead lines and a 33/0.433 kV transformer. A low-voltage distribution board (LVDB) at the end of the overhead line will supply power to the camp.

15.3.7 Communication

Voice and data communications will be established at the site data center by hardwired fiber and/or wireless connection. A complete IT and communications system will be provided for the Project to meet the requirements of a modern operating facility.

Primary operational communications on site will be via conventional two-way radio, which will be installed in vehicles, at specific fixed locations, and as portable radios for use by personnel. The radio system will support individual (one-to-one) calls, group (one-to-many) calls and broadcasting (one-to-all) calls.

Cell phone reception is currently available throughout most of the site. Repeater stations will be installed to ensure coverage in all required areas.

The communications system will provide internet and email access to the operational workforce, as well as telephone facilities amongst the workforce and to the outside world using Voice over Internet Protocol (VoIP). The system will manage data transfer to ensure that large file transfers will not affect telephone communication. The system will also allow service providers remote access to the company's servers to perform routine maintenance functions. Additionally, suppliers of equipment with communication-enabled hardware and software will be able to access their systems remotely and perform fault analyses.

A redundant fiber optic network will be installed throughout the plant for the process control network, extending from the central control room to all substations around the site. This network will be used for communications between process control system (PCS) servers, controllers and associated equipment. This network will be segregated from all others to ensure security and performance levels are maintained. Sufficient additional fibers have been included in this fiber ring to allow for the administration network (to include VOIP telephones, data and CCTV) and security network (access control systems, fire and security equipment). Additionally, fiber optic cable will be installed in a star topology to locations requiring the administration and/or security network, but not the process control network.

15.3.8 Security

Security monitoring (including CCTV) will be managed from the main guard house located at the main gate. Fencing will extend from access roads into landforms to inhibit uncontrolled entry to the site by people, domestic animals and wildlife.

15.3.9 Waste Disposal

Sewage waste disposal for the Project will be treated in three areas. A 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 WWTP with an associated spray field will also be installed at the construction camp, connected to the camp's sewer reticulation system. A smaller system will be installed near the gatehouse to service it, along with the administration and emergency services buildings.

Solid waste will be collected and disposed of within the mine site at the designated solid landfill waste area.

15.4 Project Infrastructure

This section provides a description of the Project Infrastructure required for the construction and operation of the process plant.

15.4.1 Site Preparation

Bulk earthworks for the process plant will be designed to minimize the import of fill material. Where fill material is required, material from the existing ROM pad ramp and from the existing stockpiles located adjacent to the Tollis and Golf Pits will be utilized.

The site will be prepared with 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 runoff around the plant. This will also reduce underground drainage and depth of open channels required on 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 collect water alongside the unsealed plant roads and direct them beneath the roads via corrugated steel culverts to prevent road scouring. All stormwater runoffs 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, with the new channel sized to accommodate a 1:100-year storm. Plant pad earthworks will be above the 1:100-year flood level within the new channel.

Some demolition of existing infrastructure will be required for construction of the new processing plant, and associated costs and time have been included in the capital cost estimate.

15.4.2 Emergency Services and Gatehouse Security

The Emergency Services Facilities will consist of a transportable building used by first aid and fire/emergency services personnel. It will be located at the site entrance from the main access road and will measure 12 m x 3 m. This area will include a covered ambulance bay and parking for additional services. Gatehouse/Security

The gatehouse/security facilities will consist of a single transportable building used by security personnel to record site access and conduct 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 at the site entrance, adjacent to the emergency services buildings.

15.4.3 Access Roads, Parking and Laydown

Access to the Project is via high quality, two-lane paved roads from the Stuart Highway, the main artery within the territory. While road infrastructure to the site is adequate for current exploration and development activities, no active railway or port infrastructure directly serves the Project area.

The existing plant access road is suitable for the current design, with minor road repairs to be carried out.

Additional light vehicle access roads will be constructed to access the mine workshop, permanent camp and process plant. All other existing roads will be utilized and upgraded with minor works, including the road network to the tailings and raw water dams.

Laydown areas for process plant construction have been identified and space allocated in this Technical Report Summary.

15.4.4 Permanent Accommodation

Vista will provide accommodation for 250 staff and visitors in the permanent village, located near Mt Todd as shown in Figure 104. Each room will include one bed and a private ensuite bathroom. The mess (kitchen and dining hall) facilities will be separated from the main accommodation blocks.

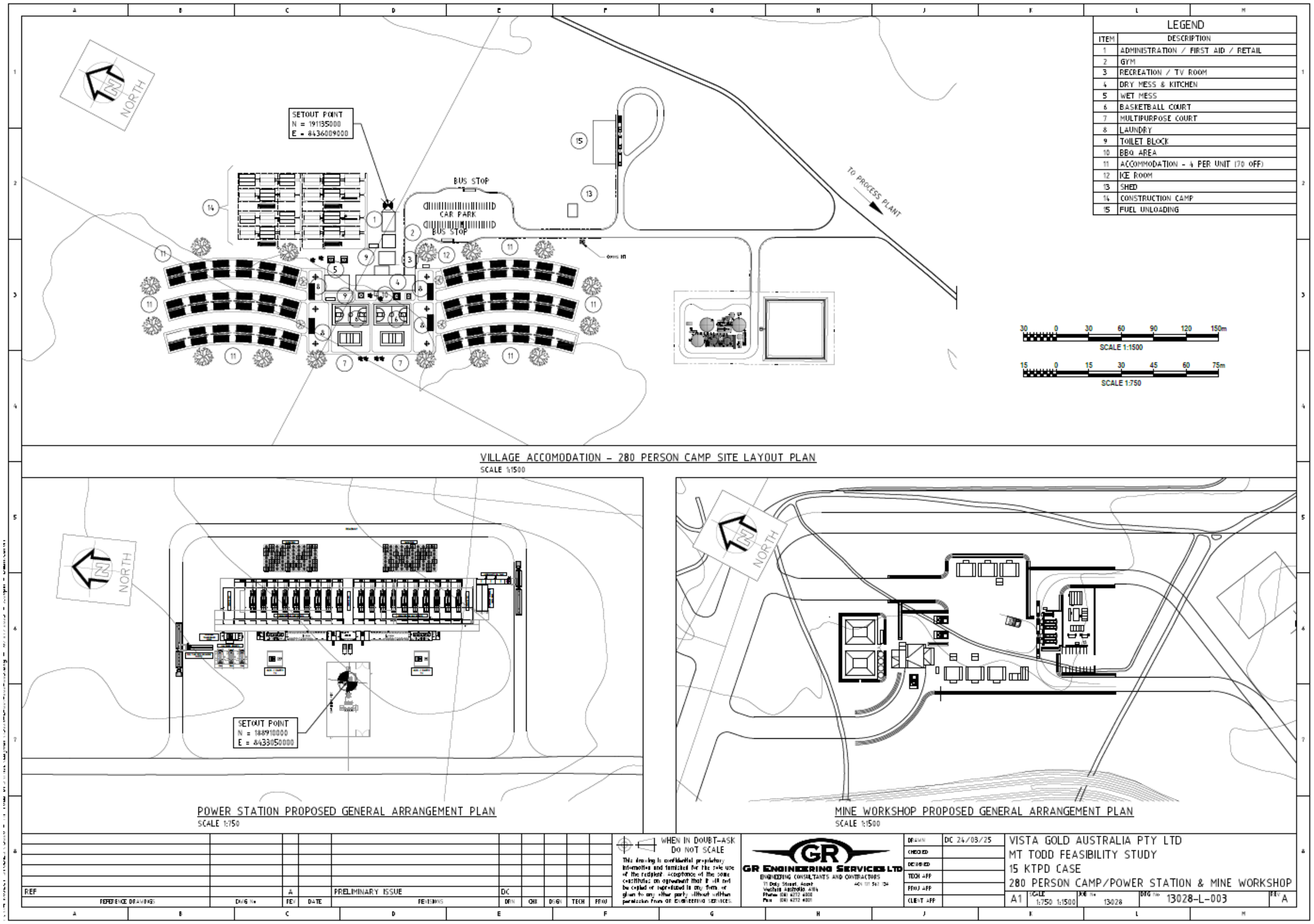


Figure 104 Permanent Camp

Vista's strategy is to provide bus transport for most of the workforce to and from camp each day.

A car parking area will be provided; however, car parking will be limited to site vehicles, and private vehicles will be discouraged.

15.4.4.1 Village Infrastructure

The village will include the following facilities:

- Accommodation units.
- Mess (kitchen and dining hall) building.
- Administration, first aid and storage building.
- Laundry buildings.
- Recreation building.
- Gymnasium.
- Gatehouse with boom gate.
- Bus stop shelter.
- Ablution buildings.

15.4.4.2 Village Services

15.4.4.2.1 Village Potable Water Supply System

Raw water will be supplied from the freshwater system to feed a dedicated RO potable water treatment plant for the village potable water system.

15.4.4.2.2 Village Wastewater System

Wastewater generated from the village will be collected in centralized pits and pumped to a dedicated Wastewater Treatment Plant (WWTP) located close to the village boundary.

Treated effluent will not be used within the village boundary. Clear effluent from the WWTP will be transferred to a nominated waste drying pond area located adjacent to the WWTP. Sludge from the WWTP will be collected and removed from the site as required on a regular basis by others.

15.4.4.2.3 Village Fire Water and Fire Protection System

Raw water will be supplied to the village boundary to feed the fire water system. Fire protection systems will comply with Northern Territory standards and regulations. Fire protection systems will include water storage tanks equipped with electric, jockey and diesel backup pump skids, instrumentation and controls, ring main piping, hose reels, hydrants, fire extinguishers, fire blankets, smoke detectors, alarms and all signage. All firefighting equipment will be visible at night using reflective high-visibility strips and localized lighting.

Fire hydrants, hose reels and extinguishers will be located throughout the village and portable fire extinguishers, and sprinkler systems will be installed throughout the buildings.

Fire detection and alarm systems will be included in all village buildings in accordance with the relevant regulations. Smoke detectors and alarms will be installed in each room or sleeping quarter in each building and throughout the amenities buildings.

15.4.4.2.4 Village Electrical Power Distribution and Communications

Power will be supplied to the permanent accommodation village via an overhead 33 kV cable from the processing plant, feeding a 2,000 kVA, 33 kV/433 V transformer. This transformer will supply the village main distribution board which will distribute power across the village as required. Village buildings will generally be equipped with a single distribution board for all internal building power distribution.

The village communications will be connected to the overall site communications.

15.4.5 Personnel Transport

A series of coaches will transport personnel from the village to the mine and process plant site. There will be a series of designated parking bays for mining, process plant and administration.

15.5 Tailings Storage Facilities

The tailings storage facilities (TSF1 and TSF2) are designed to have a combined capacity of 172.8 Mt, which exceeds the 159 Mt included in the current mine plan. TSF1 will be expanded using upstream embankment raises to contain an additional 90.9 Mt of tailings. TSF2 will be constructed on the eastern side of the Project site and will have a tailings capacity of 81.9 Mt. Based on a mill feed rate of 15 ktpd, the tailings storage facilities are expected to have a combined operational life of 33 years. Table 112 summarizes the parameters used for designing the tailings storage facilities.

Design Parameter	Value	Units
TSF1 Expansion		
Design Tailings Storage Capacity	90.9	million tonnes
Design Life	19	years
TSF2		
Design Tailings Storage Capacity	81.9	million tonnes
Design Life	14	years

Table 112 Tailings Storage Facility Design Parameters

Tailings storage capacity is calculated assuming an average tailings density of 1.5 tonnes per cubic meter (t/m^3) for the slurry. This density aligns with previous feasibility studies and is based on tailings densities using sub-aerial method of deposition.

15.5.1 Design Criteria

Design criteria are based on the Global Industry Standard for Tailings Management (GISTM) and the regional guidelines developed by the Australian National Committee on Large Dams (ANCOLD). ANCOLD has developed guidelines on the planning, design, construction, operation and closure of tailings dams, providing requirements based on the consequence category of the facility. The previous feasibility study for the Mt Todd Project assumed TSF1 would have a consequence category of High B. This assumption has been retained in this Technical Report Summary. Table 113 lists the design criteria according to the hazard classification of the site following ANCOLD guidelines.

Parameter		Value	Comments
Water Management	Extreme Storm Storage	1 in 1000 Annual Exceedance Probability (AEP)	The embankment construction schedule ensures that the raises are built at a pace to store the design storm event. Ultimate embankment height will provide sufficient storage during closure. No operational spillway is included.
	Minimum Freeboard	1 meter	
Geotechnical Stability	Static Factor of Safety (FOS)	1.5	
	Pseudo-Static FOS	1.0	ANCOLD does not provide guidelines for the minimum FOS under pseudo-static conditions. Criteria is based on engineering best practices.
	Post-Earthquake FOS	1.1	
	Design Earthquake	Safety Evaluation Earthquake (SEE): 1 in 5,000 AEP Post Closure 1 in 10,000 AEP	Peak ground acceleration (PGA) values were determined using the 2023 National Seismic Assessment (NSHA23).

Table 113 Tailings Storage Facility Design Criteria

Testing and findings from previous feasibility studies have been analyzed to evaluate the TSF1 expansion design and TSF2 designs. Material testing of embankment fill, Cone Penetration Testing (CPT) data of the currently impounded tailings, and results from laboratory testing of the new tailings grind were evaluated to assess the slope stability of both facilities. The liquefaction potential of the existing tailings was assessed through the interpretation of the CPT data. Liquefied tailings strengths were used in slope stability analyses presenting a conservative slope stability evaluation. This approach follows recommendations in GISTM and similar international guidelines.

The resulting designs for TSF1 and TSF2 meet or exceed the design requirements and recommendations stated in the GISTM and ANCOLD guidelines.

15.5.2 TSF1

TSF1 currently consists of a starter dyke (Stage 1) constructed using zoned earthfill, including an upstream low-permeability zone, a drainage/transition zone and a downstream shell constructed from mine waste. An underdrainage collection system was constructed across the entire basin of the facility and along the toe of the embankment. These drains minimize seepage loss throughout the facility foundation and improve tailings consolidation. Six decant towers were built to collect water from the supernatant pool directing flow to the return water pond at the southern embankment toe. Based on site reconnaissance, the decant system is still operational, but the status of the underdrainage system is unclear. Seepage analyses used a conservative approach assuming the underdrainage system is not functioning. A blanket drain consisting of angular waste rock or drain rock will be constructed on top of the existing tailings and extend at least 35 m from the upstream embankment face. This blanket drain will capture seepage and tailings consolidation water from new tailings deposition. The existing decant system will be raised to facilitate the reclamation of water for use in the process circuit.

The TSF1 expansion will be executed through a phased approach. Upstream raises will be constructed starting with Stage 2 (Stage 1 is the existing starter embankment) and continuing through Stage 8 with a crest elevation of 161 m. Run-of-Mine (ROM) non-acid forming (NAF) waste rock will be used for all upstream raises. A downstream waste rock shell will be constructed around the entire facility perimeter during Stage 6 construction. This waste rock shell is required to meet minimum slope stability requirements as TSF1 gets taller. Table 114 shows TSF1 stages with operational (design) life and storage capacity. TSF 1 is expected to operate for a total of 19 years.

Stage	Design Life (years)	Capacity (Mt)	
		Incremental	Cumulative
2	3.93	17.01	26.01[1]
3	2.71	13.27	39.28
4	2.71	13.26	52.54
5	2.44	11.92	64.46

Stage	Design Life (years)	Capacity (Mt)	
		Incremental	Cumulative
6	2.43	11.93	76.38
7	2.42	11.86	88.24
8	2.38	11.66	99.90

Notes: *[1] Includes the 9 million tonnes of tailings currently deposited.*

Table 114 TSF1 Construction Sequence

Figure 105 presents the typical section of the ultimate TSF1 embankment design.

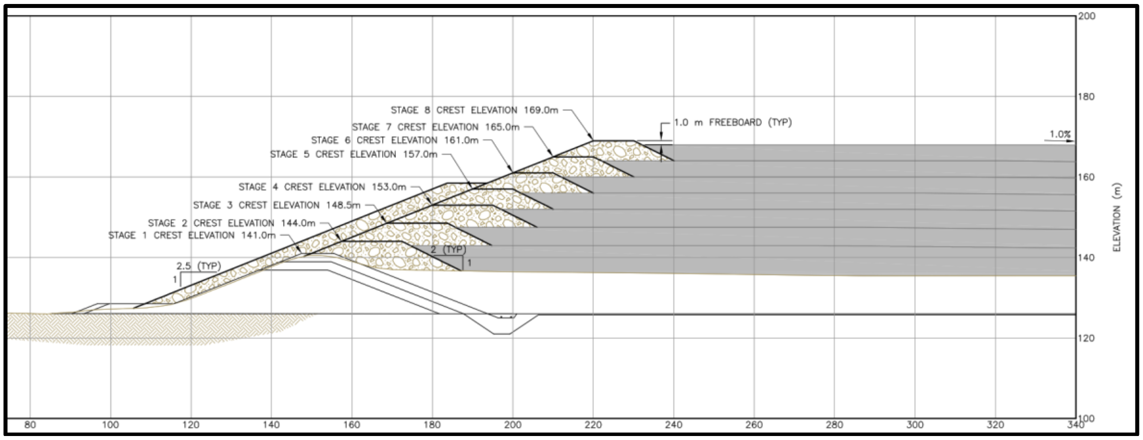


Figure 105 TSF1 Embankment Section, Tierra Group Aug 2025

15.5.3 TSF2

TSF2 will be constructed to the east of TSF1 and will cover approximately 203 hectares. The location and extent of TSF2 were designed to limit disturbance to existing waterways. TSF2 will be constructed using ROM NAF waste rock and a combination of downstream and upstream raises. A starter dyke (Stage 1) will be constructed followed by a downstream raise (Stage 2). Upstream raises will be used for Stages 3 to 5. An underdrainage system will be installed during Stage 1 construction to intercept groundwater. The basin of the impoundment and the upstream slope of the starter dyke (Stage 1) will be fully lined with a geocomposite liner system consisting of a geosynthetic clay liner (GCL) beneath a 60-mil linear low-density polyethylene geomembrane. This liner system will reduce the risk of seepage loss through the foundation and embankment. An overdrainage system on top of the geomembrane liner will be installed across the full impoundment basin and along the upstream toe to improve tailings consolidation. Figure 106 depicts the typical cross section of the TSF2 embankment.

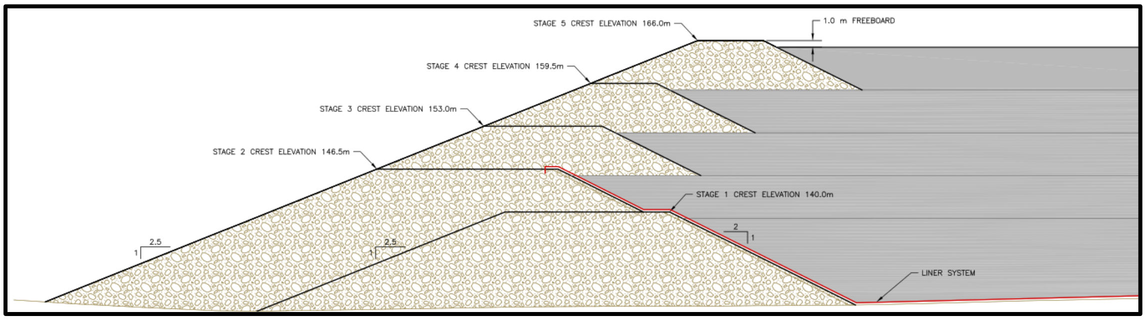


Figure 106 TSF2 Embankment Section, Tierra Group Aug 2025

TSF2 Stage 2 will be constructed as a downstream embankment. The liner system will be extended to the Stage 2 crest, forming a continuous liner with the Stage 1 liner. Stages 3 to 5 will be built using the upstream construction method. Table 115 summarizes the TSF2 construction sequence, method for each raise, and capacity.

Stage	Construction Method	Design Life (years)	Capacity (Mt)	
			Incremental	Cumulative
1	Downstream	5.18	25.35	25.35
2	Downstream	2.81	13.78	39.13
3	Upstream	2.95	14.44	53.57
4	Upstream	2.99	14.44	68.01
5	Upstream	0.06 [1]	13.87	81.88

Note: *[1] Design life is based on the expected FS mine plan. Additional storage capacity is available for* contingency.

Table 115 TSF2 Construction Sequence

Tailings will be deposited into TSF1 until it reaches full capacity in Year 19. Tailings will then be deposited in TSF2. This approach allows for the delay of TSF2 construction until sufficient mine waste is available, simplifying tailings management for both facilities and deferring construction costs until later in the mine life. Additional upstream raises can be added to TSF2 if the required tailings storage capacity exceeds current projections. TSF 2 is expected to operate for a total of 14 years.

15.6 Site Establishment and Early Works

Site establishment will occur prior to the operation of the construction camp with the hire or purchase of Engineering, Procurement and Construction (EPC) contractor and client offices/crib rooms and ablutions for the duration of the Project. The facilities will be located at the same area as the permanent camp.

Early works will include accommodation in the town of Katherine for 20 people for three months to complete the early works at the construction camp facilities and process plant area.

15.6.1 Construction Camp

A construction camp will be built at the beginning of the Project to accommodate a minimum of 200 personnel, while the permanent camp is under construction. All construction personnel will be housed in Katherine and then transferred to the camp as additional rooms become available.

Bulk earthworks and all services including power, communications, water, and sewerage will be completed prior to the arrival of the hired buildings

15.7 Power Generation

15.7.1 Power Station Design

The power station has been designed on the basis of the following loads in Table 116.

	Design Load (kW)	Hours/Year	kWhrs/year
Average	42,000	6,935	291,270,000
Peak	50,000	1,095	54,750,000
Stop	5,000	730	3,650,000
Total Export kWhrs		**8,760**	**349,670,000**

Table 116 Project Power Station Design Basis

The power station has been designed on an "N+2" basis without renewable energy sources.

- 19 x Jenbacher J620s installed (3.3 MW each), total of 64 MW:
  - 17x Jenbacher J620s required to meet peak load (56.1 MW).
  - 2x Jenbacher J620s required as back up.
- 3 x Cummins KTA50s required as black starts/back up (1 MW each) 3 x KTAs equate to 1x gas engine.

The basis of design is as follows:

- 2 x earthing transformer.
- 2 x auxiliary transformer.
- 2 x 11 kV HV/LV prefabricated switchroom (with office).
- 2 x 11 kV HV board with the following allowed for:
  - 19 x generator incomers,
  - 1 x spare incomers (spare),
  - 3 x diesel genset incomer (3 MW),
  - 1 x 11 kV client feeders,
  - 2 x auxiliary transformer feeder,
  - 2 x earthing transformer feeder.

15.7.2 Power Station Operations

The power control system will be designed to allow remote and local operator-initiated control, or fully automatic selection of the generation system depending on demand and environmental conditions. The power supply will operate and "load follow" site load demand automatically. Both power contractor operations in Perth and on-site operators will be able to control and monitor the power supply and select generation modes and sequence from a control center established in the power contractor facility.

Controlled system access will allow optimization of all generation assets to ensure efficient and reliable power delivery.

Generation control will be performed by a generation/load management system, providing local control and monitoring of individual generation equipment, with interface to common control load sharing systems.

Generation requirements to meet site demand will be automatically controlled by a centralized master control system. Control algorithms based on individual generation load capacity will support efficient operations and optimized utilization for normal day-to-day operations. This system would typically use a scheduling process of allocating power resources ahead of time to meet power reliability requirements. The power contractor's control system is flexible to allow for ongoing optimization and robust enough to provide reliable power with little or no constant operator input. Constant monitoring of load profiles will support periodic performance reviews against efficiency targets.

The thermal station will require 18 months for construction following signing of the power purchase agreement.

15.8 Overall Project Site Layout

Figure 107 below presents the overall site infrastructure layout, including roads, tailings storage facilities, water dams, power station, accommodation village, and process plant

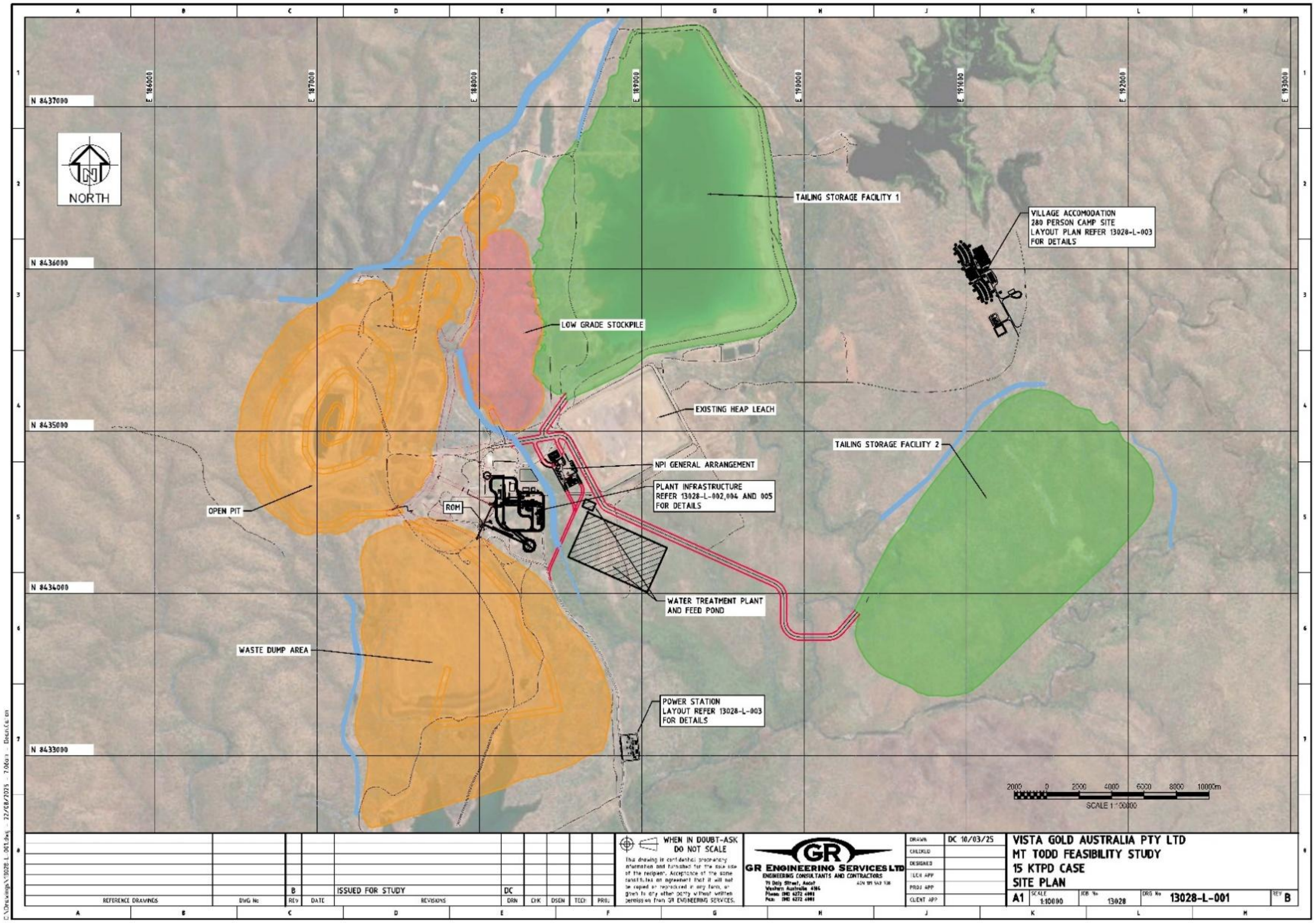


Figure 107 Overall Project Site Layout

16. MARKET STUDIES AND CONTRACTS

16.1 Introduction

Gold bullion is generally traded electronically through a mature and highly liquid market. It is bought and sold daily by traders, banks, and other financial institutions through various international marketplaces. The price of gold is driven by worldwide supply and demand.

Forecasts for gold prices used in the economic analysis are derived from a combination of sources, including consensus analyst forecasts and recent historical prices.

The Project is assumed to produce gold doré, which will be refined on behalf of Vista at international third-party refineries whose outturn of bullion meets the “Good Delivery” standard of the London Bullion Market Association. For purposes of the economic analysis, Vista’s saleable product was assumed to be “Good Delivery” bullion. The Project is, therefore, not exposed to material market risk in terms of its ability to place products or receive competitive terms.

No marketing studies have been conducted by Vista or its consultants for gold bullion.

16.2 Market Outlook

The price of gold is the primary factor in determining the Project’s profitability and cash flow from operations. The gold price of USD2,500 per gold ounce used in the economic analysis was derived from a combination of sources, including consensus forecasts reflecting a composite of financial institutions, gold prices used in various recent technical reports completed by mining companies, developers and consulting groups, and recent historical price trends.

As of the effective date of this Technical Report Summary, the spot gold price was substantially higher than recent consensus forecasts or historical trends. This variance reflects Vista’s intent to use a constant gold price that is meant to reflect long-term expectations over the life of the Project.

16.3 Contracts

Vista has no refining or bullion sales contracts in place. Commitments to deliver gold bullion or the equivalent value are presently limited to private royalty agreements in place as of the effective date of this Technical Report Summary. For purposes of the economic analysis, the value of gold associated with these royalties is included in gold sales with an offsetting royalty expense. Vista expects that terms contained within any refining, sales, or other contracts for delivery of gold bullion will be typical of, and consistent with, standard industry practices.

At the time of issuing this Technical Report Summary, Vista does not have any established contract with any third-party services supplier or contractor for the development of the Mt Todd Project.

16.4 Available Refining Options and Costs

Refining costs are typically a minor portion of total cash costs; therefore, comparative values based on actual results from producers is limited. Rates used in other technical reports are a reasonable source of information, but it must be acknowledged that most non-producers do not have refining contracts in place at the time of their technical studies.

Consideration was given to the refining costs assumptions used by other companies that published technical reports during 2025 (as filed on sedarplus.ca). A refining charge of USD5.00/oz is the most applied assumption. As such, the refining cost applied for this Technical Report Summary is USD5.00/oz payable gold.

16.5 Shipping

Transportation of gold doré from the Project to refineries will be contracted by Vista through an internationally recognized security logistics company. Outturn of gold bullion by refineries will be credited to Vista's account or as directed by Vista.

16.6 Other 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:
  - EPC and EPCM contracts,
  - Mining Contractor services,
  - Diesel and fuel oil,
  - Explosives,
  - Natural gas for the power plant,
  - Process reagents,
  - Site security services, and
  - Camp management, catering and support services.

16.7 Qualified Person Opinion

In the opinion of the QP, sales contracts that could be negotiated are expected to be within industry norms. The commodity pricing and exchange rate assumptions are suitable for use in the cash flow analysis in Section 19.

17. ENVIRONMENTAL STUDIES, PERMITTING AND PLANS, NEGOTIATIONS, OR AGREEMENTS WITH LOCAL INDIVIDUALS OR GROUPS

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 May 2024 Vista submitted a “Referral” to the NTEPA citing the changes to the Project using the EIS completed in 2014. In November 2023, this assessment was suspended at Vista request.

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 MMP was approved by the Northern Territory Government Department of Industry, Tourism and Trade (DITT). This was the last approval required before the “Mining Authorisation” can be issued by the Minister of Environment and Energy, and works can occur. The “Mining Authorisation” (0331-04) was issued August 2021. After July 1, 2024, MMPs granted under the Mining Management Act (2001) were automatically deemed to convert to a Deemed Environmental (Mining) License under the Environment Protection Act (2019), which maintains the prior approval of the MMP and the “Mining Authorisation”. The Deemed Environmental (Mining) License must be converted to an Environmental (Mining) License within four years.

The QPs for Section 17 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 26 of this Technical Report Summary, 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 Technical Report Summary.

17.1 Environmental Studies

Several environmental studies have been conducted at the Project in support of development of the EIS and EPCB Approval 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.

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 WRD, 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 217 ha with an estimated height of 160 m. 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.

17.1.1 Flora and Vegetation

Eight vegetation types covering 5,462.56 ha 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.

17.1.2 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.

17.1.3 Migratory and/or Marine Species

Fourteen EPBC Act listed migratory bird species potentially occur within 10 km of the Project area. Ten have been recorded from the licenses. Seven EPBC listed marine species potentially occur with 10 km 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.

17.1.4 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 leases and 1,337 km^2 of exploration licenses contiguous with the 55.4 km^2 of mining leases. An application for an AAPA Certificate authorizing work in additional areas associated with the 2024 FS and this Technical Report Summary has been initiated Waste and Tailings Disposal, Site Monitoring and Water Management

17.1.5 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 PAF or 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.

17.1.6 Tailings Disposal

Tailings will be disposed of in two tailings storage facilities, TSF 1 and TSF 2. TSF 1 (an existing TSF) 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) 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 15.5.

Reclamation and closure of the TSFs is described in Section 17.4.

17.1.7 Site Monitoring

Currently, surface water monitoring is conducted at various locations at the site. A comprehensive site monitoring plan has been incorporated into the environmental (mining) license.

17.1.8 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 RWD) 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.

17.1.8.1 In-situ Pit Treatment

In-situ treatment of the Batman Pit (RP3) was conducted by the 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 in accordance with the approved Waste Discharge License (WDL). As of the date of this Technical Report Summary, the pit contains approximately 3.0 GL of water. 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 as the treated water is being discharged.

17.1.8.2 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.

17.1.8.3 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 21—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 Project 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 2021 Preliminary Feasibility Study (Tetra Tech, 2021) 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 30 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.

17.1.8.4 *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 21.4—Surface Water Hydrology.

17.2 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 Project 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 117 includes a list of approvals, permits, and licenses required for the Project and their current 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. This assessment was suspended in November 2023 at Vista request.	Approved Sep. 2014	NA
Deemed Environmental (Mining) License Approval from NT Department of Primary Industry and Resources	Initial approval was granted in April 2021 based on a 50ktpd operation under the since-repealed Mining Management Act (2001). Approved mining operations were automatically granted a Deemed Environmental (Mining) License under the Environment Protection Legislation Amendment Act (2023). The Deemed Environmental (Mining) License is valid until June 30, 2028. Prior to the expiration, Vista must apply for a replacement Environmental (Mining) License.	July 1, 2024	Jun. 30, 2028
Heritage Act permit to destroy or damage archaeological sites and scatters/Aboriginal Areas Protection Authority Clearances	Authority Certificate Number 2011/15538 issued. This certificate defined restricted works areas and granted select clearances to allow for initial investigations. Additional clearances will be required for further investigations as well as prior to disturbance associated with mine development and exploration activities.	Aboriginal Areas Protection Authority dated Jun. 31 2012	NA

Approval/Permit/License	Current Status	Approval/Permit License Date	Expiration Date
Aboriginal Areas Protection Authority Certificate	Authority Certificates C2012/137 and C2021/028 issued. 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 leases and 1,337 km^2 of exploration licenses contiguous with the 55.4 km^2 of mining leases.	Jun. 7, 2021	NA
Aboriginal Areas Protection Authority Certificate	Vista applied for an Authority Certificate on June 19, 2024. The new application addresses the expanded footprints of the Batman Pit and waste rock dump plus a 200-meter buffer zone as defined in this Technical Report Summary. Consultation with aboriginal authorities has been initiated.	Pending	NA
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. Vista requested a 10-year extension of the 'substantial commencement' condition on March 23, 2023.	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. The smaller throughput operation described in this Technical Report Summary would use less water than the current license authorizes, which may require an amendment to substantiate the difference.	Jun. 01 2021	Jun. 01 2031

Approval/Permit/License	Current Status	Approval/Permit License Date	Expiration Date
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 Vista request in 2021 as not required until operational
Wastewater treatment system permits under Public Health Act 1987 and Regulations	Required for the wastewater 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.	Jan. 19, 2018	Dec. 31 2040

Table 117 Project Permit Status

In addition, permits that are required to commence construction works will be obtained prior to any construction activity.

17.3 Social or Community Requirements

The JAAC 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 areas.

Those parts of the JAAC agreement that are within the public domain are presented in this Technical Report Summary; the remaining part of the agreement, which is confidential, is not presented in this Technical Report Summary.

There are currently no legal or contractual obligations requiring engagement with local workforce and contractors within Katherine or the broader Northern Territory. Vista maintains a commitment to supporting regional businesses and expects to formalize a local industry engagement plan as the project progresses, as project development advances, local workforce participation will be incrementally increased through training initiatives and attracting previously trained local residents who may have left the area for employment elsewhere in the mining sector. For this FS, capital and operating cost estimates have been prepared on the basis that approximately 90% of the initial workforce will be sourced through FIFO arrangements. This approach reflects current market conditions and the Projects requirements. Vista expects the proportion of local employees to rise over the life of the project, providing both operational and socio-economic benefits to the region.

17.4 Reclamation and Closure

A reclamation and closure plan for the Project was developed in support of the Technical Report Summary for renewed mining operations. This reclamation and closure plan evaluates the reclamation activities that will be conducted for the landforms planned as part of mining commencement. Reclamation and closure plans and strategies for each major facility at the Project are briefly summarized in Table 118.

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 steeper than 3H:1V	X	X						
Final Overall Slopes at or gentler than 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 (TSF1 only)			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 liner system as part of cover		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

"X" denotes where the task or characteristic is applicable to the landform

Table 118 Reclamation and Closure Approach

Costs associated with reclamation and closure are provided in Section 18.4.5—Reclamation and Closure. In accordance with regulatory requirements, a 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 and closure bonding have been included in the technical economic model, excluding initial environmental bond.

17.4.1 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 **21.6**—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.

17.4.2 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 13, the WRD will be constructed with an encapsulating NAF material outer shell on each lift. Concurrent installation of a low permeability liner (i.e., bituminous geomembrane [BGM]) following attainment of final grades will serve to reduce infiltration of precipitation into the WRD core. This liner system will include placement of sorter reject material placed above and below the BGM 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 to slope 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 coarse 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 pit or WTP for treatment prior to release until it is feasible to treat this and other ARD/ML on site using passive treatment systems.

17.4.3 Tailings Disposal Facility

The TSF1 embankment and TSF1 and TSF2 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). The TSF2 embankment will be constructed of NAF material and will not require a store and release cover. Cover will include a 0.8 m NAF layer overlain by a 0.2-m thick layer of PGM. Following PGM placement, cover surfaces will be roughened and revegetated with native species. The store and release covers will serve to effectively reduce percolation of precipitation below this cover.

The majority of the impounded surface of the TSFs 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. ARD/ML collected at TSF seepage collection systems will initially be pumped to the pit or WTP for treatment prior to release until it is feasible to treat this and other ARD/ML on site using passive treatment systems.

17.4.4 Processing Plant and Pad Area

A new process plant will be built for the Project. 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, TSFs, 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 PGM and revegetated. Storm water drainage, erosion, and sediment controls will be constructed to minimize erosion.

The water treatment plan and associated pond will be left in place if needed 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.

17.4.5 Heap Leach Pad

The HLP, will be reprocessed following processing of ore and low-grade ore. Reprocessing of the HLP will be completed as part of self-funded reclamation and closure activities. Following reprocessing of the HLP material, the HLP and pond footprint will be reclaimed by cutting and removing the liner for consolidation in TSF 2. It is anticipated that the integrity of the HLP liner will have been compromised and removal of 0.5-m thick of impacted soils below the liner will be necessary. These materials would be removed and consolidated in TSF 2. The area will then be regraded to prevent ponding of water and will be covered with a 0.2 m thick layer of PGM and revegetated.

17.4.6 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 (if present), 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 collection ponds routed to the WTP.

17.4.7 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.

17.4.8 Water Storage Ponds

Prior to construction of the active WTP, a pond 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, stormwater 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 are 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 pit or 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.

17.5 Low Permeability Material

External sources of low permeability material are anticipated to be available for borrow or purchase for use in reclamation and closure activities, including store and release covers.

17.6 NAF Material for Closure

NAF waste rock material produced in the operational years will be utilized for encapsulating PAF waste rock material in the waste rock dump as part of the reclamation and closure process. NAF waste rock material will also be used for the reclamation and closure of the TSFs and other infrastructure. The costs associated with transporting and handling NAF waste rock material for reclamation and closure purposes over the life of the Project were estimated by Mining Plus, based on the assumptions and methodologies applied in the mining cost estimate.

17.7 Reclamation and Closure Cost Estimate

Costs for reclaiming and closing 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.

The closure plan includes re-processing 13 Mt of heap leach pad material from previous operations at the end of the life of mine and then placing that processed material in the TSF, operating costs associated with re-processing and tailings deposition have been treated as self-funded reclamation.

18. CAPITAL AND OPERATING COSTS

18.1 Overall Capital Cost

18.1.1 Scope of Capital and Operating Cost Estimates

Capital and operating cost estimates have been developed for a mining operation capable of treating 15 ktpd of run-of-mine (ROM) ore, comprising the following:

- A conventional open pit mine using a mine contractor, with all mining infrastructure supplied by the contractor.
- A gold processing plant utilizing a flow sheet comprising primary and secondary crushing, coarse ore stockpile, HPGR, screening, ore sorting, ball milling, tertiary grinding (VertiMill), leach tanks, carbon in leach, acid wash and elution, gold electrowinning and smelting to produce doré, tailings cyanide detoxification, tailings dewatering and discharge of tailings into a tailings storage facility.
- Tailings storage facilities and waste dump, complete with water management and drainage infrastructure.
- Access roads.
- Support infrastructure and utilities including process/admin/workshop/stores/laboratory buildings, a gas-fired power plant (supplied by a third party), accommodation, potable water supplies, a water treatment plant, raw water supply and distribution and all solid and liquid waste facilities.

All capital and operating costs detailed in the estimates below have been developed by various parties contracted by Vista and are classified as **Class 3 estimates**, consistent with the Association for the Advancement of Cost Engineering (AACE) guidelines for feasibility-level studies. Class 3 AACE estimates in a FS provide a detailed cost forecast based on significant project definition and engineering design. With an accuracy range of ±10 -15%. For detailed accuracy ranges and methodology, refer to Table 130. All cost estimates are presented in United States Dollars (USD) unless otherwise stated. There are instances where Australian dollar (AUD) has been used, as the Project is in Australia, and the exchange rates used are clearly defined – initial capital is USD:AUD at 0.66 and the life-of-mine operating exchange rate is USD:AUD at 0.67.

The capital costs estimates within this Technical Report Summary include a contingency range not exceeding 10%.

Capital and operating cost estimates based on the Project scope described in this Technical Report Summary have been peer reviewed for acceptance by the Project team.

18.1.2 Exclusions

The capital cost estimate only includes capital costs incurred from the commencement of process plant design and construction (project sanction). The following costs are excluded in the capital estimate:

- All future exploration and development drilling expenditures.
- Vista "Owner's costs" for project development and ramp-up are included, however, current site activity upkeep is excluded.
- Cost of Finance (COF), capitalized interest and standby fees from third-party lenders.
- Initial reclamation bond, to be addressed through a letter of credit.

18.1.3 Project Summary Capital Cost

The total estimated capital cost of bringing the Project into production is USD425M including risk contingency, which has been added in the financial analysis. Sustaining capital of USD256M is required over the life of the Project and accounts for infrastructure updates and replacements and tailings dam raises for TSF1 and the construction and sustaining of TSF2. The Technical Report Summary contemplates concurrent reclamation of the waste rock dump and tailings storage facilities during the life of the Project for USD121M. The closure plan includes re-processing 13 Mt of heap leach pad material from previous operations and then placing that material in the tailings storage facility and 10 years of closure and monitoring. The Project's total capital is summarized in Table 119 including the sustaining capital, reclamation and closure costs and the benefit gained from processing the heap leach pad material at the end of the mine life.

Capital Costs Summary	Initial Capital (USDM)	Sustaining Capital Years 1-30 (USDM)	Heap Leach Pad, Reclamation and Closure (USDM)[4]
Initial Capital	$ 399.48	N/A	N/A
Sustaining Capital	N/A	$231.63	$ 9.48
Heap Leach ad, Reclamation and Closure Costs	N/A	$ 109.03	$ 50.66
Combined Engineering Growth and Contingency (6-10%)	$ 25.03	$ 36.40	$ 5.76
Total Capital	**$424.51**	**$ 377.06**	**$ 65.90**

Table 119 Life of Project Capital Cost Summary

[4] Excludes cash flows from the reprocessing of HLP ore, includes sustain costs incurred during reprocessing of the HLP ore

The mining is based on a contractor supplied model. The Process Plant Capital estimate (AUD441M) or (USD292), provided in Section 18.3, has been based upon an EPC delivery process and includes contractors risk margin.

The overall estimated capital costs for the Project are summarized below in Table 120. This cost excludes sunk costs incurred to establish the exploration camp, the exploration camp access road and costs of studies conducted prior to and leading up to full Project approval.

Capital Expenditure Item	Initial Capital Cost (USDM)	Sustaining Costs (Years 1-30) Cost (USDM)	Heap Leach Pad, Reclamation and Closure Costs (USDM)[5]
Mining	$22.03	$28.01	$4.71
Process Plant	$144.80	$46.03	N/A
Project Infrastructure	$83.68	$141.23	$4.41
Site Establishment and Facilities	$36.57	$8.12	N/A
Management, Engineering, EPC Services	$ 65.22	$8.24	$0.36
Preproduction Costs and Capital Spares	$47.18	N/A	N/A
Reclamation	N/A	$109.57	N/A
Sub-total: Capital Expenditures	$399.48	$341.20	$9.48
Heap Leach Pad, Reclamation and Closure	N/A	NA	$50.66
Combined Engineering Growth and Contingency (6-10%)	$25.03	$35.86	$5.76
Total Capital Costs	**$424.51**	**$377.06**	**$65.90**

Table 120 Project Capital Cost Summary

18.1.4 Contingency and Engineering Growth

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 engineering growth refers to the increase in estimated capital costs as a project’s engineering design progresses and becomes more detailed included based on an assessment of the maturity of the design and is noted as a percentage against each line item in the estimate. Engineering growth helps project owners and decision-makers anticipate the inevitable increase from early “order of magnitude” estimates to more

[5] Excludes cash flows from the reprocessing of HLP ore, includes sustain costs incurred during reprocessing of the HLP ore.

definitive or detailed capital cost estimates. It is a critical part of risk management and budgeting in capital project planning.

Each consulting group has prepared capital costs including contingencies and engineering growth where required with responsibility as described in Section 2.6.

The capital costs estimation for this Technical Report Summary include a contingency range not exceeding 10%.

18.2 Mining Capital Cost

18.2.1 Scope of Operating and Capital Cost Estimates

This section outlines the scope, basis of preparation, key inputs, assumptions and exclusions applied to the mining cost estimates.

The mining study team collaborated with a leading Australian Tier 1 mining contractor to obtain the price estimates outlined in this document. This estimate was benchmarked against recent industry database cost estimates to ensure its reasonableness for use in this Technical Report Summary.

The mining contractor rates (costs) include primary mining operations covering the following mining activities:

- Loading and haulage from pit.
- Production drilling, charging and blasting, including provision of all explosives.
- Support activities such as dump maintenance, road construction, road maintenance (including dust suppression) and pit dewatering.
- Rehandle and stockpile activities for longer term pre-crusher stockpiles and the ROM pad crusher feed.
- Operations and technical support.
- All personnel requirements for the mining operation including operators, maintenance, support and supervision personnel.

Initially three tier 1 contractors were involved in a scoping and evaluation process to assess their suitability and capability to provide mining contractor pricing for the Project. This process identified one preferred mining contractor who was then engaged to provide detailed costing for the operation of the proposed Project open pit mining operations as outlined in this Technical Report Summary. This evaluation process considered the following criteria:

- Overall cost estimate/pricing.
- Overall timing and availability.

- Any previous Mt Todd Project experience (by company &/or team members).
- Recent Northern Territory region experience.
- General applicable recent project experience.
- Overall mining contractor capability.
- Willingness to engage with Technical Report Summary team and share expertise and information.
- Capability of proposed load and haul options.
- Capability of drilling and blasting options.

All costs were developed in AUD as of June 30, 2025 and converted to USD at the exchange rate nominated.

The estimate provides the basis for economic evaluation of the Project, and no escalation is included.

The scope of work priced by the mining contractor included a detailed mine design and scheduling plan completed during this Technical Report Summary. It reflects typical contract mining operations, as often undertaken by a Tier 1 mining contractor, and includes:

- Drilling and blasting of all material as required.
- Load and haul of all material to ROM pads, waste dumps and stockpiles as directed.
- Rehandle of low-grade material from stockpile to ROM.
- Maintenance of all equipment used in the Project.
- Provision of all consumables and operational costs.
- Provision of management, supervision, maintenance and operational labor required for the Project.
- Provision of support facilities (see Section 18.2.9 Mine Infrastructure).

18.2.2 Estimate Responsibilities and Accuracy

For clarity, a responsibility matrix was developed, see Table 121, by the FS mining Technical Report Summary team to outline the allocation of responsibilities assumed in the mining contractor pricing submissions.

18.2.3 Schedule of responsibilities

Legend: P = Principal/Vista C = Contractor

No.	Description	Design	Supply	Installation Cost	Operation Parts	Operation Labor	Maintenance Labor	Removal/ Ownership	Remarks
1	**General**								
1.1	Overall site layout	P	P	P	P	P	P	P	
1.2	Contractor's site layout	BOTH	C	C	C	C	C	P	

No.	Description	Design	Supply	Installation Cost	Operation Parts	Operation Labor	Maintenance Labor	Removal/ Ownership	Remarks
1.3	Geotechnical engineering	P	P	P	P	P	P	P	
1.4	Hydrogeological engineering	P	P	P	P	P	P	P	
1.5	Geology	P	P	P	P	P	P	P	
2	**Measurement**								
2.1	Survey equipment and materials	BOTH	C	C	C	C	C	C	
2.2	Baseline surveying	C	C	C	C	C	C	C	
2.3	Monthly void survey for payment	P	P	P	P	P	P	P	
2.4	Weekly stockpile survey	P	P	P	P	P	P	P	
2.5	Check survey	C	C	C	C	C	C	C	
2.6	Load cells for haul trucks	C	C	C	C	C	C	C	
3	**Communication**								
3.1	Radios	BOTH	C	C	C	C	C	C	P will provide a predetermined frequency
3.2	Telephone system (mobile)	C	C	C	C	C	C	C	Via local mobile provider tower accessible at site
3.3	Telephone system (landline)	N/A	N/A	N/A	N/A	N/A	N/A	N/A	
3.4	Internet access	C	C	C	C	C	C	C	C
3.5	In pit wifi mesh	P	P	P	P	P	P	P	
4	**Planning and Scheduling**								
4.1	LOM planning	P	P	P	P	P	P	P	
4.2	Annual planning	P	P	P	P	P	P	P	
4.3	Three month rolling planning	P	P	P	P	P	P	P	
4.4	One month planning	P	P	P	P	P	P	P	Contractor has sight and approval of planning
4.5	Bi-weekly plan	C	C	C	C	C	C	C	
4.6	Mining scheduling	C	C	C	C	C	C	C	Principal has sight and approval of scheduling
4.7	Drill & blast block design	P	P	P	P	P	P	P	
4.8	Drill & blast pattern design	C	C	C	C	C	C	C	
4.9	Grade control specifications & pattern design	P	P	P	P	P	P	P	
5	**Mobilization**								
5.1	Plant and equipment lists	BOTH	C	C	C	C	C	C	
5.2	Manning complements	BOTH	C	C	C	C	C	C	
5.3	Medical clearance (time and cost)	BOTH	C	C	C	C	C	C	
5.4	Employees induction (time and cost)	P	P	C	C	C	C	C	

No.	Description	Design	Supply	Installation Cost	Operation Parts	Operation Labor	Maintenance Labor	Removal/ Ownership	Remarks
5.5	Contractors' induction (time and cost)	C	C	C	C	C	C	C	
5.6	Mobilization schedule	BOTH	C	C	C	C	C	C	
6	**Site Preparation – MSA**								
6.1	Surface workshop	C	C	C	C	C	C	C	
6.2	Wash bay	C	C	C	C	C	C	C	
6.3	Laydown area – fencing	C	C	C	C	C	C	C	
6.4	Security and safety precautions	C	C	C	C	C	C	C	
6.5	Bulk power	P	P	P	P	P	P	P	
6.6	Bulk service water	P	P	P	P	P	P	P	
6.7	Bulk potable water	P	P	P	P	P	P	P	
7	**Manpower**								
7.1	Production management	C	C	C	C	C	C	C	
7.2	Engineering management	C	C	C	C	C	C	C	
7.3	Contract management	C	C	C	C	C	C	C	
7.4	General manager (legal appointment)	P	P	P	P	P	P	P	
7.5	Mine manager (legal appointment)	P	P	P	P	P	P	P	
7.6	Site management and supervision	C	C	C	C	C	C	C	Changes to management to be agreed in writing between C and P.
7.7	Labor	C	C	C	C	C	C	C	Written acknowledgement when labor numbers change
7.8	Recruitment and training	BOTH	C	C	C	C	C	C	Local employment policies to be followed
7.9	Supervisory staff accommodation (mine camp)	P	P	P	P	P	P	P	
7.10	Single quarters accommodation and feeding	C	C	C	C	C	C	C	
7.11	Ambulance	P	P	P	P	P	P	P	
7.12	First aid response by medic	P	P	P	P	P	P	P	
7.13	First aid clinic and equipment	P	P	P	P	P	P	P	
7.14	Hospitalization	C	C	C	C	C	C	C	
7.15	Safety [incl. all PPE and management]	C	C	C	C	C	C	C	
8	**Building and Furnishings**								

No.	Description	Design	Supply	Installation Cost	Operation Parts	Operation Labor	Maintenance Labor	Removal/ Ownership	Remarks
8.1	Contractor's offices, stores	C	C	C	C	C	C	C	
8.2	Equipment for 8.1 above	C	C	C	C	C	C	C	
8.3	Changehouse	C	C	C	C	C	C	C	
8.4	Equipment for 8.3	C	C	C	C	C	C	C	
9	**Facilities - Surface**								
9.1	Transportation to and from site	C	C	C	C	C	C	C	
9.2	Contractor's site transport	C	C	C	C	C	C	C	
9.3	Supervision personnel transport	C	C	C	C	C	C	C	
9.4	General personnel transport	C	C	C	C	C	C	C	
9.5	General site security	P	P	P	P	P	P	P	
9.6	Access control – perimeter	P	P	P	P	P	P	P	
9.7	Access control – pit	BOTH	C	C	C	C	C	C	
9.8	T & A clocking system	C	C	C	C	C	C	C	P requires access to all records of hours worked, annual medical validity, license validity, training recorded of all personnel on site
9.9	Housekeeping	C	C	C	C	C	C	C	General housekeeping in all areas within which the contractor operates.
10	**Services & Utilities – MSA**								
10.1	Service and potable water from point of supply	Both	C	C	C	C	C	C	
10.2	Water reticulation	Both	C	C	C	C	C	C	
10.4	Electric power point to supply	P	P	P	P	P	P	P	
10.5	Electric power from point of supply	Both	C	C	C	C	C	C	
10.6	Power reticulation – permanent	P	P	P	P	P	P	p	
10.7	Lighting – permanent	P	P	P	C	C	C	P	
10.8	Firefighting equipment – offices	BOTH	C	C	C	C	C	C	
10.9	Firefighting equipment – workshop	BOTH	C	C	C	C	C	C	
10.10	Settling ponds	P	P	P	P	P	P	P	
10.11	Pollution control dams	P	P	P	P	P	P	P	
10.12	Sewerage disposal	P	P	P	P	P	P	P	
10.13	Effluent disposal	P	C	C	C	C	C	C	
10.14	Solid waste disposal	P	P	P	P	P	P	P	

No.	Description	Design	Supply	Installation Cost	Operation Parts	Operation Labor	Maintenance Labor	Removal/ Ownership	Remarks
10.15	Hazardous waste disposal	P	C	C	C	C	C	C	Records of disposal are to be kept
10.16	Salvage	P	C	C	C	C	C	C	
10.17	Tyre disposal	P	C	C	C	C	C	C	
11	**Services & Utilities - Pit, Waste Dumps, Production Area**								
11.1	Sump and bench dewatering	C	C	C	C	C	C	C	Sufficient dewatering equipment to meet minimum annual dewatering average during wet season. The C is to maintain sufficient dewatering activities, such that water does not interfere with drill and blast operations.
11.2	Groundwater management	C	C	C	C	C	C	C	Including high wall berms, drains
11.3	Stormwater management	C	C	C	C	C	C	C	Including cut-off trenches along all dumps, stockpiles and around pit
11.4	Lighting – temporary	C	C	C	C	C	C	C	
11.5	Lighting – permanent	BOTH	C	C	C	C	C	C	If deemed necessary
11.6	Safety	C	C	C	C	C	C	C	
11.7	Sanitation	C	C	C	C	C	C	C	
11.8	Eating facility	C	C	C	C	C	C	C	
11.9	Stemming material	C	C	C	C	C	C	C	
11.10	Road capping material	C	C	C	C	C	C	C	
12	**Explosives & Blasting**								
12.1	Explosives transport - source to designated storage and distribution point	P	P	P	P	P	P	P	
12.2	Explosives transport – from designation storage and distribution point to pit	C	C	C	C	C	C	C	
12.3	Explosives accessories transport - Source to designated storage and distribution point	P	P	P	P	P	P	P	

No.	Description	Design	Supply	Installation Cost	Operation Parts	Operation Labor	Maintenance Labor	Removal/ Ownership	Remarks
12.4	Explosives accessories transport – from designated storage and distribution point to pit	C	C	C	C	C	C	C	
12.5	Explosives storage	P	P	P	P	P	P	P	
12.6	Explosives and accessories	C	C	C	C	C	C	C	
12.7	Blasting cables	C	C	C	C	C	C	C	
12.8	Blasting wire	C	C	C	C	C	C	C	
12.9	Charging units	C	C	C	C	C	C	C	
12.10	Magazine master	C	C	C	C	C	C	C	
12.11	Assistant magazine master	C	C	C	C	C	C	C	
12.12	Permanent blast monitoring	C	C	C	C	C	C	C	
13	**Plant, Equipment & Supplies**								
13.1	Drilling								
13.1.1	Blast drill equipment	C	C	C	C	C	C	C	
13.1.2	Drill steel and accessories	C	C	C	C	C	C	C	
13.1.3	Emulsion truck & trailer (backup)	C	C	C	C	C	C	C	
13.1.4	Grade control drill equipment	P	P	P	P	P	P	P	
13.1.5	Grade control drill steel, accessories and consumables	P	P	P	P	P	P	P	
13.2	Load and haul								
13.2.1	All plant	C	C	C	C	C	C	C	
13.2.2	Transport of materials, fuel and equipment into pit and between sites	C	C	C	C	C	C	C	
13.2.3	Water bowser	C	C	C	C	C	C	C	
13.2.4	Salvage	C	C	C	C	C	C	C	
13.2.5	Fleet management system	Both	C	C	C	C	C	P	
13.2.6	Collision avoidance system	N/A	N/A	N/A	N/A	N/A	N/A	N/A	
13.2.7	Firefighting equipment - plant	BOTH	C	C	C	C	C	C	
14	**Roads and Ramps**								
14.1	Road and ramp establishment	BOTH	C	C	C	C	C	C	
14.2	Brake test ramp	BOTH	C	C	C	C	C	C	
14.3	Road maintenance	C	C	C	C	C	C	C	Roads associated with C activity
14.4	Dust suppression	C	C	C	C	C	C	C	
14.5	Traffic management and signs	BOTH	C	C	C	C	C	C	
14.6	Berms	BOTH	C	C	C	C	C	C	

No.	Description	Design	Supply	Installation Cost	Operation Parts	Operation Labor	Maintenance Labor	Removal/ Ownership	Remarks
15	**Hydrocarbon Management**								
15.1	Diesel (mining)	BOTH	C	C	C	C	C	C	Fuel dispatch system
15.2	Oil and lubricants	C	C	C	C	C	C	C	Including disposal
15.3	Spill kits	C	C	C	C	C	C	C	

Table 121 Mining Cost Responsibility Matrix

The mining contractor submission assumes items and services supplied by Vista, such as diesel and water, will be provided free of charge and in a timely manner to suit operational requirements.

Additional costs, such as, mine technical services, other members of the operations owner's team, grade control assays, geotechnical monitoring, mine operations fuel and owner's team light vehicles (OPEX and CAPEX) have been included in the overall mining costs for the FS, in addition to the mining contractor cost estimate.

The overall FS cost estimates are classified as **Class 3 estimates**, consistent with the Association for the Advancement of Cost Engineering (AACE) guidelines for feasibility-level studies. For detailed accuracy ranges and methodology, refer to Table 130.

18.2.4 Exclusions

The following items are excluded from the mining contractor scope of cost estimates and are covered elsewhere in the FS:

- Mineral processing plant and associated infrastructure.
- TSF.
- Power generation and grid interconnection.
- Permanent accommodation village (supplied by the principal).
- Flights, ground transport and offsite logistics (supplied by the principal).
- Diesel (supplied by the principal).
- Exploration drilling and associated costs.
- Corporate overheads.
- Working capital provisions.
- Financing costs.
- GST and government levies (treated separately).

The overall FS mining costs include additional costs supported by the mining contractor. For example, diesel consumption is estimated by the mining contractor, assuming it is supplied by the mine's principal (Vista), with a diesel cost of AUD0.81/L (USD0.54/L), exclusive of GST, used to determine the costs incurred and included in the overall FS mining cost. Also, the people requirements on site, as estimated by the mining contractor, incur costs elsewhere in the overall Technical Report Summary – specifically for site accommodation in the mine camp and travel costs to commute to site.

Mining equipment costs provided by the mining contractor are also used to determine earthmoving costs applied elsewhere in the Technical Report Summary, including transport of rock for TSF construction and mine closure requirements.

18.2.5 Exchange Rates and Cost Escalation

The mining cost estimates are presented in USD.

Please note all mining contractor pricing and all other costs outlined in the mine cost estimate were provided in AUD and then converted to USD.

All pricing has been estimated in costs as of June 30, 2025, and no escalation has been allowed for.

18.2.6 Pre-Production Mining Capital Cost Estimate Summary

The pre-production capital estimate for the mining scope of the Project is based on a contractor-operated mining model. Under this model, the major mining equipment, supporting equipment and associated mine site infrastructure will be supplied, installed and operated by the mining contractor as part of their contractual mobilization and operating framework.

Pre-production capital costs within the mining scope include:

- **Contractor Mobilization and Site Establishment:** Lump-sum costs provided by the mining contractor to cover:
  - Mobilization of people, equipment and infrastructure to site.
  - Establishment of permanent mining-related site infrastructure in the pre-production period (see Section 21.6).
  - Site preparation including:
    - Permanent access and haul road construction to pit and ROM pad.
    - ROM pad establishment.
  - Any additional contractor pre-operational activities required prior to first mining.

Note: *The mining contractor's mobilization charge and monthly operating rates include provision for ownership and maintenance of major mining equipment, as well as supply and installation of key infrastructure items (detailed in Section 21.6). Accordingly, no separate line items are included in the owner's direct capital for mining equipment or contractor-furnished infrastructure.*

18.2.7 Mining Capital Cost Estimate

The mining capital scope is primarily based on a contractor-operated mining model. Under this approach, the mining contractor will be responsible for supplying the mining fleet and associated mine site infrastructure required to support open-pit mining operations. For consideration in this Technical Report Summary mining cost estimation, with relevant splits between capital and operating cost estimates, the ownership and maintenance of major mobile equipment is embedded within the contractor's:

- Mobilization and demobilization charges, and
- Agreed unit operating rates (AUD/BCM mined and converted to USD/BCM mined).

Accordingly, no separate line items for mining fleet capital acquisition are included in the Project's direct owner capital budget. The contractor's rates include provision for:

- Equipment ownership and leasing costs.
- Tire replacements.
- Major component rebuilds.
- Ongoing maintenance and reliability programs.

18.2.8 Mining Fleet

Based on the study's mine schedule requirements - considering mine design, site location, mining dilution and production rates - guidance was provided to the mining contractor regarding mining equipment selection and application.

An initial fleet of 10 x 190 tonne trucks will be required for the first 5 years of mine production, with a maximum fleet size of approximately 22 trucks required towards the end of mine life in 2050, due to pit depth and longer hauls to the WRD.

The drilling fleet consists of three platform drill rigs with a 30,000 kg pull-down capacity, drilling 171 mm diameter holes in various patterns.

Explosives supply costing is based on a downhole service provided by Orica, which includes delivery of bulk explosives to the Project site from the Yarwun AN plant near Gladstone, Queensland, and provision of all blasting accessories. Charging utilizes heavy ANFO blends to achieve the desired fragmentation, with 50% of explosives assumed to be emulsion due to wet holes, particularly in the wet season.

Further secondary drilling will be provided by two smaller nimble drill rigs with a knuckle boom for angled holes and horizontal holes for secondary breakage. These rigs will also be used for batter, buffer, trim shots, contour shots, toe blasting and other miscellaneous drilling.

Support equipment for primary mining will also be provided by the mining contractor and includes a fleet of track dozers, water carts, graders and other smaller equipment such as integrated tool carriers, cranes, small trucks, service trucks and light vehicles.

Up to three track dozers will be required to support the excavators and manage waste dumps and stockpiles.

Up to two water carts and two graders will be required at times to maintain optimum haul roads.

A secondary load and haul fleet is also provided and costed by the mining contractor. Rehandling of low-grade stockpile material to the ROM pad over the LOM and after mining has finished requires two 140 tonne dump trucks loaded by a 130 tonne front-end loader. The front-end loaders will also be used for ROM crusher feed when direct truck tipping is not occurring, or further ROM blending is required.

This secondary load and haul fleet is also assumed to be available for other small tasks on site, such as the planned TSF wall lifts, which are costed and discussed in other parts of this Technical Report Summary.

The mine schedule and cost estimate include a ramp-up at the commencement of mining, as outlined below:

- Pre-production period in Year -1 to allow construction of mine infrastructure to be ready for mining commencement in Year 1.
- First two months (Month 1 and Month 2, Year 1): planned production at 50 percent , equating to mobilization of one large hydraulic excavator and required trucks and support equipment.
- From the third month (Month 3, Year 1), planned production lifts to 100 percent, equating to mobilization of the second large hydraulic excavator and required trucks and support equipment.
- During month four (Month 4, Year 1), feed commences to the processing plant. This allows three months of mining to be completed to ensure adequate feed is available and ROM pad construction also is complete.

18.2.9 Mine Infrastructure

The Technical Report Summary assumes all associated mining non-productive site support infrastructure will be supplied, installed, and operated by the mining contractor as part of their contractual mobilization and operating framework. This infrastructure is planned to be constructed during the pre-production period in Year -1. The mining contractor is providing and installing the following site infrastructure required to support mining operations:

- HME Workshop using 6 domes to fit 190 tonne trucks with tray raised.

- Stores warehouse, drills workshop.
- Crib rooms, ablutions and meeting rooms.
- Wash pad, clean and dirty water ponds.
- Fuel and oil storage estimate supplied as a rate only.
- Explosive storage and facilities.
- Mine administration office and facilities.

Capital costs for this infrastructure are included in the contractor's mobilization and infrastructure establishment fees, considering engineering growth, and no additional direct owner capital is required for this scope. Vista considers that, as the pricing was provided by a highly experienced mining contractor, the associated risk factors have been adequately incorporated into the cost estimates. Accordingly, no additional contingency is deemed necessary.

Where applicable, interfaces and tie-ins with permanent site infrastructure (power supply, communications backbone, water supply and site roads) will be coordinated between Vista's project team and the contractor during the early works phase.

The site having, such a long life, will operate and retain personnel better with comfortable, safe and long-term infrastructure. A well-designed layout with concrete pads for workshops and other infrastructure based on a worst-case scenario rating for cyclonic zone B has been provided. The cost for a steel workshop structure was considerable and included utilizing an overhead crane and six bays high enough for 190 t trucks to raise trays under cover. This workshop was reconsidered by using 6 x domes on double stacked sea containers as workshop bays allowing the raised truck trays, for a considerable cost saving. The expected life span is 10 years for these so replacement costs for the dome structures were allowed for every decade and are included in the overall Technical Report Summary capital cost estimation.

Table 122 displays the summary of the mining infrastructure and mining services provided by the mining contractor. Vista considers that, as the pricing was provided by a highly experienced mining contractor, the associated risk factors have been adequately incorporated into the cost estimates. Accordingly, no additional contingency is deemed necessary

Mining Capital Costs	Initial Capital Cost (USDM)	Sustaining Capital Cost (USDM)
Mine Facilities and Light Vehicles	$19.17	$14.03
Mine Services, Mobilization and Demobilization	$2.86	$18.69
Total Mining Capital Costs	**$22.03**	**$32.72**

Table 122 Summary Mining Initial and Sustaining Capital

18.3 Process Plant and Infrastructure Capital Costs

18.3.1 Process Plant Capital Costs Summary

The process plant capital cost estimate is a **Class 3 estimate**, prepared in accordance with the standards applicable to feasibility studies (AACE doc 47R-11). This classification reflects the level of engineering definition and cost data available at the time of reporting. For a breakdown of estimate accuracy and supporting methodology, refer to Table 130.The base date for the estimate is the second quarter of calendar year 2025 (Q2 2025).

The GRES processing facility capital cost estimate has been developed based upon an Engineering, Procurement and Construction (EPC) approach for the delivery of the processing facility and includes contractor risk margin and contingency.

The capital cost estimate includes all costs associated with Project management, process engineering, design engineering, drafting, procurement, construction and commissioning services required to construct and commission the processing facility under GRES' scope.

A summary of the capital cost estimates compiled in Australian dollars for each of the processing plant elements has been listed in Table 123. Vista considers that, as the pricing was compiled by a highly experienced engineering and construction company, including a 7% engineering growth to cover costs for the inevitable increase from early order of magnitude estimates, the associated risk factors have been adequately incorporated into the cost estimates. Accordingly, no additional contingency is deemed necessary.

The figures in US dollars use an AUD:USD rate of 0.66 for pre-production capital estimation.

WBS Area and Description	Total Installed Costs (USDM)
3100 Crushing & Screening	$23.51
3200 Coarse Ore Stockpile, Reclaim & Ore Sorting	$9.80
3300 Classification and Grinding	$76.63
3400 Pre-leach Thickening, Leach & CIL	$22.66
3500 Desorption & Goldroom	$3.70
3600 Detoxification & Tailings	$4.52
3700 Reagents	$2.41
3800 Process Plant Services	$1.56
4130 Plant Wide Piping	$15.66
4200 Power Supply	$39.11
5000 Project Infrastructure	$5.97
7000 Site Establishment & Early Works	$2.10
8110 Project and Procurement Management	$8.02
8120 Engineering and Drafting	$12.06

WBS Area and Description	Total Installed Costs (USDM)
8130 Site Supervision and Management	$13.19
8140 Site Construction Cranes & Equipment	$12.14
8150 Site Construction Temporary Facilities	$0.83
8160 Construction Mobilization/Demobilization	$5.16
8170 Contractor Indirect Costs	$9.96
8300 Commissioning	$2.68
9000 Pre-Production Costs	$1.01
Engineering Growth (7%)	$18.86
Grand Total	**291.54**

Table 123 EPC Process Plant Capital Costs

The summary of the capital cost estimate, for the infrastructure items that were supplied by others and are additional to the EPC process plant capital costs listed above, these are given below in Table 124. The figures in US dollars use an AUD:USD rate of 0.66 for pre-production capital estimation. These capital estimates are based on a EPCM delivery model and are not part of the EPC Process Plant Capital Costs.

WBS Area and Description	Total Installed Costs (USDM)
4310 Communications	$0.33
5700 Power Transmission (Overhead line from power station)	$2.70
6000 Permanent Camp	$19.15
7300 Construction Camp (Leased)	$5.43
7000 Power Station Earthworks	$0.08
9200 Commissioning Expenses (First Fills/Commissioning Spares	$3.87
9300 Capital Spares	$9.83
Engineering Growth (7%)	$3.01
Grand Total	**$44.40**

Table 124 Process Plant Direct Vista Packages

18.3.2 Currency

Most direct costs were sourced in Australian dollars; however some major equipment will be sourced from foreign locations. Vista provided GRES with the reference FOREX rates to apply to the estimate as shown in Table 125.

FOREX	Currency	Rate
AUD/USD Exchange rate	USD	0.66
AUD/EUR Exchange rate	EUR	0.58

Table 125 Currency Exchange

18.3.3 FOREX Rates Initial Capital

Foreign Currency Sourced Equipment

Major equipment sourced in foreign currency values are shown in Table 126.

Currency	Value	Value (AUD)	Rate
United States Dollar	$34,588,500	$52,406,818	0.66
Euro	€1,650,000	$2,844,828	0.58

Table 126 Foreign Currency Sourced Equipment

18.3.4 Work Breakdown Structure

The work breakdown structure used in the estimate are as shown in the Table 127.

WBS Area Level 1	Description
3100	Crushing and Screening
3200	Coarse ore stockpile, reclaim and ore sorting
3300	Classification and grinding
3400	Pre-leach thickening, leach and CIL
3500	Desorption and gold room
3600	Detoxification and tailings
3700	Reagents
3800	Process plant services
4130	Plant-wide piping
4200	Power supply (electrical costs and installation around plant and infrastructure)
4800	Fuel storage and distribution
5000	Project infrastructure
7000	Site establishment and early works
8110	Project and procurement management
8120	Engineering and drafting
8130	Site supervision and management
8140	Site construction cranes and equipment
8150	Site construction temporary facilities
8160	Construction mobilization
8170	Contractor indirect costs
8300	Commissioning
9000	Pre-production costs

Table 127 Work Breakdown Structure for Process Plant Capital

18.3.5 Disciplines

The discipline codes used in the estimate are based on the GRES standard discipline codes provided in Table 128.

Discipline code	Description
ER	Earthworks
CI	Civil Works
ME	Mechanical Equipment
PL	Platework
SS	Structural Steel
EL	Electrical Installations
BD	Buildings
PI	Piping
CE	Construction Equipment
OW	Owner's Costs
TF	Temporary Construction Facilities
SCM	Supervision and Construction Management
PM	Project and Procurement Management
DE	Engineering Design
VC	Vendor Commissioning
CO	Commissioning
IF	Initial Fills
IS	Insurance Spares

Table 128 Process Plant Discipline Codes

18.3.6 Contracting Strategy

The estimates were prepared based on a contractor providing engineering, procurement and construction ("EPC contract"). This includes the provision of the project management, process engineering, design engineering, drafting, procurement, construction and commissioning services required to construct and commission the proposed processing facilities and associated infrastructure.

18.3.7 Scope Included

The section below broadly describes the scope of work for this Technical Report Summary. This scope is not intended to be definitive, for more detail refer to the process flow diagrams, the mechanical equipment list, the plot plan and general arrangement drawings, and the electrical single line diagrams.

18.3.8 Scope Inclusions

The capital cost estimate includes, broadly the following direct costs:

Design, supply, delivery, installation and commissioning of a two-stage crushing plant including:

- ROM dual dump pocket.
- Primary gyratory crusher.
- Apron feeder.
- Secondary cone crusher.
- Dust collection system.
- Screening stations.
- Conveyor systems.

Design, supply, delivery, installation and commissioning of a crushed ore stockpile and reclaim system including:

- Conveyor systems.
- Reclaim feeders.

Design, supply, delivery, installation and commissioning of a High-Pressure Grinding Rolls (HPGR) area including:

- HPGR.
- Surge and feed bins.
- Screening stations.
- Conveyor systems.

Design, supply, delivery, installation and commissioning of an ore sorting area including:

- Ore sorters.
- Fed bins.
- Product conveyor systems.
- Reject conveyor systems.

Design, supply, delivery, installation and commissioning of a grinding area including:

- Ball mill.
- Vertimills.
- Classification cyclones.
- Screening systems.
- Slurry pumping systems.

Design, supply, delivery, installation and commissioning of a pre-leach area including:

- Thickener.
- Conditioning tanks.
- Slurry pumping systems.

Design, supply, delivery, installation and commissioning of a leaching area including:

- Leach tanks.
- CIL tanks.
- Screening systems.
- Pumping systems.

Design, supply, delivery, installation and commissioning of a desorption and gold room area including:

- Elution column.
- Acid wash column.
- Electrowinning system.
- Smelting system.
- Carbon regeneration system.

Design, supply, delivery, installation and commissioning of a tailings area including:

- Detoxification system.
- Pumping system.
- TSF 1 and TSF 2.
- All decant and pipeline water returns.
- All pipework for depositing tailings to TSF 1 and TSF 2.

Design, supply, delivery, installation and commissioning of a reagents area including:

- Reagents receival systems.
- Reagents storage systems.
- Reagents distribution system.

Design, supply, delivery, installation and commissioning of Infrastructure areas including:

- Administration building.
- Workshop building.
- Warehouse building.
- Emergency services building.

- Water treatment systems.
- Waste handling for solids and water systems.
- Site access roads.

Design, supply, delivery, installation and commissioning of a power distribution system.

Design, supply, delivery, installation and commissioning of a communications system including:

- Satellite receival and transmission system.
- Fibre-optic backbone.
- Base station UHF system.

The capital cost estimate includes the following indirect costs:

- Engineering, procurement and construction (EPC) costs.
- Commissioning.
- Vendor commissioning attendance.
- General and administration charges.
- Margin, engineering and quantity growth.

18.3.9 Scope Excluded

The Process Plant Capital cost estimate broadly exclude those items listed by:

- Owner's costs such as corporate overheads, engineering services, project management, flights, staffing costs.
- Owner's contingency.
- Permanent and construction camp and all associated facilities.
- Site fencing.
- Government duties, taxes, permits and fees.
- Licence fees.
- Land cost, right of way, royalties.
- Environmental and ecological considerations (statutory approvals).
- Insurances.
- Activities of previous phases.
- Technical reviews.
- Project audits.
- Mine equipment and development.
- Mine infrastructure.
- Ore transport infrastructure.
- Lifts or expansions of existing tailings storage.

- Water supply infrastructure.
- Foreign exchange rate exposure.
- Geotechnical issues.
- Industrial disputes.
- Salvage value of temporary facilities.
- Post-commissioning activities (ramp-up).
- Inventory spares.
- Maintenance tools.
- Training of plant operators and maintainers.
- Finance and interest during construction.
- Working capital.
- Sustaining capital outside process plant which is included in operating costs.
- Tailings storage facility.
- Raw water distribution around the site.
- Site diversions and earthworks associated outside the process plant.
- WTP.
- Capitalized depreciation.
- Salvage value of permanent facilities at end.

18.3.10 Estimate Methodology

The capital cost estimate was prepared using an in-house spreadsheet-based template. The estimate was broken into the following areas:

- Activity classification.
- Activity description.
- Quantities.
- Supply cost.
- Installation cost.
- Freight cost.

18.3.11 Activity Classification

Each line item in the capital estimate was tagged with different activity and organizational classifications. These tags assist in generating different capital cost estimate output tables. Classifications in this area include:

- Work breakdown structure.
- Discipline (class code).
- Contract package (CP) number and description.
- Activity code (more detailed classification).

18.3.12 Activity Identification

The activity identification section is the primary descriptor for each line in the estimate. Each activity was grouped by the project WBS and then by common activities for each item:

WBS code and description:

- Discipline description:
  - Item description.

18.3.13 Quantities

Each item has four quantity fields:

- Quantity per unit:
  - Estimated quantity per item.
- Units:
  - The unit of measure of the quantity.
- Number of items:
  - Multiplier for number of items per unit.
- Total quantity:
  - Quantity per unit x number of items.

18.3.14 Supply Cost

Materials/equipment supply cost is calculated using three fields:

- Unit rate:
  - Cost per unit supplied.
- Base cost:
  - Net cost for all required units, based on the total quantity.
- Cost:
  - Total cost of supply (including mark-up where required).

18.3.15 Installation Cost

Installation cost is calculated using six fields:

- Hours per unit:
  - Estimated hours per unit.
- Total hours:
  - Total hours for the item.
- Labor rate:
  - Hourly labor rate for the item (including productivity factor where required).
- Equipment rate:
  - Hourly equipment rate for the item activity (small equipment only; with cranes and major equipment accounted for separately).
- Base cost:
  - Net cost of labor and equipment for erection/construction activities.
- Cost:
  - Erection cost inclusive of contractor margin.

18.3.16 Freight Cost

Freight cost is calculated using six fields:

- Quantity per unit:
  - Estimated quantity per item.
- Units:
  - The unit of measure of the quantity (mass, volume or number).
- Rate per unit:
  - Cost per unit of freight.
- Base cost:
  - Net cost of item freight.
- Cost:
  - Total cost of freight (includes mark-up where required).

18.3.17 Escalation

Escalation, such as future rises in equipment or material prices, is not included in the rates calculation. Where escalation is applied this shall be clearly defined in the estimate.

18.3.18 Engineering Growth

Engineering growth is included based on an assessment of the maturity of the design and is noted as a percentage against each line item in the estimate.

18.3.19 Engineering Basis

The engineering basis for the estimate is as called for by AACE document 47R-11 shown in Table 129 for those items relevant to the Mt Todd Gold Project. It should be noted that many of the deliverables are called for to be "complete". This should not be confused with "final". Finalization of engineering and design cannot occur prior to the delivery of certified vendor information. Such information is generally not available prior to the placement of orders. The term "complete" then shall be assumed to mean that the deliverable is complete to the extent possible considering the vendor information available.

Deliverable	Status
Project scope definition *	Defined
Mine and plant production/facility capacity *	Defined
Plant location	Specific
Geology *	Defined
Geotechnical and rock mechanics *	Defined
Metallurgical test work *	Defined
Block flow diagrams	Complete
Plot plans	Complete
Process flow diagrams	Complete
Piping and instrument diagrams	Complete
Mass balance	Complete
Mechanical equipment list	Complete
Electrical single line drawings	Complete
Specifications and data sheets	Preliminary/Complete
GA drawings	Preliminary/Complete
Discipline specific drawings	Preliminary/Complete

*Items shown * are Vista or third party provided (not GRES).*

Table 129 Engineering Status

18.3.20 Class 3 Estimate Requirements

The capital estimate was developed generally in accordance with the directions contained within AACE International Recommended Practice No. 47R-11 (Cost Estimate Classification System – As Applied in the Mining and Mineral Processing Industries). The capital estimate was developed to class 3 requirements.

A class 3 estimate shall be defined as follows:

Estimate Class	Maturity Level of Project Definition Deliverables	End Usage	Methodology	Expected Accuracy Range
Class 3	10% - 15%	Funding authorization	Semi-detailed unit costs with assembly level line items	L: -10% to -20% H: +10% to +30%

Table 130 Class 3 Estimate Accuracy

To achieve a class 3, estimate the following applies.

18.3.21 Earthworks

Rates adopted for the estimate were based on recent subcontractor submissions for similar works in Australia, with an allowance for indirect costs. The rates used were inclusive of equipment hire, operator and maintenance labor, fuel, consumables, materials, indirect costs and margin. The rates applied do account for the remote nature of the site.

18.3.22 Equipment

The process design criteria was used to develop the mechanical equipment list that defines the requirements and sizes of all mechanical equipment. Datasheets shall be developed for major process equipment only.

Preliminary (budget) inquiries were placed for major process equipment and electrical items. These inquiries were emailed including the scope of supply, specification and data sheet. Commercial conditions for these inquiries were assumed to be the vendor's standard including 10% retentions (2 x 5%) and 12 months defects liability.

Budget quotations for the following major equipment items are shown in Table 131:

Contract Package	Equipment Type	Vendors
CP001	Crushers	Metso / Sandvik / Weir
CP001A	HPGR	FLS / Metso / Weir / Citic
CP002	Screens	Joest / Oreflow / Weir
CP003	Apron feeders	Oreflow / Metso
CP010	Grinding mills	Citic / Metso

Contract Package	Equipment Type	Vendors
CP012	Slurry pumps	Weir
CP013	Process pumps	Global
CP016	Thickeners	Metso / Takraf / Roytec
CP017	Agitators	Mixtec / SPX Flow / Afromix
CP018	Cyclones	FLS / Metso / Weir
CP020	Regeneration kilns	Como / Heatsystems
CP021	Elution	TP&E / Stantil / Como
CP033	Intertank screens	Derrick / Kemix
CP100	Vertical regrind mills	FLS / Metso / STM
CP2010	MCC & switchroom	Plummers
CP2030	HV switchroom	ABB
CP2040	Transformers	Hitachi / Wilson
CP2051	LV Variable speed drives	ABB

Table 131 Contract Packages

18.3.23 Minor Equipment Items

Minor items of process equipment were priced based on recent quotations for similar equipment or actual incurred costs, this totalled to less than 10% of the total direct equipment costs.

A fixed lump sum quotation (in addition to the budget quotation) for the supply and delivery of the mill was sought. If possible (considering the timeframes) a fixed lump sum quotation for the supply and delivery of the crushing plant was also sought.

18.3.24 Structural Steel

Structural steel is estimated based on the development of material take-offs applied to preliminary layouts developed for the Project. Supply and fabrication rates for similar complexity work supplied out of Australia and Asia shall be used and applied to the MTOs. Such rates were budget rates quoted by appropriately qualified fabricators. Appropriate escalation factors were applied, if necessary, to these rates to cover the current raw steel prices. The rates are ex-works, inclusive of shop detailing, materials, fabrication labor, painting materials and labor, consumables, indirect costs and margin for three mass classifications of structural steelwork, conveyor trestle and truss steelwork, grating, handrailing and stair treads.

18.3.25 Structural Concrete

Structural concrete is estimated based on the development of material take-offs applied to preliminary layouts developed for the Project. Rates for the supply and installation of structural concrete were established based on budget quotes received from preliminary enquiries to relevant concrete suppliers, taking into account the availability of suitable materials (aggregates) and the mobilization of a batching plant.

18.3.26 Platework

The mechanical equipment list defines all items of platework required, including bins, chutes, tanks and hoppers. The mechanical equipment list displays the plate thickness and the type and thickness of wear liner.

Quantities are estimated on an item-by-item basis and/or estimated from platework used for similar applications on previous projects.

Rates adopted for the estimate are based on budget quotes from relevant fabricators for similar works, appropriately escalated, if necessary, to cover recent increases in raw steel prices. The rates are ex-works and inclusive of materials, fabrication labor, painting materials and labor, consumables, as well as indirect costs and margin.

18.3.27 Piping

Plant piping costs were developed from recent projects of a similar scope, appropriately factored to cover differences in overall scope. Consideration was given to the extents of wear lined pipe (rubber) included in the scopes.

Overland piping quantities were developed considering the cost of materials and labor required to supply and install the pipe.

18.3.28 Electrical and Instrumentation

The electrical, instrumentation and control material quantities were compiled using the single line diagrams, process flow diagrams, layout drawings, equipment list and load list. Cable and material quantities are estimated from a preliminary cable schedule.

Pricing for major electrical equipment including transformers, HV switchgear and LV switchgear and switchrooms are developed from budget inquiries placed to vendors. Rates for minor items of electrical equipment and materials are based on recent similar projects.

18.3.29 Installation Labor

Estimates for site installation labor is based on the estimated historical man-hours associated with the equipment, materials and fabricated items to be installed in each area of the plant. The estimated hours for installation reflect a labor force productivity typical of Australian construction sites for the minerals industry.

Crew rates for each trade are built up to include an appropriate mix of supervision, skilled and unskilled personnel. Each rate includes the costs of mandatory site safety meetings, meal breaks, provision of small tools, statutory labor costs, personal protective equipment, clothing, supervision, indirect costs (including flights, site messing and accommodation) and margins.

Labor rates are developed using known labor agreements & budget quoted rates from organizations familiar with working on similar remote sites which are inclusive of relevant NT levies and allowances.

18.3.30 Cranage and Equipment Costs

Costs developed for the required site cranes and construction equipment are based on estimates of the required durations to complete installation works in each area of the plant. Industry-standard budget charge-out rates for the various cranes and equipment types are applied to the estimated hire durations. The costs for mobilization and demobilization of the equipment to site are accounted for. Diesel fuel required for site construction equipment and vehicles are supplied by Vista and charged back to the contractor at the current rate paid by Vista.

18.3.31 Transport

Major equipment suppliers were requested to provide estimates of the costs of transporting their equipment to site. Where suppliers do not provide estimates of transport costs, an allowance based on the estimated size or weight of each item was made using current rates from a transport provider.

18.3.32 Commissioning Spares

An allowance of 0.5% of mechanical equipment supply was included in the estimate.

18.3.33 Insurance/Capital Spares

The following allowances were recommended to be made for insurance/capital spares; this has been included in the estimate:

- 4.0% of the mechanical supply cost.
- 4.0% of the electrical supply cost.

18.3.34 Indirect On-Site Costs

The estimate assumes that on-site facilities including the following will be provided by Vista at no cost to the project:

- Offices.
- Stores.
- Workshops.
- Communications.
- Ablution facilities.
- Crib facilities.
- Cleaning services.
- Waste removal.

An allowance for the onsite transport of construction crews and management personnel was included in the cost estimate.

The estimate assume flights, meals and accommodation costs and coordinated by Vista. Rosters are based on a three-weeks-on and one-week-off rotation during construction on a fly-in fly-out basis. An allowance of AUD1,300 per return flight (Brisbane-Katherine-Brisbane or Perth-Katherine-Perth) and AUD95/night for accommodation and messing has been allowed in the estimate based on vendor quotes provided to Vista.

18.3.35 Project Delivery Services

The estimates for project management, engineering, procurement, construction management and commissioning services were based on the hours required for a multi-disciplined project delivery team, consisting of suitably qualified and experienced personnel, to successfully complete the scope within the scheduled timeframe. The manhours are based on the execution schedule and/or engineering deliverables of similar previous projects.

The EPC contractors' personnel costs are based on current 2025 rates.

18.3.36 Contingency

Contingency was applied to the capital cost estimate. Contingency was applied to cover uncertainty in the rates and / or quantities applied to an estimate item. Contingency was established through the use of a probabilistic assessment of the capital cost estimate.

18.3.37 Tailings Storage Facility

Tierra Group provided the basis of the design and all the MTO's and are responsible for the TSF 1 and 2. GRES was provided with the MTO template from Tierra Group and applied the relevant capital cost estimate rates for these MTOs. The rates that were applied are as described in Section 18.4.1.

18.3.38 Power Station

GRES was provided with a layout and footprint for the power station by Power Contractor and provided the capital costs for the bulk earthworks only to allow Power Contractor to install their power plant.

18.3.39 Accommodation

GRES was provided a layout and footprint for the accommodation by third parties and provided the capital costs for the bulk earthworks only, to allow the third parties to install their accommodation.

The third parties were responsible for the design, supply, deliver, install and commission a Construction and Permanent Accommodation facility including:

- 250-person permanent accommodation village (partially used during construction).

- 200-person construction accommodation village.
- Messing facilities.
- Admin building.
- Laundry facilities.
- Water treatment system.
- Waste treatment system.
- Access roads.
- Parking area.

18.3.40 Owner's Costs

The Owner's costs were not included in the estimates.

Other Owner's costs such as the Owner's project management team and consultants, approvals and licenses, operational readiness, business systems, pre-production costs (including first fills, spares), insurances and inventory spares, were provided by Vista for inclusion in the estimate.

18.4 Non-Processing Infrastructure and Facilities Capital Costs

The following details those areas outside the mine and process plant capital costs, such as TSF, Water Management and Reclamation and Closure costs.

18.4.1 Tailings Storage Facilities Capital Costs

Tierra Group developed MTO's based on the engineered design of TSF components. The TSF1 and TSF2 are designed to have a combined capacity of 172.8 Mt, which exceeds the 159 Mt included in the current operating plan. Unit costs used to estimate TSF construction costs were provided by GRES.

Vista considers that given that the cost estimates for the TSF are derived from designs that exceed the necessary operational capacity and the unit rates have been established based on realistic construction and operational costs, potential risk factors have been integrated into the cost assessment. Therefore, the inclusion of additional contingency provisions is not deemed necessary at this time.

Table 132 show figures in US dollars using a AUD:USD rate of 0.66 for pre-production capital estimation and AUD:USD rate of 0.67 for production capital estimation summarizing the estimated initial and sustaining capital costs for the tailings storage facilities.

WBS No.	Description	Initial Capital (USDM)	Sustaining Capital (USDM)	Total Capital (USDM)
4410	**TSF 1**			
1	Site & Foundation Preparation	$0.21	$3.95	$4.16

WBS No.	Description	Initial Capital (USDM)	Sustaining Capital (USDM)	Total Capital (USDM)
2	Embankment Construction	$0.81	$23.12	$23.93
3	Downstream Shell Construction	$0.00	$6.11	$6.11
4	Blanket Drain	$1.34	$1.34	$2.68
5	West Drain	$0.26	$0.00	$0.26
6	Downstream Toe Drain	$0.23	$0.00	$0.23
7	Decant System	$0.00	$1.12	$1.12
8	Return Water Ponds	$0.18	$0.00	$0.18
	4410 TSF 1	**$3.03**	**$35.64**	**$38.67**
4420	**TSF 2**			
1	Site & Foundation Preparation	$0.00	$2.21	$2.21
2	Underdrain Construction	$0.00	$0.52	$0.52
3	Liner System	$0.00	$42.84	$42.84
4	Overdrain & Reclaim Sump/Pond Construction	$0.00	$1.56	$1.56
5	Embankment Construction	$0.00	$67.56	$67.56
6	Decant System	$0.00	$0.39	$0.39
7	Tailings Delivery and Return Pipelines	$0.00	$0.90	$0.90
	4420 TSF 2	**$0.00**	**$115.98**	**$115.98**
	4400 TSF 1 and 2 Total	**$3.02**	**$151.62**	**$154.65**

Table 132 Estimated TSF Capital Cost Summary

18.4.2 Mine Dewatering Capital Costs

Mine dewatering capital costs are based on direct vendor quotes or Tetra Tech in-house estimates. Dewatering will be conducted by the mining contractor therefore the costs shown in Table 133 are for the pipeline from the edge of the pit to the Water Treatment Feed Pond.

WBS No.	Description	Initial Capital (USDM)	Sustaining Capital (USDM)
7400	PPD Dewatering	$0.83	$0.95
	Engineering Growth and Contingency (10%)	$0.08	$0.09
	Total	**$0.92**	**$1.04**

Table 133 Estimated Mine Dewatering Capital Cost Summary

18.4.3 Water Treatment Plant Capital Costs

Water treatment plant capital costs are based on direct vendor quotes or Tetra Tech in-house estimates, initial capital costs are estimated at USD20.5 million. It is estimated that the plant would need a major overhaul at year 20 which is captured in sustaining capital, see Table 134.

WBS No.	Description	Initial Capital (USDM)	Sustaining Capital (USDM)
4110	Water Treatment Plant	$18.77	$11.26
	Engineering Growth and Contingency (10%)	$1.74	$1.05
	Total	**$20.51**	**$12.31**

Table 134 Estimated Water Treatment Plant Capital Cost Summary

18.4.4 Raw Water Dam and Distribution Capital Costs

Raw water dam capital costs are based on direct vendor quotes or Tetra Tech in-house estimates. These costs include upgrades to the existing raw water infrastructure and a new line from the process plant area to the camp. Initial capital costs are estimated at USD1.32 million, and a sustaining capital cost of USD1.79M, see Table 135.

WBS No.	Description	Initial Capital (USDM)	Sustaining Capital (USDM)
4120	Raw Water Dam	$1.20	$1.63
	Engineering Growth and Contingency (10%)	$0.12	$0.16
	Total	**$1.32**	**$1.79**

Table 135 Estimated Raw Water Dam Capital Cost Summary

18.4.5 Reclamation and Closure Capital Costs

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. Tetra Tech developed MTO's for clean waste (NAF) material requirements for closure activities based on the engineered design of main Project's components. Unit costs used to estimate NAF material handling for closure were provided by Mining Plus as detailed in Section 18.2.

Costs were compiled in Australian dollars, Table 136 show figures in US dollars using an exchange rate AUD:USD of 0.67.

Capital costs for reclamation are estimated at USD176 million for the life of the Project.

WBS No.	Description	Reclamation Capital (USDM)
4910	WRD	$29.47
4920	TSF1	$32.54
4931	TSF2	$18.99
4932	Heap Leach Pad	$1.44
4940	Low grade stockpile	$1.05
4950	Process Plant Area	$8.74
4960	Soil Stockpiles	$0.36
4970	Mine Roads	$0.98
4980	Batman Pit	$0.39
4990	Passive Treatment Systems	$1.05
4998	Indirect Costs	$7.01
4999	NAF material handling for closure	$58.22
	Engineering Growth and Contingency (10%)	$16.02
4900	**Total Reclamation and Closure**	**$176.26**

Table 136 Estimated Reclamation and Closure Capital Cost Summary

18.5 Owners Capital Costs

18.5.1 Pre Production Capital Costs

An allowance of was developed from the Project start date for the costs associated with the ramp up to full production and this included the following:

- Labor, (including transport and accommodation) – include Vista management, mining, process plant.
- General Expenses.
- Light Vehicles operating costs.
- Environmental Levy.
- G&A Capital for Support and Light Vehicles.

This was benchmarked against similar Australian gold operations.

18.6 Overall Operating Costs

Section 18.1.1 provides the scope covered for the overall operating costs.

Detailed mining costs have been provided by a well-established Australian contract miner. Power costs are based on a detailed power generation proposal from one of Australia's leading mine site contract power generators. This Technical Report Summary uses a fixed natural gas price of AUD8.50 per gigajoule. Vista has not yet negotiated a gas supply contract as it is not in a position currently to execute a take or pay contract.

Processing and G&A costs have been developed from first principles with quotes for major consumable supply components and competitive Australian labor rates. The operating costs contemplate that an initial 90% of the workforce will be contracted on a fly-in- fly-out basis and be housed in a 250-bed permanent camp facilities near the mine site. Operating costs before taxes and depreciation, on a unit cost basis are shown in the following Table 137.

Operating Cost Description	Units	Years 1- 15	LOM Yr 1-30
Mining Costs	USD/t processed	$18.49	$16.55
Processing Costs[6]	USD/t processed	$17.70	$17.62
G&A Costs	USD/t processed	$2.09	$2.09
JAAC Royalty	USD/t processed	$2.22	$2.08
Wheaton Royalty	USD/t processed	$0.84	$0.73
Refining Costs	USD/t processed	$0.15	$0.14
Total Cash Costs	**USD/t processed**	**$41.49**	**$39.20**

Table 137 Overall Operating Cost Summary

The subsequent sections will detail these main operating costs listed above.

18.7 Mining Operating Costs

The mining contractor has some mining engineers as they will have to provide short term and operations mine design and scheduling, with longer term planning and geological guidance provided by the Vista technical services team, which is not outlined in the mining contractor's organization chart.

Mining operating costs have been developed based on mining contractor-supplied unit rates applied to the LOM mining schedule completed within this Technical Report Summary and provided to the mining contractor. The mining operating costs scope is primarily based on a contractor-operated mining model. Under this approach, the mining contractor will be responsible for operating and maintaining the mining fleet and associated mining site infrastructure required to support open pit mining operations. The mining study team worked in partnership with the mining contractor to obtain the price estimates outlined in this Technical

[6] Incudes water management costs of approximately USD0.78/t processed.

Report Summary. This estimate has also been compared to recent industry database cost estimates to ensure reasonableness for use in this Technical Report Summary.

Mining operating costs include the contractor mining rates, and their monthly fixed costs outlined below. Also, additional costs such as mine technical services and other members of the operations owner's team, grade control assays, and geotechnical monitoring, have been included in the overall operating mining costs for this Technical Report Summary, as addition to the mining contractor cost estimate.

18.7.1 Contractor Mining Rates

Costs provided by the mining contractor in AUD/BCM mined and converted to USD/BCM mined.

- Load & Haul: mined by bench elevation based on the provided mine design and timing based on FS mine schedule.
- Drilling and Blasting: for ore, waste and waste for civil use (additional costs for waste rock blasting for rock needed at the TSF has been estimated based on mining contractor costs but included in other part of this cost estimate within this Technical Report Summary).
- Ore Stockpile Rehandle: AUD/t handled.
- ROM rehandle AUD/t handled based on 50% percentage of plant feed is rehandled on ROM by a front-end loader.

18.7.2 Monthly Fixed Costs

- Contractor site personnel salaries.
- Fixed overheads (light vehicles, small equipment maintenance, safety, training, freight).
- Site services (dewatering pumps and crew).
- Maintenance of contractor-supplied infrastructure.

18.7.3 Productivities – Load and Haul

The productivities detailed in this section are based on the experience and expertise of the tier 1 mine contractor and have also been reviewed by the Technical Report Summary team and deemed reasonable for the proposed operation.

18.7.4 Productivities - Drill and Blast

Drill penetration rates assumed for the 30,000 kg Platform rig was 25 m/60 min hour in Waste and 20 m/hr in ore.

50% of the holes are considered to be wet due to ground water and large rainfall during the summer wet season, and the patterns proposed by the Orica study were used to price Drill and Blast split only by ore and waste. The parameters are listed in Table 138.

Material Type	Bench m	BCM m^3	Pen rate (m/pwt)	Pattern	Explosive	Powder Factor
Waste dry kg bulk Fortan	12	114,103,601	25	A3	Fortan	0.57
Waste wet kg bulk Fortis	12	114,103,601	25	A2	Fortis	0.58
Ore dry kg bulk Fortan	12	31,152,384	20	C3	Fortis	0.84
Ore wet kg bulk Fortis	12	31,152,384	20	C2	Fortan	0.84

Table 138 Blasting Parameters

18.7.5 Physicals and Consumables

Estimates for physical and consumables required for the mining operations are outlined in the following section and referred tables.

- Diesel consumption estimated by contractors detailed modelling and estimation (AUD0.81/L (USD0.54/L), exclusive of GST applied for costing), see Table 139.
- Explosives consumption (included in the contractor rates), based on drill and blast pattern designs requirements see Table 139.
- Estimated Water usage, see Table 139.
- Equipment fleet breakdown by period, see Table 140.
- Contractor Mining Operations Personal count by period.

Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31
Drill Water (kL)	0	1,077	1,023	1,295	1,181	1,347	1,136	1,144	1,198	1,291	1,248	1,246	1,230	1,197	1,192	1,172	1,168	1,277	1,105	1,138	1,160	1,119	1,168	1,215	1,270	945	552	447	0	0	0	0
Potable Water (kL)	24	646,000	720,480	273,600	273,600	297,920	69,160	76,000	79,040	80,560	75,240	76,000	74,480	74,480	74,480	77,520	79,800	83,600	75,240	77,520	85,120	85,120	86,640	89,680	93,480	95,000	40,280	37,240	3,800	3,800	4,560	0
Diesel (Litres)																																
- Plant Diesel (kL)	19	10,425,894	11,208,234	13,378,802	15,323,115	16,996,736	16,051,290	17,531,906	19,341,706	19,645,539	18,431,745	17,707,882	16,979,772	17,104,218	17,091,480	18,116,312	18,507,139	20,366,635	17,355,094	17,954,261	21,775,810	21,399,299	21,852,920	23,658,299	24,797,444	14,873,628	9,129,583	7,377,345	772,062	603,820	2,534,140	0
- Explosives Diesel (kL)	0	267,734	235,549	265,312	247,117	332,024	232,449	232,523	256,833	299,097	270,477	276,888	267,257	251,022	256,456	251,582	253,203	300,834	229,404	245,929	251,119	230,845	245,338	269,361	296,045	210,563	151,995	101,241	0	0	0	0
DIESEL TOTAL	19	10,693,628	11,443,783	13,644,114	15,570,232	17,328,760	16,283,739	17,764,429	19,598,539	19,944,636	18,702,222	17,984,770	17,247,029	17,355,240	17,347,936	18,367,894	18,760,342	20,667,469	17,584,498	18,200,190	22,026,929	21,630,144	22,098,258	23,927,660	25,093,489	15,084,191	9,281,578	7,478,586	772,062	603,820	2,534,140	0
Explosives (kg)																																
- Heavy ANFO Fortan Extra 1.2	0	3,050	3,010	3,804	3,594	3,876	3,526	3,514	3,625	3,791	3,749	3,742	3,693	3,627	3,607	3,558	3,542	3,745	3,427	3,490	3,529	3,454	3,539	3,623	3,723	2,712	1,494	1,162	0	0	0	0
- Emulsion Fortis Extra 1.2	0	3,074	3,044	3,847	3,645	3,912	3,581	3,566	3,675	3,834	3,798	3,791	3,740	3,677	3,656	3,607	3,590	3,787	3,480	3,542	3,579	3,507	3,587	3,668	3,764	2,737	1,500	1,162	0	0	0	0
- Stemming	0	2,528	2,591	2,946	3,096	3,385	3,070	3,026	3,129	3,264	3,226	3,255	3,177	3,117	3,116	3,090	3,082	3,252	3,005	3,066	3,231	3,037	3,038	3,118	3,209	1,977	1,274	1,074	0	0	0	0

Table 139 Water, Diesel and Explosives Consumption by Period

Equipment #	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31
400 t excavator	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	0	0	0	0
190 t Dump Trucks	0	10	10	12	12	14	12	15	16	16	14	15	14	14	14	16	16	18	15	16	19	19	21	22	24	24	24	4	0	0	0	0
130 t FEL	1	1	1	1	1	1	1	1	1	1	1	0	1	0	0	1	1	1	1	0	1	1	1	1	1	1	0	0	0	0	1	0
140 t Dump Trucks	2	2	2	2	2	2	2	2	2	2	2	0	2	0	0	2	2	2	2	0	2	2	2	2	2	2	0	0	0	0	2	0
Dozers	1	2	2	2	3	3	3	3	3	3	3	3	3	3	2	2	2	2	2	2	2	2	2	2	2	2	2	1	0	0	0	0
Graders	1	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	3	1	0	0	0	0
Watercarts	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	3	1	0	0	0	0
Drills	0	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	1	0	0	0	0

Table 140 Equipment Profile by Period

18.7.6 *Personnel*

The Vista mining team comprising mining management and technical services, as per the positions and assumed rosters outlined below. Positions are assumed to be FIFO also for this Technical Report Summary, with a focus to place personnel locally, if possible, particularly for roles such as Geotech and Geology Technician, and Surveying assistant.

- Chief Mining Engineer (roster 8/6).
- Senior Mine Planning Engineer (roster 8/6).
- Mining Engineer (roster 8/6).
- Mine surveyor (roster 14/7).
- Surveying Helper (roster 14/7).
- Geology Superintendent (roster 8/6).
- Grade Control Geologist (roster 14/7).
- Senior Geotech Eng (roster 8/6).
- Pit Geotech Technician (roster 8/6).
- Geology Field Technician (roster 8/6).
- Contract Management Superintendent (roster 5/2).
- Project Engineer (roster 8/6).

The salaries and labor costs have derived from the 2025 Hays Salary Guide and reflect current labor market conditions and site location.

The mining owner team is assumed to predominantly be on FIFO rosters from other parts of Australia. This is based on the operational philosophy adopted by the Project. It is assumed recruitment and of personnel would likely be achieved with 50% from the Perth region and 50% from the Brisbane region.

An on-cost of 27% has been applied to the salaries as shown above to allow for insurance, leave, superannuation and payroll taxes.

Costs were compiled in Australian dollars, Table 141 show figures in US dollars using an exchange rate AUD:USD of 0.67 for an annual breakdown of labor requirements.

Vista Mining Personnel	Roster	Mining Operations (Personnel)	Costs AUD/year	Costs USD/year
Chief Mining Engineer	8/6	1	$301,357	$201,909
Senior Mine Planning Engineer	8/6	1	$275,957	$184,891
Mining Engineer	8/6	2	$496,034	$332,343
Mine Surveyor	14/7	4	$944,848	$633,048
Mine Surveyor Helper	14/7	4	$457,200	$306,324

Vista Mining Personnel	Roster	Mining Operations (Personnel)	Costs AUD/year	Costs USD/year
Geology Superintendent	8/6	1	$301,357	$201,909
Grade Control Geologist	14/7	2	$447,024	$299,506
Pit Geotech Technician	8/6	1	$180,707	$121,074
Senior Geotech Engineer	8/6	1	$295,007	$197,655
Geology Field Technician	8/6	2	$361,414	$242,148
Contract Management Superintendent	5/2	1	$314,804	$210,918
Project Engineer	8/6	1	$248,017	$166,171
Total			**$4,623,726**	**$3,097,897**

Table 141 Summary Mining Owner's Team Labor

18.7.7 Summary of Mining Operating Costs $/t mined

Based on the finalized mining cost model for this Technical Report Summary, a summary of the overall unit costs, for the summary cost areas is provided in Table 142. These numbers have been compared to recent benchmarks and industry experience in the region and are deemed as representative costs expected in the Australian mining industry for the proposed operation.

Costs were compiled in Australian dollars, Table 142 show figures in US dollars using an exchange rate AUD:USD of 0.67.

Cost Area	Unit Cost (AUD/t mined)	Unit Cost (USD/t mined)
Contractor – Monthly Overheads	$0.65	$0.44
Contractor - Drill and Blast	$0.89	$0.60
Contractor - Load and Haul	$2.30	$1.54
Contractor - Stockpile Rehandle	$0.21	$0.14
Contractor - Dewatering	$0.00 Included in the site wide costing	$0.00
Mining Contractor Sub-total	**$4.05**	**$2.71**
Grade Control Samples	$0.01	$0.01
Geotech - Wall Monitoring	$0.02	$0.01
Fuel - Diesel	$0.46	$0.31
Mine department – Labor	$0.17	$0.11
Mine department – LV's	$0.01	$0.01
Total	**$4.71**	**$3.16**

Table 142 Mining Operating Cost Summary

Operating costs reflect an assumed 365 days/year mining operation and mine schedule.

Further notes for consideration and part of this mining cost estimate:

- Mobilization and infrastructure elements are treated as pre-production capital. The infrastructure will be the property of Vista after completion and payment has been made.
- No separate Owner investment in mining fleet is required under this model.
- Ongoing performance reconciliation and KPI tracking will be implemented during operations to manage contractor performance risk.

18.8 Process Plant Operating Cost

Operating costs have been developed using the parameters specified in the process design criteria. Annual operating costs and costs per tonne milled (processed) have been developed and are summarized in Table 143. Operating costs have been estimated to an accuracy of +/- 10 - 15%. Table 143 show figures in US dollars using AUD:USD rate of 0.67 for operating costs estimation.

The costs cover the processing of ore from the ROM pad battery limit. This includes the sections covering crushing, reclaim, HPGR and ore sorting, milling, leaching and adsorption circuit, dewatering, cyanide detoxification, process plant site services (power, air, and water), and administration costs.

The operating cost estimate has been developed based on a process plant feed tonnage of 5,325,000 tpa.

The costs have been compiled from a variety of sources including:

- First principal estimates.
- Suppliers' budget quotations.
- GRES data base for similar operations.
- Metallurgical test work results.

The following Figure 108 summarizes the split in percentages of the process plant operating costs.

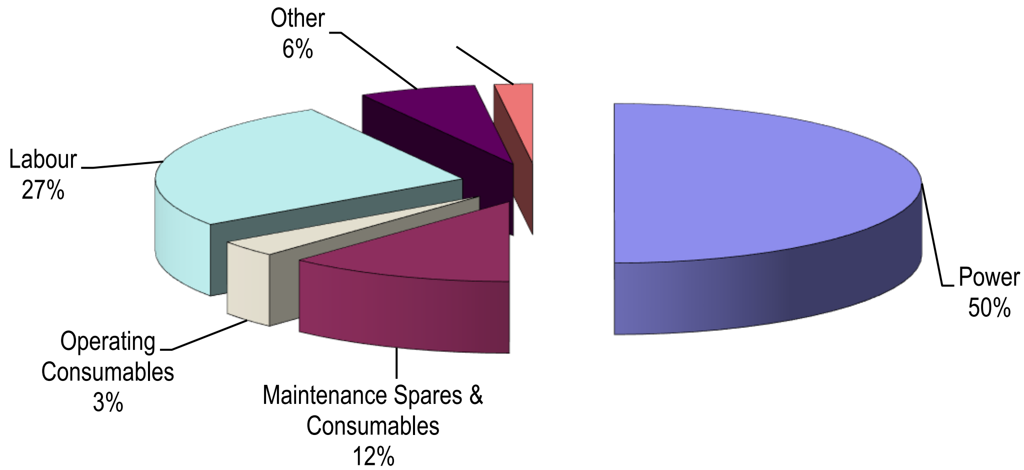


Figure 108 OPEX Item Breakdown, GRES 2025

BY COST CENTRE	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Power (USDM)	$15.93	$21.24	$21.24	$21.24	$21.24	$21.24	$21.24	$21.24	$21.24	$21.24	$21.24	$21.24
Maintenance Spares & Consumables (USDM)	$5.27	$5.97	$5.97	$5.97	$5.97	$5.97	$5.97	$5.97	$5.97	$5.97	$5.97	$5.97
Operating Consumables (USDM)	$28.40	$47.42	$47.54	$47.60	$47.66	$47.54	$47.54	$47.54	$47.66	$47.54	$47.54	$47.54
Labor (USDM)	$8.24	$10.98	$10.98	$10.98	$10.98	$10.98	$10.98	$10.98	$10.98	$10.98	$10.98	$10.98
Other (USDM)	$4.89	$3.40	$3.42	$3.34	$3.35	$3.45	$3.46	$3.46	$3.47	$3.48	$3.48	$3.48
USDM Total	**$62.73**	**$89.02**	**$89.16**	**$89.14**	**$89.20**	**$89.20**	**$89.20**	**$89.20**	**$89.32**	**$89.22**	**$89.22**	**$89.23**
Power (USD/t processed)	$5.43	$4.00	$3.99	$3.99	$3.98	$3.99	$3.99	$3.99	$3.98	$3.99	$3.99	$3.99
Maintenance Spares & Consumables (USD/t processed)	$1.80	$1.13	$1.12	$1.12	$1.12	$1.12	$1.12	$1.12	$1.12	$1.12	$1.12	$1.12
Operating Consumables (USD/t processed)	$9.68	$8.93	$8.93	$8.94	$8.92	$8.93	$8.93	$8.93	$8.92	$8.93	$8.93	$8.93
Labor (USD/t processed)	$2.81	$2.07	$2.06	$2.06	$2.06	$2.06	$2.06	$2.06	$2.06	$2.06	$2.06	$2.06
Other (USD/t processed)	$1.67	$0.64	$0.64	$0.63	$0.63	$0.65	$0.65	$0.65	$0.65	$0.65	$0.66	$0.66
USD/t processed Total	**$21.38**	**$16.77**	**$16.74**	**$16.74**	**$16.70**	**$16.75**	**$16.75**	**$16.75**	**$16.73**	**$16.76**	**$16.76**	**$16.76**

BY COST CENTRE	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24
Power (USDM)	$21.24	$21.24	$21.24	$21.24	$21.24	$21.24	$21.24	$21.24	$21.24	$21.24	$21.24	$21.24
Maintenance Spares & Consumables (USDM)	$5.97	$5.97	$5.97	$5.97	$5.97	$5.97	$5.97	$5.97	$5.97	$5.97	$5.97	$5.97
Operating Consumables (USDM)	$47.66	$47.54	$47.54	$47.54	$47.66	$47.54	$47.54	$47.54	$47.66	$47.54	$47.54	$47.54
Labor (USDM)	$10.98	$10.98	$10.98	$10.98	$10.98	$10.98	$10.98	$10.98	$10.98	$10.98	$10.98	$10.98
Other (USDM)	$3.49	$3.50	$3.50	$3.51	$3.52	$3.52	$3.52	$3.53	$3.54	$3.54	$3.54	$3.55
USDM Total	**$89.35**	**$89.24**	**$89.24**	**$89.25**	**$89.37**	**$89.26**	**$89.27**	**$89.28**	**$89.39**	**$89.28**	**$89.28**	**$89.29**
Power (USD/t processed)	$3.98	$3.99	$3.99	$3.99	$3.99	$3.99	$3.99	$3.99	$3.98	$3.99	$3.99	$3.99
Maintenance Spares & Consumables (USD/t processed)	$1.12	$1.12	$1.12	$1.12	$1.12	$1.12	$1.12	$1.12	$1.12	$1.12	$1.12	$1.12
Operating Consumables (USD/t processed)	$8.92	$8.93	$8.93	$8.93	$8.92	$8.93	$8.93	$8.93	$8.92	$8.93	$8.93	$8.93
Labor (USD/t processed)	$2.06	$2.06	$2.06	$2.06	$2.06	$2.06	$2.06	$2.06	$2.06	$2.06	$2.06	$2.06

BY COST CENTRE	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24
Other (USD/t processed)	$0.66	$0.66	$0.66	$0.66	$0.66	$0.66	$0.66	$0.66	$0.66	$0.66	$0.66	$0.66
USD/t processed Total	**$16.74**	**$16.76**	**$16.76**	**$16.76**	**$16.74**	**$16.76**	**$16.76**	**$16.76**	**$16.74**	**$16.76**	**$16.77**	**$16.77**

BY COST CENTRE	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30	Year 31	Year 32	Year 33	Total		
Power (USDM)	$21.24	$21.24	$21.24	$21.24	$21.24	$21.24	$20.66	$20.49	$20.49	$693.59		
Maintenance Spares & Consumables (USDM)	$5.97	$5.97	$5.97	$5.97	$5.97	$5.97	$5.97	$4.66	$4.66	$193.67		
Operating Consumables (USDM)	$47.66	$47.54	$47.54	$47.54	$47.66	$47.54	$50.43	$47.70	$35.28	$1,541.39		
Labor (USDM)	$10.98	$10.98	$10.98	$10.98	$10.98	$10.98	$10.98	$10.33	$10.33	$358.42		
Other (USDM)	$3.55	$3.56	$3.56	$3.56	$3.56	$3.57	$3.57	$3.52	$3.52	$116.93		
USDM Total	**$89.41**	**$89.30**	**$89.30**	**$89.30**	**$89.42**	**$89.31**	**$91.62**	**$86.70**	**$74.28**	**$2,904.00**		
Power (USD/t processed)	$3.98	$3.99	$3.99	$3.99	$3.98	$3.99	$3.88	$3.85	$5.28	$4.03		
Maintenance Spares & Consumables (USD/t processed)	$1.12	$1.12	$1.12	$1.12	$1.12	$1.12	$1.12	$0.88	$1.20	$1.13		
Operating Consumables (USD/t processed)	$8.92	$8.93	$8.93	$8.93	$8.92	$8.93	$9.47	$8.96	$9.09	$8.96		
Labor (USD/t processed)	$2.06	$2.06	$2.06	$2.06	$2.06	$2.06	$2.06	$1.94	$2.66	$2.08		
Other (USD/t processed)	$0.66	$0.67	$0.67	$0.67	$0.67	$0.67	$0.67	$0.66	$0.90	$0.68		
USD/t processed Total	**$16.74**	**$16.77**	**$16.77**	**$16.77**	**$16.75**	**$16.77**	**$17.21**	**$16.28**	**$19.14**	**$16.88**		

Table 143 Process Plant Operating Costs Yearly Breakdown

18.8.1 Qualifications and Exclusions

The operating costs presented have been calculated from first principles and budget quotations for supply of chemicals, materials and services. Process plant operating costs are considered to have an accuracy of +/- 10 - 15%. The following items have been excluded from the operating cost estimate:

- Royalties – included in financial analysis.
- All head office costs and corporate overheads.
- Exchange rate variations.
- Escalations.
- Project financing costs.
- Interest charges.
- Political Risk Insurance.
- Land compensation/landowners costs.
- Subsidies to the local community.
- Rehabilitation costs.
- Amortization and depreciation charges.

18.8.2 Basis of Estimate

The Project operating cost development methodology has been summarized in Table 144. Where there has been a deviation from the standard, the reasoning has been detailed.

All process plant operating costs are compiled in Australian dollars (AUD), when presented in US dollars (USD) an exchange rate of AUD:USD 0.67 for operating costs estimation was applied for reporting purposes. All operating costs exclude GST.

Item	Minimum Standard	Method Used	Comments
General			
Typical Accuracy Range	±5% to 10% for known operations; ±10% to 15% for new operations	±10% to 15%	
Contingency	Not applied	Not included	
A – Operating Cost Methodology			
A1 Staffing			
A1.1 Staffing Levels	Detailed manning schedule developed for Operations and Maintenance	Detailed manning schedule developed for Operations and Maintenance personnel.	Leave entitlements based on the Commonwealth's *Fair Work Act 2009*.

Item	Minimum Standard	Method Used	Comments
A1.2 Cost Rates	Detailed	Detailed and based on known rates (Hays) and on costs for superannuation, payroll tax, insurance, etc.	
A2 Consumables	Calculated	Detailed estimate	Based on reagent and equipment vendor submissions and recommended annual consumables or from first principles.
A3 Maintenance	Detailed estimate	See A1.1	
B – Operating Cost Basis			
B1 Labor			
B1.1 Labor Cost Rates	Detailed	Refer A1.2	
B1.2 Labor Burden Rates	Detailed	Refer A1.2	
B1.3 Labor Hours	Detailed	Not applicable	Refer to the OPEX estimate spreadsheet for details.
B1.4 Labor Overheads/ Management Costs	Detailed	Refer A1.2	On-costs included in labor rate build-up
B2 Utilities and Consumables			
B2.1 Power Costs	Detailed calculation based on budget quotation	Energy costs based on the preliminary tender average price of A$8.50 per GJ for gas feed in converted to 13.33 cents/kWh	Electrical consumption calculated from the electrical load list and typical load factors.
B2.2 Water Costs	Detailed calculation	Not calculated directly but indirectly through the equipment list, labor schedule, maintenance factors and power consumption.	Water supply from existing Raw Water Facility and retreated site water.
B2.3 Fuel Costs (Mobile Equipment)	Detailed calculation based on budget quotation	Fuel cost after rebate of A$0.81/L, with vehicle usage rates based on calculated run hours and typical fuel burn rates.	

Item	Minimum Standard	Method Used	Comments
B2.4 Consumables	Detailed calculation based on budget quotation	Detailed calculation based on budget quotations.	Usage based on rates from vendors or calculated from first principles. Wear liners have been included here rather than in maintenance costs.
B2.5 Supplies and Reagents	Detailed calculation based on budget quotation.	Detailed calculation based on budget quotations.	Usage based on metallurgical test work or modelling.
B3 Plant Maintenance			
B3.1 Maintenance Materials	Detailed estimate	A % of the capital cost has been applied and estimates for large contract works calculated (e.g. mill reline).	This approach is consistent with a Class 3 estimate (see Table 130)
B4 Transport and Logistics			
B4.1 Transport and Logistics	Detailed estimate	Transport for Materials/Consumables – Costs either included in factors used or rates obtained as a delivered to site rate.	
B5 – Other Operating Costs			
B5.1 Business Systems	Preliminary estimate	Allowance included for specific systems for operations, computer equipment and communications for personnel based on manning schedule.	Admin costs have been excluded from process plant OPEX and removed to G&A
B5.2 Training	Preliminary estimate	Based on 2% of annual labor cost.	Excluded from OPEX
B5.3 Auditing	Preliminary estimate	Allowance based on previous projects.	Excluded from OPEX
B5.4 Bank Charges	Preliminary estimate	Allowance based on previous projects.	Excluded from OPEX
B5.5 Communications	Preliminary estimate	Allowance based on previous projects.	Excluded from OPEX
B5.5 First Aid	Allowance per person on site	Allowance per person on site.	Excluded from OPEX

Item	Minimum Standard	Method Used	Comments
B5.6 Recruitment	Detailed estimate	Based on A$10,000 per position at 12% turnover for Year 1 and 8% thereafter.	Excluded from OPEX
B5.7 Consultants	Preliminary estimate	Allowance based on previous projects.	Included in OPEX
B5.8 Pre-operations	Preliminary estimate	Preliminary estimate	Included in Vista Owners costs
B5.9 Insurance	Preliminary estimate	Preliminary estimate as a 0.5% of the direct capital cost.	Process plant only
B5.10 Escalation	Preliminary estimate	Not Included	
B5.11 Foreign Exchange	Identify equipment and commodities to be imported, basis, values and likely currency.	No foreign exchange components, reagents and consumables identified as at the date of the estimate.	All reagents and consumables quoted in A$ (excluding GST).

Table 144 Operation Cost Methodology

18.8.3 Salaries and Labor

The salaries and labor costs have derived from the 2025 Hays Salary Guide and reflect current labor market conditions and site location.

The process plant operations team is assumed to predominantly be on FIFO rosters from other parts of Australia. This is based on the operational philosophy adopted by the Project. It is assumed recruitment and of personnel would likely be achieved with 50% from the Perth region and 50% from the Brisbane region.

The process plant operations personnel are detailed by position, roster type, shift and personnel headcount, refer Table 145.

Process Plant Personnel	Roster	Shift	Number
Plant Manager	5/2	Day	1
Senior Metallurgist	8/6	Day	1
Plant Metallurgist	8/6	Day	2
Production Superintendent	8/6	Day	2
Mill Trainer	4/3	Day	2
Shift Supervisor	14/7	Shift	3
Laboratory Supervisor	14/7	Day	2
Laboratory Technician	4/3	Day	4
Control Room Technician	14/7	Shift	3

Process Plant Personnel	Roster	Shift	Number
Gold Room Supervisor	8/6	Day	2
Gold Circuit Technician	8/6	Day	4
Process Technician	14/7	Shift	18
Maintenance Superintendent	5/2	Day	1
Maintenance Planner	8/6	Day	2
Mechanical Supervisor	8/6	Day	2
Electrical Supervisor	8/6	Day	2
Mechanical Trades	14/7	Shift	15
Electrical Trades	14/7	Shift	9
Trades Assistant	14/7	Shift	3
Total			**78**

Table 145 Detailed Process Plant Labor Headcount

The labor rates are annualized and inclusive of the following on-costs:

- Salary.
- Payroll tax (6.08%).
- Insurance, workers compensation (1.51%).
- Annual leave and sick leave (5%).
- Long service leave provision (2%).
- Superannuation (12%).

An on-cost of 27% has been applied to the salaries as shown above to allow for insurance, leave, superannuation and payroll taxes.

Costs were compiled in Australian dollars, Table 146 show figures in US dollars using an exchange rate AUD:USD of 0.67 for an annual breakdown of labor requirements.

Departments	Personnel	Cost AUD/a	Cost USD/year
Laboratory (onsite)	6	$964,478	$646,200
Process Plant Management	11	$2,697,127	$1,807,075
Process Plant Operations	27	$4,987,749	$3,341,792
Plant Maintenance	34	$7,785,086	$5,216,008
Administration (inc. Procurement, HSE)	0	$0	$0
TOTAL	**78**	**$16,394,440**	**$10,984,275**

Table 146 Summary Process Plant Labor

18.8.4 *Fuel*

A diesel price of AUD0.81/L (USD0.54/L) delivered inclusive of the fuel tax rebate has been applied to the operating cost estimate. Diesel will be required for main mining equipment, supporting equipment, maintenance vehicles and light vehicles.

Gas pricing in the Northern Territory is affected by many factors (micro and macro), as such, any projection of future pricing should consider the local market history and dynamics. Therefore, much of the information used to support Vista's gas price analysis is based on studies and reports published by groups that have expertise in gas price forecasting. Also, the Power Contractor provided a proposal for the cost of power specifically for the Project.

Base: Fixed Term 2025 Darwin Terminal Gate Price (TGP) with discount at 90.3% less GST refund and fuel tax credit (FTC).

Calculation: Net Disel Price $_{(\text{at site})}$ = (\$1.605$_{\text{TGP}}$ x 90.3%$_{\text{discount}}$)-(\$0.132$_{\text{GST}}$)-(\$0.508$_{\text{FTC}}$) = \$0.810 at site

18.8.5 *Mobile Vehicles and Equipment*

The allocation of mobile equipment to the processing and maintenance groups includes:

- Light vehicles (x9).
- 10 tonne all terrain truck with Hiab crane (x1).
- 3 tonne long tray truck (x1).
- Integrated tool carrier (x1).
- Forklift (3 t capacity) (x2).
- Skid steer loader (x2).
- Tele-handler 3 tonne (x1).
- Container reach stacker – 30 t (x1).
- 25 t mobile crane (Franna or equivalent) (x2).
- 50 t mobile crane (x1).
- Articulated boom lift (x1).
- Elevated work platform (boom lift) (x2).

For the above listed equipment, the annual run hours have been estimated based on an average usage per day.

18.8.6 *Power*

Power will be generated by a new gas turbine power plant located at the Project.

Power cost has been provided by Power Contractor based on a Build Own Operate (BOO) proposal for a thermal power station. Pricing has been built on the thermal only option with a total generated cost including capital, fuel and maintenance of AUD13.33 cents/kWh (USD8.93 cents/kWh).

Fuel costs are based on the preliminary tender average price of AUD8.50/GJ (USD5.70/GJ) for gas feed.

The thermal only option can be expanded and built with enabling infrastructure installed from day one, allowing for easy future integration of Renewable Energy should Vista prefer to take a more strategic/staged approach in introducing RE across their assets in the Northern Territory. The 19 x Primary Gas generators selected are Jenbacher J620 J-01 (Total Installed 64 M). The 3 x Back up Diesel generators are Cummins KTA50G3.

The power summary for the process plant and administration is detailed in Table 147.

Area	Power		Annual Usage
	Installed kW	Consumed kW	kWh
Processing			
Crushing and Screening	2,488	1,698	8,439,214
Reclaim/Ore Sorting	980	707	5,625,949
HPGR/Grinding & Classification	32,122	26,206	201,430,821
Carbon in Leach	2,300	1,646	10.325,422
Goldroom/Elution	346	277	1,033,442
Tailings Disposal	1,382	1,036	4,837,535
Reagents Mixing	166	116	385,019
Water & Air Services	2,379	1,665	4,324,234
Workshop	70	49	394,901
Laboratory	61	48	427,488
Administration	101	80	651,828
Totals	**42,398**	**33,533**	**237,875,853**

Table 147 Power Consumption

18.8.7 Reagents and Consumables

Reagents and consumables include the following cost elements:

- Crusher wear liners.
- Grinding mills wear liners.
- Grinding media for the grinding mills.
- All reagents used in the process.
- Fuel for mobile equipment assigned to the processing or maintenance groups.
- Lubricants, operating tools and equipment, general and operator supplies.

Reagent addition rates were derived from laboratory test work. Reagent consumption rates have been calculated on a per tonne of mill feed from the steady state mass balance.

18.8.8 Maintenance

Maintenance costs include the cost for spare parts and maintenance materials to maintain the Process Plant. The maintenance cost has been applied as a percentage of the plant area capital cost. The overall percentage factors categorized by plant area has been summarized in Table 148 for the Process Plant.

Area	Base Cost AUD	% Maintenance Cost	Total Maintenance Cost AUD	Total Maintenance Cost USD
Crushing and Screening	$29,723,743	4.5%	$1,337,568	$896,171
Reclaim/Ore Sorting	$12,098,106	3.5%	$423,434	$283,701
HPGR/Grinding and Classification	$102,817,617	5.5%	$5,654,969	$3,788,829
Carbon in Leach	$19,615,072	2.1%	$411,917	$275,984
Goldroom/Elution	$4,292,299	3.0%	$134,769	$90,295
Tailings Disposal	$3,307,917	4.0%	$132,317	$88,652
Reagent Mixing	$2,740,386	3.2%	$87,692	$58,754
Water & Air Services	$1,580,162	4.2%	$63,206	$42,348
Workshop	$300,000	3.5%	$10,500	$7,035
Laboratory	$500,000	3.5%	$17,500	$11,725
Administration	$3,427,999	1.1%	$37,708	$25,264
Total			**$8,311,580**	**$5,568,759**

Table 148 Maintenance Costs

Maintenance costs include for contract re-lining of the crushers, grinding mills and plant shutdowns. However, the liner costs are accounted for in reagents and consumables in section.

The direct labor cost for maintenance personnel has been included in the labor cost category.

An allowance has been made for maintenance costs for electrical distribution at 1.8% of base cost of AUD43,811,177 for AUD788,601 per year in maintenance costs, in US dollars using an exchange rate AUD:USD of 0.67 this equates to USD528,363 in maintenance costs.

In total, maintenance materials are approximately 4.3% of the Total Equipment Cost of the process plant.

18.8.9 Reagents and Consumables

All process plant reagent and consumables are a variable cost component except fuel costs for mobile equipment. The unit costs in Table 149 are inclusive of freight to the Project site but exclusive of GST.

Item	Consumption		Cost		
	Rate	Basis	(AUD)	(USD)	Basis
Consumables					
Primary Crusher Liners	1.5 set per year	Allowance	$239,338 per set	$160,356 per set	Vendor data
Secondary Crusher Liners	2.5 sets per year	Allowance	$344,561 per set	$230,856 per set	Vendor data
Product Screen Panels	2.5 set per year	Allowance	$200,000 per set	$134,000 per set	Vendor quotation
HPGR Liners	1.5 set per year	Allowance	$2,370,533 per set	$1,588,257 per set	Vendor quotation
Ball Mill Liners	1 set per year	Allowance	$543,000 per set	$363,810 per set	Vendor estimate
Verti Mill Liners	1 set per year / per mill	Allowance	$519,480 per set	$348,052 per set	Vendor estimate
Ball Mill Balls	1.23 kg/t ore	Calculation	$2,200 per tonne	$1,474 per tonne	Vendor quotation
Regrind Mill Media (Ceramic)	0.81 kg/t ore	Allowance	$2,240 per tonne	$1,501 per tonne	Vendor quotation
Reagents					
Quicklime	2.8 kg/t ore	Met testing	$370 per tonne	$248 per tonne	Vendor quotation
Sodium Cyanide	1.05 kg/t ore	Met testing	$4,620 per tonne	$3,095 per tonne	Vendor quotation
Carbon	30 g/t ore	Met testing	$6,450per tonne	$4,322 per tonne	Vendor quotation
Sodium Hydroxide	110 g/t ore	Allowance	$1,300 per tonne	$871 per tonne	Vendor quotation
Hydrochloric Acid	50 g/t ore	Met testing	$1,100 per tonne	$737 per tonne	Vendor quotation
Antiscalent	10 g/t ore	Allowance	$2,500 per tonne	$1,675 per tonne	Vendor quotation
Oxygen	30g/t ore	Testing	$620 per tonne	$415 per tonne	Allowance

Table 149 Reagent and Consumable Costs

18.8.10 Laboratory

Laboratory costs include the costs for assaying of various process streams and mining grade control through the on-site laboratory.

The number of process plant assays has been calculated based on selected process streams and required frequency to monitor the process plant operation, undertake metallurgical accounting and confirm final product specifications.

The Project's laboratory will be onsite and allowance has been included in the estimate at AUD316,478 (USD212,040) per year for the assaying components. Labor costs AUD964,478 (USD646,200) are on top of this and accounted for within Labor.

18.8.11 Administration

Administration costs have only covered the processing part of the plant administration, consultants, labor and administration building area maintenance. Other Administration costs have been removed to General and Administrative costs.

18.8.12 Freight

The freight cost for reagents and consumables have been applied either as ex works (EXW) or free carrier (FCA) as advised by the supplier.

18.8.13 Process Plant Sustaining and Deferred Capital

Sustaining and deferred capital will cover the funding required over the life of the Project to replace items within the process plant that have reached their maintainable and useful life or planned expenditure to modify the plant as necessary to sustain operations at the rated capacity.

Sustaining and deferred capital has been estimated as a percentage of the direct capital supply cost based on typical industry experience or from first principles. The allocations include:

- Mechanical Equipment at 3% excluding those mechanical items that have capital spares purchased.
- Electrical Equipment at 1%.
- Piping Supply at 3%.
- Building Supply at 1%.

The sustaining capital allowance excludes other areas outside the scope of the GRES supplied capital and operating costs.

The annual sustaining capital for the process plant starts in year 5 and has been estimated to be AUD2.8 M/a (USD1.88 M/a), which is roughly 1% of the total supply costs. Including the capital spares already purchased and combined with the sustaining capital for the process plant this is 5% of the total supply costs.

18.8.14 Pre-production Process Plant Costs

Pre-production process plant costs for Vista labor and vehicles have been determined to account for operation readiness and include a ramp up of personnel to full operational personnel 3 months prior to plant hand over. These costs are all included in the Pre-Production Capital costs.

18.9 Non-Process Infrastructure and Facilities Operating Costs

The following details the non-mining and process plant operating costs.

18.9.1 Tailings Storage Facilities Operating Costs

The sustaining capital costs includes all the tailings wall lifts and all other TSF operating costs are captured within the process plant operating costs.

18.9.2 Mine Dewatering Operating Costs

Mine dewatering will be performed by the mining contractor. Based on the inflows to the pit, the majority of which is precipitation in the wet season, the annual operating costs of dewatering range from USD0.6M to USD1.1M.

18.9.3 Water Treatment Plant Operating Costs

WTP operating costs are based on the results of the SWWB. The annual operating costs for the WTP are estimated to be USD3.5M. These costs will vary based on plant operation and climate and are therefore considered an estimate and could vary during the life of the Project.

18.9.4 Raw Water Dam and Distribution Operating Costs

Raw water dam capital costs are based on direct vendor quotes or Tetra Tech in-house estimates, initial capital costs are estimated at USD1.32M and with sustaining capital improvements for USD1.79M.

18.9.5 Reclamation and Closure Operating Costs

Reclamation and Closure Operating costs are included in the capital cost estimate.

18.10 General and Administration Operating Costs

Table 150 and Table 151 presents a breakdown of the general and administrative (G&A) cost components for the operation, expressed on both a unit cost per tonne processed and a total cost basis in Australian dollars and US dollars using an exchange rate of AUD:USD 0.67. These G&A costs reflect the overhead expenditures necessary to support the safe, efficient, and regulatory-compliant management of the site. They encompass personnel and associated travel and accommodation, administrative and operational support services, light vehicle operations, NT environmental levy, and insurance coverage.

General and Administrative Costs	AUD/t Processed	Total (AUD 000)
Labor G&A (Includes transport and accommodation)	$1.88	$323,525
General Expenses	$0.32	$54,315
Light Vehicles (Fire and Ambulance)	$0.07	$11,836
Environmental Levy	$0.26	$45,500
Insurances	$0.54	$93,284
Total	**$3.07**	**$435,177**

Table 150 General and Administrative Costs in AUD

General and Administrative	USD/t Processed	Total (USD 000)
Labor G&A (Includes transport and accommodation)	$1.26	$216,762
General Expenses	$0.21	$36,391
Light Vehicles (Fire and Ambulance)	$0.05	$7,930
Environmental Levy	$0.18	$30,485
Insurances	$0.36	$62,500
Total	**$2.06**	**$354,068**

Table 151 General and Administrative Costs in USD

18.11 Royalties

Royalties assumed for the Project's economics are addressed in section 19.3.3. Table 152 and Table 153 present royalties paid, expressed on both a unit cost per tonne processed and a total cost for the life of the Project basis in Australian dollars and US dollars using an exchange rate of AUD:USD 0.67

Royalties for Life of Project	AUD/t Processed	Total (AUD000s)
JAAC Royalty	$2.96	$509,761
Wheaton Royalty	$1.03	$176,776
Total	**$3.99**	**$686,537**

Table 152 Royalties Paid in AUD

Royalties for Life of Project	USD/t Processed	Total (USD000s)
JAAC Royalty	$1.99	$341,540
Wheaton Royalty	$0.69	$118,440
Total	**$2.67**	**$459,980**

Table 153 Royalties Paid in USD

18.12 Refining

Refining costs for the Project's economics are addressed in section 16.2. Table 154 and Table 155 presents Royalties paid, expressed on both a unit cost per tonne processed and a total cost basis in Australian dollars and US dollars using an exchange rate of AUD:USD 0.67.

Refining costs averages USD0.13/t processed or USD5.00/oz

Description of Cost	AUD/t Processed	Total (AUD000s)
Refining Costs	$0.19	$33,483

Table 154 Refining Costs in AUD

Description of Cost	USD/t Processed	Total (USD000s)
Refining Costs	$0.13	$22,769

Table 155 Refining Costs in USD

18.13 Overall Labor Requirements

From Month 1 of the construction period an allowance for labor has been included in the capital costs. Figure 109 shows the ramp up and designation of labor allowance for the various areas.

The Peak workforce during construction is estimated to be around the 350 – 370 personnel at around Months 21 to 23 of the construction period. The Vista owners' team will ramp up and be close to full operation at Month 27 before handover of the major project construction areas, mainly the process plant.

During Steady State Operations the labor requirements are presented in Table 156.

Average Life of Mine Personnel Requirements	Number of Employees per Shift	Total Employees Required
Sub Total - Management	3	3
Sub Total - Administrative	13	20
Sub Total - Accounts and Payroll	5	6
Sub Total - IT	3	3
Sub Total - Procurement and Supply	2	2
Sub Total - Site OHS and Environmental	7	11
General and Admin - Vista Employees	**33**	**45**
Sub Total - Mining	15	23
Sub Total - Process Met	7	10
Sub Total - Process Operations	10	30
Sub Total - Process Maintenance	13	34
Sub Total - Laboratory	3	6
Operations Vista Employees	**48**	**103**
Sub Total - Mine Operations and Mine Maintenance	77	171
Sub Total - TSF Construction and Small Civil works	9	9
Subtotal Project Services (WTP & POWER)	6	16
Sub Total - Site Camp and Logistics	20	42
Operation - Contractor Services	**112**	**238**
Total	**193**	**386**

Table 156 Total Personnel for Steady State Project Operations

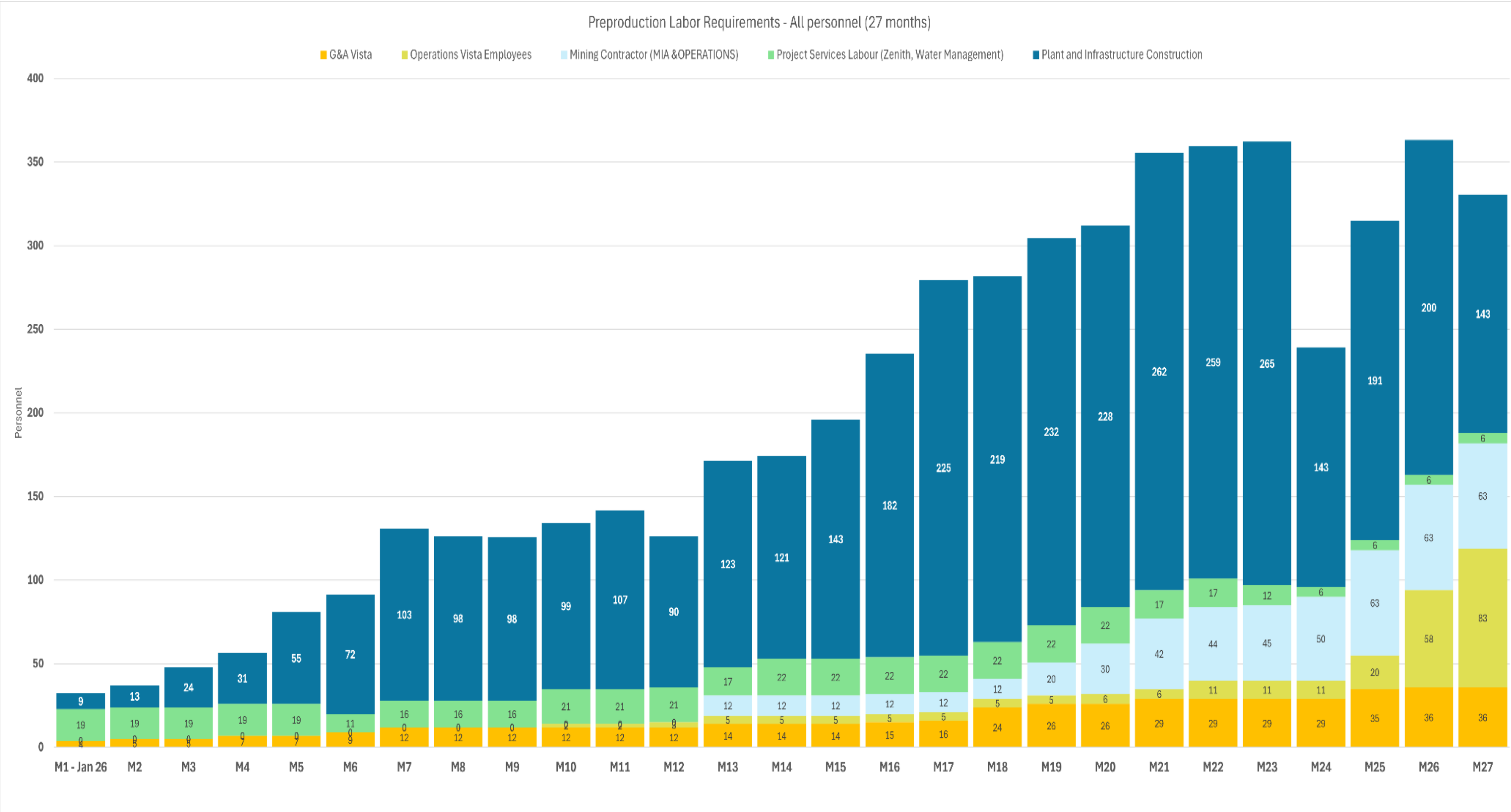


Figure 109 Labor Histogram During Construction, GRES 2025

19. ECONOMIC ANALYSIS

19.1 Introduction

Project economics for the 15 ktpd operation are based on inputs developed by GRES, Tetra Tech, Mining Plus, Tierra Group and Vista. Economic results presented in the Technical Report suggest the following conclusions, assuming a 100% equity project, and a gold price of USD2,500/oz.

▪ Mine Life	30 years
▪ Production Life	33 years
▪ Pre-Tax NPV5%	USD1,736 million, IRR: 37.3%
▪ After-tax NPV5%	USD1,060 million, IRR: 27.8%
▪ Payback (After-tax)	2.7 years
▪ JAAC Royalty Paid	USD342 million
▪ Wheaton Royalty Paid	USD118 million
▪ Northern Territory Taxes Paid (Royalty)	USD398 million
▪ Australian Commonwealth Corporate Taxes Paid	USD1,083 million
▪ Cash costs (including Private Royalties)	USD1,438/oz-Au

Project cost estimates and economics results are presented on an annual basis. Based upon design criteria presented in this Technical Report Summary, the level of accuracy of the estimate is considered ±10-15%.

Costs and economic results are presented in Q2 2025 U.S. dollars unless otherwise stated. No escalation has been applied to capital or operating costs. The 5% discount rate used is a goldmining industry standard. in North America generally used for comparability purposes among projects; it is not intended to fully reflect consideration of cost of capital, risk adjustments, or other factors.

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.

19.2 Principal and General Economic Assumptions

19.2.1 Principal Assumptions

Parameters used in the analysis are shown in Table 157. These parameters are based upon current market conditions, vendor and contractor quotes and design criteria developed by Vista and their consultants, and benchmarks against similar existing projects.

Taxes and Royalties assumed for the Project's economics are addressed in Section 22.3.3 including:

- Northern Territory Royalties and Taxation,

- JAAC Royalty,
- Wheaton Royalty,
- Australian Commonwealth corporate tax.

Principal Economic Assumptions	Unit	Parameters
Construction Period	Months	24
Commissioning & Ramp-Up	Months	3 & 12
Mine Life	Years	30
Project Life (includes re-processing of existing heap leach pad ore)	Years	33
Closure Period	Years	8
Operating Days	Days/Year	355
Gold Price	USD	$2,500
Exchange Rate Construction and Commissioning Period	AUD:USD	0.66:1
Exchange Rate Project Life	AUD:USD	0.67:1

Table 157 Principal Economic Assumptions

A diesel price of AUD0.81/L (USD0.54/L) delivered inclusive of the fuel tax rebate has been applied to the operating cost estimate. Diesel will be required for main mining equipment, supporting equipment, maintenance vehicles and light vehicles.

Power will be generated by a gas turbine power plant located at the Project site. Power costs have been provided by Power Contractor based on a Build Own Operate (BOO) proposal for a thermal power station. Pricing has been built on the thermal only option with a generated cost including maintenance of AUD13.33 cents/kWh, USD8.93 cents/kWh.

19.2.2 General Economic Assumptions

- The financial analysis was performed on Proven and Probable Mineral Reserves. All other mined material was classified as waste and assigned no economic value.
- The financial analysis was performed on Proven and Probable Mineral Reserves.
- The Project is designed at a production rate of 15 ktpd. Fresh ore production will originate from the open pit mine and will be treated using conventional CIL technology. Once ore is exhausted from the pit, the Mineral Reserves in the existing heap leach pad will then be processed.
- The Project's NPV was determined on a pre-tax and after-tax basis.
- Annual cash flows used for NPV calculations are assumed to be realized at year-end.
- An exchange rate of USD0.66:AUD1.00 for the construction period and USD0.67:AUD1.00 for the operating life of mine years and closure was assumed.
- All costs and sales do not consider inflation or escalation factors.
- All gold sales are assumed to occur in the same period as produced.

- Details of capital and operating costs are provided in Section 18 of this Technical Report Summary.
- Cash flows shown include payment of royalties.
- Progressive and final closure costs are included in the period incurred.
- The financial analysis includes working capital adjustments to provide for difference in the timing of sales and incurrence of obligations and the time of cash received and expended.
- Australian GST has been excluded.
- After-tax results and royalty payments were estimated by Vista.
- After-tax figures include Australian Commonwealth corporate tax rate of 30% applied to taxable mining income.

19.3 Economic Analysis

19.3.1 Capital Expenditures

Capital costs have been developed from first principles with quotes for all major equipment components. A turnkey engineering, procurement and construction model has been used as the basis for project construction. The Technical Report Summary contemplates a 27-month period for engineering, construction and commissioning. Contract mining and a third-party gas-fired generating plant are included in the costs. Capital costs include a permanent camp facility near the mine site with housing, dining, and recreation facilities for approximately 90% of the initial workforce. The remaining workforce is assumed to be from local communities.

The closure plan includes re-processing 13 Mt of heap leach material from previous operations and then placing that material in the tailings storage facility, the revenues from the heap leach material has been treated as self-funding reclamation. The heap leach pad material is included in Mineral Reserves.

Summaries of capital costs are shown in Table 158.

Capital Expenditure Item	Initial Capital Cost (USDM)	Sustaining Capital Years 1-30 Cost (USDM)	Heap Leach, Reclamation and Closure Costs (USDM)[7]
Mining	$22.03	$28.01	$4.71
Process Plant	$144.80	$46.03	N/A
Project Infrastructure	$ 83.68	$141.23	$4.41
Site Establishment and Facilities	$ 36.57	8.12	N/A
Management, Engineering, EPC Services	$ 65.22	$8.24	$ 0.36
Preproduction Costs and Capital Spares	$ 47.18	N/A	N/A

[7] Excludes cash flows from reprocessing of HLP ore and includes sustaining costs incurred during reprocessing of HLP ore

Capital Expenditure Item	Initial Capital Cost (USDM)	Sustaining Capital Years 1-30 Cost (USDM)	Heap Leach, Reclamation and Closure Costs (USDM)[7]
Reclamation	N/A	$109.57	N/A
Sub-total: Capital Expenditures	**$399.48**	**$341.20**	**$9.48**
Heap Leach, Reclamation and Closure	N/A	NA	$50.66
Combined Engineering Growth and Contingency (6-10%)	$25.03	$35.86	$5.76
Total Capital Costs	**$424.51**	**$377.06**	**$65.90**

Table 158 Capital Expenditures

19.3.2 Operating Expenditures

Mining costs have been provided by a prominent Australian contract miner. Power costs are based on a proposal from one of Australia's leading mine site contract power generators.

Processing and G&A costs have been developed from first principles with major consumable supply component quotes and competitive Australian labor rates. The operating costs account for around 90% of the initial workforce being hired on a fly-in-fly-out basis. This percentage decreases to 80% once the Project is established, the local workforce has been trained and attracted. Housing is provided for the fly-in-fly-out workforce in a permanent 250-bed camp facility near the mine site

Summaries of operating costs, excluding royalties, are shown in Table 159.

Operating Cost Item	Units	Years 1- 15	LOM Yr 1-30
Mining Costs	USD/t processed	$18.49	$16.55
Processing Costs[8]	USD/t processed	$17.70	$17.62
G&A Costs	USD/t processed	$2.09	$2.09
JAAC Royalty	USD/t processed	$2.22	$2.08
Wheaton Royalty	USD/t processed	$0.84	$0.73
Refining Costs	USD/t processed	$0.15	$0.14
Total Cash Costs	**USD/t processed**	**$41.49**	**$39.20**

Table 159 Operating Expenditures

[8] Include water management costs of approximately USD0.78/t processed.

19.3.3 Taxes and Royalties

19.3.3.1 Northern Territory Royalties Taxation

Effective July 1, 2024, the Northern Territory Mineral Royalties Act (MRA) 2024 replaced the Mineral Royalties Act 1982 for new mines. MRA 2024 imposes an ad valorem royalty scheme, which is calculated as:

Royalty = RR x (V – SC), where:

RR – is the royalty rate of a mineral

V – is the value of the mineral extracted from a mining operation

SC – is the amount allowed to be deducted for shipping costs within the Northern Territory

The applicable royalty rate to gold doré is 3.5%. For purposes of the economic analysis the 3.5% royalty rate was applied to total gold sales and is noted as "Northern Territory Taxes Paid"; no deduction for intra-territory shipping costs was assumed

19.3.3.2 Other Royalties

For rent of the surface rights from the current mining leases, 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 (USD33,000). In addition to the aforementioned 1% royalty to the JAAC, Vista and the JAAC agreed to replace a 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. Based on the gold price and foreign exchange rate assumptions used for the economic analysis, the sliding scale royalty is 2.0%.

The Wheaton royalty is at a rate of 1% of gross revenue if the defined completion objectives are achieved by April 1, 2028. Beginning April 1, 2028, if the completion objectives are not achieved, the rate increases annually at a rate of up to 0.13% to a maximum rate of 2%. Any annual increases beginning April 1, 2028 shall be reduced on a pro rata basis to the extent that operations have been initiated but not yet achieved a completion test at an average daily processing rate of 15,000 tonnes per day. For purposes of the economic analysis, completion is assumed to occur prior to April 1, 2029; therefore, the rate used was 1.13%. As provided by the terms of the Wheaton royalty, this rate was then reduced by one-third after production of 3.47 million gold ounces.

There is also a royalty of 5.0% 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 Reserves at the Mt Todd Project.

Summaries of royalty costs are shown in Table 160.

Royalties for Life of Project	Unit Costs USD/t processed	Costs per Payable Gold Ounce USD/oz-Au
JAAC Royalty	$1.99	$75
Wheaton Royalty	$0.69	$26

Table 160 Royalties

19.3.4 Australian Commonwealth Corporate Tax

The applicable corporate income tax rate in Australia is 30% of taxable income.

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.

Generally, 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. Vista estimated losses carried forward from prior years and available to offset assessable income to be USD91.69 million.

19.3.5 Cash Costs and All-In Sustaining Costs

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.

The average total cash cost over the life of the mine is estimated at USD1,413 per ounce of payable gold and All-in sustaining costs of USD1,499 per ounce of payable gold. Total cash cost and All-In Sustaining costs for the Project are summarized in Table 161.

Period	Cash Costs (USD/oz-Au)	Sustaining Costs (USD/oz-Au)	AISC USD/oz-Au
Years 1-15	$1,399	$49	$1,449
LOM (Years 1-30)	$1,413	$86	$1,499

Table 161 Cash Costs and All-In Sustaining Costs (USD/oz)

19.3.6 Working Capital

Working capital will vary over the mine life based on revenue, operating costs, and capital costs. Gold sales assume a customary advance payment upon shipment arrangement to be in place; therefore, 10% of the monthly value of gold produced is assumed to be in finished gold inventory and settled in the following month. The turnover rate is approximately 30 days for third-party accounts payable. Internal labor costs are assumed paid in the month incurred. All working capital is assumed to be recaptured by the end of the Project life and the closing value of the accounts is zero. First fills of consumables and other operating supplies are included in project capital. The working capital was calculated by Vista.

19.3.7 Cashflow Profile

A summary of the annual cash flows and the details of the cash flow model for the FS are presented in Table 162 and 161, including total for the life of the Project.

19.3.8 Annual Cash Flow for Years -2 to 15

Cash Flow Summary	Units	Totals	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Gold Production	**koz**	**4,554**	**-**	**-**	**100**	**181**	**181**	**182**	**151**	**150**	**150**	**150**	**150**	**151**	**150**	**150**	**152**	**150**	**151**
Gold Price	USD	$ 2,500	$ 2,500	$ 2,500	$ 2,500	$ 2,500	$ 2,500	$ 2,500	$ 2,500	$ 2,500	$ 2,500	$ 2,500	$ 2,500	$ 2,500	$ 2,500	$ 2,500	$ 2,500	$ 2,500	$ 2,500
Gold Sales	**USDM**	**$ 11,385**	**$ -**	**$ -**	**$ 250**	**$ 453**	**$ 451**	**$ 455**	**$ 377**	**$ 375**	**$ 375**	**$ 375**	**$ 376**	**$ 378**	**$ 375**	**$ 375**	**$ 380**	**$ 374**	**$ 378**
Cash Operating Costs																			
Mining	USDM	$ (2,642)	$ -	$ -	$ (66)	$ (89)	$ (93)	$ (93)	$ (92)	$ (96)	$ (103)	$ (104)	$ (102)	$ (100)	$ (99)	$ (98)	$ (98)	$ (99)	$ (101)
Processing	USDM	$ (3,071)	$ -	$ -	$ (66)	$ (93)	$ (93)	$ (93)	$ (93)	$ (93)	$ (93)	$ (93)	$ (94)	$ (93)	$ (93)	$ (93)	$ (94)	$ (93)	$ (93)
G&A	USDM	$ (354)	$ -	$ -	$ (8)	$ (11)	$ (11)	$ (11)	$ (11)	$ (11)	$ (11)	$ (11)	$ (11)	$ (11)	$ (11)	$ (11)	$ (11)	$ (11)	$ (11)
JAAC Royalty	USDM	$ (342)	$ -	$ -	$ (7)	$ (14)	$ (14)	$ (14)	$ (11)	$ (11)	$ (11)	$ (11)	$ (11)	$ (11)	$ (11)	$ (11)	$ (11)	$ (11)	$ (11)
Wheaton Royalty	USDM	$ (118)	$ -	$ -	$ (3)	$ (5)	$ (5)	$ (5)	$ (4)	$ (4)	$ (4)	$ (4)	$ (4)	$ (4)	$ (4)	$ (4)	$ (4)	$ (4)	$ (4)
Refining	USDM	$ (23)	$ -	$ -	$ (0)	$ (1)	$ (1)	$ (1)	$ (1)	$ (1)	$ (1)	$ (1)	$ (1)	$ (1)	$ (1)	$ (1)	$ (1)	$ (1)	$ (1)
Sub-total: Cash Operating Costs	**USDM**	**$ (6,550)**	**$ -**	**$ -**	**$ (151)**	**$ (212)**	**$ (216)**	**$ (217)**	**$ (213)**	**$ (217)**	**$ (223)**	**$ (224)**	**$ (223)**	**$ (221)**	**$ (220)**	**$ (219)**	**$ (219)**	**$ (219)**	**$ (222)**
Cash Operating Margin	**USDM**	**$ 4,835**	**$ -**	**$ -**	**$ 98**	**$ 241**	**$ 235**	**$ 238**	**$ 164**	**$ 158**	**$ 151**	**$ 150**	**$ 153**	**$ 157**	**$ 155**	**$ 157**	**$ 160**	**$ 155**	**$ 157**
Capital Costs																			
Initial Capex	USDM	$ (425)	$ (157)	$ (228)	$ (40)														
Sustaining Capex	USDM	$ (266)			$ (16)	$ (2)	$ (6)	$ (1)	$ (2)	$ (14)	$ (2)	$ (2)	$ (5)	$ (3)	$ (11)	$ (6)	$ (2)	$ (8)	$ (3)
Reclamation & Closure	USDM	$ (176)	$ -	$ (1)	$ (1)	$ (1)	$ (3)	$ (1)	$ (4)	$ (1)	$ (3)	$ (1)	$ (3)	$ (1)	$ (3)	$ (1)	$ (3)	$ (1)	$ (3)
Salvage	USDM	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sub-total: Capital Costs	**USDM**	**$ (867)**	**$ (157)**	**$ (229)**	**$ (57)**	**$ (2)**	**$ (9)**	**$ (2)**	**$ (6)**	**$ (15)**	**$ (5)**	**$ (3)**	**$ (9)**	**$ (4)**	**$ (14)**	**$ (7)**	**$ (5)**	**$ (9)**	**$ (6)**
Working Capital Changes	**USDM**	**$ 0**	**$ 9**	**$ 4**	**$ 2**	**$ 7**	**$ 5**	**$ 0**	**$ (5)**	**$ 1**	**$ (1)**	**$ 0**	**$ 0**	**$ 0**	**$ 0**	**$ (0)**	**$ 0**	**$ 0**	**$ 0**
Pre-Tax Cash Flow	**USDM**	**$ 3,968**	**$ (149)**	**$ (225)**	**$ 44**	**$ 245**	**$ 231**	**$ 237**	**$ 153**	**$ 143**	**$ 145**	**$ 147**	**$ 145**	**$ 153**	**$ 141**	**$ 149**	**$ 155**	**$ 146**	**$ 151**
Northern Territory Royalty	USDM	$ (398)	$ -	$ -	$ (9)	$ (16)	$ (16)	$ (16)	$ (13)	$ (13)	$ (13)	$ (13)	$ (13)	$ (13)	$ (13)	$ (13)	$ (13)	$ (13)	$ (13)
Income Taxes	USDM	$ (1,083)	$ -	$ -	$ -	$ (36)	$ (54)	$ (57)	$ (36)	$ (35)	$ (33)	$ (34)	$ (35)	$ (37)	$ (36)	$ (38)	$ (39)	$ (38)	$ (38)
After-Tax Cash Flow	USDM	$ 2,486	$ (149)	$ (225)	$ 35	$ 194	$ 161	$ 164	$ 104	$ 95	$ 99	$ 100	$ 97	$ 103	$ 91	$ 98	$ 103	$ 95	$ 100
After-Tax Cumulative Cash Flow	**USDM**		**$ (149)**	**$ (374)**	**$ (339)**	**$ (145)**	**$ 16**	**$ 180**	**$ 284**	**$ 379**	**$ 478**	**$ 578**	**$ 675**	**$ 778**	**$ 869**	**$ 967**	**$ 1,070**	**$ 1,165**	**$ 1,265**
Payback (yrs)	**Years**	**2.65**																	
Pre-Tax NPV5%	**USDM**	**$ 1,736**																	
Pre-Tax IRR	**%**	**37.3%**																	
After-Tax NPV5%	**USDM**	**$ 1,060**																	
After-Tax IRR	**%**	**27.83%**																	

Production Summary	Units	Totals	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Mining																			
Ore	kt	158,623	-	-	6,731	8,214	7,675	7,975	4,993	5,085	4,979	7,349	6,420	7,186	5,557	4,740	5,872	4,276	5,796
Waste	kt	631,493	-	-	22,726	23,785	24,325	24,024	27,095	26,915	27,020	24,651	25,667	24,814	26,442	27,260	26,215	27,724	26,204

Total Material Mined	**kt**	**790,115**	**-**	**-**	**29,457**	**32,000**	**32,000**	**32,000**	**32,087**	**32,000**	**32,000**	**32,000**	**32,087**	**32,000**	**32,000**	**32,000**	**32,087**	**32,000**	**32,000**
Stripping Ratio	**Waste:Ore**	**3.98**	**-**	**-**	**3.38**	**2.90**	**3.17**	**3.01**	**5.43**	**5.29**	**5.43**	**3.35**	**4.00**	**3.45**	**4.76**	**5.75**	**4.46**	**6.48**	**4.52**
Ore to Crushing Circuit																				
Mined Ore (Mined Direct and Stockpiled)	kt	158,623	-	-	2,934	5,309	5,325	5,325	5,340	5,325	5,325	5,325	5,340	5,325	5,325	5,325	5,340	5,325	5,325
Grade	g Au/t	0.97	-	-	1.19	1.19	1.19	1.20	0.99	0.99	0.99	0.99	0.99	1.00	0.99	0.99	1.00	0.99	1.00
Contained Gold	koz	4,959	-	-	112	204	203	205	171	170	169	170	170	171	170	170	172	169	171
Ore to Milling/CIL Circuits																				
Mined Ore (Post-Sorting)	kt	145,790	-	-	2,696	4,880	4,894	4,894	4,908	4,894	4,894	4,894	4,908	4,894	4,894	4,894	4,908	4,894	4,894
Grade	g Au/t	1.04	-	-	1.27	1.28	1.27	1.28	1.06	1.06	1.06	1.06	1.06	1.07	1.06	1.06	1.07	1.06	1.07
Contained Gold	koz	4,875	-	-	110	200	200	201	168	167	167	167	167	168	167	167	169	166	168
Heap Leach Pad Ore	kt	13,354	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Grade	g Au/t	0.54	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Gold	koz	232	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Ore to Process Plant	kt	159,144	-	-	2,696	4,880	4,894	4,894	4,908	4,894	4,894	4,894	4,908	4,894	4,894	4,894	4,908	4,894	4,894
Grade	g Au/t	1.00	-	-	1.27	1.28	1.27	1.28	1.06	1.06	1.06	1.06	1.06	1.07	1.06	1.06	1.07	1.06	1.07
Contained Gold	koz	5,107	-	-	110	200	200	201	168	167	167	167	167	168	167	167	169	166	168
CIL Recovery	%	89.2%	0.0%	0.0%	90.5%	90.5%	90.5%	90.5%	89.9%	89.9%	89.9%	89.9%	89.9%	90.0%	89.9%	89.9%	90.0%	89.9%	90.0%
Gold Production	koz	4,554	-	-	100	181	181	182	151	150	150	150	150	151	150	150	152	150	151
Overall Recovery		87.7%	0.0%	0.0%	89.0%	89.0%	88.9%	89.0%	88.4%	88.4%	88.4%	88.4%	88.4%	88.4%	88.4%	88.4%	88.4%	88.4%	88.4%

Unit Cost Metrics	Units	Totals	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Cash Operating Costs																			
Mining	$/t mined	$ (3.34)	-	-	$ (2.25)	$ (2.77)	$ (2.90)	$ (2.91)	$ (2.88)	$ (3.00)	$ (3.20)	$ (3.25)	$ (3.17)	$ (3.14)	$ (3.09)	$ (3.06)	$ (3.07)	$ (3.08)	$ (3.15)
Mining	$/t processed	$ (15.37)	-	-	$ (22.63)	$ (16.71)	$ (17.42)	$ (17.49)	$ (17.28)	$ (18.05)	$ (19.26)	$ (19.50)	$ (19.06)	$ (18.87)	$ (18.60)	$ (18.39)	$ (18.44)	$ (18.53)	$ (18.95)
Processing	$/t processed	$ (17.86)	-	-	$ (22.39)	$ (17.51)	$ (17.50)	$ (17.50)	$ (17.46)	$ (17.52)	$ (17.54)	$ (17.54)	$ (17.51)	$ (17.54)	$ (17.54)	$ (17.54)	$ (17.52)	$ (17.55)	$ (17.50)
G&A	$/t processed	$ (2.06)	-	-	$ (2.81)	$ (2.01)	$ (2.01)	$ (2.07)	$ (2.07)	$ (2.07)	$ (2.07)	$ (2.07)	$ (2.06)	$ (2.07)	$ (2.07)	$ (2.07)	$ (2.06)	$ (2.07)	$ (2.07)
Sub-total: Cash Operating Costs	**$/t processed**	**$ (35.28)**	**-**	**-**	**$ (47.82)**	**$ (36.23)**	**$ (36.94)**	**$ (37.06)**	**$ (36.81)**	**$ (37.64)**	**$ (38.87)**	**$ (39.11)**	**$ (38.64)**	**$ (38.48)**	**$ (38.21)**	**$ (38.00)**	**$ (38.02)**	**$ (38.15)**	**$ (38.52)**
Non-Operating Costs																			
JAAC Royalty	$/t processed	$ (1.99)	-	-	$ (2.55)	$ (2.56)	$ (2.54)	$ (2.57)	$ (2.12)	$ (2.11)	$ (2.11)	$ (2.11)	$ (2.11)	$ (2.13)	$ (2.11)	$ (2.11)	$ (2.13)	$ (2.11)	$ (2.13)
Wheaton Royalty	$/t processed	$ (0.69)	-	-	$ (0.96)	$ (0.96)	$ (0.96)	$ (0.97)	$ (0.80)	$ (0.79)	$ (0.79)	$ (0.79)	$ (0.79)	$ (0.80)	$ (0.80)	$ (0.80)	$ (0.80)	$ (0.79)	$ (0.80)
Refining	$/t processed	$ (0.13)	-	-	$ (0.17)	$ (0.17)	$ (0.17)	$ (0.17)	$ (0.14)	$ (0.14)	$ (0.14)	$ (0.14)	$ (0.14)	$ (0.14)	$ (0.14)	$ (0.14)	$ (0.14)	$ (0.14)	$ (0.14)
Sub-total: Non-Operating Costs	**$/t processed**	**$ (2.81)**	**-**	**-**	**$ (3.68)**	**$ (3.69)**	**$ (3.67)**	**$ (3.70)**	**$ (3.06)**	**$ (3.05)**	**$ (3.05)**	**$ (3.05)**	**$ (3.05)**	**$ (3.08)**	**$ (3.05)**	**$ (3.05)**	**$ (3.08)**	**$ (3.04)**	**$ (3.08)**
Total: Cash Costs	**$/t processed**	**$ (38.08)**	**-**	**-**	**$ (51.50)**	**$ (39.92)**	**$ (40.61)**	**$ (40.77)**	**$ (39.87)**	**$ (40.69)**	**$ (41.91)**	**$ (42.16)**	**$ (41.68)**	**$ (41.56)**	**$ (41.26)**	**$ (41.05)**	**$ (41.10)**	**$ (41.19)**	**$ (41.60)**
Cash Costs	$/oz	$ 1,438	-	-	$ 1,513	$ 1,171	$ 1,197	$ 1,192	$ 1,411	$ 1,446	$ 1,489	$ 1,498	$ 1,481	$ 1,462	$ 1,466	$ 1,456	$ 1,445	$ 1,466	$ 1,463
AISC	$/oz	$ 1,535	-	-	$ 1,685	$ 1,183	$ 1,247	$ 1,202	$ 1,451	$ 1,547	$ 1,525	$ 1,517	$ 1,538	$ 1,488	$ 1,561	$ 1,504	$ 1,481	$ 1,525	$ 1,501

Table 162 Annual Cash Flow Pre-Production to Year 15 in USD terms

19.3.9 Annual Cash Flow from Year 16 to Closure

Cash Flow Summary	Units	Totals	16-20	21-25	26-30	31-33	34-43
Gold Production	**koz**	**4,554**	**693**	**629**	**747**	**186**	**-**
Gold Price	USD	$ 2,500	$ 2,500	$ 2,500	$ 2,500	$ 2,500	$ 2,500
Gold Sales	USDM	**$ 11,385**	**$ 1,733**	**$ 1,572**	**$ 1,868**	**$ 465**	-
Cash Operating Costs							
Mining	USDM	$ (2,642)	$ (513)	$ (506)	$ (154)	$ (37)	
Processing	USDM	$ (3,071)	$ (467)	$ (469)	$ (466)	$ (263)	$ (34)
G&A	USDM	$ (354)	$ (55)	$ (56)	$ (55)	$ (25)	-
JAAC Royalty	USDM	$ (342)	$ (52)	$ (47)	$ (56)	$ (14)	-
Wheaton Royalty	USDM	$ (118)	$ (20)	$ (16)	$ (14)	$ (4)	-
Refining	USDM	$ (23)	$ (3)	$ (3)	$ (4)	$ (1)	-
Sub-total: Cash Operating Costs	**USDM**	**$ (6,550)**	$ (1,111)	$ (1,097)	$ (748)	$ (343)	$ (34)
Cash Operating Margin	**USDM**	**$ 4,835**	**$ 622**	**$ 475**	**$ 1,120**	**$ 122**	**$ (34)**
Capital Costs							
Initial Capex	USDM	$ (425)					
Sustaining Capex	USDM	$ (266)	$ (106)	$ (42)	$ (23)	$ (10)	-
Reclamation & Closure	USDM	$ (176)	$ (18)	$ (40)	$ (33)	$ (1)	$ (54)
Salvage	USDM	-	-	-	-	-	-
Sub-total: Capital Costs	**USDM**	**$ (867)**	**$ (124)**	**$ (81)**	**$ (56)**	**$ (11)**	**$ (54)**
Working Capital Changes	**USDM**	**$ 0**	**$ (8)**	**$ 9**	**$ (10)**	**$ (7)**	**$ (6)**
Pre-Tax Cash Flow	**USDM**	**$ 3,968**	**$ 490**	**$ 402**	**$ 1,053**	**$ 103**	**$ (94)**
Northern Territory Royalty	USDM	$ (398)	$ (61)	$ (55)	$ (65)	$ (16)	-
Income Taxes	USDM	$ (1,083)	$ (149)	$ (81)	$ (287)	$ (19)	-
After-Tax Cash Flow	**USDM**	**$ 2,486**	**$ 280**	**$ 266**	**$ 700**	**$ 68**	**$ (94)**
After-Tax Cumulative Cash Flow	**USDM**		**$ 1,545**	**$ 1,811**	**$ 2,511**	**$ 2,580**	**$ 2,486**
Payback (yrs)	Years	2.65					
Pre-Tax NPV5%	USDM	**$ 1,736**					
Pre-Tax IRR	%	**37.3%**					
After-Tax NPV5%	USDM	**$ 1,060**					
After-Tax IRR	%	**27.83%**					

Production Summary	Units	Totals	16-20	21-25	26-30	31-33	34-43
Mining							

Production Summary	Units	Totals	16-20	21-25	26-30	31-33	34-43
Ore	kt	158,623	22,518	25,203	18,054	-	-
Waste	kt	631,493	137,507	102,927	6,193	-	-
Total Material Mined	**kt**	**790,115**	**160,025**	**128,130**	**24,247**	**-**	**-**
Stripping Ratio	**Waste:Ore**	**3.98**	**6.11**	**4.08**	**0.34**	**-**	**-**
Ore to Crushing Circuit							
Mined Ore (Mined Direct and Stockpiled)	kt	158,623	26,640	26,654	26,640	1,178	-
Grade	g Au/t	0.97	0.92	0.83	0.98	0.58	-
Contained Gold	**koz**	**4,959**	**786**	**715**	**841**	**22**	**-**
Ore to Milling/CIL Circuits							
Mined Ore (Post-Sorting)	kt	145,790	24,484	24,498	24,484	1,083	-
Grade	g Au/t	1.04	0.98	0.89	1.05	0.62	-
Contained Gold	koz	4,875	772	702	827	21	-
Heap Leach Pad Ore	kt	13,354	-	-	-	13,354	-
Grade	g Au/t	0.54	-	-	-	0.54	-
Contained Gold	koz	232	-	-	-	232	-
Total Ore to Process Plant	kt	159,144	24,484	24,498	24,484	14,437	-
Grade	g Au/t	1.00	0.98	0.89	1.05	0.55	-
Contained Gold	koz	5,107	772	702	827	253	-
CIL Recovery	%	89.2%	89.8%	89.5%	90.3%	73.4%	
Gold Production	**koz**	**4,554**	**693**	**629**	**747**	**186**	**-**
Overall Recovery		87.7%	88.3%	88.0%	88.8%	73.3%	
Cash Operating Costs							
Mining	$/t mined	$ (3.34)	$ (3.21)	$ (3.95)	$ (4.64)	-	-
Mining	$/t processed	$ (15.37)	$ (19.27)	$ (18.97)	$ (5.77)	$ (2.54)	-
Processing	$/t processed	$ (17.86)	$ (17.55)	$ (17.59)	$ (17.49)	$ (18.08)	-
G&A	$/t processed	$ (2.06)	$ (2.07)	$ (2.11)	$ (2.06)	$ (1.75)	-
Sub-total: Cash Operating Costs	**$/t processed**	**$ (35.28)**	**$ (38.89)**	**$ (38.66)**	**$ (25.32)**	**$ (22.36)**	**-**
Non-Operating Costs							
JAAC Royalty	$/t processed	$ (1.99)	$ (1.95)	$ (1.77)	$ (2.10)	$ (0.96)	-
Wheaton Royalty	$/t processed	$ (0.69)	$ (0.74)	$ (0.61)	$ (0.53)	$ (0.24)	-
Refining	$/t processed	$ (0.13)	$ (0.13)	$ (0.12)	$ (0.14)	$ (0.06)	-
Sub-total: Non-Operating Costs	**$/t processed**	**$ (2.81)**	**$ (2.82)**	**$ (2.50)**	**$ (2.77)**	**$ (1.26)**	**-**
Total: Cash Costs	**$/t processed**	**$ (38.08)**	**$ (41.71)**	**$ (41.16)**	**$ (28.10)**	**$ (23.63)**	**-**
Cash Costs	$/oz	$ 1,438	$ 1,602	$ 1,744	$ 1,002	$ 1,846	-

Production Summary	Units	Totals	16-20	21-25	26-30	31-33	34-43
AISC	$/oz	$ 1,535	$ 1,782	$ 1,874	$ 1,007	$ 1,908	-

Table 163 Annual Cash Flow Year 16 to Year 43 (end of Project) in USD terms

19.3.10 NPV, IRR, Payback

Based on the Annual Cash Flow model results, the Project has an unlevered after-tax NPV5% of USD1,060 million, and after-tax IRR of 27.8%; and a payback period of 2.7 years at a long-term gold price of USD2500/oz. The key financial metrics of the Project are summarized in Table 164.

	Life of Project
Pre-Tax NPV5%	$1,736
IRR (%)	37.3
After -Tax NPV5%	$1,060
IRR (%)	27.8
Payback Period (years)	2.7

Table 164 Key Financial Metrics

19.4 Sensitivity Analysis

Project sensitivities are summarized in Table 165, Table 166 and Table 167 sensitivities are shown graphically in Figure 110. The Project is most sensitive gold price. Sensitivity on operating and capital cost are closely matched, with the Project being only slightly more sensitive to operating costs.

Gold Price Sensitivity	-15%	-10%	-5%	Base	5%	10%	15%
Gold Price	$2,125	$2,250	$2,375	$2,500	$2,625	$2,750	$2,875
IRR	18.8%	21.9%	25.0%	27.8%	30.7%	33.4%	36.1%
After-Tax NPV5%	$559	$724	$896	$1,060	$1,232	$1,403	$1,575
After-Tax NPV8%	$325	$440	$560	$674	$793	$913	$1,032
After-Tax NPV10%	$223	$316	$414	$506	$602	$699	$796

Table 165 Gold Price Sensitivity Analysis

CAPEX Sensitivity	-15%	-10%	-5%	Base	5%	10%	15%
After -Tax NPV5%	$1,127	$1,105	$1,082	$1,060	$1,037	$1,015	$992
After -Tax NPV8%	$734	$714	$694	$674	$654	$634	$614
After -Tax NPV10%	$562	$543	$524	$506	$487	$468	$449

Table 166 Capex Sensitivity Analysis

OPEX Sensitivity	-15%	-10%	-5%	Base	5%	10%	15%
After -Tax NPV5%	$1,345	$1,250	$1,155	$1,060	$964	$869	$773
After -Tax NPV8%	$871	$805	$740	$674	$608	$542	$476
After -Tax NPV10%	$665	$612	$559	$506	$452	$399	$346

Table 167 Opex Sensitivity Analysis

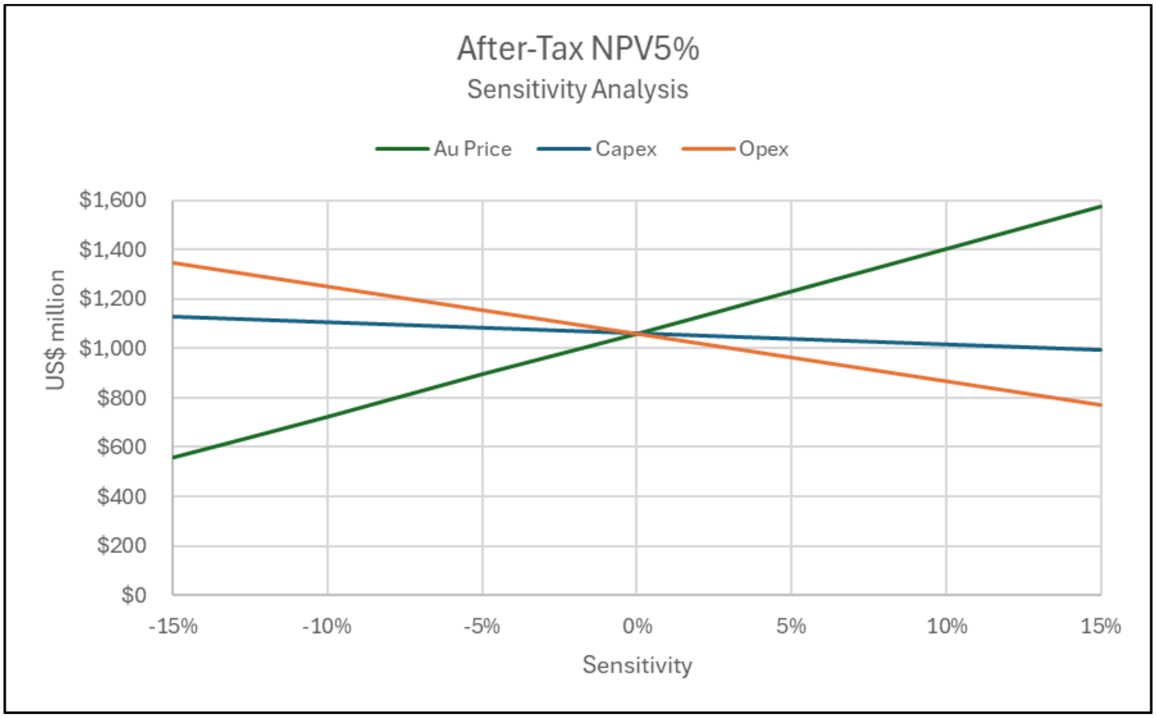


Figure 110 Project NPV5% Sensitivity Analysis, Vista 2025

20. ADJACENT PROPERTIES

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

21. OTHER RELEVANT DATA AND INFORMATION

21.1 Project Execution Plan

It is the intention of Vista to undertake the Project under several Engineering, Procurement and Construction Management style contracts that will be fixed and reimbursable. The cost estimate built up in Section 18 has been based upon engaging a mining contractor for the mining scope and an EPC contractor for the process plant and associated infrastructure. The key contracts are listed in Table 168:

Main Contract Area	Implementation Method for Delivery
Mining equipment and infrastructure	Mining contactor under a mining contract agreement
Process plant and infrastructure	EPC contract
Power plant	Under a build own operate via a selected third party
Gas Supply	Gas Purchase Agreement
Diesel Supply	Major Fuel supplier agreement
Tailings Storage Facility	EPCM contract managed by Vista
Surface Water Management	EPCM contractor, using local earthmoving as a subcontractor
Permanent Accommodation	EPCM contract from a provider and managed by Vista
Construction Accommodation	Reimbursable under the lead EPC contractor
Site Earthworks and Establishment	EPCM contractor, using local earthmoving as a subcontractor

Table 168 Key Head Contracts for Project Implementation

Vista will establish a client's representative team to manage the requirements for the delivery.

With respect to the main plant and infrastructure, the lead EPC contractor will undertake the engineering, design, drafting, procurement, construction management and specification of the process plant and infrastructure. All equipment will be purchased directly by the EPC contractor. This will ensure that equipment warranties are kept with the owner. Materials will be procured under the EPC contractor except for consumables, some bulk items and minor equipment.

The lead EPC contractor staff and mining contractor will be housed in the temporary construction and permanent accommodation village and construction contractors will be housed in the 200 bed temporary construction village and overflow catered for in Katherine.

21.2 Lead EPC Contractor

The EPC contractor will provide a range of project management, engineering, drafting, international and in-country procurement, contract management, fabrication management, logistics co-ordination, construction management and commissioning services necessary to provide a complete, safe, quality and technically compliant process plant and infrastructure. The scope of services will be established as follows:

- Project management and controls.
- Equipment and services procurement and contract management.
- Materials fabrication and delivery.
- Transport and logistics.
- Engineering and drafting.
- Construction management.
- Construction.
- Commissioning management.

21.2.1 General

The commencement date for the schedule is early January 2026, and completed over a and assumes all permits, approvals and funding are completed through August 2025 to December 2025.

21.2.2 Schedule Summary

Key Project milestones are summarized in Table 169.

Milestone	Commencement
Project commencement	Jan 26
EPC contractor engaged	Jan 26
Mining contractor engaged	Jun 27
Power Station Complete	Aug 27
Mining contractor mobilization	Dec 27
Dewatering finished	Dec 27
First ore to ROM pad	Apr 28
First ore to mill	Apr 28
Commissioning Complete and Hand Over	May 28

Table 169 Key Milestones for Project Implementation

The schedule is driven by the approvals process and resulting land access. All project areas require significant levels of preparation and earthworks before other trades can commence their activities. The schedule logic has focused on the sequential access to project development areas to open as many work fronts as possible.

21.2.3 Schedule Objectives and Scope

The key objective of the FS phase EPC schedule is to provide a Class 3, Level 3 detail Schedule with an accuracy range of ±10 - 15%.

For consistency with cost estimate classifications and accuracy definitions, refer to Table 130.

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 EPC Contractor's scope for the Process Plant and NPI, and the Owner's Works, as defined in the DFS. The schedule includes Client Activities, Mine Development, Tailings Dam, Power plant detail, Waste Water Treatment Plant and Dewatering.

21.2.4 Schedule Assumptions

The commencement date for the schedule is early January 2026, and assumes all permits, approvals and funding are completed through August 2025 to December 2025.

Siteworks has a non-working period of two weeks over Christmas/New Year, and 4 days over Easter each year.

Activity durations are based on man-hours per area/discipline, with activities outside of the critical path being spread within available float.

An initial 100-personnel construction camp will be installed during the earthworks period, available for the Civils Contractor and the Village Contractor, and then a further 100 rooms added for the construction village. The 250-personnel permanent camp will also begin early in the construction phase to accommodate the overflow for construction, mining and Owner's personnel.

- TSF Construction – 4 months.
- Dewatering requirements – Assumed to be suitable at 11 to 14 months.
- Mining equipment – Sourced and supplied by the Mining Contractor.
- The diversion channels can be constructed at any time during the Project, with a completion date prior to Ore Commissioning.
- Power Station – Assumed to be a BOO Contract.

21.2.5 Critical and Sub-Activities

There is a dual critical path for:

- Dewatering (including procurement and installation) of the Water Treatment Plant, with a significant period (remainder of the construction period) for dewatering. The total available time for dewatering is therefore only 14 months for dewatering after a 20-week lead time for a WTP and 6-month installation period.

- The award of the Mining Contractor with subsequent sourcing and mobilization of mining equipment, followed by four months of pre-production mining activities prior to having Ore ready for commissioning of the Process Plant Month 4 of Year 1.
- The Process Plant is sub-critical (only two weeks behind) of the mining contract and dewatering due to the lead time and construction of the tertiary grinding mills. The tertiary grinding mills have been ordered very early in the program otherwise they will be on the critical path and the timing of placing this order will determine the construction of the process plant.

21.2.6 Schedule Interfaces

The EPC construction 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 this Technical Report Summary 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. For consistency with cost estimate classifications and accuracy definitions, refer to Table 130.This schedule will only be baselined with the approval of Vista, EPC Contractor, and the Construction Contractor.

21.3 Water Management

21.3.1 Site-wide Water Balance

A site-wide water balance (SWWB) was developed within the GoldSim® software platform (Version 15) to simulate 2 years of preproduction, 33 years of mine production (27 active mining years and 3 additional year processing stockpiles, and 3 years of re-processing Heap Leach Pad ore), and 5 years of closure at the Vista Project Site.

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

- Verify water treatment plant (WTP) capacity and Edith River discharge quantities.
- Estimate enhanced evaporation system requirements.
- Determine Water Treatment Feed Pond (WTP Feed Pond) sizing.
- Quantify make up water requirements from the raw water dam (RWD) for the processing plant, dust suppression, and other water needs, and
- Determine overtopping event frequency.

21.3.2 Site-wide Water Balance Model

21.3.2.1 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:

Change in Storage = Inputs – Outputs

Information provided to the model and the rules by which the site features interacted are summarized below.

21.3.2.2 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).
- Water Treatment Plant Feed Pond (WTP Feed Pond).
- Water Treatment Plant (WTP).
- Process Plant (PP).
- Process Water Pond (PWP).
- Clean Water Pond (CWP).
- Dust Control.
- Tailings Storage Facility 1 (TSF1).
- Tailings Storage Facility 2 (TSF2).

21.3.2.3 General Assumptions

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

- RP1, LGRP, RP3, PRP, HLP, and a dry season TSF bleed stream report to the WTP Feed Pond which feeds makeup water to the Process Plant and discharges water through the WTP to the Edith River during production.
- TSF1 dewaters by pumping to the WTP Feed Pond during preproduction to allow for construction of embankment raises. TSF1 dewaters by pumping to the WTP Feed Pond during year 19 to 24 of production. TSF2 dewaters to the pit during closure.

- The WTP Feed Pond received water only if it was not at risk for overtopping. Given this logic, overtopping events were allowed to occur at the retention ponds.
- Inputs to all 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 WTP Feed Pond which feeds Process and the WTP except for the PRP which is a sediment pond and allowed to overtop. The WTP Feed Pond's Capacity at 1 m freeboard is 185,394 m^3.
- A dry season TSF decant bleed stream of 100 m^3/hr is sent to the WTP Feed Pond to maintain proper chemistry of the process circuit. This occurs in the dry season to allow for optimal control of the RP water levels and reduce the chance of overtopping events.
- Process makeup water is prioritized to come from WTP Feed Pond first, then TSF Decant, and then the RWD to reduce discharges to the Edith River while minimizing RWD inflows.
- During production, no permitted discharges to the Edith River (Edith) were allowed from any of the RPs with the exception of the PRP which is allowed to discharge. WTP effluent is allowed to discharge to the Edith at a dilution ratio of 19:1 (Edith/WTP).
- The HLP and LGOS were run through the PP at the end of the LOM to recover gold from these ore stockpiles.
- Initial dewatering of the Pit occurred during preproduction and consisted of a treat and release system which will release at a 1:19 ratio (discharge : Edith River flow) with a maximum flowrate of 1000 m^3/hr, and an enhanced evaporation system removing 1.73 GL per year.
- Enhanced evaporation will occur on TSF1 during preproduction, years 1-19 when TSF1 was operating, and years 19-24 when TSF1 was in closure. The evaporation rate was 0.47 GL per year.
- Enhanced evaporation occurred on TSF2 during production years 19-30 when TSF2 was operating, and years 30-32 when TSF2 was in closure.

21.3.2.4 Initial Conditions

- TSF1, the Batman Pit, LGRP, and the HLP were assigned initial water surface elevations based on the water elevation for 1 September 2024, and
- The PRP is assumed to be empty at the beginning of production.

21.3.2.5 Flow Rates

These rates represent the mean flows throughout the simulation unless stated otherwise:

- Potable water need was 20.9 m^3/hr over the Life of the Project and potable water came from the RWD.
- Gland water was specified as 17.48 m^3/hr and comes from the WTP Feed Pond, TSFs, or RWD.

- Reagent water was specified as 62.41 m^3/hr and comes from the WTP Feed Pond, TSFs, or RWD.
- TSF Decant return varied seasonally from 212 m^3/hr to 612 m^3/hr.
- Process Plant makeup water flows were 655 m^3/hr and come from either the WTP Feed Pond, TSFs, or RWD depending on availability.
- Dust suppression requirements varied between 220 and 1,153 m^3/day.
- WTP rate was set at 300 m^3/hr.

21.3.2.6 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 170.

Month	Rainfall (mm)
January	309
February	284
March	214
April	46
May	5
June	2
July	1
August	1
September	7
October	34
November	121
December	242

Table 170 Mean Monthly Precipitation

- 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 2024 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.

21.3.2.7 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 2 years of preproduction, the 30-year LOM, and 2 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.

21.3.2.8 Results

Under the modeled conditions described previously the SWWB model results indicate that:

- With a Treat and Release rate of a maximum of 1000 m^3/hr, and a dilution ratio of 1:19 in addition to an enhanced evaporation system capable of removing 1.73 GL annually, the pit can be dewatered within 2 years of preproduction. The Batman Pit will see minor water storage during the wet season and later in the Life of the Project. This is because of increased groundwater inflows and a large catchment area near the end of the Life of the Project.
- With a WTP rate of 300 m^3/hr and the ability to release to the Edith River at a 1:19 dilution ratio when need in addition to the enhanced evaporation systems on the TSFs, the site safely manages site contact water.
- The mean water (process and dust suppression) draw required from the RWD varied from 500 m^3/day to 3,900 m^3/day with the latter occurring late in the dry season and later in the Life of the Project. The estimated total water volume required from the RWD over the Life of the Project is 3.5 GL. RWD requirements were found to be the most dependent upon the TSF levels and site contact water available for makeup.
- The RP3, CWP, PWP, TSF1, and TSF2 showed no potential overtopping events through the Life of the Project.
- RP1 shows a less than 1% chance of overtopping events through the Life of the Project.
- The Heap Leach Pad (HLP) shows approximately a 5% probability of having an overtopping event in the latter portion of the Life of the Project.

- The Low-Grade Ore Stockpile Retention Pond (LGRP) shows approximately a 15% probability of having an overtopping event in the latter portion of the Life of the Project.

21.4 Geochemistry

Tetra Tech was commissioned by Vista to conduct geochemical characterization studies and predictive modelling in support of the Project Technical Report Summary.

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 wt. % 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 less than or equal to 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 less than or equal to 11 kilograms of calcium carbonate per tonne of rock (kg $CaCO_3$/tonne rock). An acid base accounting (ABA) neutralization potential ratio (NPR) screening criteria of less than 2 suggests that a majority of the waste rock samples are either potentially acid forming (PAF) 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 (greater than 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 modelling 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 sulphate 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 a low NP of 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 modelling 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 Life of the Project site water management planning.

Inputs to the Process Water Pond included precipitation and inputs from ponds/facilities from across the site including:

- RP 1 – WRD Retention Pond.
- RP 2 – Low Grade Ore Stockpile Retention Pond (LGRP).
- RP 3 – Batman Pit.
- RP5 – Plant Site Runoff Settling Pond.
- HLP – Heap Leach Pad Pond.
- RP7 – Tailings Storage Facility 1 (TSF1) Pond.
- RP8 – Tailings Storage Facility 2 (TSF2) Pond.
- 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 RP3 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. Additional water removal is also anticipated through the use of evaporators.

21.5 Surface Water Hydrology

The Project Site is drained by the perennial Edith River, located approximately 0.9 km south of RP1 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 (RP1).

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. 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 RP1.

Drainage from the Project Site enters the Edith River at two locations: discharge point for RP 1 and West Creek. The RP1 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 RP1 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 RP1. 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 (RP1), 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 1000-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.

21.6 Regional Ground Water (including 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 previous Sections; the geologic information in this section is presented from a hydrogeologic perspective as it relates to groundwater flow and pit dewatering.

21.6.1 *Regional and Site Hydrogeology*

In the Project 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 Project 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 Project 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.

21.6.2 Regional Numerical Groundwater Flow Model

Tetra Tech constructed a regional numerical groundwater flow model to estimate groundwater inflows to the open pit at the Project 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 and related hydrologic features such as streams.

For this Technical Report Summary, 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.

21.6.3 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.

21.6.4 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 171.

Mining Year	Nov-Jan (Wet Season) Mean Monthly Inflow Volume (m^3)	Jun-Aug (Dry Season) Mean Monthly Inflow Volume (m^3)
1	57,736	37
5	97,549	4,115
10	144,092	8,066
20	230,745	6,674
30	331,479	5,240

Table 171 Seasonal Inflow Volumes and for Mine Dewatering Design

21.6.4.1 Mine Dewatering

Dewatering of the proposed 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. The 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 WTP Feed Pond. Depending on the depth of the pit and routing of the pipeline to the crest of the pit, water will be pumped from the pit floor would first go through a pair of pumps mounted in the pit sump and then through skid mounted booster pumps. Lifts with booster pumps will be added in stages with increasing pit depth. Once at the surface, the water would be piped to the WTP Feed Pond.

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.

21.7 Process Plant Geotech

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 Project 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 equipment locations of heavy vibrating equipment comprising the primary and secondary crushers, screens, HPGR's 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.

22. INTERPRETATION AND CONCLUSIONS

22.1 General

Vista retained GRES, to coordinate several consultants under the supervision of Vista to prepare this Technical Report Summary for the Project. This Technical Report Summary evaluates a development scenario of a 15,000 tonne per day (15 ktpd) processing facility.

The 15 ktpd operation includes:

- Estimated Proven and Probable Mineral Reserves of 4.96 Moz of gold (158.6 Mt at 0.97 g Au/t) at a cut-off grade of 0.5 g Au/t plus an additional 0.23 Moz of gold (13.4 Mt at 0.54 g Au/t) from the historical Heap Leach Pad, to be reprocessed at the end of the life of mine as a form of self-funded reclamation.
- Average annual production of approximately 150 koz of gold per year over the 30 year mine life.
- A 33-year operating life (include 3 years at end of open pit mining to process Heap Leach Pad material).

In the opinion of the Qualified Persons engaged to deliver this Technical Report Summary, the data and analysis presented in this Technical Report Summary support the interpretations and conclusion that the Project is technically achievable and economically viable. The risks and uncertainties identified may affect the reliability of the Mineral Resource and Reserve Estimates and the projected economic outcomes. These risks will be managed through ongoing technical studies, monitoring, and adaptive planning.

22.1.1 Geology and Mineral Resources

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

Gold mineralization in this area is constrained to a single mineralization event and the deposits are classified as orogenic gold deposits in the subdivision of thermal aureole gold style. The Batman deposit has characteristics of an intrusion related gold system making it the primary Mineral Resource.

The Batman deposit is defined by approximately 8.5 Moz of gold within 316 Mt of Measured and Indicated Mineral Resource at an average grade of 0.83 g Au/t and a cut-off grade of 0.4 g Au/t and 1.4 Moz of gold within 54.3 Mt of Inferred Mineral Resource at an average grade of 0.78 g Au/t.

The Quigleys deposit is defined by approximately 0.4 Moz of gold within 10.7 Mt of Measured and Indicated Mineral Resource at an average grade of 1.26 g Au/t and a cut-off grade of 0.4 g Au/t and 0.06 Moz of gold within 2.7 Mt of Inferred Mineral Resource at an average grade of 0.71 g Au/t.

The Heap Leach Pad is defined by approximately 0.2 Moz of gold within 13.4 Mt of Indicated Mineral Resource at an average grade of 0.54 g Au/t and a cut-off grade of 0.4 g Au/t.

22.1.2 Mineral Reserve Estimate

The Project is at a FS stage based on a conventional open pit, truck and hydraulic excavator operation feeding a nominal 15 ktpd processing plant. The Mineral Reserve is supported by this Technical Report Summary.

This Technical Report Summary presented to support the Mineral Reserve that is developed to a Feasibility Study level. This includes a mine plan that is technically achievable and economically viable. Mine optimization, mine design, mine schedule, and mining costing were undertaken by Mining Plus. Specialist expertise was also utilized in the mining study team, comprising the following partners and consultants engaged by Vista, all with recent experience on the Project and/or relevant experience in the NT:

- A tier 1 Australian mining contractor to provide mine cost analysis and pricing for the mining operations and mine site infrastructure.
- Blast fragmentation study specific to the rock conditions at Project completed by Orica. This provided guidance for drill and blast designs and equipment requirements, and subsequent costing. Orica also provided pricing to the mining contractor for the provision of all explosives and a full down-hole service.
- This Technical Report Summary was undertaken by a team of industry professionals involving numerous consultants and professionals focused on technical areas including infrastructure, approvals, environmental, governance, community, local considerations, operations readiness, geochemistry, hydrogeology, geotechnical engineering, metallurgical, and FS discounted cashflow models

A large number of mine schedule runs have been run at both the strategic optimization and detailed FS mine schedule level to ensure the pit development is economically robust and viable, and practical to mine with consideration of the bounds of this Technical Report Summary.

Mine production constraints were imposed to ensure that mining was not overly aggressive with respect to the equipment anticipated for use at Project and pit geometry considerations such as sink rates, while ensuring adequate mine production to allow blending to ensure quality feed and maximize value through the processing plant. The schedule has been produced using mill targets and stockpiling strategies to enhance the Project economics, while also considering the ramp-up and commissioning requirements at of both the processing plant and mine mobile equipment. The constraints and limits are reasonable to support the Project economics which are used to justify the statement of Mineral Reserves.

In summary, the final result was a total of 6 pit designs incorporating the final ultimate pit design and 5 interim stages. During mine scheduling further delineation of the ultimate pit design occurred. The ultimate pit design delineated four geologically and spatially distinct mining zones, each of which could be developed and mined independently. To enhance operational flexibility and optimize sequencing during mine scheduling, these zones were individually coded as discrete mining stages—Stages 6, 7, and 8.

This Technical Report Summary shows that the mine plan is technically achievable and economically viable taking into consideration all material Modifying Factors

Mineral Reserves are reported using the following guidelines as defined by Subpart 229.1300 of Regulation S-K 1300 and are based on open pit mining methods. The Mineral Reserves are forward-looking information and actual results may vary. The risks regarding Mineral Reserves are summarized in this Technical Report Summary and in the risk Section 22.2.

Areas of uncertainty that may materially impact the Mineral Reserve estimates include: changes to long-term metal price assumptions; changes to include operating, and capital assumptions used, including changes to input cost assumptions such as consumables, labor costs, royalty and taxation rates; variations in geotechnical, mining, dilution, and processing recovery assumptions; including changes to pit phase designs as a result of changes to geotechnical, hydrogeological, and engineering data used; and changes to environmental, permitting and social license assumptions.

WSP was engaged by Vista to conduct the geotechnical assessment of the Batman pit slopes which forms part of this Technical Report Summary. Open pit slope recommendations were provided to guide pit optimization and mine design by the Mining Plus team.

22.1.3 Mining Methods

Mineral Reserves were estimated for the Project assuming open pit mining methods with conventional equipment for drilling, blasting, loading and haulage appropriate for the proposed production rates. This Technical Report Summary assumes a contract miner model for mining operations, involving the provision of a full mining operations service and construction and operation of the mining infrastructure required.

A 12 m bench height was used aligning with the Mineral Resource estimate block height and based on the required production rate and appropriately sized equipment. Drilling will involve 171 mm diameter holes for all production shots, with modified blasting required to meet geotechnical wall control recommendations. Furthermore, the use of backhoes in final wall areas will assist with final wall quality outcomes.

Pit designs use 12 m benches for mining. This aligns to the Mineral Resource model block heights and is believed to be reasonable with respect to dilution and equipment geometry anticipated to be used in mining. In areas where the material is inconsistently ore or at the surround ore boundary, when dilution may be an issue, benches may be mined in 6 m heights based on geological recommendations, and the contractor will mine selectively utilizing hydraulic excavators in a backhoe configuration. The dilution study completed during this Technical Report Summary on the initial Mineral Resource model shows the need for this is minimal, with overall findings of the dilution study incorporated sufficiently within the update Mineral Resource model used for the final optimization, mine design and scheduling used as the basis of the FS, and the Mineral Reserves.

The mining operations will be executed by a tier 1 Australian contract mining company, selected for its capability to manage large-scale operations and maintain high equipment availability. The contractor will utilize a fleet comprising 400 tonne class hydraulic excavators and 190 tonne class rigid-frame haul trucks, supported by ancillary equipment including loaders, dozers, graders, and water carts.

Reasonable mine designs, mine production schedules, and mine costs have been developed for the Project. Blasting costs are high comparable to other mining projects due to the high powder factor for ore blasting to assist with ore processing, and to allow for wall trim blasting. Costs for mining and mining infrastructure have been provided based on the analysis of this Technical Report Summary mine schedule completed by a tier 1 Australian mining contractor and consider local site and NT requirements, and availability of Mineral Resources such as equipment and labor.

The mine production schedule is based on commencement of mining in Month 1 of Year 1, with mill feed to commence in Month 4 of Year 1. A mining ramp-up is included in the schedule considering mobilization and commissioning of the mining contractor to site, with first two months of mining at 50% of production, which then lifts to 100% in Month 3 of year 1. Prior to the commencement of mining, all mining infrastructure will be constructed by the mining contractor in year -1, to facilitate efficient mine start-up. However, any delays in the commencement in the Project may delay the start date of mining in which case, the dates provided in the schedule and this Technical Report Summary are indicative only and cannot be relied upon.

The WRD required for the mine are substantial, with detailed work on the waste rock dump design and developed completed by Tierra Group, and Mine Closure by Tetra Tech.

22.1.4 Metallurgy, Mineral Processing Plant and Infrastructure

Historical metallurgical test work programs have shown the Batman ore is hard and competent. Gold is fine grained (<30 µm) and associated with sulfide minerals and quartz but is amenable to extraction by conventional cyanidation processes. The results of tests done from 2017 to 2018 and 2018 to 2019 have confirmed that fine grinding to a P_{80} size of 40 µm was required to sufficiently liberate the low-grade gold bearing ore to achieve moderately high leach gold extractions and enhance the leach kinetics. An average gold extraction of 90% was achieved after 30 hours of leaching in the 2018 to 2019 test work program on samples that had two stages of grinding to P_{80} sizes of <53 µm.

The gold leach extractions varied with gold head grade but were relatively consistent within the expected gold head grade range included for processing in the production schedule. The test work included pre-aeration and conditioning of the slurry with lime and lead nitrate prior to cyanidation to reduce the hindering effect of iron sulfide minerals present in the ore and this has been included in the plant design. The design has incorporated oxygen instead of air to be added in to pre-conditioning tanks and the leach and adsorption circuit. The ore has moderate to high cyanide consumption due to the presence of iron sulfide and copper minerals.

The processing plant has been designed to treat 5.325 Mtpa (15 ktpd) of hard ore from the Batman open pit and is described in Section 14 in detail. The crushing circuit design utilizing a primary gyratory crusher, secondary cone crusher and HPGR tertiary crushing is a robust, proven technology to generate a suitable grinding circuit feed. The two-stage grinding circuit configuration includes a primary overflow ball mill and four secondary vertical stirred grinding mills. The secondary grinding mill type selected is a Vertimill using hi-chrome steel media. The processing facility unit processes are based on well proven technologies for both gold recovery and treatment of hard ore.

During the initial phases of the Technical Report Summary GRES reviewed the previous designs and raised several queries in relation to the ore sorting, grind size, recovery method and historical test work. GRES addressed these queries during this Technical Report Summary with some revised approaches.

A fine leach feed P_{80} size of 40 µm has been retained from the previous studies to maximize gold extraction and leach kinetics and the ore is of high competency. Thus, most of the capital and operating costs are within this front end of the plant. The plant has a restricted front end layout due to the limited available land and so is restricted in the ability to expand this area of the plant due to the waste dumps, water course and restricted areas. The post grinding areas can be expanded relatively easily and will require some extensive demolition of the remaining existing facility to utilize this layout space.

When laying out this plant the remanent facilities on site were avoided as much as possible to minimize any major demolition and reduce front end capital costs.

22.1.5 Tailings Storage Facilities

The design of the TSF has been influenced by the interpreted material properties of the existing tailings in TSF1. A new grinding process expected from the mill may result in varied material properties. Lower quartile interpretations of the liquefied undrained shear strength ratio for the tailings have been used in the geotechnical analysis to ensure a conservative design for TSF1 and TSF2.

22.1.6 Results of the Site-wide Water Balance Model

Under the modeled conditions, the SWWB model results indicate that:

- The WTP rate of 300 m^3/hr and WTP feed pond of 185,000 m^3 of storage with enhanced evaporation systems appear adequate for safely managing site contact water. It is estimated that 11.5 GL of water will be discharged to the Edith River during production, with over 2/3 of that occurring when TSF2 is active.
- Site water needs including Process Plant makeup water required from the RWD varied from approximately 500 m^3/day to 3,900 m^3/day with the latter occurring late in the dry season and early in the Life of the Project. The estimated total water volume required from the RWD over the Life of the Project is 3.5 GL. The makeup water requirements were most dependent upon the amount of site contact water available to provide makeup water to the Process Plant and on TSF decant volumes.
- With a Treat and Release rate of a maximum of 1000 m^3/hr, and a dilution ratio of 1:19 in addition to an enhanced evaporation system capable of removing 1.73 GL annually, the pit can be dewatered within 2 years of preproduction. The Batman Pit will see minor water storage during the wet season and later in the Life of the Project. This is because of increased groundwater inflows and a large catchment area near the end of the Life of the Project.
- The mean water (process and dust suppression) draw required from the RWD varied from 500 m^3/day to 3,900 m^3/day with the latter occurring late in the dry season and later in the Life of the Project. The estimated total water volume required from the RWD over the Life of the Project is 3.5 GL. RWD requirements were found to be the most dependent upon the TSF levels and site contact water available for makeup.
- The RP3, CWP, PWP, TSF1, and TSF2 showed no potential overtopping events through the Life of the Project.
- RP1 shows a less than 1% chance of overtopping events through the Life of the Project.
- The Heap Leach Pad (HLP) shows approximately a 5% probability of having an overtopping event in the latter portion of the Life of the Project.
- The Low-Grade Ore Stockpile Retention Pond (LGRP) shows approximately a 15% probability of having an overtopping event in the latter portion of the Life of the Project.

22.1.7 Environmental and Social Considerations

22.1.7.1 Existing Body of Work

Several 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.

22.1.7.2 Environmental Impact Study and Approvals

The 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. Modifications to align existing approvals with the 15 ktpd Project have been initiated.

22.1.7.3 Social or Community Impacts

The JAAC have strong involvement in the planning 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 Certificates for Mt Todd covers the 1,337 km^2 of exploration licenses contiguous with the 55.4 km^2 mining leases. An application for an AAPA Certificate authorizing work in additional areas associated with the 2024 FS and this Technical Report Summary has been initiated.

22.1.7.4 Community-based Staffing Discussion

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 regarding 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 several of them will find the benefits of employment that allows them to be home every night to be very attractive.

22.2 Project Risk

Significant risks and uncertainties that could reasonably affect the reliability or confidence in the Project outcome are provided in Table 172.

The Project is an advanced-staged development project that has undergone engineering and permitting for several years. 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.

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

Risk	Description	Probability	Severity
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 17***	Low	Medium
Property Holdings	Vista has secured the Project concession holdings as described in ***Section 3.*** 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 Mineral Resource	The Project viability relies upon historical drilling as well as recent drilling to develop and assess the Mineral Resource model. New drill results could adversely affect the interpretation of parts of the deposit, with impacts to Mineral Resources and production estimates.	Low	Low
Capital Costs	Some areas are well defined and others not as much and there are unknowns that can affect the price, including equipment and commodity price movements.	Low	Medium
Power Plant Estimated Capital and Operating costs	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. The power supply is based on gas and there is no gas contract in place.	Medium	Medium-High
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
Mobile Equipment Capital	Mobile equipment prices are an important part of the Project capital. Significant increases could impact the Project economics.	Low	Low-Medium

Risk	Description	Probability	Severity
Process Technology	Extensive testing has been completed to identify the most suitable technology and equipment in the process. 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

Table 172 Project Risks

22.2.1 Mineral Resource Estimates

Risks to be considered for the Mineral Resource estimation include the mineralization consisting of thin veins, which could vary during the mining and recovery processes. Changes in factors, such as metal prices, recovery, and costs may affect the cut-off grade, which would alter the reported Mineral Resource numbers. Geotechnical parameters could also have an impact on the pit shell used to report the Mineral Resource.

22.2.2 Mining and Mineral Reserves Risks

There are unknown risks and uncertainties, which could have a material adverse effect; this should not be considered an exhaustive list, but the key risks are included.

For the mining area, there are two critical risks identified that need to be a focus for future planned work. These are related to the geotechnical design parameters for pit walls, and the engagement of the mining contractor in sufficient time to be ready as per the current Project schedule.

The risk related to geotechnical design parameters for pit walls, could be seen as a risk or a potential opportunity if pit walls could steepen, and is part of the geotechnical consultant's recommendations from their FS geotechnical assessment. The focus of the geotechnical future work is initially obtaining improved geotechnical engineering data, particularly in key rock structural domains, through planned drilling and laboratory testing. Geotechnical analysis and geotechnical assessment completed to focus on the pit wall slope design parameters and overall optimal configuration for mine design and operations considering ramp location, pit scheduling and life expectancy of walls for interim stages. This work will hopefully result in steeper pit slopes to assist Project economics but could results in flatter walls in some areas. Also, the objective is to overall improve the geotechnical assessment and overall risk profile for the Project.

Regarding the geotechnical risks identified, this Technical Report Summary mining study considers and recognized these risks necessitate robust geotechnical investigations, ongoing monitoring, and adaptive mine design to ensure safe and efficient operations. Within this Technical Report Summary this includes modified blasting for pit wall control, equipment selection such as small diameter drills for pre-split and/or perimeter wall holes, and backhoe configured hydraulic excavators for pit wall excavation. The FS cost model and operations plan includes technology for pit wall monitoring and also within the personnel positions for geotechnical engineers and geotech field assistants.

Regarding the risk of ensuring the mining contractor is engaged with sufficient time to be ready as per the current Projects schedule this is easily addressed by planning to have commercial discussions and negotiations as part of a formal tender process with mining contractors at least 12-18 months prior to required site mobilization. This should be easily implementable based on current Project timing, and recent experiences and interactions with three of Australia's leading Tier 1 mining contractors and other service provides as part of this Technical Report Summary, that are aware of the Project and enthusiastic to be involved.

The are other identified risks with controls and actions for the execution works outlined in detail as part of this Technical Report Summary. Also, there is planned future work, particularly to be completed as part of the planned execution and pre-production stage, will be outlined. Other items related to the mining area for future work, particularly to be addressed in the planned execution and pre-production, to address identified risks are outlined below:

- Blasting Parameters – Post the completion of initial execution work involving rock mass data for the geotechnical and hydrogeological studies, this information be utilized to update drill and blast designs and assumptions, including overall pattern designs, assumptions for percentage of trim blasts required, use of emulsion assumption for wet holes, etc.
- Stockpile design and operations – Mine scheduling focused on the use of stockpiles to best deliver feed to the processing plant to maximize metal recovery and overall value. Within the detailed design execution works, further simulation of the stockpile operation and management will be required to ensure practical and safe operation of pre-crusher stockpiles to obtain mine planning requirements.
- Grade control plan – Related to point above but more focused to the in-pit operation is further definition of drilling plans and sampling requirements to achieve required ore definition and overall grade control. An allowance in this Technical Report Summary has been developed and is reasonable for this level of Technical Report Summary assuming sampling of blast holes.

- Waste rock characterization – Forward works for this area are detailed in other sections of this Technical Report Summary, in summary this forward work would include the following: further geochemical sampling and analytical Testing Program as well as an update of the ARD Classification Criteria to improve definition of PAF/NAF and reduce amount of unclassified material from an ARD perspective.
- Operational Readiness – Forward works in this area focused on consideration of the broader project approach required for attraction and retention of people, and strategic procurement of key consumables and equipment. This is addressed through implementing an engagement strategy, procurement and tender plan for the mining contractor, that specialize is maintained both of their equipment procurement and people recruitment and attraction pipelines.
- Mineral Resource Estimation and Orebody Definition – Geological modelling underpins resource estimation and mine planning. Inaccuracies in the interpretation of lithology, grade distribution, structural controls, or mineral continuity can result in misclassification of ore and waste, leading to suboptimal mine design and financial underperformance. Mitigation requires rigorous geological data collection, validation, and continuous model refinement throughout the LOM.
- Cost Overruns and Budget Blowouts – Unexpected increases in construction, labor, fuel, or equipment costs can lead to budget overruns. These are particularly common in remote regions like the Northern Territory, where logistics and supply chain constraints can amplify costs. Early engagement with potential suppliers, service provides and partners that has already occurred within this Technical Report Summary assists to address this risk.
- Workforce Availability and Retention – Attracting and retaining skilled labor in remote areas is a persistent challenge. Competition from other mining operations and lifestyle factors can lead to high turnover and increased training costs. As outlined by Vista team members during this Technical Report Summary, they will continue to engage with both local - Katherine, and regional – NT communities to identify locals to employ and have part of the operations team which will assist with retention of people. This has also been discussed and is part of the mining contractor's operations philosophy and strategy.
- Environmental and Climate Risks – The region is subject to extreme weather events, including cyclones and seasonal flooding, which may disrupt operations, damage infrastructure, and delay production schedules. This is considered as if, for example, the pit is flooded due to significant rainfall, there is sufficient stockpiles on the ROM pad to ensure continued mill feed for over a month. Environmental compliance, particularly around water management and biodiversity, is also a critical operational consideration which is part of this broader Technical Report Summary.
- Health, Safety, and Equipment Reliability – Mining operations carry inherent risks related to worker safety, equipment failure, and operational hazards. Maintaining high safety standards and ensuring equipment reliability are essential to avoid costly downtime and reputational damage. For example, safety in design has been considered as part of the haul road design and road layout in this Technical Report Summary and will also require a detailed traffic management plan to be developed in execution works.

- Community and Indigenous Engagement – The Project and Vista must continue to navigate complex stakeholder relationships, including continued engagement with Indigenous communities and landholders. Failure to manage these relationships effectively can result in delays, legal challenges, or loss of social license to operate

22.2.3 *Metallurgical Test Work*

No new metallurgical test work has been completed post 2019, and the review conducted revisited the testwork that has been previously reported in 2024.

The key concerns with the tests to date:

- The nominated P_{80} of 40 µm grind size has not been optimized based on capital and operating costs considerations. However, the basis for design has assumed that the previous basis will be valid. The use of vertical Metso tower mills generally requires Metso “Jar tests” and these have not been done to date.
- The oxygen requirements for the leach test and oxygen consumption have not defined in the historical RDi tests, while oxygen was added it was not recorded and historical data from previous General Gold operations suggest a high oxygen demand will be required. This, coupled with the presence of pyrrhotite in the ore, suggested that a higher oxygen demand will be required.

22.2.4 *Recovery Methods*

The proposed comminution circuit is considered suitable for treatment of the hard ore, however the overall circuit complexity and large number of equipment items to be commissioned and optimized increases risk of an extended ramp-up period to reach design capacity and the target metallurgical performance. The main areas of risk for the Project during ramp-up period are in the materials handling, crushing, ore sorting and grinding areas, specifically due to the number of unit operations and conveyors, transfer points, and wear areas located in the crushing and HPGR circuits.

The plant will use a hybrid leach-CIL circuit which is well known and common within the Australian mining industry. Risk is mainly associated with liberation of the fine gold within the Batman pit with gold particle size being <25 µm in size. A target grind P_{80} size of 40 µm was selected for the Project. A finer grind to further increase liberation would incur additional regrinding capital and operational costs, as well as increasing the slurry viscosity with an increasing finer grind particle size distribution.

The risk of not achieving the recoveries noted in this Technical Report will result in lower revenues and economic indicators. The additional metallurgical testing on fresh samples at different operating conditions will increase the level of confidence in the gold recoveries.

22.2.5 Water Management/Water Treatment

In review of the SWWB, geochemical modelling and the Water Discharge License, conclusions reached for the WTP 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.

22.2.6 Environmental, Permitting and Social Considerations

The JAAC has strong involvement in the planning 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 1989.

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 Certificates for Mt Todd covers the 1,392 km² of exploration licenses contiguous with the mining leases. An application for an AAPA Certificate authorizing work in additional areas associated with the 2024 FS and this Technical Report Summary has been initiated.

22.2.6.1 Community-based Staffing Discussion

Vista has worked closely with community and territory leaders in designing involvement for community-based project. Many mining operations in Australia are FIFO and it is the generally accepted approach to attraction and retention of a skilled workforce.

The Project site is readily accessible (approximately 250 km from Darwin) and conveniently located near well-established population centers. Project 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 expressed concerns about the influx of construction workers and Vista has selected to accommodate the majority of its construction workforce on the mine site for the Project development although during early works, notably construction camp construction there will be minimal accommodation requirements in the Katherine region.

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. Vista will seek to prioritize local engagement where skill development opportunities are available.

The full-time employees at the peak, during both construction and ongoing operations will be accommodated in purpose-built accommodation on the mine site with facilities typical of those provided by similar operations throughput Australia. It is anticipated that approximately 90% of the initial workforce will be engaged on a FIFO basis.

The workforce will be engaged from key points of hire that are typical of the skill sources in Australia. FIFO teams will commute via regular chartered flights to hub out of Kathrine and be transported by scheduled bussing arrangements to the accommodations and site facilities at the Project site.

22.2.7 *Project Capital and Operating Costs*

Project risks that could impact the capital cost include:

- Major changes in EPC scope of works.
- Weather.
- Permitting issues.
- Site geotechnical conditions and resulting earthworks and civil works designs.
- Incorrect trade contractor selection or contracting strategy.
- Inadequately understood environmental risks that require reactive design.
- Risk of project team integration problems/project management and site control issues.
- Input quality issues.
- Social impact of construction workforces.
- Import risks and unforeseen taxes.

22.3 Opportunities

22.3.1 *Mineral Resource Estimates*

- The Batman deposit potentially extends along strike both to the north and south.
- The South Cross Lode zone has the potential to be expanded to the north and potentially at depth.
- Additional deep infill drilling should be used to help define potential deep mineralization not detected by earlier drill holes.

- The Quigleys deposit is has the potential to be expanded through drilling and exploration.
- Additional exploration licenses are under Vista's control, with several showing geophysical and geochemical anomalies.
- Opportunity also exists in utilizing higher metal prices to create the Mineral Resource shell used to constrain the Mineral Resource.

22.3.2 Mining and Ore Mineral Reserves

There is a normal scope of work for an open pit mining project execution, there are some other areas of future works proposed, many related to opportunities identified during this Technical Report Summary for consideration.

The list below outlines those opportunities within open pit mining that are not yet defined to this Technical Report Summary standard and as such are not built into the FS economic model for this Technical Report Summary. These will be further evaluated and ruled in or out during the next stage of project development.

- Pit Slopes – In-conjunction with geotechnical recommendations from this Technical Report Summary, investigating the potential for steeper slopes based on improved geotechnical data and consideration of the life of the walls based on the mine schedule.
- Blasting Parameters – Updating blasting designs based on further blast fragmentation work, and new geotechnical and hydrogeological data to optimize drilling and blasting operations while ensuring adequate breakage of the rock is delivered with the high powder factor blasting currently assumed. Also include updating blasting designs for modified blasting for pit wall control.
- Mining Costs – Investigating opportunities to reduce mining costs through mining contractor discussions and negotiations.
- Lower grade material – Including lower grade material in the mine schedule that is in Mineral Resource Estimate and block model to potentially increase revenue with consideration of processing recovery and economics, particularly towards the end of the mine life. This would require full economic and technical evaluation.
- Mineral Resource Expansion – Drilling to expand and upgrade the Mineral Resource, particularly at the north-east side of the Batman pit and exploring new targets at depth.
- Stockpile Design and Operations – Developing a detailed plan for stockpile operations to maximize feed to the processing plant.
- Grade Control Plan – Refining drilling plans and sampling strategies to ensure proper ore definition and grade control.
- Waste Rock Characterization – Performing geochemical testing and risk assessments to address ARD issues, with a focus on improved identification of PAF/NAF due to current significant amount of unclassified material from an ARD perspective.

- Operational Readiness – Addressing procurement and personnel readiness, including the timely engagement of the mining contractor and ensuring the availability of skilled workers and required equipment.
- Underground Mining Potential – Significant Mineral Resources extend well below the current optimized life of mine pit limits. Further studies may identify opportunities to extend the mine life through underground mining operations. No such studies are known to the authors to have been completed to date.

22.3.3 Process Plant and Infrastructure

There exists opportunity to assess the following area to improve the Hill of Value of the Project such as:

- Utilize the existing buildings as offices, workshops – as an example the existing flotation building could be converted to a workshop or stores facility.

22.3.4 Recovery Methods

- Optimize the grind size and reduce the capital and operating costs by considering not only the difference in gold extraction effect on overall cash flow but also the associated operational, capital cost impacts.

22.4 Qualified Persons Opinions

This Technical Report Summary shows that the mine plan is technically achievable and economically viable taking into consideration all material modifying factors. The resultant Mineral Reserves is also reasonable and achievable.

The mining operations will be executed by a tier 1 Australian contract mining company, selected for its capability to manage large-scale operations and maintain high equipment availability. The contractor will utilize a fleet comprising 400 tonne class hydraulic excavators and 190 tonne class rigid-frame haul trucks, supported by ancillary equipment including loaders, dozers, graders, and water carts. The contractor will also provide site mining infrastructure and all personnel to operate and maintain all mining equipment, while ensuring its supervision and operations management.

Reasonable mine designs, mine production schedules, and mine costs have been developed for the Project. Costs for mining and mining infrastructure have been provided based on the analysis of FS mine schedule completed by a tier 1 Australian mining contractor and consider local site and NT requirements, and availability of resources such as equipment and labor. These mining costs have incorporated within the overall Project financial model.

Several opportunities and risk have been identified within this Technical Report Summary, which can be managed as the Project progresses its development through to execution and pre-production stages of development towards commencement of the mining operation.

The processing plant has been designed to treat 5.325 Mtpa (15 ktpd), as described in Section 14. During the initial phases of this Technical Report Summary, GRES reviewed the previous designs and raised several queries in relation to the ore sorting, grind size, recovery method and historical test work. GRES addressed these queries during this Technical Report Summary with some revised approaches.

Most of the capital and operating costs are within the front end of the plant. The plant has a restricted front end layout due to the limited available land and so is restricted in the ability to expand this area of the plant due to the waste dumps, water course and other restricted areas. The post grinding areas can be expanded relatively easily and will require some extensive demolition of the remaining existing facility to utilize this layout space.

When laying out this plant the remanent facilities on site were avoided as much as possible to minimize any major demolition and reduce front end capital costs.

23. RECOMMENDATIONS

All required work is complete for this Technical Report Summary and no additional work is necessary for this phase. This Technical Report Summary 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.

23.1 Mineral Resource and Exploration

The Geology and Mineral Resources of the Batman, Quigleys, and Heap Leach Pad areas are well understood by the Qualified Person. Accordingly, the following section provides recommendations intended to identify potential opportunities for further study or advancement. No budget has been assigned to these recommendations at this time.

- The Batman deposit potentially extends along strike to the north beyond the South Cross Lode. Step out drilling should be evaluated, and drilling should continue to define the northeastern portion of the deposit to define the extents.
- Additional deep infill drilling should be used to help define potential deep mineralization not detected by earlier drill holes.
- Infill drilling within and exploration drill holes along the trend of the Quigleys deposit is recommended.
- Additional drilling exploring the exploration licenses, following up on geophysical and geochemical anomalies.

23.2 Mining

All required work is complete for this Technical Report Summary and no additional work is necessary for this phase. This Technical Report Summary presents a Project that is ready to progress to the next phase. The next phase of the Project is detailed design, to be completed as part of the planned execution and pre-production stage.

For the mining area, there are two critical risks identified that are recommended to be a focus for future planned work. These are related to the geotechnical design parameters for pit walls, and the engagement of the mining contractor in sufficient time to be ready as per the current Project schedule.

The focus of the geotechnical future work is initially obtaining improved geotechnical engineering data, particularly in key rock structural domains, through planned drilling and laboratory testing. Geotechnical analysis and geotechnical assessment completed to focus on the pit wall slope design parameters and overall optimal configuration for mine design and operations considering ramp location, pit scheduling and life expectancy of walls for interim stages. This work will hopefully result in steeper pit slopes to assist Project economics but could results in flatter walls in some areas. Also, the objective is to overall improve the geotechnical assessment and overall risk profile for the Project.

For the recommendation regarding the mining contractor ensuring it is engaged with sufficient time to be ready as per the current projects schedule this is easily addressed by planning to have commercial discussions and negotiations as part of a formal tender process with mining contractors at least 12-18 months prior to required site mobilization. This should be easily implementable based on current Project timing, and recent experiences and interactions with three of Australia's leading Tier 1 mining contractors and other service provides as part of this Technical Report Summary, that are aware of the Project and enthusiastic to be involved.

The recommendations that follow are considered part of the detailed design phase and represent the normal progression of the Project from an FS towards construction. A specific budget has not been compiled for these tasks.

- Blasting Parameters – post the completion of initial execution work involving rock mass data for the geotechnical and hydrogeological studies, this information be utilized to update drill and blast designs and assumptions, including overall pattern designs, assumptions for percentage of trim blasts required, use of emulsion assumption for wet holes, etc. The current blasting patterns have been tightened up with an increase power factor to minimize possibility of oversize. With further data, the blasting patterns can be optimized to reduce both drilling and blasting costs.
- Stockpile design and operations – mine scheduling focused on the use of stockpiles to best deliver feed to the processing plant to maximize metal recovery and overall value. Within the detailed design execution works, further simulation of the stockpile operation and management will be required to ensure practical and safe operation of pre-crusher stockpiles to obtain mine planning requirements.
- Grade control plan – related to point above but more focused to the in-pit operation is further definition of drilling plans and sampling requirements to achieve required ore definition and overall grade control. An allowance in this Technical Report Summary has been developed and is reasonable for this level of Technical Report Summary assuming sampling of blast holes.
- Waste rock characterization – forward works for this area are detailed in other sections of this Technical Report Summary, in summary this forward work would include the following: further geochemical sampling and analytical testing program as well as an update of the ARD Classification Criteria to improve definition of PAF/NAF and reduce amount of unclassified material from an ARD perspective.

- Operational Readiness – forward works in this area focused on consideration of the broader Project approach required for attraction and retention of people, and strategic procurement of key consumables and equipment. This is addressed through implementing an engagement strategy, procurement and tender plan for the mining contractor, whom specialize is maintained both of their equipment procurement and people recruitment and attraction pipelines.
- Mineral Resource Estimation and Orebody Definition - Geological modelling underpins Mineral Resource estimation and mine planning. Inaccuracies in the interpretation of lithology, grade distribution, structural controls, or mineral continuity can result in misclassification of ore and waste, leading to suboptimal mine design and financial underperformance. Mitigation requires rigorous geological data collection, validation, and continuous model refinement throughout the life of mine.
- Cost Overruns and Budget Blowouts - Unexpected increases in construction, labor, fuel, or equipment costs can lead to budget overruns. These are particularly common in remote regions like the NT, where logistics and supply chain constraints can amplify cost variances. Early engagement with potential suppliers, service providers and partners that has already occurred within this Technical Report Summary assists to address this risk and needs to occur through the next stage of Project progression.
- Workforce Availability and Retention - Attracting and retaining skilled labor in remote areas is a persistent challenge. Competition from other mining operations and lifestyle factors can lead to high turnover and increased training costs. As outlined by Vista team members during this Technical Report Summary, they will continue to engage with both local - Katherine, and regional – Northern Territory communities to identify locals to employ and have part of the operations team which will assist with retention of people. This has also been discussed and is part of the mining contractor's operations philosophy and strategy.

Several opportunities and risks have been identified within this Technical Report Summary, which can be managed considering the recommendations outlined in this section as the Project progresses. Since mining aspects of the Project are considered to be well understood by the Qualified Person. No budget has been assigned to these recommendations at this time.

23.2.1 Pit Geotech Recommendations

As described in (WSP, 2025), based on the geotechnical assessment for the Batman pit feasibility study the following work is recommended:

- Conduct additional geological, geotechnical and hydrogeological investigation to obtain the required information to complete the final detailed designs for the Batman pit slopes. All this new information (including structural interpretations and groundwater modelling work) needs to be incorporated into slope stability models. The slope angles and geometries recommended by WSP in this Technical Report Summary are then to be reassessed. This work needs to be conducted before the final design stage for the Batman pit. The work can be summarized as follows:

- Develop adequate geological (lithological) 3D wireframes models. This information is required to refine geotechnical domains for the Batman pit.
- Develop a 3D geological large structure wireframe model. This geological structure model should be developed by an experienced and suitably qualified structural geologist using drillhole interpretations and multi-bench scale mapping data. Understanding the continuation/persistence of geological structures is critical for slope stability assessment purposes. It is recommended that the structural geologist also map all the exposed slopes to obtain geotechnical information relating to the geological structures including the persistence of these features.
- Design and conduct a geotechnical drilling program that includes at least six drill holes that are all geotechnically logged, drill core oriented, and scanned with acoustic televiewer (ATV)/optical televiewer (OTV). These drill holes should extend at least 50 m to 60 m behind the final LOM pit design slopes. Geotechnical core samples can then be selected for laboratory strength testing. It is expected that at least 100 core samples be collected for UCS testing at an accredited geomechanics laboratory. Point load testing at regular (5 m to 10 m) intervals along the core is recommended. This testing data can be used to develop a geotechnical laboratory testing database.
- Conduct a hydrogeological study for the Batman pit slopes. Develop a detailed hydrogeological numerical model to estimate groundwater drawn down surfaces for the pit slopes and determine possible inflows in the pit. The hydrogeological model should be calibrated with groundwater monitoring data and field testing as required. The effectiveness in the use of horizontal weep holes or vertical dewatering wells to reduce groundwater levels in the pit slopes should be assessed.

▪ Implement smooth wall blasting practices along final pit slopes. This is to reduce blast damage that can contribute to slope instability and rock falls. The use of pre-split and adequately designed trim blasts will be required. Berm crests should be protected to ensure berms remain effective in containing rock falls and failed material. Incorporate geotechnical information such as rock strength and geological structures in blast designs.

▪ Make budget allowance for the installation of ground support along pit slopes where geotechnical conditions require it. The requirements for ground support should be determined during mining. Ground support includes rock bolts, cables, mesh and rockfall catch fences to help stabilize local areas and manage the risk of rock falls. The northeast slope may require the installation of systematic ground support due to persistence of the bedding.

▪ Due to relatively more favorable geotechnical conditions, it is recommended to locate the pit access ramp along the northeast, east and south slopes.

- Manage the rock fall risk by the implementation of smooth wall blasting practices, maintaining wider berms and limiting crest damage, and where appropriate install rock fall fences. Further rockfall assessment is required ahead of the next study stage to help manage this risk for the Batman pit slopes.
- It is recommended to incorporate 25 m wide geotechnical/safety berms at regular 96 m vertical intervals in all the design sectors in the pit. Haul road/ramp can be used in lieu of geotechnical/ safety berm.
- Ongoing geotechnical risk management of the Batman pit during mining is recommended. This includes the establishment of a site geotechnical department that support daily mining operations. Routine geotechnical inspections, the installation of monitoring instrumentation such as slope movement prisms, borehole extensometers, wireline extensometer, inclinometers, photogrammetry and the use of radar systems should be considered. Groundwater monitoring includes the installation of borehole piezometers along the pit slopes and behind the pit crests. Develop a ground control management plan (GCMP) for Batman pit.
- It is recommended that the Batman pit slope angles and geometries are adjusted considering actual slope performance. Due to the expected variation in bedding and vein orientations bench face angles, and inter-ramp angles may have to be adjusted accordingly during further mining. Ongoing geotechnical and hydrogeological data collection and slope stability assessment will therefore be required during the mining of the Batman.

Based on the preceding technical assessment, the following tasks are identified as the recommended next steps. While no formal bidding process has commenced, cost estimates for these activities have been informed by Vista's previous experience with drilling campaigns and verbal consultation with reputable Australian experts. The anticipated budget required to complete the tasks outlined in Table 173 is approximately USD1.5 million. This budget has not been included in the financial analysis presented in Section 19.

Task Description
Geotechnical Drilling Campaign- 4 deep holes
Develop structural 3D model for the Batman pit
Develop Geotech/hydro site investigation program/plan
Conduct a desktop hydrogeological study for the Batman pit slopes
Geological/geotechnical field mapping of exposed slopes
Geotechnical site supervision, training and core logging with contracted geotech logging/sampling etc.
ATV/OTV downhole surveying/processing
Geotechnical laboratory program and testing ~ 100 UCS testing
Hydrogeological fieldwork and testing for the Batman pit
Develop/review 3D lithological wireframe model
Develop/review 3D geological large structure wireframe model

Task Description
Develop a detailed calibrated hydrogeological numerical model to estimate groundwater drawdown surfaces for the pit (design pit)

Table 173 Pit Geotechnical (main task for detailed engineering phase)

23.3 Process Plant

23.3.1 Metallurgical Test Work

- Additional drilling and sampling are recommended to enable fresh metallurgical samples to be generated for some confirmatory test work (approximate costs have been estimated at USD250,000).
- Additional grind sensitivity test work is recommended to support a trade-off study to more clearly define the optimum grind size based on economic considerations in addition to gold extraction (approximate costs have been estimated USD15,000).
- Metso "Jar tests" are recommended to confirm the secondary grinding mill power requirements and selection (approximate costs have been estimated at USD5,000).
- Oxygen uptake tests are recommended for confirmation of oxygen requirements (approximate costs (approximate costs have been estimated at USD5,000).
- Additional leach optimization and variability test work is recommended on fresh samples to provide more confidence in the metallurgical recoveries (approximate costs have been estimated at USD30,000).
- Vendor specific thickening test work to verify pre-leach thickener selection (approximate costs have been estimated USD15,000).
- Additional detailed analyses to better define the levels and deportment of cyanide soluble copper within the Batman pit (approximate costs have been estimated USD5,000).
- Verify the blending strategy to avoid high spikes in cyanide soluble copper plant feed when oxide/transition ore is scheduled for mining. (It is expected that this will be developed as part of the owner's development costs).

23.4 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. Specific budgets have not been prepared for these tasks.

- 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.

23.5 Regional Groundwater Hydrology

The following work is recommended with respect to groundwater hydrology and mine dewatering, a specific budget has not been prepared:

- 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.
- Since the update of the regional groundwater hydrology model required real operating data that can only be accurately collected during operations, no budget has been particularly assigned for this work during the detailed engineering phase.

23.6 Tailings Facility Design

The following studies and investigations are recommended for future phases of the Project. Results of these investigations and analyses may identify additional work items as detailed design progresses. A specific budget has not been prepared.

The current list of recommended work items includes the following:

- Additional laboratory testing of the waste rock is recommended, including, but not limited to, proctor compaction, hydraulic conductivity, and shear strength testing.

- A formal Failure Mode and Effects Analysis (FMEA) should be completed for both tailing storage facilities to determine the failure consequence category of both facilities. This analysis must be done before the detailed design phase to determine the design requirements each facility must achieve.
- A site-specific Seismic Hazard Analysis (SHA) is required for future planning to update or verify the seismic hazards associated with the loading conditions utilized in the slope stability analysis for the tailings storage facilities.
- Large scale consolidation tests should be conducted on bench-scale samples of the proposed process tailings to determine hydraulic conductivities as a function of effective stress.
- The condition of the existing toe drains, underdrains, and decant towers must be investigated to confirm their condition prior to re-commissioning of TSF1.
- A reclaim alternatives evaluation is recommended in case the decant system cannot function as intended, and
- After operations have restarted, and before the first upstream raise is constructed, a test pad should be built on the tailings to assess upstream raise constructability, settlement, pore pressure generation, and density.

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25. RELIANCE ON INFORMATION PROVIDED BY THE REGISTRANT

This Technical Report Summary has been prepared by GRES and the QP Technical Report Summary authors for Vista. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to GRES and the QP Technical Report Summary authors at the time of preparation of this Technical Report Summary;
- Assumptions, conditions, and qualifications as set forth in this Technical Report Summary; and
- Data, reports, and other information supplied by Vista and other third-party sources.

GRES and the QP Technical Report Summary authors have not researched property title or mineral rights for Vista as we consider it reasonable to rely on Vista legal counsel who is responsible for maintaining this information.

GRES and the QP Technical Report Summary authors have relied upon Vista and its management to prepare the market analysys, owner costs, closure and reclamation security bond, and the applicable taxes and royalties used in the economic analysis (Section 19) and in various subsections throughout the Technical Report Summary. GRES and the QP belive the data presentyed by Vista appear in line with the current market conditions.

Tetra Tech and the QP, relied upon the following experts to prepare portions of Section 17.

- 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 .
- Mt Todd Gold Mine – Status of Key Approvals, Permits, and Licences (21 February, 2025) was relied upon for permit/license status.

The QP Technical Report Summary authors have taken all appropriate steps, in their professional opinion, to ensure that the above information from Vista is sound.

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26. DATE AND SIGNATURE PAGE

The S-K 1300 Technical Report Summary - Mt Todd Gold Project – 15 ktpd Feasibility Study, Northern Territory, Australia" with an effective date of July 29, 2025 and an issue date of September 11, 2025, was prepared and signed by:

Name	Responsible for Sections	Signature	Date
TETRA TECH	Sections 1.3, 1.4, 1.5, 1.11, 3, 4, 5.1 to 5.4, 6, 7, 8, 9, 11, 13.3, 13.8.1,15.3,1 to 15.3.4, 17, 18.4.2 to 18.4.5, 18.9.2 to 18.9.5, 20, 21.3 to 21.7, 22.1.1, 22.1.6, 22.1.7, 22.2.1, 22.2.5, 22.2.6, 22.3.1,23.1, 23.4, and 23.5	/s/	September 11,2025
GRES	Sections 1.1, 1.2, 1.8, 1.9, 1.10, 1.12, 1.13, 1.14, 2, 5.5, 5.6, 10, 14, 15.2, 15.3.5 to 15.3.9, 15.4, 15.6 to 15.8, 16, 18.1, 18.3, 18.5, 18.6, 18.8, 18.10 to 18.13, 19, 21.1, 21.2, 22.1, 22.1.4, 22.2.3, 22.2.4, 22.2.7, 22.3.3, 22.3.4, 23.3 , 24, and 25	/s/	September 11,2025
Mining Plus	Sections 1.6, 1.7, 12, 13.1, 13.4 to 13.7, 13.10, 13.11, 15.1, 18.2, 18.7, 22.1.2, 22.1.3, 22.2.2, 22.3.2 and 23.2	/s/	September 11,2025
Deepak Malhotra, PhD, SME RM	Section 12.7	/s/	September 11,2025
TIERRA GROUP	Sections 13.8,13.9, 15.5, 18.4.1, 18.9.1, 22.1.5 and 23.6	/s/	September 11,2025
WSP	Sections 13.2 and 23.2.1	/s/	September 11,2025