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

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Prepared by

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ACRONYMS, ABBREVIATIONS AND SYMBOLS

acid base accounting	ABA
acid rock drainage and metal laden leachates	ARD/ML
Alice Springs Minerals	ALS
ampere	A
annual deduction	AD
annum (year)	a
Ammonium nitrate	AN
Ammonium nitrate emulsion	ANE
Ammonium nitrate fuel oil	ANFO
Australian Drinking Water Guidelines	ADWG
Australian and New Zealand Environment Conservation Council	ANZECC
Australian and New Zealand Marketing Academy	ANZMARC
Bateman Kinhill and Kilborne	ATWAL
Bateman Kinhill and Kilborne	BKK
Batman pit	BP
below ground surface	bgs
billion	B
billion tonnes	Bt
billion years ago	Ga
Bond ball mill work index	BWi
Canadian Institute of Mining, Metallurgy, and Petroleum	CIM
capital expenditure or capital expense	CAPEX
closed circuit television	CCTV
Canadian	CDN
Capital Cost Estimate	CCE
capital recognition deduction	CRD
carbon-in-pulp	CIP
carbon-in-leach	CIL
centimeters	cm
centipoise	mPa·s
chart of accounts	CoA
Crusing work index	CW_i
cubic centimeter	cm^3
cubic feet per second	ft^3/s
cubic foot	ft^3
cubic inch	in^3
cubic meter	m^3
cubic meter(s) per hour	m^3/hr
cubic yard	yd^3
day	d
days per week	d/wk
days per year (annum)	d/a

degree	°
degrees Celsius	°C
degrees Fahrenheit	°F
Department of Regional Development, Primary Industry, Fisheries and Resources	DRDPIFR
Department of Resources	DoR
diameter	ø
Diamond drillhole	DDH
Dissolved oxygen	DO
dollar (American)	US$
dollar (Australian)	AUD$
dollar (Canadian)	CDN$
drillhole	DH
dry metric ton	dmt
Drop Weight index	DW_i
dust suppression	DUST
eligible exploration expenditure	EEE
Environmental Impact Statement	EIS
Environmental Management Plan	EMP
equalization pond	EQP
exploration licenses	EL
Engineering procurement construction management	EPCM
Enhanced Factored Cost Estimate	EFCE
Feasibility Study	FS
foot	ft
Free In Store	FIS
gallons per minute (US)	gpm
geosynthetic clay liner	GCL
Gigajoule	GJ
gigawatt	GW
gold	Au
gram	g
gram per cubic meter	g/m^3
grams per liter	g/L
grams per tonne	g/t
greater than	>
gross realization	GR
heap leach pad	HLP
hectare (10,000 m^2)	ha
hertz	Hz
high pressure grinding rolls	HPGR
hour	hr
hours per day	h/d
hours per week	h/wk
hours per year	h/a
88.9 mm drill rod (outer diameter)	HQ
Heavy vehicles	HV

hanging wall	HW
inch	in
Intermediate bulk containers	IBC
Inductively Coupled Plasma Atomic Emission Spectroscopy	ICP
Inductively Coupled Plasma Optical Emission Spectroscopy	ICP-OES
Internal Rate of Return	IRR
interim trigger values	ITV
Jawoyn Association Aboriginal Corporation	JAAC
kilo (thousand)	k
kilogram	kg
kilograms per cubic meter	kg/m^3
kilograms per hour	kg/h
kilograms per square meter	kg/m^2
kilometer	km
kilometers per hour	km/h
kilo-ounce	koz
kilopascal	kPa
kilotonne	kt
kilovolt	kV
kilovolt-ampere	kVA
kilovolts	kV
kilowatt	kW
kilowatt hour	kWh
kilowatt hours per tonne	kWh/t
kilowatt hours per year	kWh/a
less than	<
Laboratory information system	LIMS
life of mine	LoM
liter	L
liters per minute	L/m
low grade ore stockpile	LGOS
low-permeability material	LPM
maintenance and repair contract	MARC
Mine Development Associates	MDA
megabytes per second	Mb/s
megapascal	Mpa
megavolt-ampere	MVA
megawatt	MW
meter(s)	m
meters above sea level	masl
meters below sea level	mbsl
meters per minute	m/min
meters per second	m/s
microns	µm
milligram	mg

milligrams per liter	mg/L
milliliter	mL
millimeter	mm
million	M
million bank cubic meters	Mbm^3
million bank cubic meters per annum	Mbm^3/a
million tonnes	Mt
million tonnes per year	Mtpy
Mineral License Number	MLN
Mining Management Plan	MMP
minute (plane angle)	‘
minute (time)	min
Mobile processing unit	MPU
Montgomery Watson Harza	MWH
month	mo
Mining & Resource Technology Pty Ltd	MRT
Mt. Todd Gold Project Preliminary Feasibility Study	MTPY
National Instrument	NI
National Health and Medical Research Council	NHMRC
Natural Resource Management Ministerial Council	NRMMC
Natural Resources, Environment, the Arts and Sport	NRETA
Net Present Value	NPV
Net Proceeds Return	NPR
Newtons per square millimeter	N/mm^2
Northern Australian Laboratories	NAL
Northern Territory	NT
NT Environmental Laboratories	NTEL
Notice of Intent	NOI
69.9 mm drill rod (outer diameter)	NQ
operating expenditure or operating expense	OPEX
operating costs	OC
open rotary holes	OP
optical ground wire	OPGW
ounce	oz
ounces/annum	oz/a
ounces/day	oz/d
Pascal	Pa
Pincock Allen and Holt	PAH
Porphyry copper gold	PCG
parts per million	ppm
parts per billion	ppb
percent	%
plant growth medium	PGM
potentially acid generating	PAG
pound(s)	lb
Power and Water Corporation	PWC

Preliminary Feasibility Study	PFS
Public Environmental Report	PER
Qualified Persons	QP
RKD (Company Name)	RKD
raw water dam	RWD
Resource Development Inc.	RDi
Sample name	RL
retention pond	RP
reverse circulation drilling method	RVC
revolutions per minute	rpm
runoff pond	RO
second (plane angle)	"
second (time)	s
semiautogeneous ball crusher	SABC
Successive alkalinity producing systems	SAPS
short ton (2,000 lb)	st
short tons per day	st/d
short tons per year	st/y
sodium metabisulfite	SMBS
SAG mill comminution	SMC
Society for Mining, Metallurgy, and Exploration, Inc.	SME
SPX company name	SPX
specific gravity	SG
square centimeter	cm^2
square foot	ft^2
square inch	in^2
square kilometer	km^2
square meter	m^2
tailings storage facility	TSF
three-dimensional	3D
tonne (1,000 kg) (metric ton)	t
tonnes per cubic meter	t/m^3
tonnes per day	tpd
tonnes per hour	tph
tonnes per year	t/a
The University of Newcastle Research Associates	TUNRA
Trigger value	TV
The Winters Company	TWC
Unconfined compressive strength	UCS
voice over Internet protocol	VoIP
volt	V
Waste Discharge License	WDL
waste rock dump	WRD
waste water treatment plant	WWTP
water treatment plant	WTP
weak acid dissociable	WAD

week	wk
weight/weight	w/w
Western Australia	WA

UNITS OF MEASURE

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

Weight

1 oz (troy) = 31.1035 g

Analytical Values

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

ABBREVIATIONS OF THE PERIODIC TABLE

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

1.0 SUMMARY

1.1 INTRODUCTION

Vista Gold Corp.(Vista) retained Tetra Tech along with Mine Development Associates (MDA), Resource Development Inc. (RDi), Proteus EPCM Engineers, a Tetra Tech Company (Proteus), and Power Engineers, Inc. to prepare this preliminary feasibility study (PFS) for its Mt. Todd Gold Project (the Project) in Northern Territory, Australia. The PFS evaluates the Base Case, a development scenario of a 50,000 tonne per day (tpd) processing facility. In addition, an Alternate Case was considered at 33,000 tpd with higher grades presented under *Section 24 – Other Relevant Data and Information*.

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

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

The primary purpose of this Technical Report is to provide updated material, scientific, and technical information based on additional data obtained from drilling conducted in 2012; and describe the Base Case mine plan at 50,000 tpd which contains ore mined from the Batman open pit plus the ore from the existing heap leach pad.

The Base Case mine plan contains 209.5 million tonnes (Mt) at 0.84 g Au/t of ore mined from the Batman open pit plus 13.4 Mt at 0.54 g Au/t of ore from the existing heap leach pad that will be processed through the mill at the end of the mine life. Together, an estimated 222.8 Mt of ore containing 5.901 million ounces (Moz) of gold at an average grade of 0.82 g Au/t are processed over the 13-year operating life. Total gold recovered is expected to be 4.808 Moz. Average annual gold production over the life of mine (LoM) is estimated to be 369,850 ounces. During operating years 1 through 5, gold production will average an estimated 481,316 ounces per year; year 1 production is estimated to be 580,472 ounces. Commercial production would begin following two years of construction and commissioning subject to receipt of all regulatory approvals.

Prior to the completion of this Technical Report, Tetra Tech, Inc. (Tetra Tech) was commissioned by Vista in September 2009 to prepare a PFS in accordance with NI 43-101 at an ore processing rate of 6.77 million tonnes per year (Mtpy) for the Project. The PFS study at 6.77 Mtpy was issued October 1,

2010. Subsequently, Vista commissioned a second PFS at an ore processing rate of 10.65 Mtpy, which was issued January 28, 2011.

Prior to these two PFS studies, an initial NI 43-101 Technical Report was completed on June 26, 2006; a Preliminary Economic Assessment Technical Report was completed on December 29, 2006; and an update to the resource report was completed in May 2008 and February 2009 based on additional exploration drilling completed by Vista during 2007 and 2008. The following is a list of reports Tetra Tech has completed on behalf of Vista:

- NI 43-101 Technical Report Resource Update Mt. Todd Gold Project Northern Territory, Australia (effective date: September 4, 2012, issued date: October 4, 2012);
- Amended and Restated NI 43-101 Technical Report Resource Update Mt. Todd Gold Project Northern Territory, Australia (effective date: September 6, 2011, issued date: April 11, 2012);
- NI 43-101 Technical Report, Resource Update, Mt. Todd Gold Project, Northern Territory, Australia (effective date: September 6, 2011, issued date: October 19, 2011);
- 10.65 MTPY Preliminary Feasibility Study, NI 43-101 Technical Report, Mt. Todd Gold Project, Northern Territory, Australia (effective date: October 1, 2010, issued date: January 28, 2011);
- Preliminary Feasibility Study NI 43-101 Technical Report Mt. Todd Gold Project, Northern Territory, Australia (effective date: October 1, 2010, issued date: October 1, 2010);
- Mt. Todd Gold Project, Updated Preliminary Economic Assessment Report, Northern Territory, Australia (effective date: May 15, 2009, issued date: June 11, 2009); and
- Mt. Todd Gold Project, Preliminary Economic Assessment Report, Northern Territory, Australia (effective date: December 29, 2006, issued date: December 29, 2006).

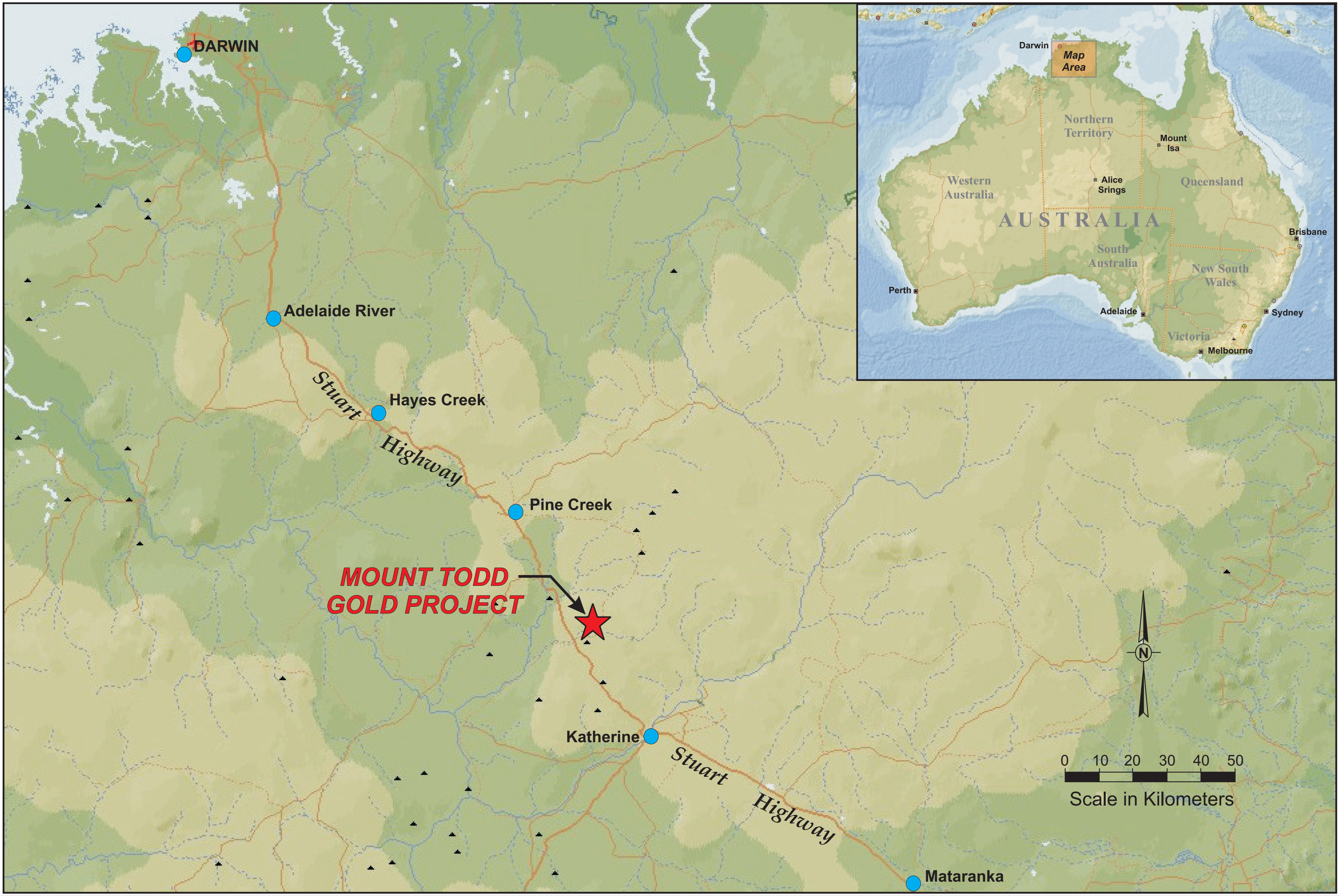
This PFS (Technical Report) notes material items which may, or may not, affect the market price of Vista's securities. Updated information contained herein includes scientific, technical and economic information deemed material to the Project.

1.2 LOCATION

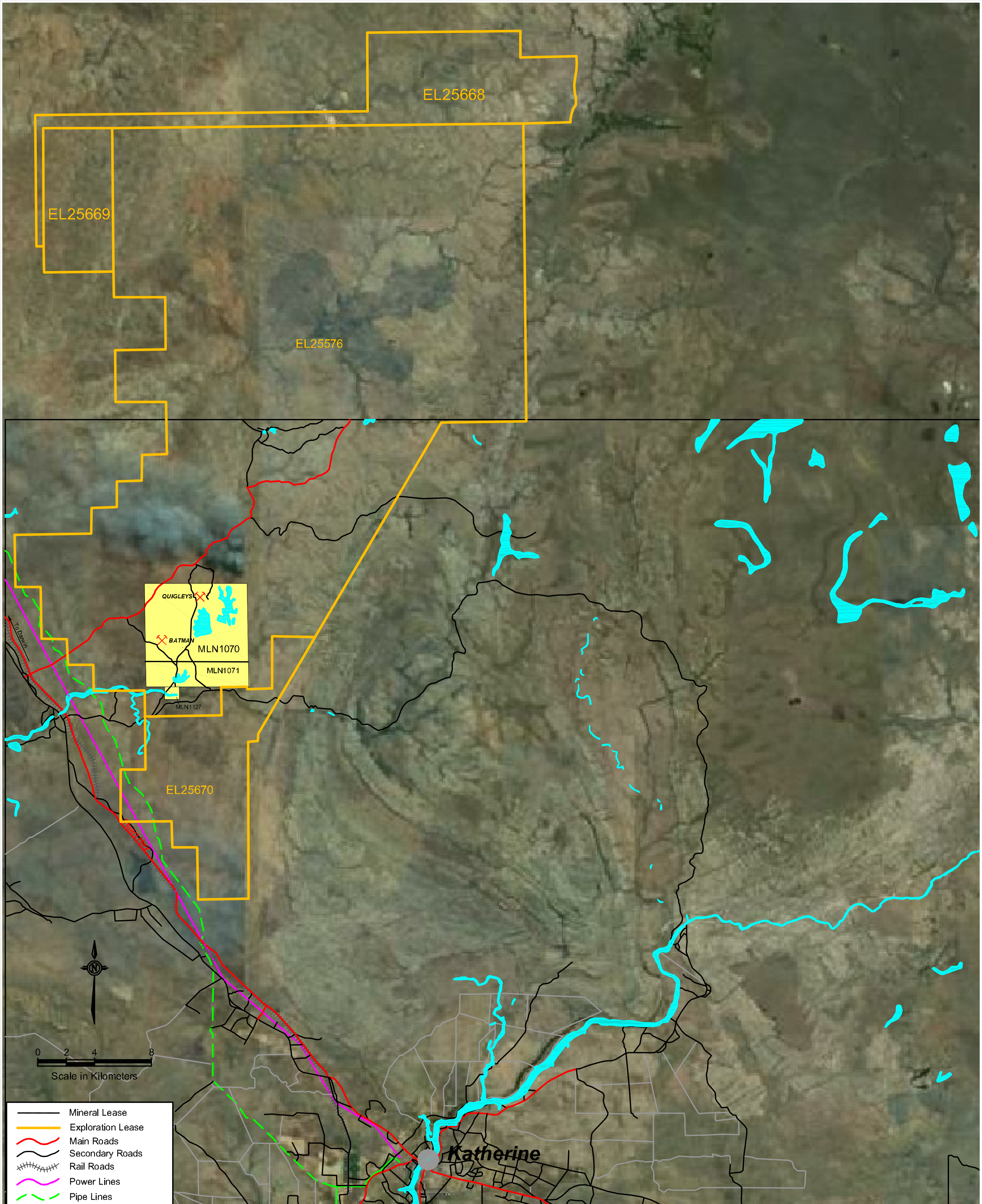
The Mt. Todd Project is located 56 kilometers (km) by road northwest of Katherine, and approximately 290 km southeast of Darwin in NT, Australia (**Figure 1-1)**. Access to the property is via high quality, two-lane paved roads from the Stuart Highway, the main arterial within the territory.

1.3 PROPERTY DESCRIPTION

Vista Australia holds three mineral licenses (MLN 1070, MLN 1071, and MLN 1127) comprising approximately 5,389 hectares (ha). In addition, Vista Australia controls exploration licenses (EL) EL 25668, EL 25669, EL 25576, EL 25670 and EL 28321 comprising approximately 134,838 ha. **Figure 1-2** illustrates the general location of the tenements and the relative position of the Batman deposit.



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Vista Gold Corp.*	Fig1-1.cdr	**Figure 4-1** **General Location Map** **Mt. Todd Gold Project**
	Project: Mt. Todd Gold Project	Project Number: 114-311156	
	Project Location: Northern Territory, Australia	Date of Issue: October 2010	



Issued by: TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Prepared for: **Vista Gold Corp.**	File Name: Fig1-2_4-2.dwg	**Concessions and Infrastructure Map**
	Project: Mt Todd Gold Project	Project Number: 114-311156	
	Project Location: Northern Territory, Australia	Date of Issue: September 2010	

1.4 Geology and Mineralization

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

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

The deposits are similar to other gold deposits of the PCG and are classified as orogenic gold deposits in the subdivision of thermal aureole gold style. The Batman deposit shares some characteristics with intrusion-related gold systems, especially in terms of the association of gold with bismuth and reduced ore mineralogies. This makes the 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 Mt. Todd area. Of greatest known economic significance are auriferous quartz-sulfide vein systems. These vein systems include the Batman, Jones, Golf, Quigleys and Horseshoe prospects, which occur within a north-northeast trending corridor, and are hosted by the Burrell Creek Formation. Tin occurs in a north-northwest trending corridor. The tin mineralization comprises cassiterite, quartz, tourmaline, kaolin, and hematite bearing assemblages, which occur as bedding 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 2200 m along strike, 400 m across dip and drill tested to a depth of 800 m. Drilling indicates the Batman mineralization to be open along-strike and down-dip.

1.5 Mineral Resource Estimate

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

1. Batman deposit;
2. Existing Heap Leach Pile; and
3. Quigleys deposit.

The table below illustrates the updated mineral resource estimate for the Project. The effective date of the Batman deposit resource estimate is March 18, 2013. The effective date of the heap leach resource estimate is May 29, 2013. The effective date of the Quigleys deposit is September 8, 2010.

The resource estimation of the Batman deposit is updated from the October 4, 2012 *Technical Report Resource Update* prepared by Tetra Tech. This report includes an estimate of gold contained in a historic heap leach pile adjacent to the Batman deposit. In addition, this report contains the resource estimation of the Quigleys deposit which has not been updated for this report. The resource update for the Quigley deposit was published in the "10.65 MTPY Preliminary Feasibility Study NI 43-101 Technical Report Mt. Todd Gold Project" prepared by Tetra Tech in January 2011. The Mt. Todd Gold Project updated resource estimates are shown in **Table 1-1,** grade tonnage curve for the measured and indicated resource for the Batman deposit is presented in **Figure 1-3**. The location for these resources is shown in **Figure 14-1**.

Table 1-1: Statement of Mineral Resources

Classification	Resource (kt)	Grade (g-Au/t)	Contained Au (koz)
MEASURED			
Batman Deposit	77,793	0.88	2,193
Quigleys Deposit	571	0.98	18
INDICATED			
Batman Deposit	201,792	0.80	5,209
Heap Leach Pad	13,354	0.54	232
Quigleys Deposit	6,868	0.82	181
MEASURED & INDICATED			
Batman Deposit	279,585	0.82	7,401
Heap Leach Pad	13,354	0.54	232
Quigleys Deposit	7,439	0.83	199
INFERRED			
Batman Deposit	72,458	0.74	1,729
Heap Leach Pad	0	0.00	0
Quigleys Deposit	11,767	0.85	320

Notes:
(1) Only measured and indicated resources were used to estimate proven and probable reserves, respectively.
(2) Batman and Quigleys Resources as of March 18, 2013. Heap Leach Resource as of May 29, 2013.
(3) Batman and Quigleys Resources at 0.40 g-Au/t cut-off grade.
(4) Measured and Indicated Mineral Resources are inclusive of Proven and Probable Mineral Reserves.
(5) Heap Leach Resource is average grade of heap, no cut-off grade is applied.
(6) Differences in the table due to rounding are not considered material.
(7) Rex Bryan, Tetra Tech, is the Qualified Person responsible for the Statement of Mineral Resources.

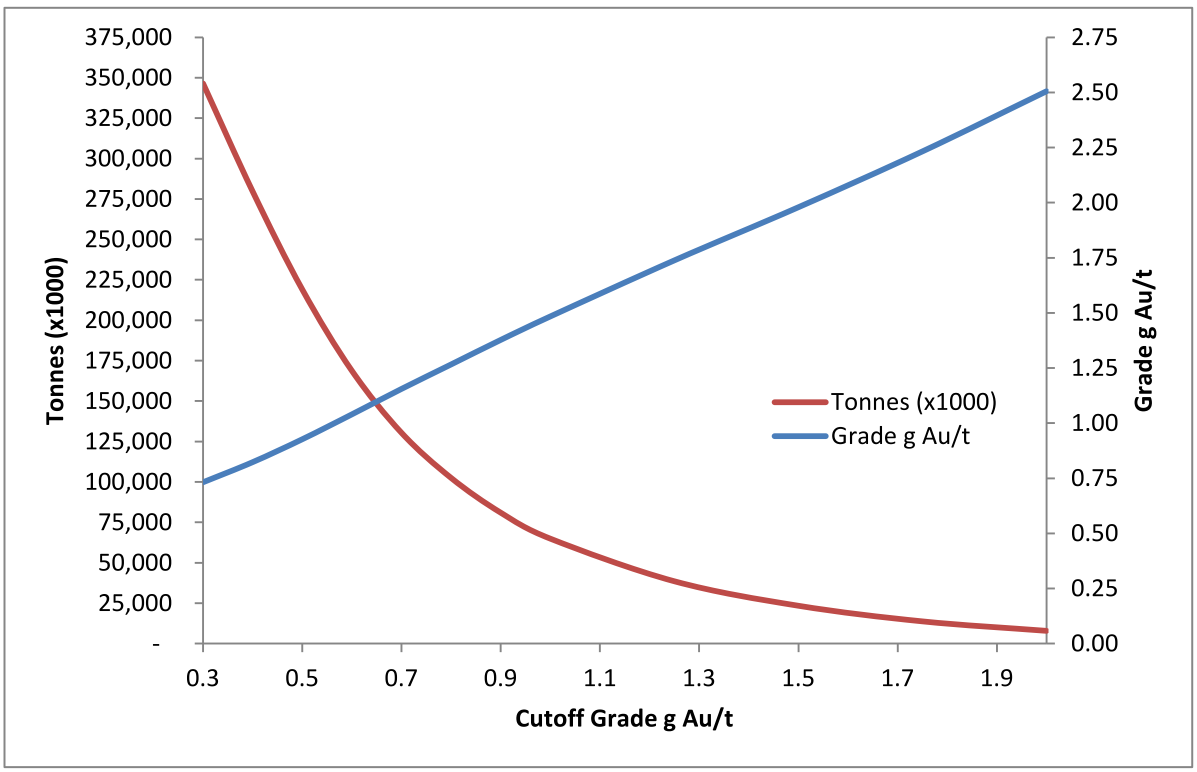


Figure 1-3: Grade Tonnage Measured and Indicated Resources – Batman Deposit

1.6 Mineral Reserve

Mine Development Associates (MDA) has used measured and indicated resources provided by Tetra Tech on March 18, 2013 to estimate mineral reserves. Pit optimization was done using Geovia's Whittle software to define pit limits with input for economic and slope parameters.

Optimization used only Measured and Indicated Resources for processing. All Inferred Resource was considered as waste.

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

The statement of mineral reserve estimates is shown in **Table 1-2**.

Table 1-2: Statement of Mineral Reserve Estimate

Classification	Reserve (kt)	Grade (g-Au/t)	Contained Au (koz)
Proven			
Batman Deposit	72,495	0.88	2,057
Probable			
Batman Deposit	136,955	0.82	3,612
Heap Leach Pad	13,354	0.54	232
Proven & Probable			
Batman Deposit	209,451	0.84	5,669
Heap Leach Pad	13,354	0.54	232

Notes:
(1) Statement of Mineral Reserves as of May 29, 2013.
(2) Mineral Reserves for Batman Deposit are reported using a 0.40 g-Au/t cut-off grade.
(3) Mineral Reserves for Heap Leach Pad have no cut-off grade applied.
(4) Based upon 50,000 tpd production rate and include mining recovery & dilution, and 82" LoM average CIL process recovery.
(5) Economic Analysis conducted only on Proven and Probable Mineral Reserves.
(6) Differences in the table due to rounding are not considered material.
(7) Thomas Dyer, Mine Development Associates, is the Qualified Person responsible for the Statement of Mineral Reserves for the Batman Deposit.
(8) Deepak Malhotra, Resource Development, Inc., is the Qualified Person responsible for the Statement of Mineral Reserves for the Heap Leach Pad.

1.6.1 Heap Leach Reserve

In addition to the mineral reserves assumed to be mined from the Batman open pit, the mine plan assumes that the 13.4 Mt of mineral reserves from the existing heap leach pad (HLP) will be processed through the mill at the end of the mine life.

Testwork indicated the following:

- Cyanidation leach tests on "as is" material on the heap will extract ± 30% of the gold.
- CIL 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.

1.7 Mining Methods

The Mt. Todd project has been planned as an open-pit truck and shovel operation.

1.7.1 Open Pit Mine

The Project is designed to be a conventional, owner-operated, large open-pit mining operation that will use large- scale mining equipment in a blast/load/haul operation.

A Base Case gold price of $1,360 per ounce has been assumed for use in mineral reserve estimation and mine planning. However, various gold prices from $300 to $2,000 per ounce were used to determine different optimized pit shells. This gold price is conservative based on the 3-year rolling average, which at the time of this report is near $1,545 per ounce Au, though the recent prices (May, 2013) have averaged $1,413. Note that while the gold price does significantly impact the cash-flow, it does not tend to have an impact on the resulting reserve definition because the pit optimizations used elevated cutoff grades.

Economic parameters are provided in **Table 1-3**. Initially, several iterations of pit optimizations were reviewed for the final determination of pit limits.

Table 1-3: Economic Parameters

Parameter	Base Case
Gold Recovery	82% Sulfide 78% Transition 78% Oxide
Payable Gold	99.9%
Overall Mining Cost	$1.90 per tonne
Processing Cost	$9.779 per tonne processed
Tailings	$0.985 per tonne processed
Water Treatment	$0.09 per tonne processed
Royalty	1% NPR (Jawoyn)

The mining costs used were varied by bench. An incremental cost of US$0.010 was added for each 6-m bench below the 145-m elevation. This represents the incremental cost of haulage for both waste and ore for each bench that is to be mined. A reference mining cost of US$1.64 was determined based on truck operating costs, truck cycle time to haul and return through a 6 m increase in differential elevation, and truck capacity. The reference mining cost was determined using first principles from previous studies.

Processing, tailings construction, tailings reclamation, waste dump rehabilitation, and general and administrative (G&A) costs were provided by Vista and based on previous studies and reviewed and approved for inclusion in this report. Note that site G&A costs are included with the processing costs for Whittle optimization. At Vista's request, MDA used a minimum cutoff grade of 0.40 g Au/t. This was done to maintain higher grades with respect to material allowed to be processed.

The breakeven and internal cutoff grade calculated using the economic parameters are shown in **Table 1-4**. The internal cutoff grade assumes that mining is constrained to an economic pit and does not include the mining cost.

Table 1-4: $1,360 Calculated Gold Price Cutoff Grades (g Au/t)

	Base Case		
	Sulfide	***Transition***	***Oxide***
Breakeven	0.36	0.38	0.39
Internal	0.31	0.32	0.32
Cutoff Grade Used	0.40	0.40	0.40

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

1.8 METALLURGY

The flowsheet consists of primary crushing, closed circuit secondary crushing, closed circuit tertiary crushing using high pressure grinding rolls (HPGRs), ball milling, cyclone classification, pre-leach thickening, leach and adsorption, elution electrowinning and smelting, carbon regeneration, tailings detoxification and disposal to conventional tailings storage facility (TSF). **Figure 1-4** provides the schematic diagram of the flowsheet.

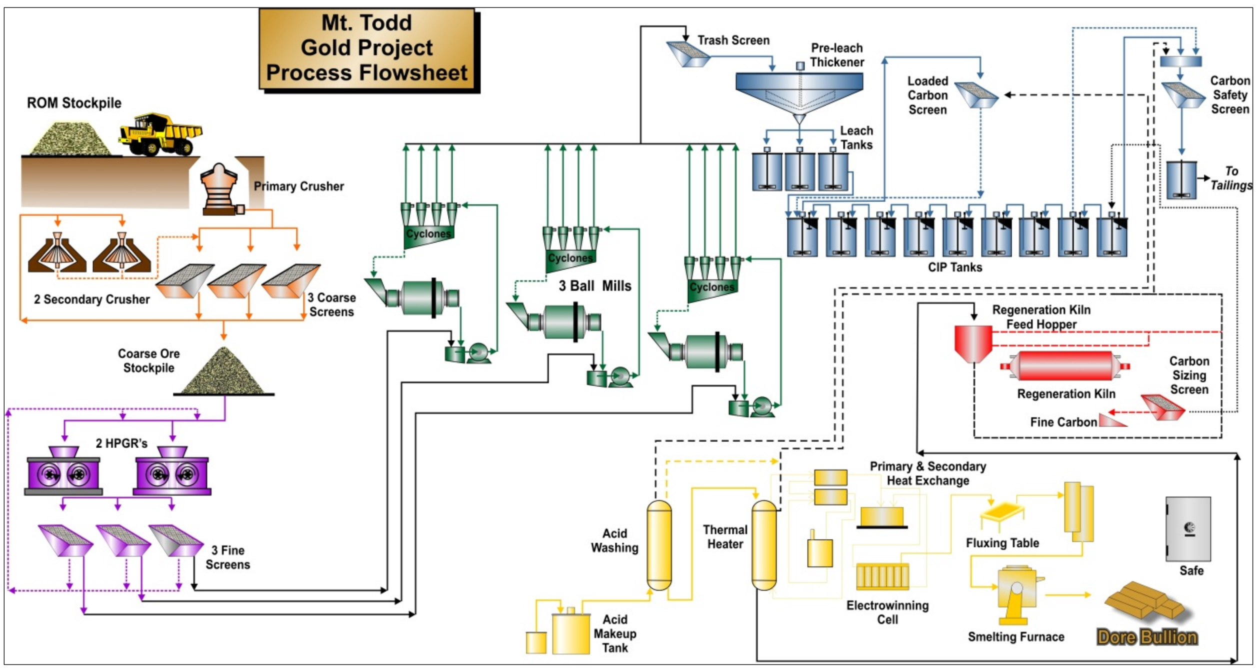


Figure 1-4: Mt. Todd Flowsheet

1.9 MINERAL PROCESSING

Detailed design criteria have been developed for the process plant. The nominal headline design criteria are listed in **Table 1-5** below.

Table 1-5: Headline Design Criteria

	Unit	Base Case
Annual Ore Feed Rate	Mt/a	17.75
Operating Days per Year	d/y	355
Daily Ore Feed Rate	tpd	50,000
Crushing Rate (6637 hours per year availability)	tph	2674
HPGR & Milling Rate (7838 hours per year)	tph	2264
Gold Head Grade	g/t	0.84
Copper Head Grade	%	0.055
Cyanide Soluble Copper	%	0.015
Ore Specific Gravity		2.76
Grind P_{80} to Leach	µm	90
Gold Recovery	%	82
Gold Production (nominal)	oz/d	1100
Gold Production (nominal)	oz/a	391,000

The testwork results collated from the 2011 and 2012 testing campaigns, together with the process design criteria, were utilized to develop the process flow sheet and mass balance.

1.10 Project Infrastructure

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

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

Planned infrastructure for the site includes the following:

- Ammonium Nitrate and Fuel Oil (ANFO) Facility;
- Mine Support Facilities (Heavy Vehicle (HV) Workshop, Lube Farm, Washdown and Tire Change, Warehouse, Fuel Farm, Mining Offices, Core Storage Facility);
- Heap Leach Facility;
- Accommodation Camp;
- Water Treatment Plant (WTP);
- Power Supply;
- Pit Dewatering;
- Mine Services;
- Communications;
- Gatehouse; and
- Future Tailings Dam/Decant Dam.

1.11 Market Studies and Contracts

1.11.1 Markets

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

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

Demand is presently high with prices showing remarkable increases during recent times. The 36-month average London PM gold price fix through May 2013 is US$1,545/oz.

1.11.2 Contracts

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

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

1.12 Social and Environmental Aspects

1.12.1 Existing Environmental and Social Information

A number of environmental studies have been conducted at the Mt. Todd Gold Project 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.

The following reports provide descriptions of the existing social and environmental conditions at the Mt. Todd Gold Project:

- Waste Discharge License 135 (EPA Northern Territory, 2005);
- Mt. Todd Environmental Management Services – Report 1: Environmental Assessment (MWH, 2006a);
- Mt. Todd Environmental Management Services – Report 2: Water Management (MWH, 2006b);
- Mt. Todd Gold Project Preliminary Economic Assessment (Gustavson, 2006);
- Environmental Management Plan (EMP) (Vista, 2007a);

- Mt. Todd Waste Discharge License Report, 2006 – 2007 (Vista, 2007b);
- Mt. Todd Water Treatment Plant Commissioning Report (Vista, 2009);
- Mt. Todd Blueprint Rehabilitation Strategy Report (Department of Regional Development, Primary Industry, Fisheries and Resources (DRDPIFR), 2008b);
- Mt. Todd Strategic Rehabilitation Reference Group: Status Update Papers in lieu of Meeting 11 (DRDPIFR, 2008c);
- Mt. Todd Mine Site Status Report, April 2008 to October 2008 (Vista, 2008); and
- Mt. Todd Project Draft Environmental Impact Statement (EIS) planned submittal mid-2013.

1.12.2 Social or Community Requirements

The Jawoyn people have strong involvement in the planning for the future of the Mt. Todd Project. Vista has a good relationship with the Jawoyn. Areas of Aboriginal Significance have been designated, and the mine plan has avoided development in these Restricted Works Areas.

1.12.3 Approvals, Permits and Licenses

The Project will require approvals, permits and licenses for various components of the Project. **Table 1-6** includes a list of approvals, permits, and licenses required for the project and their current status.

Table 1-6: Mt. Todd Permit Status

Approval/ Permit/ License	Current Status	Approval/ Permit License Date	Expiration Date
Environmental Impact Statement (EIS)	EIS currently being drafted. Following submittal of draft, requires approval by NT EPA using the EIS approval process	Draft EIS submission planned mid-2013	NA
Mining Management Act (or Plan) Approval	Notice of Intent submitted, further action on hold pending outcome of EIS	NA	NA
Heritage Act permit to destroy or damage archeological sites and scatters/ Aboriginal Areas Protection Authority Clearances	Authority Certificate Number 2011/15538 issued. This certificate defined Restricted Works Areas and granted select clearances to allow for initial investigations. Additional clearances will be required for further investigations as well as prior to disturbance associated with mine development.	Aboriginal Areas Protection Authority dated July 31, 2012	NA
Dangerous Goods Act (1988) permit for blasting activities	Waiting on final mine plan	NA	NA
Extractive Permit (under DME Guidelines) for development of borrow pits outside of approved mining areas	Would be required for PGM or LPM borrow areas. Permit application not yet in progress pending final selection of borrow areas	NA	NA
Waste Discharge License (under Section 74 of the Water Act 1992) for management of water discharge from the site	WDL 178-2 licensing discharge of waste water into the Edith River from the Mt. Todd mine site, granted with conditions	Feb. 5, 2013	Sept. 30, 2014
Waste water treatment system permits under Public Health Act 1987 and Regulations	May be required for the waste water treatment system for the construction and operations accommodation village. Permit application not yet in progress pending design and siting of accommodation village.	NA	NA
Approval to Disturb Site of Conservation Significance (SOCS)	Batman pit expansion will disturb SOCS as breeding / foraging habitat for the Gouldian finch, pending determination on EIS.	NA	NA

1.13 Capital and Cost Estimates

1.13.1 Capital Cost Estimates — Base Case

LoM capital cost requirements are estimated at US$1,405 million as summarized in **Table 1-7**. Initial capital of US$1,046 million is required to commence operations. At the end of operations, the Project will receive an estimated US$83 million credit for remaining on-site mine mobile equipment, a US$41 million salvage credit for process plant equipment and the return of the US$15 million reclamation bond.

Table 1-7: Capital Cost Summary (US$000s)

Area	Initial Capital	Sustaining	Credit/ Salvage	Total
Capitalized Costs	57,847	134,441	0	192,289
Mine & Process Mobile	138,895	151,460	(82,973)	207,381
CIL Process Plant	409,731	0	(40,651)	369,080
Tailings Dams	19,560	184,092	0	203,652
Power Supply	90,615	0	0	90,615
Water Supply	19,192	0	0	19,192
Owner's Costs	310,544	27,788	(15,000)	323,332
Total Capital	**1,046,383**	**497,782**	**(138,624)**	**1,405,540**

1.13.2 Operating Cost Estimates— Base Case

LoM operating costs requirements are estimated to be US$16.32/t-milled as summarized in **Table 1-8**.

Table 1-8: LoM Operating Costs

Opex Summary	US$/t-mined	US$/t-milled	Total (US$000s)
Open Pit Mining	2.005	6.946	1,547,607
CIL Process Plant	-	8.780	1,956,195
Water Treatment Plant	-	0.096	21,313
G&A	-	0.495	110,310
Total Opex Summary	**-**	**16.317**	**3,635,425**

1.13.3 Capital Cost Estimates — Alternate Case

LoM capital cost requirements for the Alternate Case have been estimated using the similar parameters as used for the Base Case. The Alternate Case results are estimated at US$972 million as summarized in **Table 1-9**. Initial capital of US$761 million is required to commence operations. At the end of operations, the model assumes the Project will receive a US$47 million credit for remaining on-site mine mobile equipment, a US$30 million salvage credit for process plant equipment and the return of the US$15 million reclamation bond.

Table 1-9: Capital Cost Summary, Alternate Case

Capex Summary	Initial Capital	Sustaining Capital	Salvage Credit	Total
Capitalized Costs	24,303	132,254	0	156,557
Mine & Process Mobile	77,336	73,150	(46,539)	103,947
CIL Process Plant	310,308	0	(30,324)	279,984
Tailings Dams	19,373	85,667	0	105,040
Power Supply	64,307	0	0	64,307
Water Supply	11,136	0	0	11,136
Owner's Costs	254,429	12,027	(15,000)	251,455
Total Capex Summary	**761,192**	**303,097**	**(91,862)**	**972,427**

1.13.4 Operating Cost Estimates — Alternate Case

LoM operating cost estimates are summarized in **Table 1-10**. The operating costs will average US$15.99 over the LoM.

Table 1-10: Operating Cost Summary, Alternate Case

Opex Summary	US$/t-mined	US$/t-milled	Total (US$000s)
Open Pit Mining	2.037	5.485	678,670
CIL Process Plant	-	9.507	1,176,245
Water Treatment Plant	-	0.083	13,983
G&A	-	0.739	91,431
Jawoyn Royalty	-	0.339	41,921
Refining Costs	-	0.075	9,234
Power Credit	-	-0.235	(106,052)
Total Opex Summary	**-**	**15.992**	**1,905,432**

1.14 Financial Analysis

1.14.1 Financial Analysis — Base Case

Economic results are summarized in **Table 1-11**. The analysis suggests the following conclusions, assuming a 100% equity project at a gold price of US$1,450:

- Mine Life: 13 years;
- Pre-Tax $NPV_{5\%}$: US$1,094 million, IRR: 22%;
- Post-Tax $NPV_{5\%}$: US$591 million, IRR: 16%;
- Payback (Post-Tax): 3.5 years;
- NT Royalty Taxes Paid: US$270 million;
- Australian Income Taxes Paid: US$462 million; and
- Cash costs (including Royalty): US$773/oz-Au.

Table 1-11: Technical-Economic Results

Cash Flow Summary	LoM Cost (US$000s)	Unit Cost US$/t-milled	Unit Cost US$/oz-Au
Gold Produced	4,808	-	-
Gold Price	1,450	-	-
Gold Sales	**6,971,674**	**31.29**	**1,450.00**
Refinery Costs	(15,331)	(0.07)	(3.19)
Net Smelting Return	**6,956,343**	**31.22**	**1,446.81**
Jawoyn Royalty	(69,717)	(0.31)	(14.50)
Gross Income from Mining	**6,886,626**	**30.91**	**1,432.31**
Open Pit Mine	(1,547,607)	(6.95)	(321.88)
CIL Process Plant	(1,956,195)	(8.78)	(406.86)
Water Treatment Plant	(21,313)	(0.10)	(4.43)
G&A	(110,310)	(0.50)	(22.94)
Operating Costs	**(3,635,425)**	**(16.32)**	**(756.11)**
Power Sales Credit	93,754	0.42	19.50
Cash COGS	**(3,557,002)**	**(15.96)**	**(739.80)**
Operating Margin	**3,344,955**	**15.01**	**695.70**
Capitalized Costs	(192,289)	(0.86)	(39.99)
Mine & Process Mobile Capital	(290,355)	(1.30)	(60.39)
CIL Process Plant	(409,731)	(1.84)	(85.22)
Tailings Dams (4400)	(203,652)	(0.91)	(42.36)
Power Supply (4200)	(90,615)	(0.41)	(18.85)
Water Supply & Treatment(4100)	(19,192)	(0.09)	(3.99)
Owner Costs	(338,332)	(1.52)	(70.37)
Salvage	138,624	0.62	28.83
Capital Costs	**(1,405,540)**	**(6.31)**	**(292.33)**
Pre-Tax Cash Flow	**1,939,415**		
$NPV_{5\%}$	**1,093,859**		
IRR	**22%**		
Payback (years)	**2.85**		
Post-Tax Cash Flow	**1,230,523**		
$NPV_{5\%}$	**591,318**		
IRR	**16%**		
Payback (years)	**3.50**		

1.14.2 Financial Analysis — Alternate Case

Economic results are summarized in **Table 1-12**. The analysis suggests the following conclusions, assuming a 100% equity project at a gold price of US$1,450:

- Mine Life: 11 years;
- Pre-Tax $NPV_{5\%}$: US$1,222 million, IRR: 22%;
- Post-Tax $NPV_{5\%}$: US$440 million, IRR: 17%;
- Payback (Post-Tax): 3.2 years;
- NT Royalty Taxes Paid: US$173 million;
- Australian Income Taxes Paid: US$307 million; and
- Cash costs (including Jawoyn Royalty): US$684/oz-Au.

Table 1-12: Economic Results, Alternate Case

Cash Flow Summary	LoM Cost (US$000s)	Unit Cost $/t-milled	Unit Cost $/oz-Au
Gold Produced	2,891	-	-
Gold Price	1,450	-	-
Gold Sales	**4,192,069**	**33.88**	**1,450.00**
Refinery Costs	(9,234)	(0.07)	(3.19)
Net Smelting Return	**4,182,834**	**33.81**	**1,446.81**
Jawoyn Royalty	(41,921)	(0.34)	(14.50)
Gross Income from Mining	**4,140,914**	**33.47**	**1,432.31**
Open Pit Mine	(678,670)	(5.49)	(234.75)
CIL Process Plant	(1,176,245)	(9.51)	(406.85)
Water Treatment Plant	(13,983)	(0.11)	(4.84)
G&A	(91,431)	(0.74)	(31.63)
Operating Costs	**(1,960,329)**	**(15.84)**	**(678.06)**
Power Sales Credit	106,052	0.86	36.68
Cash Cost of Goods Sold	**(1,863,511)**	**(15.06)**	**(644.57)**
Operating Margin	**2,286,637**	**18.48**	**790.93**
Capitalized Costs	(156,557)	(1.27)	(54.15)
Mine & Process Mobile Capital	(150,486)	(1.22)	(52.05)
CIL Process Plant	(310,308)	(2.51)	(107.33)
Tailings Dams (4400)	(105,040)	(0.85)	(36.33)
Power Supply (4200)	(64,307)	(0.52)	(22.24)
Water Supply & Treatment(4100)	(11,136)	(0.09)	(3.85)
Owner Costs	(266,455)	(2.15)	(92.16)
Salvage	0	0.00	0.00
Capital Costs	**(1,064,289)**	**(8.60)**	**(368.13)**
Pre-Tax Cash Flow	**1,222,348**		
$NPV_{5\%}$	**776,874**		
IRR	**22%**		
Payback (years)	**2.80**		
Post-Tax Cash Flow	**849,794**		
$NPV_{5\%}$	**440,183**		
IRR	**17%**		
Payback (years)	**3.20**		

1.15 Conclusions and Recommendations

1.15.1 Feasibility Study

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

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

1.15.2 Geology and Resources

Conclusions

- The Mt. Todd Project is situated within the southeastern portion of the Early Proterozoic Pine Creek Geosyncline which is comprised of the Burrell Creek Formation, the Tollis Formation, and the Kombolgie Formation.
- Gold mineralization in this area is constrained to a single mineralization event and the deposits are classified as orogenic gold deposits in the subdivision of thermal aureole gold style. The Batman deposit has characteristics of an intrusion related gold system making it the primary resource.
- The Batman deposit is defined by 7.4 million ounces of gold within 279.6 Mt of measured and indicated resource at an average grade of 0.82 g Au/t and a cutoff grade of 0.4 g Au/t.
- The progression of the Batman deposit resource at a cutoff grade of 0.4 g Au/t since the January 2011 PFS is summarized below:

Report	Category	Tonnes (x1000)	Average Grade g Au/t	Total Au Ounces (x1000)	Increase in ounces of Au from Previous Report
March 2013	Measured & Indicated	279,585	0.82	7,401	6%
September 2012	Measured & Indicated	261,400	0.83	7,007	17%
September 2011	Measured & Indicated	222,022	0.84	5,987	17%
January 2011 PFS	Measured & Indicated	190,939	0.84	5,125	-

Tonnage, grades and totals may not total due to rounding.
All estimated resources are shown using a 0.4 g Au/t cutoff grade.
Vista's first mineral resource estimate for the Batman deposit.

Recommendations

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

The estimated budget for drilling within the mining licenses is US$500,000-1,000,000 and US$500,000-1,000,000 for drilling on the exploration licenses.

1.15.3 Mineral Reserve and Mine Planning

- The Mt Todd Proven and Probable reserves have been defined based on optimization of mining plans using a gold price of $1,360 per ounce. The Proven and Probable reserves were used to create a production schedule for mining, and a positive cash-flow analysis has been done based on the production schedule by Tetra Tech. This establishes the reserves as having reasonable economics with respect to the statement of reserves under NI 43-101 regulations.
- Mine production constraints were imposed to ensure that mining wasn't overly aggressive with respect to the equipment anticipated for use at Mt. Todd. The schedule has been produced using mill targets and stockpiling strategies to enhance the project economics. The constraints and limits used are reasonable to support the project economics which are used to justify the statement of reserves.
- Pit designs were developed using six-meter benches for mining. This corresponds to the resource model block heights, and MDA believes this to be reasonable with respect to dilution and equipment anticipated to be used in mining. In areas where the material is consistently ore or waste so that dilution is not an issue, benches may be mined in 12-m heights.

1.15.4 Mineral Processing

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

- Mt. Todd (Batman) ore is one of the hardest and most competent ore types processed for mineral recovery. The most energy efficient comminution circuit has been determined to be the sequence of primary crushing, closed circuit secondary crushing, and closed circuit HPGR tertiary crushing followed by ball milling.
- The ore is free-milling, has no preg robbing problem, and is amenable to gold extraction by conventional cyanidation processes.
- The ore has relatively high specific cyanide consumption, determined to be 0.77 kg of sodium cyanide per tonne of ore. This is largely due to the presence of sulfides and cyanide consuming copper.
- The ore requires a P_{80} grind of 90 µm and 24 hr leach residence time to achieve a nominal 82% gold recovery (81.7% net of solution loss) from a global head grade of 0.86 g Au/t.

The equipment selection criteria for the large scale 50,000 tpd operation has received considerable interaction with specialist vendors to the point where there is a reasonably high degree of confidence in selected technology and process units at this PFS stage. The recommended flowsheet for FS consists of primary crushing, closed circuit secondary crushing, closed circuit tertiary crushing using HPGRs, ball milling, cyclone classification, pre-leach thickening, leach and adsorption, elution electrowinning and smelting, carbon regeneration, tailings detox and disposal to conventional tailings storage facility.

1.15.5 Infrastructure

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

- Heavy cranage is allowed for all lifts greater than 50t.
- All bulk transport will be weighed.
- Site wide communication is based on a 50m tall communication tower that will support eight (8) channels.

1.15.6 Environmental and Social Impacts

Conclusions

A number of environmental studies have been conducted at the Mt. Todd 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.

A draft EIS will be completed by mid-2013. Following submittal of the draft, approval by the Northern Territories EPA is required in accordance with the EIS approval process.

The Jawoyn people have strong involvement in the planning of the Mt. Todd Project. Areas of Aboriginal Significance have been designated, and the mine plan has avoided development in these Restricted Works Areas

Recommendations

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

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

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

1.15.7 Results of the Site Wide Water Balance Model

Conclusions

- The WTP rate of 500 m^3/hr and EQP sizing for 5 days of storage was determined to be appropriate for the 50,000 tpd production process water requirements.
- The greatest amount of make-up water required from the RWD was quantified as 24,409 m^3/day. RWD requirements were found to be the most dependent upon TSF decant volumes, with the greatest need occurring during the dry season when decant rates were low.

- The WRD pond was typically observed to overtop less than 0.3% of the time during the 12 year simulation.[1] RP2 and RP5 storage may be optimized.

Recommendations

Recommended model improvements include:

- The site wide water balance model is dependent upon the TSF water balance model, which provides decant water to the process facility, and the vadose and seepage models, which characterize seepage through the various rock piles on site (WRD, LGOS and HLP). As such, completion of these models to the greatest detail practicable affects the overall quality of the site wide water balance model results.
- Clarification on the elution/potable water requirements is needed. The current model relies on an assumed ratio of the process demand of 16%.
- Further investigation of the adequacy of RP 1 storage capacity is recommended, particularly within the early stages of the LoM when a larger fraction of the catchment reports to the pond.
- Stage-storage relationships of RWD will be included in the DFS model such that it may be modeled as a reservoir, as opposed to an infinite source.

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

1.15.8 Groundwater Hydrology and Mine Dewatering

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

- Calibration of the regional groundwater flow model should be completed, 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 completed with the calibrated model used as its basis. Output from the post-mining model should be incorporated into any geochemical modeling of post-mining pit lake formation and geochemistry.
- The pit dewatering system design and cost estimates should be refined to include groundwater inflows estimated with the calibrated model.
- A tradeoff study should be conducted to identify the optimum balance between the cost of the dewatering pumping equipment and the cost of mine pit floor and bench inaccessibility while the pit is being pumped dry after storm events.

The estimated budget for this work is US$300,000.

1.15.9 Geotechnical Investigation

Conclusions

There are no conclusions with regard to the geotechnical investigation.

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

Recommendations

Future geotechnical work is suggested in the following areas:

Grinding

This is a large vibrating structure which should be founded in rock rather than on fill to reduce dynamic effects. An accurate rock level is required to confirm foundation design and accurately estimate required concrete quantities and rock excavation. Existing structure concrete slabs are located directly over the new mill location, so it is recommended that additional new test pits around all four sides be undertaken to allow interpolation.

Leach/CIP

Large tanks to be constructed in this area, with high foundation bearing pressure. Variation in rock level will impact on differential settlement which needs to be considered. Recommend at least four additional test pits to evaluate variance in rock levels in north-south and east-west directions.

Stockpile & Reclaim

No test pit data anywhere near this area (nearest test pit is more than 200 m away). Steeply sloping ground surface exists (in excess of 5 m variation in ground level across the reclaim tunnel) so there may be considerable variation in rock levels which will effect potential settlement due to stockpile surcharge and required excavation for the concrete vault and tunnel. Recommend a new borehole on the high side (uncertain whether this high side bench is fill material, and therefore whether rock would be encountered within the limits of an excavator for a test pit) and a new test pit on low side, to determine rock levels.

Coarse Screening

Vibrating structure which should be founded on rock rather than on fill to reduce dynamic effect.
An accurate rock level is required to confirm foundation design and accurately estimate required concrete quantities and rock excavation. Existing test pits are on one side only so unable to interpolate between existing pits, the nearest existing test pit is approximately 100 m to the north so recommend a new test pit at this location.

HPGR

Vibrating structure which should be founded on rock rather than on fill to reduce dynamic effect.
Therefore need to know rock level to confirm foundation design and accurately estimate required concrete quantities and rock excavation. Nearest existing test pit is approximately 70 m away so recommend new test pit at this location.

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

1.15.10 Process Operating Costs

Two major items incurring operating costs are steel balls as grinding media and cyanide as leaching reagent. Together these items make up 65% of the plant consumables operating costs. The FS should investigate options for reducing the consumption rate and the unit costs for these consumables.

1.15.11 Geochemical Analyses

Geochemical characterization will be updated to reflect the designations of Potentially Acid Forming, Potentially Acid Forming-Low Capacity, Non Acid Forming, Acid Consuming and Uncertain in accordance with DITR (2007) guidelines.

2.0 INTRODUCTION

2.1 BACKGROUND INFORMATION

Vista Gold Corp. (Vista) retained Tetra Tech along with Mine Development Associates (MDA), Resource Development Inc. (RDi), Proteus EPCM Engineers, a Tetra Tech Company (Proteus), and Power Engineers, Inc. to prepare this preliminary feasibility study (PFS) for its Mt. Todd Gold Project (the Project) in Northern Territory, Australia. The PFS evaluates the Base Case, a development scenario of a 50,000 tonne per day (tpd) processing facility. In addition, an Alternate Case was considered at 33,000 tpd with higher grades presented under *Section 24 – Other Relevant Data and Information*.

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

- A 33,000 tpd processing facility as compared to a 50,000 tpd facility with associated lower mining rates and a smaller mining fleet; and
- Pit design is based on a pit shell calculated at US$925/oz-Au vs. US$1,360/oz-Au in the 50,000 tpd project and the application of a higher cut-off grade (0.45 g Au/t vs. 0.40 g Au/t); and
- Shorter operating life for the 33,000 tpd Alternate Case.

The Base Case includes:

- Estimated proven and probable reserves of 5.90 Moz of gold (223 Mt at 0.82 g Au/t) at a cut-off grade of 0.40 g Au/t;
- Average annual production of 369,850 ounces of gold per year over the mine life, including average annual production of 481,316 ounces of gold per year during the first five years of operations;
- LoM average cash costs of US$773 per ounce, including average cash costs of US$662 per ounce during the first five years of operations;
- A 13 year operating life;
- After-tax $NPV_{5\%}$ of US$591.3 million and internal rate of return (IRR) of 15.9% at $1,450 per ounce gold prices, increasing to US$876.6 million and 21.1%, respectively, at US$1,600 per ounce gold prices; and
- Initial capital requirements of US$1,046 million.

The Alternate Case discussed in Section 24 includes:

- Estimated proven and probable reserves of 3.56 Moz of gold (124 Mt at 0.90 g Au/t) at a cut-off grade of 0.45 g Au/t;
- Average annual production of 262,826 ounces of gold per year over the mine life, including average annual production of 294,502 ounces of gold per year during the first five years of operations;
- LoM average cash costs of US$684 per ounce, including average cash costs of US$676 per ounce during the first five years of operations;
- An 11 year operating life;
- After-tax $NPV_{5\%}$ of US$440.2 million and IRR of 16.9% at US$1,450 per ounce gold prices, increasing to US$615.6 million and 21.4%, respectively, at US$1,600 per ounce gold prices; and
- Initial capital requirements of US$761 million.

2.2 Terms of Reference and Purpose of the Report

This report was prepared as a NI 43-101 Technical Report for Vista by Tetra Tech. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in Tetra Tech's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report.

This report provides mineral resource and mineral reserve estimates, and a classification of resources and reserves in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Mineral Reserves: Definitions and Guidelines, November 27, 2010 (CIM).

2.3 Sources of Information

The principal technical documents and files relating to the Mt. Todd Gold Project were used in the preparation of this report and are listed in Section 27.1.

2.4 Units of Measure

The metric system has been used throughout this report. Tonnes are metric of 1,000 kg, or 2,204.6 lb. Gold is reported in troy ounces, equivalent to 31.1035 g. All currency is in Q4 2012 US dollars (US$) unless otherwise stated.

2.5 Detailed Personal Inspections

1) Rex Bryan visited and inspected the property from September 12th, 2011 to September 14th, 2011 and February 6th, 2013 to February 8th, 2013. Dr. Bryan spent time on site and reviewed the current database and archived supporting material, core logging, sampling procedures, handling and security measures, QA/QC procedures and inspected modern and historically collected core.

2) Thomas Dyer visited and inspected the subject property during March, 2011. Mr. Dyer toured the site along with geotechnical consultants and reviewed the pit, waste dump, tailings facility, and resource drilling sites. Previous mine production records held on site were also reviewed.

3.0 RELIANCE ON OTHER EXPERTS

The Consultants used their experience to determine if the information from previous reports was suitable for inclusion in this Technical Report and adjusted information that required amending. This report includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the Consultants do not consider them to be material.

Tetra Tech relied upon the following experts:

- Mt. Todd Environmental Management Services TSF Scoping Study (December 2006) prepared by MWH, to estimate the current tailings volume in TSF 1 (Tetra Tech, Section 18.3).
- Draft Environmental Impact Statement prepared by GHD (Tetra Tech, Section 20, 20.1, 20.2, 20.3).
- Information in the Memorandum titled "Mt. Todd Gold Project: Batman pit Slope Design Guidance in Support of the Definitive Feasibility Study" prepared by Golder Associates, September 13, 2011 (MDA, Section 15.1).

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

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

4.2 PROPERTY DESCRIPTION

Vista Australia is the holder of three mineral licenses (ML) MLN 1070, MLN 1071, and MLN 1127 comprising approximately 5,389 hectares (ha). In addition, Vista Australia controls exploration licenses (EL) EL 25668, EL 25669, EL 25576, EL 25670 and EL 28321 comprising approximately 134,838 ha. **Figure 4-2** illustrates the general location of the tenements and the position of the Batman deposit.

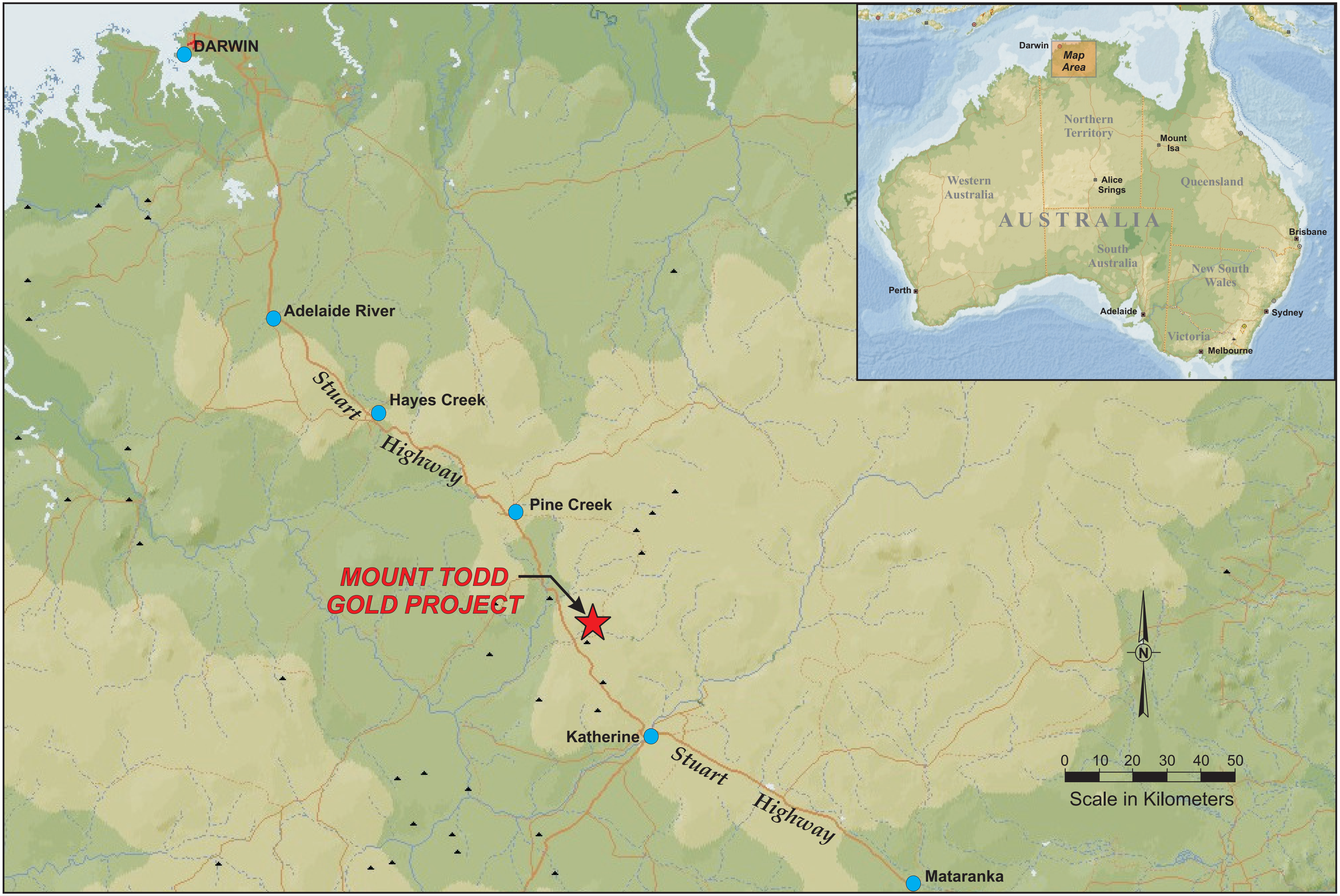
4.3 LEASE AND ROYALTY STRUCTURE

The agreement with the NT government was 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. Pursuant to the terms of the first five-year term in accordance with the conditions of the agreement, Vista Australia has undertaken a comprehensive technical and environmental review of the Project to evaluate current site environmental conditions and developed a program to stabilize the environmental conditions and minimize offsite contamination. Vista has also reviewed the water management plan and made recommendations and developed a Technical Report for the re-starting of operations. During the term of the 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 agreement. In November 2010, the NT government granted the renewal and the agreement has been extended for an additional five years until December 31, 2015.

Vista Australia paid the NT's costs of management and operation of the Mt. Todd site up to a maximum of AUD$375,000 during the first year of the term, and assumed site management and pay management and operation costs in following years. In the agreement, the NT acknowledges its commitment to rehabilitate the site and the agreement provides that Vista Australia has no rehabilitation obligations for pre-existing environmental conditions until it submits and receives approval of a Mine Management Plan for the resumption of mining operations. Recognizing the importance placed by the NT upon local industry participation, Vista Australia has agreed to use, where appropriate, NT labor and services during the period of the agreement in connection with the Mt. Todd property, and further, in connection with any proposed mining activities prepare and execute a local Industry Participation Plan.

Pursuant to an agreement with the Jawoyn Association Aboriginal Corporation (JAAC), Vista was required to issue common shares of Vista with a value of CDN$1.0 million as consideration for the JAAC entering into the agreement and for rent for the use of the surface overlying the mineral leases during the period from the effective date of the agreement until a decision is reached to begin production. Vista pays the JAAC AUD$5,000 per month in return for consulting with respect to Aboriginal, cultural, and heritage issues.



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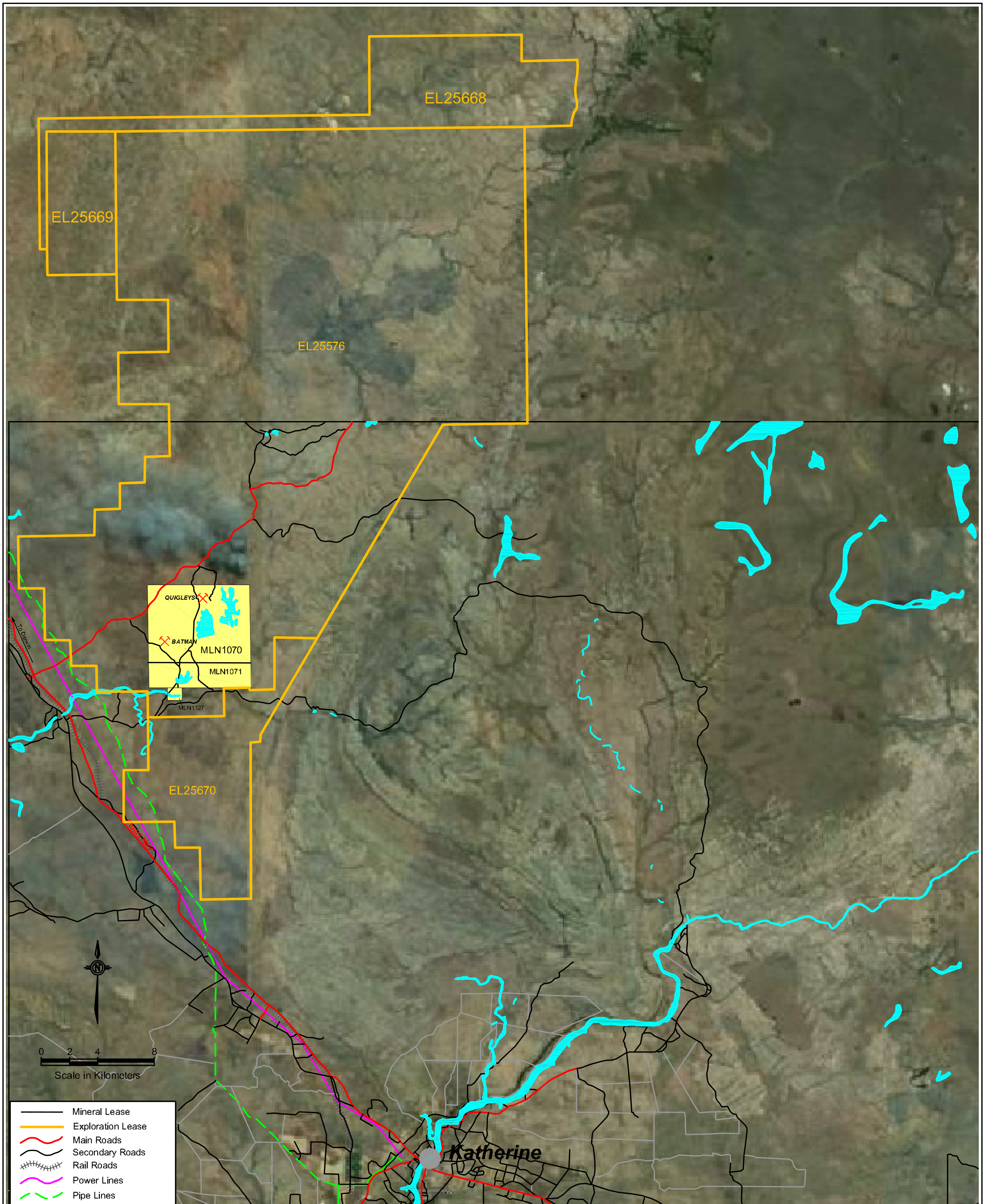
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Vista Gold Corp.*	Fig1-1.cdr	**Figure 4-1** **General Location Map** **Mt. Todd Gold Project**
	Project: Mt. Todd Gold Project	Project Number: 114-311156	
	Project Location: Northern Territory, Australia	Date of Issue: October 2010	



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TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Vista Gold Corp.*	Fig1-2_4-2.dwg	**Figure 4-2** **Concessions and Infrastructure Map**
	Project: Mt Todd Gold Project	Project Number: 114-311156	
	Project Location: Northern Territory, Australia	Date of Issue: September 2010	

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

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

5.2 CLIMATE AND PHYSIOGRAPHY

The Mt. Todd 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. The temperature usually ranges from 25° to 35°C. Between November and December, temperatures can reach 42°C. Winter temperatures in the dry season are warm in the daytime, but can drop to 10°C at night.

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

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

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

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

5.4 TOPOGRAPHY, ELEVATION AND VEGETATION

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

6.0 HISTORY

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

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

Feasibility studies for Phase I, a heap leach operation which focused predominately on the oxide portion of the deposit, commenced during 1992 culminating in an EPCM award to Minproc in November of that year. The Phase I project was predicated upon a 4 Mtpy on an annualized basis heap leach plant. This came on stream in late 1993. The treatment rate was subsequently expanded to a rate of 6 Mtpy on an annualized basis in late 1994.

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

Table 6-1: Heap Leach – Historic Feasibility Estimates vs. Historic Actual Production

Category	Historic Feasibility Study	Historic Production Actual
Tonnes Leached - million	13.0	13.2
Head Grade – g Au/t	1.2	0.96
Recovery - %	65	53.8
Gold Recovered –troy-oz	320,000	220,755
Cost/t – AUD	7.13	8.33
Cost/oz – AUS	281	500

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

Phase II involved expanding to 8 Mtpy and treatment through a flotation and carbon-in-leach (CIL) circuit. The feasibility study was conducted by a joint venture between Bateman Kinhill and Kilborne (BKK) 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 AUD$232 million (US$181 million).

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

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

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

6.1 History of Previous Exploration

The Batman gold prospect, located about 3.5 km west of Mt. Todd, is part of a goldfield that was worked from early in the 20th century. Gold and tin were discovered in the Mt. Todd area in 1889. Most deposits were worked in the period from 1902 to 1914. A total of 7.80 tonnes of tin concentrate was obtained from cassiterite-bearing quartz-kaolin lodes at the Morris and Shamrock mines. The Jones Brothers reef was the most extensively mined gold-bearing quartz vein, with a recorded production of 28.45 kg Au. This reef consists of a steeply dipping ferruginous quartz lode within tightly folded greywackes.

The Yinberrie Wolfram field, discovered in 1913, is located 5 km west of Mt. Todd. Tungsten, molybdenum and bismuth mineralization was discovered in greisenised 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 greisenised portions of the Cullen Batholith in the vicinity of the Edith River. The area has been explored previously by Esso for uranium without any economic success.

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

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

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

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

Table 6-2: Property History

1986	
October 1986 – January 1987:	Conceptual Studies, Australia Gold PTY LTD (Billiton); Regional Screening; (Higgins), Ground Acquisition by Zapopan N.L.
1987	
February:	Joint Venture finalized between Zapopan and Billiton. Geological
June-July:	Reconnaissance,
October:	Regional BCL, stream sediment sampling.
	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 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, RGP132,
June:	GP1-8, BP108, TP1-7 (202m diamond, 3090m RC); TR1-159 (501m RAB). Mining lease application (MLA's 1070, 1071) lodged.
July-Dec:	Resource estimates; mining-related studies; Batman EM-drilling: BD12, BD8690 (1375m diamond); RC pre-collars and H/W drilling, BP209-220 (1320m RC); Exploration EM and exploration drilling: Tollis, Quigleys, TP9, TD1, QP1-3, QD1-4 (1141 diamond, 278m RC); Negative Exploration Tailings Dam: E1-16 (318m RC); DR1-144 (701. RAB) (Kenny, Wegmann, Fuccenecco, Gibbs).
1990	
Jan-March:	Pre-feasibility (PFS) related studies; Batman Inclined Infill RC drilling: BP222-239 (2370m RC); Tollis RC drilling, TP10-25 (1080m RC). (Kenny, Wegmann, Fuccenecco, Gibbs)
1993 - 1997	Pegasus Gold Australia Pty Ltd reported investing more than $200 million in the development of the Mt. Todd mine and operated it from 1993 to 1997, when the project closed as a result of technical difficulties and low gold prices. The deed administrators were appointed in 1997 and sold the mine in March 1999 to a joint venture comprised of Multiplex Resources Pty Ltd and General Gold Resources Ltd.
1999 - 2000	
March - June	Operated by a joint venture comprised of Multiplex Resources Pty Ltd and General Gold Resources Ltd. Operations ceased in July 2000, Pegasus Gold Australia Pty Ltd, through the Deed Administrators, regained possession of various parts of the mine assets in order to recoup the balance of purchase price owed it. Most of the equipment was sold in June 2001 and removed from the mine. The tailings facility and raw water facilities still remain at the site.
2000 – 2006	The Deed Administrators, Pegasus Gold Australia Pty Ltd, the government of the NT, and the Jawoyn Association Aboriginal Corporation (JAAC) held the property.
2006	
March	Vista Gold Corp. acquires mineral lease rights from the Deed Administrators.

6.2 Historic Drilling

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

6.2.1 Batman Deposit

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

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

6.2.2 Drillhole Density and Orientation

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

At the time of The Winters Company's (TWC) site visit in 1997, the drillhole database numbered 730 drillholes. It is not known if any drillholes were excluded from the Pegasus exploration models. Most of the new drilling that had been added since the 1994 MRT model was relatively shallow. TWC reviewed PGA's 50 m drill sections through the Batman deposit and saw that there was a marked decrease in drillhole spacing below 1000 RL (the model has had constant 1000 m added to it in order to prevent the reporting of elevations below 0 m and have been denoted as RL for relative elevation) and another sharp break below 900 RL. The drillhole spacing in the south of 1000 N on the 954 RL bench plan approached 80 m x 80 m. Pegasus was able resolve this problem by using very long search ranges in its grade estimation. In the main ore zone, Pegasus used maximum search distances in the north and east directions of nearly 300 m.

Another potential problem related to drilling is the preferred orientation of the drillholes. Most of the drillholes in the assay database are inclined to the west to capture the vein set which strikes N10° to 20°E, dips east, and which dominates the mineralized envelope. This orientation is the obvious choice to most geologists since these veins are by far the most abundant. Ormsby (1997) discussed that while the majority of mineralization occurs in these veins, the distribution of gold mineralization higher than 0.4 g Au/t is controlled by structures in other orientations, such as east-west joints and bedding. For this reason, Ormsby stated, *"The result is that few ore boundaries (in the geological model) actually occur in the most common vein orientation.*" If this is truly the case, the strongly preferential drilling orientation has not crosscut the best mineralization and in cases may be sub-parallel to it.

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

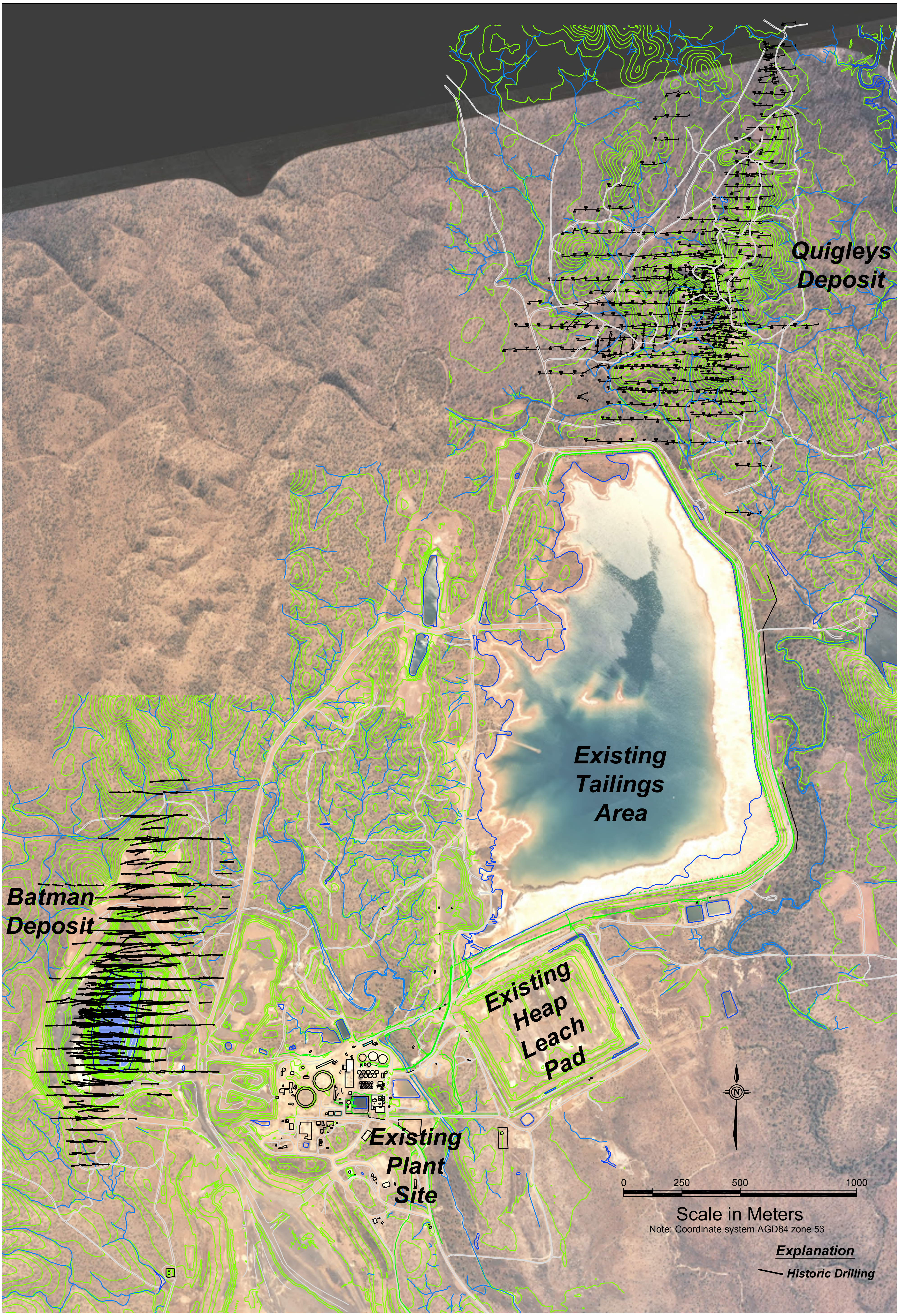
6.2.3 QUIGLEYS

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

Table 6-3: Summary of Quigleys Exploration Database

Drillholes	Gold Assays (1m approximately)	Copper Assays (1m approximately)	Lithologic Code Counts
632	49,178	41,673	51,205

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



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TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Vista Gold Corp.*	Fig6-1.dwg	**Figure 6-1**
	Project: Mt Todd Gold Project	Project Number: 114-311285	**Drillhole Location Map**
	Project Location: Northern Territory, Australia	Date of Issue: June 2013	**Batman & Quigleys Deposits**

6.3 Historic Sampling Method and Approach

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

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

6.4 Historic Sample Preparation, Analysis, and Security

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

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

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

6.4.1 Sample Analysis

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

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

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

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

6.4.2 CHECK ASSAYS

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

6.4.3 SECURITY

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

6.5 HISTORIC PROCESS DESCRIPTION

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

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

The historic design process flowsheet for the Mt. Todd Project is given in **Figure 6-2.** A brief description of the major unit operations is as follows:

- **Crushing:** Four stages of crushing were employed to produce a product having a P_{80} of 2.6 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 carbon-in-leach circuit. The leach residue was sent to the tailings pond. Approximately 60% of the gold in the flotation tailings was supposed to be recovered in the CIL circuit.
- **CIL of Flotation Concentrate:** The flotation concentrate was reground in Tower mills to 15 microns and subjected to cyanide leaching to recover the bulk of the gold in this product (94.5% of the flotation concentrate). The leach residue was sent to the tailings pond.
- **Process Recycle:** The process water was recycled to the milling circuit from the tailings pond.
  The overall gold recovery was projected to be 83.8% for the proposed circuit. However, during the initial phase of plant optimization, problems were encountered with high levels of cyanide in the recycled process water which, when returned to the mill, caused depression of pyrite and much lower recoveries to the flotation concentrate. As a result, the flotation plant was shut down and the ground ore was directly sent to the CIL circuit. The modified process flowsheet is given in **Figure 6-3**.
  Without the flotation circuit, the CIL plant recovered 72 to 75% of the gold.

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

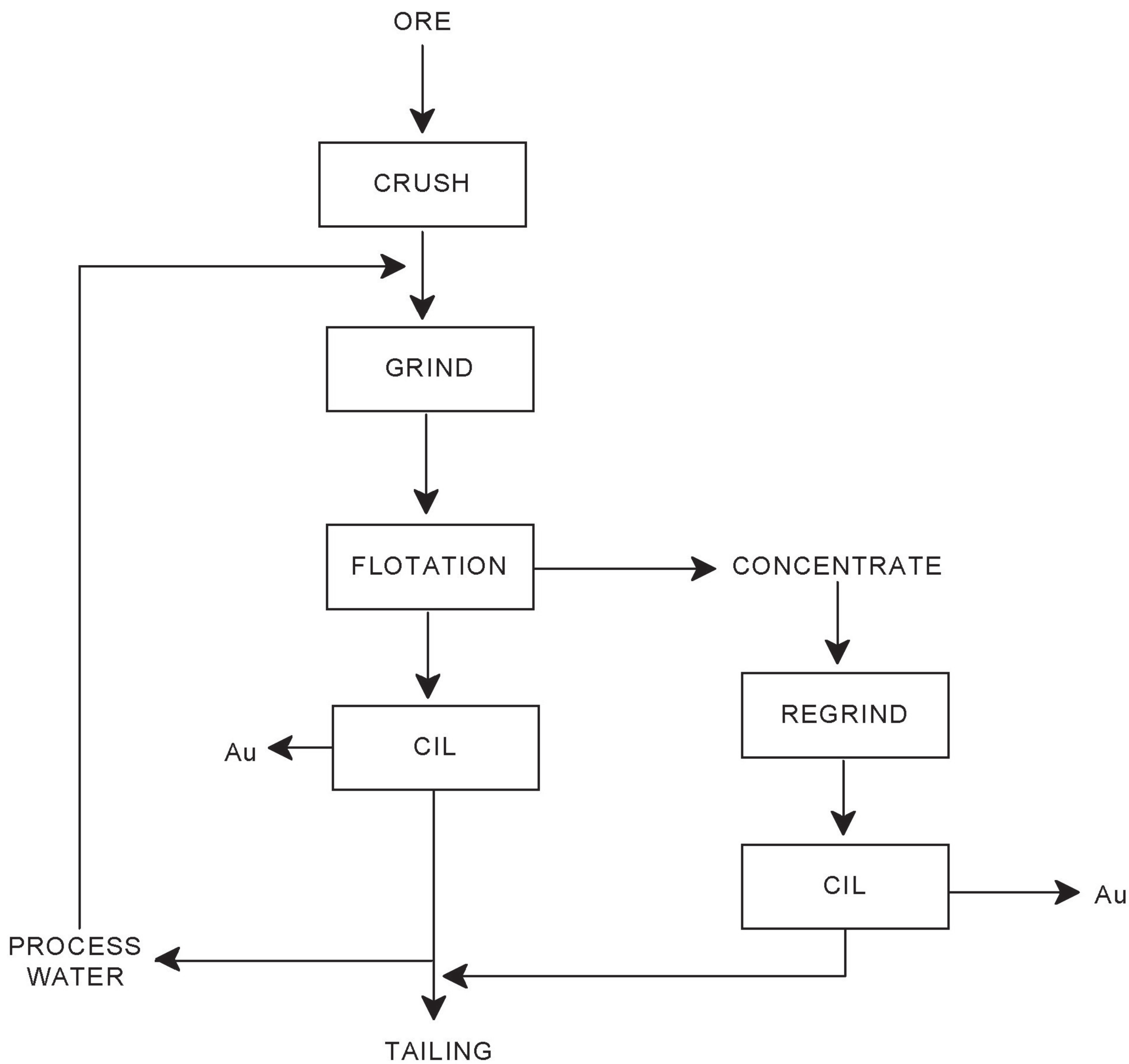
6.6 Technical Problems with Historical Process Flowsheet

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

6.6.1 Crushing

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

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



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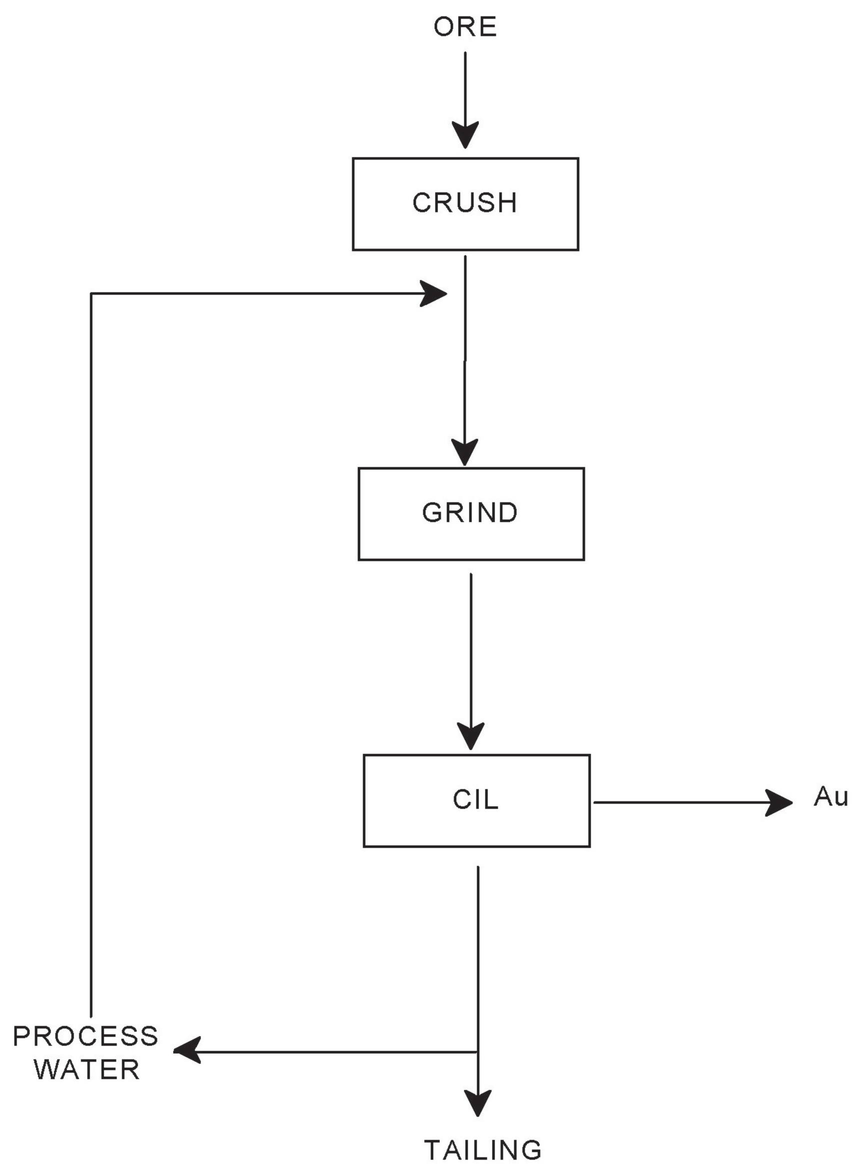
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Vista Gold Corp.*	Fig6-2.dwg	**Figure 6-2** **Plant Process Flowsheet for Mt. Todd Project (as designed)**
	Project: Mt. Todd Gold Project	Project Number: 114-311156	
	Project Location: Northern Territory, Australia	Date of Issue: 03/05/2008	



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Prepared for: Vista Gold Corp.	File Name: Fig6-3.dwg
Project: Mt. Todd Gold Project	Project Number: 114-311156
Project Location: Northern Territory, Australia	Date of Issue: 03/05/2008

Figure 6-3
Modified Plant Process Flowsheet for Mt. Todd Project

The following problems were encountered with the crushing circuit:

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

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

6.6.2 FLOTATION CIRCUIT

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

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

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

6.6.3 CIL OF FLOTATION CONCENTRATE AND TAILINGS

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

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 GEOLOGICAL AND STRUCTURAL SETTING

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

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

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

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

Late and Post Orogenic granitoid intrusion of the Cullen Batholith occurred from 1789 Ma to 1730 Ma, and brought about local contact metamorphism to hornblende hornfels facies.

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

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

7.2 LOCAL GEOLOGY

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

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

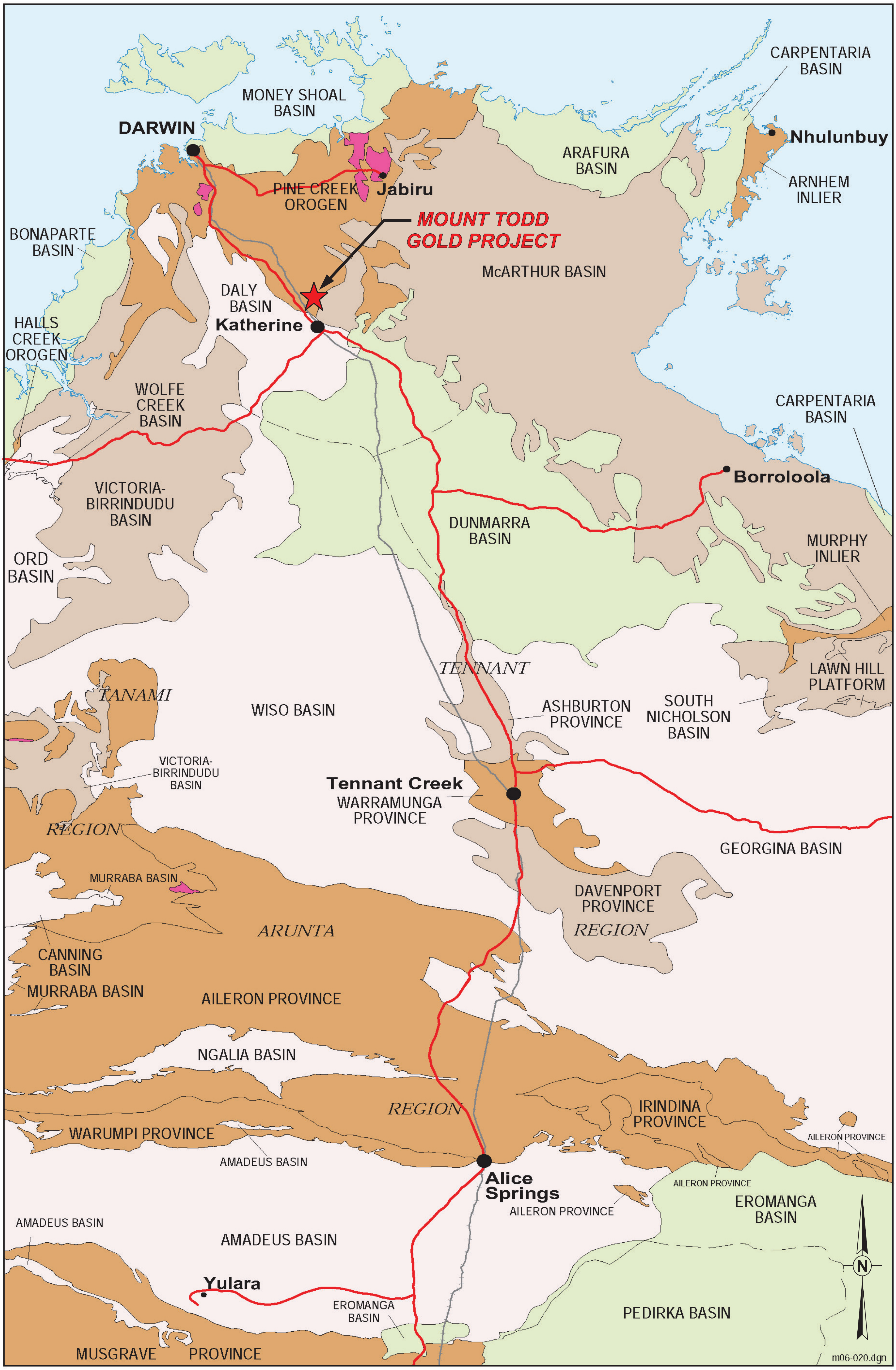
Table 7-1: Geologic Codes and Lithologic Units

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

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

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

Northerly trending quartz sulfide veins and joints striking at 0° to 20°, dipping to the east at 60° are the major location for mineralization in the Batman deposit. The veins are 1 to 100 mm in thickness with an average thickness of around 8 to 10 millimeter (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.



Produced by Cartographic Section, Northern Territory Geological Survey, March 2006.

Younger basins	
Central Australian Platform Cover	
North Australian Platform Cover	
Orogenic Domains	
Archaean Basement Inliers	

0 100 200 300 400 500 km

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TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Vista Gold Corp.*	Fig7-1.cdr	**Figure 7-1** **General Geologic Map**
	Project: Mt. Todd Gold Project	Project Number: 114-311156	
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	

7.3 MINERALIZATION

A variety of mineralization styles occur within the Mt. Todd area. Of greatest known economic significance are auriferous quartz-sulfide vein systems. These vein systems include the Batman, Jones, Golf, Quigleys and Horseshoe prospects, which occur within a north-northeast trending corridor, and are hosted by the Burrell Creek Formation. Tin occurs in a north-northwest trending corridor. The tin mineralization comprises cassiterite, quartz, tourmaline, kaolin, and hematite bearing assemblages, which occur as bedding to parallel breccia zones and pipes. Polymetallic Au, W, Mo, and Cu mineralization occurs in quartz-greisen veins within the Yinberrie Leucogranite; a late stage highly fractionated phase of the Cullen Batholith. The Batman Deposit extends approximately 2200 m along strike, 400 m across dip and drill tested to a depth of 800 m. Drilling indicates the Batman mineralization to be open along-strike and down-dip.

7.3.1 BATMAN DEPOSIT

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 chalcophyrite, 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.

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 3 major zones based on veining and jointing intensity.

Core Complex

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

Hanging Wall Zone

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

Footwall Zone

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

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

Bedding Parallel Mineralization

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

7.3.2 QUIGLEYS DEPOSIT

The Quigleys deposit mineralization was interpreted by Pegasus and confirmed by Snowden 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 reverse circulation drilling method (RC) and diamond drilling by Pegasus and previous explorers on 50 m lines with some infill to 25 m.

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

It was further thought that while the shear might be readily identified in diamond drillholes, interpretation in RC drilling, and in particular later interpretation from previously omitted RC holes, must invoke a degree of uncertainty in the interpretation.

The conclusion was 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.

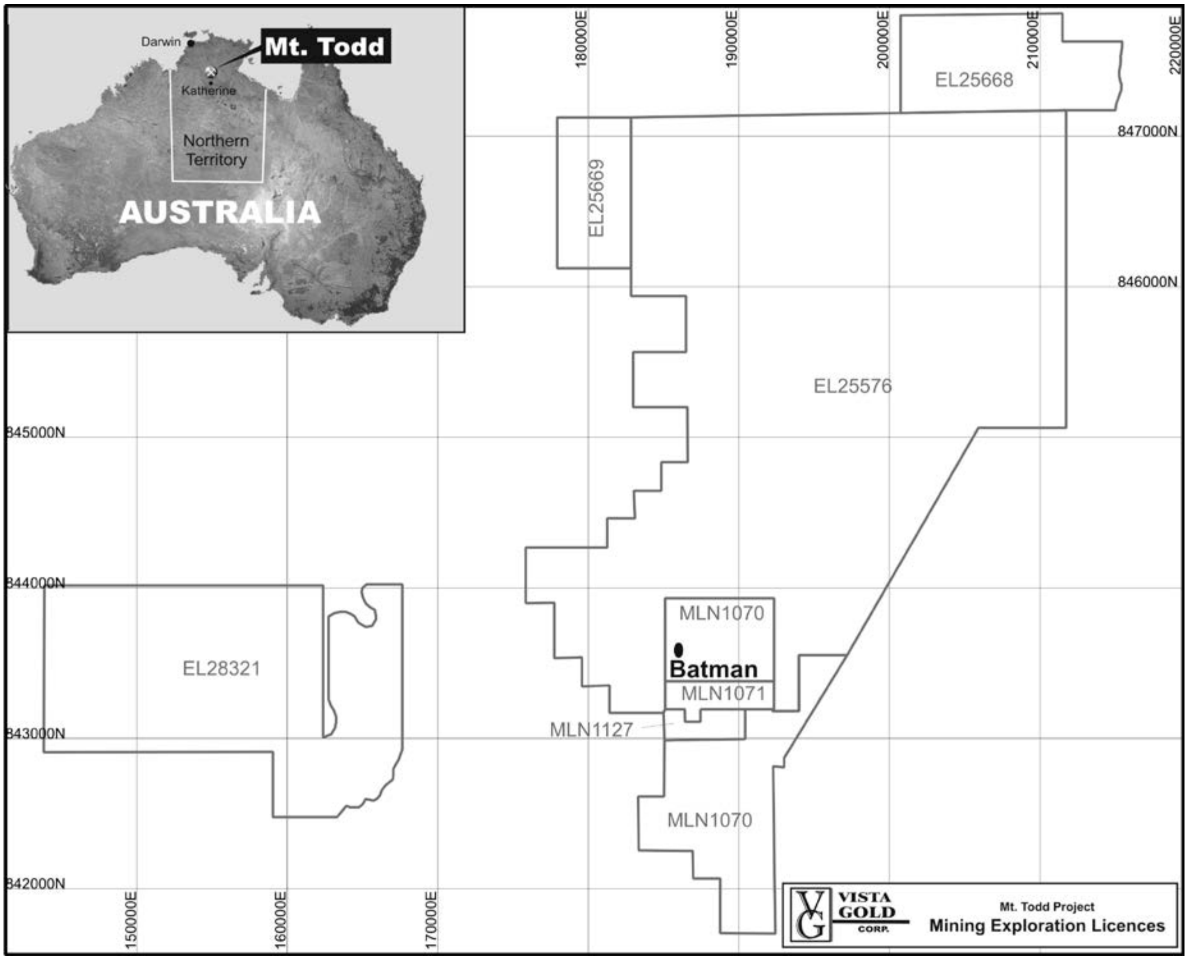


Figure 7-2: Concessions Map Mt. Todd Gold Project

8.0 DEPOSIT TYPES

According to Hein (2003), the Batman and Quigleys gold deposits of the Mt. Todd Mine are formed by hydrothermal activity, concomitant with retrograde contact metamorphism and associated deformation, during cooling and crystallization of the Tennysons Leucogranite and early in D2 (Hein, submitted for publication). It is speculated that pluton cooling resulted in the development of effective tensile stresses that dilated and/or reactivated structures generated during pluton emplacement and/ or during D1 (Furlong et al., 1991), or which fractured the country rock carapace as is typical during cooling of shallowly emplaced plutons (Knapp and Norton, 1981). 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) may have facilitated channelization of predominantly metamorphic fluid in the intermediate contact metamorphic aureole (possibly suprahydrostatic-pressured) and into the upper crust (Furlong et al., 1991; Cox et al., 2001). Rising fluids decompressed concurrent with mineral precipitation. Throttling of the conduit or fluid pathways probably resulted in over pressuring of the fluid (Sibson, 2001), this giving way to further fracturing, etc. Mineral precipitation accompanied a decrease in temperature although, ultimately, the hydrothermal system cooled as isotherms collapsed about the cooling pluton (Knapp and Norton, 1981).

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

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

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

9.0 EXPLORATION

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

The exploration effort initially focused on follow up work on targets developed by Pegasus during their tenure on the property. These included the RKD target, Tablelands, and Silver Spray. During a review of Pegasus' airborne geophysical survey data, five distinct magnetic highs were observed located within sedimentary rocks that should have a low magnetic signature. These features are remarkably similar to those at Mt. Todd, 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 explored.

Table 9-1: Exploration Sampling

Year	Soils	Samples Collected
2008	0	164
2009	1,333	45
2010	3,135	224
2011	1,925	79
2012	2,312	295
As of June, 2013	572	51
Total Samples	**9,277**	**858**

9.1 RESULTS

9.1.1 GOLDEN EYE

At Golden Eye, rock chip sampling, in an area with limited exposure, returned a 25.0 g Au/t sample from a small outcrop of banded iron formation. Further sampling returned 23.0 g Au/t and 7.7 g Au/t assays in vein and breccias located 15 m and 50 m, respectively, north of the original sample. Due to the sparse outcrop, the orientation and thickness of the mineralized zone is not currently known. A soil sampling program over the area was completed on a 20-m grid. The survey returned a strong coherent gold anomaly approximately 400 m in diameter with coincident anomalous base metals and arsenic.

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

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

A detailed ground magnetic survey was completed over the area in 2012. The survey results, combined with detailed mapping and the drillhole data is currently being reviewed and additional drilling may be recommended.

9.1.2 RKD

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

9.1.3 Silver Spray

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

9.1.4 Snowdrop

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

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

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

Additional drilling is planned in 2013.

10.0 DRILLING

10.1 DRILLING

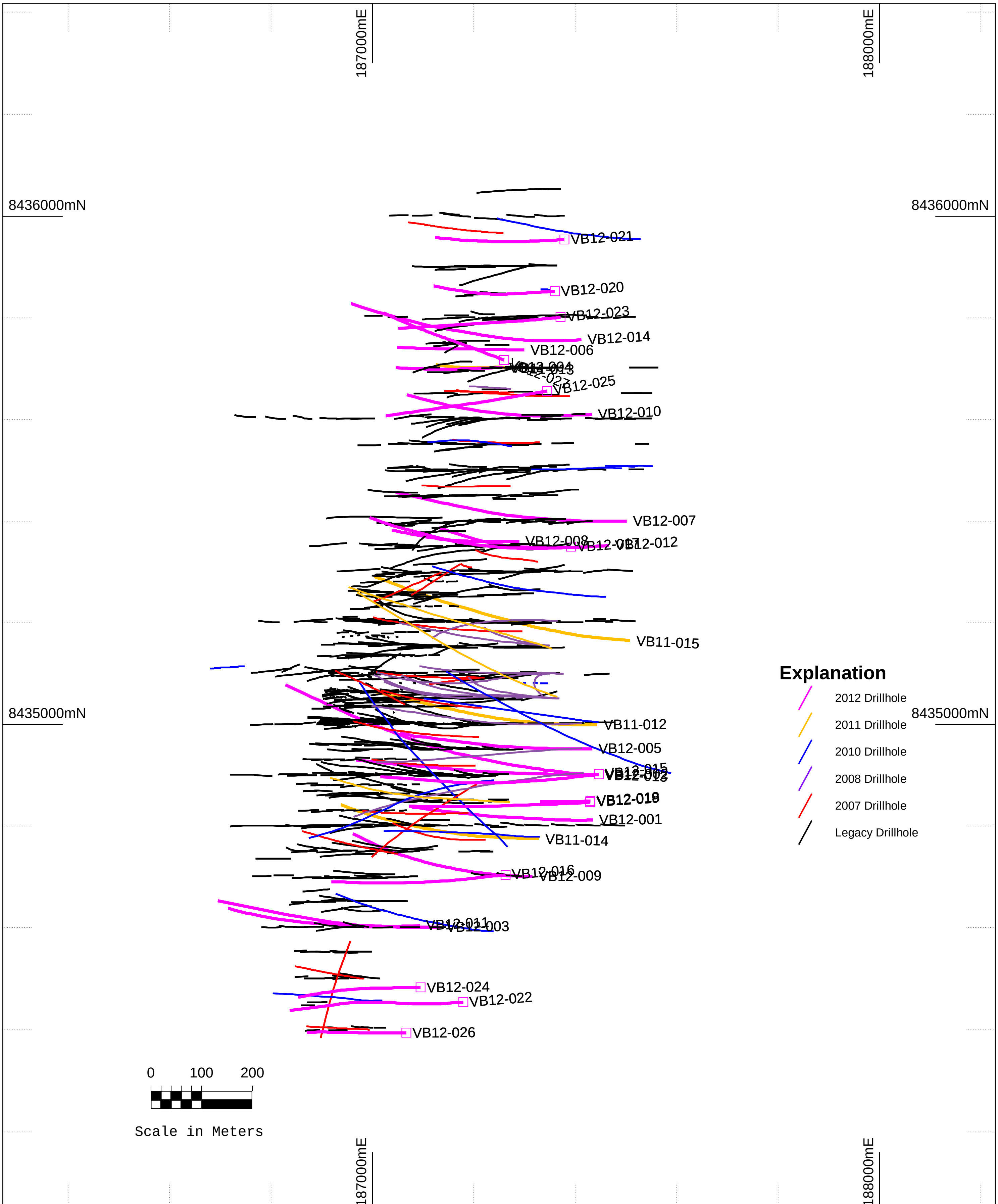
The drilling discussed in this section is limited to that completed November 2012 and used to complete the resource estimate from March 2013 drilling since the last report. No exploration drilling was completed between January and June 28, 2013.

In 2012 the Vista exploration program at the Batman deposit consisted of 13 diamond core containing 7,639.43 m that targeted both infill definitional drilling and step-out drilling. **Table 10-1** contains information for the 13 drillholes completed.

A total of 7,705 assays from 13 drillholes were added for the resource update. **Figure 10-1** is a plan map that details the locations of all exploration drillholes drilled at the Batman deposit up to and including VB12-027.

Table 10-1: Drillholes Added For Resource Update

Drillhole ID	Northing m (MGA94 z53)	Easting m (MGA94 z53)	Elevation (masl)	Bearing (°)	Dip (°)	Total Depth (m)	Drillhole Type
VB12-015	8434901.6	187446.7	144.4	268	-55	745.85	Diamond
VB12-016	8434703.6	187262.7	147.3	267	-61	713.5	Diamond
VB12-017	8435349.1	187391.2	150.8	277	-61	833.28	Diamond
VB12-018	8434849.2	187429.9	144.7	270	-56	177	Diamond
VB12-019	8434846.9	187429.4	144.8	269	-61	731.8	Diamond
VB12-020	8435852.4	187359.6	167.3	272	-67	611.9	Diamond
VB12-021	8435954.0	187378.8	149.9	271	-65	602.9	Diamond
VB12-022	8434453.4	187179.3	153.3	269	-57	647.9	Diamond
VB12-023	8435801.3	187371.0	161.3	265	-60	650.88	Diamond
VB12-024	8434482.1	187094.7	149.8	266	-58	460.14	Diamond
VB12-025	8435656.2	187344.7	158.6	261	-60	650.63	Diamond
VB12-026	8434393.4	187066.8	144.8	270	-59	378.9	Diamond
VB12-027	8435717.0	187259.7	169.8	291	-54	434.75	Diamond



Issued By:	Prepared for: *Vista Gold Corp.*	File Name: Figure 10-1.dwg	Figure 10-1: Drillhole Location Map Batman Deposit to VB12-027
TETRA TECH	Project: Mt. Todd Gold Project	Project Number: 114-311285	
	Project Location: Northern Territory, Australia	Date of Issue: June 2013	

10.2 SAMPLING

The sampling method and approach for drillholes completed in 2011 and 2012 was similar to what has historically been used at Mt. Todd. The drill core, upon removal from the core barrel, is placed into plastic core boxes. The poly core boxes are transported to the sample preparation building where the core is marked, geologically logged, geotechnically logged, photographed, and sawn into halves. One-half is placed into sample bags as one m sample lengths, and the other half retained for future reference. The only exception to this is when a portion of the remaining core has been flagged for use in the ongoing metallurgical testwork.

In the few case of short reverse circulation drillholes reverse circulation pre collars, Vista employed a rifle splitter to collect one m samples intervals the entire length drilled by reverse circulation. Vista has ceased the practice of RC pre-collaring: all 18 drillholes added for this estimate are full diamond drillholes.

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

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

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

The following section describes the sample preparation, analyses and security undertaken by Vista for the 31 exploration drillholes completed in 2011 and 2012 and included in the March 2013 resource update.

11.1 SAMPLE PREPARATION

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

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

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

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

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

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

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

11.2 Sample Analyses

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

Table 11-1: Assay and Preparation Laboratories

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

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

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

11.3 Sample Security

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

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

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

The author is satisfied with the adequacy of sample preparation, security and analytical procedures employed by Vista given the fact that Vista has completed nearly 50,000 m of core drilling to go with the approximately 98,000 m of core and rotary 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 of the mineralization estimated at the Batman deposit.

12.0 DATA VERIFICATION

12.1 DRILL CORE AND GEOLOGIC LOGS

Prior to the October 2012 resource estimate, multiple site visits were performed by Mr. Rex Bryan, QP for the resource estimation portion of this Technical Report. During that visit, Tetra Tech found a comprehensive drillhole database comprised of drill core, photographs of the drill core, assay certificates and results, and geologic logs. All data were readily available for inspection and verification. In addition, most of the subsequent companies or their consultants that have examined the project have completed checks of the data and assay results. The author reviewed drill core, drill core logs and assay certificates and found a minimal number of errors (i.e., mislabeled intervals, number transpositions), which were corrected prior to Tetra Tech's October, 2012 resource estimation. It is Tetra Tech's opinion that the databases and associated data were of a high quality in nature and valid for use in mineral resource and reserve estimation.

Tetra Tech found no significant discrepancies with the existing drillhole geologic logs and is satisfied that the geologic logging, as provided for the development of the three-dimensional geologic models, fairly represents both the geologic and mineralogic conditions of each of the deposits that comprise the Mt. Todd Project.

12.2 TOPOGRAPHY

The topographic map of the project area was delivered electronically in an AutoCAD® compatible format and represents the topography in half meter accuracy. The native coordinate system of the topography is MGA94 zone 53, and for this resource update and as the Project goes forward MGA94 zone 53 will be the used coordinate system. The surveyed drillhole collar coordinates, once translated to MGA94 zone 53 agree well with the topographic map; it is Tetra Tech's opinion that the current topographic map is accurate and accurately represents the topography of the project area. In addition, it is suitable for the development of the geologic models, mineral resource estimates, and mineral reserve estimates.

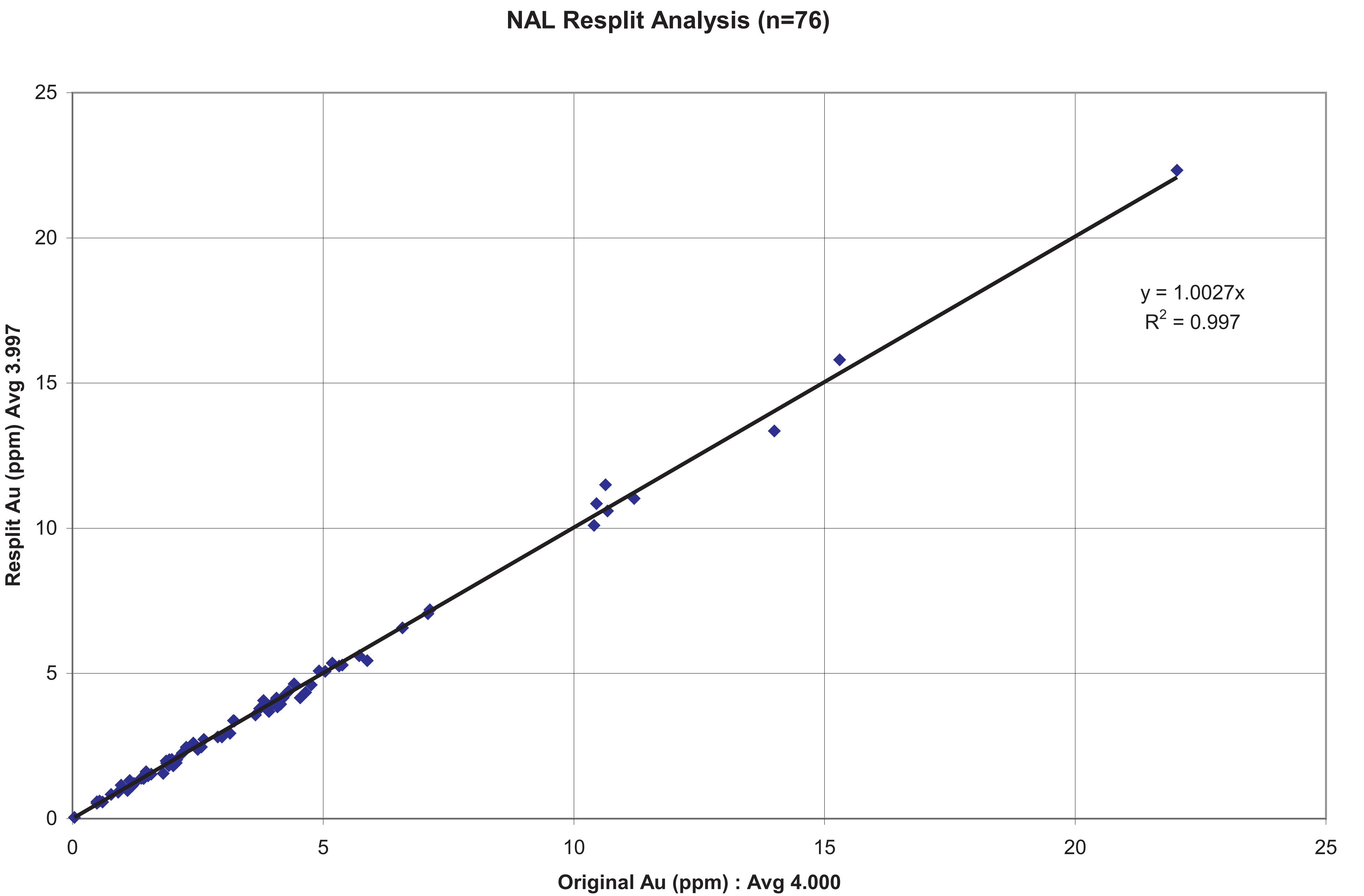
12.3 VERIFICATION OF ANALYTICAL DATA

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

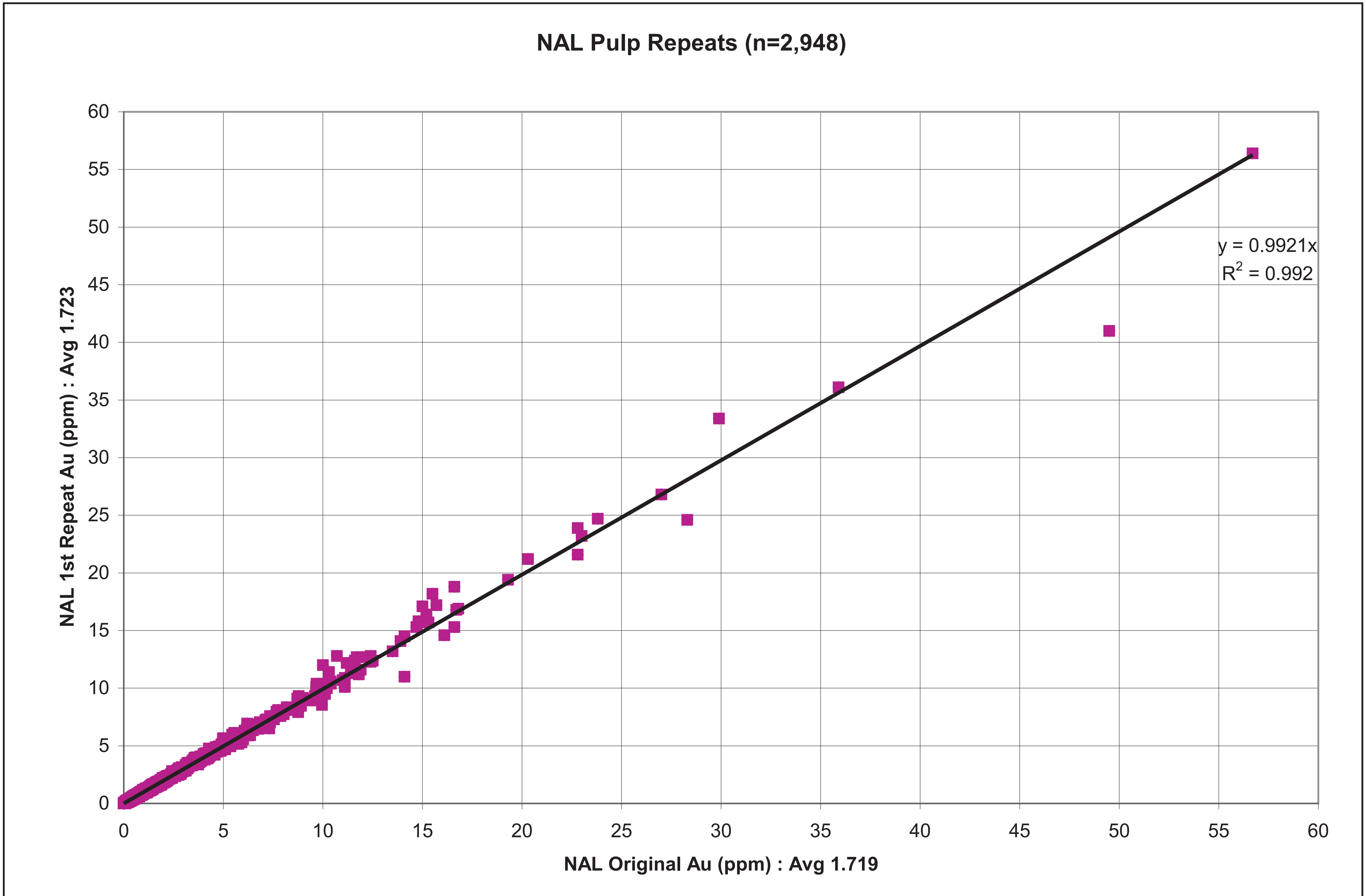
Vista completed a multi-phase program to evaluate the accuracy of gold assays generated by NAL on Mt. Todd core samples. The test involved three phases including, 1) cross checking assay standards used in the program between NAL and ALS-Chemex, 2) preparing and assaying 30, one-m intervals of remaining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

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

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

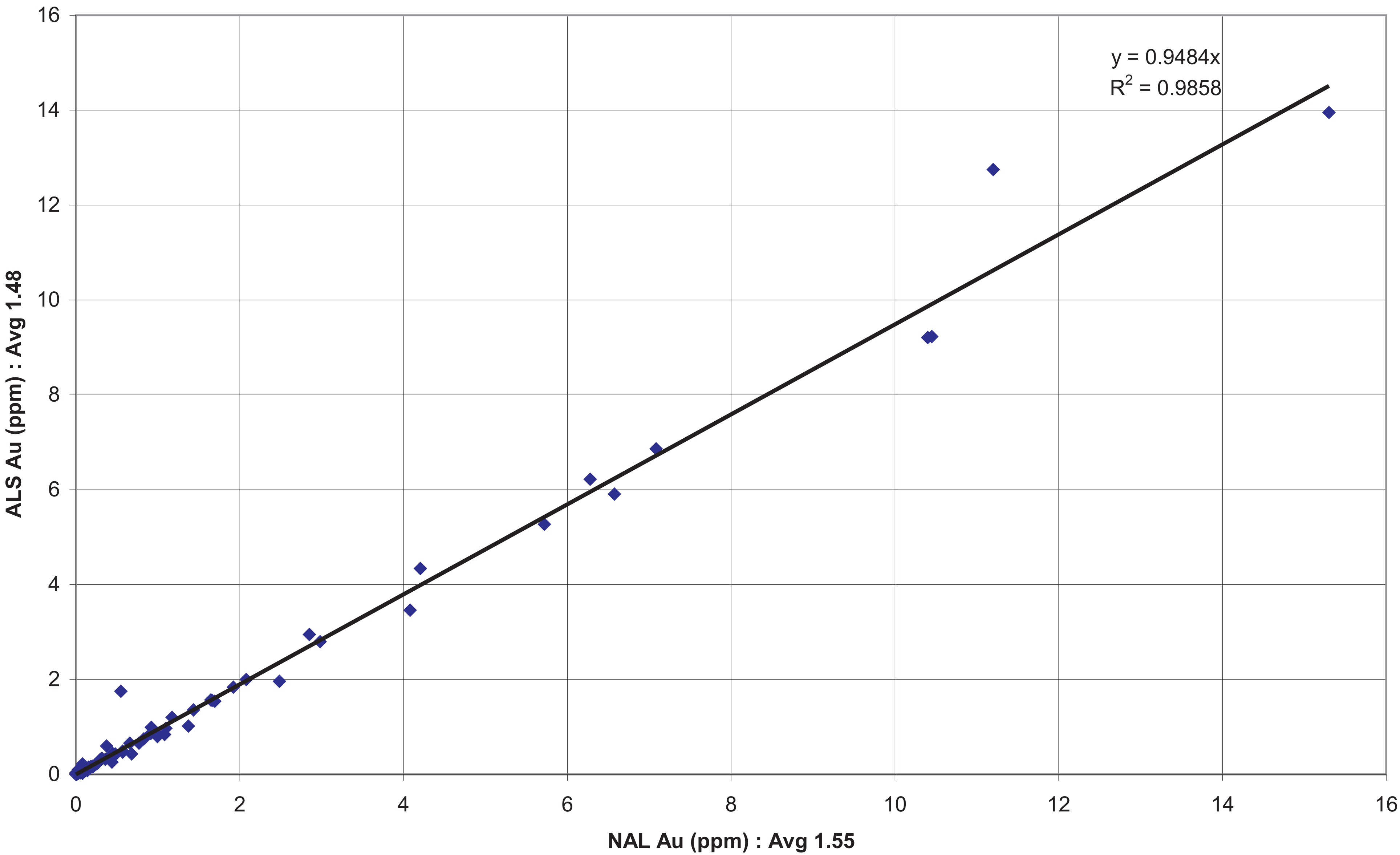


Issued by: TETRA TECH, 350 Indiana Street, Suite 500, Golden, Colorado 80401, (303) 217-5700 (303) 217-5705 fax	Prepared for: ***Vista Gold Corp.***	File Name: Fig12-1.dwg	**Figure 12-1**
	Project: Mt. Todd Gold Project	Project Number: 114-311156	**NAL Resplit Analysis (n=76)**
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Vista Gold Corp.***	Fig12-2.cdr	**Figure 12-2** **NAL Pulp Repeats (n=2,948)**
	Project: Mt. Todd Gold Project	Project Number: 114-311156	
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	

Original Pulp Cross Lab Checks (n=78)



Issued by: TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Prepared for: ***Vista Gold Corp.***	File Name: Fig12-3.dwg	**Figure 12-3** **Original Pulp Cross Lab Checks (n=78)**
	Project: Mt. Todd PreFeasibility Study	Project Number: 114-311156	
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	

12.3.1 Latest Drilling Data Verification

For the March 2013 resource estimate update, a detailed data verification procedure was undertaken by Tetra Tech which focused on the latest drilling campaign (VB12-015 through VB12-27 inclusive). This verification was accomplished by: reviewing the assay database received from Vista, comparing results with laboratory certificates received directly from the laboratory and reviewing results of the field QA/QC samples.

For the 13 drillholes, from the 2012 exploration program, there were 7,601 intervals assayed. In addition to Au and other precious metals, most intervals had multi-element and environmental test results as well. Similar to previous work, the assay interval averaged one meter with a minimum interval of 0.4 m and a maximum interval of 1.4 m. No errors were noted in the assay data received other than selenium results for one drillhole that were erroneously entered. This was corrected by Vista. A spot-check of 14% of the received database with laboratory certificates requested and received from NAL showed a 100% correct correlation of reported values. Some of the environmental results reported in the certificates had not yet been entered into the database; this was the one discrepancy noted.

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

Field blanks were also reviewed and found to be acceptable. Of 388 blank results, 6 blanks had Au concentrations greater than detection limit of 0.01. The maximum value was 0.11 ppm. Again, no drift was noted in the data over time.

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

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

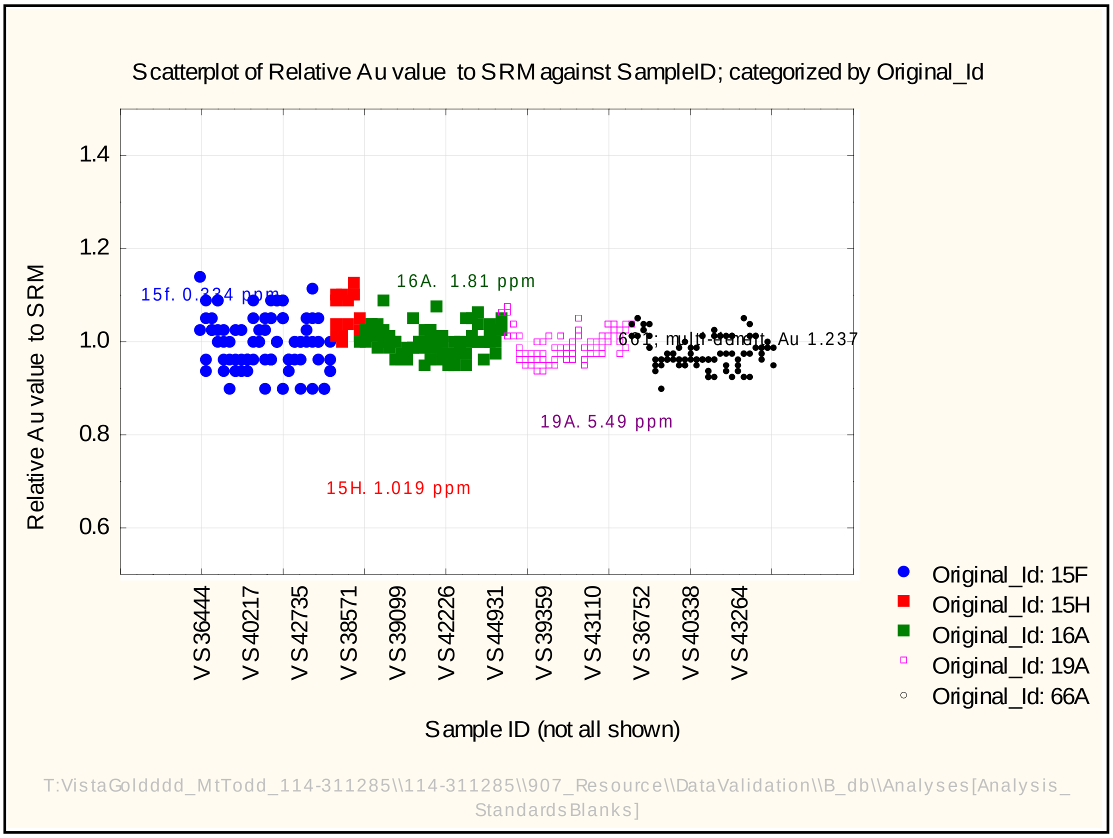


Figure 12-4: Scatterplot of relative Au Value to Certified Standard Reference Material Value

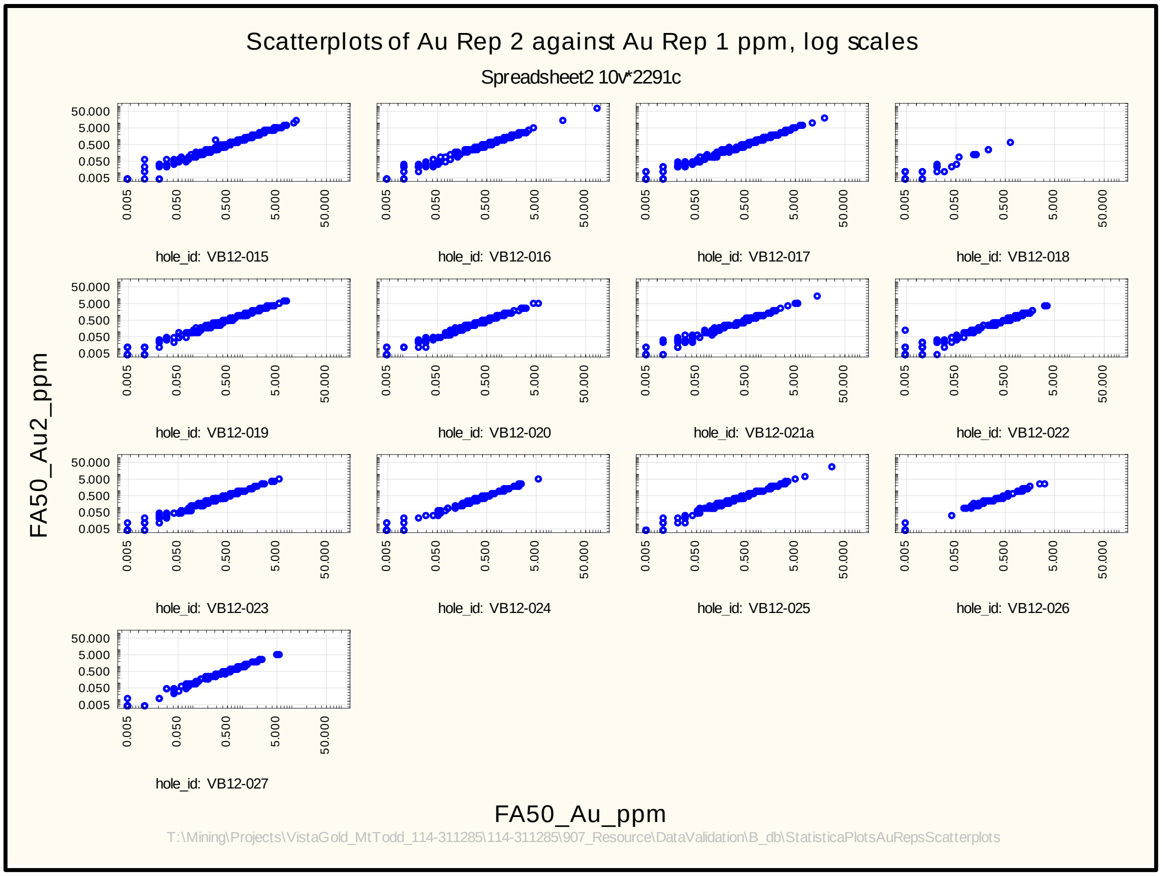


Figure 12-5: Scatterplots (log scale) of replicates by Drillhole

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

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

13.1 SUMMARY

Key conclusions drawn from the metallurgy studies to date are:

- Mt. Todd (and in particular the Batman deposit) ore is one of the hardest and most competent ore types processed for mineral recovery. The most energy efficient comminution circuit has been determined to be the sequence of primary crushing, closed circuit secondary crushing, and closed circuit HPGR tertiary crushing followed by ball milling.
- The ore is free-milling, has no preg robbing problem, and is amenable to gold extraction by conventional cyanidation processes.
- The ore has relatively high specific cyanide consumption, determined to be 0.77 kg of sodium cyanide per tonne of ore. This is largely due to the presence of sulfides and cyanide consuming copper.
- The ore requires a P_{80} grind of 90 µm and 24 hour leach residence time to achieve a nominal 82% gold recovery (81.7% net of solution loss) from a global head grade of 0.86 g Au/t of ore.

13.2 HISTORIC METALLURGICAL TEST PROGRAMS

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

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

- 1988 – 1997 metallurgical studies by previous owners (Pegasus) led to the design and construction of a treatment plant comprised of crushing, milling to a P_{80} of 150 µm, sulfide flotation, concentrate regrind and cyanidation, and, separate CIL cyanidation of flotation tailings. Operational efficiencies were lower than planned due to high degree of ore hardness, presence of cyanide-soluble copper minerals, and inefficient flotation performance resulting from the presence of free cyanide in the process water (from recycled tailings decant water). These process difficulties together with the collapse of the gold price led to the cessation of operations in the mid 1990s.
- In 2006, Vista acquired the Project with the belief that each of these challenges could be overcome through the use of current technology, adequate metallurgical testing and higher gold prices. Vista’s consultant, Resource Development Inc. (RDi), completed a study using historical metallurgical data and test results from transition ore samples. RDi proposed a flowsheet consisting of crushing and grinding followed by rougher flotation to produce a sulfide concentrate containing 85% of the gold. Rougher tailings, substantially barren of gold and sulfides, would be discarded to a

benign tailings dam. Rougher concentrate would be reground to enable upgrading in a cleaner flotation circuit to produce a saleable copper concentrate containing 50% of the gold. Cleaner tailings would be cyanide leached in a CIL circuit for gold recovery. The cleaner tailings would be subjected to cyanide destruction and stored in a separate sulfide tailings dam.

- The design incorporated energy efficient HPGR technology in the comminution circuit to handle the extremely hard ore. These processing advantages combined with a higher gold price significantly improved the viability of the proposed operation. It then became necessary to confirm if the remaining ore had the same metallurgical characteristics as the historically processed ore.
- In 2007/2008 two exploration drilling programs were completed focusing on the deeper ore beneath the existing Batman pit. The new cores were more representative of the remaining resource and samples were selected for confirmatory metallurgical test work. It was confirmed that the ore was extremely hard but it was not possible to repeat the flotation results previously achieved. The tests indicated that gold recovery into the rougher flotation concentrate was ± 80% at a grind P_{80} of 74µm but copper could not be upgraded to saleable concentrate grade of ± 20% Cu. The best results were ± 6% Cu using the same test procedure as employed for earlier core testing (2006).
- Investigations revealed that the historical core tested in 2006 was transition zone material containing copper minerals predominantly as secondary copper which is known to be a major consumer of cyanide. The major sulfide mineral was pyrite. However, the 2007 and 2008 drill core had primary copper as predominant copper species and pyrrhotite as the major sulfide mineral. Pyrrhotite is known to float readily as compared to pyrite and is significantly more difficult to depress in the flotation process. It was difficult to selectively float copper minerals and produce a copper concentrate without the dilutive effect of pyrrhotite and other gangue minerals. Consequently flotation was dropped from the flow sheet and replaced with whole ore leach.
- In 2010/2011 a confirmatory drilling campaign and metallurgical test program was conducted on the remaining Batman resource. The objective was to validate the findings of the 2007/2008 programs and to expand the level of understanding of variability of metallurgical performance within the Batman ore body.
- The test program was designed by Vista, supervised by Ausenco Limited (Ausenco), and executed by ALS Ammtec in Perth, Western Australia. The test results confirmed that gold recovery by whole ore leach was the appropriate approach to process design.

13.3 SAMPLING

This section records key details of core sampling and sub-sampling in the preparation of representative material for metallurgical test work.

13.3.1 DRILLING PROGRAM 2007-2008

The following composites were prepared for RDi's testwork conducted on the samples of the deeper Batman ore from the 2007/2008 drilling program:

- Composite 1 – 1,200 kg composite sample made up from 2007 drill core. The composite consisted of samples from five drillholes selected to be representative of a cross section of the deposit. The head assay was 1.3 g Au/t, 0.92% S and 447 ppm Cu. The sequential copper analysis indicated 80.4% of the copper in the sample was primary copper. The dominant sulfide in the sample was pyrrhotite.
- Composite 2 – 140 kg composite sample made up from 2008 drill core. The head assay was 0.89 g Au/t and 450 ppm Cu. The sequential copper analysis indicated 80.3% of the copper in the sample was primary copper. The dominant sulfide in the sample was pyrrhotite.
- Drillhole 41 sample was sourced from the oxide and transitional zones (depth of 0–65 m). The head assay was 1.78 g Au/t, 1.42% S, 448 ppm Cu.

13.3.2 Drilling Program 2010-2011

Samples used for the 2011 metallurgical testwork program were sourced from eight drillholes drilled 2010/2011. The drillholes were orientated to intersect the main Batman ore body beneath the existing pit and are representative of the ore within the PFS pit shell.

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

13.3.3 Metallurgical Test Program Samples 2011

ALS Ammtec was supplied with ninety-nine (99) composited gold ore drill core intervals originating from the Project area. Each phase of the test program was allocated a suite of composite samples designated as follows:

- **Metallurgical Composites**: Four main ore type composite samples were generated from drill core from the 2010/2011 drill program. The samples were designated as follows:
    - Master composite;
    - MHT-001 drillhole composite, used for HPGR testwork;
    - MHT-004 drillhole composite, used for HPGR testwork; and
    - 2010 HPGR composite, used for additional HPGR testwork.
- **Comminution Samples**: Twenty comminution samples were selected from drill core from the 2010/2011 drill program.
- **Variability Samples**: The variability samples were selected to be representative of the various ore zones, weathering, spatial representation and variations in grade which make up the Batman deposit. A total of 99 variability composites were constructed from the 2010/2011 drill cores.

13.4 Metallurgical Testwork Program

A leading Perth laboratory, ALS Ammtec, was commissioned in 2011 to conduct an extensive defined program of metallurgical testwork on samples of gold ore drill core composites originating from the Project as described above.

This work included the following:

- Head sample analysis – original samples and composites;
- Crushing testwork – HPGR and conventional crush;
- Comminution testwork;
- Bond impact crush work index (CWi) determination ;
- Uniaxial compressive strength (UCS) determination ;
- SAG mill comminution (SMC) testwork;
- Bond ball mill work index (BWi) determination ;
- Mineralogical analysis;
- Oxygen uptake rate determination;
- Extraction testwork;

- CIL cyanidation leaching testwork;
- Direct cyanidation time leach testwork;
- Sequential triple contact CIP testwork;
- Magnetic separation – dry and wet LIMS;
- Cyanide detoxification testwork;
- Geochemical tests;
- Mixer design testwork;
- Rheology testwork; and
- Flocculation and thickener design settling testwork.

The testwork was controlled by Vista and their consultant Ausenco under the supervision of ALS Ammtec metallurgist.

13.5 MINERALOGY

In support of the metallurgical test program, a mineralogical examination was conducted on six drill core composite specimens to determine gold deportment and identify other minerals present in the ore. The quartz vein hosting sulfides and gold was targeted. Gold deportment throughout the quartz vein was in the form of fine disseminated nuggets less than 20µm in size. Metallic bismuth also interspersed with the gold.

Several sulfides were detected in the samples, including pyrite, pyrrhotite, chalcopyrite, sphalerite and galena. In most of the samples the dominant sulfide was pyrite, with the only exception being Var-26 where a large pyrrhotite grain was detected in the measurement area. The apparent scarcity of pyrrhotite was contrary to the 2006 test conclusion that pyrrhotite replaces pyrite as the dominant sulfide at depth. The drillhole logs for the variability samples also suggest that pyrite remains the dominant sulfide species irrespective of depth. This anomaly is worth further investigation. The manual SEM EDS analysis of the detected gold grains during the gold search found that the majority of the gold occurred as both pure gold and argentian gold.

13.6 METALLURGICAL TEST RESULTS

The results of the metallurgical testing are presented in the following subsections. The work was primarily performed by ALS, but included contributions from Polysius and other vendors.

13.6.1 COMMINUTION TESTS

Twenty comminution samples were selected for SMC Tests® and Bond Ball Mill Work index (BWi) tests. Variations of tests included the following:

- BWi determinations on the master composite sample at different closing screen sizes;
- BWi comparison between conventionally crushed material and HPGR product;
- BWi determinations on selected leach variability samples at various closing screen sizes; and
- Crusher work index test on eight of the comminution samples.

The results indicated that the Batman ore is consistently very competent throughout the ore body. This is indicated in the following data bullet points:

- Average specific gravity (SG) of 2.76;
- 75th percentile A x b value of 22.1;
- 75th percentile DWi value of 12.8 kWh/t; and
- 75th percentile value BWi of 26.3 kWh/t.

Bond work index decreased by approximately 8% to 24.2 kWh/t when HPGR crushed ore was tested (ostensibly due to particle micro cracking).

Samples submitted for HPGR tests gave the following results:

- ATWAL Abrasion tests: two variability samples had very similar results, “low/medium” abrasiveness at 1% moisture and a “medium” abrasiveness at 3% moisture.
- HPGR specific throughput had a reasonably close variance across eleven samples tested. The rate varied between 259 and 284 ts/m3h (specific throughput (ts) per cubic meter hour is the capacity of a machine with a roll diameter of 1 m, roll width of 1 m and a peripheral speed of 1 m/s).
- Moisture content did not impact on the specific throughput rate.
- HPGR specific energy consumption varied between 1.33 and 2.40 kWh/t depending on the applied specific press force and moisture. At a standard press force of 3.5 N/mm^2 the specific energy consumption was between 1.8 and 1.9 kWh/t.
- HPGR product fineness of approximately 65% < 6.3 mm, 25% < 1 mm and 11% <200 µm was achieved in a single pass through the MAGRO machine from a feed of <31.5 mm at a specific press force of 3.5 N/mm^2.

The circulation factor required to produce a 6 mm product (P_{80} = 3.25 mm) was 2.45. The required net specific energy input was 1.9 kWh/t of HPGR feed including circulating load, which equates to 4.6kWh/t of finished product at a P_{80} of 90 µm (for HPGR composite sample MHT004 Test 2.1).

13.6.2 Comminution Circuit Options

The 2006 conceptual flow sheet used closed circuit HPGR tertiary crushing to replace the problematic vertical shaft impact crushers employed in the earlier as built plant. In 2011 a study was commissioned to verify that the use of HPGR was technically and economically justified.

Vista appointed Ausenco to perform a technical evaluation of comminution circuit options using JKSimMet and power-based models. Various SAG versus HPGR options were modeled with input from tests undertaken by JKTech. The tonnage throughput used for the models was 1350 tonnes per hour (11 Mtpy) and the target grind was a P_{80} of 150µm. The modeling was peer reviewed by independent consultant Dr. Steve Morrell of SMCC Pty Ltd.

The most compelling result from the study was that the HPGR circuit requires 23% less energy for the same duty as the SAG-based options. These results were similar for both the JKSimMet models and the power based models. It was confirmed that the comminution circuit would be closed circuit secondary crushing followed by closed circuit HPGR tertiary crushing and finally ball milling to finish the grind.

13.6.3 Grind Size Optimization Tests

Grind optimization tests resulted in the following conclusions:

- Gold recovery from all ore types is grind sensitive between P_{80} sizes of 53 µm and 212 µm.
- The optimum grind P_{80} for the Project mill design is 90 µm.
- Gold continues to leach up to 48 hours, however there is little benefit beyond 24 hours.

A grind P_{80} of 90 µm and a leach residence time of 24 hours were consequently selected for the Project.

13.6.4 Leach Optimization Tests

Leach Cyanide Consumption

The leach cyanide consumption was derived from the 30 variability leach tests completed at 90 µm. The average cyanide and lime consumptions were calculated for the oxide and sulfide samples for a 24 hour leach and adsorption residence time. The average consumption for oxide was 0.77 kg/t and for sulfide 0.59 kg/t.

Laboratory optimized leach CIL tests demonstrated gold recovery of a nominal 82% (81.7% net of solution loss) with a grind P_{80} of 90µm and leach and adsorption residence time of 24 hours.

To minimize the loading of copper onto carbon, a certain free cyanide concentration is maintained in the system to keep the copper as $Cu(CN)_3$ and $Cu(CN)_4$. The required ratio is 4.2 to 1 free cyanide to copper in solution to minimize dissolved copper loading onto carbon. The average Cu in solution was calculated for the sulfide samples as well as for the oxide samples at 24 hours from the variability testwork at 90 microns. The average Cu in solution for the sulfide samples was 62 g/m^3 and for the oxide samples was 220 g/m^3.

According to the mine plan, over the life of the mine oxide makes up 0.66% of plant feed, with the remaining 99.34% being either ‘mixed’ ore or sulfide ore. The average weighted cyanide consumption is calculated as 0.66% x 0.77 + (1-0.66%) x 0.59 = 0.59 kg/t.

The additional free cyanide required is calculated to be 0.14 kg/t, thus the total leach cyanide consumption is expected to average 0.77 kg/t.

The following optimal leach conditions were determined from a series of tests:

- Initial pH of 11 adjusted with 60% available CaO;
- Initial sodium cyanide concentration of 0.05% w/v maintained at > 0.025% w/v;
- Pulp density of 55% w/w;
- Air sparging to maintain dissolved oxygen (DO) level at or greater than 9 ppm, oxygen uptake after 6 hours was 0.23 mg/L/min;
- 100 g/t lead nitrate addition; and
- 24 hour leach residence time and grind P_{80} of 90µm.

13.6.5 Cyanide Detox Tests

Two leach residue samples from the master composite leach test program were used for cyanide detox tests. The following analytical methods were applied to detox head and tail solutions:

- Copper, Iron, Nickel, Zinc in solution: Direct inductively coupled plasma optical emission (ICP-OES);
- Cyanide – weak acid dissociable (WAD): Picric acid method; and
- Cyanide – free: Silver nitrate titration.

Cyanide speciation on the detoxification feed and products was performed by the Chemistry Centre of Western Australia. Several small-scale cyanide detoxification tests using the SO_2/air process were conducted. The target was to achieve a final WAD cyanide level below 10 ppm. The slurry used in the cyanide destruction testwork was taken from a bulk leach test performed under the optimized conditions at 90 µm grind.

The conclusions from the testwork are as follows:

- The air–SO_2 method successfully reduces CN_{WAD} to levels of <10 ppm.
- An SO_2:CN ratio of 4.3 is required to reduce CN_{WAD} levels to <10 ppm.
- There is sufficient dissolved copper in the leach solution for precipitation of copper iron cyanide compounds, no need for additional copper.
- One hour detox residence is sufficient for the process.
- Lime dosage requirement was determined to be a ratio of 1.3 g of lime per g of SO_2.

13.6.6 Bulk Flow Characterization Testwork

A materials handling study was undertaken by The University of Newcastle Research Associates (TUNRA) to determine the flow properties of a sample of waste composite at the as-supplied moisture level of 0.3%. The TUNRA tests provide relevant parameters for the design of efficient and reliable bulk storage and handling facilities.

The results indicate a moderately easy handling material with low bulk strength which increases slightly after 3 days undisturbed storage. The material has a very low propensity to form stable ratholes. Tests indicated moderately small critical arching dimensions with moderately steep half angles for typical wall materials. Wall friction measurements at 0.3% moisture indicate reasonably low wall friction.

13.7 Additional Testwork Conducted in 2012

13.7.1 General

Thickener and leach slurry tests performed during the 2011 campaign were based on grind P_{80} of 120 µm. For design purposes, additional tests were performed in 2012 on slurry at a grind P_{80} of 90 µm. The tests included the following:

- Thickener settling testwork, settling rate, thickener flux, final compaction achieved, and flocculant consumption.
- Leach and CIP circuit viscosity testwork to determine the optimum leach circuit density.

Leach residue samples were prepared by re-grinding previous leach test samples to a P_{80} of 90 µm, this was used for rheology tests. A newly ground sample was prepared from the remaining master composite for the thickening testwork.

13.7.2 Leach / CIP Rheology Testwork

Sample Preparation

Thirty samples from the variability composite at P_{80} of 90 µm were dispatched from ALS laboratories to SPX in Sydney for rheology testwork.

Rheology Testwork

The aim of the new testwork was to determine the maximum tolerable CIP slurry density. The previous coarser grind material had a tolerable density of 60% solids.

The addition of lime has the potential to significantly affect the viscosity of the leach slurry, therefore the tests were conducted at a pH of 10.5 by the addition of hydrated lime. The tests also provided information on the following:

- Mixing efficiency and ability to achieve a uniform solids suspension.
- Ability to re-slurry following a power failure.
- Tank design requirements (diameter to height ratio, number of baffles etc.).
- Size and budget cost information for a CIP tank agitator.

Rheology Testwork Results

Based on the rheology test results and the results of physical solids re-suspension testwork, the SPX recommendation for agitation design based on various tank sizes is shown in **Table 13-1**.

Table 13-1: SPX Agitator Size Recommendations

Item	6000 m^3 Tank	5500 m^3 Tank	5000 m^3 Tank	4500 m^3 Tank
50%wt solids	S783 – 132 kW	S783 – 132 kW	S783 – 132 kW	S783 – 132 kW
55%wt solids	S784 – 150 kW	S783 – 132 kW	S783 – 132 kW	S783 – 132 kW
60%wt solids	Hansen – 185 kW	Hansen – 185 kW	784 – 150 kW	784 – 150 kW

SPX commented as follows:

- Operating at 55%wt solids seem to be optimal in terms of power requirement;
- A dual down-pumping axial flow impeller combination is recommended;
- The optimal tank height-to-tank-diameter ratio is 1:1. Any changes to tank dimensions could have a significant impact on agitator selection, thus cost; and
- The tank would require three baffles equi-spaced at 120°.

13.7.3 Thickener Testwork

The testwork was undertaken by Outotec Oyj. (Outotec) to determine the thickener size, flux rate and flocculant consumption. Approximately 20 kg of the Master Composite 3.35 mm crushed sample was ground to 80% passing 90 µm for the thickening testwork. The outcome of the testwork results are as shown in the **Table 13-2.**

Table 13-2: Outotec Thickener Recommendations

Item	Units	Base Case	Alternate Case
Solids Feedrate	tph	2265	1495
Solids Loading	t/m^2h	1.30	1.30
Feed Slurry Density	% solids	12.5	12.5
Slurry pH	pH	7.5	7.5
Flocculant Dosage	g/t	15	15
Underflow Density	% solids	61-63	61-63
Overflow Clarity	ppm	<250	<250
Thickener Diameter	m	45	37

13.8 Historical Review

The Mt. Todd project was an operating gold mine in the 1990's. Previous operators successfully recovered gold from the oxide portion of the deposit, but encountered difficulties in processing the ore as the mine transitioned from the oxide heap leach operation to a sulfide milling operation. Some of the metallurgical challenges encountered, but not adequately addressed at that time were: hard ore (23.5-Bond ball mill work index), cyanide-soluble secondary copper minerals, and inefficient flotation sulfide mineral recovery resulting from presence of free cyanide in the process make-up water. Vista acquired the Project with the belief that each of these challenges could be overcome through the use of current technology, adequate metallurgical testing and higher gold prices.

In 2006, Vista retained RDi to evaluate the metallurgical characteristics of the Mt. Todd Project and develop a process flowsheet that would optimize the recovery of gold through the efficient use of proven processing technologies. Testwork has also been undertaken at several other testing facilities including; Krupp Polysius Research Center Germany, JK Tech Pty. Ltd. Australia, Pocock Industrial, Inc. Utah, and Kappas, Cassidy and Associates Nevada. The extensive metallurgical testwork has resulted in an economically viable process flowsheet which has overcome the metallurgical challenges encountered by earlier operators.

The process flowsheet discussed in this section has the following significant advantages over earlier processing options:

- Better characterization of the resources at site has indicated that copper may not be as important an issue as indicated by a reviewer of the historic processing challenges encountered by earlier operators. This has resulted in the development of the ore-cyanidation leach process presented in the process flowsheet;
- Incorporation of the HPGR technology in the comminution circuit to handle the extremely hard and coarsening of the grind has resulted in a significant reduction in the energy requirement for the proposed flowsheet; and
- Pre-aeration of the ground ore with lime has resulted in a reduction of the cyanide consumption in the process.

These processing advantages combined with higher gold price significantly improve the viability of the proposed operation.

13.8.1 Historical Review of Conceptual Process Flowsheet

RDi reviewed historical metallurgical testwork for the project conducted in 2006 and proposed a conceptual process flowsheet that could potentially overcome the technical problems encountered by previous operators.

The proposed flowsheet consisted of crushing and grinding the ore followed by floating the sulfides and gold in the rougher flotation. The objective of the rougher flotation step was to maximize recoveries of gold, copper and other sulfides. Rougher tailings would have negligible amounts of sulfides and would be non-acid generating thereby allowing the tailings to be sent to the existing tailings pond. Rougher concentrate containing 85% or more of the gold content in the ore would be reground and selectively floated to recover copper and gold in a cleaner concentrate which would assay over 20% Cu. The concentrate would contain approximately 50% of the gold and would be sold to a smelter. Cleaner tailings would be cyanide leached in the CIL circuit. Leach residue would be subjected to cyanide destruction and the sulfides would be sent to a separate tailings pond. The tailings pond would be constantly monitored to ensure that acid is not generated.

To confirm this flowsheet, RDi undertook a testing program in late 2006 utilizing core samples provided by Vista. The core samples consisted of approximately 3 kg each of ten drill-core reject samples stored for several years. The composite sample prepared for the study assayed 1.78 g Au/t, 448 ppm Cu, and 1.43% S_{Total}. Based on sequential copper analyses, the copper present in the composite consisted of three percent oxide copper, 63%secondary copper and 34% primary copper. The major sulfide mineral in the sample was pyrite. Froth flotation using a simple reagent suite consisting of potassium amyl xanthate, Aeropromotor 3477 and methyl isobutyl alcohol recovered approximately 82% of gold and 90% of copper in a rougher concentrate at a primary grind of P_{80} of 200 mesh. Following regrind, the rougher concentrate was upgraded to ± 19% Cu in two cleaner flotation stages. Additional cleaner stages could not be tested due to limited sample availability. Cyanide leaching of the cleaner tailings which contained ± 35% of the gold extracted 84% of the gold in the tailing. The limited open-circuit testwork indicated that the proposed conceptual process flowsheet should work for the deposit.

13.8.2 Review of Metallurgical Testwork

Vista conducted the exploration programs on the Mt. Todd Project in 2007. Part of the core from the 2007 drilling program was used for metallurgical testing to confirm the conceptual process flowsheet. The composite sample was very hard (Bond ball mill work index of 23.9 Kwh/t) and averaged 1.37 g Au/t, 447 ppm Cu and 0.92% S_{Total}. The metallurgical testwork indicated that gold recovery into the rougher flotation concentrate was ± 80% at a primary grind of P_{80} of 200 mesh. Copper in the rougher concentrate could not be upgraded to provide concentrate assaying ± 20% Cu. The best results were ± 6% Cu using the same test procedure as employed for earlier core testing (2006).

Similar metallurgical results were obtained on a composite using 2008 core samples. This composite assayed 0.89 g Au/t and 450 ppm Cu. The poor metallurgical performance results obtained on the 2007/2008 core sample composites prompted a study to determine the reasons for the differences in metallurgical response compared to the historic core. The results indicated that historical core had copper predominantly as secondary copper which is known to be a major consumer of cyanide (**Table 13-3**). The major sulfide mineral was pyrite. However, 2007 and 2008 drill core had primary copper as predominant copper species and pyrrhotite as major sulfide mineral. Pyrrhotite is known to float readily as compared to pyrite and is significantly more difficult to depress in the flotation process. Thus, it was difficult to selectively float copper minerals and produce a copper concentrate.

As a result of flowsheet changes and the incorporation of HPGR technology, power requirements have dropped.

Historical drill core stored at site, i.e., sample material used in the earlier conceptual studies, was predominantly from the transition zone. Subsequent studies have confirmed that ore with similar characteristics (i.e., transition zone sulfide minerals) accounted for less than five percent of the remaining resources at the mine. Over 95% of the resources were typical of ore encountered in 2007 and 2008 drilling. Hence, copper may not be as important an issue as indicated by a review of the historical processing challenges encountered by earlier operators.

Table 13-3: Assays of Various Composite Samples

Parameter	Historical Core	2007 Drilling	2008 Drilling
g Au/t	1.78	1.3	0.89
Cu_{Total}, ppm	448	447	450
$Cu_{AcidSol}$, ppm	14	19	24
Cu_{CNSol}, ppm	295	68	65
S_{Total}, %	1.42	0.92	
Cu Distribution, %			
Oxide	3.1	4.3	5.3
Secondary	65.8	15.3	14.4
Primary	31.1	80.4	80.3
Primary Sulfide Mineral	Pyrite	Pyrrhotite	Pyrrhotite

While this ore characterization study was on-going, the issue of ore hardness was also evaluated by RDi. It is widely recognized that the energy required to grind the material to a desired size in a conventional flowsheet increases as the hardness of the ore increases. Taking advantage of the basic principle “that it is cheaper to crush than to grind” since crushing requires less energy than grinding, testwork was undertaken to evaluate HPGR in order to reduce energy requirements for the process flowsheet. Based on subsequent laboratory studies, the energy requirements for the flowsheet shown in **Figure 13-1** was determined. The results found in **Table 13-4** indicate a significant reduction in power requirements by incorporating HPGR in the grinding circuit and changing the process to whole ore leach at a coarse grind size. As a result of flowsheet changes and the incorporation of HPGR technology power requirements dropped from 33.70 kwh/t to 18.11 kwh/t. The reduction in energy consumption was ± 25% when HPGRs were incorporated into the circuit. JK Tech Pty Ltd. conducted comminution tests on five samples of drill core from Mt. Todd Mine for Vista. This testing included SAG Mill Comminution, Bond Rod Mill Work Index, Bond Ball Mill Work Index, Bond Abrasion Index and HPGR testing. These results confirmed earlier finding that the ore was “very hard”, compared to a database of other ores, and this hardness did not exhibit a large variability across the range of samples tested.

Ausenco undertook a technical evaluation of the various comminution circuits based on the testwork undertaken by JK Tech Pty Ltd. They evaluated six different processing options and concluded that Vista should adopt a comminution flowsheet based on a secondary crush, HPGR and ball mill circuit for treating the Batman deposit. This circuit would have 23% reduction in energy requirements over the conventional semiautogeneous mill, ball mill, and pebble crusher (SABC) circuit.

Table 13-4: Energy Requirements for Different Process Flowsheets

	Flotation Process (P_{80}=200 mesh)	Direct Leach (P_{80}=100 mesh)
Conventional Crush/Grind		
Power, kwh/t	33.70	24.06
Steel, kg/t	0.72	0.66
HPGR/Grind		
Power, kwh/t	24.22	18.11
Steel, kg/t	0.79	0.72

A decision was made not to recover copper as by-product as a result of better understanding the mineralogy of the Batman deposit through the metallurgical testing completed on the drill core from the 2007 drill program. RDi evaluated a whole ore leach option to determine the viability of this flowsheet at a coarser grind. Based on past experience, pyrrhotite can be pacified with a pre-aeration of the pulp at pH 11. The process flowsheet evaluated for whole ore leach is given in **Figure 13-1**.

Testwork was systematically undertaken to evaluate and optimize the various process parameters one-at-a-time. The parameters evaluated included grind size, pre-aeration time, cyanide concentration (in both maintained and decay modes), leach time and carbon-in-pulp gold recovery (CIP). The successful completion of each subsequent test and the definition of the optimal range of the corresponding variables resulted in an improvement in the process flowsheet. As this was a process that occurred over a period of time, the CIP test was the last variable tested. Results from the CIP tests, shown in the **Table 13-5**, incorporate the optimal ranges determined by previous tests. It is important to note that the results of the CIP tests are best estimates of the expected gold recovery from the proposed process flowsheet. Carbon adsorption of the gold and subsequent gold assay of the carbon reduces the inherent sampling and assaying errors of direct measurement of low grade solutions.

The Mt. Todd Project can be expected to recover a nominal 82% of the contained gold (81.7% net of solution loss) with the proposed process flowsheet.

RDi provided cyanide leach residue to Pocock Industrial, Inc. to develop data for design of thickening and filtration equipment for the Project. The testwork undertaken included flocculant screening tests, conventional and dynamic thickening tests, viscosity tests and vacuum filtration tests to size horizontal belt filters. The highlights of the study indicated the following:

Results from particle size analyses showed the leach residue to have a P_{80} of 195 µm.

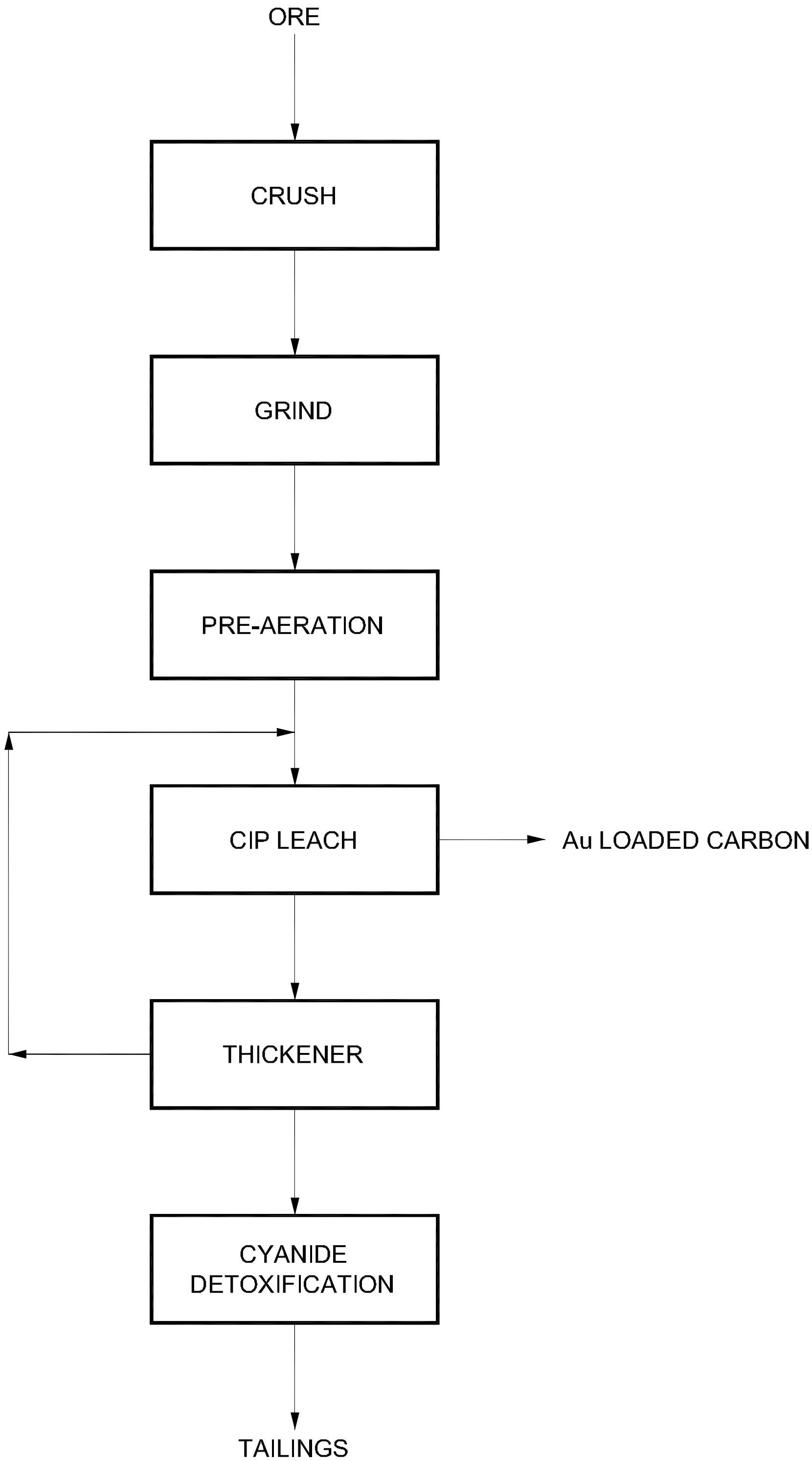
- The flocculant selected for the study was high molecular weight, low charge density anionic polyacrylamide (Hychem AF303).
- The unit area for conventional thickening was determined to be 0.125 m^2/tpd with 70% underflow solids using 10-15 g/mt of flocculant.
- The design basis for a high rate thickener was determined to be 7.33 m^3/m^2hr of feed loading with maximum 70% underflow solids.
- For paste thickening (74 to 75% solids), the recommended design basis net feed loading was determined to be 7.3 to 8.3 m^3/m^2hr.
- The horizontal belt filtration rate ranged from 65.88 to 1076 dry kg/m^2hr depending on the moisture content of the filter cake (i.e., 15 to 18%).

Kappes, Cassiday and Associates undertook limited tailing characterization testwork which included detoxification of leached tailings followed by characterization and environmental testing of the detoxified tailings. The SO_2/air process produced less than 50 ppm CN_{WAD} cyanide following the detoxification process using 2.3 g of sodium metabisulfite (SMBS) per g of total cyanide.

Table 13-5: Leach Test Results (P_{80}=100 mesh)

Test No.	Cyanide Maintain/ Decay	Leach Time, Hours	Extraction %		Residue g Au/t	Cal. Head g Au/t	NaCN Consumption Kg/t
			Au	*Cu*			
72	Decay	24	82.6	13.5	0.20	1.14	0.60
76	Decay	30	80.4	14.3	0.20	1.03	0.54
78	Maintain	30	82.2	14.5	0.17	0.93	0.60
80	Decay	36	82.2	15.0	0.14	0.79	0.54
82	Maintain	36	84.0	16.3	0.14	0.85	0.59
99	Decay	CIP 24+6	82.3	14.1	0.19	1.05	0.52
100	Decay	CIP 24+6	82.0	15.6	0.18	1.01	0.58
101	Decay	CIP 24+6	85.4	14.4	0.15	1.04	0..40
102	Decay	CIP 24+6	86.7	14.4	0.15	1.15	0.46

Leach tests at 40% solids, pH 11 with 1 g/L NaCN initial addition. CIP tests run with 20 g/L carbon added after 24 hrs. All tests have 4 hours pre-aeration.



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Vista Gold Corp.***	Fig16-1.dwg	**Figure 13-1** **Leach Process Flowsheet**
	Project: Mt Todd Gold Project	Project Number: 114-311156	
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	

14.0 MINERAL RESOURCE ESTIMATE

The resource estimation of the Batman deposit is updated from the October 4, 2012 "NI 43-101 Technical Report Resource Update Mt. Todd Gold Project" prepared by Tetra Tech. More detailed description of the method, statistics and modeling of the Batman deposit is found in the reports listed in the Reference Section.

This report includes an estimate of gold contained in a historic heap leach pile adjacent to the Batman deposit. The method of estimation is described herein. In addition, this report contains the resource estimation of the Quigleys deposit which has not been updated for this report. The resource update for the Quigley deposit was published in the "10.65 MTPY Preliminary Feasibility Study NI 43-101 Technical Report Mt. Todd Gold Project" prepared by Tetra Tech, January 2011. The Mt. Todd Gold Project's updated resource estimates are shown in **Table 14-1**. The location for these resources is shown in **Figure 14-1**.

Table 14-1: Mt. Todd Statement of Mineral Resources

	Batman Deposit			Heap Leach Pad			Quigleys Deposit		
	Tonnes (000s)	**Grade (g/t)**	**Contained Ounces (000s)**	**Tonnes (000s)**	**Grade (g/t)**	**Contained Ounces (000s)**	**Tonnes (000s)**	**Grade (g/t)**	**Contained Ounces (000s)**
Measured	77,793	0.88	2,193	-	-	-	571	0.98	18
Indicated	201,792	0.80	5,209	13,354	0.54	232	6,868	0.82	181
Measured & Indicated	**279,585**	**0.82**	**7,401**	**13,354**	**0.54**	**232**	**7,439**	**0.83**	**199**
Inferred	72,458	0.74	1,729	-	-	-	11,767	0.85	320

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

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

1. Batman deposit;
2. Existing Heap Leach Pile; and
3. Quigleys deposit.

Only the Batman deposit, the Heap Leach Pile and the Quigleys deposit currently have resource estimates classified in accordance with CIM Standards.

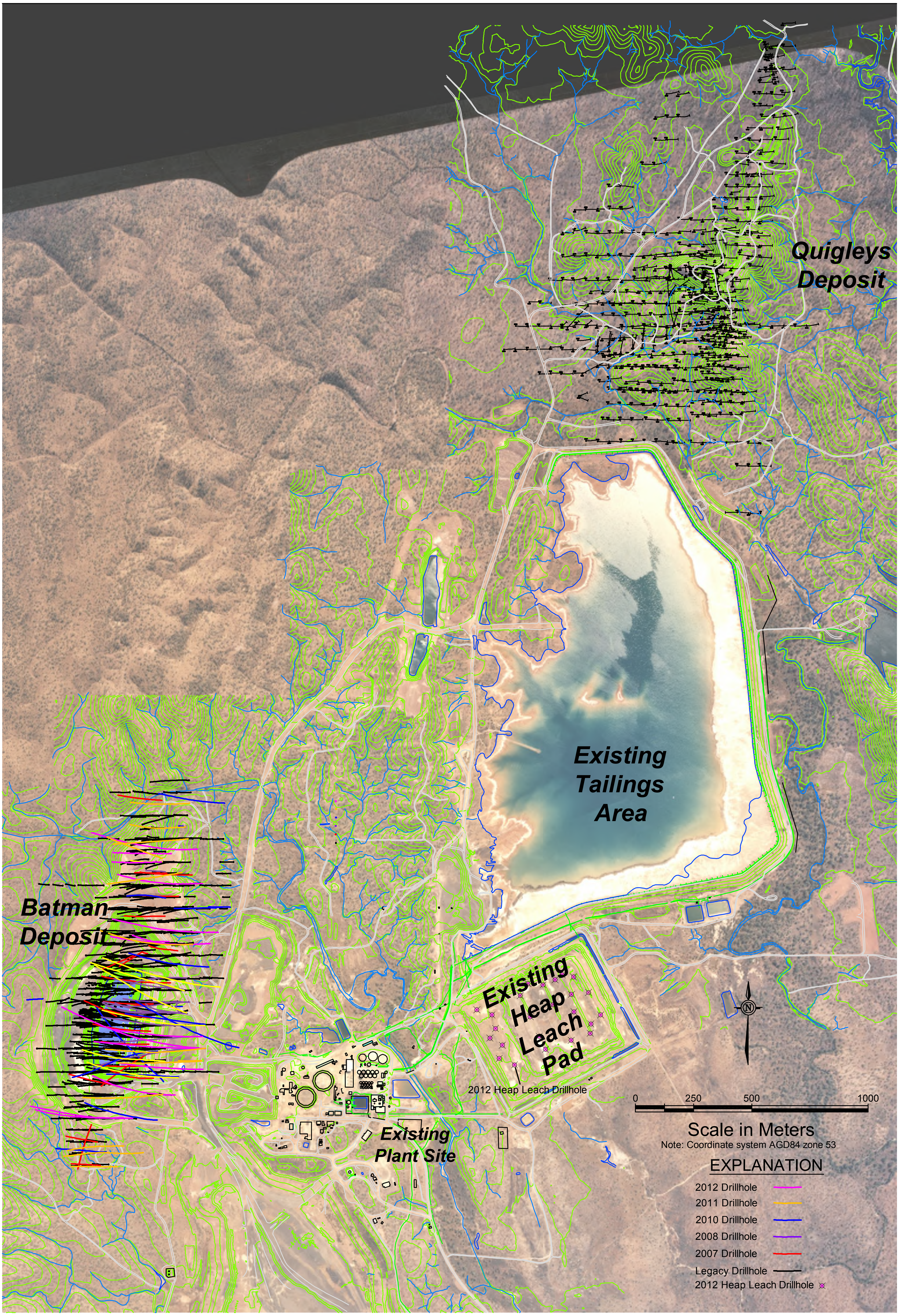
14.1 GEOLOGIC MODELING OF THE BATMAN DEPOSIT

The Batman deposit resource has been updated to reflect the increase in available data provided by drilling conducted in 2012.

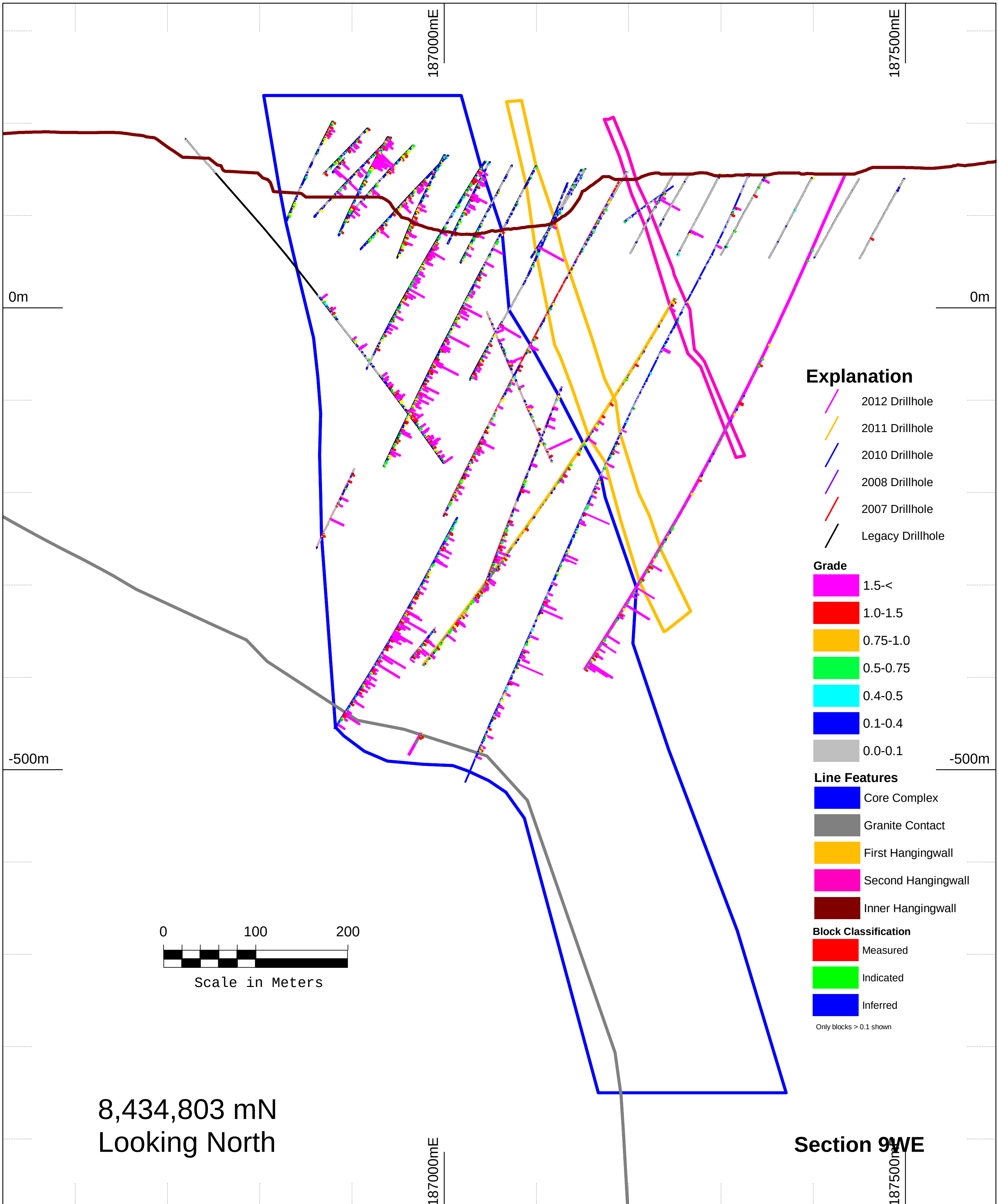
The core complex wireframe solid, which represents the main body of the mineralized shear zone, was adjusted and resized to accommodate this new data. Deep step-out drilling by Vista has indicated an inflection point in the lower footwall of the core complex previously not drill tested (**Figure 14-2**).
The flexure feature correlates well with the previously indicated higher grade plunge of the core complex. Vista site geologic staff has put considerable effort 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 which can be seen in **Figure 14-2**.

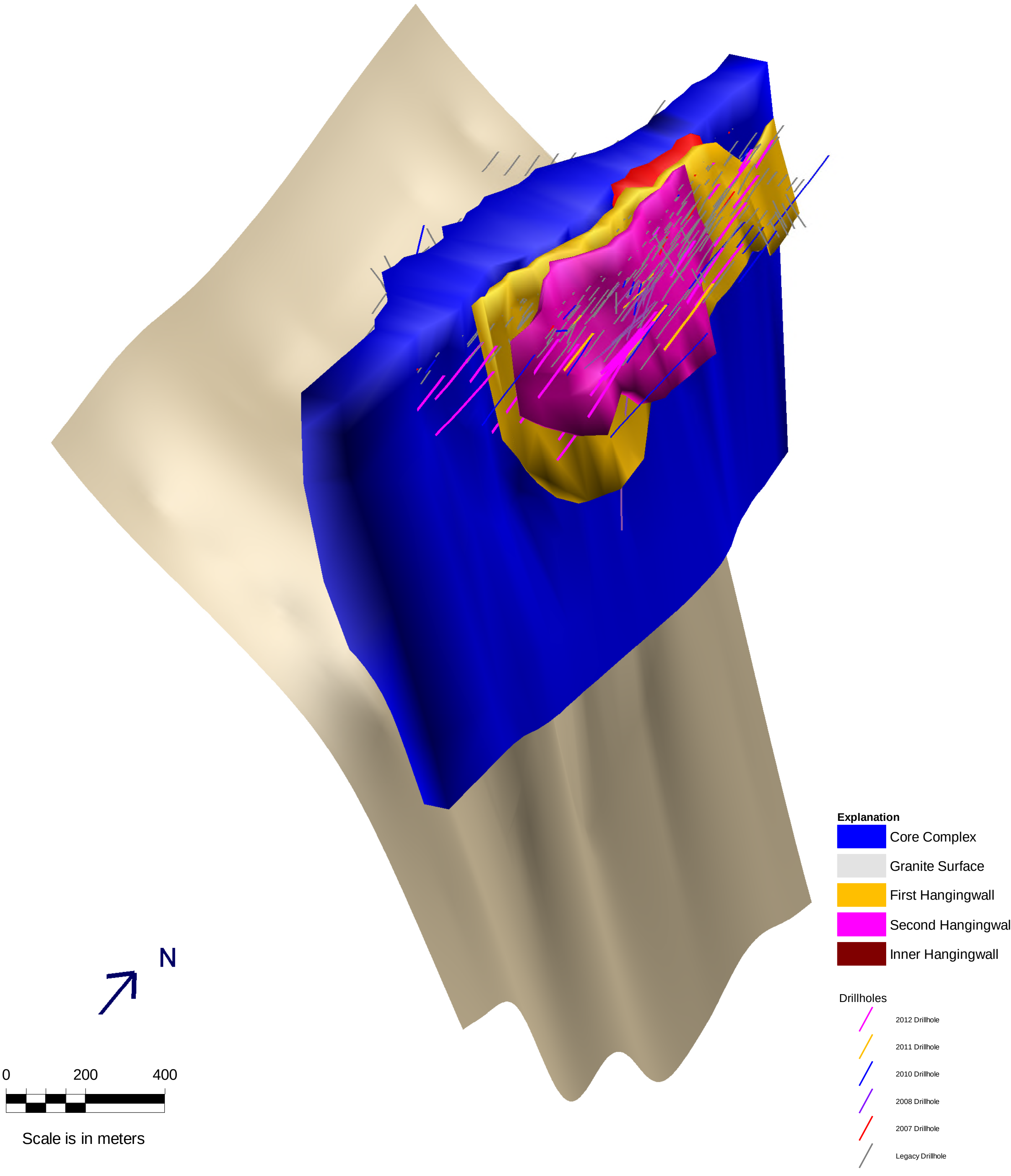
In addition to resizing the core complex wireframe solid, three structures paralleling the core complex to the east were also resized and constructed into wireframe solids and used for this resource estimate. The interpreted parallel structures represent an echoing of the main mineralization controls of the core complex nearer the surface and to the east. Wireframe solids for the parallel structures were interpreted on section using Au mineralization, veining percentage, visual sulfide percentages, structural orientations and multi element data. Deep drilling conducted in 2011 and through 2012 confirmed the existence of these structures and indicates a possible increasing definition and grade at depth. **Figure 14-3** shows the Batman deposit in a 3 Dimensional View.



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Vista Gold Corp.*	Fig14-1.dwg	Figure 14-1: Drillhole Location Map Batman & Quigleys Deposits and Heap Leach Pad
	Project: Mt Todd Gold Project	Project Number: 114-311285	
	Project Location: Northern Territory, Australia	Date of Issue: June 2013	



Issued By: TETRA TECH	**Prepared for:** ***Vista Gold Corp.***	**File Name:** Figure 14-2.pdf	**Figure 14-2: Geologic Modelling Cross Section 9WE Batman Deposit**
	Project: Mt. Todd Gold Project	**Project Number:** 114-311285	
	Project Location: Northern Territory, Australia	**Date of Issue:** June 2013	



Issued By:	Prepared for: *Vista Gold Corp.*	File Name: Figure 14-3.pdf	Figure 14-3: Geologic Modeling 3D View - Batman Deposit
TETRA TECH	Project: Mt. Todd Gold Project	Project Number: 114-311285	
	Project Location: Northern Territory, Australia	Date of Issue: Month 2013	

Batman Deposit Density Data

A total of 16,373 samples were tested for bulk density (diamond core). These bulk densities 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 resource model are presented in **Table 14-2**.

Table 14-2: Summary of Batman SG Diamond Core Data by Oxidation State

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

In addition, one hundred fist-sized grab samples (50 from 1060 level and 50 from 1040 level) were collected and sent to Assay Corp. for moisture and bulk density determination; results are presented in **Table 14-3**. Results show that the average moisture content is less than one percent and the average specific gravity for the 1060 RL (all primary) is 2.77 and 1140 RL (mixture of primary and transitional) is 2.74. These results match the predicted specific gravity within the existing and new block models.

Table 14-3: Batman Pit Sample SG Data

	1060-1068 RL		1146- 1140RL	
	SG	*Moisture%*	*SG*	*Moisture%*
Number of samples	50	50	50	50
Average bulk density (t/cm)	2.77	0.01	2.74	0
Median bulk density (t/cm)	2.78	0	2.76	0
Maximum bulk density (t/cm)	2.88	0.18	2.83	0.07
Minimum bulk density (t/cm)	2.54	0	2.52	0
Standard deviation.	0.05	0.03	0.07	0.01

14.3 DRILLHOLE DATA BATMAN

An Access® database was set up in Gemcom recreated from the previous exploration database. Tables for the grade control database were inserted into this database. **Table 14-4** shows parameters from the database used in the resource estimate.

Table 14-4: Mt. Todd Project Access® Database

Table Name	Description
tblDHCollar	Contains all collar details with drillhole coordinates in either MGA94 or AMG84, Zone 53
tblDHAlteration	Alteration logs of Batman_2010 and Goldeneye_2010 drillholes
tblDHAlteration_old	Alteration logs of Batman_2007, Batman_2008 and Pre2007 drillholes. These drillholes were logged using a different spreadsheet structure – where each key alteration mineral has its own column heading.
tblDHAssays	All gold assays, with Lab Batch number recorded where available. There is NO calculated Au field averaging the lab repeat values.
tblDHAssays_ME	All other multi-element assays with Sample_ID only, and Lab Batch number where available.
tblDHGeol	All geological logs based on the 2010 logging system. Where possible data (matching columns) from Pre2007 logs were imported into this 2010 system.
tblDHGeotech	Full geotechnical logs from Batman_2007, 2008 and 2010 and Goldeneye_2010 drilling were imported. Some Recoveries, RQD, Hardness and Fracture% data were extracted from the Pre2007 data.
tblDHQAQC	Standard, blanks, field duplicates, pulp repeats, coarse rejects assays both Au and multi-elements from Batman_2007, 2008 and 2010 and Goldeneye_2010 drilling.
tblDHStructure_Orientation	All structure data using the Batman_2010 file structure (many do not have a Beta measurement)
tblDHSulfides	Sulfide data logs of Batman_2010 and Goldeneye_2010 drillholes
tblDHSulfides_old	Sulfide data logs of Batman_2007, Batman_2008 and Pre2007 drillholes. These drillholes were logged using a different spreadsheet structure – where each key sulfide mineral has its own column heading.
tblDHSurvey	Down hole survey data from all datasets. Survey method/type was recorded wherever the data was available.
tblDHVeins	Vein data logs of Batman_2010 and Goldeneye_2010 drillholes

14.3.1 Batman Exploration Database

Table 14-5 is a summary of the Batman exploration database that formed the basis of the resource estimation of that deposit.

Table 14-5: Summary of Batman Exploration Database

Drillhole Statistics						
	Northing (m) MGA94 z53	**Easting (m) MGA94 z53**	**Elevation (m)**	**Azimuth**	**Dip**	**Depth (m)**
Minimum	8,434,393.4	187,066.8	144.4	0.0	51.6	177
Maximum	8,435,954.0	187,446.7	169.8	833	67.8	833.3
Average	8,435,150.8	187,308.8	152.8	268.1	59.8	587.6
Range	1,560.64	379.85	25.4	564.9	16.2	656.28
Cumulative Drillhole Statistics						
Total Count	826					
Total Length (m)	143186					
Drillhole Grade Statistics	**Number**	**Average**	**Std. Dev.**	**Min.**	**Max**	**Missing**
Au (g/t)	129,894	0.5415	1.262	0.001	77.70	1631
Cu (%)	34,336	0.03557	0.05837	0.000	2.4	97,189

The pre-2007 exploration database consisted of 743 drillholes, 226 diamond holes and 517 percussion holes. A total of 97,810 samples existed within that exploration database. Diamond core was a combination of NQ and HQ, with the NQ core being sawed into half splits and the HQ core being sawed into quarter splits.

Problems were identified from the original Batman exploration database:

- Only one gold field existed in the database called "Au Preferred". Au Preferred was a factored gold grade;
- Zones of non-assayed mineralized core were incorrectly coded and given 0 grade; and
- Some samples with assays below detection have been incorrectly coded as not sampled.

Original assays from logs and/or laboratory assay sheets have shown that there are up to 15 gold assay fields (five different splits with three gold fields). The Au Preferred field is usually the average of the gold assay, but with the early data, notably the Billiton data, the Au Preferred has been factored. Exactly how this factoring was calculated is uncertain. Billiton reports suggest that different laboratories along with the orientation of drillholes have impacted on the grade returned from the laboratory and factors to counter this have been applied in the calculation of the Au Preferred field.

MicroModel® files were located and contained 80% of the original assay data. Inspection of these data showed that codes, in some cases, were used for below detection (- 0.800 or - 0.008) while other times below detection was given a grade (0.005 or 0 or 0.001) instead of the code. Missing samples were given a code (- 0.900 or - 0.009 or - 0.700). Sometimes these codes have been misused with below detection codes being used instead of missing samples and vice versa. This has impacted on the Au Preferred field in the database. Original lab assay data sheets and logs were used to address this problem.

After reviewing all the logs and laboratory assays, the data were corrected and reloaded into the database. Codes were allocated, with below detection assays given a grade of 0.005, which is half the detection limit of 0.01, and missing samples were given a code –9.000.

The assays in the database have been split into different tables to save room and make the processing of the data more efficient. The gold fields have been split up into six different tables, depending on the number of duplicate samples. Gold1 is the first assay taken, Gold2 the second assay taken and so on to Gold5. An AuAv (average gold grades) table has also been added for the average gold grade from the five gold assay tables. The Au Preferred field has been retained in the present drillhole database. A separate table has also been created for the multi-element data.

The resource update described in this report is supported by 13 core holes drilled in 2012. The 13 core holes added and used for this resource update account for 7,705 Au assay intervals for 7,639 linear meters of sampling.

In 2011, Vista commissioned a third party contractor to construct a project database consolidating all available drillhole data. The database includes 13 data tables, which are shown in **Table 14-4** above. Instances where tables are tagged as "old", current data collection and interpretation and previous data collection and interpretation were incompatible and unable to be reconciled into one table. Vista has continued to use the database format described above for all exploration drilling between 2011 and December, 2012.

14.4 Batman Block Model Parameters

Table 14-6 details the physical limits of the Batman deposit block model utilized in the estimation of mineral resources.

Table 14-6: Block Model Physical Parameters – Batman Deposit

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

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

14.5 Batman Resource Estimate Over Time

The geologic model of the Batman and Quigleys deposits was originally created by General Gold and audited by Tetra Tech. The resource update of the Batman deposit the geologic model was updated by Tetra Tech to accommodate additional drilling done in 2011 and through early September 2012. The geologic model was constructed by creating three-dimensional wire-frames of the main geologic units, oxidation types, and mineralizing controls and super-imposing them on each other to create an overall numeric code that details all of the input parameters. Genera l Gold created the model based on the prior work of others, recommendations of other consultants, and General Gold's own experience. It is Tetra Tech's opinion that the General Gold geologic model and the updates made accurately portray the geologic environment of the Batman deposit.

Tetra Tech used the geologic model to guide the statistical and geostatistical analysis of the gold assay data. The analysis of the gold assays further confirmed the geologic divisions made by in the geologic model. Gold grades were estimated into the individual blocks of the model by ordinary, whole-block kriging. The estimate was prepared using MicroMine®, GEMCOM® and MicroModel® software

Variograms and kriging search parameters are the same as those used for the more detailed Appendix A of the January 28, 2011 PFS study. A more detailed discussion of the findings of a series of geostatistical studies can be found therein.

The rock model was then assigned a tonnage factor based on the oxidation state (i.e., oxidized, transition, primary). The tonnage factors were based on a number of tests from the core and, in Tetra Tech's opinion, are representative of the various rock units, and are acceptable for estimation of the in-place geologic resources.

The updated mineral resource estimate includes 13 drillholes (7,639 total m) from Vista's ongoing resource conversion drilling program at the Mt. Todd Gold Project's Batman deposit. Growth of the Batman deposit resource estimate over time is outlined in **Table 14-7**.

The estimated gold resources were classified into Measured, Indicated, and Inferred categories for both the Batman and Quigleys deposits according to the parameters detailed in **Table 14-13** through **Table 14-17**.

Table 14-7: Progression of Resource Estimate – Batman Deposit

Category	Tonnes (x1000)	Average Grade g Au/t	Total Au Ounces (x1000)
March 2013			
Measured	77,793	0.87	2,193
Indicated	73,146	0.80	5,209
Measured & Indicated	279,585	0.82	7,401
Inferred	72,458	0.74	1,729
September 2012			
Measured	75,101	0.88	2,127
Indicated	186,299	0.81	4,879
Measured & Indicated	261,400	0.83	7,007
Inferred	88,774	0.73	2,093
September 2011			
Measured	67,166	0.88	1,897
Indicated	154,836	0.82	4,089
Measured & Indicated	222,022	0.84	5,987
Inferred	103,563	0.78	2,612
January 2011 PFS			
Measured	52,919	0.91	1,543
Indicated	138,020	0.81	3,581
Measured & Indicated	190,939	0.84	5,125
Inferred	94,008	0.74	2,244
June 2006 [3]			
Measured	22,095	0.89	629
Indicated	45,715	0.88	1,294
Measured& Indicated	67,810	0.88	1,923
Inferred	61,754	0.84	1,672

Tonnage, grades and totals may not total due to rounding.
All estimated resources are shown using a 0.4 g Au/t cutoff grade.
Vista's first mineral resource estimate for the Batman deposit.

Table 14-8: Batman Resource Classification Criteria

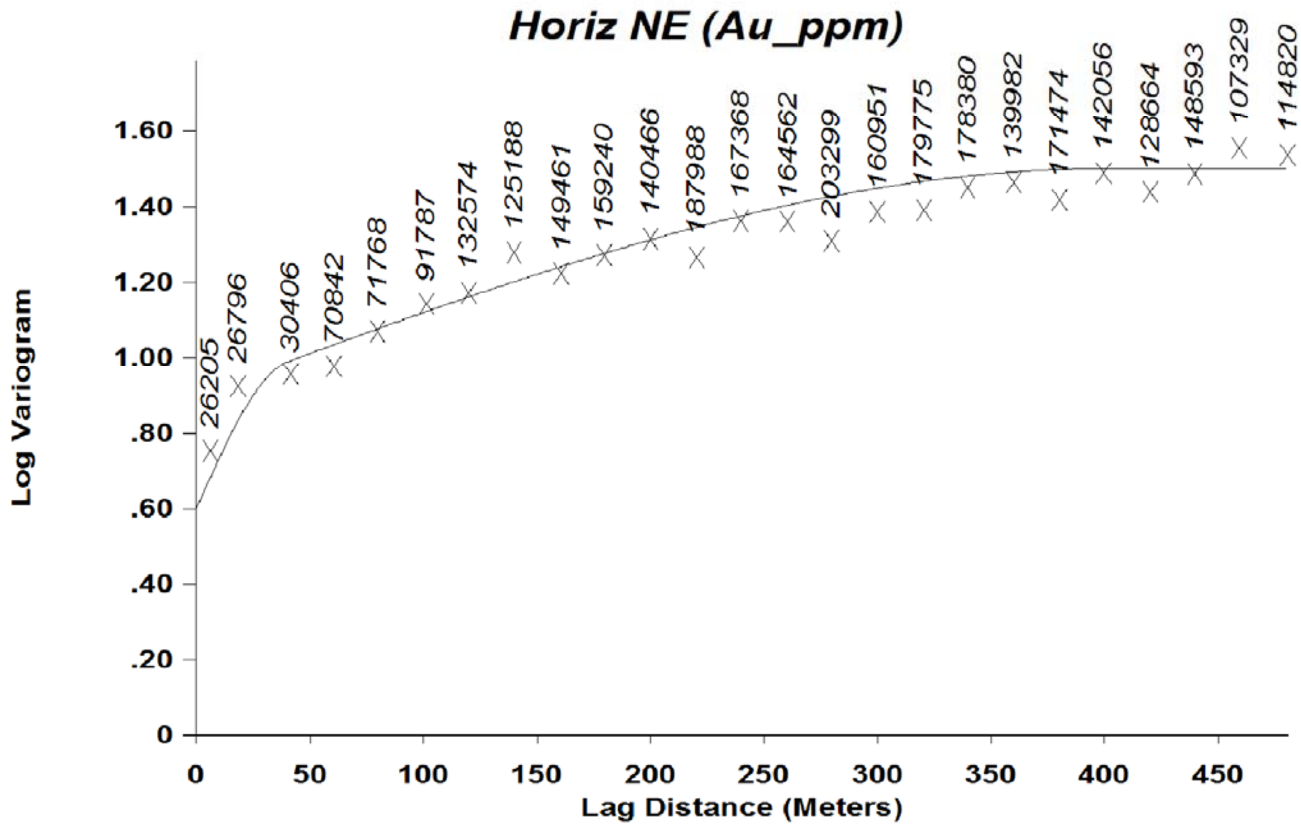
Category	Search Range & Kriging Variance	No. of Sectors/ Max Points per DH	Search Anisotropy	Min Points	Composite Codes	Block Codes	CORE
Indicated	Core Complex: 150 m & KV < 0.34 Pass 1	4/2	(1.0:0.7:0.4) [110:80:0]	2	1000	1000	CORE COMPLEX
Measured	Core Complex: 60 m & KV < 0.30) Pass 2 (overwrite Pass 1)	4/3	(1.0:0.7:0.4) [110:80:0]	4	1000	1000	
Inferred	Core Complex KV >= 0.34 Classification Step	NA	NA	NA	1000	1000	
Inferred	Outside Core Complex: 150 m & KV <= 0.45 Pass 3	4/3	(1.0:0.7:0.4) [110:80:0]	3	500/3500	500/ 3500	OUTSIDE CORE COMPLEX
Inferred	Outside Core Complex: 50 m & KV < = 0.45 Pass 4 (overwrite Pass 3)	4/3	(1.0:0.7:0.4) [110:80:0]	8	500/3500	500/ 3500	
Inferred	Primary Satellite Deposit: 150 m & KV < 0.45 Pass 5	4/3	(1.0:0.7:0.4) [110:80:0]	3	600	600	
Indicated	Primary Satellite Deposit: 50 m & KV < 0.34 Pass 6 (overwrite Pass 5)	4/3	(1.0:0.7:0.4) [110:80:0]	8	600	600	
Inferred	Secondary Satellite Deposit: 150 m & KV < 0.45 Pass 7	4/3	(1.0:0.7:0.4) [110:80:0]	3	700	700	
Indicated	Secondary Satellite Deposit: 50 m & KV < 0.34 Pass 8 (overwrite Pass 7)	4/3	(1.0:0.7:0.4) [110:80:0]	8	700	700	
Inferred	Secondary Satellite Deposit: 150 m & KV < 0.45 Pass 9	4/3	(1.0:0.7:0.4) [110:80:0]	3	800	800	
Indicated	Secondary Satellite Deposit: 50 m & KV < 0.34 Pass 10 (overwrite Pass 9)	4/3	(1.0:0.7:0.4) [110:80:0]	8	800	800	

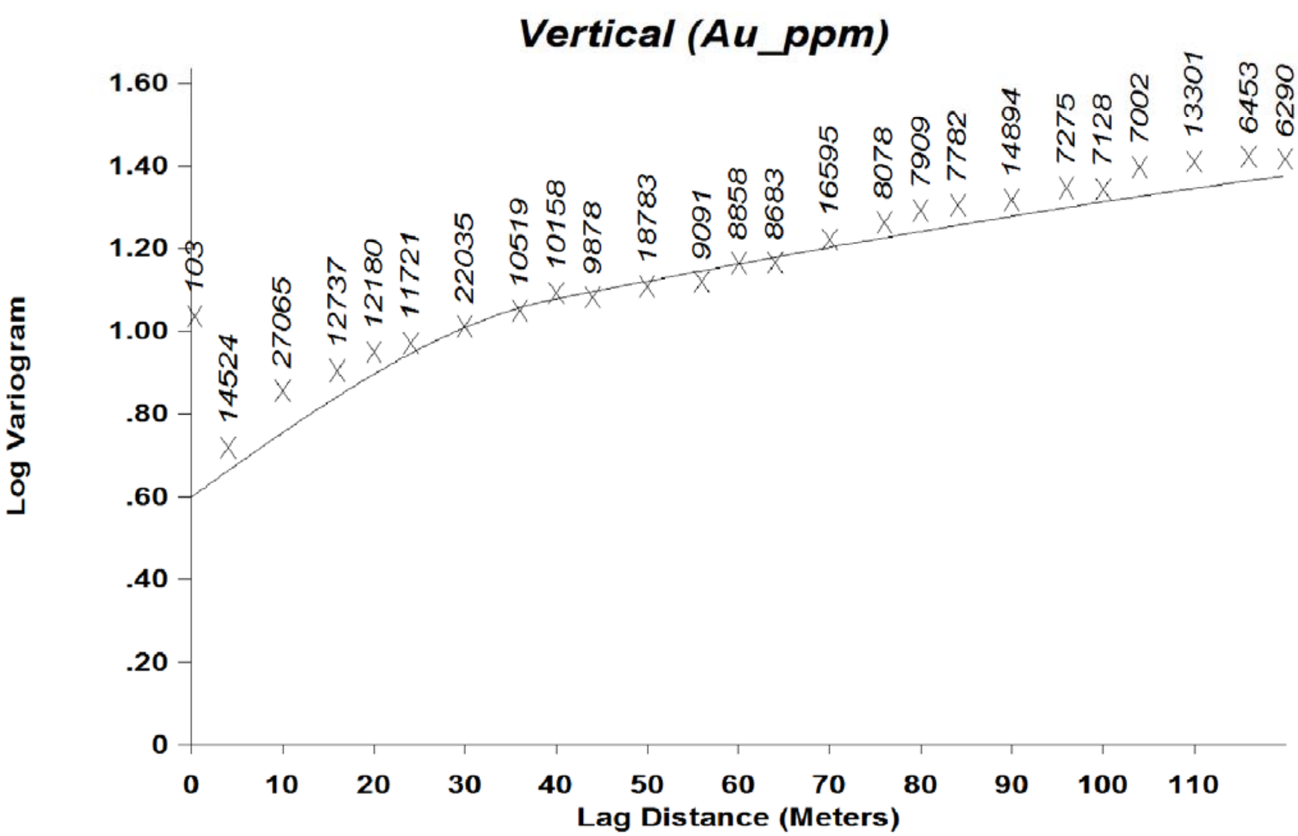
INDEX

Zone Codes	Zone Names
3500	Footwall
1000	Core Complex
800	Tertiary Satellite (between 600 and 700)
700	Secondary Satellite (in HW farthest from Core)
600	Primary Satellite (in HW Nearest to Core)
500	Hanging Wall Area

Notes

Ranges In meters (m)
KV = kriging variance,
Passes refer to multiple re-estimations of blocks with greater constraints (minimum points, search ranges, etc.) imposed.
Core and Satellites have more consistent gold grades, while the Footwall and Hanging Wall have patchy gold grades,
Search Ranges (a:b:c) Proportion of Maximum Range for: a. Primary Axis Length: b. Secondary Axis Length: c. Tertiary Axis Length
Orientation of Ellipse [1:2:3] 1. Azimuth of Primary Axis : 2. Dip of Primary Axis: 3. Rotation (Tilt) around Primary Axis





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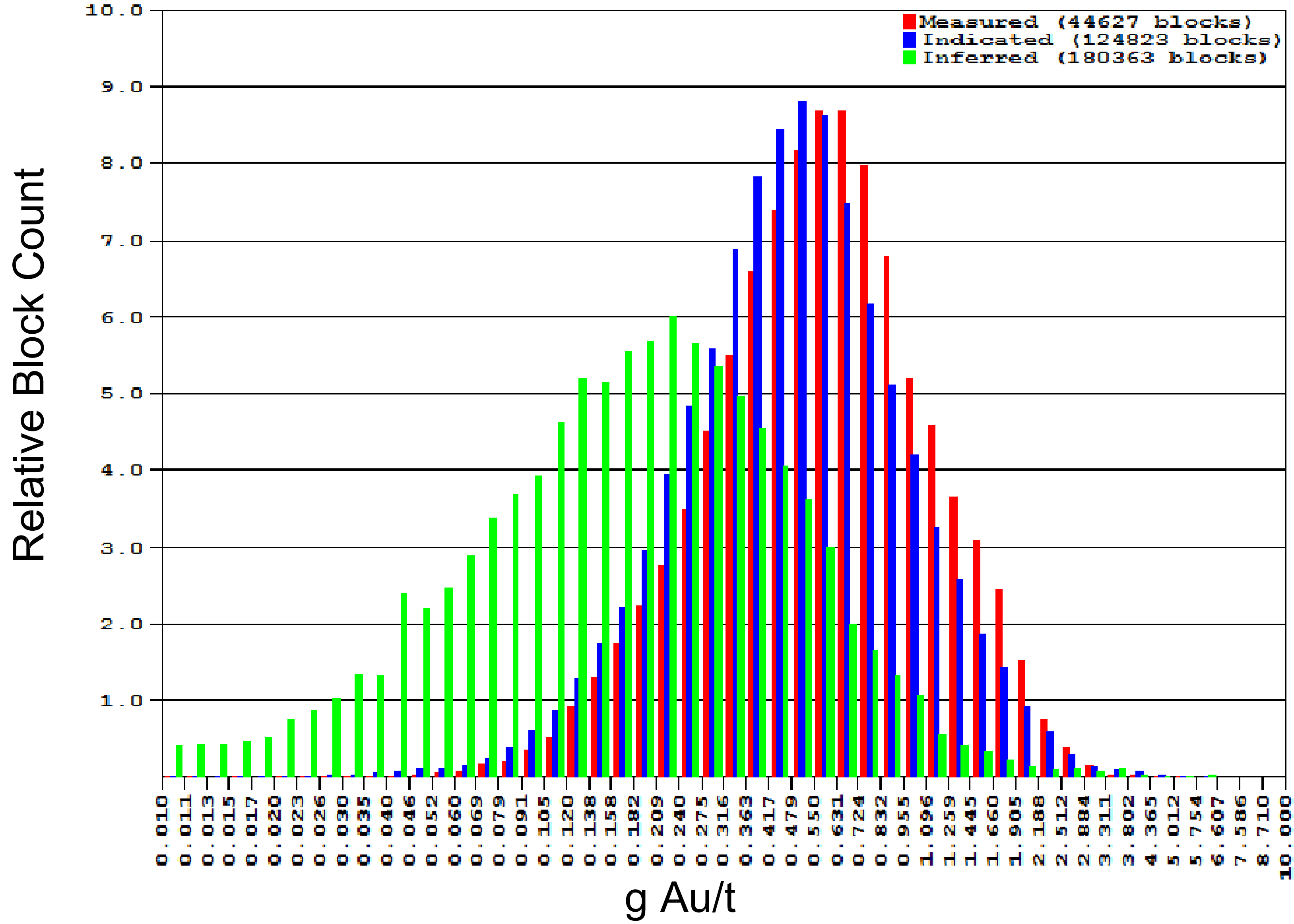
Project:	Mt Todd Gold Project
Project Location:	Northern Territory, Australia
File Name:	Figure14-2to5.dwg
Project Number:	114-311218
Date of Issue:	Sept 2012

Figure 14-
Example Log Variograms of Au within the Core Complex



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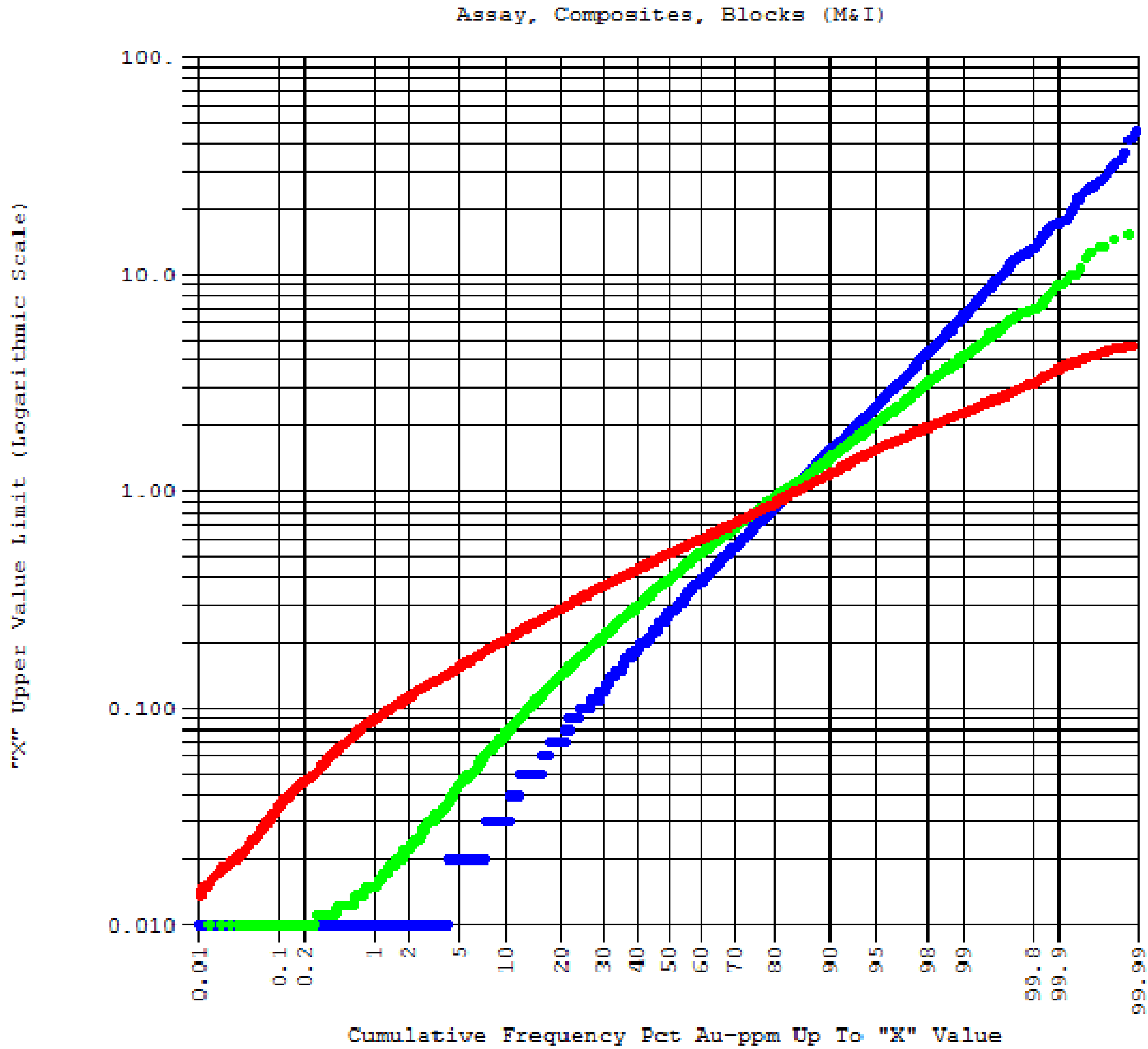
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Vista Gold Corp.***	Figure14-2to5.dwg	**Figure 14-** **Relative Block Count Histograms Measured, Indicated and Inferred - Batman Deposit**
	Project: Mt Todd Gold Project	Project Number: 114-311218	
	Project Location: Northern Territory, Australia	Date of Issue: September 2012	



	assay	composite	blocks
Number of Samples:	60194	15142	169460
Number Missing:	498	263	0
Number Below Limits:	2409	218	10
Number Above Limits:	0	0	0
Number in Range:	57287	14661	169450
Minimum Value:	0.010	0.010	0.010
Maximum Value:	65.200	15.455	5.797
Mean Value:	0.672	0.648	0.633
Median Value:	0.270	0.390	0.514
Variance:	2.301	0.752	0.213
Standard Deviation:	1.517	0.867	0.462

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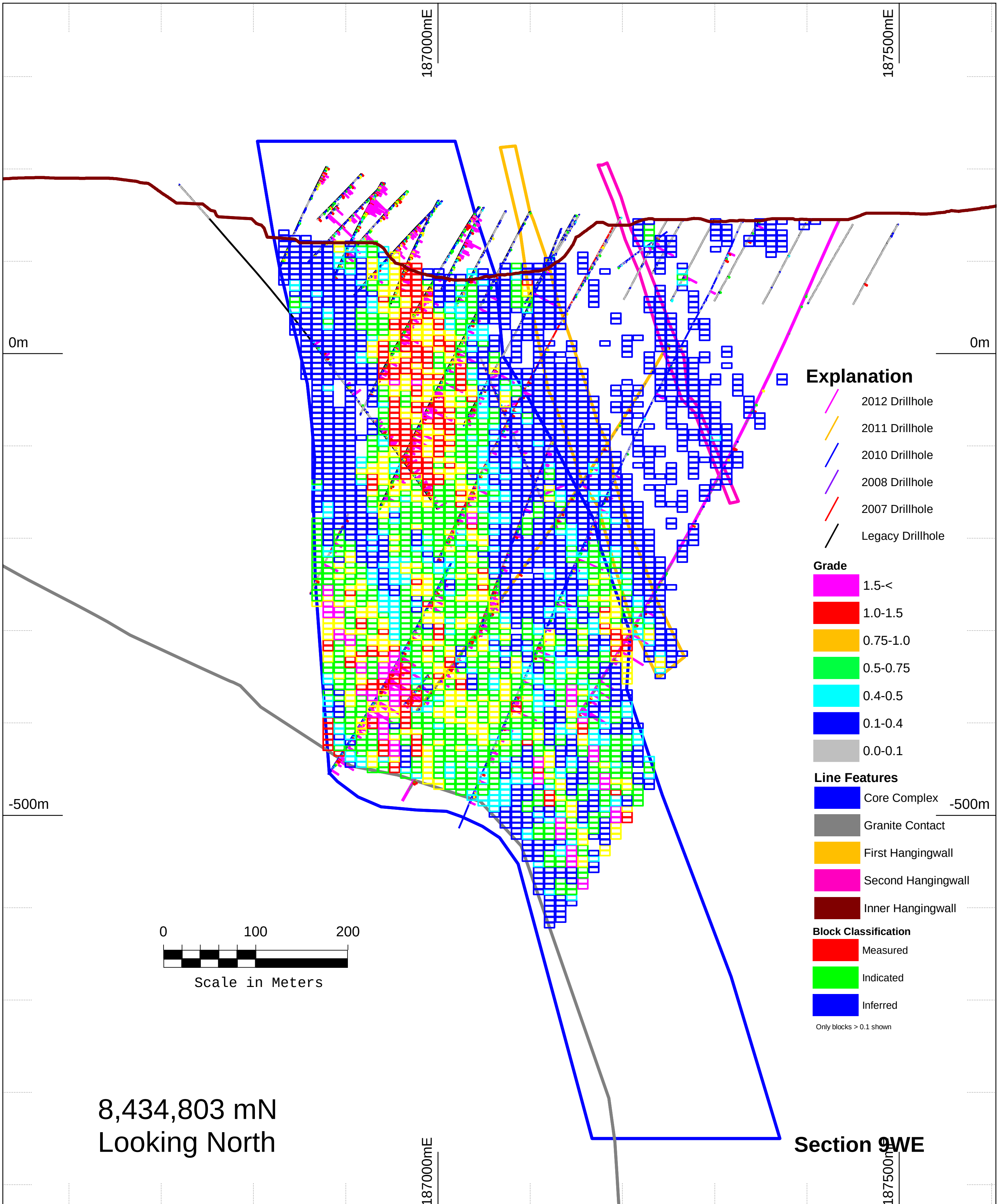
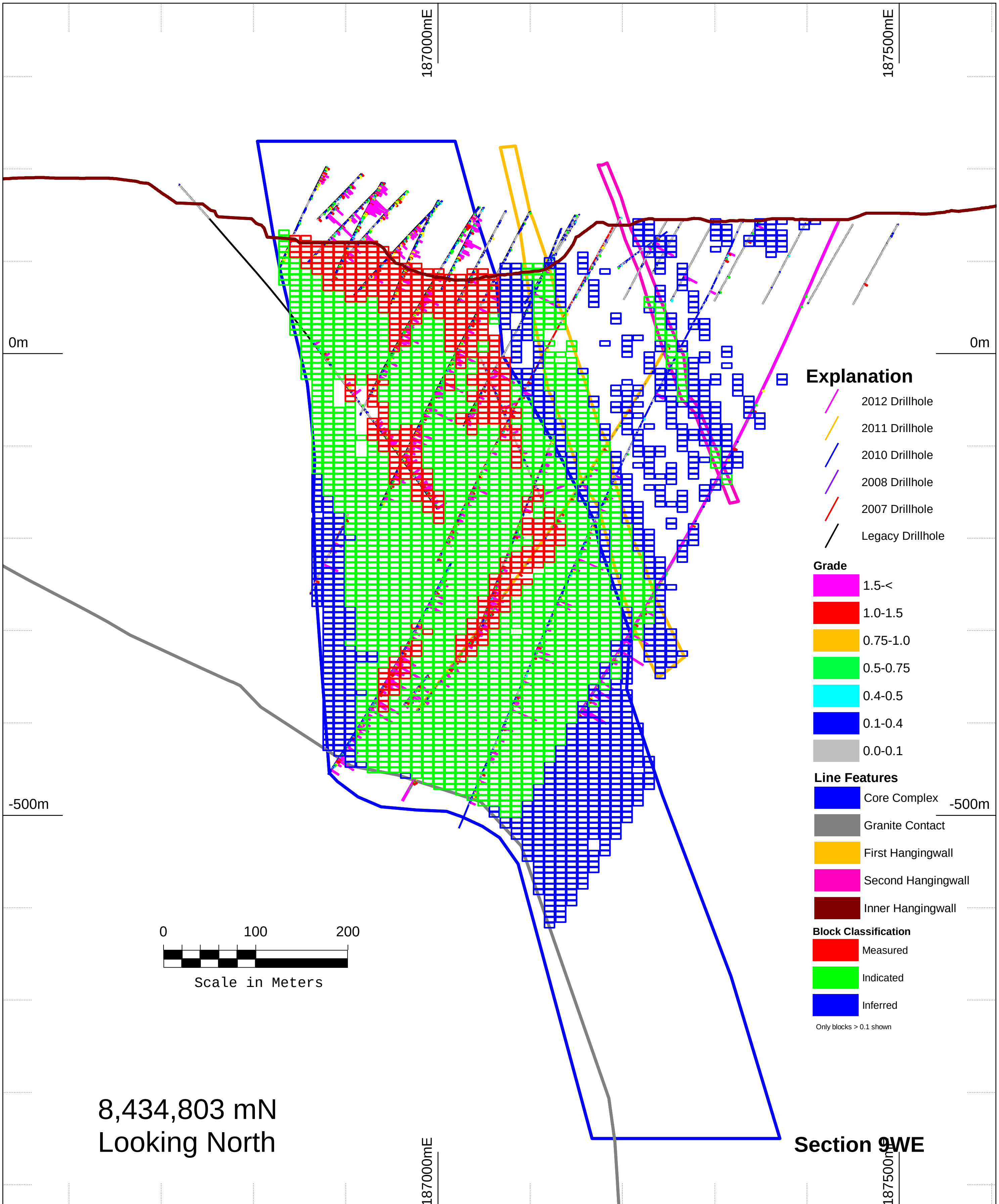
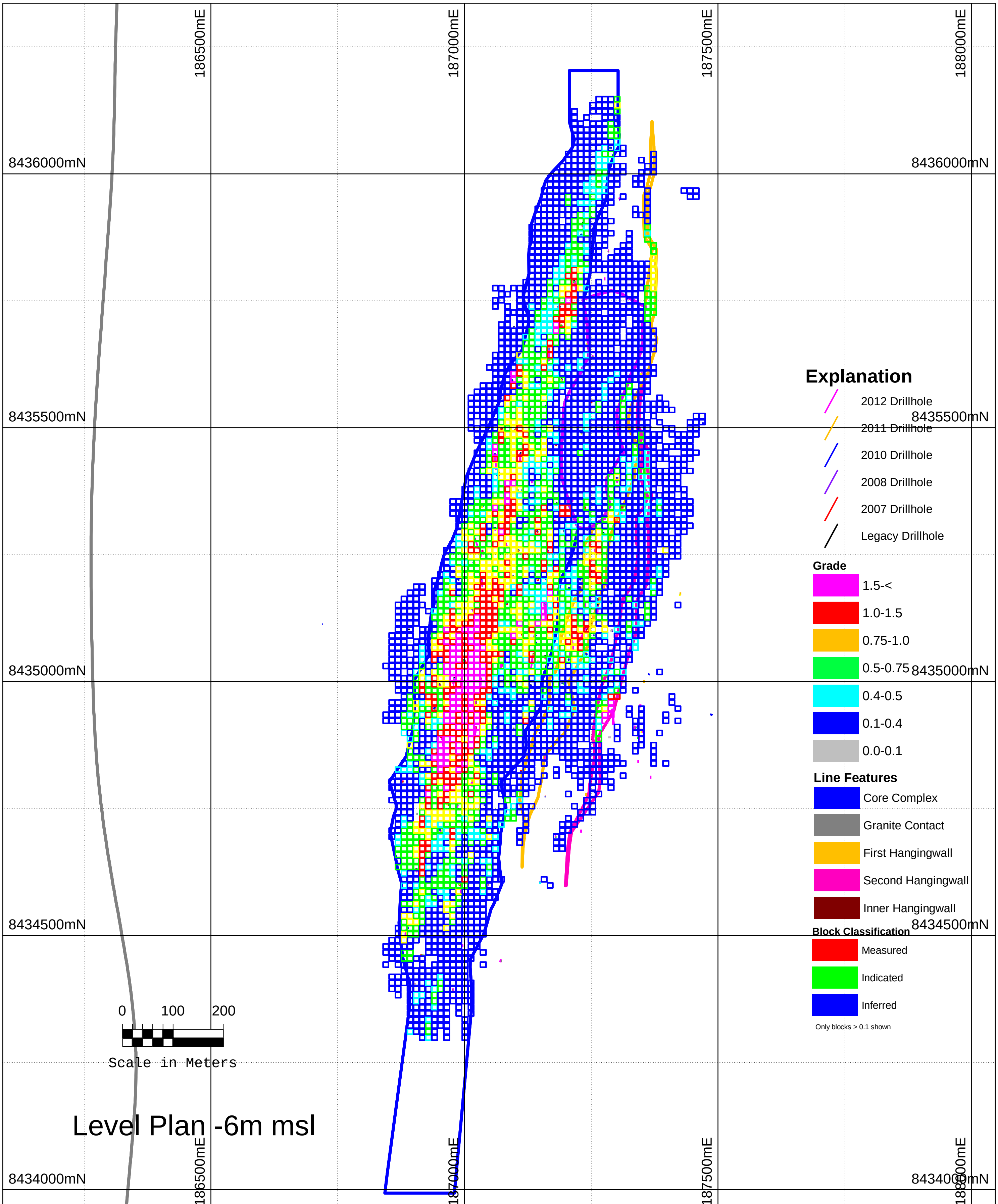
TETRA TECH, 350 Indiana Street, Suite 500, Golden, Colorado 80401, (303) 217-5700 (303) 217-5705 fax	*Vista Gold Corp.*	Figure14-2to5.dwg	**Figure 14- Cumulative Frequency Blocks, Composites, and Assays - Batman Deposit**
	Project: Mt Todd Gold Project	Project Number: 114-311218	
	Project Location: Northern Territory, Australia	Date of Issue: Sept 2012	



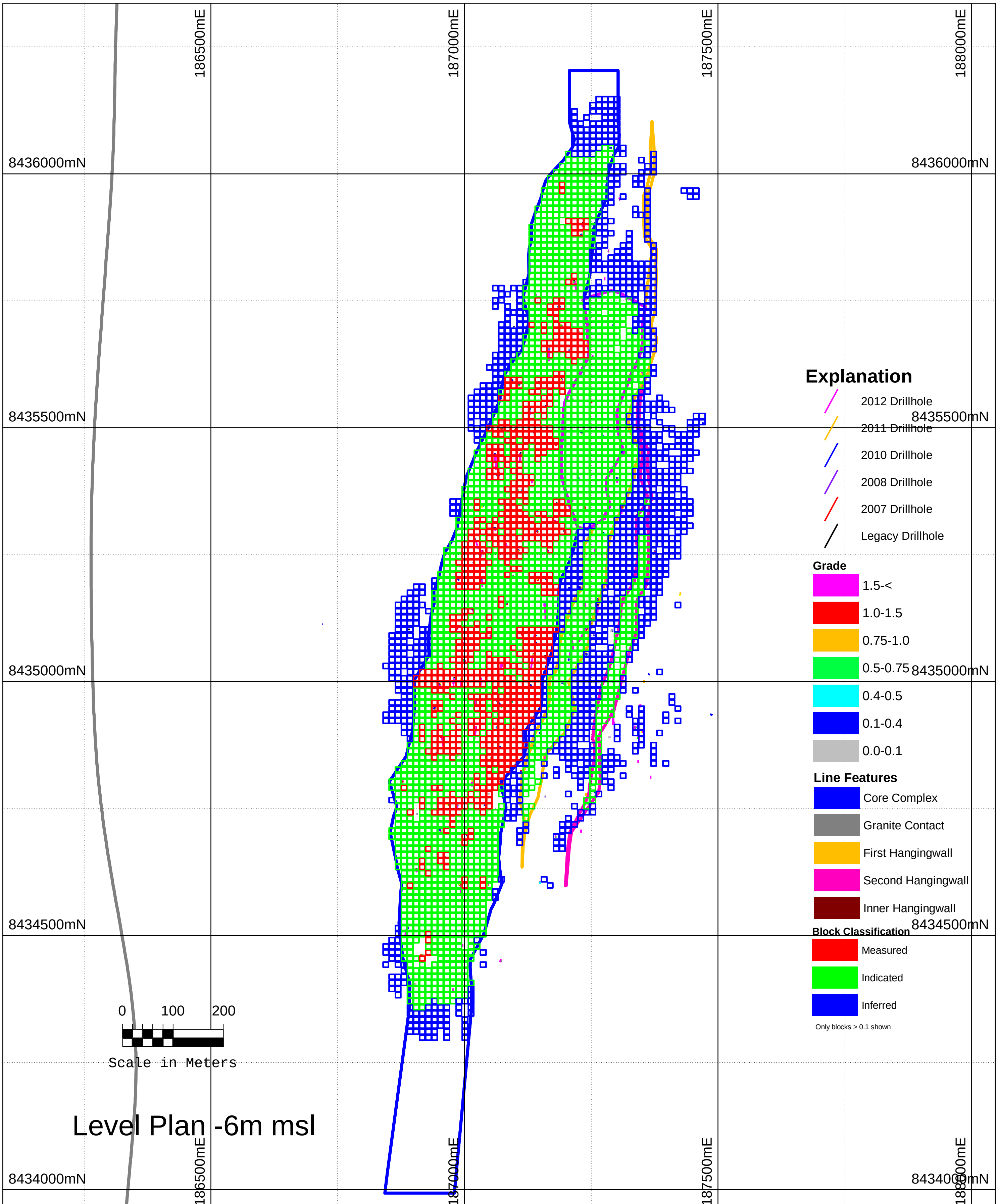
Issued By: TETRA TECH	Prepared for: **Vista Gold Corp.**	File Name: Figure 14-7.pdf	**Figure 14-7: Block Model Au - Batman Deposit**
	Project: Mt. Todd Gold Project	Project Number: 114-311285	
	Project Location: Northern Territory, Australia	Date of Issue: June 2013	



Issued By: TETRA TECH	**Prepared for:** *Vista Gold Corp.*	**File Name:** Figure 14-8.pdf	**Figure 14-8: Block Model Classification - Batman Deposit**
	Project: Mt. Todd Gold Project	**Project Number:** 114-311285	
	Project Location: Northern Territory, Australia	**Date of Issue:** June 2013	



Issued By:	Prepared for: **Vista Gold Corp.**	File Name: Figure 14-9.pdf	**Figure 14-9: Blocks kriged Au - Level Plan Batman Deposit**
TETRA TECH	Project: Mt. Todd Gold Project	Project Number: 114-311285	
	Project Location: Northern Territory, Australia	Date of Issue: June 2013	



Issued By: TETRA TECH	**Prepared for:** ***Vista Gold Corp.***	**File Name:** Figure 14-10.pdf	**Figure 14-10: Blocks Classified MIF Level Plan Batman Deposit**
	Project: Mt. Todd Gold Project	**Project Number:** 114-311285	
	Project Location: Northern Territory, Australia	**Date of Issue:** June 2013	

14.6 Batman Estimation Quality

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

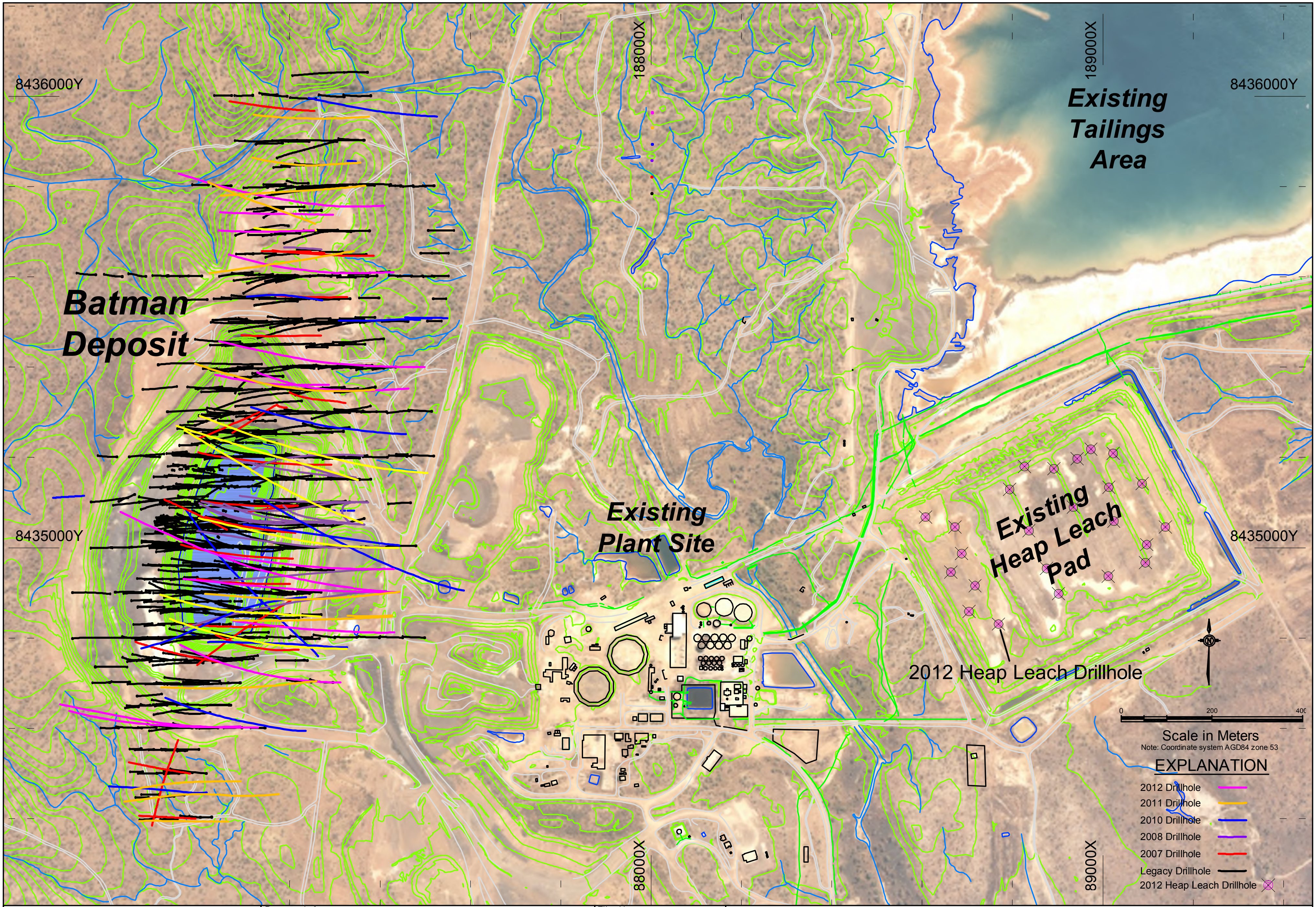
- Determine the optimum kriging search parameters;
- The overall quality of the estimation as required by classification (measured, indicated and inferred); and
- The use of visual inspection of the kriged blocks models in section and plan and the inspection of gold histograms of assays, composites and blocks.

14.7 Modeling of Existing Heap Leach Gold Resource

In addition to the in-situ gold resource for the Batman deposit, a historical 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. The concentration of gold was analyzed with 29 vertical drillholes separated by an approximately 100 meters. There were 321 assays from 1-m composites. Density of the pile was estimated from 11 drillholes using 1,162 dual density sidewall gamma probe technology. Note that the probe uses a gamma source and a scintillation detector to estimate density via the Compton Effect.

A nearest neighbor (polygon) method was employed to estimate grades since there is no apparent spatial correlation between samples. The existing heap leach pile is estimated to contain 230,000 ounces of gold within 13.4 Mt of indicated resource at an average grade of 0.54 g Au/t. No cutoff grade was applied to the heap leach pile resource as all material will be processed as part of the site rehabilitation process. While copper was also estimated, copper results are not presented here.

Figure 14-11 shows the location of the pile and placement of sample drillholes sampling the pile.



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Vista Gold Corp.	Fig14-11.dwg	**Figure 14-11: Location of Heap Leach Pad And Batman Deposit**
	Project: Mt Todd Gold Project	Project Number: 114-311285	
	Project Location: Northern Territory, Australia	Date of Issue: June 2013	

14.8 Modeling of the Quigleys Deposit

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

Zone 1 gold grades range from 0.001 to 21.75 g Au/t., averaging 0.703 g Au/t. Zone 9999 gold grades range from 0.001 to 11.318, with an average of 0.148 g Au/t. The gold grades have a lognormal distribution for both Zone 1 and 9999, with observable outlier values at the highest grades. Discussion of the capping composite gold grade values is presented in the Quigleys block modeling section.

Bulk density data were supplied by Pegasus for two ore types and waste within the oxide, transition and primary zones, based on a total of 39 samples collected from recent RC drilling. The two densities supplied were for stockwork and shear, with the density of the shear material substantially higher, particularly in the transition and primary zones. These samples were over one-m to two-m intervals and thus selected the narrow high grade portion of the shear zone as originally interpreted by Pegasus. The final mineralization envelope was much broader than this, and the bulk density was therefore estimated by assuming the final envelope contained 15% shear and 85% stockwork and weighting the density values accordingly. **Table 14-9** shows the specific gravity data assigned to the Quigleys area according to oxidation state.

Table 14-9: Quigleys Deposit Specific Gravity Data

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

More confidence in the geological interpretation would be needed to ascertain the geometry of the high-grade portion of the shear zone. Alternatively, it may be appropriate, with a more detailed density study, to weight the high-grade blocks with a higher density.

14.8.1 Quigleys Exploration Database

Table 14-10 summarizes the Quigleys exploration database.

Table 14-10: Summary of Quigleys Exploration Database

Drillhole Statistics						
	Northing (m) AMG84 z53	Easting (m) AMG84 z53	Elevation (m)	Azimuth	Dip	Depth (m)
Minimum	8,430,1876	188,445.7	129.7	0	45	0
Maximum	8,432,290	189,746.5	209.0	354.0	90	330.5
Average	8,431,129.5	189,230.8	155.9	83.4	62.5	91.3
Range	2,104.0	1,300.8	79.3	354.0	45.0	330.5
Cumulative Drillhole Statistics						
Total Count	631					
Total Length (m)	57821.28					
Assay Length (m)	1 (approx.)					
Drillhole Grade Statistics	**Number**	**Average**	**Std. Dev.**	**Min.**	**Max**	**Missing**
Au (g/t)	52,152	0.2445	0.8764	0	36.00	82
Cu (%)	40,437	0.0105	0.0305	0	2.98	11,897

14.8.2 Quigleys Block Model Parameters

Quigleys' block model parameters are shown in **Table 14-11**. The model consisted of 37,082 blocks within the modeled mineralized zones (blocks within the modeled grade zones are coded as 1). Each of the blocks is 250 m^3 (5x25x2m) with a defined density of 2.77 g/cm (692.5 tonnes).

Table 14-11: Block Model Physical Parameters – Quigleys Deposit

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

A number of absolute, log and indicator directional variograms were calculated and modeled. From these, **Figure 14-12**, showing the omni median indicator (0.3 g/t cut) gold variogram is a good example. All variograms show a large relative nugget (60% of the ultimate sill at 100 m). The experimental variogram was modeled with a nugget of 0.6 and two nested spherical models. The first spherical model has a range of 20m, and a sill of 0.3. The second model has a range of 100m and a sill of 0.1. A flat anisotropy ellipsoid with a 2:2:1 ratio of axis was used to indicate both the continuity of gold grade (variogram range) and the preferential search pattern to be used. The chosen variogram and search ellipsoid is defined with a dip of 25 at an azimuth 110°.

Composite gold grades as with the original assay a lognormal population, again with a small percentage of high grade outliers. Review of the log probability plot of the composited gold grades shows that there is a distinct break in the distribution at 12 g Au/t. All gold composites were capped at this value.

The block model was estimated using ordinary kriging. The estimation searched for four composites in a sector, allowing a maximum of three composites per drillhole. Inside the ore zone, (blocks coded a 1); composites were selected only if they also were coded as 1. In all, three separate kriging passes were done at consecutively increasing search distances. The first pass restricted points to be within 20 m as defined by the search ellipsoid axis. This estimate also required a minimum of 7 composites. This pass coded all of the estimated blocks with a provisional measured classification code (MIF code 1). The second pass used a 40 m maximum search ellipsoid with a minimum of 6 composites and assigned any un-estimated block a provisional indicated class (MIF code 2). The third pass used a 200 m search with blocks requiring a minimum of 3 composites to be estimated. Any newly estimated block was given an inferred classification (MIF code 3). 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 0.335. Hence the provisional MIF codes were then adjusted to a more restricted class when a blocks kriging error exceeded this value.

For the outside zone, a single kriging for MIF class 3 was done with a maximum search ellipse range of 25 m.

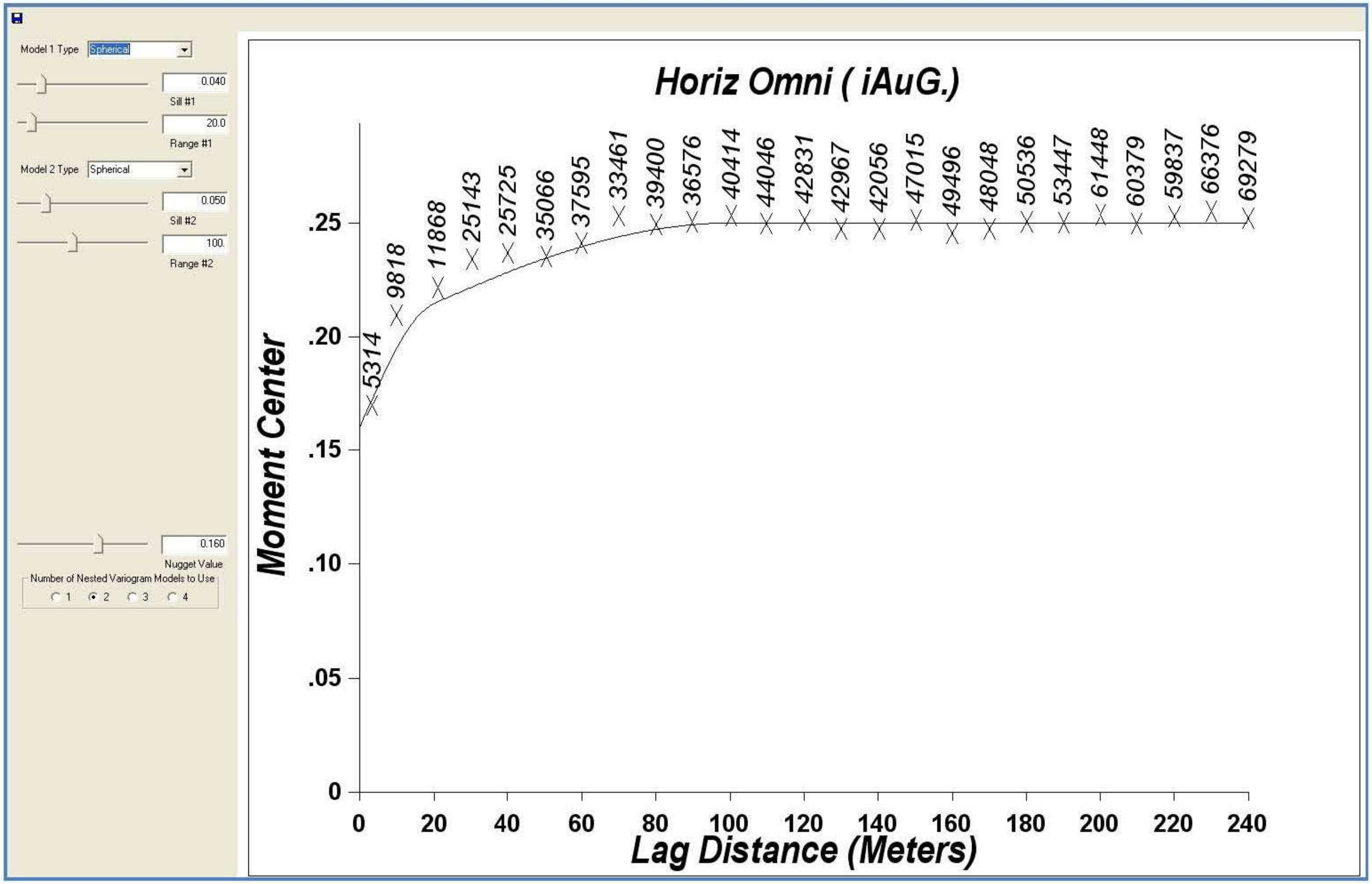


Figure 14-12: Quiqleys Median Indicator Variogram (Omni Direction)

Table 14-12: Quigleys Resource Classification Criteria

Category	Search Range & Kriging Variance	No. of Sectors/ Max Points per DH	Min Points
Measured	Zone 1: 20 m search & KV < 0.335	4/3	7
Indicated	Zone 1: 20-40 m search & KV < 0.335	4/3	6
Inferred	Zone 1 40-200 m search & KV < 0.335 Zone 9999 < 25 m	4/3	3

The classification was accomplished by a combination of search distance, kriging variance, number of points used in the estimate, and number of sectors used. **Table 14-13** through **Table 14-17** detail estimated resources by cutoff and classification. All of the resources quoted are contained on Vista's mineral leases.

Several methods used to validate the block model were used to determine the adequacy of the Quigleys resource. Cumulative frequency plots of blocks, composites, and assays were overlaid. The three overlaid plots showed the expected decrease in the variability of the gold distributions going from assays to assay composites and then to kriged blocks. Additional verification of the block model was completed by the use of jackknife studies (model validation) where known assays were estimated using surrounding samples, visual inspection of the kriged blocks models in section and plan and the inspection of gold histograms of assays, composites and blocks. More detail on the different model validation techniques employed can be found in Appendix A of the January 28, 2011 PFS.

14.9 Mineral Resources of the Mt. Todd Project

Table 14-13 details the estimated in-place resources by classification and by cutoff grade for the Batman deposit and was updated for the October 2012 Resource Update Report. **Table 14-16** details the in-place resource estimate by classification and by cutoff grade for the Quigleys deposit.

Table 14-13: Batman Deposit Measured and Indicated Gold Resource Estimate

	Cutoff Grade g Au/t	Tonnes (x1000)	Average Grade g Au/t	Total Au Ounces (x1000)
MEASURED	2.00	2,477	2.40	191.3
	1.75	4,622	2.15	319.9
	1.50	8,192	1.92	505.5
	1.25	13,212	1.71	726.3
	1.00	21,528	1.48	1,024.8
	0.90	26,497	1.38	1,176.1
	0.80	33,183	1.27	1,358
	0.70	41,615	1.17	1,561
	0.60	52,531	1.06	1,788.6
	0.50	64,655	0.97	2002.8
	0.40	**77,793**	**0.88**	**2,193.0**
	0.3	90,799	0.80	2,340.0
INDICATED	2.00	5,445	2.55	446.8
	1.75	9,159	2.27	669.4
	1.50	15,245	2.01	986.1
	1.25	25,362	1.75	1,429.7
	1.00	43,392	1.48	2,073.2
	0.90	54,488	1.38	2,411.3
	0.80	69,291	1.26	2,814.9
	0.70	88,763	1.15	3,282.5
	0.60	116,203	1.03	3,853.8
	0.50	154,333	0.91	4,524.8
	0.40	**201,792**	**0.80**	**5,209.0**
	0.30	255,602	0.71	5,813.1
MEASURED + INDICATED	2.00	7,923	2.5	638.2
	1.75	13,780	2.23	989.2
	1.50	23,437	1.98	1,491.6
	1.25	38,574	1.74	2,156.1
	1.00	64,921	1.48	3,098.0
	0.90	80,985	1.38	3,587.3
	0.80	102,474	1.27	4,172.8
	0.70	130,378	1.16	4,843.7
	0.60	168,733	1.04	5,642.5
	0.50	218,988	0.93	6,528.1
	0.40	**279,585**	**0.82**	**7,401.4**
	0.30	346,401	0.73	8,153.2

Tables above (Measured, Indicated, Measured + Indicated, Inferred) show the resources present and are not contained within a pit (i.e., all possible resources).
Tonnage, grades and totals may not total due to rounding

Table 14-14: Batman Deposit Inferred Gold Resource Estimate

	Cutoff Grade g Au/t	Tonnes (x1000)	Average Grade g Au/t	Total Au Ounces (x1000)
INFERRED	2.00	2,233	3.1155	223.7
	1.75	2,962	2.81	267.9
	1.50	3,936	2.52	318.5
	1.25	5,995	2.12	408.7
	1.00	9,845	1.72	545.8
	0.90	13,513	1.52	658.4
	0.80	18,091	1.35	783.0
	0.70	24,138	1.2	928.0
	0.60	33,316	1.04	1,117.7
	0.50	48,790	0.89	1,389.3
	0.40	**72,458**	**0.74**	**1,792.2**
	0.30	108,799	0.61	2,133.6

Tables above (Measured, Indicated, Measured + Indicated, Inferred) show the resources present and are not contained within a pit (i.e., all possible resources).
Tonnage, grades and totals may not total due to rounding

Table 14-15: Existing Heap Leach Indicated Gold Resource Estimate

	Cutoff Grade g Au/t	Tonnes (x1000)	Average Grade g Au/t	Total Au Ounces (x1000)
INDICATED	0	13,400	0.541	230

Tonnage, grades and totals may not total due to rounding

Table 14-16: Quigleys Deposit Measured and Indicated Gold Resource Estimate

	Cutoff Grade g Au/t	Tonnes (x1000)	Average Grade g Au/t	Total Au Ounces (x1000)
MEASURED	2.00	30	2.27	2
	1.75	50	2.11	3
	1.50	87	1.90	5
	1.25	136	1.71	7
	1.00	222	1.48	11
	0.90	263	1.39	12
	0.80	305	1.32	13
	0.70	355	1.24	14
	0.60	428	1.14	16
	0.50	511	1.04	17
	0.40	**571**	**0.98**	**18**
INDICATED	2.00	158	2.38	12
	1.75	273	2.17	19
	1.50	450	1.95	28
	1.25	897	1.66	48
	1.00	1,634	1.41	74
	0.90	2,057	1.32	87
	0.80	2,618	1.22	102
	0.70	3,374	1.11	121
	0.60	4,363	1.01	141
	0.50	5,565	0.91	162
	0.40	**6868**	**0.820**	**181**
MEASURED + INDICATED	2.00	188	2.36	14
	1.75	323	2.16	22
	1.50	537	1.94	34
	1.25	1,033	1.66	55
	1.00	1,856	1.42	85
	0.90	2,320	1.33	99
	0.80	2,923	1.23	115
	0.70	3,729	1.12	135
	0.60	4,791	1.018	157
	0.50	6,076	0.919	179
	0.40	**7,439**	**0.833**	**199**

The sum of measured and indicated resources as reported under NI 43-101 guidelines is equivalent to mineralized material under SEC Industry Guide 7.
Tonnage, grades and totals may not total due to rounding

Table 14-17: Quigleys Deposit Inferred Gold Resource Estimate

	Cutoff Grade g Au/t	Tonnes (x1000)	Average Grade g Au/t	Total Au Ounces (x1000)
INFERRED	2.00	335	2.35	25
	1.75	559	2.16	39
	1.50	975	1.93	60
	1.25	1,854	1.66	99
	1.00	3,193	1.43	147
	0.90	3,950	1.34	170
	0.80	4,795	1.25	193
	0.70	5,871	1.16	219
	0.60	7,473	1.05	252
	0.50	9,416	0.95	287
	0.40	**11,767**	**0.85**	**320**

Tables above (Measured, Indicated, Measured + Indicated, Inferred) show the resources present and are not contained within a pit (i.e., all possible resources).
Tonnage, grades and totals may not total due to rounding

15.0 MINERAL RESERVE

Mine Development Associates (MDA) has used the measured and indicated resource estimates provided by Tetra Tech to estimate reserves as of May 29, 2013.

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

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

15.1 PIT OPTIMIZATION

Pit optimization was done using Geovia's Whittle software to define pit limits with input for economic and slope parameters. The optimization used parameters provided by Vista and other consultants based on recent studies.

Optimization used only Measured and Indicated Resources for processing. All Inferred Resource was considered as waste.

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

15.1.1 *ECONOMIC PARAMETERS*

Economic parameters are provided in **Table 15-1**. Initially, several iterations of pit optimizations were reviewed for the final determination of pit limits.

Table 15-1: Economic Parameters

Parameter	Base Case
Gold Price	$1360 per Au oz
Gold Recovery	82% Sulfide 78% Transition 78% Oxide
Payable Gold	99.9%
Overall Mining Cost	$1.90 per tonne
Processing Cost	$9.779 per tonne processed
Tailings	$0.985 per tonne processed
Water Treatment	$0.09 per tonne processed
Royalty	1% NPR (Jawoyn)

The mining costs used were varied by bench. An incremental cost of US$0.01 was added for each 6-m bench below the 145-m elevation. This represents the incremental cost of haulage for both waste and ore for each bench that is to be mined. A reference mining cost of US$1.64 was used determined using first principles from previous studies.

Processing, tailings construction, tailings reclamation, and water treatment costs were provided by Vista and other consultants and are based on either new data or previous studies and have been accepted by MDA for inclusion in this report. Note that site G&A costs are included with the processing costs for Whittle optimization. At Vista's request, MDA used a minimum cutoff grade of 0.40 g Au/t for reserve definition. This was done to maintain higher grades with respect to material allowed to be processed.

The gold price of US$1,360 per ounce has been assumed for use in reserve estimation and mine planning. Various gold prices from $300 to $2,000 per ounce were used to determine different optimized pit shells. This gold price is conservative based on the 3-year rolling average, which at the time of this report is near US$1,545 per ounce Au, though the recent prices (May, 2013) have averaged US$1,413. Note that while the gold price does significantly impact the cash-flow, it does not tend to have an impact on the resulting reserve definition because the pit optimizations used elevated cutoff grades.

15.1.2 Slope Parameters

Slope parameters were based on studies provided by Golder Associates Ltd. (Golder) and Ken Rippere, an independent rock mechanics consultant, as detailed in a Golder's memorandum dated September 13, 2011 ("Mt. Todd Gold Project: Batman Pit Slope Design Guidance in Support of the Definitive Feasibility Study"). These recommended slopes were reduced to account for ramps required for equipment access where anticipated in the final pit design. For pit optimization, slopes were divided into five sectors in fresh rock and three additional sectors for weathered rock. The sectors are based on the wall orientation. Each sector was modeled into a zone resulting in eight zones. The recommended and adjusted inner-ramp angles are shown in **Table 15-2**.

Table 15-2: Slope Angles for Pit Optimization

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

15.1.3 Pit-Optimization Results

Whittle pit optimizations were run using the economic and slope parameters described in previous sections. Pit optimizations were completed using prices of US$300 to US$2,000 per ounce Au in order to analyze the deposit's sensitivity to gold prices for both scenarios. Results for US$100 per ounce increments from US$300 to US$2,000 per ounce of gold are shown in **Table 15-3**. A graph of the Whittle results are shown in **Figure 15-1**.

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

Gold Price (US$)	Material Processed K Tonnes	g Au/t	K Ozs Au	Waste (tonnes)	Total (tonnes)	Strip (ratio)
300	380	1.93	24	158	538	0.42
400	6,047	1.69	328	5,281	11,328	0.87
500	10,651	1.52	519	9,159	19,810	0.86
600	19,432	1.32	825	20,902	40,334	1.08
700	59,816	1.13	2,166	125,006	184,822	2.09
800	90,296	1.04	3,014	184,507	274,803	2.04
900	116,543	0.97	3,631	229,471	346,014	1.97
1,000	146,502	0.91	4,284	285,096	431,597	1.95
1,100	181,667	0.87	5,074	391,435	573,102	2.15
1,200	202,235	0.86	5,577	490,807	693,042	2.43
1,300	214,151	0.85	5,864	557,221	771,371	2.60
1,400	223,222	0.85	6,083	614,128	837,349	2.75
1,500	231,873	0.84	6,289	676,923	908,797	2.92
1,600	236,722	0.84	6,402	715,771	952,493	3.02
1,700	238,606	0.84	6,456	739,429	977,855	3.10
1,800	240,511	0.84	6,502	758,216	998,727	3.15
1,900	242,727	0.84	6,556	784,177	1,026,904	3.23
2,000	243,751	0.84	6,585	800,504	1,044,255	3.28

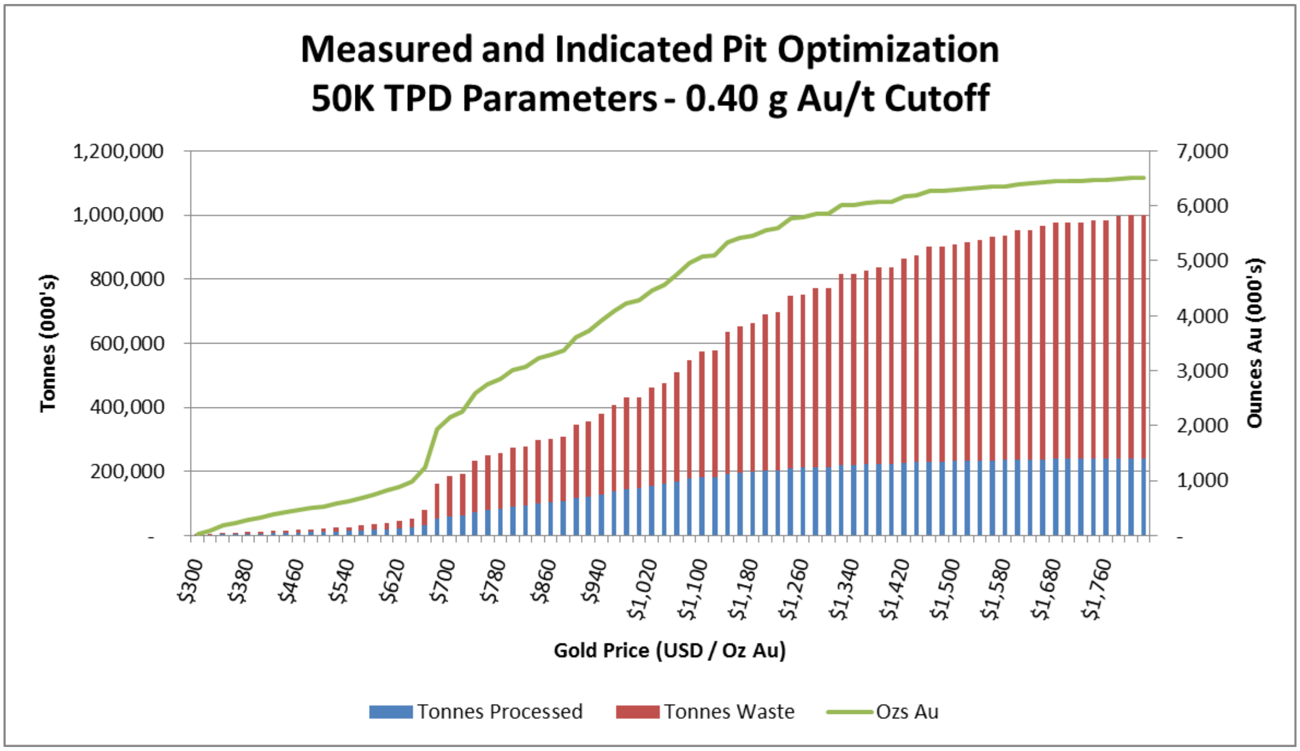


Figure 15-1: Graph of Whittle Results – Base Case Using 0.40 g Au/t Cutoff

15.1.4 Ultimate Pit Limit Selection

The ultimate pit limit was determined based on various iterations analyzed by MDA, Vista, and Tetra Tech. MDA used the parameters shown previously, but varied cutoff grades and discount rates. Using these runs, different pit shells were used as internal pit phases and production schedules were created to evaluate ultimate pit limits. The Base Case pit shell optimized at a US$1,360 per ounce Au price was chosen as the pit that best fit the ultimate corporate goals and was used to guide pit designs as described below.

15.2 Pit Designs

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

15.2.1 Bench Height

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

In future studies, Vista may consider remodeling the deposit on 7 or 8 m benches to reduce the mining costs. The deposit has relatively steeply dipping continuity with respect to gold grades. This geometry of the gold grades will diminish the impact of dilution with the increase in bench height.

15.2.2 Pit Design Slopes

Slope parameters were based on geotechnical studies provided by Golder Associates and Ken Rippere (Golder, September 13, 2011). The recommended slopes relate to five different sectors in fresh rock and three sectors in weathered rock.

The northeast and east sectors are the flattest. Due to the flatness of the slopes in those areas, the design parameters specify no safety berms. As the primary ramp is on the eastern side of each of the pit designs, the ramp is used as a safety berm in those areas. As the ramp crosses on these slopes, the ramp has been widened to allow berms to be placed along the high-wall to catch any sloughing material. The material that collects in this area would be removed on a regular basis.

The other sectors have conventional parameters applied in the form of height between catch benches (BH), safety berm widths (berms), bench face angles (BFA) and inner-ramp angles (IRA). The slope parameters along with the zone numbers are shown in **Table 15-4**.

Table 15-4: Pit Design Slope Parameters

	Northeast	East	South/ Southwest	Northwest	Weathered Rock Northeast/ East	Weathered Rock South/ Southwest	Weathered Rock Northwest
BH	150	150	24	24	30	30	30
IRA	36.1	40.5	55.1	51.4	28.7	45.7	45.7
BFA	44	50	70	65	35	60	60
Berm	50	50	8	8	12	12	12
Zone	1	2	3 & 4	5	6	7	8

Note that the northeast and east slopes in **Table 15-4** show a bench height of 150 m and berm of 50 m. The bench height reflects the approximate vertical height between the ramp crossing along the eastern wall and the 50 m berm is the width of the ramp with an added 14 m to contain material that may slough from the ultimate pit wall.

For design purposes, weathered material is considered to be the top 30 m of the surface.

15.2.3 Haulage Roads

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

Ramp width was determined as a function of the largest truck width to be used in mine planning. Mine plans use 226-t capacity trucks with operating widths of 8.30 m. For haul roads inside of the pit, a single safety berm on the inside of the roadway will be required to be at least half the height of the largest vehicle tire that uses the road. MDA has designed safety berms with a 1.5 horizontal to 1 vertical slope using run-of-mine material, a 0.33 m flat top, and a berm height of 3.10 m which provides half of the truck tire height plus 10% for the haul trucks. The 10% addition is used to ensure that the berm height exceeds half of the truck tire height in all cases. The resulting base width of safety berms is 9.54 m.

Haul-road designs inside of pits where only one safety berm is required are designed to be 36 m wide for two-way traffic. Subtracting berm widths, this provides 3.2 times the width of haul trucks for running width.

In lower portions of the pits where haulage requirements allow use of one-way traffic, haul roads are designed to have a width of 22 m. This provides 1.5 times the width of haul trucks for running width.

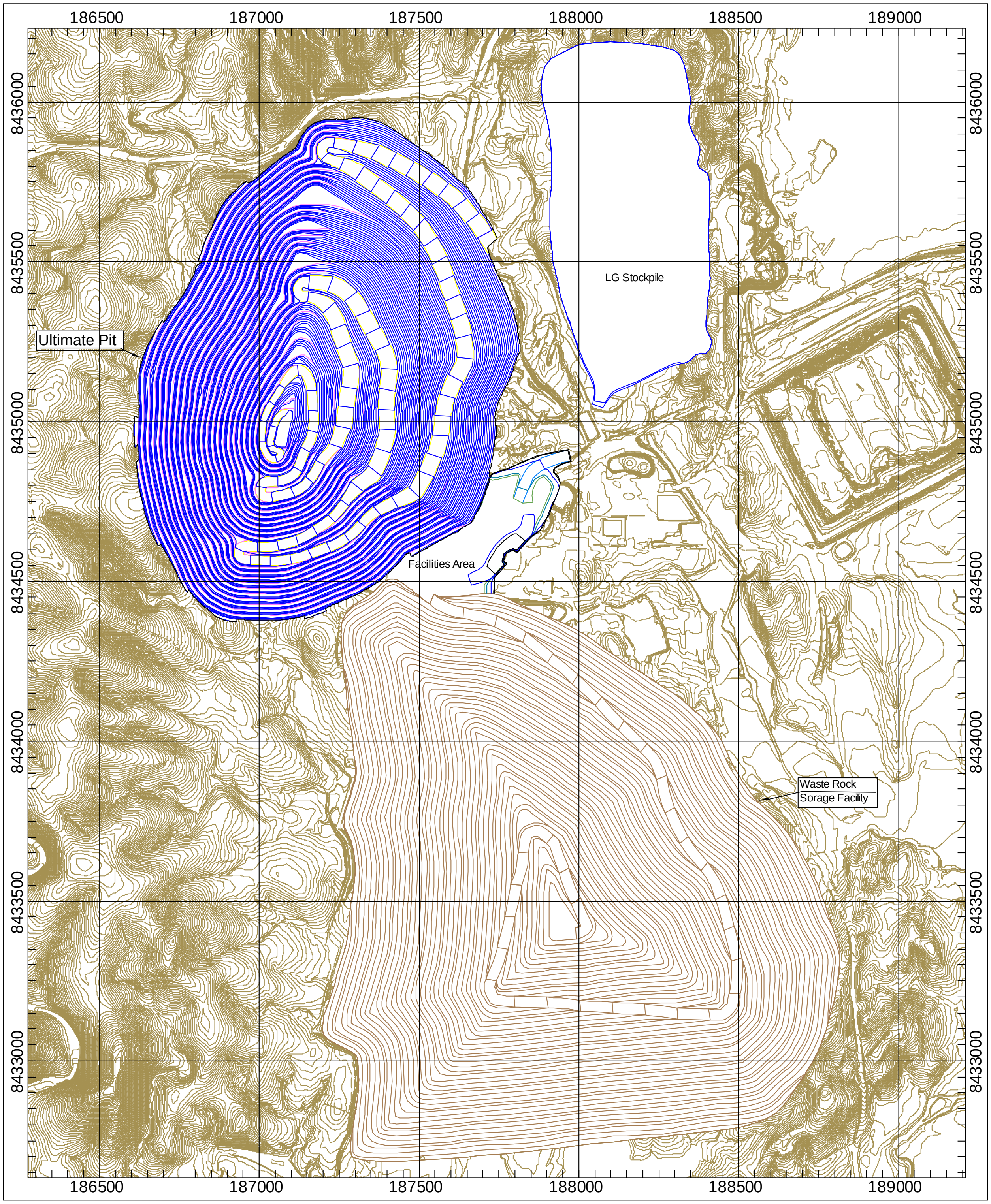
As mentioned in the previous section, where the in-pit ramps are in the northeast and east slope sectors, an additional width of 14 m has been added to the ramp so that a catch berm can be made along the high wall. This would catch slough that may slide down from the high wall and protect personnel traveling the haul roads. The area behind the catch berms will need to be cleaned out should they become full so that additional material isn't allowed to jump the catch berm and endanger traffic along the haul roads. This cleanup would be done using the production loader and trucks.

Haul roads outside of pit designs have been designed to be 48 m wide to account for an additional safety berm.

15.2.4 Ultimate Pit

As discussed in previous sections, the ultimate pit designs are based on Whittle pit shells. The final ultimate pit design incorporates switchbacks to maintain the ramp system on the east side of the pit. This allows for better traffic flow between pit phases and allows the west side of the pit to best follow the dip of the deposit. In all, there are four switchbacks in the ultimate pit designs and the lower portion of the pits spirals to achieve the ultimate pit design.

The ultimate pit designs along with the ultimate dump and stockpile designs and planned infrastructure are shown in **Figure 15-2**.



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MINE DEVELOPMENT ASSOCIATES

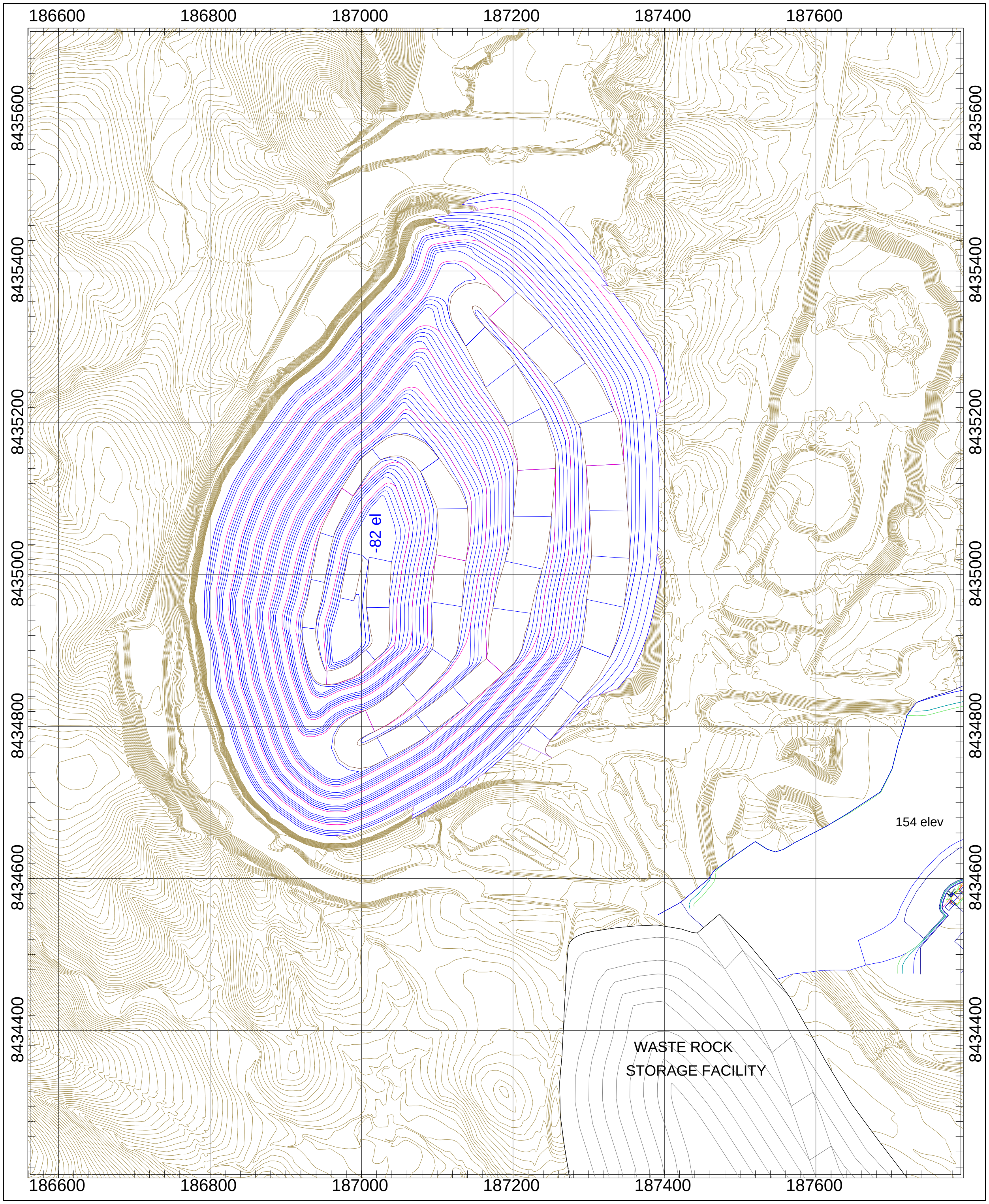
VISTA GOLD CORP.

Vista Gold
Mt Todd Ultimate Pit Map
Base Case

15.2.5 PIT PHASING

The Base Case Phase I pit design continues previous mining on the eastern side of the pit (but takes a larger pushback in comparison with the Alternate Case, reference Section 24.6). The Phase II pit design mines around Phase I, with the exception of a short common wall on the west side of the pit. Phase III expands the pit to the south and west. Phase IV establishes the ultimate pit expanding the previous phases to the north and west.

Figure 15-3, **Figure 15-4**, and **Figure 15-5** show the Base Case Phase I, II, and III. Resulting reserves for each of the phases are shown in **Table 15-6**.



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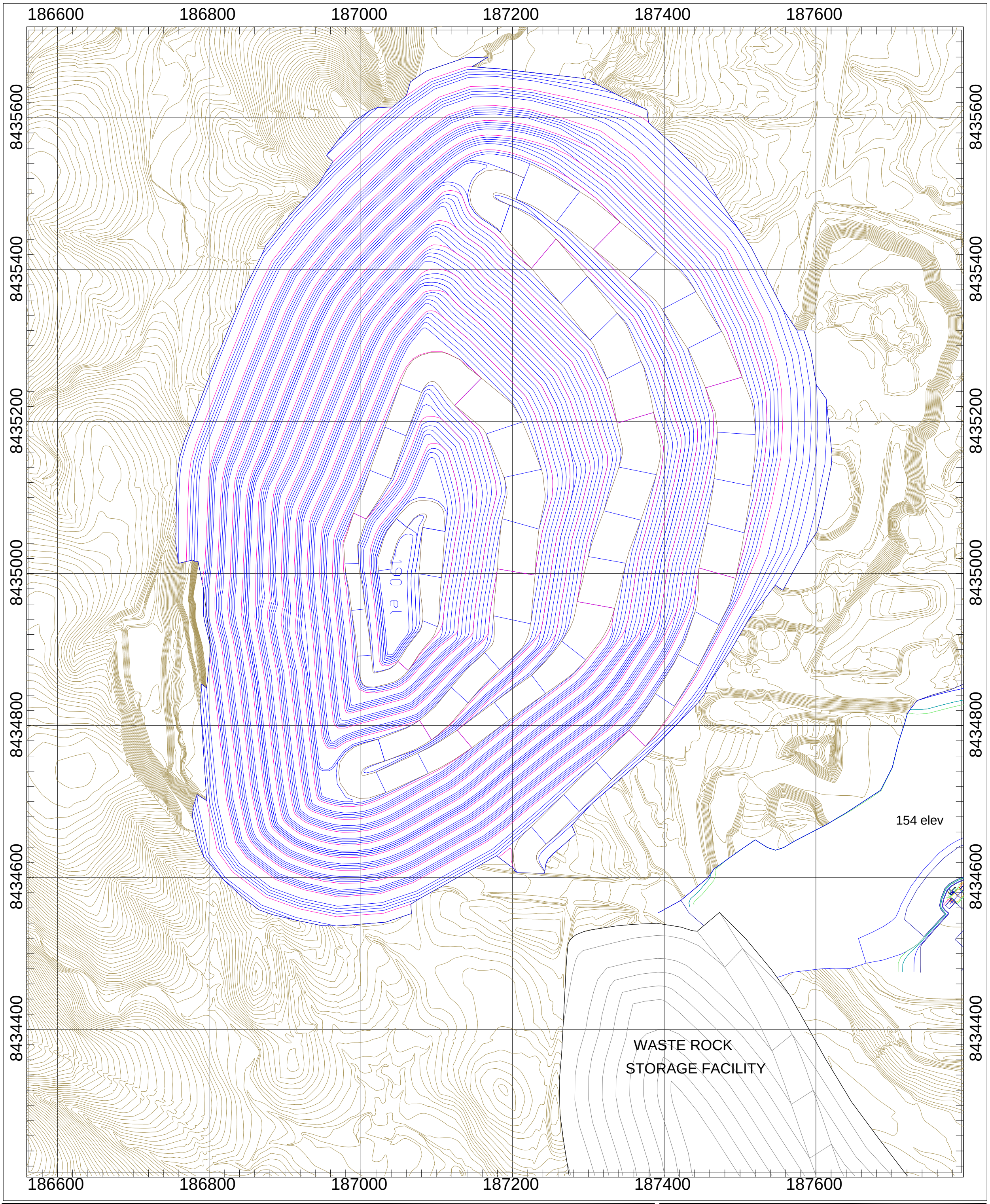
MINE DEVELOPMENT ASSOCIATES

VISTA GOLD CORP.

Vista Gold
Mt Todd Phase 1 Pit Map
Base Case



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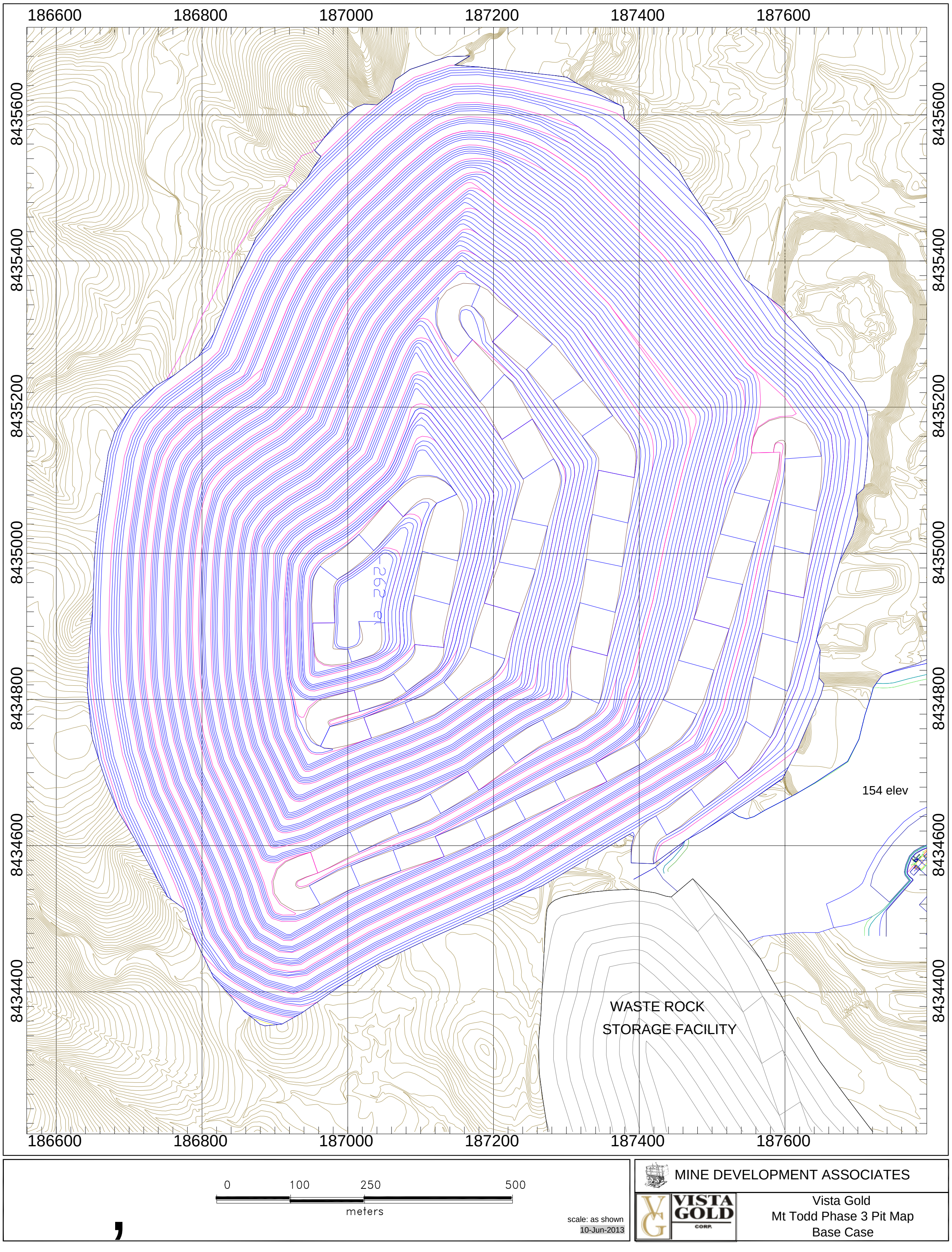
MINE DEVELOPMENT ASSOCIATES

VISTA GOLD CORP.

Vista Gold
Mt Todd Phase 2 Pit Map
Base Case



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MINE DEVELOPMENT ASSOCIATES

VISTA GOLD CORP.

Vista Gold
Mt Todd Phase 3 Pit Map
Base Case

15.3 Cutoff Grade

The breakeven and internal cutoff grade calculated using the economic parameters shown in **Table 15-1** are shown in **Table 15-5**. The internal cutoff grade assumes that mining is constrained to an economic pit and does not include the mining cost.

To enhance projects economics, the report assumes an elevated cutoff grade for reserves and scheduling. Reserves are reported using a 0.40 g Au/t cutoff grade for 50,000 tpd.

Table 15-5: $1,360 Calculated Gold Price Cutoff Grades (g Au/t)

	Sulfide	*Transition*	*Oxide*
Breakeven	0.36	0.38	0.39
Internal	0.31	0.32	0.32
Cutoff Grade Used	0.40	0.40	0.40

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

15.4 Dilution

The resource model with block sizes of 12m x 12m by 6m was used to estimate resources. The model was estimated based on this block size, and this model was used to define the ultimate pit limit and to estimate Proven and Probable reserves. MDA considers the 12m x 12m x 6m block size to be reasonable for open pit mining of the deposit and believes that this represents an appropriate amount of dilution for statement of reserves. No additional dilution or ore loss was accounted for.

15.5 Reserves and Resources

Mineral Reserves for the Project were developed by applying relevant economic criteria in order to define the economically extractable portions of the resource. MDA developed the reserves to meet NI 43-101 guidelines. The NI 43-101 guidelines rely on the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by the CIM council. CIM standards defining Proven and Probable Reserves are described below.

15.5.1 Mineral Reserve

Mineral Reserves are sub-divided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve.

A 'Mineral Reserve' is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

Mineral Reserves are those parts of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Qualified Person(s) making the estimates, is the basis of an economically viable project after taking account of all relevant processing, metallurgical, economic, marketing, legal, environment, socio-economic and government factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the treatment plant or equivalent facility. The term 'Mineral Reserve' need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

15.5.2 *Probable Mineral Reserve*

A 'Probable Mineral Reserve' is the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

15.5.3 *Proven Mineral Reserve*

A 'Proven Mineral Reserve' is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

Application of the Proven Mineral Reserve category implies that the Qualified Person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect potential economic viability.

Proven and Probable reserves are reported based on 50,000 tpd pit designs broken into individual phases. The phases represent logical mining subsets of the ultimate pit. **Table 15-6** reports the Proven and Probable reserves along with waste material for the pit designs discussed in previous sections. The reserves are shown to be economically viable based on cash-flows provided by Tetra Tech. MDA has reviewed the cash-flows and believes that they are reasonable for the statement of Proven and Probable reserves.

Table 15-6: Proven and Probable Reserves by Phase

	Proven K Tonnes	g Au/t	K Ozs Au	Probable Tonnes	g Au/t	K Ozs Au	Total P&P Tonnes	g Au/t	K Ozs Au	Waste Tonnes PAG_Wst	Un_Wst	NonPag_Wst	Total	Total Tonnes	Strip Ratio
Ph_1	23,424	0.97	730	13,972	0.98	439	37,397	0.97	1,169	24,124	6,230	5,669	36,023	73,419	0.96
Ph_2	20,615	0.80	532	24,924	0.81	649	45,540	0.81	1,181	34,392	16,269	35,705	86,366	131,905	1.90
Ph_3	13,568	0.93	407	36,223	0.82	955	49,792	0.85	1,362	54,771	39,019	63,385	157,175	206,966	3.16
Ph_4	14,887	0.81	388	61,835	0.79	1,569	76,723	0.79	1,958	119,284	39,215	124,286	282,786	359,509	3.69
Total	**72,495**	**0.88**	**2,057**	**136,955**	**0.82**	**3,612**	**209,451**	**0.84**	**5,669**	**232,571**	**100,733**	**229,045**	**562,349**	**771,800**	**2.68**

50,000 tpd reserves are reported using a cutoff grade of 0.40 g Au/t and constitute the official reserves.

15.6 In-pit Inferred Resources

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

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques for locations such as outcrops, trenches, pits, workings and drillholes.

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration.

Table 15-7 shows the inferred resources inside of the pit designs for each phase.

Table 15-7: In-Pit Inferred Resources

	K Tonnes	*g Au/t*	*K Ozs Au*
Phase I	3,214	0.61	63
Phase II	6,475	0.55	115
Phase III	2,484	0.54	43
Phase IV	9,499	0.57	173
Total	**21,672**	**0.57**	**394**

(1) Inferred resources are reported using cutoff grade of 0.4 g Au/t
(2) Mineral resources that are not mineral reserves have no demonstrated economic viability

15.7 Heap Leach Reserve

Dr. Deepak Malhotra, metallurgical qualified person, is responsible for the certification of the heap leach reserves, contained herein. Heap leach reserves are found in **Table 1-2** "Statement of Mineral Reserve Estimate". In addition to the ore mined from the Batman open pit, the mine plan contemplates processing the 13.4 Mt of ore from the existing heap leach pad through the mill at the end of the mine life.

The bottle roll and column leach test work undertaken at the ALS Metallurgy Laboratory in Australia has been reviewed (ALS report titled "CIL Extractive Testwork conducted upon heap leach (X2) and Drillhole (X2) composites from the Mt. Todd Gold Project dated April 2013). The testwork indicated the following:

- Cyanidation leach tests on "as is" material on the heap will extract ± 30% of the gold.
- CIL 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 13.4 Mt of heap leach ore meets the criteria necessary to be called "reserves" for the Mt. Todd Gold Project and should be included in the reserve tabulation based on the following:

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

16.0 MINING METHODS

16.1 MINING METHOD

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

16.2 WASTE MATERIAL DEFINITION

Some of the waste material at Mt. Todd contains sulfides which can create issues with acid generation. Tetra Tech provided MDA with classification criteria for waste material so that the resulting production schedule can include the segregation of waste types for proper handling. Waste was classified into three classes based on total sulfur content (by weight) as follows:

- Non-PAG: total sulfur <= 0.25%;
- Uncertain: total sulfur > 0.25% and <= 0.40%; and
- PAG: total sulfur > 0.40%.

Material classified as uncertain or potential acid generating (PAG) material was scheduled so that it could be placed inside of the ultimate waste dump. Non potential acid generating material (Non-PAG) was scheduled to be used to encapsulate the uncertain and PAG material as well as used for reclamation cover and construction material for TSF 1 and TSF 2.

16.3 MINE-WASTE FACILITIES

Total contained waste tonnage for 50,000 tpd is 562 Mt.

Non-PAG mine waste will be used for construction and final reclamation cover for the mine. Total non-PAG tonnage required for construction and capping material is assumed to be 60.5 Mt. The mine-waste facility has been designed to permanently contain the remaining waste material associated with reserves in the pit for both cases. This facility is an extension of existing waste dumps at site with the ultimate dump fully encapsulating the current dump. The ultimate design incorporates an angle of repose slope of 1.5 vertical to 1 horizontal with 8-m catch benches every 30 m in height. During the construction of the ultimate dump, PAG and uncertain waste materials will be dumped in the interior of each lift of the waste dump. Non-PAG material will be dumped to the outer edge of each lift. It is anticipated that at least a 10-m rind of Non-PAG material will surround all uncertain and PAG type waste material.

As each 30 m is finished, a layer of geosynthetic clay layer (GCL) material will be laid on top of the outer 15 m of the lift at an outward slope of at least 1.5%. This will promote drainage to the outer portions of the dump where drainage channels will allow storm water to drain without contacting any of the PAG material.

The mine will maintain access to the 8 m catch benches during the LoM. Once a lift is completed, and periodically through the closure of the waste dump, hydro-seeding trucks will access the benches and apply a hydro-seed solution to the slopes. Over time this is expected to increase the organic material within the fines at the surface of the waste dump and promote growth of native grasses and other plant life.

The current waste facility is approximately 24 m high located to the southeast of the pit. The ultimate dump design is 320 m above the original topography. The south end of the dump was designed to encroach on the existing RP 1 waste water storage facility. Designs are bound to the east by the process facility and to the west by another drainage basin. Designs are intended to promote any drainage to the current RP 1 waste water management dam.

In addition to the primary dump, additional waste is to be placed to level out an area to the northeast of the waste dump and extending around the crushing area. This will be placed early in the mine life to allow for road traffic to move from the pit to the shop.

A 40% swell factor and an average specific gravity of 2.67 (bank) have been assumed for volume calculations. The Base Case dump design will contain 410 Mt. With additional waste rock requirements for construction and cover, the dump is under-designed by 90 Mt of capacity. This waste is assumed to be stored as additional cover over top of the heap leach pad and tailings storage facility (TSF) areas. This will require additional design in future studies.

16.4 Mine-Production Schedule

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

Table 16-1 shows the mine-production schedule, including re-handle from stockpiles. For purpose of production scheduling, low-grade, medium-grade, and high-grade material was designated. The low-grade material used the cutoff grade of 0.40 g Au/t. Medium-grade and high-grade cutoff values used were 0.55 g Au/t and 0.85 g Au/t.

Ore material from the mine is to be sent from the pit directly to the crusher or to a mill ore stockpile. During pre-stripping, high-grade, medium-grade, and low-grade ore is stockpiled in the stockpile area northeast of the waste dump facility. High-grade and medium-grade ore is processed in the mill when mill capacity becomes available in year one.

For the purpose of scheduling, three ore stockpiles are assumed: High-grade; medium-grade; and low-grade stockpiles. The high-grade and medium-grade stockpiles are to be built within the low-grade stockpiling areas but are exhausted during the first year of processing when mill capacity becomes available. During the LoM, the low-grade stockpile is used as needed to feed the mill to full capacity. For this reason the stockpile grows and shrinks through the LoM. The maximum stockpile balance through the LoM is estimated to be 36.3 Mt.

Re-handling of stockpiled material will be done using a loader and trucks to haul ore to the crusher. **Table 16-2** shows ore stockpile balances for the end of each year.

During the construction of the waste dump, excess Non-PAG material will be placed to the north end of the waste dump. This will allow convenient re-handle of waste at the end of the mine life for capping of various facilities for reclamation. However, additional stockpiling of Non-PAG material may be done by operations where it is found to be convenient and cost effective. Waste re-handle is shown on the bottom of **Table 16-1** to account for capping and reclamation.

Ore sent to the mill is shown **Table 16-3**. This is a combination of ore shipped directly from the mine and ore that is reclaimed from stockpiles. The table summarizes the ore based on level of oxidation. The recovered ounces shown are based on the recoveries used for pit optimizations and are subject to change by Qualified Persons completing the metallurgical sections.

Table 16-1: Annual Mine Production Schedule – 50,000 tpd

			Pre-Prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Total
Total Mined	Pit to Stockpile	K Tonnes	11,764	10,619	8,318	7,299	6,596	11,268	582	-	-	-	6,581	5,111	-	68,139
		g Au/t	0.75	0.60	0.49	0.54	0.47	0.58	0.49	-	-	-	0.58	0.64	-	0.59
		K Ozs Au	284	205	130	128	100	209	9	-	-	-	123	105	-	1,293
	Pit to Crusher	K Tonnes	-	17,482	12,665	16,642	11,690	17,799	6,980	4,777	7,078	10,700	17,750	17,750	-	141,312
		g Au/t	-	1.24	0.87	1.08	0.90	1.08	0.84	0.67	0.63	0.66	0.83	1.14	-	0.96
		K Ozs Au	-	695	352	580	339	620	189	102	144	228	473	653	-	4,376
	Total Ore Mined	K Tonnes	11,764	28,101	20,983	23,941	18,285	29,066	7,561	4,777	7,078	10,700	24,331	22,861	-	209,451
		g Au/t	0.75	1.00	0.71	0.92	0.75	0.89	0.81	0.67	0.63	0.66	0.76	1.03	-	0.84
		K Ozs Au	284	900	482	708	439	829	198	102	144	228	596	758	-	5,669
	Mineralized Waste	K Tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		g Au/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-
		K Ozs Au	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Non-PAG Waste	K Tonnes	5,621	15,844	19,882	39,510	23,763	26,479	45,906	27,350	15,617	8,116	956	0	-	229,045
	PAG Waste	K Tonnes	14,392	13,622	23,174	22,292	26,512	26,187	18,680	20,449	21,727	21,779	21,204	2,554	-	232,571
	Undefined Waste	K Tonnes	4,748	4,337	12,234	16,425	21,333	5,662	6,693	6,606	8,139	8,814	5,704	38	-	100,733
	Total Waste Mined	K Tonnes	24,761	33,803	55,290	78,227	71,608	58,329	71,279	54,405	45,482	38,710	27,864	2,592	-	562,349
	Total Tonnes Mined	K Tonnes	36,525	61,905	76,273	102,169	89,893	87,395	78,840	59,182	52,560	49,410	52,195	25,454	-	771,800
	Strip Ratio	W:O	2.10	1.20	2.64	3.27	3.92	2.01	9.43	11.39	6.43	3.62	1.15	0.11		2.68
Re-Handlle Material	High-Grade Stockpile	K Tonnes	-	316	4,168	507	-	-	907	-	-	-	-	-	1,253	7,151
		g Au/t	-	1.28	1.11	1.07	-	-	1.05	-	-	-	-	-	1.07	1.10
		K Ozs Au	-	13	149	17	-	-	31	-	-	-	-	-	43	253
	Medium-Grade Stockpile	K Tonnes	-	-	917	601	6,061	-	6,117	-	-	-	-	-	4,294	17,989
		g Au/t	-	-	0.80	0.80	0.67	-	0.64	-	-	-	-	-	0.66	0.67
		K Ozs Au	-	-	23	15	131	-	125	-	-	-	-	-	91	387
	Low-Grade Stockpile	K Tonnes	-	-	-	-	-	-	3,747	12,973	10,672	7,099	-	-	8,508	42,999
		g Au/t	-	-	-	-	-	-	0.52	0.50	0.46	0.42	-	-	0.46	0.47
		K Ozs Au	-	-	-	-	-	-	63	210	157	97	-	-	126	652
	Total Re-Handle	K Tonnes	-	316	5,085	1,108	6,061	-	10,770	12,973	10,672	7,099	-	-	14,055	68,139
		g Au/t	-	1.28	1.06	0.92	0.67	-	0.63	0.50	0.46	0.42	-	-	0.58	0.59
		K Ozs Au	-	13	173	33	131	-	219	210	157	97	-	-	260	1,293
	Waste Re-handle	K Tonnes	-	-	-	-	-	-	-	-	-	-	1,000	1,000	3,000	5,000

Table 16-2: Annual Stockpile Balance – 50,000 tpd

			Pre-Prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13
High-Grade Stockpile	Added	K Tonnes	3,389	1,095	100	407	21	886	-	-	-	-	645	608	-	-
		g Au/t	1.14	1.08	1.09	1.07	1.03	1.05	-	-	-	-	1.05	1.09	-	-
		K Ozs Au	124	38	4	14	1	30	-	-	-	-	22	21	-	-
	Removed	K Tonnes	-	316	4,168	507	-	-	907	-	-	-	-	-	1,253	-
		g Au/t	-	1.28	1.11	1.07	-	-	1.05	-	-	-	-	-	1.07	-
		K Ozs Au	-	13	149	17	-	-	31	-	-	-	-	-	43	-
	Balance	K Tonnes	3,389	4,168	100	-	21	907	-	-	-	-	645	1,253	-	-
		g Au/t	1.14	1.11	1.09	-	1.03	1.05	-	-	-	-	1.05	1.07	-	-
		K Ozs Au	124	149	4	-	1	31	-	-	-	-	22	43	-	-
Medium-Grade Stockpile	Added	K Tonnes	4,930	3,474	253	1,434	108	3,496	-	-	-	-	1,853	2,441	-	-
		g Au/t	0.68	0.67	0.67	0.66	0.66	0.67	-	-	-	-	0.66	0.66	-	-
		K Ozs Au	108	75	5	30	2	75	-	-	-	-	39	52	-	-
	Removed	K Tonnes	-	-	917	601	6,061	-	6,117	-	-	-	-	-	4,294	-
		g Au/t	-	-	0.80	0.80	0.67	-	0.64	-	-	-	-	-	0.66	-
		K Ozs Au	-	-	23	15	131	-	125	-	-	-	-	-	91	-
	Balance	K Tonnes	4,930	8,404	7,740	8,573	2,620	6,117	-	-	-	-	1,853	4,294	-	-
		g Au/t	0.68	0.68	0.66	0.65	0.60	0.64	-	-	-	-	0.66	0.66	-	-
		K Ozs Au	108	183	165	180	51	125	-	-	-	-	39	91	-	-
Low-Grade Stockpile	Added	K Tonnes	3,445	6,051	7,965	5,458	6,467	6,886	582	-	-	-	4,083	2,063	-	-
		g Au/t	0.47	0.47	0.47	0.47	0.47	0.47	0.49	-	-	-	0.47	0.48	-	-
		K Ozs Au	52	92	121	83	97	104	9	-	-	-	62	32	-	-
	Removed	K Tonnes	-	-	-	-	-	-	3,747	12,973	10,672	7,099	-	-	8,508	-
		g Au/t	-	-	-	-	-	-	0.52	0.50	0.46	0.42	-	-	0.46	-
		K Ozs Au	-	-	-	-	-	-	63	210	157	97	-	-	126	-
	Balance	K Tonnes	3,445	9,495	17,460	22,918	29,385	36,271	33,106	20,133	9,461	2,362	6,445	8,508	-	-
		g Au/t	0.47	0.47	0.47	0.47	0.47	0.47	0.47	0.44	0.42	0.42	0.45	0.46	-	-
		K Ozs Au	52	144	265	348	446	550	496	286	129	32	94	126	-	-
Total	Balance	K Tonnes	11,764	22,067	25,300	31,491	32,027	43,294	33,106	20,133	9,461	2,362	8,943	14,055	-	-
		g Au/t	0.75	0.67	0.53	0.52	0.48	0.51	0.47	0.44	0.42	0.42	0.54	0.58	-	-
		K Ozs Au	284	476	433	528	497	706	496	286	129	32	155	260	-	-

Table 16-3: Annual Ore Delivery to the Mill Crusher – 50,000 tpd

		Pre-Prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Total
Sulfide Ore	K Tonnes	-	17,469	17,704	17,704	17,563	16,939	17,316	17,280	17,274	17,425	17,750	17,543	13,928	-	205,894
	g Au/t	-	1.25	0.92	1.07	0.83	1.11	0.72	0.55	0.53	0.57	0.83	1.14	0.58	-	0.85
	K Ozs Au	-	699	523	612	467	602	399	305	294	320	473	645	258	-	5,597
	Recovery	0%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	0%	82%
	K Ozs Au Rec	-	573	429	502	383	494	327	250	241	263	388	529	212	-	4,590
Mixed Ore	K Tonnes	-	272	42	16	144	565	270	313	290	222	-	207	77	-	2,418
	g Au/t	-	0.81	0.91	0.72	0.65	0.65	0.61	0.50	0.45	0.42	-	1.24	0.43	-	0.65
	K Ozs Au	-	7	1	0	3	12	5	5	4	3	-	8	1	-	50
	Recovery	0%	78%	78%	78%	78%	78%	78%	78%	78%	78%	0%	78%	78%	0%	78%
	K Ozs Au Rec	-	5	1	0	2	9	4	4	3	2	-	6	1	-	39
Oxidized Ore	K Tonnes	-	58	4	30	43	295	164	157	186	152	-	-	50	-	1,139
	g Au/t	-	1.04	1.03	0.69	0.66	0.66	0.66	0.50	0.45	0.42	-	-	0.42	-	0.58
	K Ozs Au	-	2	0	1	1	6	3	3	3	2	-	-	1	-	21
	Recovery	0%	78%	78%	78%	78%	78%	78%	78%	78%	78%	0%	0%	78%	0%	78%
	K Ozs Au Rec	-	2	0	1	1	5	3	2	2	2	-	-	1	-	17
Total Ore	K Tonnes	-	17,799	17,750	17,750	17,750	17,799	17,750	17,750	17,750	17,799	17,750	17,750	14,055	-	209,451
	g Au/t	-	1.24	0.92	1.07	0.82	1.08	0.71	0.55	0.53	0.57	0.83	1.14	0.58	-	0.84
	K Ozs Au	-	708	525	613	471	620	408	312	301	325	473	653	260	-	5,669
	Recovery	0%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	0%	82%
	K Ozs Au Rec	-	580	430	502	386	508	334	256	247	266	388	535	213	-	4,646

The additional ore projected in Years 1, 5 and 9 is due to the leap year.

16.5 EQUIPMENT SELECTION AND PRODUCTIVITIES

Vista has planned for Mt. Todd to be operated as an open-pit mine using large haul trucks, hydraulic shovels, and front-end loading equipment. Primary mine production is achieved using 31-m^3 hydraulic shovels along with 226-t haul trucks, though final equipment selection may differ.

Secondary mine production is achieved using 18-m^3 loaders along with the 226-t haul trucks. Loaders will be used mostly to mine ore from the pit to the crusher and reclamation of ore from stockpiles. Some waste production from the loader is anticipated as well.

Table 16-4 shows the maximum shovel productivity estimate based on scheduled time, availability, and truck and material parameters. This maximum productivity would require that trucks are always available; however, that is not always the case.

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

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

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

Incremental truck hours were added to waste haulage to account for waste material hauled to TSF 1 and TSF 2 for construction purposes.

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

Availability, efficiencies, operating hours and load and haul equipment requirements are shown in **Table 16-5.**

Table 16-4: Maximum Loader Productivity Estimate

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

Daily Schedule		
Shifts per Day	shift/day	2
Hours per Shift	hr/shift	12
Hours per Day	hrs/day	24
Shift Startup / Shutdown	hrs/shift	0.5
Lunch	hrs/shift	0.5
Breaks	hrs/shift	0.25
Operational Standby	hrs/shift	0.25
Total Standby / shift	hrs/shift	1.50
Total Standby / day	hrs/day	3.00
Available Work Hours	hrs/day	21.00
Schedule Efficiency	%	87.5%

Loading Parameters		**31 cm Hyd 226 T Trks**	**18 cm FEL 226 T Trks**
Shovel Mech. Avail.	%	85%	85%
Operating Efficiency	%	83%	83%
Bucket Capacity	m^3	31	18
Bucket Fill Factor	%	95%	95%
Avg. Cycle Time	sec	34	50
Truck Parameters			
Truck Mech. Avail.	%	85%	85%
Operating Efficiency	%	83%	83%
Volume Capacity	m^3	176	176
Tonnage Capacity	lt (Wet)	227	227
Truck Spot Time	sec	24	24

Shovel Productivity		**31 cm Hyd 226 T Trks**	**18 cm FEL 226 T Trks**
Effective Bucket Capacity	yd^3	29.45	17.10
Tonnes per Pass - Wet	lst (Wet)	56.8	33.0
Tonnes per Pass - Dry	lst (Dry)	52.6	30.5
Theoretical Passes - Vol	passes	5.98	10.29
Theoretical Passes - Wt	passes	4.00	6.88
Actual Passes Used	passes	4.0	7.0
Truck Tonnage - Wet	wmt/load	226	226
Truck Tonnage - Dry	dmt/load	210	210
Truck Capacity Utilized - Vol	%	67%	67%
Truck Capacity Utilized - Wt	%	100%	100%
Load Time	min	2.67	6.23
Theoretical Productivity	dst/hr	4,729	2,023
Tonnes per Operating Hour	dst/hr	3,930	1,680
Tonnes per Day	dst/day	**70,200**	**30,000**
Potential - 355 day year	t/year	**24,921,000**	**10,650,000**

Table 16-5: Annual Load and Haul Equipment Requirements – 50,000 tpd

Haulage Requirements		**Pre-Prod**	**Yr 1**	**Yr 2**	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yr 8**	**Yr 9**	**Yr 10**	**Yr 11**	**Yr 12**	**Total**
Productive Hours	Hrs	48,292	100,306	138,852	182,840	202,874	198,299	195,739	190,196	189,340	187,665	192,538	89,153	9,793	1,925,887
Operating Efficiency	%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%
Operating Hours	Hrs	58,183	120,850	167,291	220,289	244,427	238,915	235,830	229,151	228,121	226,102	231,974	107,413	11,799	2,320,345
Number of Trucks	#	14	20	27	36	38	38	38	38	38	38	38	18	3	383
Truck Availability	%	90%	89%	88%	88%	87%	86%	86%	85%	85%	85%	85%	85%	85%	
Available Operating Hours	Hrs	61,900	124,827	173,787	220,825	241,304	239,212	236,664	235,864	235,063	235,725	235,063	111,346	18,558	2,370,138
Use of Available Hours	%	98%	97%	96%	100%	101%	100%	100%	97%	97%	96%	99%	96%	64%	98%
Tonnes per Operating Hour	t/Hr	548	515	486	469	393	366	380	315	277	254	229	265	1,191	364
Hydraulic Shovel Useage															
Number of Shovels	#	2	3	3	4	4	4	3	3	3	3	2	2	-	3.2
Availability	%	48.3%	89.3%	88.6%	88.1%	87.3%	39.5%	85.7%	85.3%	85.0%	85.0%	85.0%	85.0%	0.0%	79.4%
Operating Efficeincy	%	45%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	0%	81%
Available Operating Hrs	Op Hrs	11,457	15,703	19,346	24,030	25,399	23,680	18,703	18,630	18,558	18,610	12,372	12,372	6,186	225,046
Tonnes Mined	K Tonnes	36,525	58,686	71,978	89,429	89,640	84,235	70,956	58,590	52,560	49,410	48,020	25,454	-	735,482
Operating Hours	Op Hrs	9,306	14,952	18,338	22,784	22,838	21,461	18,078	14,927	13,391	12,588	12,234	6,485	-	187,381
Use of Available Operating Hours	%	81%	95%	95%	95%	90%	91%	97%	80%	72%	68%	99%	52%	0%	83%
Front End Loaders															
Number of Loaders	#	-	2	2	2	2	2	2	2	2	2	2	2	2	2.0
Availability	%	0%	90%	89%	88%	87%	86%	86%	85%	85%	85%	85%	85%	85%	86%
Operating Efficeincy	%	0.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%	83%
Available Operating Hrs	Op Hrs	-	7,620	13,015	12,864	12,718	12,553	12,445	12,372	12,372	12,407	12,372	12,372	12,372	145,480
Tonnes Mined	K Tonnes	-	3,535	9,379	13,848	6,313	3,160	18,654	13,565	10,672	8,099	5,176	3,000	14,055	109,456
Operating Hours	Op Hrs	-	2,535	6,726	9,931	4,528	2,266	13,378	9,728	7,654	5,808	3,712	2,151	10,079	78,497
Use of Available Operating Hours	%	0%	33%	52%	77%	36%	18%	108%	79%	62%	47%	30%	17%	81%	54%

16.6 MINE PERSONNEL

Mine personnel estimates include both operating and mine-staff personnel. Operating personnel are estimated as the number of people required to operate trucks, loading equipment, and support equipment to achieve the production schedule. Mine staff is based on the people required for supervision and support of mine production. The mine-staff organizational chart is shown in **Figure 16-1**. The estimated number of mine personnel required to execute the mine plan is shown in **Table 16-6**.

Salaries for each position were estimated based on information received from Tetra Tech and Vista. Salaries include an allowance for 25% burden on top of the base salary for each position. The salaries used are shown in **Table 16-7**. The extended cost for labor by year is shown in thousands of US dollars. Note that mobile equipment labor costs are allocated to production equipment in the calculation of mining costs in later sections. In addition, a portion of the cost is allocated to construction of tailings facilities; however the table reflects the total personnel cost.

Note that the Mine Personnel tables do not include contractors. Vista anticipates using a Maintenance and Repair Contract (MARC) to maintain the mining fleet. Accordingly, no additional mechanics have been included for maintaining the mobile fleet. The mine personnel does however, include a Maintenance Superintendent, light vehicle mechanics, shop laborers, a maintenance planner, and service / fuel / and lube personnel.

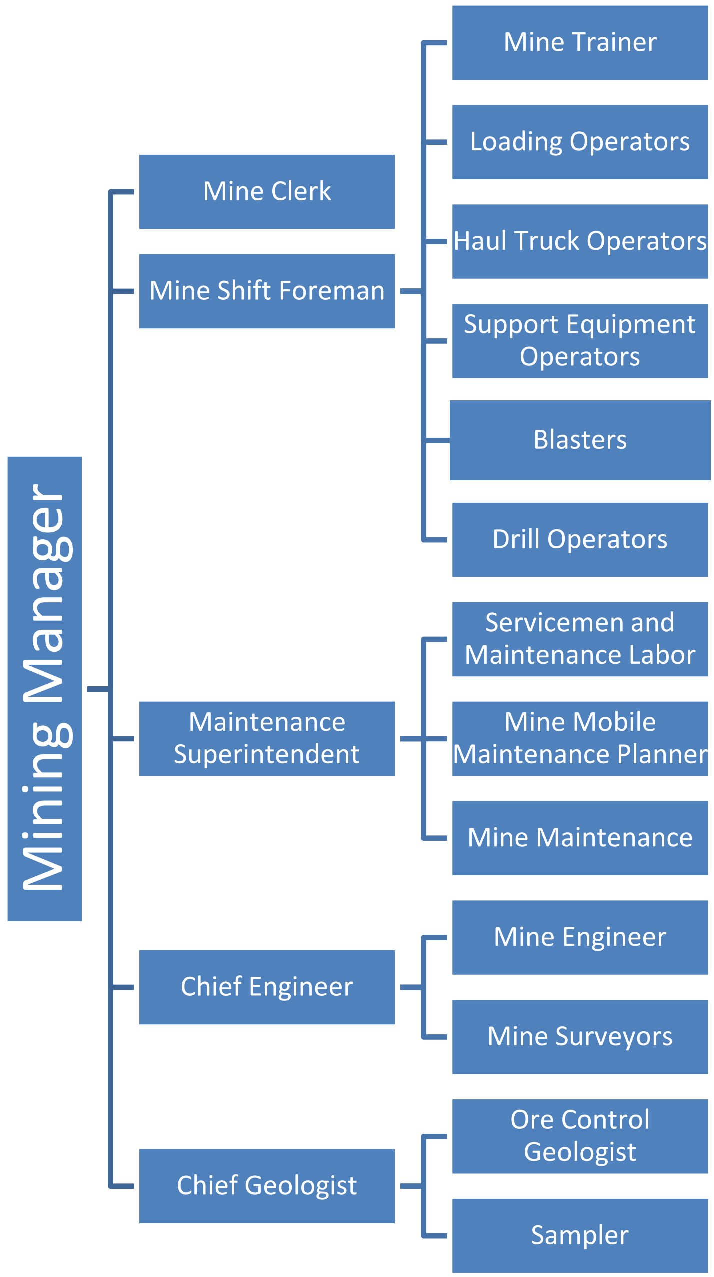


Figure 16-1: Mine Organizational Chart

Table 16-6: Mine Personnel Requirements – 50,000 tpd

Mine Overhead	**Pre-Prod**	**Yr1**	**Yr2**	**Yr3**	**Yr4**	**Yr5**	**Yr6**	**Yr7**	**Yr8**	**Yr9**	**Yr10**	**Yr11**	**Yr12**	**Yr13**
Mining Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Mine Clerk	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Mine Shift Foremen	10	10	10	10	10	10	10	10	10	10	10	10	-	-
Mine Trainer	1	1	1	1	1	1	1	1	1	1	1	1	-	-
Blaster	4	4	4	4	4	4	4	4	4	4	4	4	-	-
Blaster's Helper	4	4	4	4	4	4	4	4	4	4	4	4	-	-
Mine Production														
Loading Operators	8	12	17	22	20	17	20	20	20	16	14	12	8	-
Haul Truck Operators	54	77	108	138	152	152	152	152	152	152	152	72	12	-
Drill Operators	26	35	42	57	50	50	43	33	29	28	32	16	-	-
Support Equipment Operators	20	21	22	22	22	22	24	24	24	24	24	24	12	-
Total Mine Operating	**129**	**166**	**210**	**260**	**265**	**262**	**260**	**250**	**246**	**241**	**243**	**145**	**34**	**-**
Mine Maintenance														
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Light Vehicle Mechanics	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Tiremen	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Shop Laborers	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Maintenance Planner	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Service, Fuel, & Lube	8	8	8	8	8	8	8	8	8	8	8	8	8	-
Total Mine Maintenance	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**16**	**-**
Mine Engineering														
Chief Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Mine Surveyors	1	2	2	2	2	2	2	2	2	2	2	2	1	-
Surveyor Helper	2	2	2	2	2	2	2	2	2	2	2	2	1	-
Mine Engineer	3	3	3	3	3	3	3	3	3	3	3	3	-	-
Total Engineering	**7**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**8**	**3**	**-**
Mine Geology														
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Ore Control Geologist	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Sampler	2	2	2	2	2	2	2	2	2	2	2	2	-	-
Total Geology	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**3**	**-**
Total Mine Operations Workforce														
Mine Operations	129	166	210	260	265	262	260	250	246	241	243	145	34	-
Mine Maintenance	16	16	16	16	16	16	16	16	16	16	16	16	16	-
Mine Engineering	7	8	8	8	8	8	8	8	8	8	8	8	3	-
Geology	5	5	5	5	5	5	5	5	5	5	5	5	3	-
Total	**157**	**195**	**239**	**289**	**294**	**291**	**289**	**279**	**275**	**270**	**272**	**174**	**56**	**-**

Table 16-7: Mine Annual Personnel Costs (US$000s) – 50,000 tpd

Mine Overhead	Pre-Prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Total
Mining Manager	$308	$308	$308	$308	$308	$308	$308	$308	$308	$308	$308	$308	$308	$-	$4,000
Mine Clerk	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$100	$-	$1,301
Mine Shift Foremen	$1,169	$1,379	$1,375	$1,375	$1,375	$1,379	$1,375	$1,375	$1,375	$1,379	$1,375	$671	$-	$-	$15,600
Mine Trainer	$125	$125	$125	$125	$125	$125	$125	$125	$125	$125	$125	$61	$-	$-	$1,437
Blaster	$550	$552	$550	$550	$550	$552	$550	$550	$550	$552	$550	$268	$-	$-	$6,323
Blaster's Helper	$475	$477	$475	$475	$475	$477	$475	$475	$475	$477	$475	$232	$-	$-	$5,463
Mine Production															
Loading Operators	$855	$1,483	$2,164	$2,692	$2,560	$2,132	$2,500	$2,500	$2,500	$2,005	$1,750	$1,500	$1,000	$-	$25,641
Haul Truck Operators	$4,766	$9,656	$13,500	$17,270	$19,000	$19,052	$19,000	$19,000	$19,000	$19,052	$19,000	$9,000	$1,500	$-	$188,796
Drill Operators	$2,444	$4,367	$5,241	$7,077	$6,246	$6,268	$5,375	$4,125	$3,625	$3,510	$4,000	$975	$-	$-	$53,254
Support Equipment Operators	$2,027	$2,407	$2,477	$2,476	$2,476	$2,482	$2,700	$2,700	$2,700	$2,707	$2,700	$2,700	$1,350	$-	$31,901
Total Mine Operating	$12,818	$20,854	$26,315	$32,447	$33,214	$32,874	$32,508	$31,258	$30,758	$30,215	$30,383	$15,814	$4,258	$-	$333,715
Mine Maintenance															
Maintenance Superintendent	$234	$234	$234	$234	$234	$234	$234	$234	$234	$234	$234	$234	$234	$-	$3,041
Light Vehicle Mechanics	$235	$236	$235	$235	$235	$236	$235	$235	$235	$236	$235	$235	$235	$-	$3,057
Tiremen	$235	$236	$235	$235	$235	$236	$235	$235	$235	$236	$235	$235	$235	$-	$3,057
Shop Laborers	$235	$236	$235	$235	$235	$236	$235	$235	$235	$236	$235	$235	$235	$-	$3,057
Maintenance Planner	$150	$150	$150	$150	$150	$150	$150	$150	$150	$150	$150	$150	$150	$-	$1,951
Service, Fuel, & Lube	$940	$943	$940	$940	$940	$943	$940	$940	$940	$943	$940	$940	$940	$-	$12,228
Total Mine Maintenance	$2,029	$2,034	$2,029	$2,029	$2,029	$2,034	$2,029	$2,029	$2,029	$2,034	$2,029	$2,029	$2,029	$-	$26,391
Engineering															
Chief Engineer	$220	$221	$220	$220	$220	$221	$220	$220	$220	$221	$220	$220	$220	$-	$2,862
Mine Surveyors	$146	$293	$293	$293	$293	$293	$293	$293	$293	$293	$293	$293	$146	$-	$3,514
Surveyor Helper	$235	$236	$235	$235	$235	$236	$235	$235	$235	$236	$235	$235	$118	$-	$2,939
Mine Engineer	$408	$526	$525	$525	$525	$526	$525	$525	$525	$526	$525	$256	$-	$-	$5,919
Total Engineering	$1,010	$1,276	$1,273	$1,273	$1,273	$1,276	$1,273	$1,273	$1,273	$1,276	$1,273	$1,004	$484	$-	$15,234
Mine Geology															
Chief Geologist	$220	$221	$220	$220	$220	$221	$220	$220	$220	$221	$220	$220	$220	$-	$2,862
Ore Control Geologist	$309	$323	$323	$323	$323	$323	$323	$323	$323	$323	$323	$323	$323	$-	$4,183
Sampler	$220	$221	$220	$220	$220	$221	$220	$220	$220	$221	$220	$107	$-	$-	$2,529
Total Geology	$749	$765	$763	$763	$763	$765	$763	$763	$763	$765	$763	$650	$543	$-	$9,574
Total Mine Operations Workforce															
Mine Operations	$12,818	$20,854	$26,315	$32,447	$33,214	$32,874	$32,508	$31,258	$30,758	$30,215	$30,383	$15,814	$4,258	$-	$333,715
Mine Maintenance	$2,029	$2,034	$2,029	$2,029	$2,029	$2,034	$2,029	$2,029	$2,029	$2,034	$2,029	$2,029	$2,029	$-	$26,391
Engineering	$1,010	$1,276	$1,273	$1,273	$1,273	$1,276	$1,273	$1,273	$1,273	$1,276	$1,273	$1,004	$484	$-	$15,234
Geology	$749	$765	$763	$763	$763	$765	$763	$763	$763	$765	$763	$650	$543	$-	$9,574
Total	$16,606	$24,929	$30,379	$36,511	$37,278	$36,949	$36,572	$35,322	$34,822	$34,290	$34,447	$19,497	$7,313	$-	$384,914

17.0 RECOVERY METHODS

The key criteria used to determine the recovery method have been derived from metallurgical testwork and consultation with Vista and technology and equipment experts.

17.1 Process Design Criteria

A detailed design criteria has been developed for the process plant. The nominal headline design criteria are listed in **Table 17-1** below.

Table 17-1: Headline Design Criteria

	Unit	Base Case
Annual Ore Feed Rate	Mt/a	17.75
Operating Days per Year	d/y	355
Daily Ore Feed Rate	t/d	50,000
Crushing Rate (6637 hours per year availability)	tph	2674
HPGR & Milling Rate (7838 hours per year)	tph	2264
Gold Head Grade	g/t	0.84
Copper Head Grade	%	0.055
Cyanide Soluble Copper	%	0.015
Ore Specific Gravity		2.76
Grind P_{80} to Leach	µm	90
Gold Recovery (nominal)	%	82
Gold Production (nominal)	oz/d	1100
Gold Production (nominal)	oz/a	391,000

The testwork results collated from the 2011 and 2012 testing campaigns, together with the process design criteria, were utilized to develop the process flow sheet and mass balance.

17.2 Discussion - Flow Sheet Development

The flowsheet consists of primary crushing, closed circuit secondary crushing, closed circuit tertiary crushing using HPGR's, ball milling, cyclone classification, pre-leach thickening, leach and adsorption, elution electrowinning and smelting, carbon regeneration, tailings detox and disposal to conventional tailings storage facility. A schematic diagram of the flowsheet is presented in **Figure 17-1**.

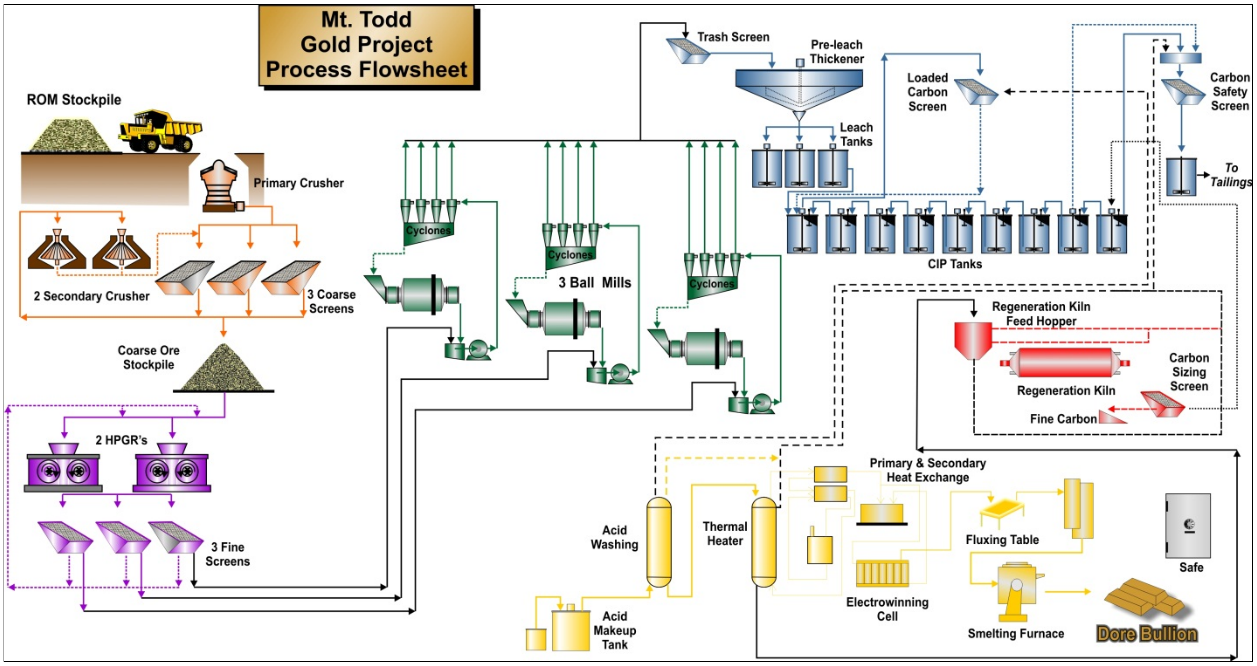


Figure 17-1: Mt. Todd Flowsheet

17.2.1 Crushing Modeling

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

Unconfined compressive strength (UCS) was measured on 16 samples. The values ranged from 39 MPa (med strong) to 174 MPa (very strong). Eighty percent of the results were in the strong to very strong designation of ore hardness.

The run of mine ore from the pit would have a maximum particle size of 1000 mm and P_{80} to the primary crusher of 450 mm. Two stages of crushing, primary and secondary, would be required to reduce particle size to a P_{80} of 31.5 mm as feed to the HPGR tertiary crushers. A single gyratory crusher would be sized for primary crushing duty in reducing ore size to a nominal P_{80} of 120 mm. Two secondary cone crushers operating in parallel and in closed circuit with two sizing screens cutting at 40 mm, will be used to produce the feed to the HPGRs at product size P_{80} of 31.5 mm.

17.2.2 Primary Crusher

The primary crusher power was calculated using both the FLSmidth (FLS) gyratory calculation model and the Metso Bruno model. Using the CW_i of 20 kWh/t and a fall through percentage of zero to simulate peak conditions, both models provided a peak primary crushing power requirements of approximately 574 kW and 576 kW respectively to reduce a feed F_{80} of 450 mm to a product P_{80} of 120 mm.

17.2.3 Secondary Crushers

Secondary crushing with closed circuit screening was modeled by Metso Corporation (Metso). Two MP1250 cone crushers operating in parallel will be used to reduce the primary crusher product to a final product P_{80} of 31.5 mm, for 50,000 tpd. The peak power demand per crusher will be 606 kW.

17.2.4 HPGR

HPGR power requirements to reduce the HPGR feed to a final product P_{80} of 3.25 mm was shown by the Polysius AG (Polysius) testwork to be 1.9 kWh/t of feed to the HPGR. The feed to the HPGR is the sum of new feed plus the recirculating load screen oversize material. The total feed to the HPGR is 2.45 times the fresh feed rate. HPGR testwork supported vendor recommendations call for two HPGR Polycom 24/17-8, each equipped with 2 x 3,300-kW drives.

17.2.5 Grinding Modeling

A variety of internal models were used to provide the estimated baseline ball mill power requirements and vendors were approached for proposals. The outcome of the investigations is that the circuit will incorporate three 18.15-MW dual pinion drive ball mills.

17.2.6 Thickener / Leach / CIP Design

Thickener

Thickener design was performed by Outotec based on the testwork results. A 45 m diameter thickener is required. Flocculant consumption of 15 g/t can be expected.

Maximum compaction of 63% solids was achieved at a bed depth of 240 mm, there would be no difficulty achieving the target of 55% solids in underflow and clear overflow.

Leach and Adsorption

The optimum leach and adsorption density as determined by the SPX testwork was 55% solids.

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

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

17.3 Description of Process Areas

Functional aspects of the sections of the process plant are shown in **Figure 17-2**, a schematic diagram of the plant layout.

Figure 17-2: Schematic Diagram of Plant Layout

17.3.1 Area 3100 - Crushing Circuit Availabilities

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

Additionally, Proteus has access to a two-year study and dynamic simulation of a large scale crusher operation in the tropics, which indicated the downtime was apportioned as follows:

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

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

Subsequently it was agreed with the mining design consultant MDA that they would allow for the costs of an extra loader and build of an emergency stockpile on the run-of-mine (RoM) pad and remove the downtime attributable to mining lack of supply in its entirety.

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

The crushing circuit was chosen based on reliability and similarity to existing mining operations. A single primary crusher in an open loop configuration and two secondary crushers in parallel in a closed loop configuration with product screens. The sized output will be conveyed to a buffer stockpile, providing two days live capacity. The primary and the secondary crushers discharge onto a common conveyor feeding the coarse ore screens. This configuration allows reduced conveyor footprint and maximum conveyor productivity.

The coarse ore screens will be fed by vibrating feeders which will be choke fed from feed bins. This arrangement will maximize the efficiency of the screens by ensuring full coverage of screen decks at controlled bed depth.

Crusher area dust will be controlled by dust collection at the screens and dust suppression in all other dust generating areas.

17.3.2 Area 3200 – Coarse Ore Stockpile, Reclaim and HPGR

The coarse ore stockpile will provide two day buffering between the crushing circuit and the HPGRs. Ore will be removed from beneath the coarse ore stockpile by two apron feeders.

A plant availability factor of 89.5% of 355 days/y has been used for the HPGRs and subsequent downstream processes, that is 7,838 operating hours per year. HPGR availability in high tonnage hard rock applications ranges from 89% to 92%. It is considered appropriate to use a conservative availability factor of 89.5% of the annual 8,760 hours for Mt. Todd ore due to its unmatched ore hardness.

Two HPGRs (2.4 m diameter rolls, 6.6 MW installed power) will operate in parallel to process 50,000 tpd.

The HPGRs are protected from damage by tramp metal by installation of metal detectors on feed conveyors.

A common HPGR product conveyor will receive the discharge from the HPGRs and convey the material to the fines screens feed bins. The HPGR fines screens are vibrating decks cutting at nominal 4 mm to produce an underflow product at P_{80} of 3.2 mm. The screens operate as wet screens with high pressure spray water applied to the decks to assist with screen efficiency. The screen overflow material, <10% moisture, will be conveyed back to the HPGR feed bins.

17.3.3 Area 3300 – Grinding and Classification

Three 18.15 MW ball mills will be used. The parallel ball mill circuits are in a conventional configuration. Fresh feed from the fines screens underflow will gravitate to the mill discharge sump and will be pumped together with the mill discharge slurry to the cyclones. The cyclone underflow will gravitate to the ball mill feed. The overflow will gravitate to linear screens for removal of trash.

An automated ball charging system will be provided to handle the dosage rate of approximately 15 t of balls per day to each mill.

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

In order to achieve the required 55% solids feed to the leach and CIP tanks, a pre-leach thickener will be used specified as a 45 m diameter unit.

The efficiency of adsorption was found to be low in the first tank if a conventional CIL circuit was utilized, therefore four hours of leaching will be provided before contact with carbon in the first adsorption tank. Six adsorption stages will be utilized. The leach and adsorption tanks will be sized to deliver the total residence time of 24 hours as determined by the test work.

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

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

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

17.3.5 Area 3500 – Desorption, Goldroom and Carbon Regeneration

Loaded carbon will be acid washed in an acid wash column prior to transfer to the elution column. Cold cyanide wash will be used to strip adsorbed copper prior to hot caustic cyanide wash to strip gold. The goldroom consisting of electrowinning, Drying and smelting facilities will be supplied as a vendor package. Stripped carbon will be regenerated using an indirect heated horizontal rotary kiln operating at up to 750^{o}C.

17.3.6 Area 3600 – Detoxification and Tailings

Two mechanically agitated, air sparged detoxification tanks in series will be used to contact the tailings slurry with sodium metabisulfite (SMBS) and provide residence time of one hour to reduce weak acid dissociable (WAD) cyanide to <10ppm.

The second detox tank will cascade overflow to a tailings pump hopper from where the tailings will be pumped to the TSF. A duty/standby configuration of series pumps will be used to ensure continuous operation.

17.3.7 Area 3700 – Reagents

SMBS will be delivered in a sea container with a semi-automated system for mixing on site. The SMBS solution will have storage for 3 days usage.

The sodium cyanide for leaching and elution will be delivered as briquettes in a bulk tanker. The solids will be dissolved in the tanker and cyanide solution will be transferred into a mixing tank to ensure full dissolution. The cyanide solution will then be transferred into four storage tanks allowing three days capacity. Additional sodium cyanide briquettes will be stored on-site to ensure back up stocks in case of delay to normal scheduled tanker deliver.

The hydrochloric acid for the acid wash column will be delivered as a 33% HCl solution and will have storage for 20 days usage.

Lime will be delivered as 92% activity quick lime powder in road tankers. The lime will be pneumatically transferred to storage silos approximately 200-t capacity. Lime will be slaked on a daily basis. Milk of lime will be distributed from a surge tank to the leach and detox tanks.

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

The lead nitrate for the leach circuit will be delivered as a powder in bulk bags and mixed to produce a 20% solution. The lead nitrate solution will have storage for 7 days usage.

17.3.8 Area 3800 – Process Plant Services

A standard suite of gold processing plant air and water services are provided to suit the Project requirements, described in more detail in the plant description section.

17.4 Process Water

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

- raw water supply;
- potable water supply;
- fire water supply;
- gland service water supply; and
- process water supply.

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

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

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

The process water system will include a 9,500 m^3 storage tank. Process water will be supplied to the plant via centrifugal pumps, one operating and one stand-by unit. This water supply will be used for process stream dilution and for use as spray water for the screens. The pre-leach thickener, tailings dam decant water and raw water all report to the process water tank.

17.4.1 Process Compressed Air

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

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

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

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

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

17.5 Plant Mobile Equipment

The plant mobile equipment will be as follows:

Table 17-2: Mobile Equipment for Process Plant

Light Vehicles	**Quantity**
Landcruiser wagon	2
Dual cab ute	10
Tray top ute	9
Troop carrier (ambulance)	1
Bus/troop carrier (15-seat)	1
Coach	3
Subtotal	**26**
Process Plant Mobile Equipment	
Loader - Cat 966G	Allowed for in mining
Tool Carrier - Cat IT28	1
Bob Cat - Mustang Case	1
Crane – 15-t Franna	1
Hiab Truck – 7-t	1
Service Truck – 2-t	1
2-t Forklift - allowance	2
25-t Container Forklift	1
80-t Crane	1
Subtotal	**9**

18.0 PROJECT INFRASTRUCTURE

18.1 Facility 2000 - Mine

The following section provides a description of the infrastructure project support facilities and project support services that have been developed to support the mining activities.

18.1.1 Area 2300 – Mine Support Facilities

Area 2300 mine support facilities consists of the buildings and services for the maintenance and repair of the mine vehicle fleet including heavy vehicles (HV). The area is located along the haul road adjacent to the proposed stockpile, between the pit and TSF 1.

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

The workshop facility will consist of six dome shelter structures mounted on sea containers with concrete floors. The sea containers come equipped as site offices, store services, store consumables, equipment repair and lube storage and dispensing facilities for the maintenance and servicing of HV's that are used for mining operations.

The workshop has been sized to service Caterpillar 793F mining trucks.

The warehouse facility will consist of one dome shelter structure mounted on sea containers with a concrete floor. The sea containers come equipped as site offices, rigging container, equipment repair workshop and stores consumable container for the storage of parts, components, spares and the like, used by the HV workshop for vehicle repair.

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

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

A mobile crane will be used externally to the dome shelters for the lifting and removal of vehicle parts.

Sub-Area 2310 Support Facilities – Fuel Farm

The fuel farm will consist of the relocated 600 kL tank complete with six new bowsers for dispensing into the HV fleet and one new 110 kL self-bunded diesel fuel tank complete with one bowser for dispensing into the LV's.

The new 110kL tank will be used for refueling the mine fleet when the 600 kL tank is not in service.

Sub-Area 2315 – Support Facilities - HV Washdown

The HV washdown facility will primarily be used for washing down the body and undercarriage of heavy vehicles prior to entering the HV workshop. The facility will consist of a single bay with raised platforms with stair access to four manually operated high pressure water cannons. The run-off water will be collected sent to the oily water separator and will include drive in concrete sumps and pits for waste water storage and recovery.

Sub-Area 2320 – Support Facilities - Crib / Ablutions / Lockers

The crib / ablutions / lockers facilities will be a transportable building used by mining personnel and is located adjacent to the HV workshop. The building will include all necessary system furniture. The crib area will also serve as a pre-start area.

Sub-Area 2325 – Support Facilities - HV Tire Change

The tire change facility will consist of one dome shelter mounted on sea containers with a concrete floor. The sea containers come equipped as equipment repair workshop and store consumables for the maintenance and changing of HV tires.

The tire change facilities will be complete with all services including power, lighting, communications, compressed air, water, specialist equipment and other services necessary for the changing of tires.

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

Sub-Area 2335 – Support Facilities - Lube Farm

The lube farm will consist of a bunded concrete slab for the storage of intermediate bulk containers (IBCs) containing oils and lubes for the servicing of HV's.

Sub-Area 2340 – Support Facilities - ANFO Facility / Magazine

The ANFO facility will be capable of storage and distribution of 10,000 tpa. It will be a secure compound for the ammonium nitrate (AN), ammonium nitrate emulsion (ANE) and diesel fuel.

The facility will include an area for AN storage, concrete hardstand for AN transfer to a mobile process unit (MPU) and containment pond for spill material.

The ANE will be tank stored on concrete plinths with air compressor and pumps for in-loading and out-loading of emulsion.

The diesel will be stored in a 110 kL self-bunded tank and includes a spill containment unit.

Magazine storage will consist of two secure modified shipping containers for the storage of detonators and high explosives. The magazines are located adjacent to the ANFO Facility and are surrounded by earth bunding and secure fencing.

A transportable building will be provided to include office / crib / ablution facilities at the facility for driver and delivery personnel.

Sub-Area 2345 – Support Facilities – Mining Offices

The mining offices will be a transportable building used by mining personnel and is located adjacent to the HV workshop. The building will include all necessary furniture and provide cellular and open planned offices along with meeting and training spaces.

Sub-Area 2355 – Support Facilities – Core Shed

The core storage facility will consist of one dome shelter mounted on sea containers with a sealed asphalt floor for the storage of core samples at the mine support area.

The sea containers will be equipped with racking for additional storage. The core storage facility will be complete with power and lighting.

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

18.1.2 Area 2400 – Mine Support Services

Mine support services consists of the services for the mine support facilities.

Sub-Area 2410 – Support Services - Potable Water

Potable water will be provided to the mine support facilities from the process plant area via pipework in common services trenching.

Sub-Area 2420 – Support Services - Raw Water

Raw water will be provided to the HV washdown storage tank at the mine support facilities via a tee off connection from raw water pipework running along the existing haul road to the process plant area.

Sub-Area 2430 – Support Services - Fire Water

The fire water main will be provided to the mine support facilities from the process plant area via pipework in common services trenching. Fire hydrants will be provided at required locations.

Sub-Area 2440 – Support Services – Air

Compressed air will be provided at the HV workshop and HV tire change facilities via suitably sized standalone air compressors and receivers.

Sub-Area 2450 – Support Services – Power

Power will be provided to the mines support facilities via a connection from the overhead power line running past the site into a kiosk substation. From the kiosk substation, power will be reticulated to the buildings and services in common services trenches.

Sub-Area 2450 – Support Services – Communications

Communications will be provided to the mine support facilities from the process plant area via a fiber optic cable in the overhead power lines and will terminate into a server room within the mining offices. Cat 6 cables will be reticulated to all required building and service locations.

18.2 Facility 4000 – Project Services

This section details the supply and distribution of services outside the process plant.

18.2.1 Area 4100 – Water Supply

Sub-Area 4110 – Water Treatment Plant (WTP)

A WTP will be fed with a combination of decant return, runoff pond water and pit dewatering discharge at a nominal rate of 500 m^3/hr. The WTP has been designed by Tetra Tech and its discharge will be returned nominally to the process plant for use as process water with excess discharged to the Edith River for disposal, pursuant to the conditions defined by Water Discharge License 178-2.

Impacted water on site is located in various retention ponds (RP) around site. Water will be pumped from each pond to an equalization pond (EQP) using barge pumps. The EQP will be located near the WTP, and will provide flow and influent chemistry equalization. The site already contains several suitable pumps, and significant lengths of pipeline. The existing equipment will be used in the design of the water supply system.

Sub-Area 4120 – Raw Water

The existing line from the RWD will be replaced with a 600-mm line approximately 4 km in length in order to handle the increased raw water requirements of the higher throughput.

Raw water will be supplied to the mine support facilities via a 1-km supply line to a storage tank in that facility. Raw water will be supplied to the power plant via a 2-km supply line. Water for the construction camp will be addressed based on its final location.

Supply of water to the construction camp via tanker was investigated and it was deemed that a supply pipeline was the most cost efficient method for transferring water to the construction camp.

Sub-Area 4130 – Potable Water

Potable water will be produced by a potable water treatment plant within the processing facility, and will be distributed to the process plant, mining, administration offices, laboratory facilities, safety showers, etc.

18.2.2 Area 4200 – Power Supply

Sub-Area 4210 – Power Generation

Power for the project will be by a natural gas powered turbine, supplemented by two reciprocating engines, located in a power station situated close to the site gatehouse. The power station will also be connected to the grid. Any surplus power will be sold back to the grid. The grid will also be used to supplement starting of large equipment such as the ball mills.

The generators will generate at a voltage of 11 kV. The existing grid connection is 22 kV. Step-up transformers, located within the power station area, will be required to bring these voltages to the site 33 kV distribution voltage.

The design of the power station and connection to the grid is by Power Engineers, Inc. (Power Engineers).

The battery limit for the power station is outgoing 33 kV terminals of a transformer or circuit breaker located within the power station.

Sub-Area 4230 – High Voltage Distribution

33 kV power distribution is via a main 33 kV switchroom, which will feed 33 kV buried cables supplying the process plant 33 kV Substations and the site wide overhead power line.

This main switchroom will be the main point of connection for incoming power from the power station as well as fiber optic communications between Telstra and the plant. The switchroom includes the main plant 33 kV switchgear feeders, metering, and an allowance for process plant power quality equipment.

It is not desirable to install overhead power lines close to the process plant, where it may cause a hazard to over-height vehicular traffic such as cranes. Therefore, in order to keep the overhead power lines away from these areas, 33 kV power to and from the process plant will be connected by buried cables. The buried cables will be connected to the main 33 kV switchroom.

Sub-Area 4232 – Overhead Power Lines

33 kV power will be provided from the power station via a single feeder. New 33 kV overhead power lines will be required to connect the power station to the process plant, which are approximately 1.2 km apart. These will be installed along a similar route as the main access road.

The 33 kV power line will also need to be distributed around site to the following new and existing locations:

- ANFO Facility;
- Heap Leach Facility;
- Accommodation Camp;
- Water Treatment Plant (WTP);
- Pit Dewatering;
- Mine Services;
- Communication Tower (depending on final location);
- Gatehouse;
- Future Tailings Dam/Decant Dam; and
- Raw Water Dam (RWD).

The total length of overhead power line required to reach the process plant and the above locations from the power station is 7.1 km. The overhead power line will incorporate a fiber optic cable into the earth conductor. Refer to Section 18.2.3.

Overhead power lines will be suitably rated for a high dust and lightning strike regions.

18.2.3 Area 4300 – Communications

Sub-Area 4310 – Fiber Optic

Two fiber optic cable rings will be installed around the process plant. These cables will generally be installed on cable ladders within the plant, although sections of the cable will be buried where cable ladder access is not available. The second cable is to provide redundancy within the process plant in case of damage to the first cable and will follow a separate route where this is practical.

The plant fiber optic cables will contain up to 72 cores and will incorporate separate networks for all data communications including those for the plant process control system, the site IT system, a site VoIP phone system, site CCTV and security networks, and fire detection systems.

Outside of the process plant, the fiber optic cables will be incorporated into the earth conductor of the overhead power lines. The optical ground wire (OPGW) has a dual function. It is designed to replace traditional earth wires on overhead power lines with the added benefit of containing optical fiber cores that can be used for communications purposes. These will connect communications equipment from locations such as the power station, waste water treatment plant (WWTP), gatehouse and ANFO facility to the plant communications network.

A Telstra communications hut will be provided outside the site gatehouse. A fiber optic cable will be installed underground between this communications hut and the site overhead power line network at the first overhead power line pole from the power station.

As it utilizes the OPGW, the fiber optic cable between the Telstra hut and process plant will not have a second redundant cable, although some redundancy will be provided by using additional fiber cores in the OPGW.

Sub-Area 4311 – Phones

Telephone communications will be via digital VoIP (Voice over IP) technology. This allows telephone calls to be made over an IP network rather than through a separate copper network. Calls can traverse the company's IT network or an external portal.

Sub-Area 4312 – Radios

This is covered in Area 5810.

Sub-Area 4313 – Telemetry

A radio telemetry system will be used to communicate to remote locations that require data exchange between the process plant and the remote location. Radio telemetry will be provided to communicate with the decant water return pump station, ANFO Facility and pit dewatering pump station.

The system will incorporate a master telemetry station, located in a switchroom of the process plant, and a number of remote telemetry stations, located in remote equipment switchboards.

The master telemetry station will communicate with the plant process control system via the preferred communications network, and will communicate with the remote locations via radio. Suitable antennas will be installed at each location.

Control of the remote equipment will be made by the plant process control system, with sufficient data exchange to ensure correct operation of the remote equipment.

18.2.4 Area 4400 – Tailings Dam

A total of 209 Mt of process tailings will be stored in two separate TSFs over a design operating mine life of 13 years at a nominal ore processing rate of 50,000 tpd. The existing TSF 1 was constructed during active mining operations between 1996 and 2000. A total of approximately 9 Mt of ore was processed during this period (MWH, 2006). Approximately 62 Mt of additional tailings will be stored in the existing TSF 1 during production years 1 through 4 of the proposed mining operations through staged vertical raises of the existing facility constructed using a combination of centerline and upstream construction techniques. TSF 2 will be constructed east of the BP using the staged upstream construction technique. A total of approximately 147Mt of tailings will be deposited in TSF 2, following tailings deposition in TSF 1. The embankments for TSF 1 and TSF 2 will be constructed using non-acid generating waste rock from the open pit operations. TSF 1 will be raised vertically by approximately 18m from its current crest elevation of 140m. The proposed TSF 2 will have a maximum height of 55 m from the downstream toe to the ultimate crest at elevation.

The design storage capacities for TSF 1 and TSF 2 were based on an assumed average in-place dry density of 1.6 t/m^3. Thickened tailings will be pumped to the TSFs at a nominal rate of 50,000 tpd and a solids content of 54% by weight. Tailings will be deposited within the TSF using subaerial deposition techniques through multiple spigot points along the crest of the TSF.

The tailings containment system for TSF 2 will consist of a geomembrane liner placed on the base and the upstream slope of the Stage 1 embankment. The geomembrane liner along the base of the facility will be underlain by a geosynthetic clay liner (GCL) as a bedding layer. An underdrainage system consisting of gravel drains is designed under the base liner to collect groundwater inflows from areas upstream of the TSF. A network of gravel overdrains is designed above the liner system to collect pore water drainage from the tailings mass.

The proposed TSF 2 will require limited construction within the existing Horse Shoe Creek and Stowe Creek drainages. Diversion channels will be constructed along the west and southeast toe of the proposed TSF to divert storm-water flows reporting to the creeks.

The existing decant structures within TSF 1 will be raised along with the staged vertical expansion to reclaim water from the supernatant pond. The decant structures will be supplemented with a barge mounted pump to provide additional decant capacity if needed. Barge mounted pumps will be used for reclaiming supernatant water from TSF 2. The reclaim water from the TSFs will be used for process plant makeup as required.

18.2.5 Area 4500 – Waste Disposal

Sewerage waste disposal will be via a WWTP installed at the process plant area. The mine support and process plant buildings will be connected to the WWTP via the sewer pipework reticulation system.

18.2.6 Area 4600 – Plant Mobile Equipment

The plant mobile equipment to be purchased will be as follows:

Table 18-1: Mobile Equipment for Process Plant

Light Vehicles	
Landcruiser wagon	2
Dual cab ute	10
Tray top ute	9
Troop carrier (ambulance)	1
Bus/troop carrier (15-seat)	1
Coach	3
SUBTOTAL	26
Process Plant Equipment	**Quantity**
Loader - Cat 966G	Allowed for in mining
Tool Carrier - Cat IT28	1
Bob Cat - Mustang Case	1
Crane - 15t Franna	1
Hiab Truck - 7t	1
Service Truck - 2t	1
2t Forklift - allowance	2
25t Container Forklift	1
80t Crane	1
SUBTOTAL	9

18.3 Facility 5000 – Project Infrastructure

This section provides a description of the Project infrastructure required for the construction and operation of the process plant.

18.3.1 Area 5100 – Site Preparation

Bulk earthworks for the process plant is designed to minimize the import of fill material. Where fill material has to be imported, material from the existing RoM pad ramp and from the existing stockpile located adjacent to the Tollis and Golf pits will be used.

The site will be prepared such that there is a mono slope fall from the proposed boundary of the pit toward the existing drainage channel on the east side of the proposed process plant.

Stormwater V-drains will be designed to collect water alongside the unsealed plant roads and direct them beneath the roads via corrugated steel culverts to prevent scouring of plant roads. All storm-water run-off will be directed toward the existing drainage channel on the east side of the proposed process plant. Rip-rap protection to earthwork embankments adjacent to the existing drainage channel on the east side of the proposed process plant will also be installed for flood protection.

18.3.2 Area 5200 – Support Buildings

The support buildings consist of the building infrastructure for the process plant.

Sub-Area 5210 –Administration Office

The administration offices will be a transportable building used by mine management and administration personnel and is located at the northern end of the process plant site. The building will include all necessary system furniture and provide cellular and open planned offices along with conference and meeting spaces.

Sub-Area 5211 –Process Plant Office

The process plant offices will be transportable buildings located within the existing flotation building. The buildings will include all necessary system furniture and provide cellular and open planned offices.

Sub-Area 5220 – Workshop / Warehouse

The workshop / warehouse will be incorporated into the existing flotation building along with the process plant offices, main control room, crib and ablutions and the light vehicle workshop. The offices / ablutions / crib facilities will be transportable building located within the annex of the building.

The existing flotation building will require modifications to steelwork and replacement of the concrete floors. The building will be complete with all services including overhead traveling crane, power, lighting, communications, compressed air, water, specialist equipment and other services necessary for the maintenance of process plant equipment and the light vehicle (LV) fleet.

Sub-Area 5230 – Reagent Store

The reagent store will consist of one dome shelter mounted on sea containers with a concrete floor. The sea containers will act as additional space for the storage of reagents.

The reagent store will be complete with all services including power and lighting. The dome shelter will be constructed of steel frame and tensile fabric with a fabric life expectancy of 10 years.

A secure and fenced hardstand area will be provided for the storage of sea containers used at the reagent store.

Sub-Area 5240 – Crib / Ablutions

The crib / ablutions facilities will be transportable buildings located within the existing flotation building. The buildings will include all necessary system furniture fixtures and fittings.

Sub-Area 5250 – Emergency Services

The emergency services facilities will be a transportable building used by the first aid and fire and emergency services personnel. It will be located adjacent to the administration offices in the process plant.

Sub-Area 5260 – Sample Preparation and Laboratory

The sample preparation and laboratory facility will be a structural steel shed with insulated metal clad walls and roof and concrete floor for the receipt and storage of samples and a transportable building containing the preparation areas, laboratory and offices for processing samples. The sample preparation and laboratory building and equipment has been sized to process 450 samples/day.

Sub-Area 5270 – Gatehouse / Security

The gatehouse / security facilities will be a transportable building used by security personnel for recording movement to and from the site and drug and alcohol testing of contractors and employees. The facility will include a boom gate and pedestrian turnstile. The gatehouse will be located along the access road to the process plant.

Sub-Area 5280 – Control Building – Crushing

The crushing control room will be a transportable building located at the primary crusher. The buildings will include all necessary system furniture for one operator.

Sub-Area 5281 – Control Building – Main Control Building

The main control room will be a transportable building located within the flotation building. The building will include all necessary system furniture for one supervisor and three operators.

Sub-Area 5282 – Control Building – CIP

The CIP control room will be a transportable building located on top of the leach tanks and subdivided into a control room and a titration room. The buildings will include all necessary system furniture for one supervisor and two operators.

Area 5300 – Access Roads, Parking and Laydown

The existing plant access road is suitable for both the Base Case and the Alternate Case. Miscellaneous road repairs will be carried out to the existing plant access road.

The existing RP 5 culvert crossing the existing drainage channel on the east side of the proposed process plant is suffering from corrosion to the existing corrugated steel culvert. These corrugated steel culverts will be replaced.

18.3.3 Area 5400 – Heavy Lift Cranage

Heavy lift cranage covers all the cranage that will be needed on site during the construction period for the heavy lifts on site, approximated as follows:

Table 18-2: Heavy Lift Cranage Requirements

Crane	Duration (Hours)
600 t	270
450 t	470
200 t	540
180 t	540
100 t	810
80 t	3090
50 t	1610

18.3.4 Area 5600 – Bulk Transport

Bulk transport in and out of site will be weighed on a weighbridge near the gatehouse. The weighbridge will be located on a dedicated off take from the main road. The site weighbridge will be capable of weighing a triple trailer tanker or truck.

18.3.5 Area 5800 – Communications

Sub-Area 5810 – Sitewide Radio Communications

The site will require radio communication for both individual division usage and also across all site personnel for emergencies. Some divisional usage will be localized, but coverage across the site will generally be required.

To cover all radio communications requirements across the site, there will be a suitably located, approximately 50 m tall communications tower complete with appropriate antenna arrays and ancillary equipment. A communication hut will be located at the base of the tower. This hut will house the necessary repeaters, servers, communications equipment and back-up batteries to provide a robust radio communications system. A maximum of eight individual radio channels will be provided.

Depending on the final location, the communications hut will either be connected to the overhead power line network or, in the case where this is not practicable, a solar powered power supply will be provided.
The communication hut back-up battery life will last for a minimum of 10 hours on loss of incoming power.

18.4 Electric Power Plant

Power Engineers, Inc. evaluated the site electrical peak power demands and estimated them to be between 70MW and 83MW. Electrical demand will be met through the installation of a Rolls Royce Trent 60 Wet Low Emissions single gas turbine generator and up to two reciprocating gas engines (such as the MAN 20V35/44SG) located near West Creek adjacent to the WRD, approximately 1 km south of the HLP along the main entrance road, gas supply pipeline and incoming power lines. Water consumption in the power plant is primarily for gas turbine inter-stage cooling and fogging, injection into the combustor for NO_x control and periodic cleaning of the turbine compressor section. The existing raw water source has the capacity and will supply the required 216,000 m^3 per year for power plant operation.

The preliminary design of the power station is based on the Rolls Royce Trent 60 WLE gas turbine with inlet fogging and inter-stage inner cooling for increased capacity and efficiency and the option to include up to two MAN 20V35/44SG reciprocating gas engines to meet increased demand above the net 57MW generated by the gas turbine. Auxiliary equipment was selected to meet the utility requirements of the gas turbine and supplement the existing facilities on the site.

Net power generated by the Rolls Royce gas turbine will be approximately 57MW supplemented by up to two MAN 20V35/44SG gas engines capable of 9.2 MW each. Peak electrical loads will be supplied by Power Water Corporation (PWC) via connection to the utility grid.

18.4.1 Generation Option Selection

Power generation equipment selection is based on past PFS performed in 2010 and 2011 for Vista analyzing equipment options for Mt. Todd. The Rolls Royce Trent 60 WLE gas turbine was chosen as the optimum configuration from the PFS because it provided the mine's entire baseload electrical demand with a single gas turbine and has the lowest installation and life cycle operating costs per kilowatt-hour of the gas turbine options. With electricity demands estimated between 56MW and 76MW, alternative scenarios with up to two reciprocating engines were added to the power station to meet increased demand.

18.4.2 Electrical

Conceptual Design

A conceptual electrical one-line diagram, **Figure 18-1,** has been created to show the electrical distribution system from the 22kV utility interconnect down to the 400V power distribution bus. The equipment ratings are preliminary and based on generator ratings provided by Rolls Royce and some preliminary calculations. The equipment ratings shown are for cost estimating purposes only. The actual equipment ratings will be determined using detailed load flow and short circuit studies during detailed design.

Plant Arrangement

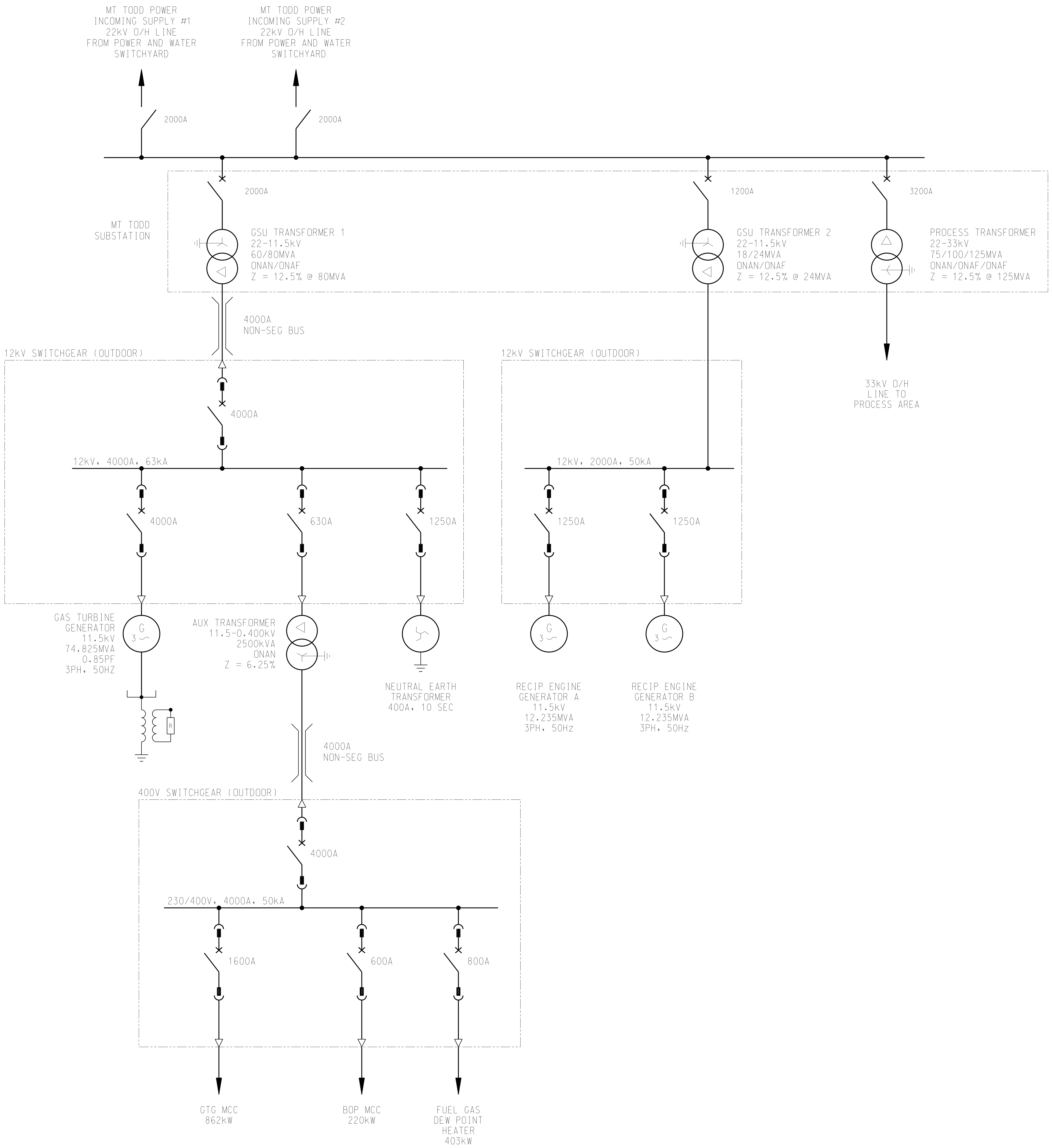
The auxiliary electrical equipment is included on the mechanical general arrangement drawing (**Figure 18-2)**. All the equipment physical sizes are based upon similar equipment from reference projects except for the GSU transformer which is based upon an equipment supplier for this project.

Step Load Capability

Generators in the 35-60MVA size category can typically accommodate a 5 to 7 MW load change and stay within 5% frequency range for 1 or 2 seconds.[2] Larger step load changes may be accommodated with a heavy inertia design generator rotor. Transient load response is also improved with 1-15 MW of load already connected to the generator. The largest loads at the Mt. Todd Mine are induction motors estimated at

[2] *Position Paper #50 - LM Gas Turbine Load Accept Guidelines,* GE Energy Aero Division, January 16, 2009

approximately 18.1 MW each. If the motors are started sequentially the generating station will likely be able to accommodate the loads. When a complete process equipment load list is available a load analysis is required to determine voltage and frequency limitations that will need to be considered in the final design of electrical equipment. If the generating station equipment is not sufficient for plant startup loads the site has the option of supplementing onsite generating capacity with the connection to the electrical grid if needed for startup loads.



NOTES

1. BLACK START CAPABILITY IS NOT CURRENTLY INCLUDED.
2. VOLTAGE SUBJECT TO CHANGE BASED UPON IEC STANDARDS.
3. ALL EQUIPMENT RATINGS ARE PRELIMINARY AND SUBJECT TO CHANGE.

INTER-DISCIPLINE REVIEW

DISC	ARCH	CIVIL	ELECT	HVAC	I&C	MECH	STRUCT
DATE	-	-	-	-	-	-	-
INIT	-	-	-	-	-	-	-

REV	REVISIONS	DATE	DRN	DSGN	CKD	APPD
C	RE-ISSUED FOR REVIEW	06/07/13	RDQ	CH	CH	GH
B	RE-ISSUED FOR REVIEW	11/13/11	RDQ	CH	RL	PS
A	ISSUED FOR REVIEW	08/17/11	RDQ	CH	BT	PS

DSGN	CH	07/18/11
DRN	RDQ	07/19/11
CKD	BT	07/21/11

SCALE: NONE

FOR 594x841 DWG ONLY

POWER ENGINEERS

VISTA GOLD

VISTA GOLD CORPORATION

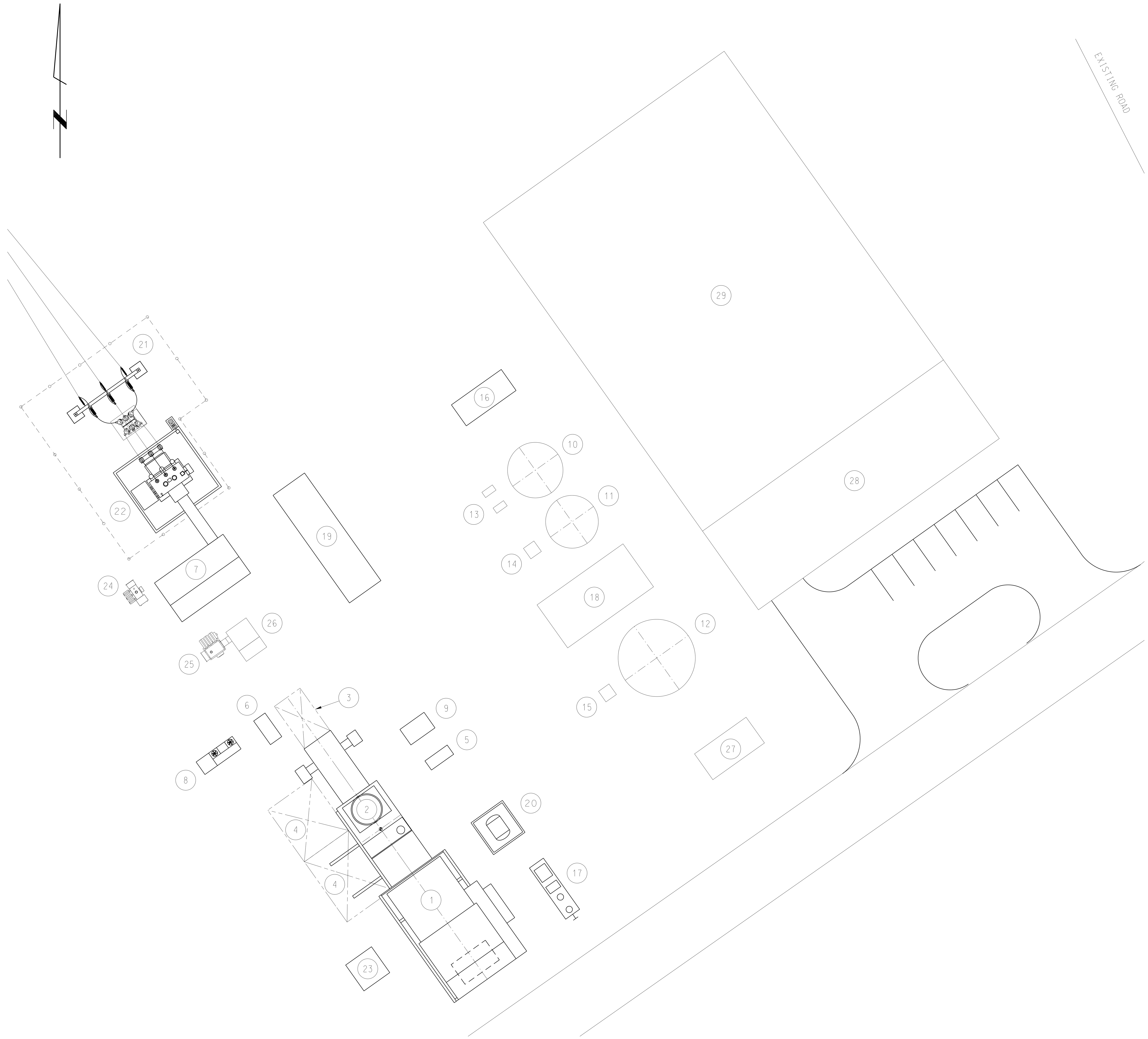
MT. TODD POWER STATION

CONCEPTUAL ONE-LINE ELECTRICAL DIAGRAM

JOB NUMBER 122329

REV C

DRAWING NUMBER E1-1



ITEM	DESCRIPTION	NOTES
1	GAS TURBINE GENERATOR (GTG)	
2	EXHAUST STACK	
3	GENERATOR REMOVAL AREA	
4	TURBINE REMOVAL AREA	
5	GAS TURBINE FIRE SUPPRESSION SKID	
6	GENERATOR LUBE OIL SKID	
7	GENERATOR SWITCHGEAR	
8	WET SURFACE AIR COOLER SKID	
9	WATER INJECTION SKID	
10	DEMIN WATER STORAGE TANK - 5.3 M DIA	$114M^3$
11	SERVICE WATER STORAGE TANK - 4.9 M DIA	$91M^3$
12	RAW WATER STORAGE TANK - 7.3 M DIA	$303M^3$
13	DEMIN WATER PUMPS	
14	SERVICE WATER PUMPS SKID	
15	RAW WATER PUMPS SKID	
16	AIR COMPRESSOR SKID	
17	OIL/WATER SEPARATOR	
18	WATER TREATMENT SKID	
19	CONTROL MODULE/PDC/MCC	
20	WASH WATER DRAINS TANK	
21	SWITCHYARD	
22	GSU TRANSFORMER	
23	ISI PUMP SKID	
24	NEUTRAL EARTH TRANSFORMER	
25	AUX TRANSFORMER	
26	400V SWITCHGEAR	
27	FUEL GAS CONDITIONING SKID	
28	CONTROL ROOM/WORKSHOP	
29	ENGINE HALL EXPANSION	

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INTER-DISCIPLINE REVIEW

DISC	ARCH	CIVIL	ELECT	HVAC	I&C	MECH	STRUCT
DATE							
INIT							

REV	REVISIONS	DATE	DRN	DSGN	CKD	APPD
C	ISSUED FOR REVIEW	06/07/13	AJG	TRC		TRC
B	ISSUED FOR REVIEW	03/08/12	KAC	KAC	TLD	TRC
A	ISSUED FOR REVIEW	09/01/11	KAC	TLD	TLD	TRC

DSGN	TLD	07/07/11
DRN	KAC	07/12/11
CKD	TRC	07/12/11

SCALE: 1:200

FOR 594x841 DWG ONLY

POWER ENGINEERS

VISTA GOLD

VISTA GOLD CORPORATION

MT. TODD POWER STATION

GENERAL ARRANGEMENT

JOB NUMBER 122329

REV C

DRAWING NUMBER M1-1

18.5 WET INFRASTRUCTURE

Section 18.2 discusses water supply inclusive of the water treatment plan (WTP), raw water, and potable water supply. Additional information regarding regulations, design criteria and receiving water is provided herein.

18.5.1 WATER TREATMENT PLANT

A WTP will be fed with a combination of decant return, runoff pond water and pit dewatering discharge at a nominal rate of 500 m^3/hr. The WTP has been designed by Tetra Tech and its discharge will be returned nominally to the process plant for use as process water with excess discharged to Edith Creek for disposal, pursuant to the conditions defined by Water Discharge License 178-2.

Impacted water on site is located in various Retention Ponds (RP) around site. Water will be pumped from each pond to an EQP using barge pumps. The EQP will be located near the WTP, and will provide flow and influent chemistry equalization. The site already contains several suitable pumps, and significant lengths of pipeline. The existing equipment will be used in the design of the water supply system.

18.5.2 WATER QUALITY STANDARDS FOR WASTE WATER DISCHARGE

The Edith River and its tributaries are protected beneficial use under the Water Act 2000 for aquatic ecosystem protection. The initial Project Waste Discharge License (WDL 135) was issued for the 2005/2006 season on December 21, 2005 and subsequently transferred to Vista (EPA, 2005). The WDL was updated by Nature Resource, Environment, The Arts and Sport (NRETAS) on December 30, 2010 (WDL 178) and allows discharge from RP 1 based on the same gauge height of 0.81 m which was equated to a 20,000 fold dilution of the impacted water (NRETAS, 2010). In February 2013 Vista was issued its current Waste Discharge License (WDL 178-2) which authorizes Vista to:

- Release water from the mine site subject to the quality of the water and the flow of the Edith River meeting strict environmental controls and monitoring requirements;
- Maintain water quality objectives outside of any agreed mixing zone when defined for the receiving waters;
- Monitor and report the results of water quality, macro invertebrates and sediment sampling and analysis; and
- Monitor just below the mixing zone to ensure water in the Edith River complies with Australian Drinking Water Guidelines for Health.

To meet these requirements, WDL 178-2 allowed controlled discharges from the RP 1 siphon that depended on minimum flows in the Edith River; specifically, water could be released from the RP 1 discharge point when the Edith River at SW4 was flowing at 12 cubic meters per second (m^3/s) and the water level was above 0.81 m. This flow was considered sufficient to ensure downstream compliance with established copper criteria, which in turn diluted other regulated constituents to acceptable levels.

Historic WDL 178 requires development of site-specific trigger values (TV) for ecosystem protection within the Edith River. To this end, GHD (2011) developed interim TV following Australian and New Zealand Environment and Conservation Council and Agriculture and Resource Management Council of Australia and New Zealand (ANZECC & ARMCANZ, 2000) guidelines. The interim TV are based on the 20th and 80th percentiles for pH and the 80th percentile for all other parameters at SW2, the upstream monitoring point on the Edith River (**Table 18-3**). The 95% species protection TV were used when ANZECC & ARMCANZ guidelines were not available or if values were higher than the 80th percentile at SW2. The magnesium and sulfate interim TV were developed based on literature values including Elphick et al. (2011) for sulfate and Van Dam et al. (2010) for magnesium (as magnesium sulfate).

Although the interim TV are most applicable to the Project and are the current basis by which site water quality is assessed, water quality guidelines that may also be relevant to the Project include:

- The Australian and New Zealand Environment and Conservation Council (ANZECC) and Agriculture and Resource Management Council of Australia and New Zealand (ARMCANZ) guidelines for recreational water quality and aesthetics. These guidelines are intended to protect waters for recreational activities such as swimming and boating and to preserve the aesthetic appeal of water bodies (ANZECC and ANZMARC, 2000).

Table 18-3: Site Specific Interim Trigger Values for Edith River

Analyte[a]	Trigger Value	Source
pH	6-8	ANZECC and ARMCANZ Table 3.3.4
Dissolved Oxygen (mg/L)	80%	ANZECC and ARMCANZ Table 8.2.10
Temperature	No Data	No ANZECC and ARMCANZ Guidelines
Conductivity (µS/cm)	20-250	ANZECC and ARMCANZ Table 3.3.5
Magnesium (µg/L)	2.5	Van Dam et al 2010 Environ Toxicol Chem 29(2):410-421
Sulfate (mg/L)[b]	129	Elphick et al 2011 Environ Toxicol Chem 30(1):247-253
Aluminum (µg/L)	149	Site derived 80th percentile
Cadmium (µg/L)[c]	0.2	High reliability TV ANZECC and ARMCANZ Table 3.4.1
Cobalt (µg/L)	90	Moderate reliability TV ANZECC and ARMCANZ pg 8.3-116
Chromium (III) (µg/L)	3.3	Low reliability TV ANZECC and ARMCANZ Table 3.4.1
Chromium (VI) (µg/L)	1	High reliability TV ANZECC and ARMCANZ Table 3.4.1
Copper (µg/L)[c]	1.4	High reliability TV ANZECC and ARMCANZ Table 3.4.1
Manganese (µg/L)	1700	Moderate reliability TV ANZECC and ARMCANZ Section 8.3.7
Nickel (µg/L)[c]	11	High reliability TV ANZECC and ARMCANZ Table 3.4.1
Lead (µg/L)[c]	3.4	High reliability TV ANZECC and ARMCANZ Table 3.4.1
Iron (µg/L)	300	Canadian Guideline ANZECC and ARMCANZ pg 8.3-123
Mercury (µg/L)	0.6	High reliability TV ANZECC and ARMCANZ Table 3.4.1
Zinc (µg/L)[c]	8.0	High reliability TV ANZECC and ARMCANZ Table 3.4.1

Note: Reprinted from GHD, December 2011
[a] Metals 0.45 µm filtered
[b] Sulfate TV is hardness dependent; displayed TV shown is for soft water (Elphick et al 2011 Environ Toxicol Chem 30(1):247-253)
[c] TV is hardness dependent

Table 18-4 summarizes background water quality values that are available in the Edith River.

Table 18-4: Background water quality in the Edith River

Parameter	Unit	Median	80%ile
pH[1]	su	6.18	6.36
Electrical Conductivity[a]	µS/cm	18.0	21.1
Aluminum[a]	µg/L	86	149
Cadmium[a]	µg/L	0.01	0.02
Copper[a]	µg/L	0.52	0.68
Zinc[a]	µg/L	0.9	1.6
Sulfate[a]	mg/L	0.05	0.2
Magnesium[b]	mg/L	0.8	0.8
Cobalt[b], [c]	µg/L	1	1
Chromium (III) [b], [c]	µg/L	1	1
Chromium (VI) [b], [c]	µg/L	1	1
Manganese[b]	µg/L	17	17
Nickel[b], [c]	µg/L	1	1
Lead[b], [c]	µg/L	1	1
Iron[b]	µg/L	790	790
Mercury[b], [c]	µg/L	0.1	0.1

[a] From GHD, December 2011, "WDL 178-1 – Interim Site Specific Trigger Values"
[b] From November 2011 sample collected by Vista
[c]Value below detection limit

Influent Water Quality and Treatment

Influent water quality and quantity at the WTP were established via a modeling effort. The water balance modeling completed indicates that over the life of the mine, the capacity of the WTP will need to be 4500 m^3/hr. Modeling of the chemistry of the feed streams to the WTP provided the influent water quality.

Table 18-5 summarizes the expected influent water quality to the WTP, the effluent goals of the WTP, and the expected effluent quality.

Table 18-5: WTP Influent Water Quality, Effluent Goals, and Expected Effluent Quality

Parameter	Unit	Influent Water Quality		Effluent Goal	Expected Effluent Quality
		Wet Season	*Dry Season*		
Magnesium	mg/L	97	195	72.4	0.49
Sulfate	mg/L	907	2,022	5436	990
Aluminum	µg/L	20,000	28,000	119	13
Cadmium	µg/L	39	60	7.6	5.3
Cobalt	µg/L	450	680	3,757	0.53
Chromium	µg/L	0.71	1.1	97.9	1.35
Copper	µg/L	3,000	4,500	30.9	0.61
Manganese	µg/L	5.5	9.6	71.0	0.32
Nickel	µg/L	470	650	423	31.9
Lead	µg/L	24	26	102	1
Iron	µg/L	2.0	5.1	240	2.66
Mercury	µg/L	0.17	0.17	21.1	0.17
Zinc	µg/L	8,800	13,000	271	124
Total Cyanide	mg/L	10	10	3.38	4.38

The selected treatment process includes a number of steps to treat dissolved constituents of interest. The WTP begins with the addition of lime to raise the pH of influent water to pH 11 in order to precipitate dissolved constituents, specifically magnesium. Ferric chloride will also be added to assist in the co-precipitation and adsorption of other metal species, specifically cadmium. The solids generated by the addition of lime and caustic will be removed in a high rate solids contact clarifier. Sulfuric acid will be used to reduce the pH back to neutral, which will facilitate the precipitation of aluminum.

Microfilters will be used to remove the precipitated aluminum. Solids collected in the clarifier will be disposed of at the TSF, and water used to backwash the microfilters will be recycled through the treatment plant. The plant will be designed in a two-train configuration. This will allow for redundancy and ease of maintenance, and allow the plant to operate more efficiently at lower flow rates. The WTP design calls for specialized equipment that may have lead times as long as 37 weeks.

Effluent goals were created by estimating the annual average stream flow of the Edith River, and calculating the available dilution. Directly meeting all of the discharge criteria set forth in WDL 178-1 would be prohibitively expensive, so the dilution from native Edith River flows is required. During the Wet Season, sufficient dilution is available to allow direct discharge of treated water. During the Dry Season, sufficient dilution is not available, so the treated effluent will be used on-site, for dust suppression (DUST), process make up water, and other uses.

Expected capital costs, are presented in **Table 18-6**.

Table 18-6: WTP Expected Capital Costs

Cost Component	Cost (US$)
Equipment Costs	$12,510,000
Labor Costs	$1,391,000
Construction Indirects/Contingency	$7,388,000
Total	**$21,289,000**

The WTP will be housed in a pre-engineered metal building with a slab on grade or spread footing foundation. Several tanks, including a lime silo, will be located near the building but outdoors. These tanks will be covered and otherwise protected against environmental conditions, and be provided with slab on grade foundations. A lined earthen EQP of approximately 60,000 m^3 will accept flows from around the site and will serve as the influent to the WTP. Submersible pumps located outside of the WTP building in a below grade concrete structure will be used to transfer influent flow to the WTP. A pipeline to the discharge at the Edith River will be required. An effluent pump station will be supplied to pump treated water to other uses (besides direct discharge) around the site, as mentioned previously. The building will require electric power, and bulk deliveries of lime, ferric chloride, sulfuric acid, sodium hypochlorite, citric acid, and caustic will be required for the treatment process.

18.5.3 RWD and Pipeline

The RWD is the sole source for potable and elution water, as it is the only freshwater source on site.
The RWD reservoir provides storage of fresh water for use at the mine and processing facility. The reservoir is on a tributary of Horseshoe Creek, located north and east of TSF 1, and retains a reservoir storage volume of approximately 4.5 million cubic m.

The RWD reservoir provides a ready supply of fresh water for several uses. The water balance indicates that process water obtained from recycled process water and TSF decant water will need to be supplemented, particularly in the dry season. The RWD reservoir can also provide water for dust control and onsite potable water supply. Dust control will be needed during the dry season on roads and exposed soil surfaces around the project site. The reservoir generally fills in the wet season (November through April) and will be used during the dry season (May through October). It can also supply wet season fresh water, if needed.

The existing dam is a 13-m high, 114-m long, zoned-embankment dam with a low level outlet and a spillway. The outlet works are connected to the fresh water pipeline that extends to the process plant. The spillway is designed for the extreme flood and discharges to Horseshoe Creek.

The existing line from the RWD will need to be replaced with a 600 mm line approximately 4 km in length in order to handle the increased raw water requirements of the higher throughput.

Raw water will be supplied to the mine support facilities via a 1-km supply line to a storage tank in that facility. Raw water will be supplied to the power plant via a 2-km supply line.

18.5.4 Potable Water

Potable water will be produced by a potable water treatment plant within the processing facility, and will be distributed to the process plant, mining, administration offices and laboratory facilities.

Drinking water quality guidelines that may be relevant to the Project include:

The Australian Drinking Water Guidelines (ADWG). These guidelines are intended to provide a framework for good management of drinking water supplies that will assure safety at point of use (NHMRC and NRMMC, 2004).

18.5.5 Sanitary Sewer System

The sanitary sewer system will consist of gravity lines conveying the sewerage to a single sewer lift station. The lift station will then pump the sewer to the septic system for treatment. The effluent will flow by gravity to a leach field.

19.0 MARKET STUDIES AND CONTRACTS

19.1 MARKETS

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

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

Demand is presently high with prices showing remarkable increases during recent times. The 36-month average London PM gold price fix through May 2013 is US$1,545/oz.

19.2 CONTRACTS

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

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

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This section discusses the environmental permitting and social impact aspects of the Project. Numerous consultant team members are in the process of finalizing the Environmental Impact Study ("EIS") for submittal in mid-2013.

20.1 ENVIRONMENTAL STUDIES

A number of environmental studies have been conducted at the Mt. Todd Gold Project 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.

The following reports provide descriptions of the existing environmental conditions at the Mt. Todd Gold Project (Chadwick T&T Pty Ltd, 2009):

- Waste Discharge License 135 (EPA Northern Territory, 2005);
- Mt. Todd Environmental Management Services – Report 1: Environmental Assessment (MWH, 2006a);
- Mt. Todd Environmental Management Services – Report 2: Water Management (MWH, 2006b);
- Mt. Todd Gold Project Preliminary Economic Assessment (Gustavson, 2006);
- Environmental Management Plan (EMP) (Vista, 2007a);
- Mt. Todd Waste Discharge License Report, 2006 – 2007 (Vista, 2007b);
- Mt. Todd Water Treatment Plant Commissioning Report (Vista, 2009);
- Mt. Todd Blueprint Rehabilitation Strategy Report (Department of Regional Development, Primary Industry, Fisheries and Resources (DRDPIFR), 2008b);
- Mt. Todd Strategic Rehabilitation Reference Group: Status Update Papers in lieu of Meeting 11 (DRDPIFR, 2008c);
- Mt. Todd Mine Site Status Report, April 2008 to October 2008 (Vista, 2008); and
- Mt. Todd Project Draft Environmental Impact Statement (EIS) submitted June 28th, 2013.

20.2 SITE WIDE WATER BALANCE

A site side water balance was developed within the GoldSim® software platform (Version 10.50) to simulate 13 years of mine production (12 active mining years and 1 additional year processing stockpiles) at the Vista Mt. Todd site for 50,000 tpd.

The site wide water balance was developed to simulate site conditions in order to:

- Identify water treatment plant capacity;
- Determine equalization pond sizing; and
- Quantify make up water requirements from the RWD for process make up water, dust control, and potable/elution needs.

20.2.1 Site Wide Water Balance Model

Water Balance Modeling

The site wide water balance model was constructed using deterministic (known with certainty) inputs, such as pond stage-storage relationships, as well as stochastic (known, but with some uncertainty) inputs, such as rainfall. Water storage within retention ponds (RPs) was modeled using the basic formula:

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

Information provided to the model and the rules by which the site features interacted are summarized below.

Model Elements

The site features (pits, facilities and associated RPs) represented within the model are:

- Waste Rock Dump (WRD, RP 1);
- Low Grade Ore Stockpile (LGOS, RP 2);
- Batman Pit (BP, RP3);
- Plant Runoff Pond (RO, RP 5);
- Heap Leach Pad (HLP);
- Raw Water Dam (RWD);
- Equalization Pond (EQP);
- Water Treatment Plant (WTP);
- Process Plant;
- Dust Control; and
- Tailings Storage Facility (TSF, RP 7).

General Assumptions

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

- RP 1, RP 2, RP 3, RP 5 and the HLP reported to the EQP which feeds the WTP;
- The EQP receives water only if it is not at risk for overtopping. Given this logic, overtopping events are allowed to occur at the RPs;
- Inputs to all ponds included precipitation, catchment runoff (where applicable) and seepage (where applicable);
- Outputs from ponds included evaporative loss, pumping and overtopping events (uncontrolled releases);
- All RPs report to the EQP that feeds directly to WTP. The EQP was sized to contain five days of capacity;
- A bleed stream of 10% of the TSF is sent directly to WTP to maintain chemistry;
- Permitted discharges to the Edith River were not allowed from any of the RPs. WTP effluent was allowed to discharge to the Edith River throughout the year, regardless of season;
- The HLP was run through process at the end of the Life of Mine (LoM); and
- WRD water that reported to RP-1 was not allowed to be used for dust control.

Initial Conditions

- RP 1, RP 2, RP 3, RP 5 and the EQP were assigned water surface elevations based on outputs from the pre-production model, which concluded at the initiation of production with initial conditions based on real site water surface elevation observations from October 2012.
- The dust suppression tank was assumed to be full at the initiation of production.

Flow Rates

- Process makeup water requirements throughout the LoM were 2,427 m^3/day. This value accounts for recycle from the thickener overflow within the Process circuit.
- TSF decant flows were typically 1,825 m^3/day.
- Dust suppression requirements varied between 220 and 1,153 m^3/day.
- WTP capacity is 500 m^3/hr.

Climatological Inputs

The Vista Mt. Todd site wide water balance model was designed to reflect weather conditions as accurately as possible, given the arid tropical climate (i.e., wet, monsoon conditions with intense, short-lived events and extended hot, dry periods). Features within the climatological section of the model included:

- Stochastic precipitation inputs, such that a range of likely scenarios may be determined, thus allowing the user an understanding of dry (5-percentile), typical (mean) and wet (95-percentile) climatological effects on the site. Precipitation was entered as a statistical probability, rather than assuming rainfall on a given day and assigning a rainfall depth. The mean monthly total precipitation values (total mm per month) provided to the model for wet season months shown in **Table 20-1**.

Table 20-1: Mean Monthly Precipitation

Month	Precipitation (mm)
January	292
February	259
March	193
April	38
October	35
November	111
December	239

- Use of synthetic data to extend the precipitation period of record and identify extreme rainfall events. Mt. Todd site precipitation data were collected from 1993 to present. A correlation of the site data to nearby Katherine gage allowed the rainfall time series to be extended to that of the Katherine gage, 138 years, thus allowing determination of extreme rainfall events that may not otherwise present within a data set spanning less than 20 years.
- Linking incidental rainfall and runoff within the Edith River using the Australian Water Balance Method. Catchment parameters within the model were adjusted to ensure optimal agreement between modeled runoff and observed Edith River flows. Real precipitation, evaporation and Edith River flow data were used to calibrate this portion of the model.
- The model used stochastic evaporation, based on a monthly time step and calculated by the Blaney-Criddle method. The Blaney-Criddle approach recommends against use of a time step smaller than one month.

Model Run

A time step of one day was selected for the model run. Use of stochastic inputs allowed a "Monte Carlo" analysis to be run wherein the 13 year LoM was simulated across 1,000 realizations (or equally likely weather scenarios), each incorporating the uncertainty associated with meteorological conditions and collectively providing an envelope of expected outcomes at the site. All RPs were subjected to the stochastic weather events as described in the previous section and reported to the WTP.

20.2.2 Results

- The WTP rate of 500 m^3/hr and EQP sizing for 5 days of storage was determined to be appropriate for the 50,000 tpd production process water requirements.
- The greatest amount of make-up water required from the RWD was quantified as 35,460 m^3/day occurring when the summation of TSF decant and WTP effluent could not satisfy process makeup water requirement.
- The WRD pond was typically observed to overtop less than 0.3% of the time during the 13-year simulation.[3] RP2 and RP5 storage may be optimized.

20.3 Waste and Tailings Disposal, Site Monitoring, and Water Management

20.3.1 Waste Rock Disposal

Waste rock will be disposed of in a WRD constructed as an expansion of the existing WRD. All waste rock will be analyzed to identify the rock as potentially acid generating (PAG) or non-PAG material before being hauled to the WRD. Non-PAG material will be stockpiled for use in reclamation covers or placed in the WRD. Construction of the WRD is described in Section 16. Reclamation and closure of the WRD is described in Section 20.6.

20.3.2 Tailings Disposal

Tailings will be disposed of in two tailings storage facilities, TSF 1 and TSF 2. TSF 1, an existing tailings storage facility, will be expanded with six additional raises to the embankment and construction of two new saddle dams at the west end of the impoundment. A second tailings storage facility, TSF 2, is to be constructed following completion of raises to TSF 1. The TSF 2 impoundment is designed to operate as a zero-discharge facility, utilizing a 60-mil linear low-density polyethylene (LLDPE) textured (double sided) geomembrane liner underlain by a Geosynthetic Clay Liner (GCL) as bedding material for tailings containment. Construction of the tailings storage facilities is described in Section 18.2.

TSF seepage will be collected and treated in the water treatment plant. Reclamation and closure of the TSFs is described in Section 20.6.

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

20.3.3 Site Monitoring

Currently, surface water monitoring is conducted at various locations at the site. A comprehensive site monitoring plan will be developed as part of future work on the Project.

20.3.4 Environmental Water Management

The primary existing environmental issue at the site is water management resulting from the project shutdown without implementation of closure or reclamation activities. The pit and existing water RPs (excluding the raw water pond) contain acidic water with elevated concentrations of regulated constituents. This water has been managed through evaporation, pumping to RP 3 for containment, and controlled discharge to streams during major flow events. Historically, average wet season rainfall has resulted in uncontrolled overflow from retention ponds to the Edith River due to the high amount of precipitation received in short periods of time coupled with insufficient pumping capabilities. Current water management strategies employed by Vista appear to be successful at preventing recurrence of historic uncontrolled discharges and are minimizing impacts on the Edith River downstream of the Mt. Todd site.

Prior to, during, and following resumed mining operations, water management at the site involves distinct water management components including in-pit treatment, seepage management, treatment of acid rock drainage and metal laden leachates (ARD/ML), and surface water management. Each of these components is discussed in the subsections below:

In-situ Pit Treatment

In-situ treatment of RP3 is being conducted by use of limestone and quicklime. Treatment has been undertaken to produce water to be discharge at rates protective of water quality in the Edith River in a suitable timeframe to meet project requirements. The treatment methodology includes raising the pH of the water within RP3 to greater than pH 7.0 using limestone and quicklime in succession to capitalize on the capabilities of the low-cost limestone and minimize the quantity of quicklime required to attain a pH sufficient to precipitate additional metals. Raining the pH to greater than 7.0 will result in the precipitation of key metals of concern including iron, aluminum, chromium, copper, lead, nickel, cadmium, cobalt and zinc.

Seepage Management

A thorough assessment of the infiltration and seepage conditions of the WRD, HLP, TSF 1, ore stockpiles, and other site facilities has not been well characterized at the current time but will be foundational to understanding the need for site water management. 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 utilized for the infiltration and seepage assessment.

Ongoing ARD/ML Water Treatment

Water treatment for the project will involve active water treatment for ARD/ML. Active water treatment will occur prior to operations, as part of rehabilitation of the site necessary to restart mining, during mining operations, and for a period following cessation of operations. Passive water treatment will be conducted at the site following closure of the active water treatment plant.

Active water treatment at the site has been described in Section 18.5.

Passive water treatment will be conducted in an anaerobic wetland or equivalent passive treatment system. The goals of the passive/semi-passive water treatment at Mt. Todd are to:

- Eliminate or drastically curtain 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.

Passive and semi-passive water treatment systems are generally appropriate for ARD/ML with discharge of between ~ 24 m^3 and ~ 48 m^3/hour, low levels of mineral acidity and sufficient space available to construct a passive or semi-passive treatment system. Passive water treatment system have successfully treated ARD/ML flows ~ ≤ 120 m^3/hour.

Previous studies conducted for the Project indicate that in year 6 of the post-closure phase, an anaerobic wetland or successive alkalinity producing systems (SAPS) should be suitable for passive treatment of ARD/ML.

Estimating flows and water quality 20 years in the future is wrought with uncertainty. These uncertainties 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 early-stage estimates at best and must be checked and updated or entirely modified as the project progresses and more information becomes available.

Surface Water Management

Surface water at the site is well-documented and its management has been the object of study by both Vista and the NT Government in recent years. Surface water management is described further in Section 24.3.

20.4 PERMITTING

On January 1, 2007, Vista became the operator of the Mt. Todd site and accepted the obligation to operate, care for, and maintain the assets of the NT Government on the site. Vista developed an Environmental Management Plan (EMP) for the care and maintenance of the Mt. Todd Mine Site in accordance with the provisions of the Mineral Leases 1070, 1071 and 1127 granted under the Mining Act (Vista, 2007a). The EMP identified the environmental risks found at the Mt. Todd 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 Mt. Todd site. As part of the agreement, the NT Government acknowledged its commitment to rehabilitate the site and that Vista has no obligations for pre-existing conditions until it submits and receives approval of a Mining Management Plan (MMP) for resumption of mining operations.

The Project will require approvals, permits and licenses for various components of the Project. **Table 20-2** includes a list of approvals, permits, and licenses required for the project and their current status.

The Department of Resources (DoR), formerly DRDPIFR, has referred the Project to the Environment, Heritage and the Arts Division of the Department of Natural Resources, Environment, the Arts and Sport (NRETA) for assessment under the NT Environmental Assessment Act. NRETA has required an EIS for the Mt. Todd Project to assist in assessing environmental impacts that are significant either in terms of site-specific issues, off-site issues and conservation values and/or the nature of the proposal.

Vista is currently working with a variety of consultants to develop the EIS (draft EIS planned for submission in mid-2013), and address environmental and permitting tasks for the project. As such, additional costs to conduct permitting activities are not included in the PFS cost model at this time.

Table 20-2: Mt. Todd Permit Status

Approval/ Permit/ License	Current Status	Approval/ Permit License Date	Expiration Date
Environmental Impact Statement	EIS currently being drafted. Following submittal of draft, requires approval by NT EPA using the EIS approval process	Draft EIS submission planned July 2013	NA
Mining Management Act (or Plan) Approval	Notice of Intent submitted, further action on hold pending outcome of EIS	NA	NA
Heritage Act permit to destroy or damage archeological sites and scatters/ Aboriginal Areas Protection Authority Clearances	Authority Certificate Number 2011/15538 issued. This certificate defined Restricted Works Areas and granted select clearances to allow for initial investigations. Additional clearances will be required for further investigations as well as prior to disturbance associated with mine development.	Aboriginal Areas Protection Authority dated July 31, 2012	NA
Dangerous Goods Act (1988) permit for blasting activities	Waiting on final mine plan	NA	NA
Extractive Permit (under DME Guidelines) for development of borrow pits outside of approved mining areas	Would be required for PGM or LPM borrow areas. Permit application not yet in progress pending final selection of borrow areas	NA	NA
Waste Discharge License (under Section 74 of the Water Act 1992) for management of water discharge from the site	WDL 178-2 licensing discharge of waste water into the Edith River from the Mt. Todd mine site, granted with conditions	Feb. 5, 2013	Sept. 30, 2014
Waste water treatment system permits under Public Health Act 1987 and Regulations	May be required for the waste water treatment system for the construction and operations accommodation village. Permit application not yet in progress pending design and siting of accommodation village.	NA	NA
Approval to Disturb Site of Conservation Significance (SOCS)	Batman pit expansion will disturb SOCS as breeding / foraging habitat for the Gouldian finch, pending determination on EIS.	NA	NA

In addition, permits that are required to commence construction works will be obtained prior to any construction activity.

20.5 Social or Community Requirements

The Jawoyn people have strong involvement in the planning for the future of the Mt. Todd Project. Vista has a good relationship with the Jawoyn. Areas of Aboriginal Significance have been designated, and the mine plan has avoided development in these Restricted Works Areas.

20.6 Mine Reclamation and Closure

A PFS reclamation plan for the Mt. Todd Project was developed in support of the PFS for renewed mining operations. This reclamation plan evaluates the reclamation activities that will be conducted for the features planned as part of the restart of mining. Test plots and fills will be installed at the site to test and refine reclamation methods. These test plots and fills will be monitored to evaluate and confirm the performance of alternative grading, storm-water drainage and cover designs, and erosion control and revegetation treatments. Reclamation plans and strategies for each major facility at Mt. Todd are briefly summarized in **Table 20-3**.

Table 20-3: Reclamation Approach

Task	Facility							
	Batman Pit	WRD	HLP	TSF1&2 Impounded Surface	TSF1&2 Dams	Process Plant and Pad	LGO 2	Mine Roads
Surface of Facility at Cessation of Production Composed of Non-PAG Material		X			X			
Final Overall Slopes > 3H:1V*	X	X						
Final Overall Slopes < 3H:1V*				X	X	X	X	X
Benches Created During Construction	X	X			X			
Install 1.0 m-Thick non-PAG Material		X		X				
Install 0.8 m-Thick Store and Release Cover				X	X	X	X	X
Install 0.2 m-Thick Plant Growth Medium (PGM) Cover				X	X	X	X	X
Revegetate with Native Seed Mix				X	X	X	X	X
Install GCL Liner (with under and overlayer of fines)		X						
Install Erosion and Sediment Controls		X		X	X	X	X	X
Construct Access Restriction Bund	X							
Additional Remedial Measures (as necessary)	X	X	X	X	X	X	X	X

* > and < indicates slopes are steeper and less steep, respectively.

20.6.1 Batman Pit

Based on a preliminary conceptual water balance model for the Batman pit (outlined in Section 20.2), no pit lake will result during the post-closure phase, making active dewatering and treatment of pit water unnecessary following closure.

A berm (also termed bund) will be constructed around the entire perimeter of the Batman pit to impede human access and reduce the inflow of surface water to the pit. The safety berm will be offset 30 m from the pit perimeter per the requirements outlined in the guidelines "Safety Bund Walls around Abandoned Open Pit Mines" from the Department of Industry and Resources in Western Australia.

20.6.2 Waste Rock Dump

The existing WRD will be significantly enlarged based on plans for the resumption of mining. The WRD will be constructed at an effective angle of 30° with interbench slopes of 34°. Each lift will be constructed with 8 m wide benches at 30 m vertical intervals on the face of the WRD.

As described in Section 16, the WRD will be constructed with non-PAG material rinds on each lift. Concurrent installation of a GCL cover following attainment of final grades will serve to reduce infiltration of precipitation into the WRD core. This GCL system will include a 0.3 m thick bedding layer of crushed rock consisting of 750 mm particle size material, followed by placement of the GCL, and capped with a 0.3-m thick protecting layer of finely crushed rock placed over the GCL. The GCL will span approximately 52 m on top of each lift, covering the 8 m bench, and running below the subsequent lift. The GCL will be installed at five percent slopes toward the outside of the WRD, and will be constructed with a 0.5-m tall berm with 1:1 side slopes at the interior edge of the GCL layer. At least a 1-m thick layer of non-PAG waste rock will cover all surfaces of the WRD to aid in erosion control.

Prior to WRD grading, a seepage collection system will be constructed along the down-gradient toe of the WRD and subsequently covered with waste rock from grading activities. ARD/ML collected by the WRD seepage collection system will initially be pumped to the New WTP for treatment prior to release until it is feasible to treat this and other ARD/ML on-site using a passive treatment system.

20.6.3 Tailings Disposal Facility

The TSF embankment and impoundment surfaces will be reclaimed at closure by installing and revegetating a 1-m thick store and release cover. The 1-m thick store and release cover will consist of a 0.8-m thick layer of blended non-PAG waste rock (40%) and low-permeability material (60%), overlain by a 0.2-m thick layer of plant growth medium (PGM). Following placement, the cover surface will be roughened and revegetated with native species. The store and release cover will serve to effectively reduce percolation of precipitation into waste rock, PAG, and/or metaliferous materials

The majority of the impounded surface of the TSF at closure will be primarily composed of thixotropic tailings (thick like a solid but flows like a liquid when a sideways force is applied) which will maintain a high degree of saturation for many years unless actively dewatered and consolidated, covered with material, or chemically treated to increase their strength. It is assumed that a 1.0-m thick layer of non-PAG waste rock on the impounded surface of the TSFs will adequately bridge the thixotropic tailings and allow for equipment to place the 1-m thick store and release cover.

To the degree possible, store and release covers will be installed concurrently during construction when portions of facilities reach final grade. Storm water drainage, erosion, and sediment controls will be constructed to minimize erosion and scour of active reclamation areas.

20.6.4 Processing Plant and Pad Area

A new process plant will be built for renewed mining. Once ore processing ceases, the process plant will be decommissioned, decontaminated, demolished and any reusable equipment and materials will be salvaged and resold. Material that cannot be treated in-situ will be excavated and disposed of in the WRD, TSF, or an off-site facility that is certified to accept and dispose of contaminated soil. Concrete foundations, building walls, and other inert demolition waste will be broken up and either:

- Placed in the WRD;
- Buried in-place; and/or
- Backfilled against cut banks and highwalls throughout the process plant and pad area, as well as other areas that will be reclaimed at Mt. Todd.

Surface and large shallow pipes will be removed and pipes at depth will be plugged with concrete or other suitable materials.

The process plant area will be graded to blend into the surrounding topography and drain towards Batman Creek. The process plant area and pad will be covered with a 1-m thick store and release cover, as described previously, and revegetated. Storm water drainage, erosion, and sediment controls will be constructed to minimize erosion.

The WTP and EQP will be left in place, up-graded if necessary, and used to treat ARD/ML during the closure and post-closure phases. These facilities will be closed when it is feasible to treat ARD/ML in passive treatment systems.

20.6.5 Heap Leach Pad and Pond

The HLP and Pond will be left in place and reprocessed following processing of ore and low grade ore. Following reprocessing of the heap material, the pad and pond footprint will be reclaimed by cutting and removing the liner for consolidation in TSF 2. It is anticipated that the integrity of the heap liner will have been compromised and removal of 0.5-m thick of impacted soils below the liner will be necessary. These materials would be removed and consolidated in TSF 2. The area will then be regraded to prevent ponding of water, and will be covered with a 1-m thick store and release cover, as described previously, and revegetated.

20.6.6 Low Grade Ore Stockpile

The existing LGOS1 will be eliminated during the expansion of the BP and it is assumed that no reclamation is required for the closure of this facility.

The LGOS2 will be located near the pit and the process plant area. Closure of LGOS2 will include removal of residual ore from the stockpile areas, regrading, covering the material with a 1-m thick store and release cover, 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.

20.6.7 Mine Roads

Mine access roads will remain in place to provide post-closure access to the area. All haul roads will be closed by grading into surrounding topography, ripping subgrade materials, placing 0.2 m of PGM (when applicable), and revegetating the areas.

20.6.8 Water Storage Ponds

Prior to construction of the active WTP, an EQP will be constructed for mixing of ARD/ML from various on-site sources prior to treatment and to temporarily store ARD/ML in case of system upset. All proposed and existing ponds at Mt. Todd will be maintained for the collection of seepage, storm water and ARD/ML until long-term quality of water collected by the WRD seepage collection system meets applicable standards, flows to the collection system cease, or an alternative passive water treatment system is installed.

Ponds remaining post-closure may be incorporated into the passive water treatment system or used as backup water storage in case treatment upset occurs.

To decommission and close ponds, residual standing water will be pumped to the WTP, and sediments and foundation materials will be tested to determine their chemical characteristics with acidic, PAG and metaliferous materials treated *in-situ* or buried in place. Following sediment testing and removal, pond liners will be cut and folded in place. Pond berms will be pushed into the pond void to cover the liners and until the area no longer impounds water. The top 0.6 m of graded material is assumed to have physical and chemical properties to support plant growth. Storm water drainage, erosion, and sediment controls will be constructed to minimize erosion and channel scour, and the areas will be revegetated.

20.6.9 Low Permeability Borrow Area

A low permeability borrow area will be developed to provide low permeability material for use in project feature construction and for use in reclamation. As portions of the low permeability borrow area are taken out of service and are no longer used to generate material, they will be reclaimed by ripping and amending the remaining soils with organic matter, constructing channels to route drainage within the borrow area footprint and revegetating the area. Some portions of the low permeability borrow area may also be used as stock water ponds.

20.6.10 Closure Cost Estimate

Costs for reclaiming major facilities at the Project were estimated using closure material quantities based on the Base Case and Alternate Case ultimate designs and following the closure plans discussed above. Costs were developed separately for the Base and Alternate Cases, accounting for differences in reclamation scheduling appropriate for each mining scenario. Closure costs are accrued and contained in the financial model.

21.0 CAPITAL AND OPERATING COSTS

Capital and operating costs are summarized in this section and are prepared by Vista's engineers and consultants as follows:

- Open Pit Mining: MDA;
- Process Plant: Proteus;
- Tailings Dam: Tetra Tech;
- Infrastructure: Proteus;
- Raw Water Dam & Water Treatment: Tetra Tech;
- Reclamation: Tetra Tech; and
- Owner's Costs: Vista.

Costs are presented in Q4 2012 US dollars and are based on an US$1.00:AUD1.00 exchange rate.

21.1 CAPITAL COST

LoM capital cost requirements are estimated at US$1,405 million as summarized in **Table 21-1**. Initial capital of US$1,046 million is required to commence operations. At the end of operations, the Project will receive a US$83 million credit for remaining on-site mine mobile equipment, a US$41 million salvage credit for process plant equipment and the return of the US$15 million reclamation bond.

Table 21-1: Capital Cost Summary (US$000s)

Area	Initial Capital	Sustaining	Credit/ Salvage	Total
Capitalized Costs	57,847	134,441	0	192,289
Mine & Process Mobile	138,895	151,460	(82,973)	207,381
CIL Process Plant	409,731	0	(40,651)	369,080
Tailings Dams	19,560	184,092	0	203,652
Power Supply	90,615	0	0	90,615
Water Supply	19,192	0	0	19,192
Owner's Costs	310,544	27,788	(15,000)	323,332
Total Capital	**1,046,383**	**497,782**	**(138,624)**	**1,405,540**

21.1.1 *MINE CAPITAL*

The mine capital cost is estimated based on the quantity of equipment required to achieve the mine production and on the costs for equipment from equipment procurement firms, estimation guides, and recent project data with which MDA has been involved.

Table 21-2 shows the estimated mine capital requirements for the 50,000 tpd case by year. The initial mine capital is estimated to be US$248.2 million US (total of year -1 and year 1). This includes US$52.1 million for pre-stripping and US$5.0 million for mining of waste material used for construction that occurs at the beginning of the mine life.

Sustaining mining capital is estimated to be $77.2 million. Summary information for the capital expenditure estimates is given in the following sections.

Table 21-2: Mine Annual Capital Costs (US$000s) – 50,000 tpd

Primary Mining Equipment	**Pre-Prod**	**Yr 1**	**Yr 2**	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yr 8**	**Yr 9**	**Yr 10**	**Yr 11**	**Yr 12**	**Yr 13**	**Total**
Atlas Copco PV235	$12,579	$6,507	$868	$2,603	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$22,556
165mm Rotory Blast Hole Drill	$ 759	$ -	$ -	$ -	$ -	$190	$569	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,518
28m3 Hyd. Shovel (PC 5000)	$25,059	$ 12,530	$3,132	$9,397	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$50,118
18m3 Front End Loader (994)	$ 1,202	$4,809	$3,607	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 9,618
180t Haul Truck	$63,172	$ 50,357	$400	$ 45,853	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 159,782
Total Primary Equipment	$ 102,771	$ 74,202	$8,007	$ 57,852	$ -	$190	$569	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 243,592
Support Equipment															
300 Kw Dozer (D9)	$ 1,889	$ -	$ -	$ -	$ -	$472	$1,417	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 3,778
230 Kw Dozer (D8)	$ 783	$ -	$ -	$ -	$ -	$196	$587	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,566
4.9 m Motor Grader (16H)	$ 1,671	$ -	$ -	$ -	$ -	$418	$1,253	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 3,342
Water Truck - 70,000 Liter	$ 3,271	$ -	$ -	$ -	$ -	$409	$1,227	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 4,906
RTD Dozer (834H)	$ 2,004	$ -	$ -	$ -	$ -	$501	$1,503	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 4,009
Rock Breaker - Impact Hammer (691 Kg m)	$39	$ -	$ -	$ -	$ -	$ 10	$ 29	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$78
Backhoe/Loader (1.5 cu m - 446D)	$ 286	$ -	$ -	$ -	$ -	$ 72	$215	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 572
2 cm excavator (Cat 392)	$ 397	$ -	$ -	$ -	$ -	$ 99	$298	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 794
Low Boy	$ 994	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 994
Flatbed	$56	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$56
Total Support Equipment	$11,390	$ -	$ -	$ -	$ -	$2,176	$6,529	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$20,095
Blasting															
Skid Loader	$53	$ -	$ -	$ -	$ 53	$ -	$ 13	$ 40	$ -	$ -	$ -	$ -	$ -	$ -	$ 160
Total Blasting	$53	$ -	$ -	$ -	$ 53	$ -	$ 13	$ 40	$ -	$ -	$ -	$ -	$ -	$ -	$ 160
Mine Maintenance															
Lube/Fuel Truck	$ 428	$ -	$ -	$ -	$ -	$ 54	$161	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 643
Mechanics Truck	$ 187	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 187
Tire Truck	$ 137	$ -	$ -	$ -	$137	$ 34	$103	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 411
Total Mine Maintenance	$ 752	$ -	$ -	$ -	$137	$ 88	$263	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,241
Other Mine Capital															
Light Plant	$51	$ 22	$ 7	$ 22	$ 7	$ 22	$ 7	$ 22	$ -	$ -	$ -	$ -	$ -	$ -	$ 161
Mobile Radios	$60	$ 17	$ 7	$ 28	$ 1	$ 7	$ 21	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 141
Shop Equipment	$ 491	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 491
Engineering & Office Equipment	$ 200	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 200
Water Storage (Dust Suppression)	$98	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$98
Base Radio & GPS Stations	$ 105	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 105
Unspecified Miscellaneous Equipment	$ 150	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 150
Access Roads - Haul Roads - Site Prep	$ 175	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 175
Light Vehicles	$ 586	$ -	$ -	$656	$ -	$ -	$493	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,735
Total Other Mine Capital	$ 1,916	$ 39	$ 14	$706	$ 8	$ 29	$521	$ 22	$ -	$ -	$ -	$ -	$ -	$ -	$ 3,256
Capitalized Mine Operating Costs															
Pre - Stripping Mining Cost	$52,127	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$52,127
Tailings Construction Costs	$ 4,984	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 4,984
Total Capitalized Mining Costs	$57,111	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$57,111
Capital Summary															
Primary Mining Equipment	$ 102,771	$ 74,202	$8,007	$ 57,852	$ -	$190	$569	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 243,592
Support Equipment	$11,390	$ -	$ -	$ -	$ -	$2,176	$6,529	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$20,095
Blasting	$53	$ -	$ -	$ -	$ 53	$ -	$ 13	$ 40	$ -	$ -	$ -	$ -	$ -	$ -	$ 160
Mine Maintenance	$ 752	$ -	$ -	$ -	$137	$ 88	$263	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,241
Other Mine Capital	$ 1,916	$ 39	$ 14	$706	$ 8	$ 29	$521	$ 22	$ -	$ -	$ -	$ -	$ -	$ -	$ 3,256
Capitalized Mine Operating Costs	$57,111	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$57,111
Total - All Mining Capital	$ 173,994	$ 74,241	$8,021	$ 58,558	$199	$2,483	$7,896	$ 62	$ -	$ -	$ -	$ -	$ -	$ -	$ 325,454

Major Mining Equipment

Capital for major mining equipment for 50,000 tpd is shown in **Table 21-2** and discussed in the following subsections.

Drilling and Blasting

Primary drilling equipment capital is based on equipment quotations for a total of eleven Atlas Copco Pit Viper 235 blast-hole drills required through the first year of production and a replacement of two drills later in the mine life. Seven of the drills will be purchased at the start of mining in year -1, an additional four drills purchased in year 1, and then two drills will be purchased in year 3 at a cost of US$1.7 million each (including shipping and assembly). The cost of the drills was provided by EMG LLC.

In addition to the production drills, smaller 45K pull-down drills will be used for pre-split drilling. These will use 165mm bits and will cost approximately US$759,000 each. One drill is purchased in year -1 and a replacement drill has been planned for year 6.

Quotes for explosives trucks, powder magazine storage, and bulk ANFO storage has been obtained by Proteus. These capital costs are included in the infrastructure costs. Additional capital expense for a skid loader is provided to be used by the blasting crew for stemming holes. The skid loader would be purchased at an estimated cost of US$53,000 during pre-production and then two additional units would be purchased in year four and six.

Loading

Capital costs for loading equipment have been quoted by EMG LLC and include four Hitachi Ex5500 hydraulic shovels and two Caterpillar 994 loaders. Two of the hydraulic shovels would be purchased during pre-production with a second being purchased during year one. The fourth shovel is purchased in year three. The estimated cost for each shovel is US$12.5 million which includes freight and assembly.

The cost of the 18-m^3 loaders is based on a quote for a Caterpillar 994 loader with the first one being purchased at the start of production in year one and the second purchased in year two at a cost of $4.5 million each.

Haulage

The 226-t haulage truck costs are based on CAT 793F trucks and were quoted by EMG LLC. Fifteen trucks are purchased during pre-production, with another 12 trucks purchased in year one. Trucks are purchased as they are required through the mine life. The trucks are staged in to allow ramp up of production through each year as they are needed to meet production requirements. The total number of trucks required by year is shown in **Table 21-3**.

Table 21-3: Trucks Required by Year

Year	Number of Trucks Added
-1	15
1	12
3	11

Throughout the mine life, a total of 38 trucks are purchased. The cost of each truck is estimated at US$4.2 million including freight and assembly.

Mine Support

Capital estimates for mine support equipment include freight and erection. The initial support equipment to be purchased in year -1 as follows:

- Two Caterpillar D9 track dozers (US$945,000 each quoted by EMG LLC);
- One Caterpillar D8 track dozer (US$783,000 each quoted by EMG LLC);
- Two Caterpillar 16H motor graders (US$835,000 quoted by EMG LLC);
- Two Caterpillar 777 truck with a 70K liter water truck (US$1.6 million quoted by EMG LLC);
- Two Caterpillar 834H rubber tire dozer (US$1.0 million quoted by EMG LLC);
- One Caterpillar 392DL excavator (US$397,000 quoted by EMG LLC);
- One low-boy trailer complete with a used 60t haul truck to tow it (US$993,000);
- One flatbed truck (US$56,000);
- One rock breaker to be attached to the 321DL excavator as needed (US$39,000); and
- Five light plants (US$15,000).

Replacements are purchased for most units in year 6.

Maintenance

Capital for mine maintenance equipment includes three fuel/lube trucks (US$214,000 each) one mechanic's truck (US$187,000 each), and three tire tucks (US$137,000). Note that requirements for mechanic's trucks are reduced due to the assumption of MARC for maintenance. The mechanic's truck is intended for support of a small number of owner operated equipment.

An additional US$491,000 has been included for shop equipment / tooling. Shop facilities were estimated by Proteus and included in facility capital.

Mine Facilities

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

Light Vehicles

Initial capital for light vehicles is estimated to be US$586,000 while sustaining light vehicle capital is US$1.1 million. Initial and sustaining light vehicle capital is shown in **Table 21-4**.

Table 21-4: Mine Light Vehicle Initial and Sustaining Capital

		Initial Capital			Sustaining Capital		
Mine Department	**Type**	**Quantity**	**Unit Cost**	**Ext. Cost**	**Quantity**	**Unit Cost**	**Ext. Cost**
Mine Superintendent	3/4 ton 4wd Pickup	1	$35,000	$35,000	2	$35,000	$ 70,000
Shift Foreman	4wd Pickup	2	$35,000	$70,000	4	$35,000	$ 140,000
Trainer	4wd Pickup	1	$29,000	$29,000	0	$29,000	$ -
Blasting	4wd Pickup	1	$33,000	$33,000	2	$33,000	$ 66,000
Blasting	1 ton 4wd Pickup	1	$29,000	$29,000	2	$29,000	$ 58,000
Crew Vans	3/4 ton Passenger Van	2	$35,000	$70,000	4	$35,000	$ 140,000
Engineering							
Chief Engineer	4wd Pickup	1	$35,000	$35,000	2	$35,000	$ 70,000
Short Range Planning	4wd Pickup	1	$29,000	$29,000	2	$29,000	$ 58,000
Survey	4wd Pickup	1	$35,000	$35,000	3	$35,000	$ 105,000
Geology							
Chief Geologist	4wd Pickup	1	$35,000	$35,000	3	$35,000	$ 105,000
Ore Control	4wd Pickup	1	$29,000	$29,000	3	$29,000	$ 87,000
Samplers	4wd Pickup	1	$29,000	$29,000	2	$29,000	$ 58,000
Mine Maintenance							
Maintenance Superintendent	4wd Pickup	2	$35,000	$70,000	3	$35,000	$ 105,000
Mechanics / Labor	4wd Pickup	2	$29,000	$58,000	3	$29,000	$ 87,000
Total		18		$586,000	35		$1,149,000

Other Capital

Other miscellaneous capital includes mobile radios for mobile equipment (US$1,000 per unit), engineering and office equipment (US$200,000), water storage for dust suppression (US$98,000), GPS stations and surveying equipment (US$105,000), and other unspecified miscellaneous equipment (US$150,000). An additional capital cost of US$175,000 has been included for roads and site preparation around the open pit and dumping areas.

21.1.2 CIL Process and Infrastructure

The process and infrastructure capital cost estimate (CCEs) are based on the Proteus scope of work (SoW) developed during the PFS phase of the Project. This section discusses the CCE summary results and describes the approach taken to prepare the key components of the estimates.

Proteus' PFS CCEs are based on an enhanced factored cost estimate (EFCE) methodology, which features higher confidence levels around contingency provision and management reserve. The capital estimates are supported by the design work carried out throughout the study including process documentation, schematics, general arrangement drawings, 3D models and calculations.

Process & Infrastructure Capital

The CCE is summarized in **Table 21-5**.

Table 21-5: Capital Cost Summary

Capital Cost	Initial Capital (US$ million)
Facility 1000 – Geology	0.00
Facility 2000 – Mine Infrastructure	15.966
Facility 3000 – Process Plant	409.110
Facility 4000 – Project Services	48.091
Facility 5000 – Project Infrastructure	38.726
Facility 6000 – Permanent Accommodation	0.086
Facility 7000 – Site Establishment & Early Works	27.548
Facility 8000 – Management, Engineering, EPCM Services	92.736
Facility 9000 – Preproduction Costs	14.015
Subtotal Direct Costs	**511.969**
Subtotal Indirect Costs	**134.299**
Contingency Provision (14.6%)	**94.367**
Total Expected Cost	**740.634**

The total capital cost, base cost plus contingency provision, represents the expected cost for the project, with approximately a 55% confidence level of completion within cost. This estimate has an accuracy range of approximately -0 to 15% based on the expected cost. At the upper limit of the accuracy range, there is an 85% confidence level of completion within cost.

Typically, the EPCM Project Manager would initially receive Owner's approval for expenditure up to the expected cost (i.e., this is the initial project budget).

Exclusions

The Proteus Scope of Work is a significant part of the overall Project scope, although other parties have compiled capital costs for other areas on behalf of Vista. The potential impacts of possible price or labor rate fluctuations or currency exchange rate fluctuations are the role of a qualified actuary and should be covered by Vista in its standard business practices.

Capital Cost Estimating Methodology

A PFS design has been developed for a 50,000 tpd plant as the basis for an EFCE. The EFCE approach uses a combination of bottom-up calculations and factoring methods for each area in the estimate. The methods used to estimate capital in the CCE are summarized in the following sections.

Enhanced Factored Estimate Approach

The EFCE for the process plant features the methodology shown in the table below.

Table 21-6: CCE Methodology for Facility 3000 – Process Plant

Bulk Commodity	Base Case
Mechanical Equipment	A detailed mechanical equipment list, with supply and installation pricing based on budget quotations and internal body of knowledge
Concrete	MTO's based on 3D model and unit rates and factored up from 33,000 tpd option and unit rates
Structural Steel	MTO's based on 3D model and unit rates and factored up from 33,000 tpd option and unit rates
Platework	MTO's from previous projects a unit rates
Tankage	MTO's based on preliminary design calculations and unit rates
Piping	Percentage factor of the supplied mechanical equipment supply price, assessed on an area by area basis.
Electrical	Percentage factor based on total mechanical equipment supply price
Instrumentation and Control	Costs factored up from 33,000 tpd case

Subsequently estimate factors, by area, were back calculated for each bulk commodity as a percentage of the mechanical equipment supply cost estimate. In turn, the resultant estimate factors were critiqued against published data and industry experience.

Other Area Capital Cost Estimates

The EFCE for Facility 2000 - Mining features:

- Area 2300 is a series of facilities which were priced largely on a building per square meter basis using all inclusive rates.
- Area 2400 was priced using preliminary quantities and unit rates.

The EFCE for Facility 4000 features:

- Sub-Area 4110 – Watery Supply-WTP and Area 4400 – Tailings Dam were estimated by Tetra Tech's Golden Office.
- Sub-Area 4120 – Raw Water Distribution was estimated by an estimate of mechanical equipment costs and factoring of bulk commodities.
- Area 4600 - Plant Mobile Equipment was estimated by vendor pricing of the proposed fleet for plant operation.
- The balance of facility 4000 was estimated by preliminary quantities and unit rates of construction.

The EFCE for Facility 5000 features:

- Area 5100 – Site Preparation was estimated based on material take-offs from preliminary drawings and rates developed from first principles.
- Area 5200 – Support Buildings was estimated based on building sizes and square meter rates for appropriate buildings.
- Area 5300 – Access Roads, Parking and Laydown was estimated based on provisional sums based on miscellaneous road and culvert repairs.

The EFCE for facility 6000 features:

- Area 6100 – Personnel transport was priced based on unit rates for bus shelters with an allowance for the small amount of concrete required.

The EFCE for Facility 7000 features:

- Area 7300 – Construction camp was priced based on material take-offs for the access roads and site works and vendor quotes for the camp and operation.

The EFCE for Facility 8000 features:

- Area 8100 – EPCM services was based on a bottom up estimate using hours and Proteus rates.
- The balance of facility 8000 was based on a factor of mechanical equipment costs for commissioning and total direct costs for all other areas.

The EFCE for facility 9000 features:

- Area 9300 – Capital spares and area 9400 – Stores and Inventories were priced based on a factor of the total mechanical equipment costs.
- The balance of facility 9000 was priced based on a factor of the total direct costs.
- Area 9800 – Contingency and area 9900.

Construction Labor Rates

The calculation is based upon current ordinary time wages for various classes of labor including direct supervision, to which the following factor may apply; site allowance, tool allowance, leave provisions, taxes and insurances, overtime etc. This develops a gang rate that is combined with costs of incumbent support equipment (such as light vehicles, light mobile cranes, small tools, consumables, first-aid facilities and accommodation) and management support to arrive at an all-purpose site gang rate.

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

- Concrete;
- Structural, Mechanical and Piping (SMP); and
- Electrical and Instrumentation (E&I).

Base Labor Rates

The base labor rate includes the direct labor allocated for the installation of equipment and bulk commodities. Base pay rates were derived from award rates for similarly sized projects currently underway in the North West of Western Australia and in the Northern Territory. These are considered to be the benchmark for the area, including Mt. Todd. Allowances were made for overtime loadings above a 36 hour week including time and a half for the initial twelve hours overtime, followed by double time for the final seventeen hours overtime, to provide for a sixty five hour working week. The rates were averaged over a standard mix of trades, to produce a composite rate per man per hour.

The base labor rates were developed to include items listed below:

- All direct payments including the site allowances and special project allowances for straight time and overtime worked for personnel;
- Overtime at penalty rates;
- Provision for holiday leave and loadings thereon;
- Provision for sick leave;
- Provision for cost of travel time to site and return travel on job completion;

- Provision for additional manpower turnover, bereavement leave and miscellaneous paid non-work days;
- Payroll tax;
- Workers compensation insurance;
- Superannuation considerations; and
- Industry redundancy payments.

An R&R burden was also added to the composite rate, together with a 15% contractors allowance for overheads and margin to produce the base labor rate for each contractor type.

Contractor Indirect Rates

The contractor indirect rate is a combination of costs associated with indirect contractor personnel, contractor vehicles, contractor overheads and construction plant equipment. An estimate of construction contract duration and installation hours was based on the EPCM schedule and bulk quantity development.

The contractor indirect rates were developed to include the items listed below:

- Project Management personnel;
- Construction Supervision personnel;
- Site Quality Assurance and Control personnel;
- Site Health, Safety, Environmental personnel;
- Other indirect labour (stores officer, surveyor etc.);
- Contractor vehicles for the Project Management team;
- Office accommodation;
- Workshop and stores facilities;
- Staff travel including airfares ;
- Office overheads; and
- Vehicle consumables.

Provisions for the accommodation and messing are also not included in the indirect contractor rates. This is allowed for in the construction camp cost estimate to supply and operate the camp.

Although they are considered indirect costs, construction plant equipment rates are estimated separately to include the following:

- Construction plant equipment mobilization / demobilization;
- Construction plant management support; and
- Construction plant and equipment.

The provision for task specific heavy lift cranes >50 tonnes were not included in the indirect contractor rates build-up; instead it was allowed for in a separable line item in the chart of accounts (CoA).

Construction Gang Rates

The overall site construction gang rates were developed by summing the base labor rate, contractor indirect rates and construction plant rates to provide an overall site construction gang rate for Concrete, SMP and E&I contractors as shown in **Table 21-7** below.

Table 21-7: Construction Gang Rate Development

Contractor	Base Labor Rate (AUD/hr)	Contractor Indirect Rate (AUD/hr)	Construction Plant Rate (AUD/hr)	Construction Gang Rate (AUD/hr)
Concrete	115.28	31.48	21.14	167.90
SMP	123.99	39.58	14.68	178.25
E&I	131.28	36.99	16.83	185.10

Mechanical Equipment

The supply costs comprise of the direct mechanical equipment cost plus the cost for freight to site. Installation costs are estimated based on an evaluation of installation hours multiplied by the SMP contractor gang rate. These estimating methods are discussed in the following sections.

Equipment Costs

The basis for estimating the mechanical equipment supply costs was largely based on budgetary pricing from vendors. The vendors were provided with preliminary specifications and/or data sheets for major equipment items. The budget quotations received from vendors are expected to have an accuracy equal to ±10%.

All other minor equipment items were priced from a Proteus' database of costs from recent similar sized projects. The basis of the supply cost estimate for each mechanical equipment line item is documented in the process plant CCE.

Freight Costs

Several methods were used to determine and validate the allowance for delivery costs of mechanical equipment to site. These methods included:

- Quotes provided by the manufacturer;
- Estimates based on the weight and volume of the load;
- Estimates based on published and in-house guides for similar installations; and
- Estimates based on a validated percentage of the mechanical equipment cost (determined to be 9% of the supply price).

Installation Hours

Several methods were used to determine and validate the installation hour allowance for mechanical equipment. These methods included:

- Quotes provided by the manufacturer;
- Estimates based on the weight of the equipment; and
- Estimates based on published and in-house guides for similar installations.

The installation hour estimates for large process equipment (>3000 man hours/ equipment) including the crushers, HPGRs, ball mills and thickener were reviewed in detail against historical records and published guidelines.

Quantity Development and Unit Rates

The basis for the development of supply and installation costs of bulk commodities is discussed in the following section. Bulk commodities include civil, concrete, structural steel etc., which will be used in the construction of the process plant. These costs were largely derived based on an estimate of material take-off

(MTO) quantities which were multiplied by a unit rate for each type of material. The unit rates were calculated using Proteus standard templates and comprise of allowances for supply of the raw material, fabrication, freight and erection.

Civil

Preliminary bulk earthwork quantities were estimated using civil 3D modeling software 12D Model. The 12D Model accurately calculates earthworks volumes utilizing the existing topography and proposed design levels. Structural excavation and backfill required for concrete structures are included in the concrete quantities. Trenching requirements for underground utilities distribution were determined from service plans. Storm-water drainage quantities were determined from the civil site plan with V-drains alongside plant roads directing surface run-off beneath roads via corrugated steel culverts. All quantities were categorized by standard type of work classification.

Unit rates for this work classification were developed from the in-house rates database. This rates database is constantly maintained so as to be current and has proven to be sufficiently accurate over several recent projects. The availability of water and local earthworks materials was taken into account in to the development of unit rates.

Concrete

Concrete quantities for foundations and ground slabs for all equipment and structures in the process plant were calculated by 3D models. Concrete quantities were categorized by standard classes of concrete including spread/pad footings, strip footings, raft footings, ring beams, ground slabs, walls, sumps and pits etc.

Unit pricing was obtained from industry sources by standard classification, each having an assessment of formwork, props, bracing reinforcing, embedment's, joints in slabs plus a miscellaneous allowance for curing, formwork hardware and other sundries. Concrete supply was costed at a rate deemed to include plant control testing, some admixtures and out of hours pouring. A wastage factor was included in the rates. A Contractor's mark-up and freight allowance was also applied to all materials. Direct labor unit man-hours were be sought from industry sources and checked against historical data and various published references.

Structural Steel

Quantities of steel required for the process plant structures were quantified using the 3D models developed by the structural drafters. Steel quantities were categorized by standard classes of steel including light, medium, heavy and very heavy. There are also provisions made for grating, hand-railing and stair-treads.

Unit rates for the supply and installation of structural steel were calculated using Proteus Standard templates. Supply of steel was based on rates quoted from Thai fabricators from a similar project. The supply rate includes provisions for steel supply, shop drawings, shop fabrication, painting and freight to site. Estimates for the installation costs of structural steel are based on estimates of erection hours and the SMP gang rate.

Platework

Quantities of steel required for custom designed platework was calculated by mechanical engineers with reference to specification sheets, outline sketches and MTO sheets from similar projects. The cost items for platework includes plate thicknesses of <10mm, 12-20mm and floor plate of 6mm and also allowances for Bisalloy or rubber lining where applicable.

Unit rates for platework were derived from quotes from Thailand fabricators as is described for structural steel.

Tankage

Quantities of steel required for custom designed tankage were quantified by structural engineers using Proteus standard spread sheets to determine the required tank shell and base thickness in accordance with the provisions of API 650. This includes allowances for the mass of steel for shell plates, top rings, and base plates.

Unit rates for supply and installation of tankage were based on Thai fabricated steel using the same methodology as is described for structural steel. There are two classes of tankage allowed for in the CCE including shop fabricated and site erected tanks (assumed to be greater than 7m in diameter).

Piping

The estimate for the supply and installation of process piping was factored based on a percentage of the supplied mechanical equipment price, assessed on an area by area basis. These percentages were based on in-house and industry typical piping allowances for similar gold plants. These factors were validated with values reported in published guidelines.

Electrical

The estimate for the supply and installation of electrical components for the process plant was factored based on a percentage of the total mechanical equipment supply and installation cost. These percentages were based on in-house and industry typical electrical allowances for similar gold plants.

Instrumental and Control

The estimate for the supply and installation of the instrumentation and process control system was initially estimated for a 33,000 tpd process plant based on preliminary piping and instrumentation diagrams (P&IDs) and equipment lists and on a highly automated gold plant with all field instruments marshaled to remote I/O cabinets. This estimate was then up-scaled for 50,000 tpd based on an expected 30% increase in equipment, I/O, programming and instrumentation.

Indirect Costs

Construction Camp

An estimate of the construction facilities was developed from previous project experience for the various scopes of work. This includes a breakdown of costs for contractor preliminaries, transportable building (supply and install) and establishing the infrastructure, power supply, communications and water supply. Provisions were also made for the operation of the camp based on a man-day rate. It also includes allowances for removal of the infrastructure following completion.

EPCM Services

Engineering, drafting and documentation functions are task and deliverable related. Hence, their estimates were based on task and deliverable identification with time estimates based in industry experience. Procurement activities were estimated from hours related to purchasing, expediting, inspection and transport functions derived by time involvements, and then checked against industry experience. Management, administrative and project engineering functions are mostly time-related and were assessed by title, rate and man-months of key personnel and other staff proposed.

To the extent possible, site office items were detailed and estimated on an item-by-item basis. Management, supervisory and administrative staffing were estimated on an hours basis.

External Consultants and Testing

Cost allowances for Environmental, Human Resources and Industrial Relations, and Health and Safety consultants are based on industry experience of required manning and market contract values.

Other Indirect Costs

The following costs were calculated based on industry validated percentages of the total direct costs of the Project:

- Owners engineering / management;
- License, fees and legal costs;
- Project insurances; and
- Pre-production labor.

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

- Commissioning Expenses;
- Capital Spares; and
- Stores and Inventories.

Contingency Provision

The contingency provision is an allowance added to an estimate to provide for costs which cannot be estimated due to inadequate information, but which are known to be implicit in the scope.

The contingency provision represents costs which are expected to be incurred to complete the project and must be regarded as part of the total funds placed under the direct control of the project manager.

The contingency provision includes an allowance for:

- Unidentified items not included in the quantity calculations or equipment lists, due to lack of knowledge, but implicit in the scope.
- Small changes, arising from detailed design, which normally occur during the course of the project, as knowledge becomes firmer.
- Design Omissions.

Changes in concept, scope or production rates which depart from those on which the estimate has been based require a new estimate. These changes are not allowed for in the contingency.

21.1.3 Electric Power Plant (Area 4200)

Capital costs compiled for this study cover direct and indirect costs of power station construction including equipment quoted from suppliers, material quantities estimated from the preliminary design, and installation labor and supervision with local material and labor costs applied to those estimates. Power plant capital costs are shown in **Table 21-8**.

Table 21-8: 76MW Installed Capital Cost Summary

Cost Component	57MW Gas Turbine	2x9.7 MW Recip Engines	Total
Generator Set	25,506,388	14,087,216	**39,593,604**
Balance of Plant Equipment	9,073,179	867,600	**9,940,779**

Mechanical, Civil, & Electrical Direct Costs	8,361,337	6,293,022	**14,654,359**
Engineering Fees	4,000,000	821,578	**4,821,578**
Contractor's Fees	2,592,346	1,064,906	**3,657,252**
Freight, Taxes, and Other Indirect Costs	8,833,803	1,985,672	**10,819,475**
Total Installed Costs	58,367,053	25,119,994	**83,487,047**
Capital Cost per Net Installed Capacity, $AUS/kW	**1,023**	**1,295**	**1,092**

21.2 Operating Costs

LoM operating costs requirements are estimated to be US$16.32/t-milled as summarized in **Table 21-9**.

Table 21-9: LoM Operating Costs

Opex Summary	US$/t-mined	US$/t-milled	Total (US$000s)
Open Pit Mining	2.005	6.946	1,547,607
CIL Process Plant	-	8.780	1,956,195
Water Treatment Plant	-	0.096	21,313
G&A	-	0.495	110,310
Total Opex Summary	**-**	**16.317**	**3,635,425**

21.2.1 Open Pit Mine

Annual mine operating costs have been estimated based on personnel requirements and equipment hourly costs. **Table 21-10** summarizes annual mine operating costs after the allocation of capitalized mining. Costs are provided based on functionality (drilling, blasting, loading, hauling, support, mine general services, mine maintenance, engineering, and geology).

Table 21-10: Annual Mine Operating Costs – 50,000 tpd

Base Case Mining Costs	Units	Pre-Prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Total
Mine General Service	K USD	$-	$2,088	$2,082	$2,083	$2,082	$2,088	$2,081	$2,080	$1,887	$2,085	$1,937	$1,314	$583	$-	$22,389
Mine Maintenance	K USD	$-	$2,470	$2,464	$2,464	$2,464	$2,471	$2,462	$2,461	$2,232	$2,467	$2,292	$2,465	$2,465	$-	$29,177
Engineering	K USD	$-	$1,343	$1,339	$1,339	$1,339	$1,343	$1,338	$1,338	$1,213	$1,341	$1,246	$1,071	$551	$-	$14,801
Geology	K USD	$-	$831	$829	$829	$829	$832	$829	$828	$751	$830	$772	$717	$610	$-	$9,488
Drilling	K USD	$-	$15,933	$18,785	$25,005	$21,875	$23,035	$19,553	$14,709	$11,953	$12,747	$13,497	$6,743	$-	$-	$183,836
Blasting	K USD	$-	$10,853	$11,925	$15,222	$13,293	$13,941	$11,082	$8,533	$7,215	$7,961	$8,835	$5,546	$-	$-	$114,406
Loading	K USD	$-	$12,444	$17,027	$21,776	$19,560	$17,759	$20,658	$16,945	$14,060	$13,169	$10,994	$6,292	$5,164	$-	$175,849
Hauling	K USD	$-	$48,063	$66,672	$87,289	$96,690	$96,571	$96,690	$95,579	$87,882	$92,936	$88,190	$40,209	$5,449	$-	$902,219
Mine Support	K USD	$-	$6,701	$6,774	$6,842	$6,842	$6,953	$7,118	$7,116	$6,455	$7,133	$6,628	$7,126	$4,184	$-	$79,871
Total Mine Cost	K USD	$-	$100,725	$127,898	$162,850	$164,976	$164,992	$161,811	$149,588	$133,648	$140,668	$134,390	$71,482	$19,005	$-	$1,532,034
Mine Cost per Tonne Mined																
Mine General Service	$/t	$-	$0.03	$0.03	$0.02	$0.02	$0.03	$0.03	$0.04	$0.04	$0.05	$0.04	$0.06	$-	$-	$0.03
Mine Maintenance	$/t	$-	$0.04	$0.03	$0.02	$0.03	$0.04	$0.03	$0.05	$0.05	$0.06	$0.05	$0.11	$-	$-	$0.04
Engineering	$/t	$-	$0.02	$0.02	$0.01	$0.01	$0.02	$0.02	$0.03	$0.03	$0.03	$0.03	$0.05	$-	$-	$0.02
Geology	$/t	$-	$0.01	$0.01	$0.01	$0.01	$0.01	$0.01	$0.02	$0.02	$0.02	$0.02	$0.03	$-	$-	$0.01
Drilling	$/t	$-	$0.26	$0.25	$0.25	$0.24	$0.35	$0.25	$0.29	$0.25	$0.31	$0.30	$0.31	$-	$-	$0.26
Blasting	$/t	$-	$0.18	$0.16	$0.15	$0.15	$0.21	$0.14	$0.17	$0.15	$0.19	$0.20	$0.26	$-	$-	$0.16
Loading	$/t	$-	$0.20	$0.22	$0.21	$0.22	$0.27	$0.26	$0.33	$0.30	$0.32	$0.25	$0.29	$-	$-	$0.25
Hauling	$/t	$-	$0.78	$0.88	$0.86	$1.08	$1.47	$1.23	$1.88	$1.85	$2.24	$1.97	$1.86	$-	$-	$1.29
Mine Support	$/t	$-	$0.11	$0.09	$0.07	$0.08	$0.11	$0.09	$0.14	$0.14	$0.17	$0.15	$0.33	$-	$-	$0.11
Total Mine Cost	$/t	$-	$1.63	$1.69	$1.60	$1.84	$2.50	$2.05	$2.94	$2.81	$3.40	$3.00	$3.30	$-	$-	$2.19
Mine Cost per Tonne Moved																
Mine General Service	$/t	$-	$0.03	$0.03	$0.02	$0.02	$0.03	$0.02	$0.03	$0.03	$0.04	$0.04	$0.06	$0.04	$-	$0.03
Mine Maintenance	$/t	$-	$0.04	$0.03	$0.02	$0.03	$0.04	$0.03	$0.04	$0.04	$0.05	$0.05	$0.11	$0.18	$-	$0.04
Engineering	$/t	$-	$0.02	$0.02	$0.01	$0.01	$0.02	$0.01	$0.02	$0.02	$0.03	$0.03	$0.05	$0.04	$-	$0.02
Geology	$/t	$-	$0.01	$0.01	$0.01	$0.01	$0.01	$0.01	$0.01	$0.01	$0.02	$0.02	$0.03	$0.04	$-	$0.01
Drilling	$/t	$-	$0.26	$0.23	$0.24	$0.23	$0.35	$0.22	$0.23	$0.21	$0.26	$0.30	$0.31	$-	$-	$0.24
Blasting	$/t	$-	$0.17	$0.15	$0.15	$0.14	$0.21	$0.12	$0.13	$0.12	$0.16	$0.20	$0.26	$-	$-	$0.15
Loading	$/t	$-	$0.20	$0.21	$0.21	$0.20	$0.27	$0.23	$0.27	$0.24	$0.27	$0.25	$0.29	$0.37	$-	$0.23
Hauling	$/t	$-	$0.77	$0.82	$0.85	$1.01	$1.47	$1.08	$1.50	$1.51	$1.92	$1.97	$1.86	$0.39	$-	$1.18
Mine Support	$/t	$-	$0.11	$0.08	$0.07	$0.07	$0.11	$0.08	$0.11	$0.11	$0.15	$0.15	$0.33	$0.30	$-	$0.10
Total Mine Cost	$/t	$-	$1.62	$1.58	$1.58	$1.73	$2.50	$1.81	$2.34	$2.29	$2.90	$3.00	$3.30	$1.35	$-	$2.00

The following subsections describe the operating cost estimate by function.

The total average mining cost for the Base Case is estimated to be US$2.19/t mined after allocations for pre-stripping and tailings construction. The overall mining cost per tonne before allocations is US$2.10/t mined. The Base Case mining costs are shown in **Table 21-10** and further described in the following sections.

Drilling

The average LoM drilling cost is estimated to be US$0.26/t mined after allocation of drilling costs for pre-stripping and tailings construction. This includes maintenance allocations based on maintenance and repair contract assumptions.

Blasting

The average LoM blasting cost is estimated to be US$0.16/t mined after allocation of blasting costs for pre-stripping and tailings construction.

Loading

The average LoM loading cost is estimated to be US$0.25/t mined or US$0.23/t moved after allocation of costs for pre-stripping and tailings construction. The cost per tonne moved includes the re-handle of ore and waste from stockpiles at the end of the mine life. Maintenance costs assume the use of a maintenance and repair contract based on costs provided by EMG LLC.

Haulage

The average LoM haulage cost is estimated to be US$1.29/t mined or US$1.18/t moved after allocation of haulage costs for pre-stripping and tailings construction. The cost per tonne moved includes re-handle of stockpiled ore and waste at the end of the mine life. Maintenance costs assume the use of a maintenance and repair contract based on costs provided by EMG LLC.

Mine Support Costs

Mine-support costs include the operation of all of the mine-support equipment. The average LoM support cost is estimated to be US$0.11/t mined or US$0.10/t moved after allocation of support costs for pre-stripping and tailings construction. The cost per tonne moved includes support during re-handling of stockpiled ore and waste at the end of the mine life. Maintenance costs assume the use of a maintenance and repair contract based on costs provided by EMG LLC.

Mine Maintenance

Most maintenance will be done under a maintenance and repair contract. The vendor with the contract will be expected to supply mechanics and maintenance parts for major equipment repair. Costs associated with the contract have been included in the equipment hourly cost.

Owner mine-maintenance costs have been included to cover items not covered by the maintenance and repair contract as well as supervision and oversight of the contract. This includes salaries for a Maintenance Superintendent and Maintenance Planner to track costs associated with the contract. Tiremen will be hired by the owner to maintain all equipment tires, and servicemen will be hired to keep equipment fueled and lubricated. An allocation for shop laborers has been included for light maintenance of facilities.

The average LoM mine-maintenance cost is estimated to be US$0.04/t mined or US$0.04/t moved after allocation of support costs for pre-stripping and tailings construction.

Mine General Services, Engineering, and Geology

Mine general costs include salaries for a mine manager, mine clerk, shift foremen, and trainers. Mine general costs also include an allocation for various supplies and office costs.

Engineering and geology services are provided to maintain surveying, mine planning, and ore control for the operations. The average LoM general services, engineering, and geology costs are estimated to be US$0.06/t after allocation of support costs for pre-stripping and tailings construction.

21.2.2 PROCESS PLANT AND G&A OPERATING

Overall the approach taken for the PFS operating cost estimate establishment was to perform the estimates at a feasibility study (FS) level of detail, leading to a higher than usual level of detail presented for the PFS. This approach was deliberated adopted to minimize rework during the FS stage, with additional information expected to be limited to the use of improved accuracy quotes for the FS cost estimate.

Final plant operating cost estimates issued for the PFS were US$165 million per year, giving a cost of US$9.31/t treated as shown in **Table 21-11.**

Cost Distribution

The distribution of operating costs was not unexpected for large scale gold operations, with the four main operating cost expenditures in descending order being:

- Reagents and Consumables;
- Power;
- Labor;
- Maintenance; and
- G&A.

Items of expenditure higher than normally expected for gold mining operations related specifically to:

- the Mt. Todd ore hardness, and included consumables (mill balls) and power consumption; and
- the high volume / low grade proposed Mt. Todd ore treatment schedule and related predominantly to reagents.

Table 21-11: Plant Operating Costs (@ Steady State)

Cost Center	US$/a	US$/t	US$/oz	%
Labor				
Total	**21,270,000**	**1.20**	**53.65**	**12.9%**
Transport and Accommodation				
Total	**2,010,000**	**0.11**	**5.07**	**1.2%**
Power				
Processing Plant	42,510,000	2.39		
Miscellaneous	330,000	0.02		
Total	**42,840,000**	**2.41**	**108.06**	**25.9%**
Fuel				
Vehicles	480,000	0.03		
Plant Gas	500,000	0.03		
Total	**980,000**	**0.06**	**2.47**	**0.6%**
Maintenance				
Fixed Plant	9,540,000	0.54		
Mobile Equipment	140,000	0.01		
Total	**9,680,000**	**0.55**	**24.42**	**5.9%**
Reagents and Consumables				
Reagent Price	57,230,000	3.22		
Annual Consumables	27,570,000	1.55		
Total	**84,800,000**	**4.78**	**213.90**	**51.3%**
Contract/General Expenses				
General Consumables	260,000	0.01		
Contract Expenses	910,000	0.05		
General Expenses	2,450,000	0.14		
Total	**3,620,000**	**0.20**	**9.13**	**2.2%**
TOTAL	**165,200,000**	**9.31**	**416.70**	**100%**

Labor

Labor Cost Development

Labor costs were developed by a build-up of base labor rates, on-costs and required work force numbers. Workforce numbers were developed from base by assessing requirements in each area, and in consultation with Vista personnel, adjusting for areas specific to Mt. Todd requirements.

Labor rates were initially taken as the Proteus standard rates (actual operating mine data from 2010), but were subsequently adjusted up by 7% in consultation with Vista. The rates were verified with an independent consultant in Australia. This review indicated labor rates for four out of the 154 categories presented required an upwards adjustment.

Labor on-costs were adjusted from the standard Proteus factor of 26.85% to Vista's favored 25.0%. The whole site labor force was presented in the Proteus operating cost analysis to ensure that there was some consistency in labor rates across the board, however mining and mining related labor costs were not included in the Proteus operating cost estimate as these costs were ultimately in the domain of the mining consultant MDA's operating cost schedule.

Labor Costs

Final process plant and general and administrative (G&A) labor cost estimates issued for the PFS were US$21.27 million per year.

Transport and Accommodation

Accommodation Cost Development

Taking on board the Vista model for labor force accommodation of a workforce self-funded housing scheme based in Katherine and Pine Creek, the requirements for on-going use of any camp post the construction period was estimated as follows:

- Accommodation allowance to cover personnel recruitment, assuming a 20% turnover of the entire workforce annually, and assuming these personnel would consist of a four unit family requiring accommodation in the camp for an average of two months before sourcing their own accommodation. This provided an estimated requirement for 54 rooms in the camp per annum.
- Accommodation for contractors flying to site, largest of which would predominantly consist of the mill reline crew. Assuming a nominal sum of 10 other contractors throughout the year, and assuming these could be staggered to require accommodation for periods other than during mill relines, gave an estimated requirement for an additional 18 rooms.
- Accommodation for miscellaneous visitors etc., where accommodation for whatever reason could not be mutually exclusive with mill relines provided a nominal requirement for seven rooms.
- For the total on-going accommodation estimate of 69 rooms per annum, a requirement for 70 rooms was anticipated.

An allowance of $67.50 per man per day was made for a continuation of the partial construction camp.

Transport Cost Development

Using the numbers developed for the accommodation requirement, flights to Darwin from Perth were estimated at 204 return flights per annum. Allowing a 42% / 17% / 42% split between Low, Shoulder and High seasons respectively, and assuming all flights were at fully flexible fares provided the basis for annual flight expenditures.

Transport and Accommodation Costs

Final transport and accommodation cost estimates issued for the PFS were US$2.010 million per year.

Power Requirements

Power usage was developed by a combination of methods, namely:

- Significant power consuming items had power consumptions calculated from base formulae and models, and included the following items:
  - Primary crusher;
  - Secondary crushers;
  - Ball mills;
  - HPGR units; and
  - For smaller or steady state power consumers the power consumed was calculated as a factor of installed power, with the factor varying on known vendor motor oversizing propensities.
- Nominal allowances were made for some areas where actual installed power was estimated based on usual loads for such duties, and included items such as the air conditioners, lighting and small power etc.

The total power consumption was 649 GWh/year. Assuming power cost of $0.066/kWh, power costs were estimated at US$42.840 million per year.

Fuel

Fuel consumption estimates were developed for each item of process plant mobile equipment, by estimating an annual operating hours and using vendor documented or estimated fuel consumptions for each equipment item.

Plant fuel consumption accounts for utility mobile equipment and vehicles. Fuel costs were estimated at US$1.288 per liter as bulk supply, then allowing for a US$0.319 federal government rebate to provide a net cost of US$0.968 per liter. Fuel cost estimates issued for the PFS was US$0.980 million per year.

Maintenance

Maintenance costs were developed by applying factors to free in store (FIS) equipment costs. The Proteus maintenance cost estimating methodology is consistent with that of the Australasian Institute of Mining and Metallurgy (Cost Estimation Handbook For The Australian Mining Industry, AUSIMM, 1993). Proteus factors have been developed over a period of time and fall within the AUSIMM guidelines.

Large wear items (crusher wear liners, ball mill lifters / liners) were identified and listed separately in the consumables section.

An additional allowance of 2% was applied across the site equipment to allow for sustaining capital expenditure.

The maintenance cost estimate issued for the PFS are US$9.68 million per year.

Reagents

Reagent costs were estimated by applying the ALS determined consumption rates by quoted cost of delivered reagents to site. Instances where consumption rates were altered from the original ALS testwork or previous assumptions included:

- Consumption of carbon was increased from 15 g/t to 20 g/t based on Proteus industry experience.
- Flocculant consumption was reduced from 18 g/t to 15 g/t based on recent Outotec thickener settling testwork at P_{80} of 90 µm.

Reagent prices were obtained from quotes from relevant suppliers. For the PFS only one vendor quote for the majority of reagents was available, with multiple additional quotes still pending.

Two quotes were sourced for the highest expenditure reagent (sodium cyanide), with the Australian Gold Reagents Pty. Ltd. (AGR) chosen as the most cost effective supplier. Further price sourcing from overseas suppliers was on-going at the time of writing.

Transport costs of reagents to site were sourced from reagent suppliers, in addition to an independent quote from a transport agency. As the majority of the reagents supplied by Orica Limited (Orica), with the exception of cyanide, were delivered to Darwin, transport costs from Darwin to site were required. The most economical of the quotes for delivery from Darwin to Katherine was chosen as the cost to be used in the PFS, in this case it was from Seatram Australia Pty. Ltd. (Seatram).

The reagent cost estimate for the PFS is US$57.23 million per year.

Consumables

Consumable costs were estimated by calculating or estimate consumable consumption rates coupled with quotes or estimates for unit prices.

Consumption of mill balls was calculated based on the ore abrasion index, and since this item was the largest expenditure in the consumable category three quotes were received, with the most cost effective being Shandong Humain (China).

Where possible, transport costs were sourced from suppliers, however if they were not provided costs were sourced from other quotes. The quote from Shandong Huamin only included shipping to Darwin, transport costs from Darwin to Katherine were sourced from the Moly-Cop quote.

In some instances where vendor advice was not received in a timely fashion, consumable quotes were scaled from previous studies, and included the primary crusher liners.

Optimization opportunities as identified in Proteus' Independent Assessment report were maintained, and included items like the use of two refurbished HPGR tires for every new tire purchased.

Consumable cost estimates issued for the PFS was US$27.570 million per year.

Contract/General Expenses

Proteus standard factors were used for general expenses and general consumables, some items of which are a standard allowance and others which are linked to site personnel numbers (clothing, medical supplies etc.). General expenses and consumables allowed for included:

- General consumables; office and general supplies, tools and equipment, communications maintenance materials, sampling and analysis consumables;
- Contract expenses; environmental monitoring costs, contracting electrical expenses; and
- General expenses; emergency supply, personnel recruitment, legal/compliance, office communications, safety supplies.

Proteus' standard allowances were included for contract expenses, with the adjustments specific for Mt. Todd including:

- Additional allowances for environmental monitoring costs as advised by Vista; and
- Additional contract electrical costs to allow for the complexity of interaction and maintaining dual source high voltage power supplies.

General / Contract Expenses in addition to General Consumables cost estimates issued for the PFS were US$3.620 million per year.

21.2.3 POWER PLANT

Fuel Costs

Fuel cost analysis is based on baseload operation of the power station calculated using generating equipment Higher Heating Value (HHV) heat rates operating at the average annual condition (27.2˚C and 24% relative humidity). The fuel gas rate of AUD$5.475/GJ is given in the draft terms of supply provided by a qualified natural gas supplier dated February 28, 2013.

Table 21-12 lists the fuel gas requirements for the Trent 60 WLE gas turbine for baseload power and the incremental requirements for one MAN 20V35/44SG reciprocating engine.

Table 21-12: Fuel Cost Summary

	Rolls Royce Trent 60 WLE	MAN 20V35/44SG
Gross Output at 27.2C	58,192	10,400
Net Output at 27.2C	57,053	9,700
Auxiliary Loads	1,139	700
Availability	98%	95%
LHV Heat Rate (kJ/kWh)	8844	7738
HHV Net Heat rate (kJ/kWh)	9,790	8,566
Thermal Efficiency	40.71%	46.52%
Annual Fuel Gas Consumption (GJ/yr)	4,890,912	741,374
Annual Fuel Cost for Baseload ($AUD/yr)	**26,777,741**	**4,059,023**

Operating Costs

Operating costs for the power station are more than 90% fuel costs with the remainder scheduled maintenance and parts replacement. Fuel costs are based on higher heating value (HHV) gross heat rates of 9,904 kJ/kWh for the gas turbine and 8,566 for the reciprocating engines at average ambient condition (27.2˚C and 24% relative humidity) and fuel cost of AUD$5.475/GJ listed in the draft term sheet of key terms and conditions to supply natural gas to Vista provided by the PWC and dated February 28, 2013.

Operating costs include a minimal staff dedicated to the power station and based in a new control room/workshop building. Personnel staff will consist of three swing shifts at two operators per 12 hour shift with a mechanic and instrumentation/electrical technician on one shift per day. One of the day shift operators can also serve as the control room manager. Labor costs provided are referenced from the 2010 Hays Salary Guide for energy sector trades in Northern Territory. Plant personnel costs are shown in **Table 21-13**.

Table 21-13: Personnel Costs, Power Plant

	Annual Salary	Number on Staff
Control Room Manager/Operator	$100,000	1
Control Room Operator	$75,000	5
Instrumentation/ Electrical Technician	$80,000	1
Mechanic	$80,000	1
Total	**$635,000**	**8**

Properly maintained, aeroderivative gas turbines can operate up to 24,000 hours (approximately 3 years) between maintenance overhauls with annual inspections. Annual inspections require 96 hours of turbine downtime and consist of function checks of the gas turbine package systems, borescope of the gas generator/power turbine and safety checks of the equipment and control system. Overhauls alternate between a hot section inspection and major inspection and require approximately 96 hours of turbine downtime with the use of a support unit. It is expected that power station downtime will be coordinated with other major process turnarounds so the site can operate on electricity provided by the two 22kV connections to the utility grid. PWC has indicated these lines are rated for up to 10MVA each but may have additional capacity pending inspection and electrical tests of the lines and interconnecting equipment.

The gas turbine maintenance plan schedule and budgetary quote provided by Rolls Royce follows a 6 year cycle of annual inspections with a mid-life overhaul of hot section components at 3 years and a major overhaul of the gas turbine and generator on the 6th year. Contracted maintenance includes all scheduled spare parts replacement and unit health monitoring to help plan maintenance for optimum performance of the gas turbine and auxiliaries.

To minimize downtime during overhauls, an exchange engine is available for installation when the turbine is shipped offsite for inspection and refurbishment. For the core annual fee and exchange engine fee, the exchange engine is guaranteed to be available for installation when the gas turbine is offline for its scheduled maintenance. Without an exchange engine available the gas turbine will be shipped offsite for 13 weeks at mid-life inspections and 15 weeks for major overhauls.

An additional lease engine fee will provide availability of an exchange engine for unscheduled outage events and comprehensive service coverage may be purchased to assure power station availability. At this point is it recommended to include the lease engine fee in lifecycle cost estimate but omit the comprehensive services Fee until a final price is negotiated with Rolls Royce. The gas turbine maintenance cost schedule is shown in **Table 21-14**.

Table 21-14: Gas Turbine Maintenance Cost Schedule

Year	Annual Inspection	Mid-Life Inspection	Major Overhaul	Core Annual Fee	Exchange Engine Fee (Scheduled Maintenance)	Lease Engine Fee (Unscheduled Maintenance)	Comprehensive Services
1	$115,463	-	-	$218,427	$325,467	$135,375	$165,015
2	$115,463	-	-	$218,427	$325,467	$135,375	$165,015
3	-	$2,894,105	-	$218,427	$325,467	$135,375	$165,015
4	$115,463	-	-	$218,427	$325,467	$135,375	$165,015
5	$115,463	-	-	$218,427	$325,467	$135,375	$165,015
6	-	-	$5,746,181	$218,427	$325,467	$135,375	$165,015

Note: All estimated are shown in $AUS

Table 21-15 lists the annual operating costs for the Rolls Royce Trent 60 WLE gas turbine to generate nominal net output of 76MW for the 17 year operating life of the mining project. This estimate includes operating overhead personnel to meet the mine's base power demand.

Table 21-15: Power Station Annual Operating Costs

Year	Gas Turbine Fuel Costs	Water Treatment Costs	Gas Turbine Maintenance	On-Site Personnel	Recip Engine Fuel Costs	Recip Engine Maintenance
1	26,777,741	48,983	794,733	635,000	8,118,046	1,094,514
2	26,777,741	48,983	794,733	635,000	8,118,046	1,094,514
3	26,777,741	48,983	3,573,375	635,000	8,118,046	1,094,514
4	26,777,741	48,983	794,733	635,000	8,118,046	1,094,514
5	26,777,741	48,983	794,733	635,000	8,118,046	1,094,514
6	26,777,741	48,983	6,425,451	635,000	8,118,046	1,094,514
7	26,777,741	48,983	794,733	635,000	8,118,046	1,094,514
8	26,777,741	48,983	794,733	635,000	8,118,046	1,094,514
9	26,777,741	48,983	3,573,375	635,000	8,118,046	1,094,514
10	26,777,741	48,983	794,733	635,000	8,118,046	1,094,514
11	26,777,741	48,983	794,733	635,000	8,118,046	1,094,514
12	26,777,741	48,983	6,425,451	635,000	8,118,046	1,094,514
13	26,777,741	48,983	794,733	635,000	8,118,046	1,094,514
14	26,777,741	48,983	794,733	635,000	8,118,046	1,094,514
15	26,777,741	48,983	3,573,375	635,000	8,118,046	1,094,514
16	26,777,741	48,983	794,733	635,000	8,118,046	1,094,514
17	26,777,741	48,983	794,733	635,000	8,118,046	1,094,514
Subtotal	**455,221,597**	**832,711**	**33,107,818**	**10,795,000**	**138,006,782**	**18,606,738**
					TOTAL:	656,570,646

The reciprocating gas engines require semi-annual maintenance when operated in continuous duty. Each engine will be offline for 220 hours per maintenance inspection, making the engines available about 95% of the year. Industry guidelines estimate maintenance costs at approximately $0.0063 per MWh for an engine similar to the MAN 20V35/44SG. Preventative maintenance plans can be purchased in advance from the manufacturer, reducing the need for full time staff to perform these semi-annual overhauls. **Table 21-15** also shows the incremental operating costs of two MAN 20V35/44SG engines each with 9.7MW net output.

22.0 ECONOMIC ANALYSIS

Project economics for the Base Case are primarily based on inputs developed by MDA, Proteus, and Tetra Tech. Economic results presented in the report suggest the following conclusions, assuming a 100% equity project:

- Mine Life: 13 years;
- Pre-Tax $NPV_{5\%}$: US$1,094 million, IRR: 22%;
- Post-Tax $NPV_{5\%}$: US$591 million, IRR: 16%;
- Payback (Post-Tax): 3.5 years;
- NT Royalty Taxes Paid: US$270 million;
- Australian Income Taxes Paid: US$462 million; and
- Cash costs (including Royalty): US$773/oz-Au.

All costs and economic results are presented in Fourth Quarter, 2012 US dollars. No escalation has been applied to capital or operating costs.

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

22.1 PRINCIPAL ASSUMPTIONS

Parameters used in the analysis are shown in **Table 22-1**. These parameters are based upon current market conditions, vendor quotes, design criteria developed by Vista and their consultants, and benchmarks against similar existing projects.

Mt. Todd will commence at a production rate of 50,000 tpd. Fresh ore production will originate from the open pit mine and will be treated using conventional CIL technology. Once ore is exhausted from the pit, the reserves in the existing heap leach pad will then be processed.

Projected revenues from the sale of gold doré are based upon a market price of US$1,450/oz-Au. Vista is in discussion with the Perth Mint for the sale of its product. It is too early to enter into definitive agreement with refiners as of the date of this PFS. However, refinery assumptions used in the technical economic model (TEM) are indicative of current refiner rates.

Table 22-1: TEM Principal Assumptions

Principal Assumptions	Unit	Parameter
Pre-Production Period	Years	2
Mine Life	Years	13
Operating Days	Days / Year	355
Gold Price	USD	$1,450
NPR Royalty – Jawoyn	%	1
Exchange Rate	AUD:USD	1:1
Diesel Fuel	AUD/L	$0.969
Natural Gas	AUD/GJ	$5.475
Electric Power – From Grid	AUD/kWh	$0.220
Electric Power – From Plant	AUD/kWh	$0.066

Refining costs are summarized in **Table 22-2** resulting in an all in refining cost of US$3.19/Au-oz over the LoM.

Table 22-2: Refining Costs

Cost Component	Cost (US$)
Refining Fee	0.75
Gold Retention	0.10%
Purchase Discount-Gold	0.50
Assay Fee	95.00
Environmental Fee	50.00
Freight & Insurance	0.20

The Project will be subject to a 30% corporate income tax and will also pay a royalty to the Northern Territory Government of 20%. The cost of capital for the purpose of this report is 5%, assuming a 100% equity project.

22.2 LoM Production

Ore will be mined using open pit mining methods. Production over the LoM is summarized in **Table 22-3**.

Table 22-3: LoM Ore Production

Production	kt	g/t	Contained Au (koz)
Waste	562,349	-	-
Ore	209,451	0.84	5,669
Heap Leach	13,354	0.54	232
Total Production	**785,154**	**1.38**	**5,901**

The Mt. Todd Gold Project has been planned as an open-pit truck and shovel operation. Open pit ore totals 209 Mt grading 0.84 g/t and contains 5.7 Moz of gold. Open pit production will have a 2.7:1 strip ratio over the 13-year LoM. Upon completion of conventional mining, the existing heap leach pad will be processed.

Ore is planned to be processed in a large comminution circuit consisting of a gyratory crusher, two cone crushers, two HPGR crushers, and three ball mills as discussed in greater detail below. Vista plans to recover gold in a conventional carbon-in-leach ("CIL") recovery circuit. Process recovery was determined based on ore types. Three ore types, sulfide, mixed, and oxide were identified for the open pit and will have recoveries of 82.0%, 78.0%, and 78.0%, respectively. The heap leach facility will have a recovery of 70.0%.

22.3 CAPITAL COSTS

LoM capital cost requirements are estimated at US$1,406 million as summarized in **Table 22-4.** Initial capital of US$1,046 million is required to commence operations. Sustaining capital of US$498 million is required over the LoM and accounts for capitalized stripping in the open pit, mine equipment additions and replacements, and tailings dam raises. Also included are salvage values for mining and process equipment totaling US$124 million and credit for the reclamation bond of US$15 million.

Table 22-4: LoM Capital Costs (US$000s)

Capex Summary	Initial Capital	Sustaining Capital	Salvage Credit	Total
Capitalized Costs	57,847	134,441	0	192,289
Mine & Process Mobile	138,895	151,460	(82,973)	207,381
CIL Process Plant	409,731	0	(40,651)	369,080
Tailings Dams	19,560	184,092	0	203,652
Power Supply	90,615	0	0	90,615
Water Supply	19,192	0	0	19,192
Owner's Costs	310,544	27,788	(15,000)	323,332
Total Capital Costs	**1,046,383**	**497,782**	**(138,624)**	**1,405,540**

22.3.1 CAPITALIZED COSTS

Capitalized costs consist of mine pre-stripping, dewatering, and closure. Capitalized costs are estimated at US$192 million as summarized in **Table 22-5**. Initial capitalized costs are estimated at US$58 million.

Table 22-5: Capitalized Costs (US$000s)

Capitalized Costs	Initial Capital	Sustaining Capital	Salvage Credit	Total
2100-Mine Pre-Stripping	57,338	32,144	0	89,482
2500-Mine Dewatering	0	7,920	0	7,920
2900-Mine Closure	509	94,378	0	94,887
Total Capitalized Costs	**57,847**	**134,441**	**0**	**192,289**

22.3.2 MINE & PROCESS MOBILE

Open pit mine and process mobile capital costs estimated at US$207 million over the LoM are shown in **Table 22-6**. Initial capital is estimated to be US$139 million. Salvage credit of US$83 million was provided by Vista as a provision to account for mobile equipment which will still be usable at the end of the mine life. A detailed discussion of mine capital costs is presented in Section 21.1.1.

Table 22-6: Mine & Process Mobile Capital Costs (US$000s)

Mine & Process Mobile	Initial Capital	Sustaining Capital	Salvage Credit	Total
1000-Geology	0	0	0	0
2200-Mine Mobile Equipment	116,883	151,460	(82,973)	185,370
2300-Support Facilities	14,249	0	0	14,249
2400-Mine Support Services	1,707	0	0	1,707
4300-Communications	920	0	0	920
4600-Plant Mobile Equipment	5,136	0	0	5,136
Total Mine & Process Mobile	**138,895**	**151,460**	**(82,973)**	**207,381**

22.3.3 CIL Process Plant

CIL process plant capital costs are shown in **Table 22-7**. Initial capital totaling US$410 million will be required for the CIL process plant. Salvage credit of US$41 million accounts for an estimate of equipment only which will be saleable at the end of the mine life. A detailed discussion of process costs is presented in Section 21.1.2.

Table 22-7: CIL Process Plant Capital Costs (US$000s)

CIL Process Plant	Initial Capital	Sustaining Capital	Salvage Credit	Total
3000-Process Plant	409,731	0	(40,651)	369,080
4800-Fuel storage and distribution	0	0	0	0
Total CIL Process Plant	**409,731**	**0**	**(40,651)**	**369,080**

22.3.4 Tailings Dams

Tailings dams capital costs are estimated to total US$204 million over the LoM as shown in **Table 22-8**. The cost includes estimates for both TSF 1 and TSF 2. Approximately 62 Mt of additional tailings are planned for TSF 1. The remaining tailings will be placed in TSF 2, which has a total capacity of 147 Mt. TSF 2 will be constructed in production years 4 and 5, and will become operational in year 5.

Table 22-8: Tailings Dams Capital Costs (US$000s)

Tailings Dams	Initial Capital	Sustaining Capital	Salvage Credit	Total
4400-Tailings Dams 1 & 2	19,560	184,092	0	203,652
Total Tailings Dams	**19,560**	**184,092**	**0**	**203,652**

22.3.5 Power Supply

Power supply capital costs are estimated to total US$91 million over the LoM as shown in **Table 22-9**. These costs occur in pre-production. The power plant will consist of 1- 57 MW gas turbine and 2- 9.7 MW reciprocating units, totaling 76 MW. A detailed discussion of the power supply costs is presented in Section 21.1.3.

During the construction period, power will be supplied by the local utility via the existing transmission line and substation. On-site power generation will commence during the first production year.

Table 22-9: Power Supply Capital Costs (US$000s)

Power Supply	Initial Capital	Sustaining Capital	Salvage Credit	Total
4200-Power supply	90,615	0	0	90,615
Total Power Supply	**90,615**	**0**	**0**	**90,615**

22.3.6 Water Supply

Water supply capital costs are estimated to total US$19.2 million over the LoM as shown in **Table 22-10**. These costs occur in pre-production. The water supply consists of the water treatment plant for both the plant and potable water and the RWD. The water treatment plant is fed by decant water, runoff pond water, and pit dewatering. The RWD will provide approximately 4.5 million m^3 of storage.

Table 22-10: Water Supply Capital Costs (US$000s)

Water Supply	Initial Capital	Sustaining Capital	Salvage Credit	Total
4100-Water supply	19,192	0	0	19,192
Total Water Supply	**19,192**	**0**	**0**	**19,192**

22.3.7 Owner's Costs

Owner's costs primarily occur during pre-production and include contingency. Owner's costs are estimated at US$323 million. These costs are shown in **Table 22-11**.

Table 22-11: Owner's Costs (US$000s)

Owner Costs	Initial Capital	Sustaining Capital	Salvage Credit	Total
4500-Waste Disposal	357	0	0	357
5000-Project Infrastructure	38,118	0	0	38,118
6000-Permanent Accommodation	86	0	0	86
7000-Site Establishment & Early Works	27,548	0	0	27,548
8000-Management, Engineering, EPCM Services	123,236	5,040	(15,000)	113,276
9000-Preproduction Costs (excl. cont. & man. Res.)	14,015	0	0	14,015
Contingency	107,185	22,748	0	129,933
Total Owner Costs	**310,544**	**27,788**	**(15,000)**	**323,332**

22.4 Operating Costs

LoM operating costs are summarized in **Table 22-12**. The operating costs will average US$16.68/t-milled over the LoM.

Table 22-12: LoM Operating Costs

Opex Summary	US$/t-mined	US$/t-milled	Total (US$000s)
Open Pit Mining	2.005	6.946	1,547,607
CIL Process Plant	-	8.780	1,956,195
Water Treatment Plant	-	0.073	21,313
G&A	-	0.495	110,310
Jawoyn Royalty	-	0.313	69,717
Refining Costs	-	0.069	15,331
Total Opex Summary	**-**	**16.676**	**3,720,473**

22.4.1 Open Pit Mining Costs

Open pit mining costs are shown in **Table 22-13**. Costs will average US$2.01/t-ore (US$6.95/t-milled) over the LoM. Hauling is the highest cost item, US$1.17/t-ore, and US$4.05/t-milled respectively. Note also that unit costs per tonne milled include 13.4 Mt of heap leach ore which is not mined. A detailed discussion of mining costs is presented in Section 21.2.1.

Table 22-13: Open Pit Operating Costs

	US$/t-mined	US$/t-milled	Total (US$000s)
Mine General Service	0.029	0.100	22,389
Mine Maintenance	0.038	0.131	29,177
Engineering	0.019	0.066	14,801
Geology and Grade Control	0.024	0.083	18,384
Drilling	0.238	0.825	183,836
Blasting	0.148	0.513	114,406
Loading	0.228	0.789	175,849
Hauling	1.169	4.049	902,219
Mine Support	0.103	0.358	79,871
Rehandle HLP	0.009	0.030	6,677
Total Open Pit Mining	**2.005**	**6.946**	**1,547,607**

22.4.2 CIL Process Plant Costs

CIL process plant operating costs averaging US$8.78/t-milled are shown in **Table 22-14**. A detailed discussion of process and G&A costs is presented in Section 21.2.2.

Table 22-14: CIL Process Plant Operating Costs

	US$/t-milled	Total (US$000s)
Labor	0.906	201,835
Trans. & Accom.	0.113	25,214
Power	2.312	515,025
Diesel Fuel	0.055	12,301
Maintenance	0.545	121,427
Reagents & Consumables	4.777	1,064,443
HLP Reagent Factor	0.072	15,950
Equipment Hire	0.000	0
Total CIL Process Plant	**8.780**	**1,956,195**

22.4.3 Water Treatment Plant Costs

Water treatment plant operating costs averaging US$0.07/t-milled are shown in **Table 22-15**.

Table 22-15: Water Treatment Plant Operating Costs

	US$/t-milled	Total (US$000s)
Chemicals		
Lime	0.011	2,356
Ferric Chloride	0.028	6,270
Sulfuric Acid	0.004	853
Sodium Hypochlorite	0.003	640
Caustic	0.000	7
Citric Acid	0.000	46
Dewatering		
Electricity	0.000	0
Power		
Electricity	0.008	1,835
Labor		
Manager	0.004	780
Operator	0.014	3,120
Maintenance	0.002	390
Total Water Treatment Plant	**0.073**	**16,298**

22.4.4 General & Administrative

G&A will average US$0.50/t-milled over the LoM as shown in **Table 22-16**.

Table 22-16: G&A Operating Costs

	US$/t-milled	Total (US$000s)
Labor	0.291	64,900
Contract-General Expenses	0.204	45,410
Total G&A	**0.495**	**110,310**

22.4.5 Jawoyn Royalty

Jawoyn Royalty costs averaging US$0.31/t-milled are shown in **Table 22-17**.

Table 22-17: Jawoyn Royalty Costs

	US$/t-milled	Total (US$000s)
Jawoyn Royalty	0.313	69,717
Total Jawoyn Royalty	**0.313**	**69,717**

22.4.6 Refining Costs

Refining costs averaging US$0.07/t-milled are shown in **Table 22-18**.

Table 22-18: Refining Costs

	US$/t-milled	Total (US$000s)
Refining Cost	0.069	15,331
Total Refining Cost	**0.069**	**15,331**

22.4.7 Operating Cost Inputs

Inputs used to estimate operating costs are summarized in this section.

Labor

The labor breakdown shown in **Table 22-19** represents the personnel contingent at steady state operations. Labor rates are fully burdened, are presented in Australian Dollars, and are based upon recent Australian labor rate surveys. Specific staff titles, numbers, and costs are provided in the TEM.

Table 22-19: Process Labor Costs

Labor Summary	No. Of Staff	Annual Cost (AUD)
Management	7	2,048,750
Administration	10	937,500
Finance	2	263,750
Information Technology	2	285,000
Processing Technical	10	1,671,000
Processing Production	50	6,837,500
Fixed Plant Maintenance	36	5,235,865
Laboratory	14	1,601,827
Mining Technical	15	2,233,750
Mining Production	181	23,445,000
Mobile Maintenance	18	2,493,750
Surveying	3	381,250
Geology	5	762,500
Supply	8	996,250
SHE	11	1,362,500
Training	2	256,250
Camp / Messing	6	480,000
Power Plant	8	1,262,500
Water Treatment Plant	6	802,500
Reclamation	6.1	762,500
Tailings Management	2.12	264,125
Projects	0	0
Office	0	0

Reagents

Reagent consumption rates and costs are shown in **Table 22-20**. Consumption rates are based upon metallurgical testwork and prices are based on vendor quotes, including a delivery to site.

Table 22-20: Process Reagents

Reagent	Consumable Rate	Unit	Annual Cost (AUD)
Quick Lime	910	g/t ore	4,864,565
Sodium Cyanide	770	g/t ore	38,077,655
Sodium Hydroxide	40	g/t ore	521,039
Flocculant	15	g/t ore	920,921
Sodium Metabisulfite (SMBS)	730	g/t ore	7,017,127
Hydrochloric Acid	81	g/t ore	780,535
Lead Nitrate	100	g/t ore	4,178,438
Activated Carbon	20	g/t ore	857,875
Borax	150	kg/t conc.	4,039
Silica	150	kg/t conc.	1,474
Soda Ash	100	kg/t conc.	1,365
Potassium Nitrate	30	kg/t conc.	557

Consumables

Consumable consumption rates are based upon benchmark data and vendor information given the ores processed at the site. Costs for consumables are based upon vendor quotes including delivery to site. These costs are shown in **Table 22-21**.

Table 22-21: Process Consumables

Consumables	Unit Cost	Unit	Consumable Rate	Unit
Crushing				
Primary Crusher mantle	61,299	USD/mantle	3,550	kt/mantle
Primary Crusher concaves	146,103	USD/concave	10,650	kt/concave
Secondary Crusher wear liners	74,000	USD/set	3,625	kt/set
Grinding Mill Balls				
68mm	944	USD/t	0.95	kg/t ore
65mm	952	USD/t		kg/t ore
50mm	960	USD/t		kg/t ore
40mm	976	USD/t		kg/t ore
Mill Liners	1,149,720	USD/set		
HPGR				
HPGR tires, new	1,610,083	USD/set	6,000	hr/set
HPGR tires, refurbished	1,046,554	USD/set		
Lime Slaker - Mill Balls				
40mm	1,063	USD/t	0.5	kg/t lime

22.5 ECONOMIC RESULTS

Project cost estimates and economics are prepared on an annual basis. Based upon design criteria presented in this report, the level of accuracy of the estimate is considered ±25%.

Economic results are summarized in **Table 22-22**. The analysis suggests the following conclusions, assuming a 100% equity project at a gold price of US$1,450:

- Mine Life: 13 years;
- Pre-Tax $NPV_{5\%}$: US$1,094 million, IRR: 22%;
- Post-Tax $NPV_{5\%}$: US$591 million, IRR: 16%;
- Payback (Post-Tax): 3.5 years;
- NT Royalty Taxes Paid: US$270 million;
- Australian Income Taxes Paid: US$462 million; and
- Cash costs (including Royalty): US$773/oz-Au.

Table 22-22: Technical-Economic Results

Cash Flow Summary	LoM Cost (US$000s)	Unit Cost US$/t-milled	Unit Cost US$/oz-Au
Gold Produced	4,808	-	-
Gold Price	1,450	-	-
Gold Sales	**6,971,674**	**31.29**	**1,450.00**
Refinery Costs	(15,331)	(0.07)	(3.19)
Net Smelting Return	**6,956,343**	**31.22**	**1,446.81**
Jawoyn Royalty	(69,717)	(0.31)	(14.50)
Gross Income from Mining	**6,886,626**	**30.91**	**1,432.31**
Open Pit Mine	(1,547,607)	(6.95)	(321.88)
CIL Process Plant	(1,956,195)	(8.78)	(406.86)
Water Treatment Plant	(21,313)	(0.10)	(4.43)
G&A	(110,310)	(0.50)	(22.94)
Operating Costs	**(3,635,425)**	**(16.32)**	**(756.11)**
Power Sales Credit	93,754	0.42	19.50
Cash Cost of Goods Sold (COGS)	**(3,557,002)**	**(15.96)**	**(739.80)**
Operating Margin	**3,344,955**	**15.01**	**695.70**
Capitalized Costs	(192,289)	(0.86)	(39.99)
Mine & Process Mobile Capital	(290,355)	(1.30)	(60.39)
CIL Process Plant	(409,731)	(1.84)	(85.22)
Tailings Dams (4400)	(203,652)	(0.91)	(42.36)
Power Supply (4200)	(90,615)	(0.41)	(18.85)
Water Supply & Treatment(4100)	(19,192)	(0.09)	(3.99)
Owner Costs	(338,332)	(1.52)	(70.37)
Salvage	138,624	0.62	28.83
Capital Costs	**(1,405,540)**	**(6.31)**	**(292.33)**
Pre-Tax Cash Flow	**1,939,415**		
$NPV_{5\%}$	**1,093,859**		
IRR	**22%**		
Payback (years)	**2.85**		
Post-Tax Cash Flow	**1,230,523**		
$NPV_{5\%}$	**591,318**		
IRR	**16%**		
Payback (years)	**3.50**		

22.5.1 Taxes, Royalties

Royalties

Northern Territory Royalty

Mineral royalties are levied in compliance with the Minerals Royalty Act of the Northern Territory. Royalties are based on the net value of production from a mine, irrespective of the nature of the land holding. A flat rate of 20% is applied. No royalty is payable on the first AUD$50,000 of net value.

Net value of production for the purposes of calculating the royalty is based the formula:

GR – (OC + CRD + EEE + AD)

where -

GR is the gross realization from the production unit in the royalty year;

OC is the operating costs of the production unit for the royalty year;

CRD is the capital recognition deduction;

EEE is the eligible exploration expenditure, if any; and

AD is any additional deduction.

Other Royalties

If the Mt. Todd Gold Project proves feasible, Vista has agreed to offer the Jawoyn Association Aboriginal Corporation (JAAC) the opportunity to establish a Joint Venture Company with Vista holding 90% and the JAAC holding a 10% participating interest, to finance and develop the Project.

For rent of the surface rights from the current mining licenses, including the mining license on which the Batman deposit is located, the JAAC is entitled to an annual amount equal to 1% of the value of production with a minimum annual payment of AUD$50,000.

There is also a royalty of 5% of the gross value of gold or other metals commercially extracted from certain mineral concessions which are located outside the zone of mineralization currently defined as the Batman deposit (the "Denehurst royalty").

Taxes

Australian Federal Income Tax

The rate of corporate income tax in Australia is currently 30%. There is no alternative minimum tax.

Taxable income is based on assessable income less allowable deductions. Assessable income generally includes gross income from the sale of goods, the provision of services, dividends, interest, royalties and rent. Assessable income may also include capital gains after offsetting capital losses. Normal business expenses are deductible.

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

Thin capitalization provisions can limit the deductibility of interest and other "debt deductions" in certain cases. In general, a deduction will be partly disallowed if the company's debt exceeds three times its equity.

Transfer pricing rules apply to international transactions/dealings between separate legal entities. Covered cross-border transactions include those involving tangible or intangible property, the provision of services and financing. There are several generally accepted transfer pricing methods available in Australia.

Consolidation allows wholly owned corporate groups to operate as a single entity for income tax purposes.

Australia operates a full imputation system for the avoidance of double taxation of dividends. Under this system, the payment of company tax is imputed to shareholders so that shareholders are relieved of their tax liability to the extent profits have been taxed at the corporate level. Dividends paid out of profits on which corporate tax has been paid are said to be "franked" and generally entitle shareholders to an offset for the corporate tax paid.

22.5.2 Sensitivity

Project sensitivities are summarized in **Table 22-23** and shown graphically in **Figure 22-1**. As seen, the Project is most sensitive to gold production and gold price. Sensitivity on operating and capital cost is closely matched, with the Project being only slightly more sensitive to operating costs.

Table 22-23: Project Sensitivity

Parameter	-15%	-10%	-5%	Base	5%	10%	15%
Market Price	$174,333	$314,110	$453,192	**$591,318**	$729,466	$867,130	$1,004,175
Opex	$806,093	$734,950	$663,360	**$591,318**	$519,276	$447,081	$374,016
Capex	$736,496	$688,105	$639,569	**$591,318**	$543,067	$494,816	$446,678

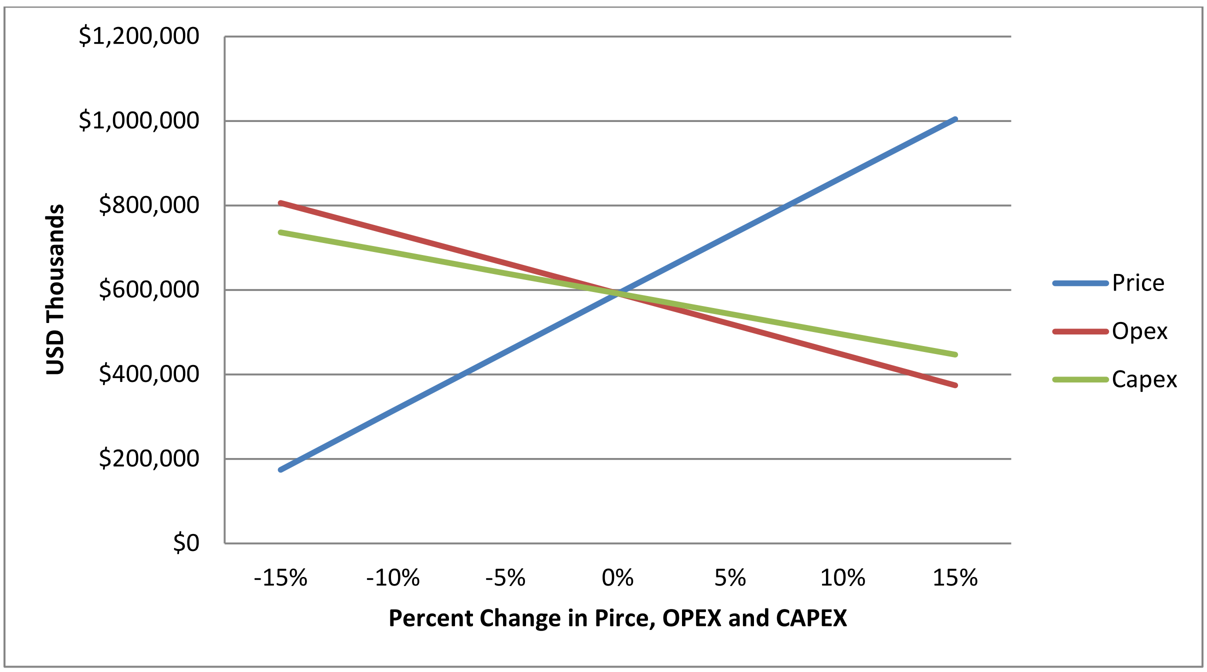


Figure 22-1: Project Sensitivity

23.0 ADJACENT PROPERTIES

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

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 GEOTECHNICAL

Bulk earthworks for the process plant is designed to minimize the import of fill material and excavation of rock. Where fill material is required to be imported, material from the existing RoM Pad ramp and from the existing stockpile located adjacent to the Tollis and Golf Pits will be utilized. The civil basis of design took into consideration the following geotechnical information:

- Geotechnical Desktop Study Mt. Todd Process Plant DFS undertaken by Coffey Geotechnics in December 2012. The study reviewed previous Soil and Rock Engineering (SRE) geotechnical data from December 1992 and April 1993 for the original Mt. Todd development. The study also reviewed SRE earthworks monitoring data for construction of the original Mt. Todd development. Recent April 2012 and December 2012 Tetra Tech geotechnical test pit data was also reviewed. The study focused on foundations for heavy vibrating loads including the crusher and mill as well as screening structures and ancillary plant buildings. A review of potential borrow material in close proximity to the proposed plant site suitable for structural fill and pavement construction was also included.
- Foundation Recommendations report produced by Tetra Tech in April 2012. The geotechnical test pit investigation was conducted in October 2011 at the then proposed location of the process plant. A summary of the test pit investigation and preliminary recommendations for foundation design at the site were provided.
- Technical Memorandum regarding "Results of Test Pit Excavation Program and Borrow Source Investigation, Mt. Todd Project, Vista Gold Corporation, Northern Territory, Australia" from Tetra Tech dated 20 December 2012. A summary of the test pit results and potential borrow sources were provided.
- Foundation Recommendations report produced by Tetra Tech in February 2013. The report reviewed previous Soil and Rock Engineering (SRE) geotechnical data from December 1992 and April 1993 for the original Mt. Todd development. The report also reviewed previous Foundation Recommendations report produced by Tetra Tech in April 2012 and the previous geotechnical test pit investigation conducted in December 2012 at the proposed location of the process plant. A summary of the test pit investigation and recommendations for foundation design at the site were provided.

Further geotechnical investigation is recommended during the design phase of the project to obtain geotechnical data in the final location of foundations for heavy vibrating loads including the crusher and mill as well as screening structures and ancillary plant buildings and also to confirm fill material and rock excavation requirements.

24.2 GEOCHEMISTRY

Tetra Tech was commissioned by Vista to conduct geochemical characterization studies and predictive modeling in support of the Mt. Todd Project PFS.

Waste rock samples were selected from the three distinct rock units identified from the 18 mapable rock codes, specifically:

- Greywacke;
- Shale; and
- Mixed greywacke/shale (interbedded).

Eighty-seven waste rock samples were subjected to acid-base accounting (ABA). Nine samples, including three samples from each of the three distinct units were selected for kinetic testing using humidity cell tests. Mineralogy by quantitative x-ray diffraction (XRD) was conducted on the nine humidity cell test samples.

The greywacke waste rock sample average nitric acid (HNO_3) extractable (sulfide) sulphur content of 0.19 wt. % was comparatively low with interbedded and shale samples containing 0.51 and 0.31 wt. %, respectively. Hydrochloric acid (HCl) extractable (sulfate) sulfur was largely absent suggesting that minimal sulfide oxidation occurred prior to geochemical characterization. On average, insoluble sulphur made up approximately 30% of the sulphur distribution in the 87 samples that underwent ABA testing. The average sulphur content of the waste rock samples was ≤ 0.51 wt. % HNO_3 extractable sulfide sulphur, however, the potential for acid formation remains a concern due to the limited amount of neutralization potential (NP). On average, samples contained NP ≤ 11 kg $CaCO_3$/tonne rock. A neutralizing potential ratio (NPR) ABA screening criteria < 2 suggests that a majority of the waste rock samples are either potentially acid generating or highly likely to generate acid whereas approximately 30% of the samples were highly unlikely to generate acid. The samples contained high insoluble sulphur (> 30 wt. %) which may be from sulfidic species that are resistant to HNO_3 digestion such as sphalerite (ZnS) and/or galena (PbS).

Preliminary sulphur cutoff criteria were developed based on ABA and Net Acid Generation (NAG) pH results, to assist with waste rock management and closure planning. The specific sulphur cutoff values are:

- Non-PAG waste rock is defined by total sulphur 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 sulphur;
- The total sulphur content of PAG waste rock is > 0.4 wt. %; and
- Waste rock with > 1.5 wt. % sulphur was considered to be likely acid generating.

The cutoffs were used for geochemical modeling of the WRD seepage and pit lake wall rock runoff and can be used in combination with the total sulphur block model based on the exploration database to assist with proper routing of waste rock.

The nine waste rock samples selected for kinetic testing were subjected to humidity cell testing. Weekly leachate quality results were obtained for pH, acidity, alkalinity, electrical conductivity, and sulfate over the entire test duration. Monthly leachate composites for dissolved constituent concentrations were also obtained over the testing period. Of the nine samples subjected to kinetic testing, a shale sample with 0.43 wt. % HNO_3 extractable sulfide sulphur and low NP = 3.7 kg $CaCO_3$/tonne rock produced acidic leachate (pH < 6) from the initiation of testing. Elevated copper, lead, nickel, and zinc levels were observed in leachate from the acid generating cell. The remaining humidity cells produced circumneutral pH values, with relatively low concentrations of metals. However, it is anticipated that given ample time these cells will likely produce acidic leachate and concomitant increased metal concentrations.

Geochemical characterization of two tailings samples was also conducted including ABA, mineralogy, water leaching, and supernatant analysis. Humidity cell testing has been initiated on one of the samples. The samples contain 1.25 wt. % and 1.13 wt. % total sulphur with net acid production potential (NAPP) and NPR values that show the tailings have potential to eventually generate acid. However, the tailings supernatant and water leach testing produced alkaline pH values. Concentrations of some metals/metalloids, major ions, and cyanide in the tailings supernatant were above ANZECC water quality guidelines, whereas levels were lower in the water leachate but some metals and metalloids and cyanide remained elevated above the guidelines. After 32 weeks, kinetic testing of one of the samples shows a neutral pH with low concentration of metals. Calculations suggest that abundant sulfide sulphur still remains, suggesting the sample may produce acidic leachate given ample time.

Predictive geochemical modeling was conducted to determine the production phase water quality of the WTP Equalization Pond. The water quality estimates were used as a basis for the WTP design and further assist with LoM site water management planning.

Inputs to the Equalization Pond included precipitation and inputs from ponds/facilities from across the site including:

- RP 1 - WRD Pond;
- RP 2 - LGOS Pond;
- RP 3 - Batman Pit;
- RP 5 - Plant Site Runoff Pond;
- HLP - Heap Leach Pad Pond; and
- RP 7 or RP 8 - the TSF 1 or TSF 2 Ponds.

Biannual water quality estimates suggest the Equalization Pond will predominantly 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 Equalization Pond.

In order for Vista to re-start mining activities, RP3 must be dewatered. Treatment of RP3 by micronized lime has been conducted with success, with pH levels becoming circumneutral with a general decrease in metal concentrations.

24.3 Surface Water Hydrology

The Mt. Todd site is drained by the perennial Edith River, located approximately 1 km south of RP 1, and also drained by several ephemeral streams. Batman Creek, which bisects the center of the site and Horseshoe Creek, is located east of the site. Both Batman and Horseshoe feed Stow Creek, which enters the Edith River at a location south of the discharge point from RP 1.

Horseshoe Creek and Batman Creek catchments are approximately 45 and 11 km^2, respectively. The RWD was built across Horseshoe Creek immediately above the mine, forming a sub-catchment covering about 55% of the Horseshoe Creek catchment. The remainder of the Stow Creek catchment is approximately 144 km^2 and is not impacted by mining activity. Stow Creek flows for a short distance after its confluences with both Batman Creek and Horseshoe Creek, prior to joining the Edith River. The catchment area of the Edith River upstream of Stow Creek confluence is approximately 540 km^2.

Surface water at the site is well-documented and its management has been the object of study by both Vista and the NT Government in recent years. Historically, flows from the mine have exceeded the capacity of the water management system, thus allowing uncontrolled discharges to the Edith River. The effectiveness of the water management system has improved as a result of revisions to the pumping systems, installation of a stage height and telemetry station at SW4 and a flow meter on the siphon and pumping outlets from RP 1. Additionally, the NT Government recently completed a raise of the spillway crest and dam at RP 1 by 1.5 m.

Drainage from the Mt. Todd site enters the Edith River at two locations: discharge point for RP 1 and West Creek. The RP 1 discharge point is located 0.8 km below the Stow Creek and the Edith River confluence and is discussed further below. 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 western side of the WRD via the Western Diversion Drain, and overflow from the RP 1 spillway. The West Creek catchment is small and it is reported that the creek only delivers mine water to the Edith River after substantial rainfall events exceed capacity at RP 1. During the wet season (approximately November to April) uncontrolled discharges to the Edith River occur from RP 1, RP 2 and RP 5 during high-rainfall events. However, for a large part of the year (approximately May to October), no runoff from the mine area enters the Edith River.

24.4 Regional Groundwater Model and Mine Dewatering

The Mt. Todd Gold Project will enlarge the existing Batman pit significantly below the water table. After the existing pit has been emptied, the pit is expected to require additional dewatering as mining progresses. Historical data indicate that the primary driver for dewatering will likely be runoff entering the pit from precipitation during the wet season, rather than groundwater inflow.

The following sections provide a brief summary of the applicable hydrogeologic information, historical observations, and conceptual pit inflow model. This information provides the basis for the dewatering cost estimate. Geologic information related to the geological setting, mineralization and exploration of the project site was presented in Sections 7.0, 8.0, and 9.0; the geologic information in this section is presented from a hydrogeologic perspective as it relates to groundwater flow and pit dewatering.

24.4.1 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[4] indicates that the formations in the vicinity of the BP are the Finniss River Group (Burrell Creek and Tollis Formations) and the Cullen Batholith (specifically the Yenberrie and Tennysons Leucogranites). The Finnis 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 Mt. Todd Gold 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.

[4] *National Geoscience Mapping Accord, Katherine (NT), Sheet SD 53-9, Second Edition, 1994.*

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[5]. These bores are located both near the pit and up to 4 km north and south of the pit. In addition, Vista has advanced a number of boreholes both for exploration and geotechnical evaluation. The borehole logs generally indicate that the upper 3 m are unconsolidated. Below that, weathering typically extends to approximately 30 m below ground surface (m bgs), with the degree of weathering decreasing with depth.

The Mt. Todd area experiences heavy rainfall during the wet season. The average rainfall is 1,129 mm/year, but more than 80% of it falls from December through March. Thus, anecdotally, sheet flow of precipitation runoff occurs as the thin crust of soil and alluvial material reaches saturation. During heavy rain events and for some time afterward, numerous ephemeral streams develop in the valleys. These subsequently stop flowing during the dry season.

The conceptual model of groundwater flow is that nearly all of the precipitation becomes runoff. Of the precipitation that does infiltrate, most flows within the upper 3 meters of unconsolidated material toward the nearest valley, where it feeds the stream system. Within the valleys, flow occurs as surface water in the streams and also within the thin layer of alluvium beneath and adjacent to the streams. Within bedrock, most water is believed to flow in the weathered profile, through fractures. The regional flow of groundwater is generally from higher to lower elevations.

24.4.2 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. Although preliminary calibration of the regional groundwater model has been completed, additional calibration is being conducted and the model has not yet been finalized. Thus, only preliminary estimates of groundwater inflow to the expanded Batman pit are currently available. The model will be finalized and used to generate updated estimates of dewatering flows and dewatering effects on the groundwater system for the feasibility study. The model will also be expanded to simulate post-mining recovery of the groundwater system.

For this PFS, Tetra Tech developed preliminary estimates of groundwater discharge into the pit based on preliminary model output coupled with historical observations. Historical observations are discussed in Section 24.4.2. Preliminary estimates from the groundwater modeling conducted to date suggest that groundwater inflows should initially be approximately 3 m^3/hr and will gradually increase as the pit is

[5] *Rockwater, 1994. Mt Todd Gold Mine, Bore Water Supply Expansion Programme Bore Completion Report.*

enlarged and deepened, reaching approximately 105 m^3/hr during the latter part of Phase IV. Under expected normal conditions, a portion of the groundwater inflow would be removed by evaporation from the pit walls and floor. Pit dewatering is expected to lower groundwater levels in the vicinity of the pit. The preliminary modeling suggests that dewatering-related water level declines of 1m or more should not extend farther than approximately 300 m from the pit.

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. Considering that no dewatering had been done in the intervening six years, groundwater inflow is expected to be small and therefore a relatively minor component of dewatering.

While the groundwater inflow component is expected to be relatively minor, precipitation during the wet season has historically been significant, especially on a short-term basis. Monthly reports prior to June 2000 indicate that on several occasions large storm events generated sufficient storm-water inflow to interrupt mine operations. One event in particular resulted in the pit floor being inaccessible for approximately a month (General Gold Operations Pty Ltd (GGO). 2000). Thus, a dewatering plan will be required to ensure that surface water runoff and precipitation inflows do not significantly hamper consistent mine operation.

24.4.3 INFLOW ESTIMATES

As noted above, groundwater inflow is expected to be a relatively minor component of dewatering. Therefore, for the PFS level estimate, groundwater inflow has been assumed to be negligible. However, the large amount of precipitation and storm-water runoff has historically been a cause for concern.

Thus, Tetra Tech based the conceptual dewatering plan on the 10-year recurrence interval, 72-hour and 100-year recurrence interval, 24-hour duration storm events. The precipitation values for those storm events, as obtained from the Bureau of Meteorology (BOM. 2012), are 3.47 mm/hr for 72 hours, which results in a total amount of 249.84 mm, and 10.7 mm/hr for 24 hours, which results in a total amount of 256.80 mm. The precipitation is assumed to fall uniformly over the pit and its catchment area. As the pit increases in size during mine development, the catchment area outside the pit would decrease until the pit comprises the entire drainage area in Phase II. Total volumes of storm water runoff inflow to the pit at the end of each phase of mine development are listed in **Table 24-1**. The volumes were calculated using the SCS Curve Number method (USDA 1996).

Table 24-1: Catchment and Pit Areas, Inflow Volumes, and Pumping Rates for Mine Dewatering Design

Mine Phase	Catchment Area (m^2)	Inflow Volume (m^3)	Design Pumping Rate (m^3/hr)
Phase I	489,510	120,654	1,014
Phase II	738,787	182,095	1,530
Phase III	972,235	239,635	2,014
Phase IV	1,242,534	306,258	2,574

24.4.4 Mine Dewatering

Dewatering of the proposed Mt. Todd mine is anticipated to be through passive collection of water in strategically placed sumps. The sumps would collect surface water pit wall run-off and precipitation, as well as any minor groundwater inflow encountered. Given the large volume of water expected from the design storm events, it is not currently anticipated that groundwater inflow from intercepting additional fractures and bedding planes will be significant relative to the effects of storm-water runoff and precipitation on design of the dewatering system. **Table 24-1** shows the design pumping rates for each phase of pit development. The rates are based on the design stipulation that all stormwater be removed from the pit within 48 hours of the end of the storm.

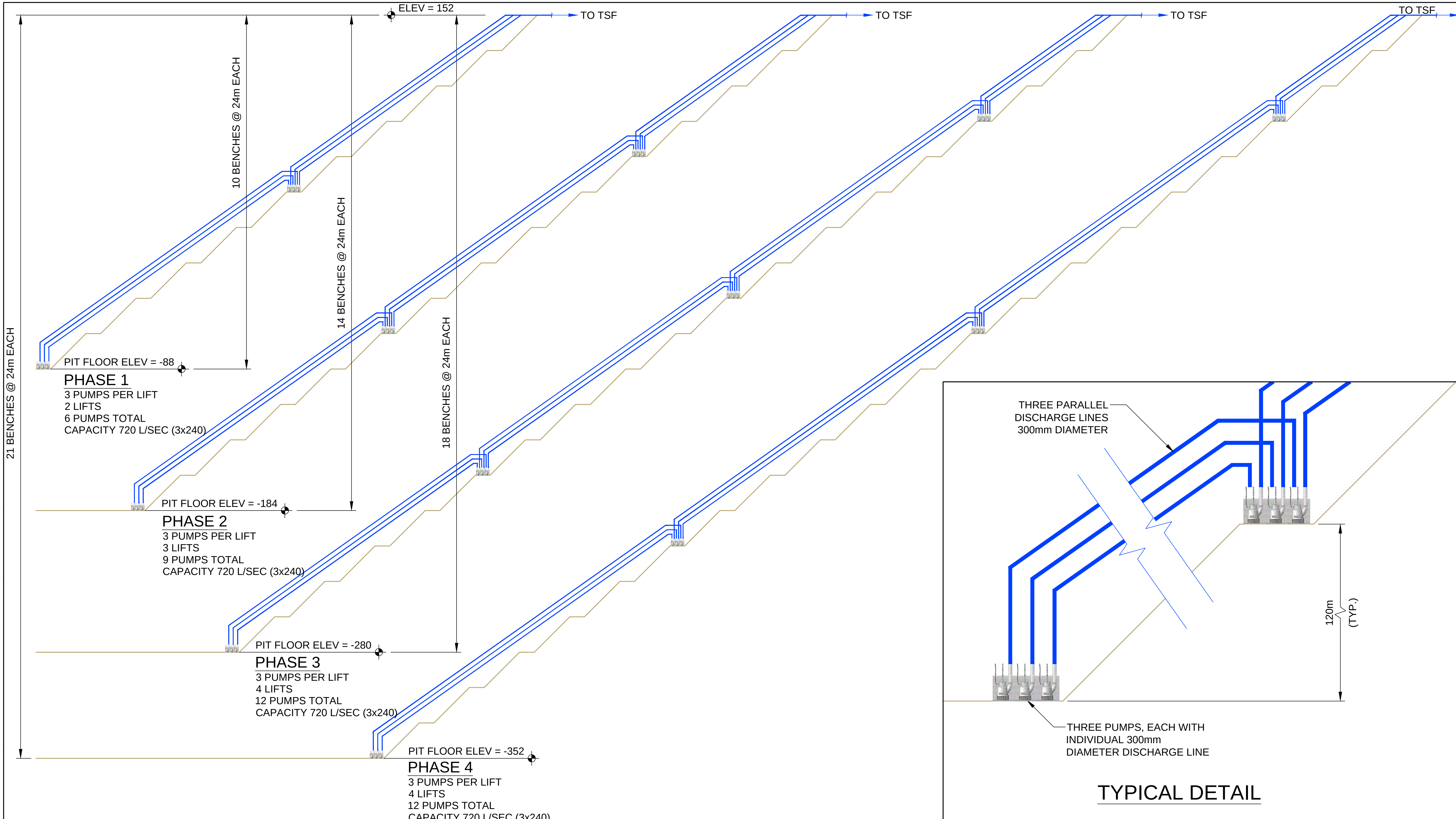
Sump water would be removed through pumping and discharge lines to the pit rim. Pumping from the sumps would be progressively increased with booster pumps added in stages with increasing pit depth. Once at the surface, the stormwater would be piped the TSF 1. **Figure 24-1** shows the sump pump, booster pump, and pipeline conceptual design, and **Figure 24-2** shows the conceptual layout of the dewatering system.

Total LoM costs for the development and operation of the open pit dewatering system as described are estimated to be:

- CAPEX: $8,869874; and
- OPEX: $2,006,780.

CAPEX costs are based on direct vendor quotes or Tetra Tech in-house estimates and include a contingency of 12%. OPEX costs are related to power consumption; no manpower expense is included, as supervision of the dewatering system is planned for existing mine and/or environmental staff and will not require dedicated personnel. Monitoring costs are included within the Environmental Program plan.

The mine dewatering system may require modification and refinement as empirical data become available during advanced exploration and initial mine construction and operation. In particular, groundwater-related mine inflow estimates should be refined based on numerical model updates incorporating observed groundwater inflow rates to the pit and observed water level changes in groundwater monitoring bores at the site.



NO	DATE	DESCRIPTION	PREPARED	REVIEW	DESIGN	AUTHORIZE
REFERENCE DRAWINGS			DRAFTING		ENGINEERING	

SUB CONSULTANT(S):

PERMIT STAMP

SEAL

DESIGNED BY:	PREPARED BY:	REVIEWED BY:
R. HOSIER	D. MONTEZ	K. THOMPSON
AUTHORIZED BY:	DATE: June 2013	SCALE: NONE

CLIENT:

VISTA GOLD

TETRA TECH

PROJECT NAME:
MT. TODD GOLD PROJECT

DRAWING DESCRIPTION:
FIGURE 24.6-1
OPEN PIT DEWATERING CONCEPTUAL DESIGN

DRAWING NO:

REV: 0

ANSI D (22x34)

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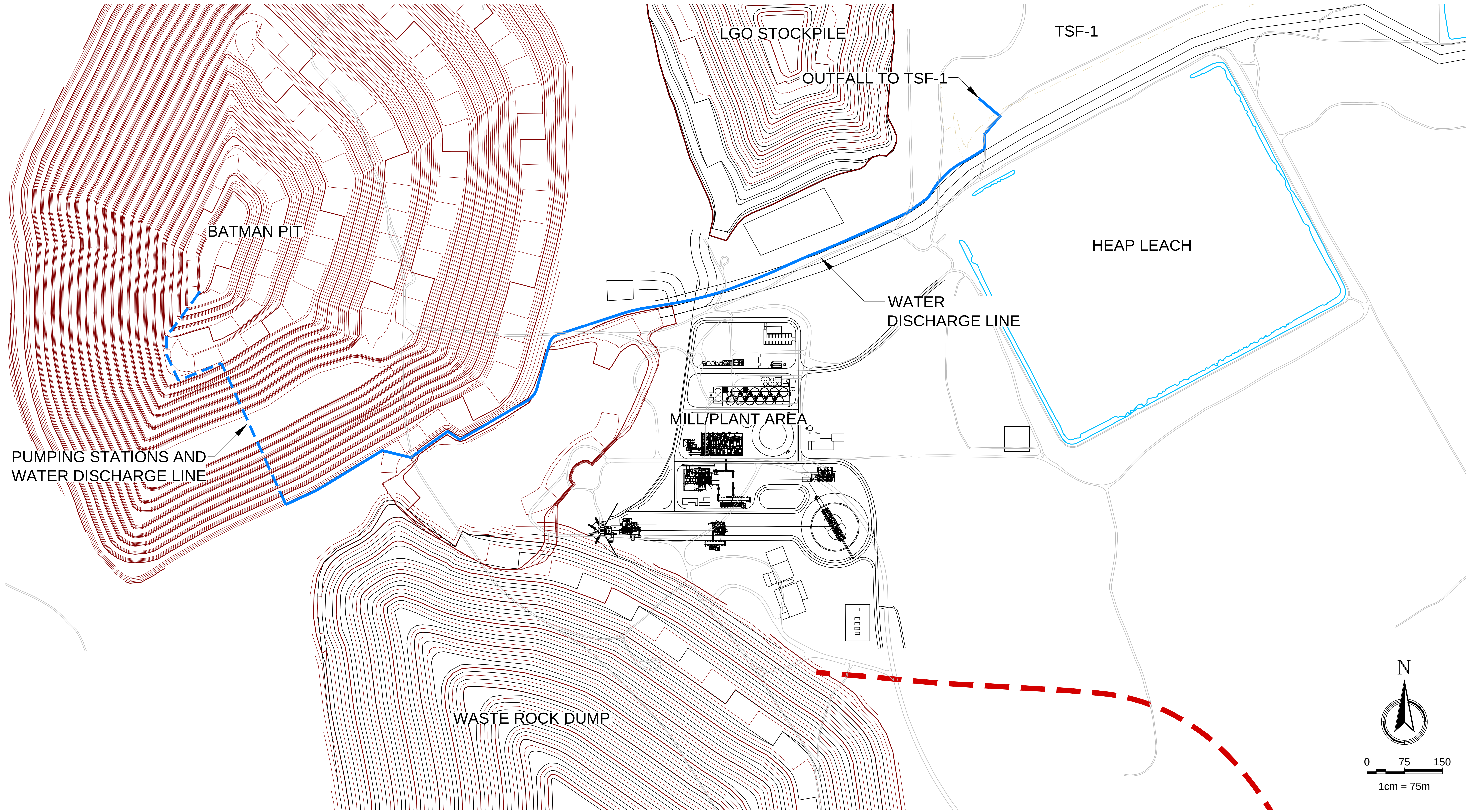

NO	DATE	DESCRIPTION	PREPARED	REVIEW	DESIGN	AUTHORIZE
REFERENCE DRAWINGS			DRAFTING		ENGINEERING	

SUB CONSULTANT(S):

PERMIT STAMP

SEAL

DESIGNED BY:	PREPARED BY:	REVIEWED BY:
R. HOSIER	D. MONTEZ	K. THOMPSON
AUTHORIZED BY:	DATE: January 2013	SCALE: AS SHOWN

CLIENT:

VISTA GOLD

TETRA TECH

PROJECT NAME:
MT. TODD GOLD PROJECT

DRAWING DESCRIPTION:
FIGURE 24.6-2
OPEN PIT4DEWATERING SYSTEM
CONCEPTUAL LAYOUT

DRAWING NO:

REV:
0

ANSI D (22x34)

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24.5 Project Implementation

24.5.1 Project Implementation Strategy

This section outlines a high level Project Implementation Strategy, which will be further developed during the next study phase of the Project.

The PFS definitions of scope, cost and schedule have been established on the presumption that Vista will implement the Project utilising the engineering, procurement and construction management (EPCM) execution model.

Vista will appoint an EPCM contractor (Engineer) with the prerequisite capability and experience to undertake the work.

To complement the EPCM approach, Vista will adopt design and construct (D&C) implementation strategies, for select areas of the Project.

Properly executed, the EPCM Execution strategy will afford Vista the following benefits:

a) Lower capital cost outcomes;
b) Project implementation flexibility;
c) Fast-track execution opportunities;
d) Flexible project funding strategies; and
e) Optimal project quality outcomes.

24.5.2 Project Organization

EPCM Contracts

Vista's Project Manager will direct all activities including EPCM and D&C contractors.

For the EPCM Scope, two organization charts are developed:

a) EPCM Stage 1 – Design & Procure. Refer Diagram 1
b) EPCM Stage 2 – Construct & Commission. Refer Diagram 2

The proposed EPCM scope is for services in relation to the process plant and other items as defined in Items 17.0 and 18.0.

Figure 24-3: EPCM Stage 1 – Design & Procure. Refer Diagram 1

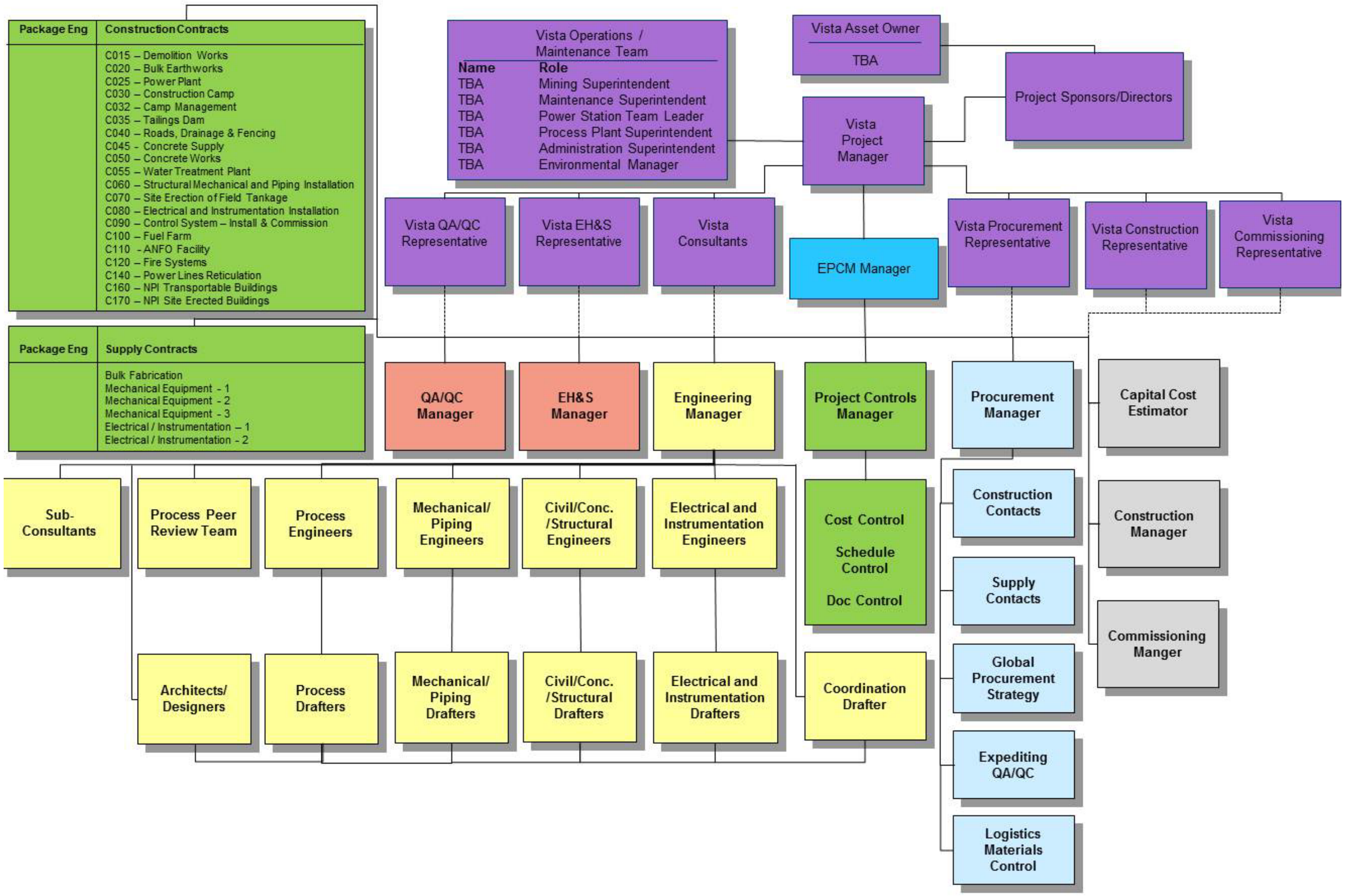
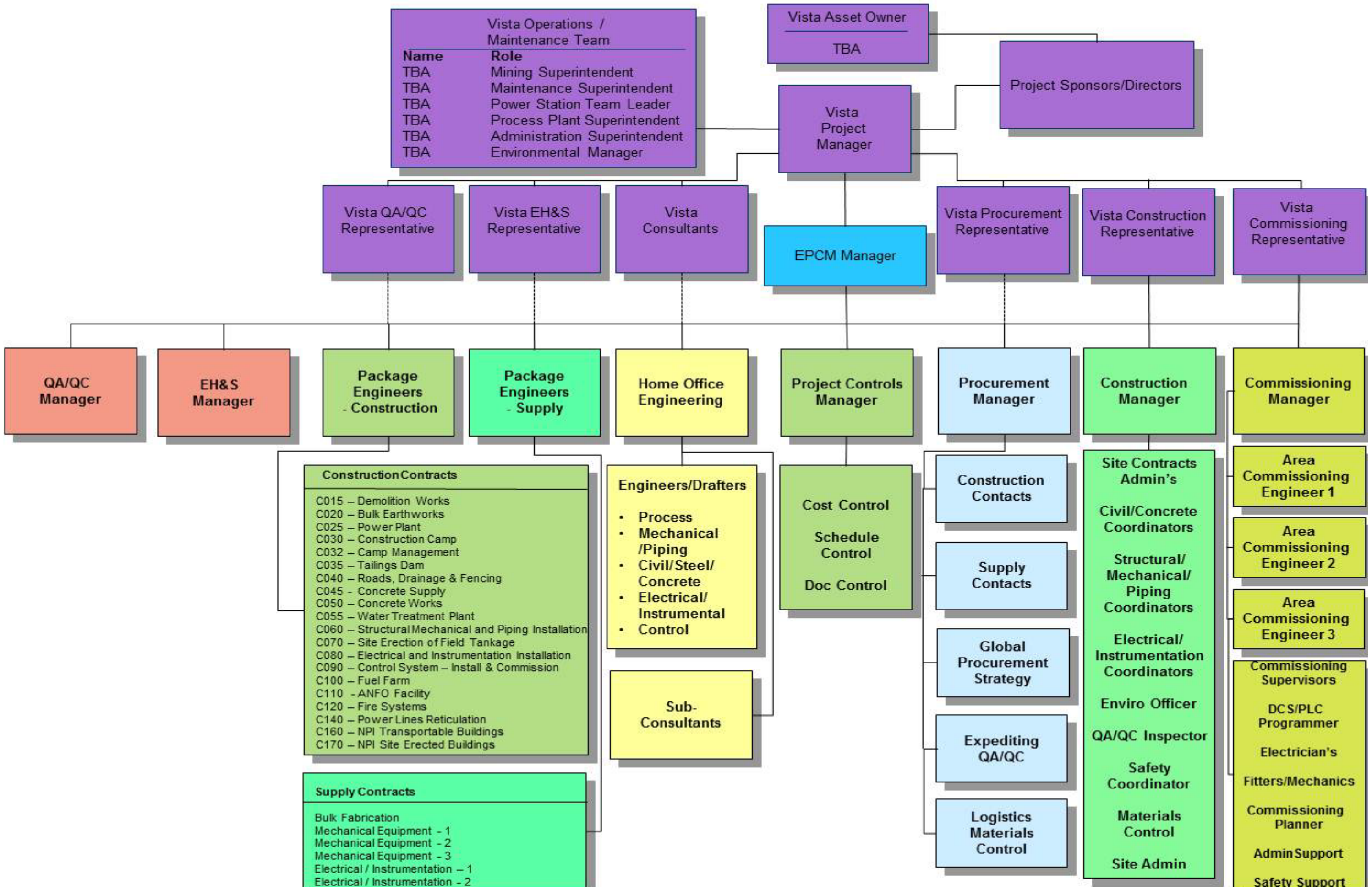


Figure 24-4: EPCM Stage 2 – Construct & Commission. Refer Diagram 2



D&C Contracts

Two D&C contracts are proposed, specifically:

a) NPI – Transportable Buildings (Package C160)

b) NPI – Site Erected Buildings (Package C170)

EPCM Contract Scope of Services

Generally, the Engineer will perform the following tasks:

a) Detailed process, civil, structural, mechanical and electrical design

b) Preparation of specification documentation

c) Calling and review of tenders for supply and installation of equipment

d) Contract evaluation, negotiations and documentation

e) Preparation of Purchase Orders and Contracts

f) Quality audits of major contractors and manufacturers

g) Equipment and site inspections

h) Cost control, procurement, scheduling and planning, contract administration

i) Regular reporting on progress against schedule and cost against budget

j) Site testing and commissioning

k) Preparation and review of Operation and Maintenance manuals

24.5.3 PROCUREMENT

Procurement Strategy

The key procurement aims and objectives are to:

a) Contribute to achieving the project objectives of earliest possible completion, cost-effective execution, quality workmanship and high degree of safety from suppliers.

b) Adhere to the project plan, aims and schedule.

c) Ensure that commercial and schedule risks are at acceptable levels.

d) Provide a purchasing environment that minimises claims and protracted disputes.

e) Provide a procurement arrangement that encourages suppliers to be innovative and efficient.

f) Carry out the procurement function for the project in an ethical and professional manner.

Key success factors are to:

a) Meet or exceed expectations for health and safety requirements.

b) Meet or better the project schedule.

c) Meet or better the project budget.

d) Meet project quality objectives.

Construction Packages

The Engineers Procurement Manager will be responsible for the development of a construction contracting strategy.

A preliminary strategy is documented in the contracting and procurement plan.

The following construction packages are envisaged as a minimum:

Table 24-2: Construction Packages

Package No	Package Description
C015	Demolition Works
C020	Bulk Earthworks
C025	Power Plant
C030	Construction Camp
C032	Camp Management
C035	Tailings Dam
C040	Roads, Drainage & Fencing
C045	Concrete Supply
C050	Concrete Works
C055	Water Treatment Plant
C060	Structural Mechanical and Piping Installation
C070	Site Erection of Field Tankage
C080	Electrical and Instrumentation Installation
C090	Control System - Install & Commission
C100	Fuel Farm
C110	ANFO Facility
C120	Fire Systems
C140	Power Lines Reticulation
C160	NPI Transportable Buildings
C170	NPI Site Erected Buildings
C180	Communications - Telstra Interface
C190	Communications - Temporary

Supply Packages

The Engineers Procurement Manager will be responsible for the development of an Equipment and Services Supply Contracting strategy. A preliminary strategy is documented in the Contracting and Procurement Plan.

The following supply packages are envisaged as a minimum:

Table 24-3: Supply Packages

Package No	Package Description
P001	Ball Mill
P002	Primary Crusher
P003	Secondary Crusher
P004	HPGR
P005	Dry Screens
P006	Wet Screens
P007	Slurry Pumps

Table 24-3: Supply Packages

Package No	Package Description
P008	Solution Pumps
P009	Apron Feeder
P010	Belt Feeder
P011	Cyclone Cluster
P012	Agitators
P013	Thickener
P014	Inter Tank Screens
P015	Carbon Transfer Pump
P016	Gold Room
P017	Vibrating Feeders
P018	Container Tipper
P020	Flocculant Mixing Package
P021	Lime Slaker
P023	Potable Water Plant
P024	Mill Relining Machine
P025	Overhead Travelling Cranes
P026	Air Compressors, Driers & Receivers
P028A	Conveyor Drives
P028B	Conveyor Pulleys
P028C	Conveyor Idlers
P028D	Conveyor Belts & Splicing
P028E	Conveyor Skirts
P028F	Conveyor Scrapers & Ploughs
P031	Wet Scrubber
P032	Isolation Gates
P033	Ventilation Fans
P034	Screw Feeders
P035	Rotary Valves
P036	Filters
P038	Hoists
P039	Ball Charging Magnet
P040	Tramp Magnets
P041	Sump Pumps
P042	Firewater System
P043	Weightometer
P045	Samplers
P046	Analysers
P047	Rock Breaker
P048	Blowers - Detox
P049	Metal Detectors
P050	FRP Tanks
P051	Winches
P053	Manual Valves
P054	Laboratory Equipment
P055	Bag Splitters
P056	Safety Showers
P057	Pressure Relief Valves
P058	Pressure Regulators
P060	Weighbridge
P101	HV Switchgear
P102	HV Cables
P103	Transformers

Table 24-3: Supply Packages

Package No	Package Description
P104	Motor Control Centers
P105	HV Variable Speed Drives
P106	Neutral / Earth Resistors
P107	Overhead Power Lines
P108	Control System - Supply
P109	Instruments
P110	Switchrooms/MCC's
P111	LV Variable Speed Drives
P112	Power Factor Correction / Harmonic Filters
P113	Control Valves
P114	CCTV
P115	2 way Radio
P116	Plant Fire Detection System
P117	RMU's/Kiosk Substations
P118	Spare
P119	Telemetry
P120	Emergency Power
P121	Security
P122	UPS
P123	WAD Cyanide Analysers
P124	HCN Monitors
P125	Data Room
P126	Motors
P200	Fabricated Structural Steel Work
P210	Fabricated Platework
P220	Fabricated Site Erected Tankage
P230	Fabricated Pipe Work

Indirect Packages

The Engineer's Procurement Manager, in collaboration with Vista, will establish and manage a series of Indirect Packages.

The Indirect Packages are envisaged as a minimum:

a) EPCM Services
b) Environmental Consultants
c) HR & IRC Consultants
d) HSEC Consultants
e) Commissioning
f) Licensee, Fees and Legals
g) Project Insurances
h) Pre-Production Costs
i) Capital Spare
j) Stores and Inventories
k) Heavy Lift Cranage

24.5.4 Schedule

Schedule Objectives and Scope

The key objective of the PFS phase EPCM schedule is to provide a Class 3, Level 3 detail Schedule with an accuracy range of ±15%.

Class of Schedule defines the degree of completeness required for schedule development, Class 5 being a low degree of completeness, and Class 1 being a high degree of completeness. Level of Schedule defines the degree of detail for communication, reporting, and execution, Level 1 being a low degree of detail and Level 5 being a high degree of detail.

The scope included in the Schedule is that which is included in the EPCM contractors scope, as defines in the PFS. Consequently, Client Activities, Mine Development, Tailings Dam, Power plant detail, or Waste Water Treatment Plant are excluded from Schedule.

Schedule Assumptions

For the Project, the specific schedule assumptions include:

a) The Northern Territory wet season runs from ~1st December to ~17th April when heavy rains can impact construction activities at times, particularly civil and concrete works
b) No disruptions to scheduled work (IR or otherwise)
c) Open access to all work fronts is available
d) Transportation to and from site (both air and land) is without delay
e) The schedule has assumed that project approval will be given by Vista on or about October 1, 2013. Start up, as defined by handover after completion of commissioning is scheduled to commence early 2016

Critical Activities

The Critical Path of the EPCM Schedule runs through the Vista approval process and the purchase packages and contracts for Area 3300 (Classification and Grinding) as follows:

- P001 Ball Mills Scope Development and Tender Period 11 weeks
- P001 Ball Mills Manufacture and Delivery 79 weeks
- P001 Ball Mills SMP Construction 36 weeks (Total)
- Area 3300 Verification and Commissioning 5 weeks

The above critical activities determine a critical path of 119 weeks duration after Project approval to proceed has been given.

A copy of the Critical Path schedule has been attached in Appendix M.3 and M.6 for the respective throughput cases.

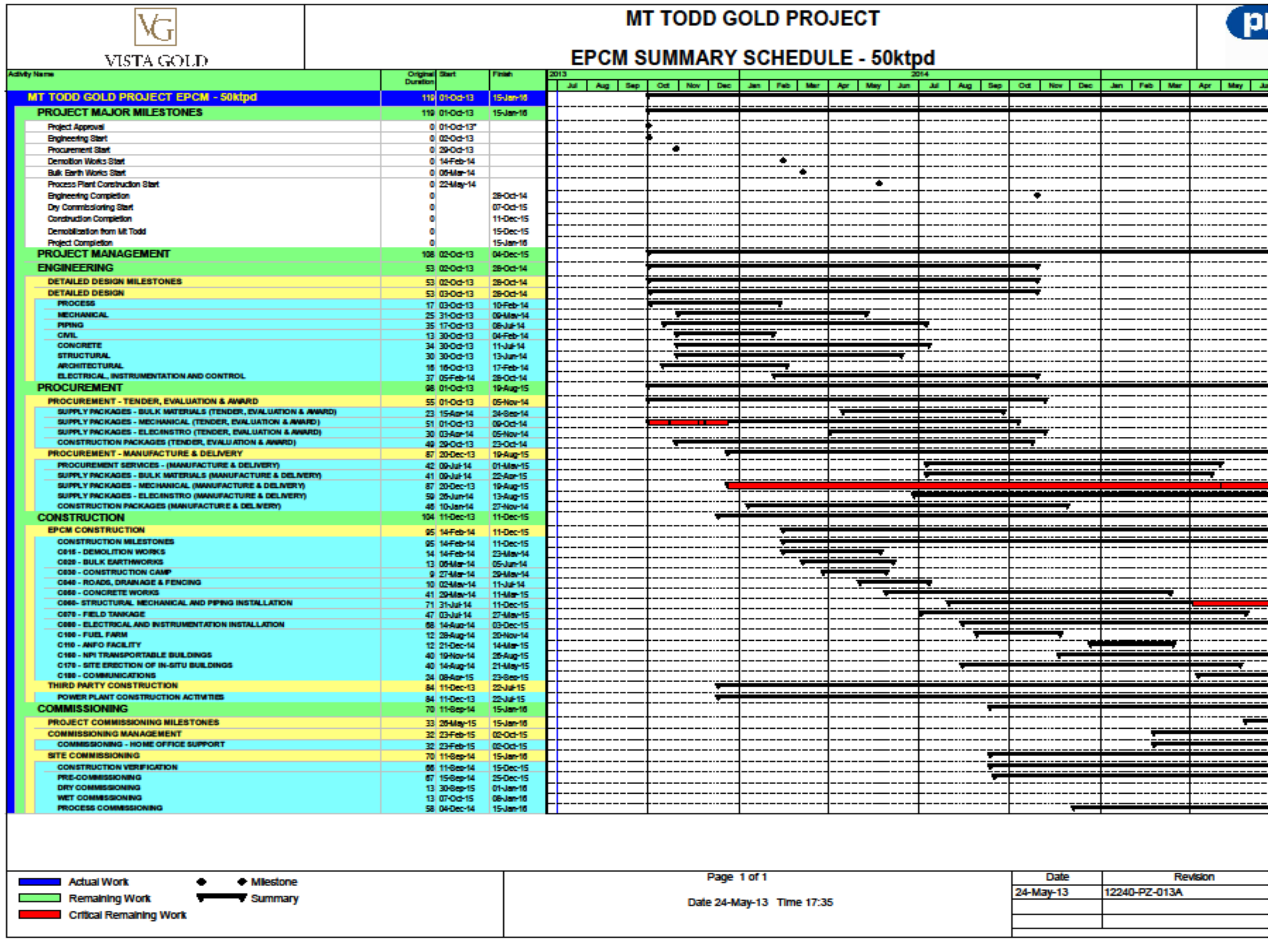
Significant Activities

Major procurement packages with a lead time ex-works greater than 40 weeks are:

Table 24-4: Supply Packages with Significant Lead Times

Package	Lead Time
P001 – Ball Mills	71 weeks
P002 – Primary Crusher	62 weeks
P003 – Secondary Crusher	56 weeks
P004 – HPGR's	62 weeks
P006 – Wet Screens	40 weeks
P009 – Apron Feeder	48 weeks
P012 - Agitators	49 weeks
P013 - Thickener	44 weeks
P021 – Lime Slaker	40 weeks
P024 – Mill Relining Machine	52 weeks
P110 – Switchrooms	52 weeks

Figure 24-5: EPCM Summary Schedule



24.6 ALTERNATE CASE

Vista also prepared an Alternate Case which considers a smaller and higher-grade project. Key differences between the Base Case and the Alternate Case include:

- A 33,000 tpd milling facility vs. a 50,000 tpd facility with associated lower mining rates and a smaller mining fleet; and
- A pit design based on an optimized shell calculated at US$925/oz-Au vs. US$1,360/oz-Au in the 50,000 tpd option and the application of a higher cut-off grade (0.45 g Au/t vs. 0.40 g Au/t).
- Shorter LoM (11 years).

Results of the Alternate Case are presented in this Section.

24.6.1 MINING

Mineral Resource

The same Mineral Resource, as used in the Base Case is used in the Alternate Case.

Mining

Pit Optimization

Separate pit designs were completed for the Base and Alternate Cases. Reserves are based on the 50,000 tpd ultimate pit design as this pit is larger and encompasses all material inside of the Alternate Case ultimate pit design. Economic Parameters are shown in **Table 24-5**.

Table 24-5: Economic Parameters, Alternate Case

	Alternate Case
Gold Recovery	82% Sulfide 78% Transition 78% Oxide
Payable Gold	99.9%
Overall Mining Cost	$2.16 per tonne
Processing Cost	$10.330 per tonne processed
Tailings	$0.89 per tonne processed
Water Treatment	$0.09 per tonne processed
Royalty	1% NPR (Jawoyn)

As with the Base Case, the mining cost was varied using an additional US$0.010 per each 6 m bench below the 145 m elevation. The reference mining cost of US$1.86 was used for the Alternate Case. Processing, tailings construction, tailings reclamation, and water treatment costs were provided by Vista and based on previous studies. Note that site G&A costs are included with the processing cost

A minimum cutoff grade of 0.45 g Au/t was used for 33,000 tpd. This was done to maintain higher grades with respect to material allowed to be processed.

Pit optimizations were completed using prices of US$300 to US$2,000 per ounce Au in increments of US$20 per ounce in order to analyze the deposit's sensitivity to gold prices for both scenarios. Results for US$100 per ounce increments from US$300 to US$2,000 per ounce of gold are shown in **Table 24-6**.

Table 24-6: Whittle Pit Optimization Results – Alternate Case Using 0.45 g Au/t Cutoff

Gold Price (US$)	Material Processed			Waste (tonnes)	Total (tonnes)	Strip (ratio)
	K Tonnes	*g Au/t*	*K Ozs Au*			
300	360	1.94	22	144	505	0.40
400	5,226	1.70	286	4,477	9,704	0.86
500	9,613	1.53	474	9,377	18,991	0.98
600	15,467	1.36	676	15,728	31,196	1.02
700	26,362	1.20	1,015	31,043	57,405	1.18
800	77,682	1.04	2,602	163,659	241,341	2.11
900	101,864	0.98	3,208	207,889	309,753	2.04
1,000	122,360	0.93	3,674	247,275	369,635	2.02
1,100	143,442	0.92	4,231	336,378	479,821	2.35
1,200	156,391	0.91	4,581	407,222	563,613	2.60
1,300	172,691	0.90	5,021	509,672	682,363	2.95
1,400	185,349	0.90	5,341	591,157	776,506	3.19
1,500	196,248	0.89	5,625	675,062	871,310	3.44
1,600	203,535	0.89	5,814	738,269	941,805	3.63
1,700	208,525	0.89	5,947	787,999	996,524	3.78
1,800	209,602	0.89	5,972	798,237	1,007,840	3.81
1,900	212,661	0.89	6,058	835,862	1,048,523	3.93
2,000	213,613	0.89	6,085	849,442	1,063,056	3.98

Graphs of the Whittle results are shown in **Figure 24-6**.

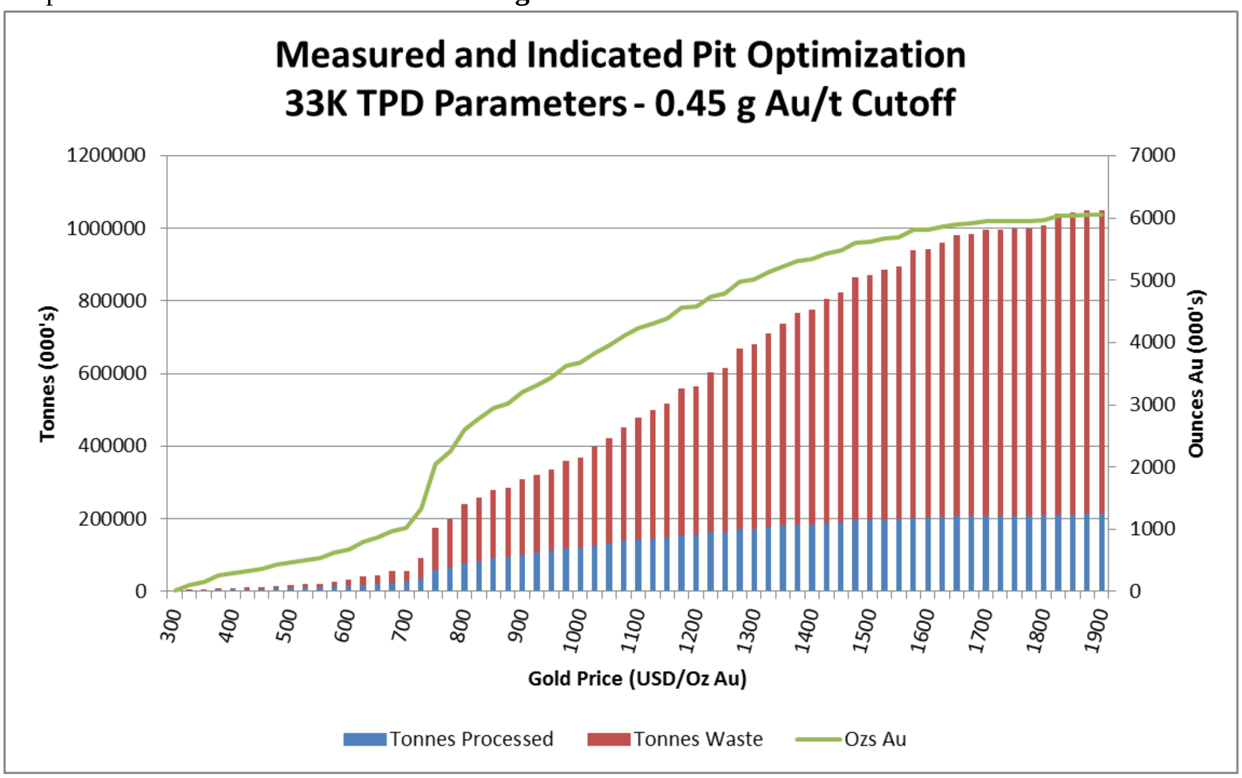


Figure 24-6: Graph of Whittle Results – Alternate Case Using 0.45 g Au/t Cutoff

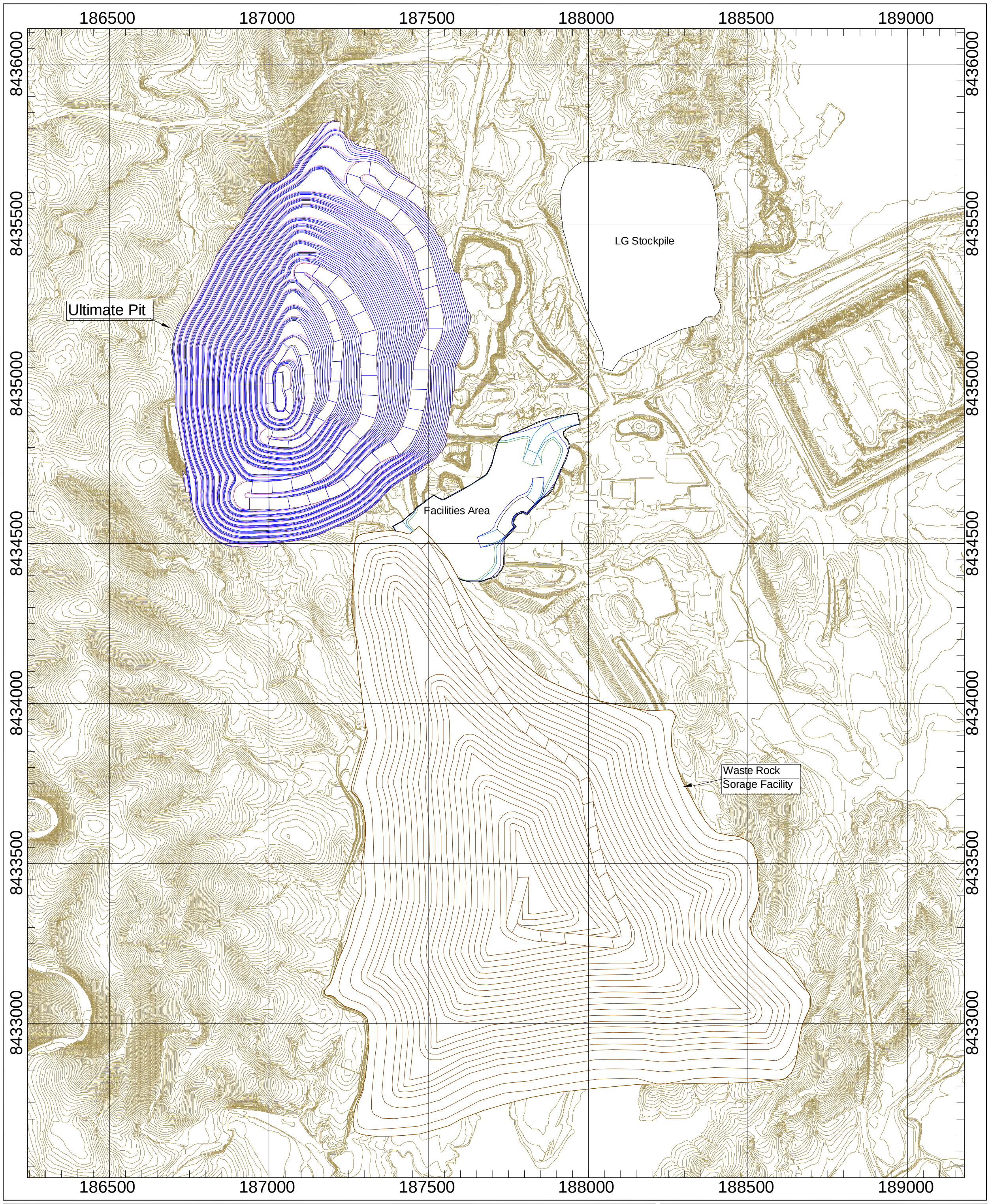
The ultimate pit limit for the Alternate Case was based on trying to reduce the capital requirements and the general overall size of the project. The ultimate pit limit is based on the 33,000 tpd pit shell optimized at a US$925 per ounce Au price using a 0.45 g Au/t cutoff grade.

Pit Designs

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

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

The ultimate pit design along with the ultimate dump and stockpile designs and planned infrastructure is shown in **Figure 24-7**.



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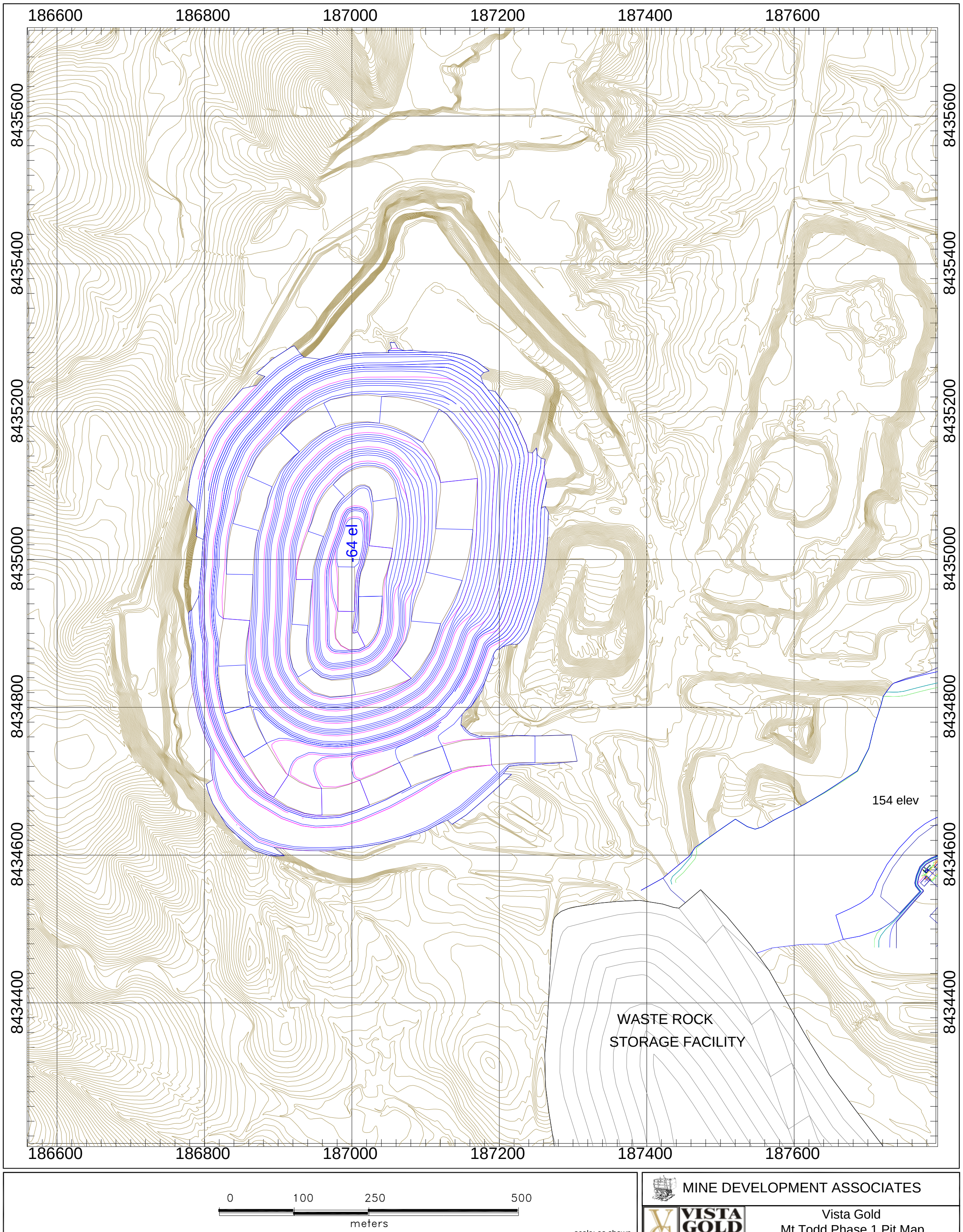
MINE DEVELOPMENT ASSOCIATES

VISTA GOLD CORP.

Vista Gold
Mt Todd Ultimate Pit Map
Alternate Case

Separate phase designs for the Alternate and Base Cases were created. The Alternate Case Phase I design essentially continues the western wall down from existing mining that was done by prior operators and leaves a ramp on the east side of the pit. Phase II expands the pit in almost all directions, though there is a bit of a common wall with Phase I on the west side of the pit. Phase III mines around Phase II in all directions, but has a common wall with the ultimate pit on the west.

Figures 24-8 to **24-10** show the Alternate Case Phase I, II, and III pit designs. The Alternate Case Phase IV design is depicted in **Figure 24-7** as the ultimate pit. Resulting reserves for each of the phases are shown in **Table 24-8**.



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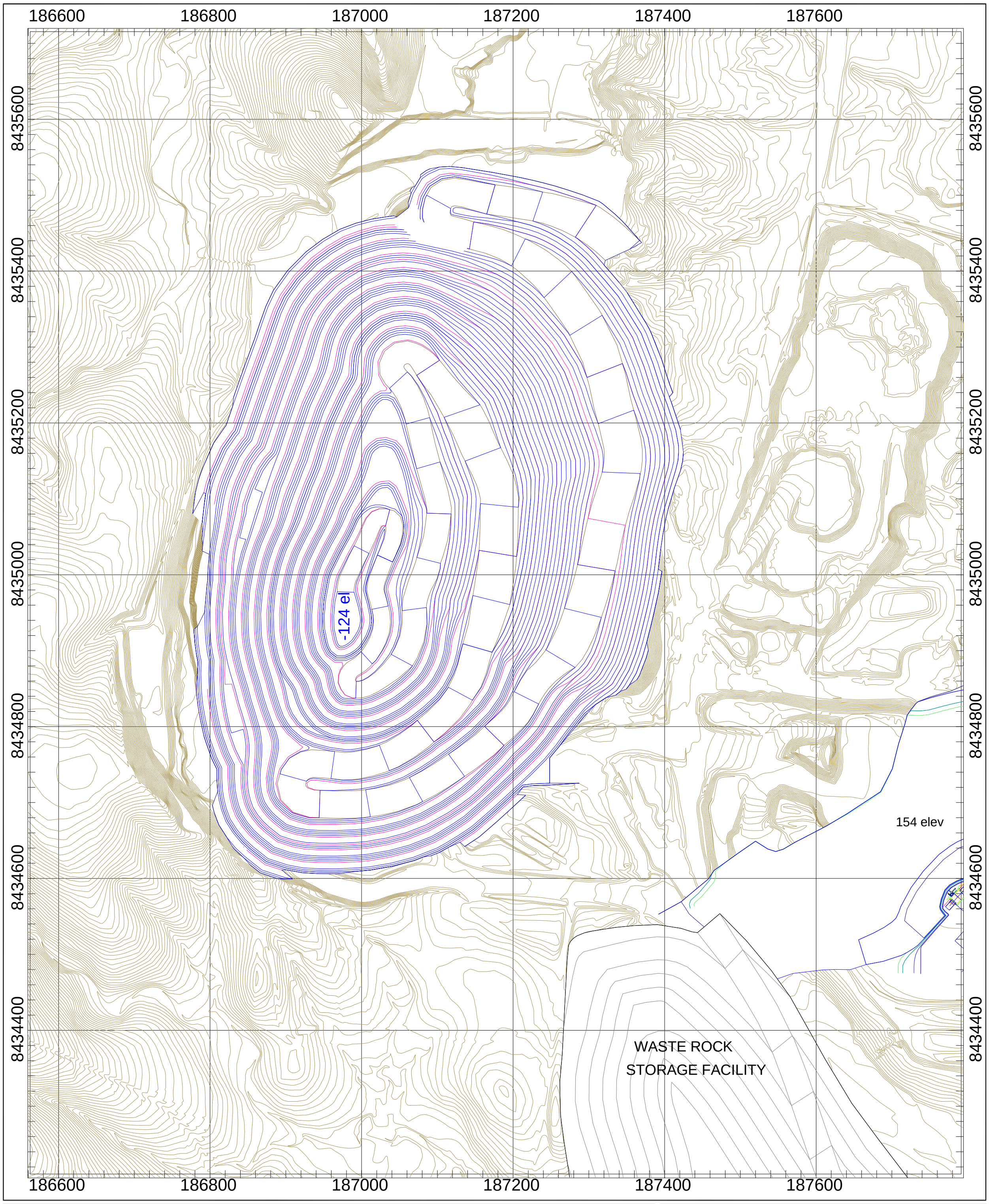
MINE DEVELOPMENT ASSOCIATES

VISTA GOLD CORP.

Vista Gold
Mt Todd Phase 1 Pit Map
Alternate Case



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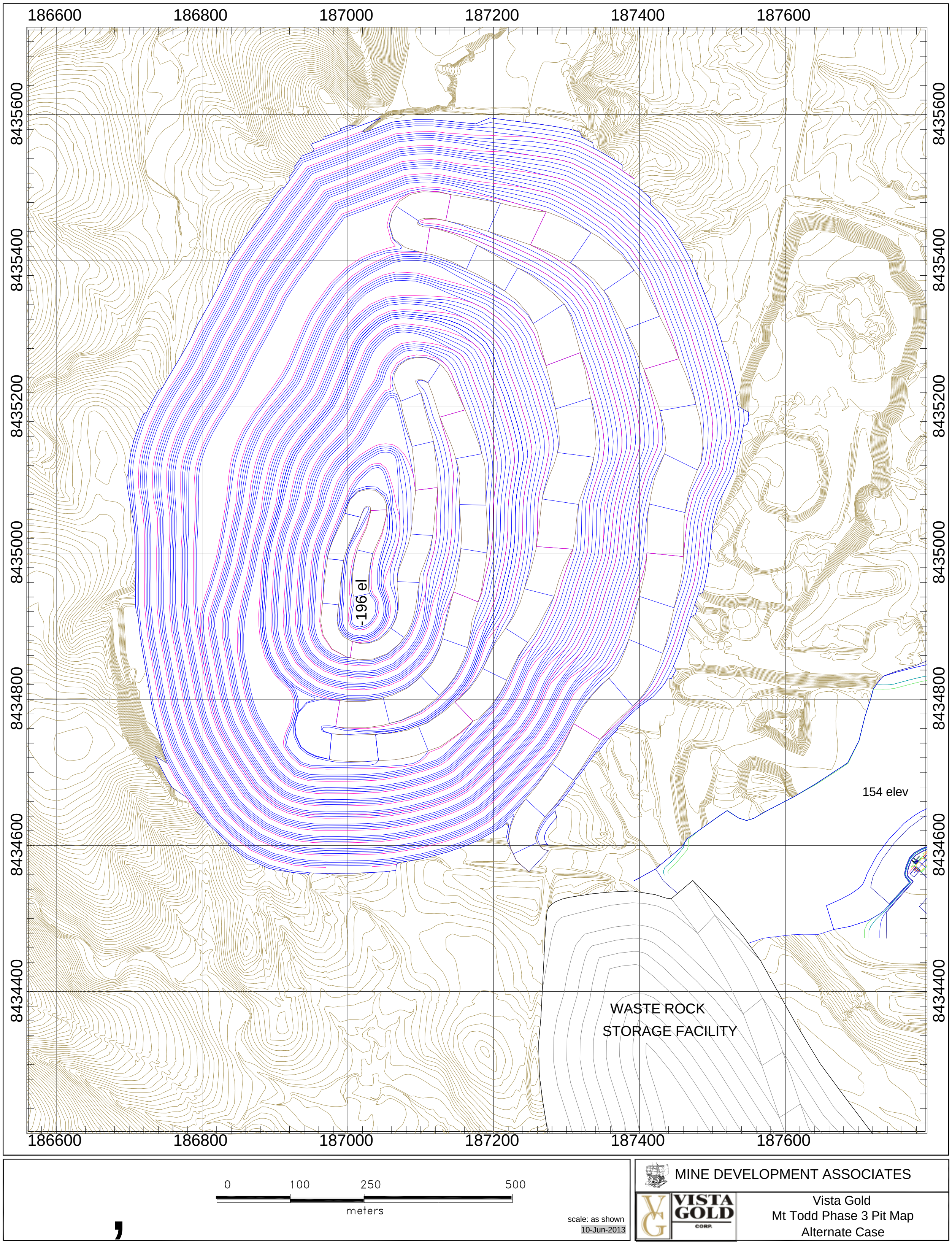
MINE DEVELOPMENT ASSOCIATES

VISTA GOLD CORP.

Vista Gold
Mt Todd Phase 2 Pit Map
Alternate Case



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MINE DEVELOPMENT ASSOCIATES

VISTA GOLD CORP.

Vista Gold
Mt Todd Phase 3 Pit Map
Alternate Case

Cut-Off Grade

The breakeven and internal cutoff grades calculated using the economic parameters shown in **Table 24-5** are shown in **Table 24-7**. The internal cutoff grade assumes that mining is constrained to an economic pit and does not include the mining cost.

To enhance projects economics, Vista used a higher cutoff grade for reserves and scheduling than what operating costs would have predicted. Reserves are reported using a 0.45 g Au/t cutoff grade for 33,000 tpd.

Table 24-7: $1,360 Calculated Gold Price Cutoff Grades (g Au/t)

	Sulfide	Transition	Oxide
Breakeven	0.38	0.40	0.40
Internal	0.32	0.34	0.34
Cutoff Grade Used	0.45	0.45	0.45

For purposes of production scheduling, low-grade, medium-grade, and high-grade material was designated. The low-grade material used a 0.45g Au/t cutoff grade. Medium-grade and high-grade cutoffs used was 0.75 and 1.00 g Au/t.

Mineral Reserves

Mineral Reserves are shown in **Table 24-8**. Note that these Proven and Probable Reserves are contained within the Base Case pit designs used to state the full official reserves. Thus, the Alternate Case Reserves are a subset of the official reserves.

Table 24-8: Alternate Case Proven and Probable Reserves by Phase

	Proven K Tonnes	g Au/t	K Ozs Au	Probable Tonnes	g Au/t	K Ozs Au	Total P&P Tonnes	g Au/t	K Ozs Au	Waste Tonnes PAG_Wst	Un_Wst	NonPag_Wst	Total	Total Tonnes	Strip Ratio
Ph_1	13,954	1.10	494	6,078	1.14	224	20,032	1.11	718	14,499	3,032	1,068	18,599	38,631	0.93
Ph_2	14,750	0.83	394	14,857	0.92	439	29,607	0.87	832	26,707	6,568	10,159	43,434	73,041	1.47
Ph_3	13,177	0.90	382	16,898	0.91	496	30,075	0.91	878	39,479	16,617	23,300	79,396	109,471	2.64
Ph_4	11,704	0.93	349	18,957	0.91	553	30,660	0.92	902	31,443	13,766	36,094	81,303	111,963	2.65
Total	**53,584**	**0.94**	**1,619**	**56,789**	**0.94**	**1,711**	**110,374**	**0.94**	**3,330**	**112,127**	**39,984**	**70,621**	**222,732**	**333,106**	**2.02**

The 33,000 tpd reserves are reported using a cutoff grade of 0.45 g Au/t and are a subset of the Proven and Probable Reserves.

Inferred Mineral Resources

Table 24-9 shows the inferred resources inside of the Alternate Case pit designs for each phase.

Table 24-9: In-Pit Inferred Resources

	K Tonnes	g Au/t	K Ozs Au
Phase I	1,930	0.65	44
Phase II	3,172	0.56	68
Phase III	1,021	0.62	22
Phase IV	1,968	0.67	40
Total	**8,092**	**0.62**	**175**

(1) Inferred resources are reported using cutoff grade of 0.4 g Au/t
(2) Mineral resources that are not mineral reserves have no demonstrated economic viability

Mine Waste Facilities

Total contained waste tonnage is 223 Mt for the Alternate Case. The current waste facility is approximately 24 m high located to the southeast of the pit. The ultimate dump design is about 175 m. A 40% swell factor and an average specific gravity of 2.67 (bank) have been assumed for volume calculations. The 33,000 tpd waste dump design has a capacity to contain approximately 188 Mt. The remaining waste material will be used for tailings construction and reclamation.

Mine Production Schedule

Table 24-10 shows the mine-production schedule, including re-handle from stockpiles. For purpose of production scheduling, low-grade, medium-grade, and high-grade material was designated. The low-grade material used a cutoff grade of 0.45 g Au/t. Medium-grade and high-grade cutoffs used were 0.75 and 1.00 g Au/t.

Table 24-10: Annual Mine Production Schedule – Alternate Case

			Pre-Prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Total
Total Mined	Pit to Stockpile	K Tonnes	3,407	6,343	7,020	7,382	3,679	5,921	5,596	-	-	850	-	-	40,199
		g Au/t	0.79	0.63	0.58	0.60	0.57	0.58	0.69	-	-	0.68	-	-	0.63
		K Ozs Au	87	129	132	143	68	111	124	-	-	19	-	-	812
	Pit to Crusher	K Tonnes	-	10,529	4,993	10,392	1,243	4,410	11,715	2,681	8,501	11,747	3,964	-	70,175
		g Au/t	-	1.38	1.15	1.23	1.00	1.07	1.17	0.66	0.79	0.99	1.38	-	1.12
		K Ozs Au	-	466	184	410	40	152	442	57	216	375	176	-	2,518
	Total Ore Mined	K Tonnes	3,407	16,872	12,013	17,775	4,921	10,331	17,311	2,681	8,501	12,597	3,964	-	110,374
		g Au/t	0.79	1.10	0.82	0.97	0.68	0.79	1.02	0.66	0.79	0.97	1.38	-	0.94
		K Ozs Au	87	595	316	554	107	263	566	57	216	394	176	-	3,330
	Mineralized Waste	K Tonnes	332	1,227	1,546	1,216	1,022	1,361	1,233	696	1,281	946	17	-	10,876
		g Au/t	0.43	0.42	0.43	0.42	0.43	0.43	0.42	0.42	0.42	0.42	0.43	-	0.42
		K Ozs Au	5	17	21	17	14	19	17	9	17	13	0	-	149
	Non-PAG Waste	K Tonnes	1,059	6,440	3,728	9,188	12,130	1,981	14,504	18,835	2,752	2	-	-	70,621
	PAG Waste	K Tonnes	5,194	12,335	14,715	9,042	14,154	15,181	5,647	6,356	13,727	4,868	33	-	101,251
	Undefined Waste	K Tonnes	1,899	3,713	3,621	3,514	7,885	5,539	2,550	5,743	5,129	392	-	-	39,984
	Total Waste Mined	K Tonnes	8,483	23,714	23,611	22,960	35,191	24,062	23,934	31,629	22,889	6,209	49	-	222,732
	Total Tonnes Mined	K Tonnes	11,891	40,587	35,624	40,735	40,112	34,393	41,245	34,310	31,390	18,806	4,014	-	333,106
	Strip Ratio	W:O	2.49	1.41	1.97	1.29	7.15	2.33	1.38	11.80	2.69	0.49	0.01		2.02
Re-Handle Material	High-Grade Stockpile	K Tonnes	-	665	237	89	-	-	-	434	-	-	68	-	1,493
		g Au/t	-	1.27	1.30	1.25	-	-	-	1.34	-	-	1.25	-	1.30
		K Ozs Au	-	27	10	4	-	-	-	19	-	-	3	-	62
	Medium-Grade Stockpile	K Tonnes	-	554	803	63	-	-	-	747	-	-	145	-	2,312
		g Au/t	-	0.85	0.85	0.84	-	-	-	0.85	-	-	0.86	-	0.85
		K Ozs Au	-	15	22	2	-	-	-	20	-	-	4	-	63
	Low-Grade Stockpile	K Tonnes	-	-	5,682	1,171	10,472	7,337	-	7,853	3,214	-	666	-	36,394
		g Au/t	-	-	0.59	0.59	0.59	0.58	-	0.59	0.59	-	0.58	-	0.59
		K Ozs Au	-	-	107	22	197	137	-	149	61	-	12	-	686
	Total Re-Handle	K Tonnes	-	1,218	6,722	1,322	10,472	7,337	-	9,034	3,214	-	878	-	40,199
		g Au/t	-	1.08	0.64	0.64	0.59	0.58	-	0.65	0.59	-	0.68	-	0.63
		K Ozs Au	-	42	139	27	197	137	-	188	61	-	19	-	812
	Waste Re-handle	K Tonnes	-	-	-	-	-	-	-	-	1,000	1,000	3,000	-	5,000

Table 24-11: Annual Stockpile Balance – Alternate Case

			Pre-Prod	**Yr -1 Prod**	**Yr 1**	**Yr 2**	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yr 8**	**Yr 9**	**Yr 10**	**Yr 11**
High-Grade Stockpile	Added	K Tonnes	627	274	-	89	-	-	434	-	-	68	-	-	-
		g Au/t	1.27	1.30	-	1.25	-	-	1.34	-	-	1.25	-	-	-
		K Ozs Au	26	11	-	4	-	-	19	-	-	3	-	-	-
	Removed	K Tonnes	-	665	237	89	-	-	-	434	-	-	68	-	-
		g Au/t	-	1.27	1.30	1.25	-	-	-	1.34	-	-	1.25	-	-
		K Ozs Au	-	27	10	4	-	-	-	19	-	-	3	-	-
	Balance	K Tonnes	627	237	-	-	-	-	434	-	-	68	-	-	-
		g Au/t	1.27	1.30	-	-	-	-	1.34	-	-	1.25	-	-	-
		K Ozs Au	26	10	-	-	-	-	19	-	-	3	-	-	-
Medium-Grade Stockpile	Added	K Tonnes	943	414	-	63	-	-	747	-	-	145	-	-	-
		g Au/t	0.85	0.85	-	0.84	-	-	0.85	-	-	0.86	-	-	-
		K Ozs Au	26	11	-	2	-	-	20	-	-	4	-	-	-
	Removed	K Tonnes	-	554	803	63	-	-	-	747	-	-	145	-	-
		g Au/t	-	0.85	0.85	0.84	-	-	-	0.85	-	-	0.86	-	-
		K Ozs Au	-	15	22	2	-	-	-	20	-	-	4	-	-
	Balance	K Tonnes	943	803	-	-	-	-	747	-	-	145	-	-	-
		g Au/t	0.85	0.85	-	-	-	-	0.85	-	-	0.86	-	-	-
		K Ozs Au	26	22	-	-	-	-	20	-	-	4	-	-	-
Low-Grade Stockpile	Added	K Tonnes	1,837	5,655	7,020	7,230	3,679	5,921	4,415	-	-	637	-	-	-
		g Au/t	0.60	0.58	0.58	0.59	0.57	0.58	0.60	-	-	0.58	-	-	-
		K Ozs Au	35	106	132	138	68	111	85	-	-	12	-	-	-
	Removed	K Tonnes	-	-	5,682	1,171	10,472	7,337	-	7,853	3,214	-	666	-	-
		g Au/t	-	-	0.59	0.59	0.59	0.58	-	0.59	0.59	-	0.58	-	-
		K Ozs Au	-	-	107	22	197	137	-	149	61	-	12	-	-
	Balance	K Tonnes	1,837	7,492	8,830	14,890	8,096	6,680	11,095	3,242	28	666	-	-	-
		g Au/t	0.60	0.59	0.58	0.59	0.58	0.59	0.59	0.59	0.59	0.58	-	-	-
		K Ozs Au	35	141	166	282	152	126	211	62	1	12	-	-	-
Total	Balance	K Tonnes	3,407	8,532	8,830	14,890	8,096	6,680	12,276	3,242	28	878	-	-	-
		g Au/t	0.79	0.63	0.58	0.59	0.58	0.59	0.63	0.59	0.59	0.68	-	-	-
		K Ozs Au	87	173	166	282	152	126	250	62	1	19	-	-	-

Table 24-12: Annual Ore Delivery to the Mill Crusher – Alternate Case

		Pre-Prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Total
Sulfide Ore	K Tonnes	-	11,666	11,612	11,599	11,512	11,657	10,779	11,258	11,567	11,747	4,841	-	108,241
	g Au/t	-	1.35	0.86	1.16	0.63	0.77	1.22	0.65	0.74	0.99	1.25	-	0.94
	K Ozs Au	-	506	321	434	234	288	422	236	274	375	195	-	3,285
	Recovery	0%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	0%	82%
	K Ozs Au Rec	-	415	264	356	192	236	346	193	225	308	160	-	2,694
Mixed Ore	K Tonnes	-	60	68	85	155	69	807	217	77	-	1	-	1,540
	g Au/t	-	0.94	0.55	0.89	0.55	0.55	0.65	0.61	0.56	-	0.56	-	0.65
	K Ozs Au	-	2	1	2	3	1	17	4	1	-	0	-	32
	Recovery	0%	78%	78%	78%	78%	78%	78%	78%	78%	0%	78%	0%	78%
	K Ozs Au Rec	-	1	1	2	2	1	13	3	1	-	0	-	25
Oxidized Ore	K Tonnes	-	21	34	30	48	21	129	240	70	-	1	-	593
	g Au/t	-	1.25	0.60	1.01	0.56	0.56	0.68	0.66	0.56	-	0.56	-	0.67
	K Ozs Au	-	1	1	1	1	0	3	5	1	-	0	-	13
	Recovery	0%	78%	78%	78%	78%	78%	78%	78%	78%	0%	78%	0%	78%
	K Ozs Au Rec	-	1	1	1	1	0	2	4	1	-	0	-	10
Total Ore	K Tonnes	-	11,747	11,715	11,715	11,715	11,747	11,715	11,715	11,715	11,747	4,843	-	110,374
	g Au/t	-	1.35	0.86	1.16	0.63	0.77	1.17	0.65	0.73	0.99	1.25	-	0.94
	K Ozs Au	-	509	323	438	237	289	442	245	277	375	195	-	3,330
	Recovery	0%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	0%	82%
	K Ozs Au Rec	-	417	265	359	194	237	361	201	227	308	160	-	2,729

For the purpose of scheduling, three ore stockpiles are assumed: High-grade ore stockpile for high-grade ore; medium-grade stockpile for medium-grade ore; and a low-grade stockpile for low-grade ore. The high-grade and medium-grade stockpiles are to be built within the low-grade stockpiling areas but are exhausted during the first year of processing when mill capacity becomes available. During the LoM, the low-grade stockpile is used as needed to feed the mill to full capacity. For this reason the stockpile grows and shrinks through the LoM. The maximum stockpile balance through the LoM is estimated to be 12.3 Mt.

Re-handling of stockpiled material will be done using a loader and trucks to haul ore to the crusher. **Table 24-11** shows the Alternate Case ore stockpile balances for the end of each year.

During the construction of the waste dump, excess Non-PAG material will be placed to the north end of the waste dump. This will allow convenient re-handle of waste at the end of the mine life for capping of various facilities for reclamation. However, additional stockpiling of Non-PAG material may be done by operations where it is found to be convenient and cost effective. Waste re-handle is shown on the bottom of **Table 24-10** to account for capping and reclamation.

Ore sent to the mill is shown in **Table 24-12**. This is a combination of ore shipped directly from the mine and ore that is reclaimed from stockpiles. These tables summarize the ore based on level of oxidation. The recovered ounces shown are based on the recoveries used for pit optimizations and are subject to change by Qualified Persons completing the metallurgical sections

Equipment Selection and Productivities

Availability, efficiencies, operating hours and load and haul equipment requirements are shown in **Table 24-13**.

Table 24-13: Annual Load and Haul Equipment Requirements – Alternate Case

Haulage Requirements		**Pre-Prod**	**Yr 1**	**Yr 2**	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yr 8**	**Yr 9**	**Yr 10**	**Total**
Productive Hours	Hrs	13,574	62,292	74,019	80,368	71,890	77,295	81,367	77,932	81,034	53,604	12,447	685,823
Operating Efficiency	%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%
Operating Hours	Hrs	16,354	75,050	89,180	96,829	86,614	93,127	98,033	93,894	97,632	64,584	14,996	826,293
Number of Trucks	#	5	13	15	15	15	16	16	16	16	12	10	148
Truck Availability	%	90%	90%	89%	88%	87%	86%	85%	85%	85%	85%	85%	
Available Operating Hours	Hrs	21,753	80,096	94,206	95,900	94,808	100,404	99,192	99,047	98,974	74,440	27,424	886,242
Use of Available Hours	%	87%	94%	95%	101%	91%	93%	99%	95%	99%	87%	55%	93%
Tonnes per Operating Hour	t/Hr	681	557	475	434	584	448	421	462	365	307	526	458
Hydraulic Shovel Useage													
Number of Shovels	#	1	2	2	2	2	2	2	2	2	2	2	2.0
Availability	%	58%	90%	89%	88%	87%	86%	85%	85%	85%	85%	85%	85%
Operating Efficeincy	%	53%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	82%
Available Operating Hrs	Op Hrs	4,908	13,063	12,881	12,736	12,590	12,480	12,372	12,372	12,372	12,407	5,485	123,664
Tonnes Mined	K Tonnes	11,891	40,587	35,624	40,735	40,112	34,393	41,245	34,310	31,390	18,806	4,014	333,106
Operating Hours	Op Hrs	3,029	10,340	9,076	10,378	10,219	8,762	10,508	8,741	7,997	4,791	1,023	84,866
Use of Available Operating Hours	%	62%	79%	70%	81%	81%	70%	85%	71%	65%	39%	19%	69%
Front End Loaders													
Number of Loaders	#	-	1	1	1	1	1	-	1	1	1	1	1.0
Availability	%	0%	90%	89%	88%	87%	86%	0%	85%	85%	85%	85%	86%
Operating Efficeincy	%	0%	83%	83%	83%	83%	83%	0%	83%	83%	83%	83%	83%
Available Operating Hrs	Op Hrs	-	6,568	6,477	6,404	6,331	6,276	6,186	6,186	6,186	6,203	2,742	59,560
Tonnes Mined	K Tonnes	-	1,218	6,722	1,322	10,472	7,337	-	9,034	4,214	1,000	3,878	45,199
Operating Hours	Op Hrs	-	874	4,821	948	7,510	5,262	-	6,479	3,022	717	2,781	32,414
Use of Available Operating Hours	%	0%	13%	74%	15%	119%	84%	0%	105%	49%	12%	101%	54%

Mine Personnel

The estimated number of mine personnel required to execute the mine plan is shown in **Table 24-14**.

Table 24-14: Mine Personnel Requirements – Alternate Case

Mine Overhead	Pre-Prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11
Mine Manager	1	1	1	1	1	1	1	1	1	1	1	1
Mine Clerk	1	1	1	1	1	1	1	1	1	1	1	1
Mine Shift Foremen	10	10	10	10	10	10	10	10	10	10	10	10
Mine Trainer	1	1	1	1	1	1	1	1	1	1	1	1
Blaster	4	4	4	4	4	4	4	4	4	4	4	4
Blaster's Helper	4	4	4	4	4	4	4	4	4	4	4	4
Mine Production												
Loading Operators	8	12	17	22	20	17	20	20	20	16	14	12
Haul Truck Operators	54	77	108	138	152	152	152	152	152	152	152	72
Drill Operators	26	35	42	57	50	50	43	33	29	28	32	16
Support Equipment Operators	20	21	22	22	22	22	24	24	24	24	24	24
Total Mine Operating	129	166	210	260	265	262	260	250	246	241	243	145
Mine Maintenance												
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1
Light Vehicle Mechanics	2	2	2	2	2	2	2	2	2	2	2	2
Tiremen	2	2	2	2	2	2	2	2	2	2	2	2
Shop Laborers	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Planner	1	1	1	1	1	1	1	1	1	1	1	1
Service, Fuel, & Lube	8	8	8	8	8	8	8	8	8	8	8	8
Total Mine Maintenance	16	16	16	16	16	16	16	16	16	16	16	16
Engineering												
Chief Engineer	1	1	1	1	1	1	1	1	1	1	1	1
Mine Surveyors	1	2	2	2	2	2	2	2	2	2	2	2
Surveyor Helper	2	2	2	2	2	2	2	2	2	2	2	2
Mine Engineer	3	3	3	3	3	3	3	3	3	3	3	3
Total Engineering	7	8	8	8	8	8	8	8	8	8	8	8
Mine Geology												
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	1
Ore Control Geologist	2	2	2	2	2	2	2	2	2	2	2	2
Sampler	2	2	2	2	2	2	2	2	2	2	2	2
Total Geology	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**
Total Mine Operations Workforce												
Mine Operations	129	166	210	260	265	262	260	250	246	241	243	145
Mine Maintenance	16	16	16	16	16	16	16	16	16	16	16	16
Engineering	7	8	8	8	8	8	8	8	8	8	8	8
Geology	5	5	5	5	5	5	5	5	5	5	5	5
Total	**157**	**195**	**239**	**289**	**294**	**291**	**289**	**279**	**275**	**270**	**272**	**174**

Salaries for each position were estimated based on information received from Tetra Tech and Vista. Salaries include an allowance for benefits at a rate of 25% of the base salary for each position. Note that mobile equipment labor costs are allocated to production equipment in the calculation of mining costs in later sections. In addition, a portion of the cost is allocated to construction of tailings facilities.

24.6.2 Process Facility

Design Criteria

The nominal headline design criteria are listed are shown in **Table 24-15.**

Table 24-15: Headline Design Criteria

	Unit	Alternate Case
Annual Ore Feed Rate	Mt/a	11.72
Operating Days per Year	d/y	355
Daily Ore Feed Rate	t/d	33,000
Crushing Rate (6637 hours per year availability)	tph	1765
HPGR & Milling Rate (7838 hours per year)	tph	1495
Gold Head Grade	g/t	0.94
Copper Head Grade	%	0.055
Cyanide Soluble Copper	%	0.015
Ore Specific Gravity		2.76
Grind P_{80} to Leach	µm	90
Gold Recovery	%	82
Gold Production (nominal)	oz/d	815
Gold Production (nominal)	oz/a	289,256

Flowsheet

The Alternate Case flowsheet is the same as that of the Base Case, as shown in **Figure 17-1** with the following equipment differences:

- Cone Crushers: 2- MP1000 secondary crushers;
- HPGRs: 2- HPGR Polycom 20/17-7, each equipped with 2 x 2,000 kW drives, 2.0 m diameter rolls;
- Ball Mills: 2- 18.15 MW dual pinion drive ball mills;
- Thickeners: 27m diameter; and
- Leach/CIP: Carbon movement requirements will be in the order of 20 tpd.

24.6.3 Infrastructure

Project support infrastructure and services are similar to the Base Case, but with the following differences:

Area 2300 – Mine Support Facilities

<u>Support Facilities – HV Workshop/Warehouse</u>. The HV Workshop will decrease in size for the Alternate Case from six bays to four bays. The smaller facility will require a significant reduction in the amount of concrete. There will be no difference in size between the Base Case and the Alternate Case for the Warehouse. The Alternate Case will have a reduced turnover of stock/spares.

Fuel Farm: The fuel farm for Alternate Case will require four bowsers as opposed to six.

Lube Farm: The Lube Farm area for the Alternate Case will have a reduced footprint for the storage of lubricants.

Area 4000 – Project Services

4100 Water Supply. Water supply to the Project does not differ greatly between the two cases except that the water treatment plant is sized to handle a smaller throughput and several supply pumps will be decreased in size.

Raw water will be brought down to the raw water tank in the process plant through the existing 400 mm poly line at the RWD.

4200 Power Supply. The current carrying capacity of the buried cable between the overhead power lines and the Process Plant will be decreased to suit the approximate 45% decrease in plant load.

4200 Overhead Power Lines. The current carrying capacity of the overhead power lines between the Power Station and the Process Plant will be decreased to suit the approximate 45% decrease in plant load. The overhead power line network will not be required to extend to the RWD as the existing system will provide adequate raw water to the plant.

Area 5000 – Project Infrastructure

5100 Bulk Earthworks. The site footprint for the Alternate Case will be smaller than the Base Case and as a result will have lower earthworks, road works and drainage costs.

5230 Reagent Store: The Alternate Case will require a reduced inventory of reagents and therefore the footprint of the Reagents Stores will be reduced.

5260 Sample Preparation and Laboratory: The Alternate Case requires a reduced inventory of sample processing laboratory equipment to process 300 samples/day.

5400 Heavy Lift Cranage. The heavy lift cranage durations will be reduced for the Alternate Case based on a reduced SMP contract schedule.

24.6.4 Site Wide Water Balance Model

An Alternate Case analyzed. Differences between the Base Case and the Alternate Case water balance models are:

- Process plant for the alternative is 1,703 m^3/day;
- Production occurs over 13 years;
- Water is treated at a rate of 300 m^3/day;
- Equalization pond will hold a 5 day volume of 36,000 m^3;
- Maximum draw from the RWD for make-up water is 30,479 m^3/day;
- Batman pit shell evolution extents are less for the ultimate pit; and
- Other water requirements (potable, reagent mixing, gland, etc.) are consistent with lower production rates.

24.6.5 Capital Costs, Alternate Case

LoM capital cost requirements for the Alternate Case have been estimated using the similar parameters as used for the Base Case. The Alternate Case results are estimated at US$972 million as summarized in **Table 24-16**. Initial capital of US$761 million is required to commence operations. At the end of operations, the model assumes the Project will receive a US$47 million credit for remaining on-site mine mobile equipment, a US$30 million salvage credit for process plant equipment and the return of the US$15 million reclamation bond.

Table 24-16: Capital Cost Summary, Alternate Case

Capex Summary	Initial Capital	Sustaining Capital	Salvage Credit	Total
Capitalized Costs	24,303	132,254	0	156,557
Mine & Process Mobile	77,336	73,150	(46,539)	103,947
CIL Process Plant	310,308	0	(30,324)	279,984
Tailings Dams	19,373	85,667	0	105,040
Power Supply	64,307	0	0	64,307
Water Supply	11,136	0	0	11,136
Owner's Costs	254,429	12,027	(15,000)	251,455
Total Capex Summary	**761,192**	**303,097**	**(91,862)**	**972,427**

24.6.6 Operating Costs, Alternate Case

LoM operating cost estimates are summarized in **Table 24-17**. The operating costs will average US$15.99 over the LoM.

Table 24-17: Operating Cost Summary, Alternate Case

Opex Summary	US$/t-mined	US$/t-milled	Total (US$000s)
Open Pit Mining	2.037	5.485	678,670
CIL Process Plant	-	9.507	1,176,245
Water Treatment Plant	-	0.083	13,983
G&A	-	0.739	91,431
Jawoyn Royalty	-	0.339	41,921
Refining Costs	-	0.075	9,234
Power Credit	-	-0.235	(106,052)
Total Opex Summary	**-**	**15.992**	**1,905,432**

24.6.7 Economic Results, Alternate Case

Project cost estimates and economics are prepared on an annual basis. Based upon design criteria presented in this report, the level of accuracy of the estimate is considered ±25%.

Economic results are summarized in **Table 24-18**. The analysis suggests the following conclusions, assuming a 100% equity project at a gold price of US$1,450:

- Mine Life: 11 years;
- Pre-Tax $NPV_{5\%}$: US$1,222 million, IRR: 22%;
- Post-Tax $NPV_{5\%}$: US$440 million, IRR: 17%;
- Payback (Post-Tax): 3.2 years;
- NT Royalty Taxes Paid: US$173 million;
- Australian Income Taxes Paid: US$307 million; and
- Cash costs (including Jawoyn Royalty): US$684/oz-Au.

Table 24-18: Economic Results, Alternate Case

Cash Flow Summary	LoM Cost (US$000s)	Unit Cost $/t-milled	Unit Cost $/oz-Au
Gold Produced	2,891	-	-
Gold Price	1,450	-	-
Gold Sales	**4,192,069**	**33.88**	**1,450.00**
Refinery Costs	(9,234)	(0.07)	(3.19)
Net Smelting Return	**4,182,834**	**33.81**	**1,446.81**
Jawoyn Royalty	(41,921)	(0.34)	(14.50)
Gross Income from Mining	**4,140,914**	**33.47**	**1,432.31**
Open Pit Mine	(678,670)	(5.49)	(234.75)
CIL Process Plant	(1,176,245)	(9.51)	(406.85)
Water Treatment Plant	(13,983)	(0.11)	(4.84)
G&A	(91,431)	(0.74)	(31.63)
Operating Costs	**(1,960,329)**	**(15.84)**	**(678.06)**
Power Sales Credit	106,052	0.86	36.68
Cash Cost of Goods Sold	**(1,863,511)**	**(15.06)**	**(644.57)**
Operating Margin	**2,286,637**	**18.48**	**790.93**
Capitalized Costs	(156,557)	(1.27)	(54.15)
Mine & Process Mobile Capital	(150,486)	(1.22)	(52.05)
CIL Process Plant	(310,308)	(2.51)	(107.33)
Tailings Dams (4400)	(105,040)	(0.85)	(36.33)
Power Supply (4200)	(64,307)	(0.52)	(22.24)
Water Supply & Treatment(4100)	(11,136)	(0.09)	(3.85)
Owner Costs	(266,455)	(2.15)	(92.16)
Salvage	0	0.00	0.00
Capital Costs	**(1,064,289)**	**(8.60)**	**(368.13)**
Pre-Tax Cash Flow	**1,222,348**		
$NPV_{5\%}$	**776,874**		
IRR	**22%**		
Payback (years)	**2.80**		
Post-Tax Cash Flow	**849,794**		
$NPV_{5\%}$	**440,183**		
IRR	**17%**		
Payback (years)	**3.20**		

25.0 INTERPRETATION AND CONCLUSIONS

25.1 GEOLOGY AND RESOURCES

- The Mt. Todd Project is situated within the southeastern portion of the Early Proterozoic Pine Creek Geosyncline which is comprised of the Burrell Creek Formation, the Tollis Formation, and the Kombolgie Formation.
- Gold mineralization in this area is constrained to a single mineralization event and the deposits are classified as orogenic gold deposits in the subdivision of thermal aureole gold style. The Batman deposit has characteristics of an intrusion related gold system making it the primary resource.
- The Batman deposit is defined by 7.4 million ounces of gold within 27.96 Mt of measured and indicated resource at an average grade of 0.82 g Au/t and a cutoff grade of 0.4 g Au/t.
- The progression of the Batman deposit resource at a cutoff grade of 0.4 g Au/t since the January 2011 PFS is summarized below:

Report	Category	Tonnes (x1000)	Average Grade g Au/t	Total Au Ounces (x1000)	Increase in ounces of Au from Previous Report
March 2013	Measured & Indicated	279,585	0.82	7,401	6%
September 2012	Measured & Indicated	261,400	0.83	7,007	17%
September 2011	Measured & Indicated	222,022	0.84	5,987	17%
January 2011 PFS	Measured & Indicated	190,939	0.84	5,125	-

Tonnage, grades and totals may not total due to rounding.
All estimated resources are shown using a 0.4 g Au/t cutoff grade.
Vista's first mineral resource estimate for the Batman deposit.

25.2 MINERAL RESERVE AND MINE PLANNING

- The Mt Todd Proven and Probable reserves have been defined based on optimization of mining plans using a gold price of $1,360 per ounce. The Proven and Probable reserves were used to create a production schedule for mining, and a positive cash-flow analysis has been done based on the production schedule by Tetra Tech. This establishes the reserves as having reasonable economics with respect to the statement of reserves under NI 43-101 regulations.
- Mine production constraints were imposed to ensure that mining wasn't overly aggressive with respect to the equipment anticipated for use at Mt. Todd. The schedule has been produced using mill targets and stockpiling strategies to enhance the project economics. The constraints and limits used are reasonable to support the project economics which are used to justify the statement of reserves.
- Pit designs were created to use six-meter benches for mining. This corresponds to the resource model block heights, and MDA believes this to be reasonable with respect to dilution and equipment anticipated to be used in mining. In areas where the material is consistently ore or waste so that dilution is not an issue, benches may be mined in 12-m heights.

25.3 Mineral Processing

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

- Mt. Todd (Batman) ore is one of the hardest and most competent ore types processed for mineral recovery. The most energy efficient comminution circuit has been determined to be the sequence of primary crushing, closed circuit secondary crushing, and closed circuit HPGR tertiary crushing followed by ball milling.
- The ore is free-milling, has no preg robbing problem, and is amenable to gold extraction by conventional cyanidation processes.
- The ore has relatively high specific cyanide consumption, determined to be 770 g of sodium cyanide per tonne of ore. This is largely due to the presence of sulfides and cyanide consuming copper.
- The ore requires a P_{80} grind of 90 µm and 24 hr leach residence time to achieve a nominal 82% gold recovery (81.7% net of solution loss) from a global head grade of 0.96 g of gold per tonne of ore.

The equipment selection criteria for the large scale 50,000 tpd operation has received considerable interaction with specialist vendors to the point where there is a reasonably high degree of confidence in selected technology and process units at this preliminary feasibility study stage. The recommended flowsheet for FS consists of primary crushing, closed circuit secondary crushing, closed circuit tertiary crushing using HPGRs, ball milling, cyclone classification, pre-leach thickening, leach and adsorption, elution electrowinning and smelting, carbon regeneration, tailings detox and disposal to conventional tailings storage facility.

25.4 Infrastructure

25.4.1 Site Preparation

Bulk earthworks are designed to minimize the import of fill materials.

25.4.2 Support Buildings

- Administration offices, gatehouse/security facilities, cribs/ablutions are planned to be transportable buildings.
- The process plant offices, workshop and warehouse are located inside the existing Flotation Building.
- Sample preparation and laboratory will have a purpose-built steel shed.

25.4.3 Access Roads Parking and Laydown

The access road is based on the repaired existing road.

25.4.4 Heavy Lifts

Heavy cranage is allowed for all lifts greater than 50t.

25.4.5 Bulk Transport

All bulk transport will be weighed.

25.4.6 Communications

Site wide communication is based on a 50m tall communication tower that will support eight (8) channels.

25.5 Environmental and Social Conclusions

25.5.1 Existing Body of Work

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

25.5.2 Draft EIS

A draft EIS will be completed by mid-2013. Following submittal of the draft, approval by the Northern Territories EPA is required in accordance with the EIS approval process.

25.5.3 Social or Community Impacts

The Jawoyn people have strong involvement in the planning of the Mt. Todd Project. Areas of Aboriginal Significance have been designated, and the mine plan has avoided development in these Restricted Works Areas

25.5.4 Results of the Site Wide Water Balance Model

- The WTP rate of 500 m^3/hr and EQP sizing for 5 days of storage was determined to be appropriate for the 50,000 tpd production process water requirements.
- The greatest amount of make-up water required from the RWD was quantified as 24,409 m^3/day. RWD requirements were found to be the most dependent upon TSF decant volumes, with the greatest need occurring during the dry season when decant rates were low.
- The WRD pond was typically observed to overtop less than 0.3% of the time during the 12 year simulation.[6] RP2 and RP5 storage may be optimized.

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

26.0 RECOMMENDATIONS

26.1 FEASIBILITY STUDY

A FS is recommended to advance the Project to a place where any additional detailed information necessary provide support of capital and operating cost estimates which lead to a potential project development decision.

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

26.2 RESOURCE AND EXPLORATION

- The Batman deposit potentially extends along strike both to the north and south. Step out drilling should be used to explore this.
- Additional deep infill drilling should be used to help define potential deep mineralization not detected by early historical shallow drillholes.
- Infill drilling within and exploration drillholes along the trend of the Quigley deposit is recommended.
- Additional drilling exploring the exploration licenses, following up on geophysical and geochemical anomalies.

The estimated budget for drilling within the mining licenses is US$500,000-1,000,000 and US$500,000-1,000,000 for drilling on the exploration licenses.

26.3 ENVIRONMENTAL STUDIES

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

- Erosion analyses;
- Waste and cover material hydraulic properties characterization and analysis;
- Acid-base accounting on waste rock and tailings;
- Ongoing aquatic, benthic and wildlife studies;
- Comprehensive vegetation survey;
- Archaeological and historical assessments for all areas to be disturbed;
- Hydrogeologic investigations and site-wide hydrogeologic characterization; and
- Precipitation, stream flow, and watershed data.

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

26.3.1 Site Wide Water Balance

Recommended model improvements include:

- The site wide water balance model is dependent upon the TSF water balance model, which provides decant water to the process facility, and the vadose and seepage models, which characterize seepage through the various rock piles on site (WRD, LGOS and HLP). As such, completion of these models to the greatest detail practicable affects the overall quality of the site wide water balance model results.
- Clarification on the elution/potable water requirements is needed. The current model relies on an assumed ratio of the process demand of 16%.
- Further investigation of the adequacy of RP 1 storage capacity is recommended, particularly within the early stages of the LoM when a larger fraction of the catchment reports to the pond.
- Stage-storage relationships of RWD will be included in the DFS model such that it may be modeled as a reservoir, as opposed to an infinite source.

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

26.4 Groundwater Hydrology and Mine Dewatering

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

- Calibration of the regional groundwater flow model should be completed, 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 completed with the calibrated model used as its basis. Output from the post-mining model should be incorporated into any geochemical modeling of post-mining pit lake formation and geochemistry.
- The pit dewatering system design and cost estimates should be refined to include groundwater inflows estimated with the calibrated model.
- A tradeoff study should be conducted to identify the optimum balance between the cost of the dewatering pumping equipment and the cost of mine pit floor and bench inaccessibility while the pit is being pumped dry after storm events.

The estimated budget for this work is US$300,000.

26.5 Geotechnical Investigation Recommendations

For the DFS future geotechnical work is suggested in the following areas:

Grinding

This is a large vibrating structure which should be founded in rock rather than on fill to reduce dynamic effects. An accurate rock level is required to confirm foundation design and accurately estimate required concrete quantities and rock excavation. Existing structure concrete slabs are located directly over the new mill location, so it is recommended that additional new test pits around all four sides be undertaken to allow interpolation.

Leach/CIP

Large tanks to be constructed in this area, with high foundation bearing pressure. Variation in rock level will impact on differential settlement which needs to be considered. Recommend at least four additional test pits to evaluate variance in rock levels in north-south and east-west directions.

Stockpile & Reclaim

No test pit data anywhere near this area (nearest test pit is more than 200m away). Steeply sloping ground surface exists (in excess of 5 m variation in ground level across the reclaim tunnel) so there may be considerable variation in rock levels which will effect potential settlement due to stockpile surcharge and required excavation for the concrete vault and tunnel. Recommend a new borehole on the high side (uncertain whether this high side bench is fill material, and therefore whether rock would be encountered within the limits of an excavator for a test pit) and a new test pit on low side, to determine rock levels.

Coarse Screening

Vibrating structure which should be founded on rock rather than on fill to reduce dynamic effect. An accurate rock level is required to confirm foundation design and accurately estimate required concrete quantities and rock excavation. Existing test pits are on one side only so unable to interpolate between existing pits, the nearest existing test pit is approximately 100m to the north so recommend a new test pit at this location.

HPGR

Vibrating structure which should be founded on rock rather than on fill to reduce dynamic effect. Therefore need to know rock level to confirm foundation design and accurately estimate required concrete quantities and rock excavation. Nearest existing test pit is approximately 70m away so recommend new test pit at this location.

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

26.6 PROCESS OPERATING COSTS

Two major items incurring operating costs are steel balls as grinding media and cyanide as leaching reagent. Together these items make up 65% of the plant consumables operating costs. The FS should investigate options for reducing the consumption rate and the unit costs for these consumables.

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

26.7 GEOCHEMICAL ANALYSES

Geochemical characterization will be updated to reflect the designations of Potentially Acid Forming, Potentially Acid Forming-Low Capacity, Non Acid Forming, Acid Consuming and Uncertain in accordance with DITR (2007) guidelines.

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

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28.0 CERTIFICATE OF QUALIFIED PERSON

28.1 QUALIFICATIONS OF CONSULTANTS

The Consultants preparing this Technical Report are specialists in the fields of geology, exploration, mineral resource and mineral reserve estimation and classification, underground mining, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the Consultants or any associates employed in the preparation of this report has any beneficial interest in Vista. The Consultants are not insiders, associates, or affiliates of Vista. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Vista and the Consultants. The Consultants are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered Qualified Persons (QP) as defined in the NI 43-101 standard, for this report, and are members in good standing of appropriate professional institutions.

This Technical Report was prepared by the following QPs, Certificates and consents of which are contained herein:

Name	Title, Company	Responsible for Sections
Rex Bryan, PhD	Senior Geostatistician Tetra Tech, Inc.	6, 7, 8, 9, 10, 11, 12, 14 and portions of 25-27
Patrick Donlon, FSAIMM, FAusIMM, NHD Ext Met	Principal Metallurgist Proteus EPCM Engineers, a Tetra Tech company	13 and portions of 17
Thomas L. Dyer, P.E.	Senior Engineer Mine Development Associates	16 and portions of 15, 21, 25, and 26
Deepak Malhotra, PhD	President Resource Development Inc.	Section 15.7
Nick Michael, B.S., MBA	Principal Mineral Economist Tetra Tech, Inc.	1, 2, 3, 4, 5, 19, 22 and portions of 21, 24-27
David M. Richers, PhD, PG	Geochemist, Geologist Tetra Tech, Inc.	20
Lachlan Walker, FIEAust, CPEng	Director Proteus EPCM Engineers, a Tetra Tech company	18 and portions of 17, 21, 24, 26 and 27

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To accompany the Report Entitled: "NI 43-101 Technical Report, Mt. Todd Gold Project 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia" (Technical Report), effective May 29, 2013, issued June 28, 2013.

I, Rex Clair Bryan, PhD, do hereby certify that:

1) I am a Senior Geostatistician with Tetra Tech, Inc. with a business address at 350 Indiana Street, Suite 500, Golden, Colorado 80401, USA.
2) I graduated with a degree in Engineering (BS with honors) in 1971 and a MBA degree in 1973 from the Michigan State University, East Lansing. In addition, I graduated from Brown University, Providence, Rhode Island with a MS degree in Geology in 1977, and The Colorado School of Mines, Golden, Colorado, with a graduate degree in Mineral Economics (Ph.D.) in 1980. I have worked as a resource estimator and geostatistician for a total of thirty-one years since my graduation from university; as an employee of a leading geostatistical consulting company (Geostat Systems, Inc. USA), with large engineering companies such as Dames and Moore, URS, and Tetra Tech and as a consultant for more than 30 years. I am a Registered Member (#411340) of the Society for Mining, Metallurgy, and Exploration, Inc.
3) I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
4) I have visited and inspected the subject property from September 12th, 2011 to September 14th, 2011 and February 6th, 2013 to February 8th, 2013.
5) I am responsible for Sections 6, 7, 8, 9, 10, 11, 12, 14 and portions of 25-27 of the Technical Report.
6) I satisfy the requirements of independence according to Section 1.5 of NI 43-101.
7) I have had prior involvement with Vista Gold Corp. on the property that is the subject of this Technical Report. My involvement has consisted of acting as an expert who was relied upon for previous Technical, Preliminary Economic Assessment, and Prefeasibility Reports.
8) I have read National Instrument 43-101, Form 43-101F1, and 43-101CP, and the Technical Report has been prepared in compliance with that instrument, form, and companion policy.

9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 28th day of June 2013

"Rex Clair Bryan" – Signed
Signature of Qualified Person

Rex Clair Bryan, PhD________
Print Name of Qualified Person

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To accompany the Technical Report titled "NI 43-101 Technical Report, Mt. Todd Gold Project 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia" (Technical Report), effective May 29, 2013, issued June 28, 2013.

I, Patrick Donlon, FAusIMM, FSAIMM, Ext Met NHD, do hereby certify that:

1) I am a Principal Metallurgist at Proteus EPCM Engineers, a Tetra Tech company, at 370 Murray Street, Perth, Western Australia 6000.

2) I graduated with an Extraction Metallurgy National and Higher National Diploma from Johannesburg University's Technikon Witwatersrand School of Mining, South Africa, in 1986. I have 26 years' experience in technical support and management of metallurgical operations, and as a consultant, lead design engineer and principal metallurgist engaged in the design of metallurgical processes. My expertise is in mineral processing and recovery of metals including gold, platinum, uranium and ferrous metals.

3) I am a Fellow of the Australian Institute of Mining and Metallurgy (member number 308860), and I am a Fellow of the South African Institute of Mining and Metallurgy (member number 701397). I am in good standing with these professional institutions.

4) I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5) I have never visited the subject property.

6) I am responsible for the preparation of Sections 13 and 17 of the Technical Report.

7) I satisfy the requirements of independence according to Section 1.5 of NI 43-101.

8) I have had previous involvement with Vista Gold Corp. on the property that is the subject of this Technical Report. I previously contributed to an Independent Assessment of selected elements of a Feasibility Study prepared by Ausenco Limited. The findings are documented in "Independent Assessment of Process Plant Feasibility Study, Doc. No 12121-RG-001B."

9) I have read National Instrument 43-101, Form 43-101F1, and 43-101CP, and the Technical Report has been prepared in compliance with that instrument, form, and companion policy.

10) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

11) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 28th day of June 2013

"Patrick Donlon, FAusIMM, FSAIMM, Ext Met NHD" – Signed
Signature of Qualified Person

Patrick Donlon, FAusIMM, FSAIMM, Ext Met NHD
Print Name of Qualified Person

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To accompany the Report Entitled: "NI 43-101 Technical Report, Mt. Todd Gold Project 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia" (Technical Report), effective May 29, 2013, issued June 28, 2013.

I, Thomas L. Dyer, P.E., do hereby certify that:

1) I am currently employed as Senior Engineer by Mine Development Associates, Inc., 210 South Rock Blvd., Reno, Nevada 89502.
2) I graduated with a Bachelor of Science degree in Mine Engineering from South Dakota School of Mines & Technology in 1996. I have worked as a Mining Engineer for 18 years since graduation. I am registered as a Professional Engineer – Mining in the State of Nevada (# 15729). I am also a Registered Member of the Society of Mining, Metallurgy, and Exploration (# 4029995RM) in good standing.
3) I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
4) I have visited the subject property during March, 2011.
5) I am responsible for Sections 1.6 (excluding 1.6.1), 1.7, 1.15.3, 15.1 through 15.6, all of Section 16, Section 21.1.1, 21.2.1, 24.6.1 and 25.2 of the Technical Report.
6) I satisfy the requirements of independence according to Section 1.5 of NI 43-101.
7) I have had previous involvement with Vista Gold Corp. on the property that is the subject of this Technical Report. I previously completed a mining study and reserve statement which was included into the technical report entitled: "Preliminary Feasibility Study - NI 43-101 Technical Report – Mt Todd Gold Project – Northern Territory, Australia" (October 1, 2010).
8) I have read National Instrument 43-101, Form 43-101F1, and 43-101CP, and the Technical Report has been prepared in compliance with that instrument, form, and companion policy.
9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 28th day of June 2013

"Thomas L. Dyer" - Signed
Signature of Qualified Person

Thomas L. Dyer, P.E.
Print Name of Qualified Person

CERTIFICATE OF AUTHOR

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To accompany the Report Entitled: "NI 43-101 Technical Report, Mt. Todd Gold Project 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia" (Technical Report), effective May 29, 2013, issued June 28, 2013.

I, Deepak Malhotra, PhD, do hereby certify that:

1) I am currently employed as President by Resource Development Inc. (RDi) at:
   11475 W. I-70 Frontage Road North
   Wheat Ridge, Colorado 80033

2) I graduated with an M.S. in Metallurgical Engineering and a Ph.D. in Mineral Economics from the Colorado School of Mines in 1974 and 1978, respectively. I have worked as a metallurgist/mineral economist for a total of 35 years and have been involved with the preparation of numerous reports, feasibility studies, and NI 43-101 documents. I am a Registered Member of the Society for Mining, Metallurgy, and Exploration, Inc. (SME) and the Canadian Institute of Mining (CIM). I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

3) I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4) I have not visited or inspected the subject property.

5) I am responsible for Section 15.7.

6) I satisfy the requirements of independence according to Section 1.5 of NI 43-101.

7) I have had prior involvement with Vista Gold Corp. on the property that is the subject of this Technical Report. My involvement has consisted of acting as the Qualified Person for previous Technical, Preliminary Economic Assessment, and Prefeasibility Reports.

8) I have read National Instrument 43-101, Form 43-101F1, and 43-101CP, and the Technical Report has been prepared in compliance with that instrument, form, and companion policy.

9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 28th day of June 2013

_*"Deepak Malhotra"- Signed*____

Signature of Qualified Person

Deepak Malhotra, PhD_________

Print Name of Qualified Person

CERTIFICATE OF AUTHOR
Nick Michael, B.S., MBA
Principal Mineral Economist
Tetra Tech, Inc.
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To accompany the Report Entitled: "NI 43-101 Technical Report, Mt. Todd Gold Project 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia" (Technical Report), effective May 29, 2013, issued June 28, 2013.

I, Nick Michal, do hereby certify that:

1) I am a Principal Mineral Economist with Tetra Tech, Inc. with a business address at 350 Indiana Street, Suite 500, Golden, Colorado 80401, USA.

2) I am a graduate of the Colorado School of Mines in Golden, Colorado USA in mining engineering (1983) and received and received an MBA from Willamette University (1986). I have practiced my profession continuously since 1987. Since 1990, I have completed valuations, evaluations (technical-economic models), and have audited a variety of projects including exploration, pre-production (feasibility-level), operating and mine closure projects. I have also served as expert witness with respect to technical-economic issues.

3) I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

4) I have never visited the subject property.

5) I am a co-author and responsible for the preparation of Sections 1, 2, 3, 4, 5, 19, 22 and portions of Sections 21, 24-27 of the Technical Report.

6) I satisfy the requirements of independence according to Section 1.5 of NI 43-101.

7) I have had previous involvement with Vista Gold Corp. on the property that is the subject of this Technical Report. I previously contributed to technical economic modeling in support of previous preliminary feasibility studies for Vista Gold.

8) I have read National Instrument 43-101, Form 43-101F1, and 43-101CP, and the Technical Report has been prepared in compliance with that instrument, form, and companion policy.

9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 28th day of June 2013

"Nick Michael" – Signed
Signature of Qualified Person

Nick Michael, B.S., MBA
Print Name of Qualified Person

CERTIFICATE AND CONSENT

David M. Richers, PhD, PG
Geochemist, Geologist
Tetra Tech, Inc.
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To accompany the Technical Report titled "NI 43-101 Technical Report – Mt. Todd Gold Project, 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia" (Technical Report), effective May 29, 2013, issued June 28, 2013.

I, David M. Richers, PhD, PG, do hereby certify that:

3) I am currently employed by Tetra Tech, Inc. at: 350 Indiana Street, Suite 350 Golden, Colorado 80401.

4) I have been practicing my profession as a geologist/geochemist for 39 years since receiving my BS degree in Geology from Pennsylvania State University in 1974. I also received an MS degree in Geology/Geochemist in 1977 from University of Kentucky, and a PhD degree in Geology/Geochemistry from University of Kentucky in 1980. My relevant experience as a geologist and geochemist includes geochemical site characterization services and mine geology. I have worked on mining projects in the United States, Australia, Spain, and Canada including both surface and underground operations. My duties routinely included participation in geochemical studies and programs aimed at protecting the environment including quantification of geochemical processes for engineering design, closure planning and impact analysis. My background also includes extensive work with acid rock drainage and metal leaching (ARD/ML) and the associated fate and transport. I also have expertise in geologic computer mapping and 3D GIS.

5) I have read the definition of "qualified person" set out in *National Instrument 43-101 Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" within the meaning of NI 43-101.

6) I have not visited and have not inspected the subject property.

7) I am responsible for Section 20 of this Technical Report.

8) I am independent of Vista Gold Corp. as defined by Section 1.5 of the Instrument.

9) I have had prior involvement with Vista Gold Corp. on the property that is the subject of this Technical Report. My involvement has consisted of acting as an expert who was relied upon for waste rock characterizations and geologic assessments in previous reports.

10) I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.

11) As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12) I consent to the public filing of the Technical Report entitled "NI 43-101 Technical Report, Mt. Todd Gold Project, Preliminary Feasibility Study, Northern Territory, Australia", effective May 29, 2013, issued June 28, 2013.

13) The undersigned does hereby consent to any extracts from, or summary of, the Technical Report in Vista Gold's press release dated May 29, 2013.

14) The undersigned does hereby certify that I have read the press release dated May 29, 2013 filed by Vista Gold Corp. and that the press release fairly and accurately represents the information in the sections of the Technical Report for which I am responsible.

Dated this 28th day of June 2013

"David M. Richers" - Signed

Signature of Qualified Person

David M. Richers, PhD, PG

Print name of Qualified Person

CERTIFICATE OF AUTHOR
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To accompany the Technical Report titled "NI 43-101 Technical Report, Mt. Todd Gold Project 50,000 tpd Preliminary Feasibility Study, Northern Territory, Australia" (Technical Report), effective May 29, 2013, issued June 28, 2013.

I, Lachlan Walker, FIEAust, CPEng, do hereby certify that:

11) I am a Director at Proteus EPCM Engineers, a Tetra Tech company, at 370 Murray Street, Perth, Western Australia 6000.

12) I graduated with a Bachelor Degree in Civil Engineering from the University of Western Australia in 1971. I have 40 years' experience in Studies, Design and EPCM Execution – initially as a design engineer and finally as a Principal Project Manager. Major focus has been chemical, mineral processing and materials handling projects in Gold, Iron Ore, Minerals Sands, Alumina, Base Metals and Diamonds.

13) I am a Chartered Professional Engineer and a Fellow of the Institution of Engineers Australia (Mem. No. 493597). I am a member of the Australian Cost Engineering Society and a (lapsed) Member of the Project Management Institute, USA. All memberships in good standing.

14) I have read the definition of "qualified person" set out in National Instrument 43-101 - Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

15) I have never visited the subject property.

16) I am responsible for the preparation of the following Sections of the Technical Report:
    - 18.2.1, 18.2.3, 18.2.6, 18.3.1, 18.3.2, 18.3.3, 18.3.5
    - 21.1.2, 21.2.2, 24.5, 24.6.3, 24.6.5, 24.6.6

17) I satisfy the requirements of independence according to Section 1.5 of NI 43-101.

18) I have had previous involvement with Vista Gold Corp. on the property that is the subject of this Technical Report. I previously completed an Independent Assessment of selected elements of a Feasibility Study prepared by Ausenco Limited. The findings are documented in "Independent Assessment of Process Plant Feasibility Study, Doc. No 12121-RG-001B."

19) I have read National Instrument 43-101, Form 43-101F1, and 43-101CP, and the Technical Report has been prepared in compliance with that instrument, form, and companion policy.

20) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

21) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 28th day of June 2013

***"Lachlan Walker, FIEAust, CPEng."* -** Signed
Signature of Qualified Person

Lachlan Walker, FIEAust, CPEng
Print Name of Qualified Person