10.65 Mtpy Preliminary Feasibility Study
NI 43-101 Technical Report
Mt. Todd Gold Project

Northern Territory, Australia

Prepared for:

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1.0 SUMMARY

Vista Gold Corp (Vista) purchased the Mt. Todd property on March 1, 2006, and the acquisition was completed on June 16, 2006 when the mineral leases transferred to Vista and funds were released from escrow. Tetra Tech, Inc. (Tetra Tech) was commissioned by Vista in September 2009 to prepare a NI 43-101 compliant Preliminary Feasibility Study (PFS) at an ore processing rate of 6.77 million tonnes per year (Mtpy) for the Mt. Todd Gold Project located in Northern Territory (NT), Australia. 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 is the subject of the study presented herein and issued January 28, 2011.

Prior to these two PFS studies, an initial NI43-101 Technical Report was completed on June 26, 2006. A Preliminary Economic Assessment 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 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 these have had historic mining, with Batman having the most production and exploration completed. Currently, only the Batman and Quigleys deposits have Canadian Institute of Mining, Metallurgical, and Petroleum (CIM) compliant reported resources and only the Batman deposit has CIM compliant reportable mineral reserves.

1.1 Location

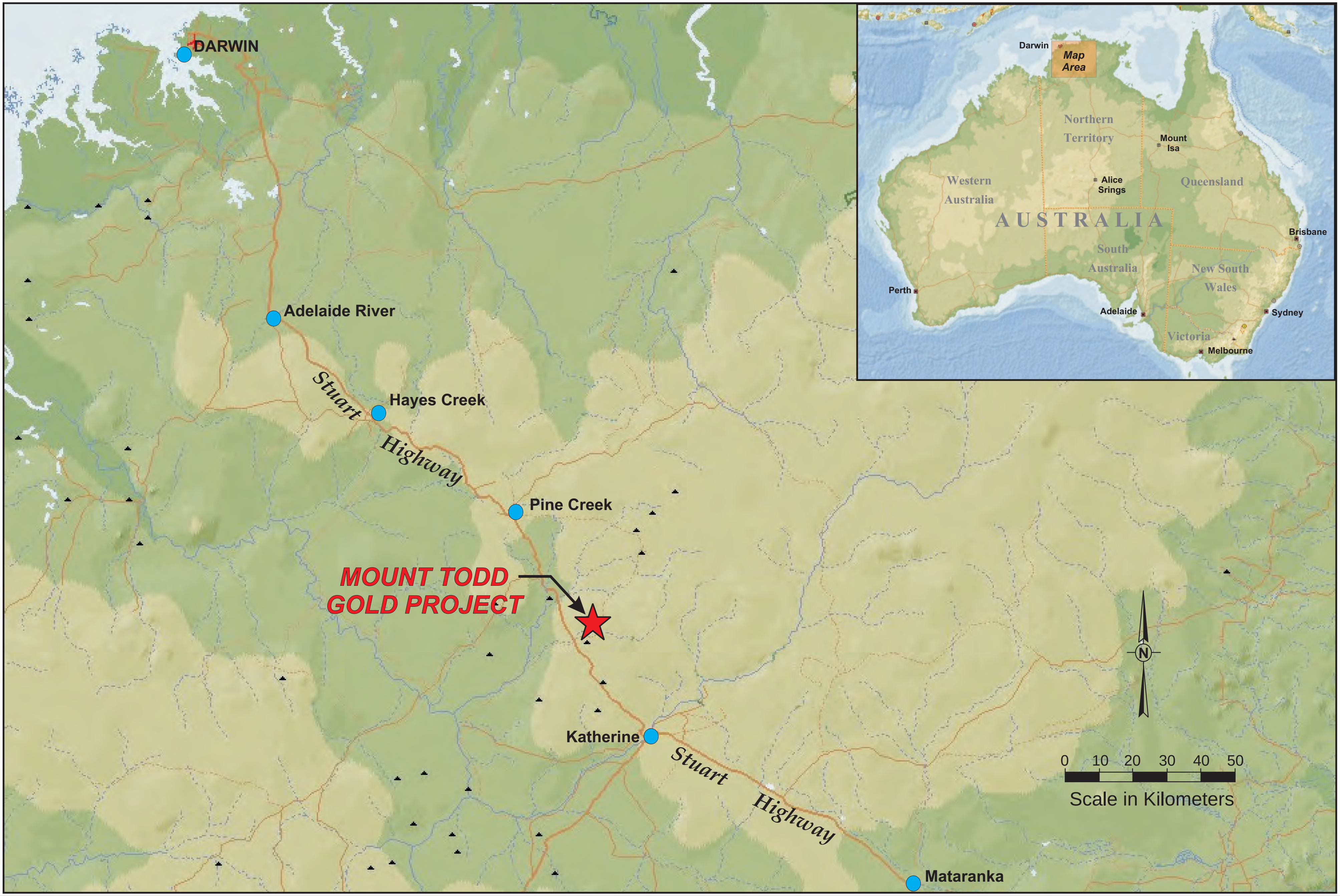
The Mt. Todd Project is located 56 kilometers (km) by road northwest of Katherine, and approximately 250 km southeast of Darwin in NT, Australia (FIGURE 1-1). Access to the property is via high quality, two-lane paved roads from the Stuart Highway, the main arterial within the territory.

1.2 History

The Mt. Todd Gold Project has a long, well-documented history as presented in TABLE 1-1. In addition, it has a well-preserved and meticulously maintained database and supporting file system. The care and quality of these data speak well to the trust and integrity of the resultant studies that have been completed since the deposit was discovered.

While the property operated and closed due to bankruptcy, the failure of the project was not a result of a failure of the deposit and/or the resource estimate. The failure of the project was primarily a result of improper crushing and grinding, accompanied by poor recovery which resulted in higher than expected operating costs, and low gold prices. Had proper bulk sampling and testing been completed, a different process plant would have been built which would have been more appropriate for the deposit conditions.

The Batman resource estimate reconciled very well on a “global” basis, but had difficulties on a local basis. This was primarily due to improper modeling techniques that “over-smoothed” the grades and poor sampling techniques of the blast holes. The improper modeling of the resource was rectified in Vista’s original Technical Report (dated June 26, 2006) when the entire deposit was remodeled. Vista has continued to use modeling procedures that ensure the continued integrity of the resource estimates. Prior to closure in 2000, it appears that all of the sampling problems, as specified by the various consultants and reports, had been addressed and corrected.



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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.dwg	**Figure 1-1** **General Location Map** **Mt. Todd Gold Project**
	Project: Mt. Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	

TABLE 1-1: PROPERTY HISTORY
VISTA GOLD CORP. – MT TODD GOLD PROJECT
May 2009

1986 October 1986 – January 1987:	Conceptual Studies, Australia Gold PTY LTD (Billiton); Regional Screening; (Higgins), Ground Acquisition by Zapopan N.L.
1987 February: June-July: October:	Joint Venture finalized between Zapopan and Billiton. Geological Reconnaissance, Regional BCL, stream sediment sampling. Follow-up BCL stream sediment sampling, rock chip sampling and geological mapping (Geonorth)
1988 Feb-March: March-April: May: May-June: July: July-Dec:	Data reassessment (Truelove) Gridding, BCL grid soil sampling, grid based rock chip sampling and geological mapping (Truelove) Percussion drilling Batman (Truelove) - (BP1-17, 1475m percussion) Follow-up BCL soil and rock chip sampling (Ruxton, Mackay) Percussion drilling Robin (Truelove, Mackay) - RP1-14, (1584m percussion) 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: June: July-Dec:	Batman diamond and RC drilling:BD72-85 (5060m diamond); BP101-208, (8072m RC). Penguin, Regatta, Golf, Tollis Reef Exploration Drilling : PP1-8, PD1, RGP132, GP1-8, BP108, TP1-7 (202m diamond, 3090m RC); TR1-159 (501m RAB). Mining lease application (MLA's 1070, 1071) lodged. Resource Estimates; mining-related studies; Batman EM-drilling: BD12, BD8690 (1375m diamond); RC pre-collars and H/W drilling, BP209-220 (1320m RC); Exploration EM and exploration drilling: Tollis, Quigleys, TP9, TD1, QP1-3, QD1-4 (1141 diamond, 278m RC); Negative Exploration Tailings Dam: E1-16 (318m RC); DR1-144 (701. RAB) (Kenny, Wegmann, Fuccenecco, Gibbs).
1990 Jan-March:	Pre-feasibility 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.	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, 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	Ferrier Hodgson (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 concession rights from the Deed Administrators.

1.3 Ownership

The mineral leases (ML) consist of three individual tenements, MLN 1070, MLN 1071, and MLN 1127 comprising some 5,365 hectares. In addition, Vista controls exploration leases (EL) EL25668, EL25669, EL25576, and EL25670 comprising approximately 117,632 hectares. FIGURE 1-2 illustrates the general location of the tenements and the relative position of the two primary mineral deposits: Batman and Quigleys.

The agreement with the NT is for an initial term of five years commencing January 1, 2006, with an extension of five years at Vista's option and three additional years possible at the option of the NT. During the first five-year term, Vista must undertake a comprehensive technical and environmental review of the project to evaluate current site environmental conditions to develop a program to stabilize the environmental conditions and minimize offsite contamination. Vista must also review the water management plan and make recommendations and produce a technical report for the re-starting of the operations. During the term of the agreement, Vista must 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 entire Mt. Todd Project site.

As part of the agreement, the NT has 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 Mine Management Plan for resumption of mining operations. Vista provided notice to the NT Government in June 2010 that it wished to extend the agreement. In November, the NT Government acknowledged that Vista had fulfilled its obligations for the initial term, and the agreement has been extended for five years until December 31, 2015.

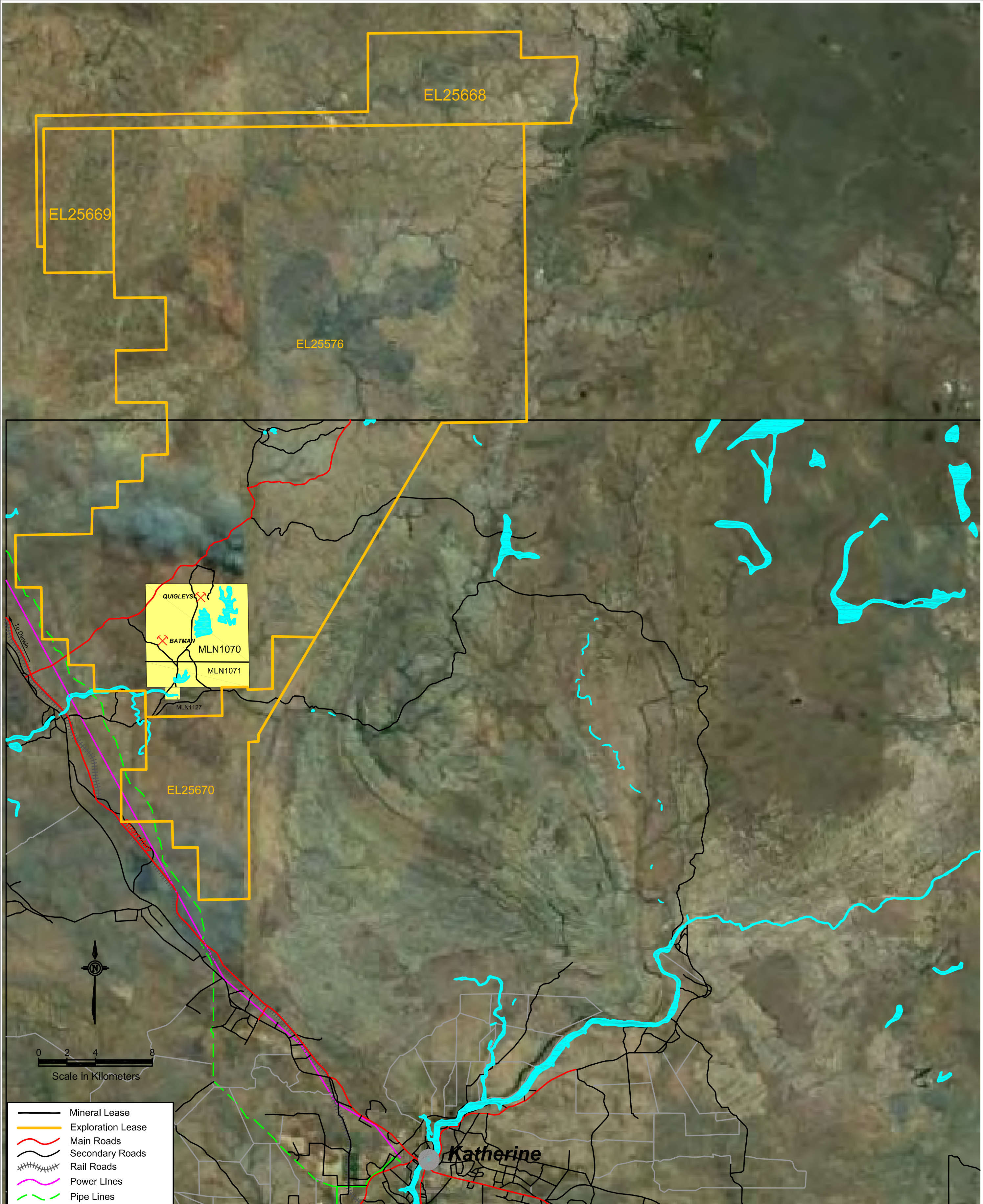
1.4 Geology

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

The 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^o, dipping at 40^o to 60^o 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 Pine Creek Geosyncline (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.



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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 1-2** **Concessions and Infrastructure Map**
	Project: Mt. Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	

1.5 Estimated Resources

At the present time, resources have only been estimated for the Batman and Quigleys deposits. Other deposits are known to be located and, in some cases, possess limited drill hole and other geologic information, but have not been investigated by Vista. Tetra Tech created three-dimensional computerized geologic and grade models of the Batman and Quigleys deposits. While the global model area also contains the Golf-Tollis and several other smaller deposits, no resources have been estimated for these deposits.

Tetra Tech used the geologic model that has evolved over the last few years, as adjusted by each exploration program, to guide the statistical and geostatistical analysis of the gold assay data. This model is a combination of lithologic and alteration data. The rock model was assigned a tonnage factor based on the oxidation state (i.e., oxidized, transition, primary). The tonnage factors are 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.

Estimation has been completed by using whole-block kriging techniques. This is the same estimation procedure as the previous Tetra Tech resource models, adjusted according to each successive drilling program. The estimation is completed as a "two-pass" process. That is, the first pass is for the resources within the main core complex using only data from this zone. The second pass is for the material outside of the main core complex using only assays from outside the core complex. The estimated gold resources were classified into measured, indicated, and inferred categories. The classification was accomplished by a combination of kriging variance, number of points used in the estimate, and number of sectors used. TABLE 1-2 details the results of the classification.

TABLE 1-2: RESOURCE CLASSIFICATION CRITERIA
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

BATMAN (March 2008 & February 2009)			
Category	**Search Range & Kriging Variance**	**No. of Sectors/ Max Pts per DH**	**Min Pts**
Measured	Core Complex: 60 m & KV < 0.30	4/3	4
Indicated	Core Complex: 150 m search & KV >= 0.30 and <0.55	4/2	2
Indicated	Outside Core Complex: 50 m search & KV <0.45	4/3	8
Inferred	Core Complex: 150 m & KV >0.55	4/3	2
Inferred	Outside Core Complex: 150 m & KV <0.45	4/3	3
QUIGLEYS (October 2010)			
Category	**Search Range & Kriging Variance**	**No. of Sectors/ Max Pts per DH**	**Min Pts**
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 & < 0.335 Zone 9999 < 25 m	4/3	3

TABLE 1-3 details the estimated in-place resources by classification and by cutoff grade for the Batman deposit. TABLE 1-4 details the in-place resources by classification and by cutoff grade for the Quigleys deposit. All of the resources quoted are contained on Vista's mineral leases. The Reserve Case cutoff for the resource reporting is 0.4 grams of gold per tonne (g Au/tonne) and is bolded in the table. This cutoff value was determined using the three-year average gold price of $950 in September 2010 and accompanying parameters as presented in TABLE 18-1 of this report. It is important to note that the change in the cutoff grade has resulted in the reporting of significantly more contained gold ounces; however, the gold grade model is unchanged from the February 27, 2009, Technical Report.

TABLE 1-3: BATMAN DEPOSIT CLASSIFIED GOLD RESOURCES
VISTA GOLD CORP. – MT TODD GOLD PROJECT
May 2009

Cutoff Grade g Au/tonne	Tonnes (x1000)	Average Grade g Au/tonne	Total Au Ounces (x1000)
MEASURED (1)			
2.00	1,977	2.38	151
1.75	3,676	2.14	253
1.50	6,469	1.91	398
1.25	10,163	1.71	560
1.00	16,119	1.49	774
0.90	19,764	1.39	885
0.80	24,262	1.29	1,007
0.70	29,616	1.19	1,136
0.60	36,700	1.09	1,284
0.50	44,645	0.99	1,424
0.40	**52,919**	**0.91**	**1,543**
INDICATED (1)			
2.00	3,238	2.49	259
1.75	5,773	2.21	410
1.50	10,140	1.95	637
1.25	17,532	1.70	961
1.00	30,873	1.45	1,437
0.90	39,308	1.34	1,694
0.80	50,410	1.23	1,996
0.70	64,371	1.13	2,332
0.60	82,412	1.02	2,707
0.50	105,936	0.92	3,121
0.40	**138,020**	**0.81**	**3,581**

MEASURED + INDICATED (1, 2)			
2.00	5,215	2.45	410
1.75	9,449	2.18	663
1.50	16,609	1.94	1,035
1.25	27,695	1.71	1,521
1.00	46,992	1.46	2,210
0.90	59,072	1.36	2,578
0.80	74,672	1.25	3,003
0.70	93,987	1.15	3,468
0.60	119,112	1.04	3,991
0.50	150,581	0.94	4,545
0.40	**190,939**	**0.84**	**5,125**

INFERRED RESOURCES			
Cutoff Grade g Au/tonne	**Tonnes (x1000)**	**Average Grade g Au/tonne**	**Total Au Ounces (x1000)**
2.00	2,058	2.76	183
1.75	3,056	2.47	242
1.50	4,808	2.16	333
1.25	7,936	1.84	470
1.00	14,280	1.52	696
0.90	18,878	1.38	836
0.80	25,593	1.24	1,018
0.70	35,885	1.10	1,266
0.60	48,503	0.98	1,529
0.50	66,725	0.86	1,849
0.40	**94,008**	**0.74**	**2,244**

NOTE (1): The sum of measured and indicated resources as reported under NI 43-101 is equivalent to mineralized material under SEC Industry Guide 7.

NOTE (2): These tables contain the resources present that are contained within and without of the pit detailed later (i.e. all possible resources).

At the PFS economic cutoff grade of 0.4 g Au/t, and exclusive of proven and probable resources, the Batman deposit contains some 3,958,000 tonnes of measured resources averaging 0.88 g Au/t containing approximately 112,000 toz of gold; 37,106,000 tonnes of indicated resources averaging 0.76 g Au/t containing approximately 900,000 toz of gold; and, 94,008,000 tonnes of inferred resources averaging 0.74 g Au/t containing approximately 2,244,000 toz of gold. In addition, the Batman deposit also contains some 48,961,000 tonnes of proven reserves averaging 0.91 g Au/t containing approximately 1,431,000 toz of gold and 100,914,000 tonnes of probable reserves averaging 0.83 g Au/t containing approximately 2,681,000 toz of gold.

TABLE 1-4: QUIGLEYS DEPOSIT CLASSIFIED GOLD RESOURCES
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Cutoff Grade g Au/tonne	Tonnes (x1000)	Average Grade g Au/tonne	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 (1)			
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**

NOTE (1): The sum of measured and indicated resources as reported under NI 43-101 is equivalent to mineralized material under SEC Industry Guide 7.

INFERRED RESOURCES			
Cutoff Grade g Au/tonne	**Tonnes (x1000)**	**Average Grade g Au/tonne**	**Total Au Ounces (x1000)**
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**

Exploration Potential

The following discussion details by deposit some of the more important areas that have been identified by Tetra Tech that are likely to result in increases in either the confidence of the resource estimate and/or the amount of the resource estimate for the individual deposits located on the Mt. Todd mineral leases.

Batman Deposit

One of the results of the statistical and geostatistical analysis of the blasthole gold data and resulting creation of independent gold, copper, silver, lead, zinc, iron, and sulfur grade models was the identification of areas within the existing defined deposit that continue to be "under drilled" with regard to classification of the estimated resources. In general, as the depth of the main mineralized host and structure increases, the density of drilling decreases, although the 2008 exploration program did improve the deep drilling. This has resulted in a number of areas that contain no estimated resources, but in all likelihood, based on the geology and surrounding drill hole data, are mineralized and would contain resources if additional drilling were completed.

In addition to these areas, the Batman deposit continues to be open in both the north and south directions. The last fence on the north and south sides of the deposit are mineralized and suggest that more "step-out" drilling is still needed.

Another feature that came to light from the 2007 and 2008 exploration-drilling program is the potential existence of new "parallel and/or sub-parallel" structures and mineralization to the east of the main core complex at the Batman deposit. Both of these parallel and/or sub-parallel structures warrant additional exploration drilling to better define these zones.

Quigleys and Golf-Tollis Deposits

The Quigleys and Golf-Tollis deposits appear to be more structurally controlled than the Batman deposit with the mineralization occurring in narrower bands. Because of this, additional work will need to be undertaken in order to develop a more accurate geologic model and mineralization controls. Tetra Tech proposes that the following items be considered when preparing the work plan:

- Surface mapping and subsequent re-interpretation of the footwall contact relationship to the shear zone mineralization is recommended. Any additional structural complexity that results should, where appropriate, be used to refine the mineralized envelope upon which modeling updates are based;
- Optimization of the resource provides a focus to define areas requiring further investigation or infill drilling. Due to the high degree of variability in the deposit, infill drilling is best targeted at key areas of geological complexity;
- A model should be developed for the area outside the shear zone. This will require separation of areas of mineralization from unmineralized areas using a suitable constraining envelope; and
- The cause of an apparent bias between some of the old and new reverse circulation (RC) drilling should be confirmed to validate the inclusion of all samples in the resource calculation.

1.6 Reserve Case Mine Plan and Mineral Reserves

Potentially mineable pit shapes were evaluated using a Lerchs-Grossman (LG) analysis performed with the GEMS® Whittle pit optimization software and the Mt. Todd mineral resource model. The optimization is an iterative process with initial parameters coming from the Mt. Todd October 1st, 2010 PFS. The final parameters incorporate mining costs developed during this study. The optimization runs used only Measured and Indicated material for processing. All Inferred material was considered as waste. The parameters assumed for the LG analyses are summarized in TABLE 1-5.

TABLE 1-5: RESERVE CASE PARAMETERS FOR LERCHS-GROSSMAN ANALYSES
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Overall Pit Slopes	33^o from pit centered azimuth ranging 10^o – 150^o 55^o from pit centered azimuth ranging 150^o – 10^o
Gold Price	US$1000 per toz Au
Gold Recovery	82 percent
Mining Cost	US$1.40 per tonne mined
Processing Cost	US$7.60 per tonne processed
Tailings Construction	$1.00 per tonne processed
Tailings Reclamation	$1.14 per tonne processed
Waste Dump Rehabilitation	$0.12 per tonne waste
General and Administrative Cost	US$0.60 per tonne processed

The Reserve Case LG shell is defined by the economic factors listed in TABLE 1-5. 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.

Using the Reserve Case, the ultimate pit was designed as an open-pit mine using large haul trucks, hydraulic shovels, and front-end loading equipment. Primary production is achieved using 21 cubic meter hydraulic shovels along with 180 tonne haul trucks. This equipment is used primarily for the movement of waste material.

Secondary production is achieved using a CAT 992 loader and smaller CAT 785C trucks. The 992 loader is assumed to have a 12 cubic meter bucket, and the CAT 785C trucks have a rated payload of 140 tonnes. The loader and smaller trucks are used primarily to move ore from the pit to the crusher and for reclaiming ore from stockpiles. Waste production from the 992 loader and 785C trucks is anticipated as well.

After the ultimate pit was designed, pits or phases within the ultimate pit were designed to enhance the project by providing higher-value material to the process plant earlier in the mine life. The design includes smoothed pit walls, haulage ramps, benches, and pit access. Phase 1 and phase 2 pit designs remain unchanged from the previous PFS work. Phase 3 was designed to the ultimate pit limit on the south, while phase 4 (the final pit phase) is used to achieve the ultimate pit in the north.

TABLE 1-6: CLASSIFICATION OF RESERVE CASE MINEABLE RESERVES
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Class	Ore Tonnes (x 1000)	Average Gold Grade (gm/t)	Contained Gold (oz x 1000)	Waste Tonnes (x 1000)	Total Tonnes (x 1000)	Stripping Ratio (W:O)
Proven	48,961	0.91	1,431			
Probable	100,913	0.83	2,681			
Proven + Probable	149,874	0.85	4,112	271,480	421,354	1.81

Note: Reserves are reported using a 0.40 g Au/t cutoff grade.

The Reserve Case production schedule for this PFS assumes a 10.65 Mtpy ore production rate, resulting in a 14-year operating life, as shown in TABLE 1-7.

TABLE 1-7: RESERVE CASE PRODUCTION SCHEDULE
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Year	"Ore" Tonnes (x 1000)	Avg. Grade (g Au/tonne)	Waste Tonnes (x 1000)	Stripping Ratio (W:O)
PP1	1,084	0.68	6,287	5.80
1	12,210	0.86	22,965	1.88
2	13,584	0.90	25,048	1.84
3	11,997	0.90	24,400	2.03
4	10,650	0.95	25,578	2.40
5	6,200	0.71	27,824	4.49
6	8,175	0.67	25,041	3.06
7	13,198	0.79	24,662	1.87
8	11,158	0.76	24,710	2.21
9	8,990	0.66	22,655	2.52
10	13,626	0.78	20,386	1.50
11	12,102	0.86	14,158	1.17
12	13,379	0.93	5,940	0.44
13	11,310	1.09	1,805	0.16
14	2,213	1.40	22	0.01
Total	**149,875**	**0.85**	**271,480**	**1.81**

1.7 Limestone Quarry and Lime Production

Limestone is currently commercially produced near Katherine by quarrying the Katherine limestone beds. The Mt. Todd operation plans to ensure a supply of economic lime is available for use in the processing and water treatment areas of the operation. A limestone quarrying operation will be developed by mining a nearby outcropping of Katherine Limestone; a lime kiln plant will be established at the quarry to convert the limestone into lime.

1.8 Power Supply

The Power Engineers report, "Mt. Todd Power Station, Phase 3 Pre-Feasibility Study," dated September 30, 2010, provides a detailed discussion of the generation equipment options available for onsite electrical supply to meet the power requirements of the re-commissioned Mt. Todd Gold Mine in NT, Australia operated by Vista. The site electrical power demands are a fixed constant operating load estimated at 46 megawatts (MW) with a minimum of startup/shutdown cycles. This load falls between gas turbine size categories so surplus generating capacity is expected if the load is met with a single turbine.

The cost analysis for this study is based on a 13-year operating plant life without annual pricing index. Fuel costs are based on a rate of $5.75 (AUS) per gigajoule. Calculated 13-year project life costs (includes all capital and operating costs) are estimated $0.0710 to 0.0950 (AUS) per kilowatt-hour for the 46 MW site demand compared to the commercially purchased electricity rate of $0.1636 per kilowatt-hour (kWh) (adjusted for demand) for the same time period.

Five options were considered for generation of power at Mt Todd. A Rolls Royce Trent 60 WLE was selected for use in this study. This unit will generate power at a direct operating cost averaging $0.0629 (AUD) per kWh over the life of the project.

1.9 Processing and Process Flowsheet

The Mt. Todd gold recovery process evolved both historically and through studies commissioned by Vista from Resource Development, Inc. (RDi). The evolved process uses proven technologies to recover 82 percent of the contained gold by carbon in leach (CIL) leaching. For purposes of this PFS, an ore feed grade of 1.08 grams per tonne (g/t) and an Ausenco adjusted plant feed rate of 1,427 tonnes per hour (t/h) (nominally 30,000 tonnes per day [tpd] or 10.65 Mtpy) was used. Note that Ausenco frequently describes their work as the "11Mtpy Engineering and Cost Study."

Testwork at RDi on samples provided by Vista supports a process using conventional coarse crushing followed by HPGR crushing and ball mill grinding to produce a leach feed at P_{80} 150 micrometer (µm) (100 mesh Tyler). The resulting pulp is then pre-aerated and subjected to CIL leaching followed by adsorption, desorption, and recovery (ADR) leading to gold doré. The CIL tailings are detoxified and sent to an impoundment, from which plant process water is recycled. The process is robust.

1.10 Tailings Disposal

A tailings disposal tradeoff study was completed in early 2010 in order to explore several options for tailings disposal, such as a dry stack facility, new tailings storage facility (TSF) designs for both thickened and conventional tailings, and several raises to the existing TSF. The 60 million tonne capacity raise to the existing TSF design (TSF1) was originally selected based on economic tradeoff studies and the relatively low cost per tonne of tailings stored. Since the total required tailings storage for the project is 150 million tonnes, a new TSF (TSF2) has been designed to provide an additional 100 million tonnes of tailings storage. This provides extra storage as a contingency.

The design for the raises to TSF1 was adapted from the MWH design completed in 2006, with some modifications to accommodate the projected capacity of the facility. The facility will be constructed in six separate stages, using centerline construction techniques for the first raise and upstream construction techniques for subsequent raises. The embankments will be constructed with 2.5:1 (horizontal [H] to vertical [V]) downstream slopes and 2:1 (H:V) upstream slopes. Three saddle dams will be constructed to contain the tailings on the west side of the facility. It was assumed that all of the existing toe drains, under-drains, and decant towers installed at the existing facility will be fully operational when tailings deposition begins and that minimal construction will be required to raise or extend the drains and towers to the required elevation at each stage.

TSF2 will be completed in four construction stages using upstream raise construction methods. The embankments will be constructed with 3:1 (H:V) upstream and downstream inter-bench slopes and five-meter wide benches at the downstream crest of each stage, yielding an overall slope of 3.2:1 (H:V). The crest will be 30 m wide and will slope at 0.5 percent from the high point in the southeast corner to the tie-in with existing ground near Mt. Todd. The facility will be fully lined and will include a system of toe drains, under-drains, and over-drains, as well as a new water reclaim system. A small surface water diversion will be constructed at the southwest corner of the proposed facility to direct Horseshoe Creek away from the new TSF footprint.

1.11 Environmental Conditions

The primary environmental issue at the Mt. Todd site is water management resulting from the project shutdown without implementation of closure or reclamation activities. All of the water retention ponds (excluding the raw water pond) and the pit contain acidic (~pH 3-4.5) water with elevated concentrations of regulated constituents.

1.11.1 Permitting

In 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. 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 Notice of Intent (NOI) for resumption of mining operations. A decision on the appropriate permitting route will be initiated by submission of an NOI to the Department of Regional Development, Primary Industry, Fisheries and Resources (DRDIPFR), now the Department of Resources (DoR).

A referral and assessment process will determine how the Environment Protection and Biodiversity Conservation Act (EPBC Act) will be applied. The EPBC Act addresses the protection of matters of national environmental significance which include flora, fauna, ecological communities and heritage places. If significant impacts are likely to occur, the project will require formal assessment either through preparation of a Public Environmental Report (PER) or an Environmental Impact Statement (EIS).

1.11.2 Water Management

Current and historic evidence indicates that Mt. Todd waste rock, ore, and tailings contain sulfides capable of generating acid and metal laden leachates (ARD/ML). ARD/ML currently occurs or is found in the waste rock dump and associated pond (RP1), the lean ore stockpile and associated pond (RP2), exposed pit walls and associated pit lake (RP3), the heap leach pad (HLP) and associated pond and moat, the plant runoff pond (RP5), and within the tailings storage facility (RP7).

The Edith River and tributaries are protected beneficial use under the Water Act 2000 for aquatic ecosystem protection. As a result, discharges from the site are regulated under the Mt. Todd Project Waste Discharge License (WDL 135) which allows controlled discharges from RP1 to the Edith River during high flow events. The impacted water is sufficiently diluted during high flow events to ensure downstream compliance with established copper criteria which in turn dilutes other regulated constituents to acceptable levels. Improvements to the water management system have reduced uncontrolled discharges during the wet seasons.

In August 2009, Vista commissioned a water treatment plant (WTP) to treat ARD/ML water at a capacity of 193 cubic meters per hour (m^3/hr). Pilot studies showed that lime treatment removed 98 percent of the cadmium, 98.8 percent of aluminum, and greater than 99 percent of the copper and zinc in acidic water from the waste rock dump pond (RP1). The treated solution including the reaction by-products (gypsum and metal hydroxide compounds) flows by gravity to the tailings storage facility (RP7). Testing is underway to define the operational conditions required to meet standards to discharge treated water after clarification either on a continuous basis or during the wet season. Based on recent measurements (flow meter installed in the Existing WTP influent pipe in December 2010), ARD/ML is treated at a rate of approximately 360 m^3/hr (HydroGeoLogica, Inc. and Tetra Tech, 2010).

1.11.3 Baseline Studies

Site characterization studies were conducted at the Mt. Todd site in support of the 1992 Draft EIS (Zapopan, 1992). Vista is conducting additional baseline studies as required by the site waste discharge license and to support design, permitting, operations, and closure. Baseline studies currently being conducted or to be implemented include:

- Surface water and groundwater characterization;
- Soils;
- Geochemical characterization;
- Biological resources (aquatic and benthic, vegetation and wildlife);
- Cultural and archaeology; and
- Socio-economics.

These environmental baseline studies can be completed within one year or less.

1.11.4 Reclamation and Closure

The major and immediate environmental challenges for Mt. Todd are the management of ARD/ML currently contained in several water storage facilities and the management of precipitation and surface water runoff reporting to mine-related surface disturbance. ARD/ML is currently managed through a combination of practices including evaporation, active water treatment, pumping excess water to the Batman Pit, and controlled and uncontrolled discharges to creeks in the vicinity of Mt. Todd and the Edith River during major flow events. Recent upgrades to the pumping system have reduced the frequency of uncontrolled effluent releases from the ponds to the Edith River and its tributaries.

Throughout the mine-life, Vista should anticipate, plan, design for, and implement effective plans for:

- Year-round collection, containment, and treatment of all ARD/ML prior to effluent release;
- Identification of potentially acid-generating (PAG) and non-PAG materials, as well as materials that have the potential to leach constituents in concentrations above applicable water quality-based effluent standards (metalliferous);
- Selective handling of PAG and non-PAG material and potentially direct treatment of PAG materials throughout the mine-life to prevent or reduce the generation of ARD/ML;
- Separation of unimpacted surface and ground water from PAG and metalliferous materials, and ARD/ML;

- Short- and long-term hydrologic isolation of PAG and metaliferous materials from ground and surface water;
- Facility and site-wide closure; and
- Control of storm-water to prevent excessive erosion and sedimentation.

Specific recommendations related to these and other closure and water treatment needs are provided in Section 21-Recommendations.

The major facilities that currently exist at Mt. Todd, which are included as part of the 10.65 Mtpy mine plan, are as follows:

- Batman Pit;
- Batman Pit Lake (RP3);
- Waste Rock Dump (WRD);
- WRD Pond (RP1) and pumping system;
- TSF;
- TSF Pond (RP7);
- Process Plant and Operations Area;
- Process Plant Runoff Pond (RP5) and pumping system;
- HLP;
- HLP Pond and pumping system;
- Low Grade Ore Stockpile (LGO);
- LGO Pond (RP2) and pumping system;
- Existing Water Treatment Plan (WTP); and
- Mine roads and other ancillary facilities (e.g., pipelines).
- The new facilities proposed for closure and the mine-life water treatment system are as follows:
- Run-on diversions up-gradient of the RP1, TSF1, and WRD;
- New WTP;
- Linear Low Density Polyethylene (LLDPE) (or equivalent)-Lined Equalization Pond;
- LLDPE (or equivalent)-Lined Sludge Disposal Cell;
- TSF1 and TSF2 Closure Spillways;
- Modified TSF1 Decant Ponds;
- Modified TSF2 Sumps;
- LLDPE (or equivalent)-Lined TSF1 Collection Ditch;
- LLDPE (or equivalent)-Lined TSF2 Collection Ditch;
- LLDPE (or equivalent)-Lined LGO2 Collection Ditch;
- LLDPE (or equivalent)-Lined LGO2 Sump;
- Collection Ditch at toe of closed WRD;

- Modified HLP Seepage Collection Pump and Pipeline;
- Pumps and pipelines;
- Clay Borrow Area; and
- Three Anaerobic treatment wetlands (or equivalent passive/semi-passive water treatment system).

A PFS-level Closure Plan (PFCP) is included as an appendix (Appendix J) to the PFS. The PFCP includes descriptions, approximate dimensions, and performance criteria for proposed facilities. Arrangements and design drawings and details for these facilities have not been completed at this stage of the planning process.

The closure and water management goals for Mt. Todd include:

- Control acid-generating conditions;
- Reduce or eliminate the acid and metal loads of seepage and runoff water;
- Minimize adverse impacts to the surface and ground water systems surrounding Mt. Todd;
- Physical and chemical stabilization of mine waste and other mine-related surface disturbances;
- Protect public safety;
- Comply with the WDL and applicable Edith River water quality-based effluent standards; and
- Comply with NT Government regulations governing mine development and closure.

Closure plans and strategies for each major facility at Mt. Todd and the mine-life water treatment system are summarized in Appendix J.

Closure and water treatment costs were estimated at a ± 25 percent level of accuracy based on the following:

- 10.65 Mtpy mine plan and existing engineering and data presented in the PFS;
- Geochemical testing program and results (Appendix H);
- Mine-life (i.e., pre-production phase of 2 years, production phase of 15 years, closure phase of 3 years, post-closure phase of 6 years) water balance simulations, water quality estimates, and water management plans (Appendix I);
- Use of existing and new water management systems and infrastructure;
- Estimates of environmental conditions throughout the mine-life;
- NT Government mine closure and environmental protection regulations and guidelines;
- Published unit costing references;
- Tetra Tech’s recent mine closure and water treatment costing experience; and
- Best professional judgment.

As summarized in TABLE 1-8 the PFS-level cost estimates for implementing the closure and mine-life water treatment plans are $67,864,000 and $36,590,000, respectively.

TABLE 1-8: PREFEASIBILITY-LEVEL CLOSURE AND MINE-LIFE WATER TREATMENT COST ESTIMATE
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Area	Cost[1]
Tailings Storage Facility 1 (TSF1)	$ 9,101,000
Tailings Storage Facility 2 (TSF2)	$ 19,018,000
Heap	$ 2,585,000
Process Plant And Pad Area	$ 11,280,000
Batman Pit	$ 205,000
Waste Rock Dump	$ 8,620,000
WRD Retention Pond	$ 1,709,000
Low Grade Ore Stockpile 1 (LGO1)	$ 128,000
Low Grade Ore Stockpile 2 (LGO2)	$ 244,000
Mine Roads	$ 3,786,000
Clay Borrow Area	$ 135,000
Sludge And Equalization Pond Closure	$ 273,000
Total Direct Closure Cost	**$ 57,084,000**
Mobilization/Demobilization (Assume On-Site Mining Equipment Fleet Used)	$ 0-
Incidentals (Communication, Misc. Supplies, Etc.) = 0.5 % Of Total Direct Cost	$ 385,000
Haul Road Maintenance During Closure = 0.5 % Of Total Direct Cost	$ 385,000
Engineering Re-Design = 2 % Of Total Direct Cost	$ 1,540,000
Contingency = 8 % Of Total Direct Cost	$ 6,160,000
Total Indirect Cost [2]	**$ 8,470,000**
Annual Site Maintenance and Monitoring For 6 Years Post Closure	$ 2,310,000
Total Closure Cost	**$ 67,864,000**
Water Treatment System Facility/Component	
Active Water Treatment And Sludge Disposal System Construction	$ 4,169,000
Passive Water Treatment System #1, #2 & #3	$ 15,314,000
Total Direct Water Treatment Construction Cost	**$ 19,483,000**
Pre-Production Period (Years -2 and -1) Water Treatment O&M, Reagent and Pumping[3]	$ 5,545,000
Production Period (Years 1 through 15) Water Treatment O&M, Reagent and Pumping[3]	$ 6,125,000
Closure Period (Years 16 through 18) Water Treatment O&M, Reagent and Pumping[3]	$ 2,612,000
Post-Closure Production Period (Years 19 through 24) Water Treatment O&M, Reagent and Pumping[3]	$2,825,000
Total Mine-Life Water Treatment O&M, Reagent and Pumping[3]	**$ 17,107,000**
Total Mine-Life Water Treatment Costs	**$ 36,590,000**

1 Cost rounded to nearest $1,000 in current $.

2 Includes indirect costs associated with the construction of Water Treatment System

3 Includes Plant O& M, Lime, and Water and Sludge Pumping

The major closure and water treatment assumptions used for the development of the closure plan are provided in Appendix J and summarized in Section 5.4-Environmental Conditions.

1.12 Economic Evaluation

The financial results presented in this PFS have been developed co-operatively between Vista, Tetra Tech, and other consultants. The financial results are presented in constant dollars with the mine and mill capital having been estimated in the second and fourth quarters of 2010, respectively. A five percent discount rate has been applied to the financial analysis. Besides the Reserve case, sensitively analyses were completed using varying gold prices, currency exchange rates, capital cost estimates and operating cost estimates. Unless otherwise noted, an US/AUD conversion rate of 0.85 was used. Unless specifically noted, all monetary values in the entire document are in US dollars.

1.12.1 Reserve Case

The Reserve Case project entails mining 149,875,000 ore tonnes over a 15-year period. The scenario requires that 10.65 Mtpy ore be mined and processed assuming $1,000/toz Au, an exchange rate of 0.85 US/AUD dollars, and metallurgical recoveries of 82 percent. Note that the actual 3-year average gold price is $1,023/toz Au; however, both Tetra Tech and Vista agreed to use $1,000/toz Au for the Reserve Case analysis.

1.12.2 Capital Costs

Estimated capital expenditures for the life-of-mine Reserve Case are estimated to be $851.1 million; this being a combination of $589.6 million start-up capital and $261.5 million sustaining capital, both including working capital and contingency. TABLE 1-10 provides a summary of the project capital over the life of the proposed operation.

1.12.3 Mine Operating Costs

Mine operating costs have been estimated for each year of operations based on production requirements with the estimates comprising labor, fuel, material, equipment, and maintenance. A summary of the mine operating costs per tonne ore processed are presented in TABLE 1-9 for the 10.65 Mtpy Reserve Case.

1.12.4 Process Operating Costs

The Reserve Case process operating costs range from $6.76 to $6.79/t ore during the years of operation. Included in these costs are operating expenses for the water treatment and tailings facilities. The process plant operating costs by year are given in TABLE 1-11.

TABLE 1-9: MINE OPERATING COST SUMMARY (000) PER TONNE ORE PROCESSED
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Ore Mined	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	775
Total mining costs	50,882	55,947	55,555	55,046	49,107	41,713	59,865	46,330	32,800	58,451	23,991	39,725	29,086	9,747	1,145
Mine Operating Cost / tonne	$4.78	$5.25	$5.22	$5.17	$4.61	$3.92	$5.62	$4.35	$3.08	$5.49	$2.25	$3.73	$2.73	$0.92	$1.48

TABLE 1-10: SUMMARY OF PROJECT CAPITAL COSTS (000)
VISTA GOLD CORP. - MT. TODD GOLD PROJECT
January 2011

CAPITAL ($000'S)	LOM	INITIAL	SUSTAINING
MINE CAPITAL			
Primary:			
Open Pit Mine Equipment	98,792	46,483	52,309
Lime Operation Mine Equip	5,617	5,617	0
Sub-Total Primary	**104,409**	**52,100**	**52,309**
Ancillary:			
General Surface Mobil Equipment	18,596	8,404	10,191
Sub-Total Ancillary	**18,596**	**8,404**	**10,191**
Miscellaneous:			
Mine Office, Shop and Warehouse	2,268	2,268	0
Mining Development Supply and Labor Op Costs	9,394	9,394	0
Sub-Total Miscellaneous	**11,662**	**11,662**	**0**
TOTAL MINE CAPITAL (Before Contingency)	**134,667**	**72,166**	**62,500**
Mine Capital Contingency	**9,759**	**5,615**	**4,144**
PLANT CAPITAL			
Process Plant	269,243	269,243	0
Onsite Infrastructure	22,503	22,503	0
Mobile Equipment, Spares, First-Fills	11,223	11,223	0
Power Generating Station	37,678	37,678	0
Site Demolition	3,664	3,664	0
TAILING STORAGE FACILITIES CAPITAL			
Pre-production WTF + Tailings Management	4,777	4,777	0
TSF Fine Grading, Equipment, Piping, Drains	71,304	5,258	66,046
TSF Bulk Earthwork	88,555	4,193	84,362
TOTAL PLANT + TAILINGS STORAGE	**508,948**	**358,539**	**150,408**
INDIRECT PROCESS			
Temporary Construction Facilities	6,999	6,999	0
Commissioning	5,599	5,599	0
Total Indirect Process	**12,598**	**12,598**	**0**
TOTAL PLANT + TAILING + INDIRECT CAPITAL (Before Contingency)	**521,546**	**371,137**	**150,408**
Plant Capital Contingency	**60,208**	**51,202**	**9,006**
EPCM TOTAL (PLANT & TAILING)	**73,504**	**68,600**	**4,904**
OTHER CAPITAL			
Off-site Infrastructure / Accommodation Village	16,268	16,268	0
Excess Water Treatment Facility	17,985	0	17,985
Permitting	2,500	2,500	0
Recruit and Training	1,700	1,500	200
Lime Kiln/Processing	6,158	6,158	0
Total Other Capital	**44,611**	**26,426**	**18,185**
Other Capital Contingency	**6,692**	**3,964**	**2,728**
Total Contingency	**76,659**	**60,781**	**15,878**
TOTAL CAPITAL	**850,987**	**599,111**	**251,876**
TOTAL WORKING CAPITAL CHANGES	**102**	**(9,528)**	**9,630**
TOTAL CAPITAL + WORKING CAPITAL CHANGES	**851,088**	**589,583**	**261,506**

NOTE: Some rounding may occur due to truncation of the numbers.

TABLE 1-11: PROCESS OPERATING COST SUMMARY (000)*
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Ore Processed	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	775
Total processing costs	72,159	72,109	72,120	72,080	72,169	72,200	72,366	72,286	72,277	72,213	72,213	72,201	72,019	72,068	5,535
Ore Processing Cost / tonne	$6.78	$6.77	$6.77	$6.77	$6.78	$6.78	$6.79	$6.79	$6.79	$6.78	$6.78	$6.78	$6.76	$6.77	$7.14

*Note: Gold doré refining, transport and treatment charges are $4.50/toz Au, but are included separately in the cash flow analyses.

1.12.5 Cash Flow Analyses

The cash flow analysis developed for the Reserve Case includes all mining, processing, tails disposal, and reclamation.

Cash flow analyses at $1,000/toz Au and a US/AUD exchange rate of 0.85 results in a project pretax NPV of $385.336 million and a pre-tax Internal Rate of Return of 13.9 percent and a post-tax rate of return of 10.7 percent, both evaluated at a 5 percent discount rate. Note that 3,371,914 toz Au are recovered during the operating life. TABLE 1-12 is the cash flow associated with the Reserve Case scenario.

1.12.6 Sensitivity Gold Price Sensitivities

Gold Price sensitivity analyses were performed on the Reserve Case reflecting Au prices from $850 to $1,150 in increments of $50. A graph showing the results of these sensitivities is shown in FIGURE 1-3.

TABLE 1-12: MT TODD 10.65 MTPY RESERVE CASE, VISTA GOLD CORP - MT TODD GOLD PROJECT, January, 2011

Mt. Todd - 10.65Mtpa (28 January 2011)

PRETAX:	
IRR	13.9%
NPV0 (000'S)	$964,514
NPV5 (000'S)	$385,336
AVG ANNUAL CF (000's) PRODUCTION YEARS	$97,094
AVG ANNUAL CF (000's) LIFE OF MINE	$56,016
STRIPPING RATIO (WST:ORE)	1.81

AFTER-TAX:	
IRR	10.7%
NPV0 (000'S)	$584,562
NPV5 (000'S)	$184,312
AVG ANNUAL CF (000's) PRODUCTION YEAR	$71,764
AVG ANNUAL CF (000's) LIFE OF MINE	$41,403
PAYBACK PERIOD (YRS) FROM : START OF PRODUCTION	7.2
POST CLOSURE NET CASH FLOW:	$92,460

CAPITAL	
INITIAL CAPITAL (000'S)	$538,330
CONTINGENCY	$60,781
SUB-TOTAL	599,111
WORKING CAPITAL - YR -2 TO YR 1	(9,528)
INITIAL CAP, PRE-PROD DEV & WORKING CAP	$589,583
SUSTAINING CAPITAL (000'S)	235,998
CONTINGENCY	15,878
TOTAL SUSTAINING CAPITAL	251,876
WORKING CAPITAL - YR 2 TO YR 15	9,630
TOTAL MINE LIFE CAPITAL	$851,088

COSTS	
CASH OPER COST PER OUNCE	$520
TOTAL CASH COST PER OUNCE	$530
CAPITAL COST PER OUNCE	$231
TOTAL PRODUCTION COST PER OUNCE	$761
UNIT COSTS	
MINING COST ($/TONNE MINED)	$1.68
MINING COST ($/TONNE ORE)	$4.07
PROCESSING COST ($/TONNE ORE)	$6.847
G&A Cost ($/TONNE ORE)	$0.55
TOTAL OPERATING COSTS $/TONNE ORE	$11.47

PROJECT PRODUCTION SCHEDULE / GOLD GRADES AND CONTENT

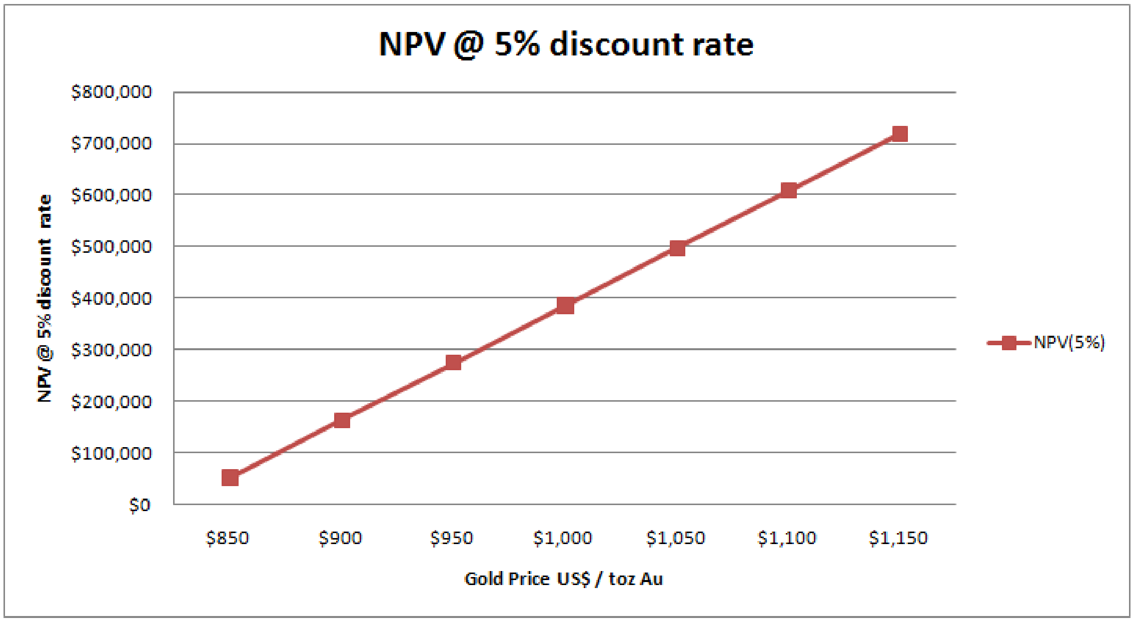
		Total LOM	Project Year -2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
MINE																													
ORE TONNAGE TO CRUSHER (000's)	ore tonnes	149,875			10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	775										
ORE GRADE	g Au/tonne	0.853			0.93	1.02	0.95	0.95	0.61	0.62	0.87	0.77	0.63	0.86	0.92	1.04	1.13	0.66	0.47										
	toz Au/tonne	0.027			0.030	0.033	0.031	0.031	0.020	0.020	0.028	0.025	0.020	0.028	0.030	0.034	0.036	0.021	0.015										
CONTAINED GOLD	g Au	127,900,394			9,950,173	10,876,180	10,118,272	10,143,927	6,472,261	6,611,008	9,267,222	8,213,125	6,701,086	9,193,713	9,779,633	11,127,334	12,003,292	7,081,749	361,418										
	toz Au	4,112,090			319,905	349,677	325,310	326,135	208,088	212,549	297,948	264,058	215,445	295,585	314,422	357,752	385,914	227,683	11,620										
WASTE TONNAGE MINED (000's)	waste tonnes	271,480		6,287	22,965	25,048	24,400	25,578	27,824	25,041	24,662	24,710	22,655	20,386	14,158	5,940	1,805	22											
CAPITALIZED TONS (included in total material mined)	kt	57,954		6,287		700	340	360	5,795	10,200		6,972	13,200		13,833	267													
TOTAL MATERIAL MINED	total tonnes	421,354		6,287	33,615	35,698	35,050	36,228	38,474	35,691	35,312	35,360	33,304	31,036	24,808	16,590	12,455	10,672	775										
STRIPPING RATIO	waste : ore	1.8			2.16	2.35	2.29	2.40	2.61	2.35	2.32	2.32	2.13	1.91	1.33	0.56	0.17	0.00											
MILL																													
ORE TONNAGE TO MILL (000's)	ore tonnes	149,875			10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	775										
MILL FEED GRADE	g Au/tonne	0.853			0.93	1.02	0.95	0.95	0.61	0.62	0.87	0.77	0.63	0.86	0.92	1.04	1.13	0.66	0.47										
	toz Au/tonne	0.027			0.030	0.033	0.031	0.031	0.020	0.020	0.028	0.025	0.020	0.028	0.030	0.034	0.036	0.021	0.015										
CONTAINED GOLD	g Au	127,900,394			9,950,173	10,876,180	10,118,272	10,143,927	6,472,261	6,611,008	9,267,222	8,213,125	6,701,086	9,193,713	9,779,633	11,127,334	12,003,292	7,081,749	361,418										
	toz Au	4,112,090			319,905	349,677	325,310	326,135	208,088	212,549	297,948	264,058	215,445	295,585	314,422	357,752	385,914	227,683	11,620										
MILL RECOVERY @ 82%	% recovery of Au	82%			82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%										
GOLD RECOVERED	g Au	104,878,323			8,159,142	8,918,467	8,296,983	8,318,020	5,307,254	5,421,027	7,599,122	6,734,763	5,494,890	7,538,845	8,019,299	9,124,414	9,842,699	5,807,034	296,362										
	toz Au	3,371,914			262,322	286,735	266,754	267,430	170,632	174,290	244,317	216,527	176,665	242,379	257,826	293,357	316,450	186,700	9,528										
REFINERY																													
PAYABLE GOLD TO REFINERY	g Au	104,878,323			8,159,142	8,918,467	8,296,983	8,318,020	5,307,254	5,421,027	7,599,122	6,734,763	5,494,890	7,538,845	8,019,299	9,124,414	9,842,699	5,807,034	296,362										
	toz Au	3,371,914			262,322	286,735	266,754	267,430	170,632	174,290	244,317	216,527	176,665	242,379	257,826	293,357	316,450	186,700	9,528										

		Total LOM	Project Year -2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
GOLD PRICE	$/oz	$1,000			$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000										
WASTE TONNES	000's	271,480		6,287	22,965	25,048	24,400	25,578	27,824	25,041	24,662	24,710	22,655	20,386	14,158	5,940	1,805	22											
TONNES ORE TO MILL	000's	149,875			10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	775										
STRIPPING RATIO	waste:ore	1.81			2.16	2.35	2.29	2.40	2.61	2.35	2.32	2.32	2.13	1.91	1.33	0.56	0.17	0.00											
OUNCES PAYABLE	toz Au.	3,371,914			262,322	286,735	266,754	267,430	170,632	174,290	244,317	216,527	176,665	242,379	257,826	293,357	316,450	186,700	9,528										
GOLD GRADE	g/tonne	0.853			0.934	1.021	0.950	0.952	0.608	0.621	0.870	0.771	0.629	0.863	0.918	1.045	1.127	0.665	0.466				-	-	-	-	-	-	
GROSS GOLD SALES	$000's	$3,371,914			$262,322	$286,735	$266,754	$267,430	$170,632	$174,290	$244,317	$216,527	$176,665	$242,379	$257,826	$293,357	$316,450	$186,700	$9,528										
RENTAL INCOME/POWER INCOME	$000's	$208,312			$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$16,159	$16,256	$16,265	$16,478	$16,478	$16,478	$16,490	$16,531		
GROSS REVENUE	$000's	$3,580,225			$267,467	$291,880	$271,899	$272,575	$175,777	$179,435	$249,462	$221,672	$181,810	$247,524	$262,971	$298,501	$321,595	$191,845	$14,673	$16,159	$16,256	$16,265	$16,478	$16,478	$16,478	$16,490	$16,531		
LESS REFINING, TRANS. & TREATMENT	$000's	15,174			1,180	1,290	1,200	1,203	768	784	1,099	974	795	1,091	1,160	1,320	1,424	840	43										
REVENUE FROM SALES	$000's	3,565,052			266,287	290,590	270,698	271,372	175,009	178,650	248,363	220,698	181,015	246,433	261,811	297,181	320,171	191,005	14,630	16,159	16,256	16,265	16,478	16,478	16,478	16,490	16,531		
LESS ROYALTY JAAC	$000's	33,719			2,623	2,867	2,668	2,674	1,706	1,743	2,443	2,165	1,767	2,424	2,578	2,934	3,164	1,867	95										
NET REVENUE	$000's	$3,531,333			$263,664	$287,722	$268,031	$268,697	$173,303	$176,908	$245,920	$218,533	$179,248	$244,010	$259,233	$294,248	$317,006	$189,138	$14,535	$16,159	$16,256	$16,265	$16,478	$16,478	$16,478	$16,490	$16,531		
NET REVENUE AFTER PRODUCTION	$131,138																												

		Total LOM	Project Year -2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
OPERATING COSTS																													
MINE	$000's	609,389			50,882	55,947	55,555	55,046	49,107	41,713	59,865	46,330	32,800	58,451	23,991	39,725	29,086	9,747	1,145										
MILL	$000's	1,026,251	2,291	3,254	72,159	72,109	72,120	72,080	72,169	72,200	72,366	72,286	72,277	72,213	72,213	72,201	72,019	72,068	5,535	944	838	830	377	377	377	364	317	268	
G&A	$000's	82,786		5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	548										
RECLAMATION	$000's	67,864			2,560	161	526	124	511	393	4,114	17,190	3,406	1,149	1,378		278	34	2,056	10,478	10,166	10,755	385	385	385	385	385	658	
TOTAL OPERATING COSTS		$1,786,290	$2,291	$8,737	$131,084	$133,699	$133,683	$132,731	$127,268	$119,788	$141,827	$141,289	$113,966	$137,295	$103,064	$117,408	$106,866	$87,331	$9,284	$11,423	$11,004	$11,585	$763	$763	$763	$749	$702	$927	
MILL OPERATING COSTS AFTER PRODUCTION	4,693																												
RECLAMATION COSTS AFTER PRODCTION	33,985																												
OPERATING MARGIN	$000's	$1,745,043	($2,291)	($8,737)	$132,579	$154,023	$134,348	$135,966	$46,034	$57,119	$104,092	$77,244	$65,282	$106,714	$156,169	$176,839	$210,140	$101,807	$5,251	$4,737	$5,253	$4,681	$15,716	$15,716	$15,716	$15,741	$15,829	($927)	
CAPITAL COSTS																													
MINE EQUIPMENT	$000's	134,667		72,166	21,930	4,933		3,249	15,932	2,863		413	2,836	7,482	2,836	27													
PLANT EQUIPMENT & CONSTRUCTION	$000's	361,686	30,779	330,906			0	(0)																					
TSF Fine Grading, Equipment, Piping, Drains	$000's	71,304		5,258		505	247	267	252	192	34,980	23,192		4,940			1,472												
TSF Bulk Earthwork	$000's	88,555		4,193		1,057	496	527	9,485	17,240			24,818		30,127	614													
OTHER/CONTINGENCY/EPCM	$000's	194,774	15,279	140,528	1,942	376	62	270	1,322	194	8,745	3,259	779	2,074	142	4	7,620			133	426	1,260	555					9,804	
SUB-TOTAL	$000's	$850,987	$46,059	$553,052	$23,872	$6,871	$804	$4,312	$26,991	$20,488	$43,725	$26,864	$28,433	$14,496	$33,104	$645	$9,091			$133	$426	$1,260	$555					$9,804	
SALVAGE VALUE	$000's	(70,559)																	(57,372)									(13,187)	
TOTAL CAPITAL	$000's	$780,427	$46,059	$553,052	$23,872	$6,871	$804	$4,312	$26,991	$20,488	$43,725	$26,864	$28,433	$14,496	$33,104	$645	$9,091		($57,372)	$133	$426	$1,260	$555					($3,383)	
CHANGES TO WORKING CAPITAL	$000's	102	2	3,635	(13,164)	2,533	(148)	94	(89)	578	(840)	810	787	(1,685)	122	(1,215)	1,177	291	6,553	585	15	7	(9)	(4)		(0)	(0)	68	
PRE-TAX CASH FLOWS	$000's	$964,514	($48,352)	($565,423)	$121,872	$144,620	$133,692	$131,560	$19,132	$36,053	$61,208	$49,570	$36,062	$93,903	$122,943	$177,410	$199,872	$101,516	$56,071	$4,019	$4,812	$3,413	$15,169	$15,719	$15,716	$15,741	$15,830	$2,388	
CUMM. PRE-TAX CASH FLOWS	$000's	$964,514	($48,352)	($613,775)	($491,903)	($347,283)	($213,591)	($82,031)	($62,899)	($26,846)	$34,362	$83,931	$119,993	$213,896	$336,839	$514,249	$714,121	$815,637	$871,707	$875,726	$880,538	$883,951	$899,120	$914,839	$930,555	$946,296	$962,126	$964,514	
DD&A	$000's	850,987	9,212	117,199	121,974	123,348	123,509	115,159	14,071	13,394	20,765	25,977	30,801	27,362	29,886	21,270	17,715	12,028	9,129	2,535	2,491	1,486	475	475	448	363	111	9,804	
PROFIT BEFORE TAX	$000's	894,056	(11,503)	(126,949)	7,752	25,087	5,720	15,095	26,278	38,370	81,754	62,763	32,524	75,502	123,665	152,898	190,697	88,094	(1,947)	12,680	12,927	13,949	15,626	15,626	15,653	15,763	16,104	(10,073)	
INCOME TAX - Australian & Northern Territories	$000's	379,952								4,163	35,909	21,802	14,417	34,160	54,809	67,695	83,912	38,747			326	958	4,572	4,572	4,580	4,613	4,715		
NET INCOME AFTER TAXES	$000's	$514,105	($11,503)	($126,949)	$7,752	$25,087	$5,720	$15,095	$26,278	$34,207	$45,845	$40,960	$18,107	$41,342	$68,856	$85,202	$106,786	$49,346	($1,947)	$12,680	$12,601	$12,991	$11,054	$11,054	$11,073	$11,150	$11,388	($10,073)	
AFTER-TAX CASH FLOW	$000's	$584,562	($48,352)	($565,423)	$121,872	$144,620	$133,692	$131,560	$19,132	$31,890	$25,299	$27,767	$21,645	$59,743	$68,133	$109,715	$115,960	$62,768	$56,071	$4,019	$4,486	$2,455	$10,597	$11,147	$11,135	$11,128	$11,114	$2,388	
CUMM. AFTER-TAX CASH FLOW	$000's	$584,562	($48,352)	($613,775)	($491,903)	($347,283)	($213,591)	($82,031)	($62,899)	($31,009)	($5,710)	$22,057	$43,702	$103,446	$171,579	$281,293	$397,254	$460,022	$516,093	$520,111	$524,598	$527,053	$537,650	$548,797	$559,932	$571,060	$582,174	$584,562	

FIGURE 1-3: Sensitivity of Pretax Net Present Value to Gold Price @ 5 % Discount Rate (000's)

PRICE ($/oz)	$ 850	$ 900	$ 950	$ 1,000	$ 1,050	$ 1,100	$ 1,150
NPV(5%)	$51,470	$162,759	$274,047	$385,336	$496,625	$607,914	$719,202

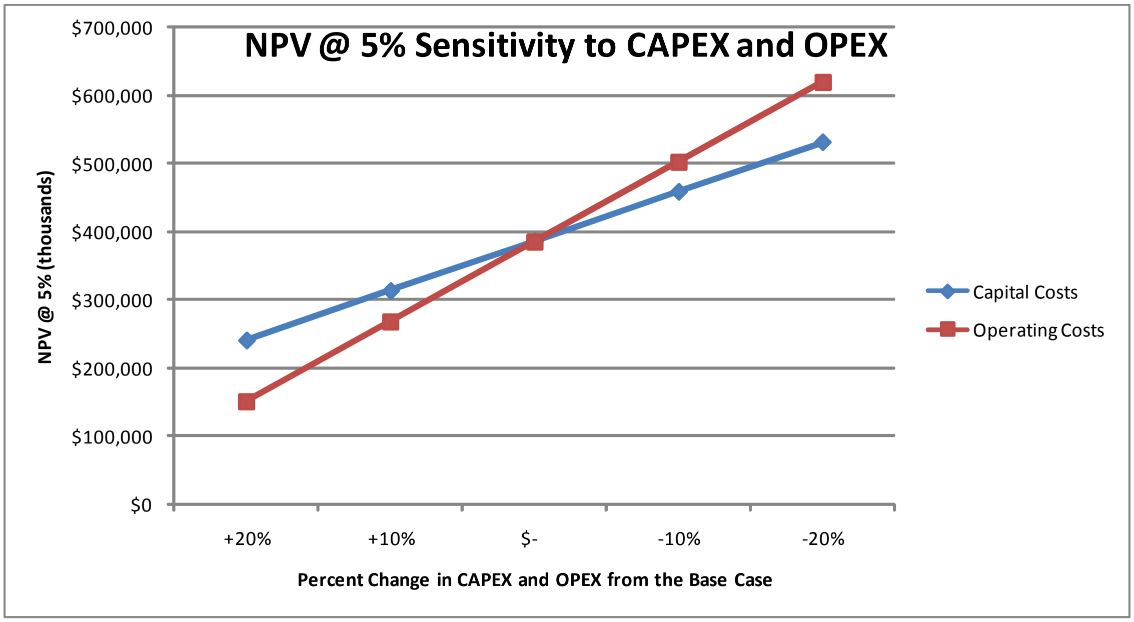


Capital and Operating Cost sensitivity analyses were performed on the Pretax Reserve Case reflecting mutually exclusive increases and decreases of 10 percent and 20 percent for both. A graph showing the results of these sensitivities is shown in FIGURE 1-4.

FIGURE 1-4: Sensitivity of Pretax Net Present Value to CAPEX and OPEX @ 5% Discount Rate (000's)

Sensitivity of Pretax Net Present Value @5% to CAPEX and OPEX Costs

	+20%	+10%	$ -	-10%	-20%
Capital Costs	$240,261	$312,798	$385,336	$457,874	$530,412
Operating Costs	$150,854	$268,095	$385,336	$502,577	$619,819



1.12.7 Sensitivities Deviating from the Reserve Case

Sensitivity analysis performed on the Reserve Case scenario at a Au price of $1,350/toz Au and 1.00 US/AUD exchange rate yielded an after tax NPV of $944.470 million at a five percent discount rate (note that this sensitivity is outside the range of those shown in Figure 1-3).

A second sensitivity considered a Au price of $950/toz Au and 0.85 US/AUD exchange rate. The analysis resulted in an after tax NPV of $274.047 million at a five percent discount rate.

1.13 Conclusions

Vista's exploration and development work on the Mt. Todd Gold Project, and specifically the Batman and Quigleys deposits, continues to provide strong justification for additional expenditures and efforts to develop a new mine at this site and progress the project through full feasibility. In addition to the Batman and Quigleys deposits, other known deposits/areas that warrant addition exploration include the following.

Golf and Tollis Deposits

While the Quigleys and the Golf Tollis deposits have had limited drilling and some surface production, they have not been explored using the lessons learned at Batman. The exploration to date has concentrated on near-surface oxide gold mineralization with few, if any, deep drill holes existing. In addition, the Batman structural interpretation has not been applied to these deposits either. Since these deposits are known to contain gold mineralization, a more systematic exploration program is warranted.

Exploration Leases

A significant portion of the exploration leases is yet to be systematically explored and evaluated. The broad structural and geologic trends that host the Batman, Quigleys, and Golf Tollis deposits may well host other deposits. Much of what Vista has learned from more detailed exploration of the Batman deposit has yet to be applied to these other areas; therefore, these areas remain highly prospective.

1.14 Recommendations

Based on Tetra Tech's review of the database, previous studies and work products, and as an outgrowth of the recent mineral resource modeling and PFS update, Tetra Tech provides the following list of recommendations for Vista's consideration.

1.14.1 Geology and Exploration

Batman Deposit

While not yet totally defined by drilling, the Batman deposit continues to warrant both exploration and development work. Additional exploration work is justified in locating the extents of the deposit as it is currently open on all sides. In addition, more development drilling is warranted to increase the measured and indicated resource base as the project moves toward a feasibility study. In support of the feasibility study, the following work items form the next logical progression in the development scenario:

- Additional exploration drilling, as the deposit is still open to the north, south, and at depth.
- The 2007 and 2008 exploration drill hole programs have identified what appear to be parallel and/or sub-parallel structures to the east of the main core complex. Additional exploration and definition of these structures is warranted.
- Completion of additional geologic and geotechnical mapping to increase the understanding of the larger system.
- Additional metallurgical sampling and testing. Additional metallurgical samples are needed to ensure that all of the potential deposit variability is accounted for and considered in the process design phase.
- Additional testwork on the HPGR component of the process design is needed to reach feasibility-level results. Initial testwork has proven that this is highly likely to work at Mt. Todd and results in significant energy and capital savings.
- Additional geotechnical logging and drilling to confirm the pit slope recommendations of this report.
- Additional geochemical characterization analyses. Historic and recent geochemical characterization of waste rock and tailings has provided a basis for the environmental considerations of this report; however, additional testing will be required for the feasibility study.

Quigleys and Golf-Tollis Deposits

The Quigleys and Golf-Tollis Deposits appear to be more structurally controlled than Batman with the mineralization occurring in narrower bands. Tetra Tech proposes that the following items be considered when preparing the ongoing work plan:

- Surface mapping and subsequent re-interpretation of the footwall contact to the shear zone mineralization are recommended. Any additional structural complexity that results should, where appropriate, be used to refine the mineralized envelope upon which modeling updates are based.
- Optimization of the resource provides a focus to define areas requiring further investigation or infill drilling. Due to the high degree of variability in the deposit, infill drilling is best targeted at key areas of geological complexity.
- A model should be developed for the area outside the shear zone. This will require separation of areas of mineralization from unmineralized areas using suitable envelope constraints.
- The cause of the apparent bias between some of the old and new RC drilling should be confirmed to validate the inclusion of all samples in resource calculations.

Other Mineralized Occurrences

Several other known mineral occurrences are found on the concession; these are Golf, Tollis, and Horseshoe deposits. There are some indications of prior exploration work, based on maps and minor references that have involved geologic, geochemical, geophysical, and drilling. While a lower priority than Batman and Quigleys, efforts should be undertaken that:

- Locate all available data and confirm, if possible, the validity;
- Re-assess the data to determine if additional exploration work is warranted; and
- Develop appropriate programs that systematically attempt to define the size and tenor of the mineralization present.

1.14.2 Metallurgy/Process Engineering

Tetra Tech, RDi, and Ausenco recommend additional metallurgical testwork and process studies in working toward the feasibility stage of development to validate key metallurgical information, explore possible process improvements, and to reduce process risk.

- Process testwork is proposed on samples representing different rock/ore types within the resource to include extremes in grade, hardness, and associative mineralogy. Such work should be performed for all deposit areas that may ultimately become minable reserves.
- Ore variability testing for the whole ore flowsheet (i.e., transition ore, oxide zone), including ore grade variation and blending should be conducted. Of specific interest in addition to gold leaching and recovery is the copper constituent and potential for deleterious copper loading on the activated carbon, potentially beyond current circuit design capacity.
- Several commercial scale HPGR applications have begun operation in the past 18 months. Undoubtedly, manufacturers and the mining industry have learned from these efforts. A study to benchmark the commercial operations against the envisioned

application at Mt. Todd including specific energy requirements, circuit design, and wear/maintenance issues is recommended.

- Efforts to optimize the crushing and grinding circuit in general should be continued considering that comminution in total defines a major proportion of both the project capital and operating costs.
- Development of improved blasting techniques to safely produce the finest feed for the crushing circuit has the potential to reduce comminution costs. With regard to comminution, as crushing is more efficient than grinding, so is blasting more efficient than crushing.
- Use of the grind thickener as a precursor to the pre-aeration unit operation should be optimized. Often the residence time inherent with a grind thickener allows an opportunity for significant geochemical precursor reactions to occur or be in place before the actual pre-aeration step. This is a logical step in addition to optimization of the entire pre-aeration process so as to minimize the consumption of lime.
- Additional metallurgical testwork should include optimization of oxygen and cyanide concentration in the CIL circuit. Such tests would consider leaching under conditions of decay versus maintenance of NaCN concentration and oxygen content. Further, whole ore leach (WOL) tests should be performed using material crushed by HPGR to investigate if there is any potential improvement.
- Confirmation CIL extraction testwork using site water and cyanide destructed tailing water as process leach water should be conducted as a continuation of the metallurgical testwork.
- Carbon loading and stripping tests should be performed.
- Detoxification process studies on CIL tailings should be performed to investigate different commercial approaches, reagent consumption, and overall effectiveness of such processes on the different ore types that might be encountered.
- Slurry rheology tests should be conducted as a component of the metallurgical testwork program as the project moves into the feasibility phase of development. This should include testwork on the thickening of ground material before pre-aeration. Such tests will also give information pertinent to slurry pumping, pipelines, and the selection and design / layout thereof.
- The ore(s) should be tested for mercury and, if found in significant quantities, provisions should be made in the process flowsheet for mercury capture and condensation.
- The size of the coarse ore storage facility should be studied to determine optimum capacity. Appropriately sized storage will assist in preventing mine delay when the crusher is down and, conversely, crusher delay when the mine is down. Coarse ore storage capacity is tied directly to the mobility of equipment in the pit and the flexibility of the mine plan to switch production from ore to waste.
- Elemental tests of the fuels to be used in the kiln should be performed so as to ensure the selection of the best material for the kiln shell. Some fuels are higher in specific elemental constituents detrimental to specific metals and alloys.
- Historical review of metallurgical related process development and operational issues shows that when ore containing significant amounts of copper are processed there can be cyanide solubility related impacts to performance. Copper dissolution is generally undesirable during leaching because it can consume cyanide and dissolved oxygen,

retard gold dissolution rates, interfere with subsequent recovery processes, and contaminate the final product. Care should be taken in mine operations to avoid ore containing copper to ensure that carbon loading and stripping, in particular, are not problematic. If future exploration, resource development, and mine plans encounter elevated copper ores, then additional metallurgical testwork will be warranted to develop alternative process techniques capable of managing high copper containing ores.

1.14.3 Water Treatment

The following water treatment studies are recommended:

- Obtain NT Government approval to permit effluent releases from the existing WTP to Batman Creek.
- Initiate dialog with the NT Government to determine if additional numeric standards will apply to WDL or water quality-based effluent standards for the Edith River.
- Construct run-on diversions up gradient of the TSF and WRD Pond that satisfy specific temporal and performance criteria.

Address all water treatment information and data gaps identified in the PFS and summarize as follows:

- Complete a water treatment and sludge management study (including bench-scale testing and regulatory review) to define, optimize, and cost the following:
  - Water treatment plant capacity, design, and location;
  - Impacted water collection, conveyance, and storage;
  - Site-wide pipeline and pumping system requirements; and
  - Sludge conveyance, storage, and containment.
- Inventory all existing water management facilities to determine overall system arrangement, facility capacity, operation and maintenance status, and remaining functional life.

1.14.4 Closure

The following closure studies are recommended:

- Complete a waste and cover material hydraulic properties analysis;
- Complete a tailings trafficability study;
- Improve precipitation-watershed yield characterization of the site;
- Complete a precipitation-watershed yield study;
- Complete a waste rock management plan;
- Complete a tailings management plan;
- Complete a site-wide soils, closure cover, and reclamation materials inventory and characterization study; and
- Complete a waste and closure cover erosion and sediment control study.

1.14.5 Tailings Storage Recommendations

The following studies and investigations are recommended for further design reports for both proposed facilities:

- The material properties used in the geotechnical modeling must be confirmed through additional drilling, laboratory testing, and tailings testing;
- The geotechnical modeling must be updated and expanded to include consolidation analysis, liquefaction analysis, and deposition modeling;
- A seismic hazard analysis should be updated even though the site is in a relatively inactive seismic area;
- The TSF water balance must be updated to optimize the size of the water pool to accommodate the water requirements of the processing facilities; and
- Engineering analyses should be performed to verify the liner and drain systems can support loads from the tailings and embankment.

For TSF1, the following site-specific studies and investigations should be performed for further design reports:

- The spillway design must be revised to account for any required change in cross-section at different stages of the impoundment; and
- The condition of the existing toe drains, under-drains, and decant towers must be investigated to confirm their efficacy when tailings deposition resumes.

Further design reports for TSF2 should include the studies listed above, as well as a tradeoff study to determine the feasibility of using a side-hill decant structure instead of a floating barge pump for water reclaim.

1.15 Limitations

Tetra Tech is not aware of any potential limitations to the project that would materially change any of the data, resource estimates, environmental considerations, socio-economic factors, or conclusions presented within this report that are outside of the normal factors that may impact mining projects, such as, price variability, exchange rates, permitting time, etc. With respect to the Mt. Todd Gold Project, the land tenure is secured by agreement with all of the potentially affected parties, the existing environmental liabilities are well documented and have been adequately addressed, potential new environmental issues are part of this and future studies and are not anticipated to materially impact the path forward, the site has good existing infrastructure, power and water, exploration and development drilling will continue, and metallurgical testing and analyses continues to occur.

2.0 INTRODUCTION

Vista Gold Corp ("Vista") purchased the Mt. Todd property on March 1, 2006, and the acquisition was completed on June 16, 2006 when the mineral leases transferred to Vista and funds were released from escrow. Tetra Tech, Inc. ("Tetra Tech") was commissioned by Vista in September 2009 to prepare a NI 43-101 compliant Preliminary Feasibility Study (PFS) at an ore processing rate of 6.77 million tonnes per year (Mtpy) for the Mt. Todd Gold Project (the "Project") located in Northern Territory ("NT"), Australia. 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 is the subject of the study presented herein and issued January 28, 2011.

Prior to these two PFS studies an initial NI43-101 Technical Report was completed on June 26, 2006. A Preliminary Economic Assessment report was completed on December 29, 2006, 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 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 these have had historic mining, with Batman having the most production and exploration completed. Currently, only the Batman and Quigleys deposits have CIM compliant reported resources and only the Batman deposit has CIM compliant reportable mineral reserves.

2.1 Terms of Reference

This report has been prepared in accordance with the guidelines provided in National Instrument 43-101, Standards of Disclosure for Mineral Projects. The Qualified Person responsible for this report is Mr. John W. Rozelle, P.G., Principal Geologist at Tetra Tech. Other Qualified Persons who had significant input to this report are presented in TABLE 2-1.

TABLE 2-1: LISTING OF QUALIFIED PERSONS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Qualified Person	Firm	Report Section
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Dr. Steve Krajewski, P.G. Mr. John W. Rozelle, P.G.	Tetra Tech, Inc,	Section 17: Mineral Resources
Mr. Ed Lips, P.E. Mr. Thomas L. Dyer, P.E.	Tetra Tech, Inc. Mine Development Associates	Section 18: Mine Engineering Portions of Section 19
Mr. Erik Spiller Dr. Deepak Malhotra	Tetra Tech, Inc. Resource Development Inc.	Section 16: Mineral Processing

Neither Tetra Tech nor any of its employees and associates employed in the preparation of this report has any beneficial interest in Vista or in the assets of Vista. Tetra Tech will be paid a fee for this work in accordance with normal professional consulting practice.

2.2 Scope of Work

The Mt. Todd Mine property is made up of several gold deposits occurring in an area of 5,365 hectares in the NT of Australia. The most prominent of these deposits are the Batman and Quigleys deposits. The other mineral occurrences do not have sufficient data available at this time to develop classified mineral resource estimates.

The scope of work undertaken by Tetra Tech involved an update of the PEA from December 29, 2006, based on the recent update of the gold resource model completed in February 2009, which included exploration, geology, and assay work completed by Vista as part of their 2007 and 2008 exploration program. Based on these additional data, Tetra Tech re-estimated the capital and operating costs and re-developed pit designs and production schedules for the Batman deposit.

2.3 Effective Date

The effective date of the mineral resource and mineral reserve statements in this report is January 28, 2011.

2.4 Units

All dollars are presented in US dollars unless otherwise noted. For the purpose of this report the exchange rates are \$0.85 = AUD\$1.00 except as needed for the sensitivity analysis. Common units of measure and conversion factors used in this report include:

Linear Measure

1 inch = 2.54 centimeters
1 foot = 0.3048 meter
1 yard = 0.9144 meter
1 mile = 1.6 kilometers

Area Measure

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

Capacity Measure (liquid)

1 US gallon = 4 quarts = 3.785 liter
1 cubic meter per hour = 4.403 US gpm

Weight

1 short ton	= 2000 pounds	= 0.907 tonne
1 pound	= 16 oz	= 0.454 kg
1 oz (troy)	= 31.103486 g	

Analytical Values

	percent	**grams per metric tonne**	**troy ounces per short ton**
1%	1%	10,000	291.667
1 g/tonne	0.0001%	1.0	0.0291667
1 oz troy/short ton	0.003429%	34.2857	1
10 ppb			0.00029
100 ppm			2.917

Frequently used acronyms and abbreviations

AA	=	atomic absorption spectrometry
Ag	=	silver
Au	=	gold
°C	=	degrees Centigrade
CIC	=	Carbon-in-column
CIM	=	Canadian Institute of Mining, Metallurgical, and Petroleum
CIP	=	Carbon-in-pulp
°F	=	degrees Fahrenheit
FA	=	Fire Assay
ft	=	foot or feet
g	=	gram(s)
g/kWh	=	grams per kilowatt hour
g/t	=	grams per tonne
h	=	hour
ICP	=	Inductively Coupled Plasma Atomic Emission Spectroscopy
km	=	kilometer
kV	=	kilovolts
kWh	=	Kilowatt hour
kWh/t	=	Kilowatt hours per tonne
L	=	liter
m	=	meter(s)
m^2	=	square meter(s)
m^2/t/d	=	square meters per tonne per day
m^3	=	cubic meter(s)
m^3/h	=	cubic meter(s) per hour
mm	=	millimeter
Mtpy	=	million tons or tonnes per year
MW	=	megawatts
NSR	=	net smelter return
toz Ag/t	=	troy ounces silver per short ton (oz/ton)
toz Au/t	=	troy ounces gold per short ton (oz/ton)
ppm	=	parts per million
ppb	=	parts per billion
RC	=	reverse circulation drilling method
SAG	=	semi-autogenous grinding
ton	=	short ton(s)
tonne	=	metric tonne
t/m^3	=	tonne per cubic meter
tpd	=	metric tonnes per day
tph	=	tonnes per hour
µm	=	micron(s)
%	=	percent
tpy	=	tons (or tonnes) per year
tpm	=	tons (or tonnes) per month

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	

2.5 Basis of Report

Tetra Tech has prepared this report exclusively for Vista. The information presented, opinions and conclusions stated, and estimates made are based on the following information:

- Information available at the time of the preparation of the report as provided by Vista;
- Assumptions, conditions, and qualifications as set forth in the report;
- Data, reports, and opinions from prior owners and third-party entities; and
- Data, reports, and opinions from Vista exploration work and consultants.

Effective March 1, 2006, Vista and their subsidiary, Vista Gold Australia Pty Ltd (Vista Australia) entered into agreements with Ferrier Hodgson, the Deed Administrators for Pegasus Gold Australia Pty Ltd, the NT Government, and the Jawoyn Association Aboriginal Corporation (JAAC) to purchase a 100 percent interest in the Mt. Todd Gold Mine (i.e., Mining Licenses) and acquire the rights to the surface in the area of the mining licenses and exploration licenses. Tetra Tech has reviewed this information and information that shows that Vista is current with all obligations that are part of these agreements and is satisfied that they have all the necessary legal and financial rights to explore and develop the Mt. Todd Gold Project.

3.0 RELIANCE ON OTHER EXPERTS

The Mt. Todd mining property, having been an operating mine for several years, has been the subject of numerous written reports. The Trustee for the NT has provided Vista with an inventory of the available documentation for the property. Many of these reports and other documents were prepared by mining consulting firms on behalf of the operators of the mine/property at the time. Tetra Tech has used a number of the references in the preparation of the mineral resource estimate detailed herein. The reports referenced have each been reviewed for materiality and accuracy, as they pertain to Vista's plans for property development. Specific experts that had an important role in the preparation of this report include:

Dr. Rex C. Bryan

Graduated with a Mineral Economics Ph.D. from the Colorado School of Mines, Golden, Colorado, in 1980. Graduated in 1976 from Brown University, in Providence, Rhode Island, with M.Sc. Geology. Graduated from Michigan State University with a MBA (1973) and a BS in Engineering (1971).

Is a member of the Society for Mining, Metallurgy, and Exploration, Inc. (SME).

He has worked as a geostatistical reserve analyst and mineral industry consultant for a total of 26 years since graduating from Colorado School of Mines. He is an expert witness to industry and for the U.S. Department of Justice on ore-grade control, reserves, and mine contamination issues. He is currently a consultant to the industry.

Ken Rippere

Graduated with a BS degree in Geological Engineering from the Colorado School of Mines in 1966; is a member of the American Institute of Professional Geologists (CPG No. 6023), The Society of Mining, Metallurgy, and Exploration (SME), and is registered to practice geology in Arizona and Georgia; has worked on the geotechnical aspects of rock slopes, including both design and failure management, particularly for open pit mines, for 41 years, nearly equally divided between consulting and mine operations at properties around the world.

Dr. Richard W. Jolk, P.E.

Graduated with a PhD in Mining Engineering 2007, an MS in Environmental Engineering 1993, an MS in Mining Engineering 1986, and a BS in Metallurgical Engineering 1978, all from the Colorado School of Mines, Golden, Colorado.

He is a Registered Professional Minerals Engineer (PE Colorado 24448), a Certified Minerals Appraiser (CMA 2010-1), and a member of the Society for Mining, Metallurgy, and Exploration, Inc. (SME).

He has been professionally involved in the minerals extraction and beneficiation industries internationally for over 32 years including work in mineral project valuation, feasibility, development, operations and closure. Experience has included working for both mine operating companies and engineering firms.

Mr. John W. Rozelle, P.G., has personally reviewed the available reports and the extracted data in order to ensure that these items meet all of the necessary reporting criteria as set out in the NI43-101 guidelines.

4.0 LOCATION AND PROPERTY DESCRIPTION

4.1 Location

The Mt. Todd Project is located 56 km by road northwest of Katherine, and approximately 250 km southeast of Darwin in the NT of Australia. Access to the property is via high quality, two-lane paved roads from the Stuart Highway, the main arterial within the territory (FIGURE 4-1).

4.1.1 Tenements

The concession consists of three individual mineral leases, MLN1070, MLN1071, and MLN1127 comprising some 5,365.27 hectares. In addition, Vista controls exploration leases, EL25668, EL25669, EL25576, and EL25670 comprising approximately 117,632 hectares. FIGURE 4-2 illustrates the general location of the tenements and the relative position of the two primary mineral deposits: Batman and Quigleys.

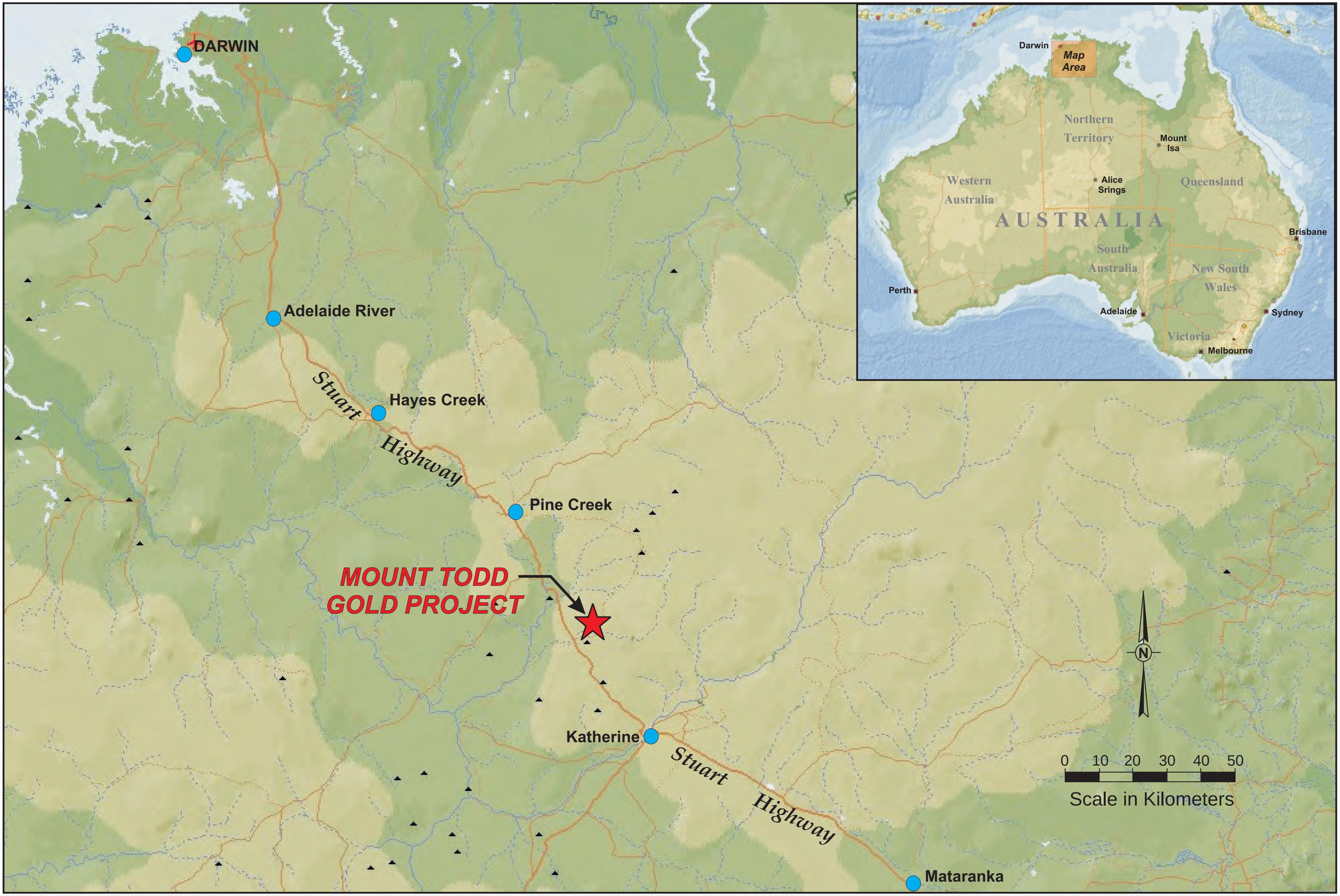
4.1.2 Lease and Royalty Structure

The agreement with the NT is for an initial term of five years commencing January 1, 2006, with an extension of five years at Vista's option and three additional years possible at option of the NT. During the first five-year term in accordance with the conditions of the agreement, Vista 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 implemented recommendations. Vista has developed a technical and economic report for the re-starting of operations.

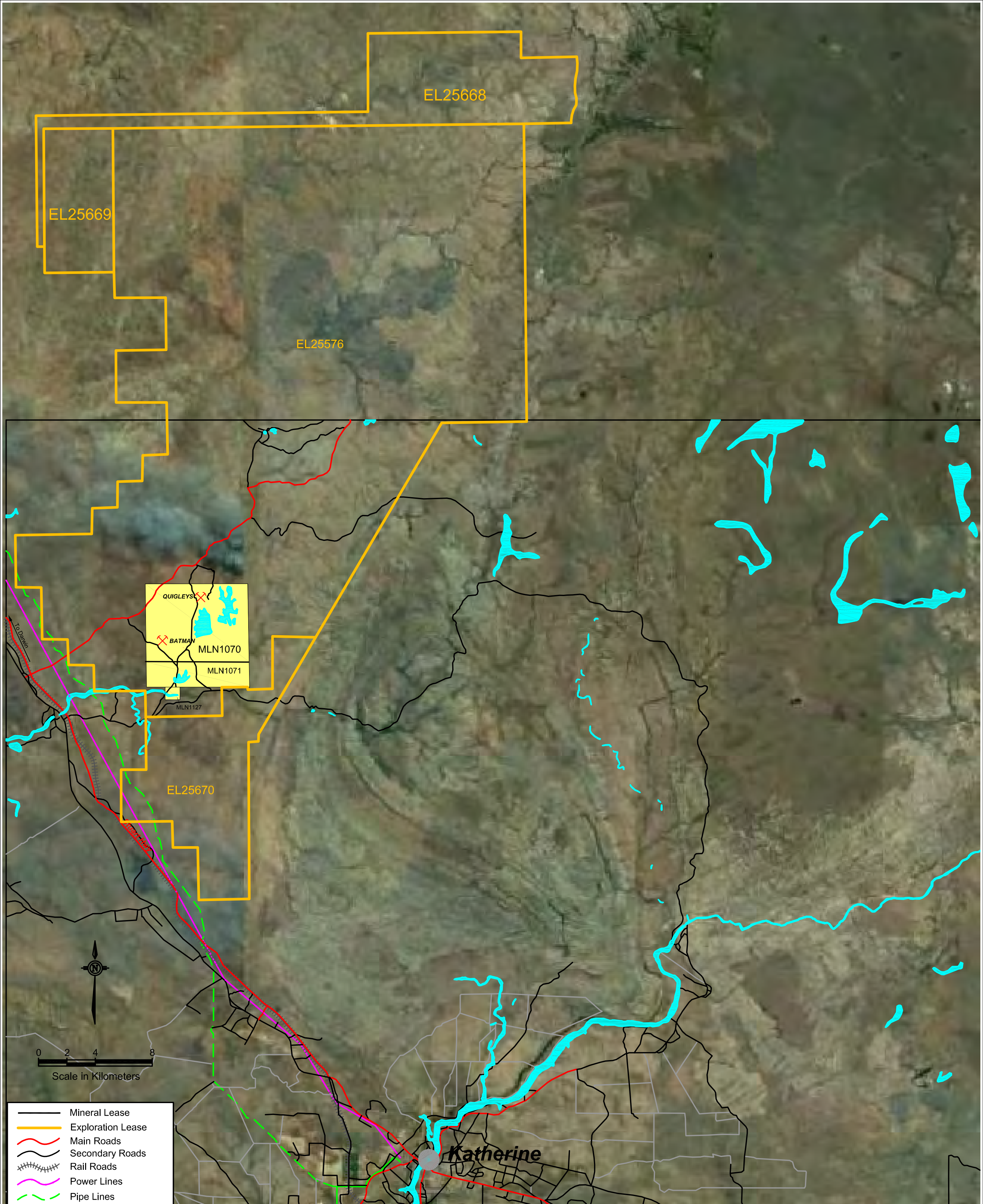
Vista provided notice to the NT government in June 2010 that it wished to extend the agreement. In November, the NT government acknowledged that Vista had fulfilled its obligations for the intial term, and the agreement has been extended for five years until December 31, 2015.

Vista 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 that Vista has no rehabilitation obligations for pre-existing 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 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, that when a production decision is reached, to prepare and execute a local Industry Participation Plan.

The agreement with the JAAC called for Vista to issue common shares of Vista with a value of CAD$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 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.



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.*	Fig4-1.dwg	**Figure 4-1** **General Location Map** **Mt. Todd Gold Project**
	Project: Mt. Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	



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Prepared for: Vista Gold Corp.	File Name: Fig1-2_4-2.dwg
Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a
Project Location: Northern Territory, Australia	Date of Issue: Sept/2010

Figure 4-2
Concessions and Infrastructure Map

If the Mt. Todd Project proves feasible for economic development of the mineral leases including a fully funded site reclamation bond, Vista will establish a technical oversight committee with representatives of the NT and the JAAC. Additionally, Vista will offer the JAAC the opportunity for joint venture participation in the operation on a 90 percent Vista/10 percent JAAC basis. For rent of the surface during production, Vista (or the Joint Venture if formed) will pay the JAAC an annual amount equal to one percent of the annual value of production with an annual minimum of AUD$50,000. As part of the agreement, Vista will endeavor to use services and labor provided by the JAAC when feasible. Vista and the JAAC may form a 50/50 exploration joint venture to explore JAAC lands outside the mineral leases.

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 250 km southeast of Darwin in the NT of Australia (see FIGURE 4-1). 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

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 (77° to 95° F). Between November and December, temperatures can reach 40° C (104° F). Winter temperatures in the dry season are warm in the daytime, but can drop to 10° C (50° F) 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. Because the area has both historic and current mining activity, the area contains a skilled mining workforce. In addition, Katherine offers all of the necessary support functions that are found in a medium sized city with regard to supplies, hotels, communications, etc.

The property has an existing high-pressure gas line and an electric line that was used by previous operators. In addition, both 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 also 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.

5.4 Environmental Conditions

5.4.1 Permitting

In 2006, Vista, including its wholly-owned subsidiary Vista Gold Australia Pty Ltd., acquired the Mt. Todd Project through various contracts executed with the NT Government, Ferrier Hodgson as the deed administrator for Pegasus Gold Australia Pty Ltd., and the Jawoyn Association Aboriginal Corporation (JAAC). These contracts gave Vista the right to explore and develop the mineral resources of the associated Mining Licenses.

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 identifies the environmental risks found at the Mt. Todd site in its present state of operations and defines the actions that Vista is taking 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 Notice of Intent (NOI) for resumption of mining operations.

The first step in formal mine permitting will be submission of a NOI to the NT Government. This document are intended to cover all the major issues relating to the mine development and provide sufficient information (background and technical) to allow a preliminary assessment by the Department of Resources (DoR), formerly Department of Regional Development, Primary Industry, Fisheries and Resources (DRDPIFR). Ultimately, the adequacy of the Mt. Todd Project NOI will be assessed against the following requirements:

- Description of mining activities;
- Description of the existing environment;
- Safety, health and environmental issues relevant to the mining activities and the management system to be implemented;
- Description of current and proposed mine workings and infrastructure; and
- A plan and costing of closure activities.

Simultaneously, an Environment Protection and Biodiversity Conservation Referral (EPBC Referral) will be submitted to the Commonwealth's Department of Sustainability, Environment, Water, Population and Communities (SEWPaC). SEWPaC will assess the EPBC Referral and make recommendations about whether or not the project should be approved to proceed.

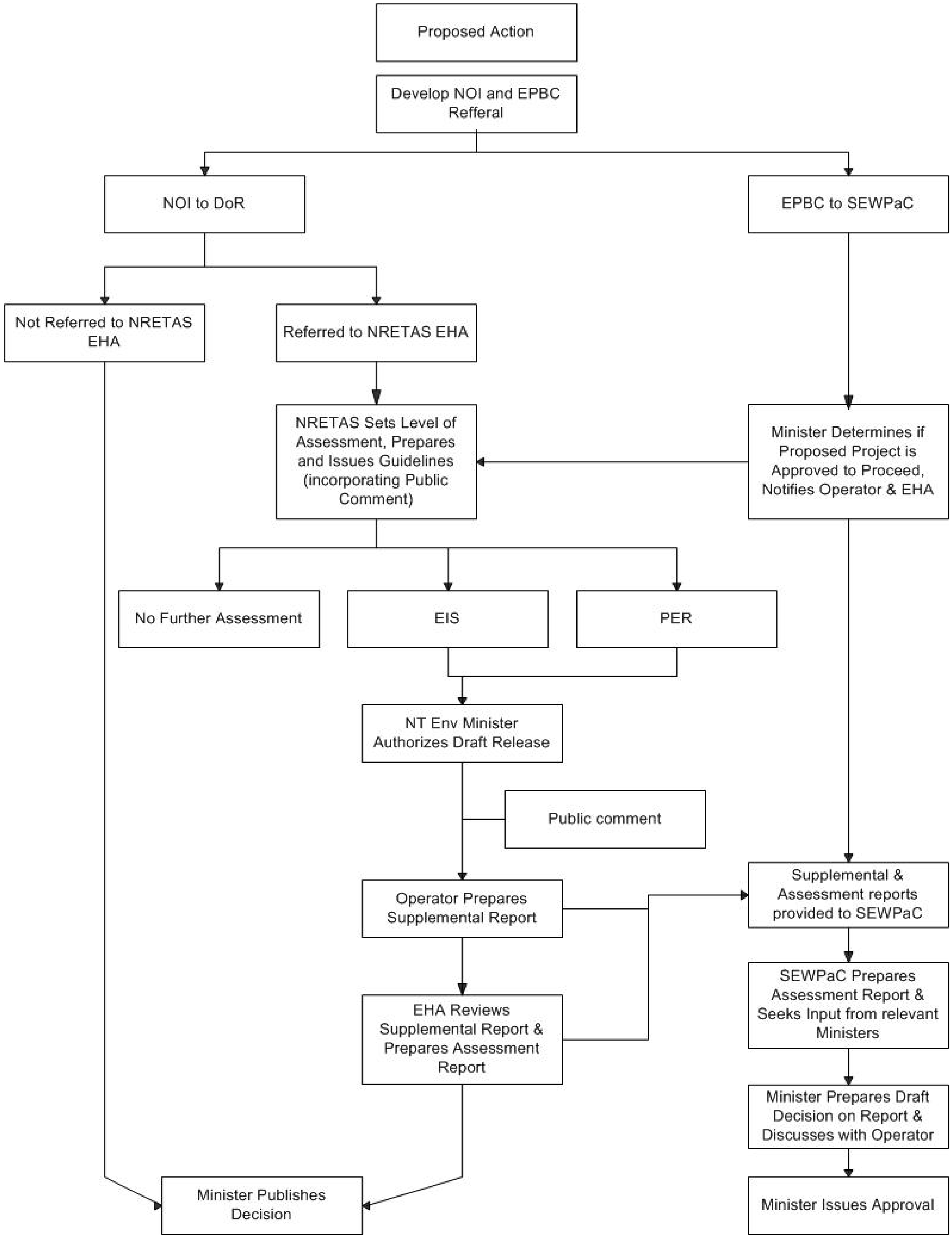
DoR will determine if the proposed project should be referred to the Environment, Heritage and the Arts Division (EHA) of the Department of Natural Resources, Environment, the Arts and Sport (NRETAS) for assessment under the NT Environmental Assessment Act as detailed in FIGURE 5-1 (DRDPIFR, 2008a). If the DoR recommends referral, NRETAS, with input from the SEWPaC regarding the EPBC Referral, will advise on the requirement for either a Public Environmental Report (PER) or an Environmental Impact Statement (EIS).

The guidelines provided by NRETAS indicate that:

- A PER is required to assist in assessing environmental impacts that are considered significant but limited in extent; while
- An EIS is required 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.

An NT Environmental Minister will review the PER or EIS and authorize a draft release with a public comment period. A Supplemental Report will be prepared for review by EHA which addresses concerns from the public. Both the Supplement Report and an Assessment Report to be prepared by EHA based on the Supplement Report will be issued to SEWPaC which will prepare a draft decision for issuance and approval prior to publication of the final decision.

The estimated costs and timing of the possible paths associated with the environmental assessment process are provided in TABLE 5-1. These costs are based on estimates provided by Gustavson (2006) updated assuming an 18 percent increase in costs since 2006 (Engineering News Record, 2006) and guidance from GHD (GHD, 2010a). An allocation of $650,000 for permitting and $1,850,000 for baseline studies for the Mt. Todd Project has been included in the project budges for the next stage. This estimate assumes the permitting process will include an EIS; however, it is unclear at this time whether DoR will refer the project.



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TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***Vista Gold Corp.***	Fig5-1.jpeg	**Figure 5-1** **MT. TODD ENVIRONMENTAL ASSESSMENT PROCESS**
	Project: Mt. Todd Gold Project Pre-Feasibility Study	Project Number: 310912A	
	Project Location: Northern Territory, Australia	Date of Issue: 01/26/2011	

TABLE 5-1: ESTIMATED MINE DEVELOPMENT PERMITTING COSTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Task	Estimated Time[1]	Cost ($)[2]
Case 1: Assessment under the Mining Management Act (not referred to NRETAS)		
Mining Management Plan or Notice of Intent	1 month	$50,000
Total	1 month	$50,000
Case 2: Referred to NRETAS, Public Environmental Review Required		
Mining Management Plan or Notice of Intent	1 month	$50,000
Public Environmental Report	3 months	$300,000
Total	4 months	$350,000
Case 3: Referred to NRETAS, Environmental Impact Statement Required		
Mining Management Plan or Notice of Intent	1 month	$50,000
Environmental Impact Statement	9 months	$600,000
Total	10 months	$650,000

Note: [1]Preparation time only, does not include time for government approval process
[2]if preparation is outsourced

5.4.2 Existing Environmental Conditions

The following description of the existing environmental conditions at the Mt. Todd site is taken from Chadwick T&T Pty LTD (2009):

- Waste Discharge License 135 (EPA NT, 2005);
- Draft Waste Discharge License 178 (NT Government, 2010);
- 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 (Vista, 2007a);
- Mt. Todd Waste Discharge License Report, 2006 – 2007 (Vista, 2007b);
- Mt. Todd Blueprint Rehabilitation Strategy (BRS) Report (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);
- Mt. Todd Water Treatment Plant Commissioning Report (Vista, 2009);
- Mt. Todd Water Management Plan, 2010 – 2011 (Vista, 2010); and
- Mt. Todd Water Balance - Care and Maintenance Model Calibration and Forward Modeling Predictions (HydroGeoLogica, Inc. and Tetra Tech, 2010).

5.4.3 Surface Water Hydrology

The Mt. Todd site is drained by the perennial Edith River, located approximately 1 km south of RP1, and also drained by several ephemeral streams. Batman Creek runs through the center of the site and Horseshoe Creek is located on the eastern side of the site. Both Batman and Horseshoe feed Stow Creek, which enters the Edith River at a point south of the discharge point from RP1. These hydrologic features are shown on FIGURE 5-2.

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. Flows from the mine have exceeded the capacity of the water management system, allowing the release of contaminated water 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 flow meters on the siphon and pumping outlets from RP1 and Existing WTP inlet pipe. A map illustrating the general locations of the surface water monitoring locations is provided in FIGURE 5-2.

Horseshoe Creek and Batman Creek catchments are approximately 45 and 11 km^2, respectively. The Raw Water Dam was built across Horseshoe Creek immediately above the mine, forming a sub-catchment covering about 55 percent 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.

Controlled mine water drainage enters the Edith River from the discharge point for RP1 and a minor creek known as West Creek. The RP1 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 waste rock dump via the Western Diversion Drain, and overflow from the RP1 spillway. The West Creek catchment is small and it is reported that the creek only delivers mine water to the Edith River after substantial rainfall events cause RP1 to overflow. During periods of wet season base flow (approximately January to May) uncontrolled flow to the Edith River occurs. Mine water contributing this flow originates from a number of possible sources on site including uncontrolled overflow from RP1, RP2, and RP5 during high-rainfall events and potentially from various surface seeps in much smaller quantities. However, for a large part of the year (roughly May to December), no mine water enters the Edith River through surface flow.

5.4.4 Updated Water Balance

A site-wide water balance was developed by the NT Government Department of Mines and Energy (NTG DME) in 2001 based on earlier models developed by Bateman Kinhill Kilborn (BKK, 1996) and General Gold Operations (General Gold Resources N.L., 1999). MWH (2006a) used the Goldsim platform (www.goldsim.com) to update the model to assist Vista with decision-making regarding water management options and to provide a starting point for future developments.

The Goldsim water balance model was updated and calibrated with site measurements collected over the last two years. A wide range of precipitation scenarios were developed, to evaluate the site water management protocols and the potential for Batman Pit and pond overflows assuming an extended care and maintenance period. The updated and calibrated model was used to size a new water treatment plant with sufficient capacity to partially dewater Batman Pit to permit in-pit preparation activities (lay backs) prior to the initiation of mining. The model was also used to estimate water treatment requirements during all phases of the project including current, pre-production, production, closure, and post-closure.

5.4.5 Water Quality

The Mt. Todd Project Report 1: Environmental Assessment provides a summary of the hydrochemistry of site waters sampled from ponds, pits, dams, streams, and groundwater for 12 to 24 months up to and including June 2006 with a key objective of providing baseline water quality data to Vista prior to assuming responsibility for the site (MWH, 2006a). Water management on site has changed since mid-2006 including most significantly the transition from RP7 as a repository for impacted water from other facilities to RP3 as the primary repository. A comparison of average water quality from the historic data and samples collected during the 2008/2009 wet and dry season from RP1 and RP3 shows that all constituent concentrations that were measured have decreased with the exception of arsenic (Appendix I). Although water quality has improved, ARD and constituent concentrations above applicable guideline values remains a primary water management issue at the Mt. Todd site.

The Edith River and tributaries are protected beneficial use under the Water Act 2000 for aquatic ecosystem protection. The Mt. Todd Project WDL 135, issued on December 21, 2005, for the next two wet seasons and transferred to Vista on January 1, 2007, states that pollutant discharges from point sources is governed by the following principles (EPA NT, 2005):

- Must not prejudice water quality objectives outside of any agreed mixing zone when defined for the receiving waters; and
- Pollutant discharges must be reduced to the maximum extent by Best Practice Environmental Management (BPEM) in accordance with the hierarchy of waste management (i.e., reduce, reuse, recycle).

The performance of the water management system is assessed against the WDL criteria through the monitoring and evaluation of on site, downstream and upstream water quality, sediment geochemistry, and macro-invertebrate sampling.

To meet these objectives, WDL 135 allows controlled discharges from the RP1 siphon that depend on minimum flows in the Edith River; specifically, water can be released from the RP1 discharge point when the Edith River at SW4 is flowing at 12 m^3/s and the water level is above 0.81 meters (m). This flow is considered sufficient to ensure downstream compliance with established copper criteria which in turn dilutes other regulated constituents to acceptable levels. TABLE 5-2 contains the ANZECC and ANZMARC (2000) guidelines for aquatic ecosystem protection.

A new waste discharge license is being prepared by NRETAS to replace the expired WDL 135. The new WDL is expected to contain similar terms regulating discharges from the site. Vista intends to revise the monitoring program to meet any requirements of the new license.

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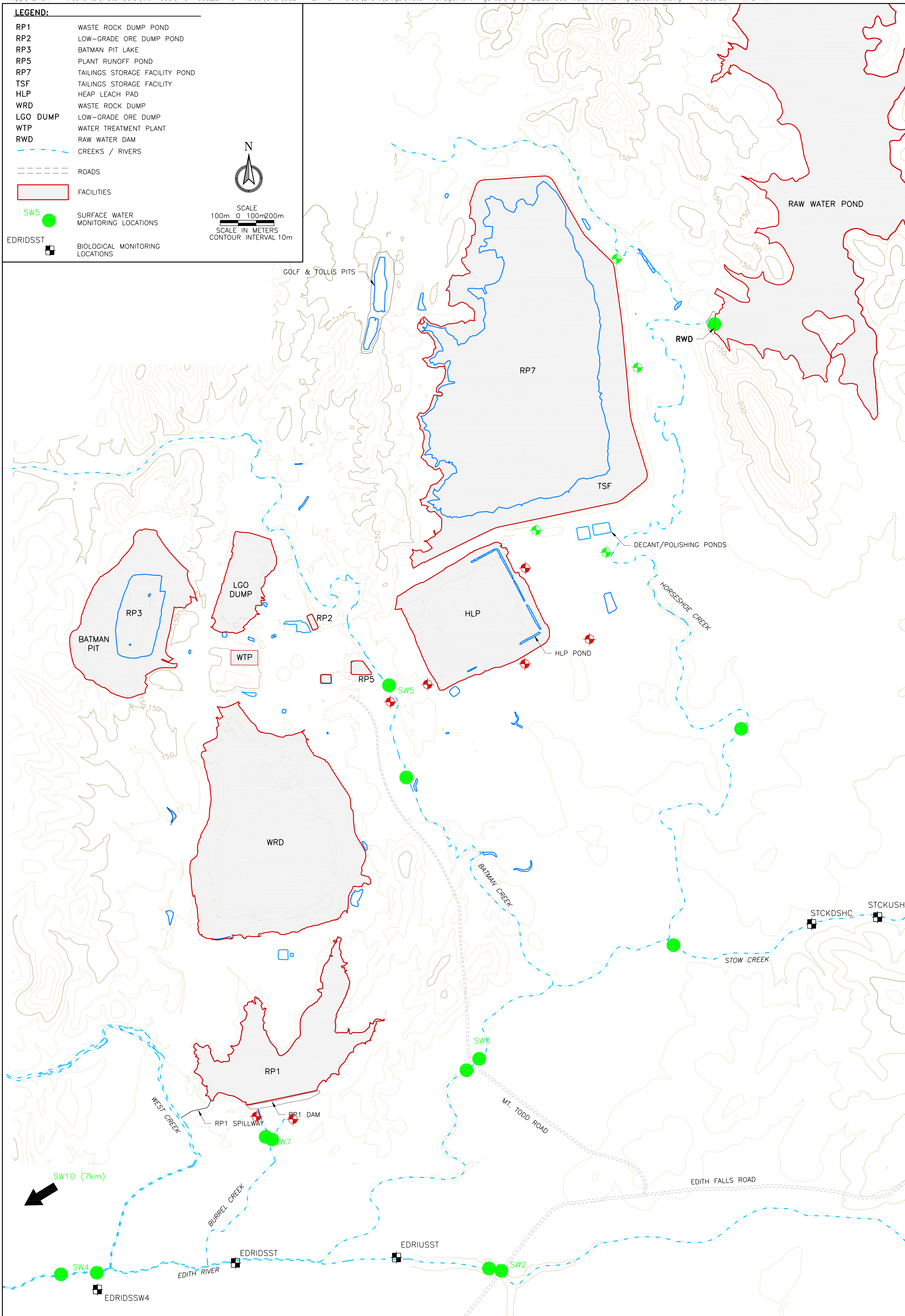

VISTA GOLD CORP.

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Golden, Colorado 80401
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Title:
SURFACE WATER MONITORING LOCATIONS

Project:
Mt TODD GOLD PROJECT
PREFEASIBILITY STUDY

Project no.:
310912A

Location:
NORTHERN TERRITORY, AUSTRALIA

Date:

FIGURE 5-2

TABLE 5-2: TRIGGER VALUE FOR FRESH WATER
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Analyte		(μgL^{-1}) Level of protection (% species) 99%	95%	90%	80%
Aluminum	pH > 6.5	27	55	80	150
Aluminum	pH < 6.5	ID	ID	ID	ID
Antimony		ID	ID	ID	ID
Arsenic (As III)		1	24	94 ^{C}	360 ^{C}
Arsenic (As V)		0.8	13	42	140 ^{C}
Cadmium		0.06 ^{H}	0.2 ^{H}	0.4 ^{H}	0.8 $^{C, H}$
Copper		1 ^{H}	1.4 ^{H}	1.8 $^{C, H}$	2.5 $^{C, H}$
Lead		1	3.4	5.6	9.4 $^{C, H}$
Manganese		1200	1900 ^{C}	2500 ^{C}	3600 ^{C}
Nickel		8 ^{H}	11 ^{H}	13 ^{H}	17 $^{C, H}$
Zinc		2.4 ^{H}	8.0 $^{C, H}$	15 $^{C, H}$	31 $^{C, H}$

C = May not protect key test species from chronic toxicity (this refers to experimental chronic figures or geometric mean for species)

H = Trigger value determined based on hardness of 30 mg/L $CaCO_3$; value is higher for harder water

ID = Insufficient data to derive a reliable trigger value

The quantitative discharge limits outlined in WDL 135 require that filterable copper concentrations at SW10 be no greater than 10 µg/L over SW2 background concentrations. This criterion was breached several times during each of the 2002 through 2009 wet seasons. The 2005/2006 wet season exceedances were partially attributed to delays in installation of the water management infrastructure. The water management strategy implemented by Vista from November 2006 through October 2007 appears to have reduced uncontrolled discharge from RP1 to four days from 17 days during the previous reporting period. Additional works undertaken by Vista during the 2007 dry season included the installation of a stage height and telemetry station at SW4 and flow meters on the siphon and pumping outlets from RP1 which allowed for enhanced discharge management during the 2007/2008 wet season. However, during 2008/2009 wet season, numerous copper exceedances occurred that were attributed to the inability of the pumping system to maintain the level of RP1 below the spillway during storm events. To minimize the potential for future uncontrolled discharges, Vista initiated pumping earlier to drawdown RP1 prior to onset of the wet season. This corrective action combined with the low rainfall volumes resulted in no exceedances during the 2009/2010 wet season.

In August 2009, Vista commissioned the WTP to treat water from RP1 and RP3 using milk-of-lime at a capacity of 193 m^3/hr (Vista, 2009). Lime treatment removed 98 percent of the cadmium, 98.8 percent of aluminum, and greater than 99 percent of the copper and zinc in RP1 water (TABLE 5-3). The sulfate concentration of treated RP1 water was largely unaffected (~1400 mg/L) due to the relatively high solubility of gypsum in water.

As currently configured, the treated solution including the reaction by-products (gypsum and metal hydroxide compounds), flow by gravity to RP7. Pending approval from the NT Government, this approach will continue during the wet season, whereas water and reaction by-products will be pumped from RP3 to the WTP and discharged to RP7 during the dry season. Once sufficient evidence has been obtained to verify that the water is of acceptable quality, Vista will seek additional authorization to discharge this water directly to Batman Creek or other acceptable discharge location.

Discharge of treated water without dilution such as could be envisioned during the dry season may result in elevated sulfate concentrations at the compliance point (SW10) compared to levels observed upstream of the Mt. Todd site at SW2. Review of the available 2007 surface water quality data shows that sulfate levels upstream of the Mt. Todd site at SW2 range from <0.1 to 2.2 mg/L with a median sulfate level of 0.10 mg/L (average = 0.37 mg/L) whereas sulfate concentrations range from 1.1 to 12.0 mg/L (median = 3.7 mg/L; average = 4.3 mg/L) at SW10.

Katherine Region groundwaters have also been declared a beneficial use for the protection of raw water for drinking water supply, agricultural, and industrial purposes. The groundwater monitoring network is limited to the immediate vicinity of RP1, RP7, and the HLP. Groundwater quality results exceeded the ANZECC and ANZMARC (2000) aquatic and/or recreational guideline levels for electrical conductivity (EC), sulfate, arsenic, cadmium, copper, iron, manganese, and zinc (MWH, 2006a).

TABLE 5-3: TREATMENT PILOT SYSTEM WATER QUALITY
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Settling Time	Number of Data Points	Statistics	pH	Electrical Conductivity	Filterable Aluminum	Filterable Cadmium	Filterable Copper	Filterable Zinc	Filterable Sulfate
			Standard Units	mS/cm	µg/L	µg/L	µg/L	µg/L	mg/L
Prior to Treatment	17	Average	N/A	2097	44529	125	10046	28571	1416
		Median		2100	42600	126	9410	28400	1350
		Range	3.8 to 4.0	1990 to 2250	41100 to 48600	110 to 130	9100 to 11300	27300 to 31500	1290 to 1590
0	17	Average	N/A	2105	529	2.52	90	8.6	1302
		Median		2110	449	0.42	4.0	1.8	1320
		Minimum	6.9 to 9.6	1990 to 2260	18.5 to 1860	0.10 to 22	2.6 to 1290	0.1 to 86	4.0 to 1570
24	15	Average	N/A	2081	471	4.26	88	41	1303
		Median		2090	176	0.59	2.4	0.5	1340
		Minimum	6.6 to 10.4	1970 to 2200	2.5 to 2240	0.10 to 42	1.4 to 1270	0.5 to 562	7.5 to 1570
48	15	Average	N/A	2096	437	7.02	88.8	22	1337
		Median		2090	162	0.48	2.5	0.5	1450
		Minimum	6.5 to 10.5	1950 to 2250	6.3 to 2540	0.10 to 64	1.5 to 1290	0.5 to 295	109 to 1590
72	15	Average	N/A	2061	354	3.05	91	22	1338
		Median		2070	100	0.56	3.3	1.0	1490
		Minimum	6.4 to 10.5	1960 to 2200	0.5 to 2040	0.10 to 31	1.7 to 1320	0.5 to 291	1.3 to 1580

5.4.6 Environmental Baseline Studies

Site characterization studies were conducted at the Mt. Todd site in support of the 1992 Draft EIS (Zapopan, 1992). Additional baseline data collection is ongoing as required by the site waste discharge license and to support development of required environmental and operational permits. Current baseline studies focus on water quality and geochemical characterization of mine waste. Cost estimates for each environmental discipline are provided in Section 21-Recommendations.

Climate and Meteorology

The north of the NT has two distinctive seasons; the monsoonal wet season which usually starts in November and ends in April and the dry season (May to October). Rainfall ranges from 15 mm falling during the dry season to 958.1 mm falling during the wet season and temperature ranges of 12.8°C to 35.8°C during the dry season and 20.3°C to 37.8°C during the wet season based on Bureau of Meteorology data. Evaporation rates are relatively consistent throughout the year with mean daily evaporation ranging between 4.8 mm to 7.8 mm. Daily pan evaporation and precipitation data have been measured on-site from December 1993 to October 2010. Existing data is adequate to characterize the resource.

Geology and Geochemistry

The 1992 Draft EIS identified three types of waste rock with Type I being considered non-acid producing (Zapopan, 1992). The original WRD was designed to encapsulate Type II (potentially acid generating, < 1% sulfur) and Type III (potentially acid generating, > 1% sulfur) with Type I waste rock which was expected to be amenable to revegetation and was not anticipated to have long-term problems with acidity or metal leaching. The project shut down without implementation of closure or reclamation activities; as a result, ARD/ML generation has become a primary environmental issue at the Mt. Todd site.

Tetra Tech was commissioned by Vista to conduct PFS-level geochemical characterization of Mt. Todd Project waste rock. Preliminary characterization of a composite tailings sample was also conducted. The primary objective of the geochemical testing program was to further understand the potential for ARD/ML associated with waste rock exposed to water and oxygen due to the proposed mining activities. The information obtained from this program can be used to further develop waste management criteria and predict drainage chemistry to assist with site water management. Detailed data analysis is provided in the Mt. Todd Gold Project Geochemical Characterization Program Report (Appendix J).

Waste rock samples were selected from the three distinct rock units identified from the 18 mappable rock codes, specifically:

- Greywacke
- Shale
- Mixed greywacke/shale (interbedded)

A total of 87 waste rock samples were subjected to acid-base accounting (ABA). Three subsamples from each of the three distinct units were selected for humidity cell testing.

The greywacke waste rock sample average HNO_3 extractable sulfide sulfur content of 0.19 wt. percent is comparatively low with interbedded and shale samples containing 0.51 and 0.31 wt. percent, respectively. HCl extractable sulfate sulfur was largely absent suggesting that minimal sulfide oxidation occurred prior to humidity cell testing. Although the sulfur content of Mt. Todd waste rock samples is relatively low (≤ 0.51 wt. percent average HNO_3 extractable sulfide sulfur), the potential for acid formation remains a concern due to the limited amount of neutralization potential (NP); on average NP $\leq$ 11 kg $CaCO_3$/tonne rock. A neutralization potential ratio (NPR) ABA screening criteria < 3 suggests that a majority of the waste rock samples are either potentially acid generating or highly likely to generate acid whereas approximately 30 percent of the samples are highly unlikely to generate acid. Field segregation methods will be developed to assist with handling waste rock during operations.

Nine waste rock samples, including three samples from each major rock unit, were subjected to up to 40 weeks of 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 elemental analysis were also obtained over the testing period. Of the nine samples subjected to kinetic testing, a shale sample with 0.43 wt. percent HNO_3 extractable sulfide sulfur and low NP = 3.7 kg $CaCO_3$/tonne rock produced acidic leachate (pH < 6) from initiation of testing. Elevated copper, lead, nickel and zinc levels were observed in leachate from the acid generating cell. Cells producing neutral pH leachate showed comparatively high levels of arsenic and antimony suggesting meteoric water contact could result in release of these constituents. The kinetic test results in combination with site wide water quality and quantity data can be used to predict drainage chemistry which will help guide site water management strategies.

The Mt. Todd tailings sample contained 1.25 wt. percent total sulfur of which the majority is sulfide sulfur from pyrrhotite and/or pyrite. The NNP of -20.2 kg $CaCO_3$/tonne and NPR < 1 shows that the tailings are likely to eventually generate acid. However, the tailings supernatant and water leach testing of the tailings have alkaline pH. Concentrations of some regulated constituents are elevated in the tailings supernatant and water leach extraction fluid. Kinetic testing of tailings samples will provide additional information about the long-term potential to generate acid and leach metals.

Biological Resources

Considerable work was done to establish this baseline in earlier permitting efforts, including the 1992 Draft EIS, although additional work will be necessary to fully characterize resources in the immediate vicinity of the Mt. Todd Project (Chadwick T&T Pty LTD, 2009). Interest in biological resources is driven by three factors: the impact assessment of proposed operations, the planning and design of reclamation and closure activities, and compliance with the EPBC Act. The EPBC Act addresses the protection of “matters of national environmental significance,” which include flora, fauna, ecological communities and heritage places.

Aquatic and Benthic. Fish kills observed in the Edith River in 2004 have led to additional monitoring of water discharging from the Mt. Todd site. DRDPIFR (now DoR) has conducted regular sampling of freshwater fish and macroinvertebrates in the Edith and Fergusson Rivers, and in the Stow Creek since 2003 (following the wet season). The DoR sampling effort focuses on gathering sufficient information to develop an understanding of environmental impacts resulting from existing conditions at the site and to support further development of a closure and rehabilitation strategy. Under the program, electrofishing was conducted in August 2008, which included collection of tissue samples for metals analyses. Results from the 2008 fish sampling study were not available for review. Interpretation of the macroinvertebrate results from 2003 through 2007 indicate that discharge processes and uncontrolled discharges from Mt. Todd Mine site during the 2006/2007 wet season did not cause detectable detriment to macroinvertebrate communities in the Edith River (Vista, 2007b). Vista also noted that the high overall similarity of the downstream sites on the Edith River with the upstream reference site is consistent with the last two years (2006 and 2005) of monitoring and markedly different than that displayed in 2004, when an impact was recorded.

Wildlife. A database search of EPBC resources identified the potential for a number of listed species (or their habitats) to occur in the vicinity of the project area; however, to date only the Gouldian finch, classified as "Endangered" under the EPBC has been documented to occur in the area (O'Malley, 2006). Major threats to the finch population within and around the Yinberrie Hills have been identified as inappropriate fire regimes and feral pigs; therefore, no specific conservation practices have been required at Mt. Todd for the finch (O'Malley, 2006).

Vegetation. There are three main land units within the mine site area which are described as low hills with open woodland, open grassland with an open shrub layer and tall closed woodland along riparian zones (Vista, 2007b).

Cultural/Archaeology

The majority of the project area was inventoried for cultural and archaeological sites for the 1992 Draft EIS. The following list of archaeological and historical significant areas was derived from the available reports (Chadwick T&T Pty LTD, 2009):

- Aboriginal sites of significance are known to exist on the lease area. Vista is working with the Jawoyn people to ensure that the appropriate measures are taken to protect these aboriginal areas;
- Sections of the Overland Telegraph Line remain at the site and should be protected as historically significant assets;
- Historical mines exist outside the lease area. It is unknown whether or not any historical mines are located within the current lease area; and
- Other historical assets may include graves, accommodation, fires/cooking pits, rubbish dumps (historical).

For future permitting efforts, the previous investigations into archaeological and historical assessments undertaken at the site should be compiled and mapped for easy recognition of significant areas.

Socio-economics

It is recognized that the community changes to Katherine will be substantial upon the commencement of the mine, with possible impacts listed as follows:

- Economic revenue;
- Demographic structure;
- Population increase and the associated impact to infrastructure;
- Disruption to the community through community concern; and
- Change in the level of community resources.

Socio-economic impacts have been a prime consideration by Vista who, in conjunction with the NT Government, has held regular community meetings and site visits. The DRDIFER (now the DoR) formed the Mt. Todd Rehabilitation Reference Group in 2005 with members from the Jawoyn Association, Amateur Fisherman's Association, Charles Darwin University, Environment Centre, Minerals Council, various other Government departments and the Katherine and Pine Creek town councils. Vista joined the Reference Group in 2006. Topics discussed include environmental monitoring and investigations, relevant site visits, updates on the management and status of the site and sharing communication/reports. The Mt. Todd BRS Report includes a risk-based management methodology that has identified the priority of work and discusses the above-mentioned communication and consultation strategy (DRDPIFR, 2008b).

5.4.7 Comments on Known Liabilities

The primary environmental issue at the Mt. Todd site is water management resulting from the project shutdown without implementation of closure or reclamation activities. All of the water retention ponds (excluding the raw water pond) and the pit contain acidic water with elevated concentrations of regulated constituents, including:

- Batman Pit (RP3);
- WRD retention pond (RP1);
- TSF and pond (RP7);
- The HLP;
- The plant runoff pond (RP5); and
- Low grade ore stockpile pond (RP2).

This water has been managed through a combination of evaporation, pumping to RP3 for containment, and controlled discharge to streams during major flow events. Historically, average wet season rainfall in the area results in uncontrolled overflow from RP1, RP2, and RP5 to the Edith River due to the high amount of precipitation received in short periods of time coupled with insufficient pumping capabilities. Other uncontrolled discharges to the Edith River during the wet season include surface seeps from the heap leach facility and surface seeps and underflow from the TSF dam. Vista adopted the water management plan developed by MWH (2006b) which appears to be successful at minimizing impacts on the Edith River downstream of the Mt. Todd site.

The existing water treatment plant (Existing WTP) is being used to raise the pH and reduce metals concentrations in water from site retention ponds prior to its discharge into the TSF. Pending approval, the water management plan will be further refined to optimize the ability to discharge water and eliminate the reliance on RP3 as a repository for contaminated waters. The challenges posed by ARD/ML are significant but are believed to be manageable.

Additional hydrogeologic investigations will be necessary to improve the understanding of operational dewatering requirements as well as fully develop the site water balance. These investigations will provide the necessary information to characterize the existing groundwater conditions and develop a more rigorous groundwater monitoring program for the site. It is noted that dewatering was minimal and very manageable during previous operations at the Mt. Todd site. However, the hydrogeology of the mining area has not been investigated in sufficient detail to comment conclusively on the future dewatering requirements or provide a dewatering cost estimate at this time.

Additional information will need to be gathered to assess the quantity of salvageable soil from new disturbances (e.g., expansion of WRD and Batman, TSF2), verify that sufficient quantities of growth medium will be available for closure of proposed and existing facilities. The adequacy of available soils for supporting plant growth and suitability for use as liner/cap material also needs to be evaluated.

The 1992 Draft EIS identified the following as the specific environmental issues to be considered for the project (Zapopan, 1992):

- Control of ARD;

- Heap leach solution containment;
- Tailings containment;
- Water management;
- Conservation of the Gouldian finch (*Erythrura gouldiae*) in the Yinberrie Hills;
- Impacts on Aboriginal sites of cultural significance;
- Impacts on historical and Aboriginal archaeological sites;
- Rehabilitation planning;
- Impacts of noise, dust, and blasting;
- Impacts on vegetation and fauna;
- Impacts on regional urban and social infrastructure; and
- General site management issues, such as weeds, mosquito-borne diseases, wildlife, and workforce behavior.

The Gouldian finch was classified as "Endangered" in 2001 by the NT Parks and Wildlife Commission (MWH, 2006a). The conservation of the Gouldian finch was an important consideration at the start of mining operations in 1993, when it was thought that the finch was confined to the Yinberrie Hills. However, the range of the finch is now believed to be broader than initially identified and less emphasis is being placed by the NT Government on this issue. There are currently believed to be no specific conservation practices enforced at the Mt. Todd site for the finch.

The Jawoyn people have strong involvement in the planning for the future of the Mt. Todd Project. Vista Gold has a good relationship with the Jawoyn, and at this time they have raised no concerns about re-opening the mine.

5.4.8 Reclamation and Closure

Tetra Tech was retained by Vista to develop a PFCP for the Mt. Todd Project in support of the overall PFS for renewed mining operations. This PFCP evaluates the closure liabilities that will transfer to Vista should a decision be made to restart mining operations at Mt. Todd and is supported by information and data provided in Appendices H, I, and J.

The major and immediate environmental challenges for Mt. Todd are the management of ARD/ML currently contained in several water storage facilities, and the management of precipitation and surface water runoff reporting to mine-related surface disturbance. Mt. Todd contains several facilities that capture and store ARD/ML including WRD Pond (RP1), the Low Grade Ore Stockpile Pond (RP2), the Batman Pit Lake (RP3), the Process Plant Runoff Pond (RP5), the TSF Pond (RP7), and the HLP Pond and Moat (Heap Leach Ponds). Several small unidentified ponds and seeps are scattered throughout the Mt. Todd.

ARD/ML is currently managed through a combination of practices as follows:

- Passive evaporation;
- Pumping excess water from RP1 to RP3 (previously pumped to RP7);
- Active water treatment in the Existing WTP; and
- Controlled and uncontrolled effluent discharges to creeks in the vicinity of the mine and the Edith River during major flow events.

Pumps installed at the Heap Leach Ponds, RP2, RP5, and the pumping capacity at RP1 has been increased, which has significantly reduced the frequency of uncontrolled effluent releases from these ponds to the Edith River and its tributaries.

Throughout the mine-life, Vista should anticipate, plan and design for, and implement effective plans for:

- Year-round collection, containment and treatment of all ARD/ML prior to effluent release;
- Identification of PAG and non-PAG materials, as well as materials that have the potential to leach constituents in concentrations above applicable water quality-base effluent standards (metaliferous);
- Selective handling of PAG and non-PAG material and potentially direct treatment of PAG materials throughout the mine-life to prevent or reduce the generation of ARD/ML;
- Separation of unimpacted surface and ground water from PAG and metalliferous materials, and ARD/ML;
- Short- and long-term hydrologic isolation of PAG and metalliferous materials from ground and surface water;
- Facility and site-wide closure; and
- Control of storm water to prevent excessive erosion and sedimentation.

Specific recommendations related to these and other closure and water treatment needs are provided in Section 21-Recommendations.

The major facilities that currently exist at Mt. Todd, which are included as part of the 10.65 Mtpy mine plan are as follows:

- Batman Pit (RP3);
- WRD;
- RP1 and pumping system,
- TSF1 (RP7);
- Process Plant and Operations Area;
- RP5 and pumping system;
- HLP;
- HLP Ponds and pumping system;
- LGO Stockpile;
- RP2 and pumping system;
- WTP; and
- Mine roads and other ancillary facilities (e.g., pipelines).

The new facilities proposed for closure and the mine-life water treatment system are as follows:

- Run-on diversions up-gradient of the RP1, TSF1, and WRD;
- New WTP;
- Equalization Pond;
- Sludge Disposal Cell;

- TSF1 and TSF2 Operational (and Closure) Spillway;
- Modified TSF1 Decant Ponds and Modified TSF2 Sumps;
- TSF1 Collection Ditch;
- TSF2 Collection Ditch;
- New Low Grade Ore Stockpile (LGO2) Collection Ditch;
- LLDPE (or equivalent)-Lined LGO2 Sump;
- Collection Ditch at toe of closed WRD;
- Modify HLP Seepage Collection Pump and Pipeline;
- Pumps and pipelines;
- Clay Borrow Area; and
- Three anaerobic treatment wetlands (or equivalent passive/semi-passive water treatment system).

The PFCP includes descriptions, approximate dimensions and performance criteria for proposed facility. Facility arrangements, and design drawings and details have not been completed at this stage of the planning process.

The closure and water management goals for the Mt. Todd Project include:

- Control acid-generating conditions;
- Reduce or eliminate the acid and metal loads in seepage and runoff water through appropriate treatment;
- Minimize adverse impacts to the surface and ground water systems surrounding Mt. Todd;
- Physically and chemically stabilize mine waste and other mine-related surface disturbances;
- Protection of public safety;
- Comply with applicable water quality-based effluent standards and the WDL; and
- Comply with NT Government regulations governing mine development and closure.

Closure plans and strategies for each major facility at Mt. Todd and the mine-life water treatment system are briefly summarized below. Appendix I also includes facility dimensions, volumes, expansion plans and other pertinent quantities.

Batman Pit

Based on water balance modeling of the Batman Pit (See Appendix I), a hydrologic sink should be maintained passively during the post-closure phase. Therefore, the need for active pit dewatering and treatment of pit water is assumed to be unnecessary following the closure phase.

Scaling and blasting of select pit benches and walls will be completed during the production phase to reduce the potential of human injury due to rock fall, and improve pit wall stability and aesthetics. A berm will be constructed around the entire perimeter of the Batman Pit primarily to impede human access to pit and also reduce the inflow of surface water to the pit.

Waste Rock Dump

A preliminary WRD design was completed by MDA. As designed, these WRD expansion plans include:

- Avoid placing waste rock in RP1;
- Avoid grading of waste rock at the end of the mine-life;
- Incorporate concurrent reclamation in 14 of the planned 16 years of WRD construction;
- Incorporate concurrent reclamation of the entire WRD by Planning Year 15; and
- Create a 'geomorphic' final surface that includes:
    - Highly dissected, non-uniform, and complex slopes;
    - Opportunities for dispersing rather than concentrating runoff from the surface of the WRD; and
    - Final WRD configuration similar to the surrounding undisturbed topography.

To achieve a stable WRD configuration over time, and limit erosion, channel scour and overtopping we anticipated that engineered armored channels will be located along preferential flow paths near the center of concave slopes. As designed the final surfaces of the WRD will include horizontal benching. The WRD will be benched appropriate to geotechnical stability constraints. Storm-water drainage, erosion, and sediment controls will be designed and constructed to minimize erosion and channel scour and the area will be revegetated. Stormwater collected on benches will be conveyed to the toe of the WRD through the engineered channel located near the center of the concave slopes. A surface water collection ditch will be constructed along the down-gradient toe of WRD following placement of the store and release cover at closure. The surface runoff from reclaimed WRD will be routed around RP1 to avoid the comingling of ARD/ML seepage from the WRD with non contact water.

Concurrent installation of a store and release cover following attainment of final grades is proposed for the closure of the WRD (as well as other facilities at Mt. Todd). The goal of a store and release cover is to effectively reduce percolation of precipitation into the waste rock (as well as other PAG and metalliferous materials) and thereby, reduce steady-state seepage following closure; long-term water treatment and sludge disposal costs; and potential adverse impacts to ground and surface water. The proposed store and release cover includes a multi-layer cover, with deliberate revegetation of the surface and balancing of the percolation of meteoric waters through the cover with the water storage necessary to establish vegetation, while avoiding exposure of mine waste due to erosion. The basic design of the store and release cover proposed for the closure of the WRD and estimate hydrologic properties are discussed in the PFCP and are applicable to other capped facilities at Mt. Todd.

Prior to cover-placement, subgrade material will be deep ripped to reduce compaction and prevent slippage of the cover. The first lift of the store and release cover will be low-permeability material (clay) incorporated into the graded non-PAG surface of the WRD. This will be followed by placement of a 0.7 meter-thick blended non-PAG waste rock-clay cover that is suitable plant growth medium. The primary source of cover material for closure at Mt. Todd will be non-PAG waste rock from the Batman Pit and clay from borrow areas.

The same cover design described above is proposed for the closure of the Low Grade Ore Stockpiles 1 & 2, Process Plant and Pad Area, and mine roads. Closure of the TSF1 and TSF2, HLP, and Sludge Disposal Cell will include a 1.0 m-thick blended non-PAG waste rock-clay cover that is suitable for revegetation. Schematics drawings of the cover systems proposed at Mt. Todd are provided in the PFCP.

An evaluation of the hydraulic performance of the cover was carried out using the variably-saturated flow model, in a one-dimensional mode under average climatic conditions at the site (PFCP, Attachment A). Based on the preliminary results of this analysis, it appears that constructing a “net zero flux” cover at Mt. Todd site is feasible by constructing a 1 meter cover composed of greywacke waste rock. To be conservative the estimated net flux of annual precipitation through the store and release covers proposed for the WRD (i.e., 0.7 m-thick) and HLP (1.0 m-thick) was increased to five percent. The PFCP presents a brief summary of the results of the analyses and provides some recommendations for the design of the soil covers at Mt. Todd.

The estimated net flux of annual precipitation through the store and release covers, as well as other assumed properties of the WRD, HLP, and TSF1 and TSF2 over the mine-life (e.g. incident precipitation, foot print area, catchment area runoff, total and drainable porosity, saturated volume, runoff rate, evaporation rate) were used to estimate seepage and runoff rates from the WRD, Heap and TSF1 and TSF2 (See Appendix I) and as a basis for the development of mine-life water treatment system requirements and costs (See discussion below).

Based on the geochemical testing and analysis program conducted for the PFS (Appendix H), approximately 30 percent of the waste rock excavated during renewed mining will be non-PAG. A Waste Rock Management Plan (WRMP) will be developed that specifies how waste rock is to be handled to minimize the potential for ARD/ML and maximize the use of non-PAG waste rock for closure.

Additional analysis and design will be required to confirm that the planned final configuration of the WRD and drainage control approach discussed above is suited to the hydrologic conditions; and uniformity, dimensions and durability of waste rock at Mt. Todd as well as the design of the store and release cover. In addition, the results and conclusions of WRMP will likely require modifications to the WRD and cover design discussed above.

Tailings Disposal Facility

Slurry tailings will be disposed of in the TSF1 and TSF2. Soil will be salvaged from the footprint of TSF2 and temporarily stockpiled prior to the construction of the tailing dam and inundation with tailing. Initiation of closure activities at TSF1 and TSF2 are anticipated in planning years 8 and 16, respectively.

Tetra Tech anticipates that the impounded tailings surface conditions in TSF1 and TSF2 at the end of tailings deposition activities will be similar to the current conditions. Currently, beach sands cover only a narrow strip near the inside crest of the existing TSF1 dam and slimes cover the remainder of the surface of the TSF1. As such, Tetra Tech has assumed that at closure the majority of the impounded surface of the TSF1 and TSF2 will be primarily composed of thixotropic tailings (thick like a solid but flows like a liquid when a sideways force is applied) that will maintain a high degree of saturation for many years, unless they are actively dewatered and consolidated, covered with material (i.e., increase surcharge) or chemically treated to increase their strength.

A small pool will likely remain post-closure on the impounded surface of the TSF1 and TSF2 near the spillway. Surface runoff from the impounded and capped surface of the TSF1 and TSF2 will be conveyed via surface drainage channels to the tailings pool. The operational spillway constructed during the production phase will be modified at closure to safely convey surface runoff from the TSF1 and TSF2 pools to Horseshoe Creek.

At and following closure, the release of water from the TSF1 and TSF2 pools to Horseshoe Creek will depend on pool water quality. Plans will be developed for the treatment of TSF1 and TSF2 surface discharge prior to release to Horseshoe Creek if there is unacceptable risk that WDL or water quality-based effluent standards will be exceeded. To evaluate this risk, Tetra Tech recommends regular monitoring and review of TSF1 and TSF2 pool water quality.

A 1 m-thick cover composed of non-PAG waste rock will be installed on the impounded surface of the TSF1 and TSF2 to bridge thixotropic tailings and allow equipment access for the installation of the 1m-thick store and release cover (discussed previously). Instead of clay, the soil removed from the footprint of the TSF2 will be hauled to TSF1 (potentially hauled directly to the TSF1 without stockpiling) and used in the store and release cover. This assumes that use of soil instead of clay will not substantially increase the net flux of precipitation through the TSF1 cover.

On the outside slopes of the main dams of TSF1 and TSF2 dam and saddle dams of TSF1 the 1 m-thick store and release cover will be installed. Prior to cover placement on the TSF1 and TSF2 dam faces, subgrade materials will be deep ripped to reduce compaction and prevent slippage of the store and release cover. To increase the erosion resistance of the store and release cover on the TSF1 and TSF2 dams it may be necessary to apply erosion control fabric to the cover surface, increase the content of coarse rock fragments in the top of the cover, and/or roughen the cover surface. To the degree practicable, the store and release cover will be installed concurrently on the TSF1 and TSF2 dams.

The TSF1 decant pipes will be plugged with concrete and a seepage collection ditch or a series of ditches will be constructed at closure to collect seepage from the toe of the TSF1. The bottom and down-gradient interior side slope of collection ditch will be lined with LLDPE (or equivalent). The Decant Ponds will be modified to receive seepage from the TSF1 collection ditch via gravity and the TSF1 foundation drains. This same design approach will be applied at TSF2. However, a reclaim barge will be used for the operation of the TSF2. Therefore, decants are absent and foundation drains will drain to small sumps at the toe of the tailings dam. TSF2 sumps will be modified to receive seepage from the TSF2 foundation drains and TSF2 collection ditch via gravity.

Tetra Tech estimates the tailings disposed of in the TSF1 and TSF2 will be PAG and seepage from the TSF1 and TSF2 will be ARD/ML (Appendix H). Tetra Tech also assumed that the operation of the Process Plant during the production phase will use all excess water from the TSF1 and TSF2. As such, TSF seepage will only be collected and treated during and following the closure each TSF. ARD/ML collected in the Decant Ponds and sumps will be initially pumped to the New WTP for treatment prior to release. The estimated rates of drain-down and steady-state seepage from the TSF1 and TSF2 are provided in Appendix I. These estimates were used as inputs to estimate water treatment requirements during and following closure of each TSF. The Decant Ponds, sumps, and the collection ditches will be maintained until it is feasible to treat this and other ARD/ML on-site using passive treatment systems (see discussion below).

Process Plant and Pad Area

A new process plant will be built at the current Process Plant and Pad Area. Tetra Tech does not anticipate the existing process plant and pad disturbance will change significantly due to the construction of the new process plant. Once mineral processing ceases, the Process Plant will be decommissioned, decontaminated, demolished and any reusable equipment and materials will be salvaged and resold.

The current operating assumption is that the Process Plant or portions thereof, will be demolished (disassembled), removed (salvaged) or hauled to a solid waste landfill or other suitable locations on-site, capped and reclaimed. Some buildings will remain to support closure operations and post-closure site maintenance.

Concrete foundations, walls and bridges and other non-reactive, non-combustive, non-corrosive and non-hazardous demolished waste will be broken up and either:

- Placed in the WRD; and/or
- Buried in-place or backfilled against cutbanks 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 and Pad Area will be graded to blend into the surround topography and drain towards Batman Creek. Stormwater drainage controls, and erosion and sediment controls will be designed and constructed to minimize erosion and channel scour. The Process Plant Area and Pad will be covered with the 0.7m-thick store and release cover (described previously) to prevent exposure, non-reactive, non-combustive, non-corrosive and non-hazardous waste. Prior to cover placement subgrade materials will be deep ripped to reduce compaction and prevent slippage of the cover. The cover will be revegetated and protected from erosion as described previously. We assume that the Process Plant and Pad Area will no longer be a source of ARD/ML following closure.

The New WTP, Equalization Pond and Sludge Disposal Cell will be constructed in Planning Year -2 and be located on or immediately adjacent to the Process Plant and Pad Area. These water treatment and sludge disposal facilities will remain in place during the production phase, up-graded if necessary and used to treat ARD/ML during the closure and post-closure phases. These facilities will be closed as discussed below when it is feasible to treat ARD/ML in passive treatment systems. RP2 and RP5, will be closed as described below if it is determined they are no longer needed for ARD/ML storage or containment during the closure and post-closure phases.

Heap Leach Pad and Pond

The Heap is not needed for renewed mining at Mt. Todd and can be closed immediately. Due to extent of exposure to precipitation, Tetra Tech assumes the Weak Acid Dissociable (WAD) Cyanide concentration of Heap effluent and pore water meets applicable standards. Therefore, Tetra Tech assumes deliberate rinsing of the Heap prior to initiation of closure activities is not required. While not confirmed by test results, the material in the Heap is likely PAG due to the acidic nature of seepage stored in the Heap Leach Ponds. These assumptions must be verified prior to closure of the Heap.

Vista will assume the responsibility to close the Heap with the goals of reducing acid and metal loads to ground and surface water and the New WTP. The Heap will be closed in the first year of the production phase in a manner similar to the WRD. All grading will occur within the confines of the existing liner.

Portions of the Heap will be used to install test plots and fills. 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. Conclusions regarding the performance of closure alternatives tested on the Heap will be used in the develop of final closure plans and designs at Mt. Todd, and to validate vadose zone and water balance models to improve the prediction of long-term water treatment requirements and adverse impacts to surface and ground water in the vicinity of Mt. Todd.

The Heap Leach Ponds will be modified to continually receive seepage and runoff from the Heap during and following grading, capping and closure. Tetra Tech estimates that seepage from the Heap will be ARD/ML and continue following closure. Therefore, ARD/ML collected in the Heap Leach Ponds will be initially pumped to the WTP for treatment prior to release. The estimated mine-life rate of seepage from the Heap is provided in Appendix I. These estimates were used to estimate water treatment volumes prior to and following Heap closure. The modified Heap Leach Ponds will be maintained until long-term seepage is feasible to treat this and other ARD/ML on-site using a passive treatment system (see discussion below).

Low Grade Ore Stockpiles

The LGO1 will no longer be needed for mineral processing after approximately year five of the production phase. The LGO2 will be located adjacent to the northeast quadrant of the WRD. While not confirmed by test results, the material below the LGO1 is assumed to be PAG. We have also assumed that material below the LGO2 will also be PAG. These assumptions must be verified prior to closure of the LGO1 and LGO2.

A seepage and runoff collection ditch and sump will be constructed at the toe of LGO2 to ARD/ML. The ARD/ML collected in the sump will be conveyed to the New WTP for treatment prior to release.

Closure of the the LGO1 and LGO2 will included removal of residual ore from the stockpile areas. Tetra Tech assumed these areas will not be graded. However, stormwater drainage, erosion, and sediment controls will be designed and constructed to minimize erosion and channel scour. Prior to cover placement subgrade materials will be deep ripped to reduce compaction and prevent slippage of the cover. This will be followed by placement of a 0.7-m thick store and release cover described previously. The cover will be amended and revegetated as described previously.

We assume that the LGO1 and LGO2 will no longer be sources of ARD/ML following closure.

Mine Roads

Mine roads will either remain in place and be reduced in size for local access, or be reclaimed and abandoned. Mine roads will be graded to blend into the surrounding topography. Prior to cover placement subgrade materials will be deep ripped to reduce compaction and prevent slippage of the cover. Storm water drainage controls, and erosion and sediment controls will be designed and constructed to minimize erosion and channel scour and the area will be revegetated.

Water Storage Ponds

During the pre-production phase, the Equalization Pond will be constructed for mixing of ARD/ML from various on-site sources prior to treatment and to temporarily store ARD/ML in case of system upset (i.e. ARD/ML surge due to extreme storm events or shut-down of the New WTP). The pond will be LLDPE-lined (or equivalent). The cells will likely included a spillway or decant system and containment structures to address overflows. In the event of a system failure or shut-down for maintenance of the New WTP, the Equalization Pond would provide approximately 3 days of ARD/ML storage at an inflow rate equal to the treatment capacity of the New WTP or 1,000 m^3/hour.

As previously discussed two small sumps will be constructed at the toe to TSF2 and one small sump will be constructed at the toe of the LGO2. A small LLDPE-lined sump will be constructed in a similar manner as the Equalization Pond to temporarily store ARD/ML from the LGO2.

Prior to the closure of the heap, moat, pond, pumps and pipeline will be modified according to the closure grading plans. The modifications will be completed to allow continued collection and conveyance of seepage from HLP to the New WTP. Tetra Tech anticipates that RP2 and RP5 will remain during the production phase. Two pumps will added to RP2 and RP5 to reduce the potential for overflows from these ponds (HydroGeoLogica, Inc. and Tetra Tech, 2010). RP2 and RP5, will be closed when it is determined they are no longer needed for ARD/ML storage or containment during the closure and post-closure phases.

Tetra Tech anticipates that RP1 will remain during the pre-production and production phases. Therefore, ARD/ML from the WRD collected in RP1 during the pre-production and production phase will be pumped to the New WTP for treatment prior to release.

All other existing ponds as well as the Equalization Pond are anticipated to remain through closure. Pumps and pipelines at water storage ponds will be modified and moved as necessary to convey seepage and runoff according to mine-life water handling plans (See Appendix I).

During the closure of TSF1 beginning in planning year 8, the Decant Ponds will be modified to receive seepage via gravity from TSF1 collection ditch and foundation drains. Passive Water Treatment System #1 will be constructed near the HLP and TSF1.

Seepage from the WRD will likely be ARD/ML and continue following closure. Passive Water Treatment System #2 will be designed to treat ARD/ML from the WRD (and potentially other ARD/ML sources) to meet applicable water quality standards during the closure and post-closure phases. The construction and decommissioning of RP1 and the construction of Passive Water Treatment System #2 may be delayed based on the quality of runoff from the surface of the reclaimed WRD; and the erosional stability of the store and release cover on the surface of the WRD.

Residual water stored in RP1 will be pumped to the New WTP prior to construction of Passive Water Treatment System #2 and reclamation of the remainder of RP1. Sediments accumulated behind the RP1 dam will be tested and if appropriate removed or stabilized in place. The RP1 dam will be breached to ensure it no longer impounds water and the remaining area within RP1 will be reclaimed. If sufficient excess treatment capacity exists, the flow of ARD/ML reporting to RP1 may be routed to Passive Water Treatment System #1 instead of constructing Passive Treatment System #2.

During the closure of TSF2 the sumps will be modified to receive seepage via gravity from TSF2 collection ditch and from the TSF foundation drains.

All ponds at Mt. Todd will be maintained for the collection of seepage, stormwater and ARD/ML until long-term quality of water collected in the ponds meet applicable standards; flows to collection system cease; or alternative passive water treatment system are installed. The ponds anticipated to remain in the post-closure phase until site-wide passive treatment of ARD/ML is feasible include RP2; RP5; HLP Ponds; the TSF2 sumps; the Decant Ponds; and the Equalization Pond. These ponds may be incorporated into the passive water treatment system or used as backup water storage in case treatment upset occurs.

Sludge Disposal Cell

During the pre-production phase the Sludge Disposal Cell will be constructed adjacent to the New WTP for the permanent disposal of water treatment sludge. We have assumed that the sludge would be conveyed in an HDPE pipe to a lined Sludge Disposal Cell. The cell will be

LLDPE-lined (or equivalent) and will likely included a spillway or decant system and containment structures to address overflow resulting from storms in excess of the design event. Excess water from sludge consolidation may be evaporated within the sludge disposal facility. Additional sludge storage cells may be required, depending on the volume of water treated and the properties of the sludge.

When the Sludge Disposal Cell reaches its storage capacity or passive treatment systems are installed and the New WTP is decommissioned, a 1 meter-thick blended, non-PAG waste rock (or other available and similar material)-clay cover will be installed on the surface of the cell. Grading will be completed to promote rapid surface runoff from the surface of the closed cell. Storm water drainage controls, and erosion and sediment controls will be designed and constructed to minimize erosion and channel scour and the area will be revegetated.

Clay Borrow Area

Based on drill core data collected from the development of groundwater monitoring wells down gradient of the toe of the TSF, sources of clay may be available on site. However, significant uncertainties exist related to the sources, quantity and quality of clay reasonably available at Mt. Todd. Clay or other low-permeability materials are a critical component of the proposed store and release cover design that will, in large part, control the moisture retention and release properties of the store and release cover. Therefore, it is essential that all viable sources of clay (as well as other reclamation materials such as rip rap and drain rock) be inventoried and tested during the pre-production phase to determine the suitability and quantity of clay sources at Mt. Todd.

To estimate the borrow area closure cost, Tetra Tech assumed that storm water drainage controls, and erosion and sediment controls will be designed and constructed to minimize erosion and channel scour within the Clay Borrow Area. The surface of the excavated pit will be amended with organic matter to improve revegetation performance, seeded and mulched and crimped. Following this, the borrow area will be graded and reclaimed (on an interim or final basis) or remain open until closure activities are completed on site.

Water Treatment

There are two fundamental approaches to water treatment – active and passive treatment systems. The first phase of water treatment at Mt. Todd will include active water treatment in the New WTP. This phase will be initiated during pre-production phase of the project and will continue to until ARD/ML flow and water quality properties are conducive to treatment in passive/semi-passive water treatment systems. The second phase of ARD/ML treatment, which include passive water treatment in an anaerobic wetland (or equivalent passive/semi-passive treatment system) is anticipated to be initiated during the production phase and completed six years following the closure phase

We anticipated that a passive treatment system will be constructed to treat drain-down and steady-state flow from the reclaimed TSF1 and HLP during production in planning year 13. ARD/ML from the WRD is anticipated to be treated in a passive treatment systems immediately following the closure phase in planning year 19. ARD/ML from the TSF2 is anticipated to be treated in a passive treatment system six years following the closure phase in planning year 24. The basic design of these treatment systems are discussed below.

Active Water Treatment: The goals of active water treatment and sludge disposal at Mt. Todd are:

- Partial dewatering of the Batman Pit by approximately planning year -1 to permit in-pit preparation activities (lay backs) prior to the initiation of mining while meeting WDL and Edith River water quality–base effluent standards.
- Year-round collection, containment and treatment of all ARD/ML prior to effluent release;
- Ensure that treated ARD/ML complies with the WDL numeric water quality standards;
- Use neutralization reagents (reagents) and flocculants efficiently;
- Minimize the volume and water content of sludge produced from water treatment;
- Provide adequate long-term storage and containment of sludge in the on-site disposal facility; and,
- Promote rapid sludge consolidation.

During the pre-production, production, closure and post-closure phases ARD/ML will be continuously collected and pumped to the treatment facility prior to release. Active water treatment will occur in a fixed facility (New WTP) of pipes, metering pumps, automated delivery systems, agitator, reaction/mixing vessels, and clarifiers. The New WTP should be located on or adjacent to the Process Plant and Pad Area. This will likely be a good location for the New WTP. To the degree possible equipment from the existing WTP (inflow pipes, on-site utilities and offices, lime silo) will be salvaged and incorporated into the New WTP.

Appendix I presents a summary of major ARD/ML sources, source water quality and inflows to the New WTP during each mine-life phase. These flow and water quality estimates were derived from the mine-life water balance model simulations and the predicted water quality of each ARD/ML source provided in Appendix I. Flow estimates are based on minimum and maximum average annual flows from each ARD/ML source anticipated during each mine-life phase and do not consider high flows that could be generated by high intensity storm events or extreme annual precipitation.

The recommend capacity of the New WTP is approximately 1000 m^3/hour. The recommended capacity was driven almost exclusively by the constraints as follows:

- Dewater the Batman Pit to the extent necessary to allow in-pit preparation activities during the pre-production phase and regular pit production activities during the first few years of the production phase until the Batman Pit is completely dewatered (based on the estimated volume of water in the Batman Pit as of October 2010 is approximately 8.0 million m^3);
- At a minimum, maintain pit water elevation 5 m below the lowest planned pit operation anticipated during each year of pit preparation and production;
- Maintain the pit in a dewatered condition following the first few years of the production phase; and
- Comply with all site water management system requirement such as:
    - Avoid overtopping of on-site ponds; and
    - Maintain RP7 at an elevation less than or equal to 136 m to permit construction of the first three upstream raises of the TSF dam.

A complete description of the water management constraints used in water balance model is provided in Appendix I.

Appendix J describes Tetra Tech's methodologies and the estimates of hydrated lime consumption and sludge production during the 26-year period when the New WTP is anticipated to be in operation.

Based on the estimated time-variable flow rates and water quality of the ARD/ML sources at Mt. Todd (Appendix I) during the mine-life approximately 62 million m^3 of ARD/ML will be treated in the New WTP. Water treatment is estimated to consume approximately 17,000 m^3 of hydrated lime and produce 80,000 m^3 of sludge. Approximately 40 million m^3 of ARD/ML will be treated in the New WTP during the pre-production and production phase. However, due to the rapid dewatering of the Batman Pit during the pre-production phase, treatment rates are approximately 12 million m^3/year, compared to the 2 million m^3/year during the production phase. At closure, average treatment rates increase to approximately 4 million m^3/year. This is largely attributable to dewatering RP1 to permit the removal of sediments and breaching of the RP1 Dam and draining RP7 and TSF2 to permit installation of the closure cover on the impounded surface of the TSF1 and TSF2. Average treatment rates decline to approximately 1.6 million m^3/year during the first six years of the post-closure phase. Therefore, excess treatment capacity in the New WTP will exist following the pre-production phase.

These conclusions and design recommendations are based on preliminary estimates and must be confirmed based on detailed hydraulic investigations and Vista's design and risk tolerances. Until experienced water treatment engineers and chemists conduct a thorough engineering estimate as part of the feasibility study, all conclusions and recommendations provided here and in Appendix J should be viewed as early-stage planning products.

Passive/Semi-Passive Water Treatment: The goals of the passive/semi-passive water treatment at Mt. Todd are:

- Eliminate or drastically curtain the costs and continual inputs (e.g. reagents, power, staff) required to operate and maintain the New WTP;
- Eliminate sludge disposal cell operations and maintenance;
- Year-round collection, containment and treatment all ARD/ML prior to effluent release; 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 with discharge of between ~ 24 m^3 to~ 48 m^3/hour, low levels of mineral acidity and sufficient space available to construct passive or semi-passive treatment system. Passive water treatment system have successfully treated ARD/ML flows ~ ≤ 120 m^3/hour (See discussion in Appendix J).

Tetra Tech estimates the following:

1. During the production phase in planning year 13 of the post ARD/ML flows from the reclaimed TSF1 and HLP will be approximately ≤ 120 m^3/hour. These flows will be treated in Passive Treatment System #1;
2. Immediately following the closure phase in planning year 19 ARD/ML flows from the reclaimed WRD will be approximately ≤ 120 m^3/hour. These flows will be treated in Passive Treatment System #2; and
3. In year six of the post-closure phase (planning year 24) ARD/ML flows from the TSF2 will be approximately ≤ 120 m^3/hour. These flows will be treated in Passive Treatment System #3.

At or near these times during the mine-life, anaerobic wetlands or SAPS may be suitable for passive treatment of ARD/ML at Mt. Todd based on the estimated flows and water quality post-closure (Appendix I). Flow-weighted average acidity of all sources when total post-closure flows are approximately ≤ 120 m^3/hour may be on the order of 100 to 400 mg/L Non-CO_2 Acidity (as

$CaCO_3$ mg/L). The estimated area of the Passive Treatment System #1, #2 and #3 necessary to treat ARD/ML is 4, 0.3 and 6 hectares, respectively.

These estimates are based on ARD/ML treatment to 0.0 mg/L Non-CO_2 Acidity (as $CaCO_3$ mg/L) and an anaerobic wetland treatment efficiency of 16.4 g acidity/m^2/day (Skousen. J and P. Ziemkiewicz, 2005).

Estimating flows and water quality tens of years in the future is wrought with uncertainty and the introduction of potentially significant error. These uncertainties and errors may also be magnified due to changes in the mine plans, changes in closure plans and designs, climatic conditions, unforeseen material characteristics, etc. Therefore, the estimates and recommendations provided above 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.

5.4.9 Closure Cost Estimate

Tetra Tech estimated quantities (e.g. facility dimension, material/fluid volumes, surface areas, disturbance footprints) for the closure of major facilities at Mt. Todd and mine-life water treatment based on closure and water treatment plans discussed above. Closure and water treatment costs were estimated at a ± 25 percent level of accuracy based on the following:

- 10.65 Mtpy mine plan, and existing engineering and data presented in the PFS below;
- Mine-life (i.e. pre-production, production, closure and post-closure project planning phases) water balance simulations and water quality estimates (Appendix I);
- Geochemical testing and analysis program (Appendix H);
- Use of existing and new water management systems and infrastructure;
- Estimates of environmental conditions throughout the mine-life;
- NT Government mine closure and environmental protection regulations and guidelines;
- Published unit costing references;
- Tetra Tech's recent mine closure and water treatment costing experience; and
- Best professional judgment.

Tetra Tech used a 2010 estimate for the demolition of a 40,000 tpd Silver-Zinc Mill (US$12.3 Million) process plant as the basis for estimating the cost to demolish the Process Plant at Mt. Todd since detailed information regarding the design of the planned plant was not available in time to develop design-specific demolition costs.

Based on the costing approach described above, the PFS-level cost estimate for implementing this closure plan is $ 67,864,000 as summarized in TABLE 5-4 this cost estimate includes closure of the sludge disposal cell, the equalization pond, the clay borrow pit, contingency, engineering re-design, road maintenance during closure activities, incidentals and annual site maintenance and monitoring for the first 6 years of the post-closure phase. TABLE 5-4 also includes pre-feasibility level cost estimate for implementing the mine-life water treatment plan of $ 36,590,000. This cost estimate includes construction of the water treatment and sludge disposal system, and Mine-Life water treatment operation and maintenance, lime and pumping of water and sludge.

TABLE 5-4: PREFEASIBILITY-LEVEL CLOSURE AND MINE-LIFE WATER TREATMENT COST ESTIMATE
VISTA GOLD CORP. – Mt TODD GOLD PROJECT
January 2011

Area	Cost[1]
Tailings Storage Facility 1 (TSF1)	$ 9,101,000
Tailings Storage Facility 2 (TSF2)	$ 19,018,000
Heap	$ 2,585,000
Process Plant And Pad Area	$ 11,280,000
Batman Pit	$ 205,000
Waste Rock Dump	$ 8,620,000
WRD Retention Pond	$ 1,709,000
Low Grade Ore Stockpile 1 (LGO1)	$ 128,000
Low Grade Ore Stockpile 2 (LGO2)	$ 244,000
Mine Roads	$ 3,786,000
Clay Borrow Area	$ 135,000
Sludge And Equalization Pond Closure	$ 273,000
Total Direct Closure Cost	**$ 57,084,000**
Mobilization/Demobilization (Assume On-Site Mining Equipment Fleet Used)	$ 0-
Incidentals (Communication, Misc. Supplies, Etc.) = 0.5 % Of Total Direct Cost	$ 385,000
Haul Road Maintenance During Closure = 0.5 % Of Total Direct Cost	$ 385,000
Engineering Re-Design = 2 % Of Total Direct Cost	$ 1,540,000
Contingency = 8 % Of Total Direct Cost	$ 6,160,000
Total Indirect Cost [2]	**$ 8,470,000**
Annual Site Maintenance and Monitoring For 6 Years Post Closure	$ 2,310,000
Total Closure Cost	**$ 67,864,000**
Water Treatment System Facility/Component	
Active Water Treatment And Sludge Disposal System Construction	$ 4,169,000
Passive Water Treatment System #1, #2 & #3	$ 15,314,000
Total Direct Water Treatment Construction Cost	**$ 19,483,000**
Pre-Production Period (Years -2 and -1) Water Treatment O&M, Reagent and Pumping[3]	$ 5,545,000
Production Period (Years 1 through 15) Water Treatment O&M, Reagent and Pumping[3]	$ 6,125,000
Closure Period (Years 16 through 18) Water Treatment O&M, Reagent and Pumping[3]	$ 2,612,000
Post-Closure Production Period (Years 19 through 24) Water Treatment O&M, Reagent and Pumping[3]	$2,825,000
Total Mine-Life Water Treatment O&M, Reagent and Pumping[3]	**$ 17,107,000**
Total Mine-Life Water Treatment Costs	**$ 36,590,000**

1 Cost rounded to nearest $1,000 in current $.

2 Includes indirect costs associated with the construction of Water Treatment System

3 Includes Plant O& M, Lime, and Water and Sludge Pumping

5.4.10 Major Closure and Water Treatment Assumptions

The assumptions and estimated quantities (e.g., facility dimension, material/fluid volumes, surface areas, disturbance footprints) used for the development PFCP are provided as appropriate in the text, tables, and figures provided in Appendix J. A summary of the major assumptions is provided below. These assumptions should be verified as part of the feasibility study.

Closure

- Sufficient quantities of suitable clay or other low-permeability materials will be available within or immediately adjacent to Mt. Todd for the closure of the LGO1, LGO2, WRD, TSF1, TSF2, HLP, Process Plant Pad Area and other mine-related surface disturbance.
- Sufficient quantities of non-PAG waste rock will be selectively handled during mining so as to be available for the closure of the LGO1, LGO2, WRD, TSF1, TSF2, HLP, Process Plant Pad Area and other mine-related surface disturbance.
- Non-PAG waste rock in combination with clay will be suitable as a store and release cover material and a plant growth medium.
- Applying a 1m-thick cover of non-PAG waste rock on the impounded surface of the TSF1 and TSF2 is adequate to bridge thixotropic tailings to permit the installation of the 1m-thick store and release cover on the TSF1 and TSF2.
- The operational TSF1 and TSF2 spillways will require modification at closure to safely pass peak flows produced by the design storm events.
- The channel dimensions and rock armoring assumed for the closure stormwater management system are adequate to safely pass peak flows produced by the design storm event.
- Attaining final cut and fill slopes of a maximum overall slope gradient of approximately 3H:1V will be adequate to ensure long-term geotechnical stability.
- Vista will assume the responsibility to close the HLP.
- WAD cyanide levels in pore water and seepage from HLP are below maximum allowable concentration limits. Therefore, the HLP will not require rinsing or treatment with oxidants prior to grading and closure.
- The HLP will be reclaimed at the beginning of the production period and used to test closure design alternatives at Mt. Todd.
- Leached ore in the HLP will remain within the current HDPE-lined area following grading to attainment 3H:1V slopes. It may be necessary to pull the existing perimeter crest back towards the center of the HLP until 3H:1V slopes are attained.
- The HLP HDPE-liner is fully functional (and will remain this way for the foreseeable future) and devoid of significant leaks.
- Sediments removed from ponds during pond decontamination and closure activities, and HDPE piping from the removal of the tailings delivery and reclaim pipelines will be disposed of in an adequately designed and operated on-site disposal facility.
- Concrete foundations, walls and bridges and other non-reactive, non-combustible, non-corrosive and non-hazardous demolition waste will be broken up and either:
    - Placed in the WRD; and/or

- Buried in-place or backfilled against cutbanks and highwalls throughout the Process Plant and Pad Area, as well as other areas that will be reclaimed at Mt. Todd.

- Sufficient quantities of adequately-sized durable, non-slakeable and angular rock are available on or immediately adjacent to Mt. Todd to produce rip rap on-site for the armoring surface drainage channels and the construction of foundation drains.
- The equipment fleet used for mining will be used for closure.
- Process Plant demolition cost estimates include the following assumptions:
  - Salvage value will equal the removal cost for all Process Plant equipment and prefabricated items.
  - All structural steel and building skeletons will be disassembled/cut and removed and sold as scrap.
  - Steel stockpiled, along with pre-fabricated items, will be transported for salvage to a central location(s) at Mt. Todd.
  - Explosive/implosive demolition of concrete slabs and footers will be conducted after all steel infrastructure is removed.

Mine-Life Water Treatment

- Vista will obtain approval from the NT Government to permit effluent releases (that comply with the WDL and water quality-based effluent standards established for the Edith River as currently approved) from the Existing WTP and New WTP to Batman Creek.
- Numeric standards for sulfate, arsenic and other oxyanion will not be applied by the NT Government to the WDL or water quality-based effluent standards for the Edith River.
- Vista will construct run-on diversion(s) to achieve, at a minimum, the performance criteria as follows:
  - Divert approximately 70 percent of the surface runoff from the RP7 catchment area between planning years -2 and -1;
  - Divert approximately 29 percent of the surface runoff from the RP1 catchment area between planning years -2 and -1; and
  - Divert the majority of the surface runoff that would report to the WRD that lies outside (west) of the existing RP1 catchment area.
- By June Planning Year -2 Vista will commission the following facilities:
  - New WTP with a minimum ARD/ML treatment capacity of approximate 1000 m^3/hour;
  - LLDPE-lined (or equivalent) equalization pond;
  - LLDPE-lined (or equivalent) sludge disposal cell for the dispose of water treatment sludge produced by the New WTP;
  - Two pumps at RP2 (total capacity approximately 600m^3/hr);
  - Two pumps at RP3 (total capacity approximately 600m^3/hr); and
  - 1 Pumps at RP5 (total capacity approximately 300m^3/hr).

- Following the production phase the Batman Pit will be a passively-maintained hydrologic sink. Therefore, active pit dewatering and treatment of pit water will be unnecessary following the closure phase.
- Groundwater inflows to and outflows from the Batman Pit are insignificant.
- The TSF1 and TSF2 is a closed hydrologic system.
- TSF1 and TSF2 foundation drains will be operational and maintained during closure and post-closure.
- Operation of the Process Plant during the production period will use all excess water from the TSF1 and TSF2. If necessary, ARD/ML collected for treatment during the production phase will be used for the operation of the Process Plant.
- Adverse impacts to ground water from previous and planned mining and processing activities at Mt. Todd do not and will not occur. As such groundwater remediation due to ARD/ML contamination from Mt. Todd is not necessary.
- The Process Plant and Pad Area, LGO1, and LGO2 will no longer be a source of ARD/ML immediately following closure.
- RP2 and RP5 and collection systems at the toe of LGO2 will be maintained until no longer need for site-wide management of ARD/ML.
- The lime-precipitation treatment of the ARD/ML generated at Mt. Todd in a properly designed water treatment plant is adequate to meet currently approved WDL standards.
- Lime with a neutralizing efficiency equal to 90 percent will be produced at or adjacent to Mt. Todd in quantities sufficient to meet mine-life water treatment requirements (Approximately 17,000 tonnes of hydrated lime with a neutralizing efficiency equal to 90 percent).
- Existing pumps and Existing WTP inflow pipes with upgrades, and the existing lime silo and utility installations and offices on-site will be adequate for the New WTP.
- Sludge solids approximately equal to 50 percent slurry density.
- Passive water treatment systems #1, #2, and #3 (most likely anaerobic wetlands or Successive Alkalinity Producing Systems - SAPS) may be activated and effective at meeting WDL or water quality-based effluent standards for the Edith River when the total post-closure ARD/ML flows directed to each treatment system is approximately ≤ 120 m^3/hour.
- The passive water treatment systems will treat ARD/ML to 0.0 mg/L acidity and treatment efficiency will be ≥ 16.4 g acidity/m^2/day.

6.0 HISTORY

The Mt. Todd Project area has significant gold deposits located on it and is located 250 km southeast of Darwin in the NT of Australia. 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, 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. 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 designed to recover 90,000 ounces per year on an annualized basis over a life of 4 years. This came on stream in late 1993. The treatment rate was subsequently expanded to a rate of 6 million tonnes per year on an annualized basis in late 1994.

A comparison of actual and predicted production figures is printed in TABLE 6-1.

TABLE 6-1: HEAP LEACH – FEASIBILITY ESTIMATES VS. ACTUAL PRODUCTION
VISTA GOLD CORP. – MT TODD GOLD PROJECT
June 2009

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

Note: All tonnages and grades shown in TABLE 6-1 are historical numbers and are not NI43-101 compliant.

Phase II involved expanding to 8 Mtpy and treatment through a flotation and CIL circuit. The feasibility study was conducted by a joint venture between Bateman Kinhill and Kilborne (BKK) and was completed in June 1995. The feasibility study indicated that treatment of transitional and primary ore from the Batman pit would provide an 8-year mine life to recover 2 million ounces at a cost of AUD$369 (US$266) per ounce. Capital cost for Phase II was estimated at AUD$207.8 million.

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 $400 in early 1996 to below $300 per ounce during 1997. According to the 1997 Pegasus Gold Inc. 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 agreed to form a joint venture with Multiplex Resources and Pegasus Gold Australia to own, operate, and explore the mine. Initial equity participation in the joint venture was General Gold two percent, Multiplex Resources 93 percent, and Pegasus Gold Australia five percent. The joint venture appointed General Gold as mine operator, which contributed the operating plan in exchange for a 50 percent 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. 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 drill holes were drilled at Quigleys Reef. 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 drill hole program, with an aggregate meterage of 676.5 m, to test the gold content of Quigleys Reef over a strike length of 800 m. Following this program CRAE did not proceed with further exploration.

During late 1986, Pacific Gold Mines NL 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 CIP plant owned by Pacific 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 gold **(Historic reported quantity, not NI43-101 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
VISTA GOLD CORP. – MT TODD GOLD PROJECT
June 2009

1986 October 1986 – January 1987:	Conceptual Studies, Australia Gold PTY LTD (Billiton); Regional Screening; (Higgins), Ground Acquisition by Zapopan N.L.
1987 February: June-July: October:	Joint Venture finalized between Zapopan and Billiton. Geological Reconnaissance, Regional BCL, stream sediment sampling. Follow-up BCL stream sediment sampling, rock chip sampling and geological mapping (Geonorth)
1988 Feb-March: March-April: May: May-June: July: July-Dec:	Data reassessment (Truelove) Gridding, BCL grid soil sampling, grid based rock chip sampling and geological mapping (Truelove) Percussion drilling Batman (Truelove) - (BP1-17, 1475m percussion) Follow-up BCL soil and rock chip sampling (Ruxton, Mackay) Percussion drilling Robin (Truelove, Mackay) - RP1-14, (1584m percussion) 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: June: July-Dec:	Batman diamond and RC drilling:BD72-85 (5060m diamond); BP101-208, (8072m RC). Penguin, Regatta, Golf, Tollis Reef Exploration Drilling: PP1-8, PD1, RGP132, GP1-8, BP108, TP1-7 (202m diamond, 3090m RC); TR1-159 (501m RAB). Mining lease application (MLA's 1070, 1071) lodged. Resource Estimates; mining-related studies; Batman EM-drilling: BD12, BD8690 (1375m diamond); RC pre-collars and H/W drilling, BP209-220 (1320m RC); Exploration EM and exploration drilling: Tollis, Quigleys, TP9, TD1, QP1-3, QD1-4 (1141 diamond, 278m RC); Negative Exploration Tailings Dam: E1-16 (318m RC); DR1-144 (701. RAB) (Kenny, Wegmann, Fuccenecco, Gibbs).
1990 Jan-March:	Pre-feasibility 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	Pegasus Gold Australia Pty Ltd reported investing more than $200 million in

Australia Pty Ltd.	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, 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	Ferrier Hodgson (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 concession rights from the Deed Administrators.

6.2 Historic Drilling

The following discussion centers on the historic drill hole 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 drill-hole databases used as the bases of this report contain all of the available data. Tetra Tech is unaware of any drill hole data that have been excluded from this report.

6.2.1 Batman Deposit

There are 730 historic drill holes in the Batman Deposit assay database. FIGURE 6-1 shows the drill hole locations for the Batman Deposit. These holes 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 percent. The Central area of the deposit was extensively core-drilled. Outside of the Central area, most of the drill holes were RVC and OP holes. All drill holes collars were surveyed by the mine surveyor. Down-hole surveys were conducted on most drill holes using an Eastman single shot instrument. All holes were logged on site.

A series of vertical RVC infill holes were drilled on a 25-meter-by-12.5-meter grid in the core of the deposit to depths between 50 and 85 m below the surface. Zapopan elected to exclude these holes from modeling the Batman Deposit because the assays from these holes seemed to be downwardly biased and more erratic compared to assays from inclined RVC holes. Of the possible reasons cited as to why vertical RVC holes might report lower grades and have a more erratic character, the 1992 Mining & Resource Technology Pty Ltd ("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 blastholes 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 problem of drill hole density within the Batman Deposit. According to Pegasus management, the decision to not drill out the lower portion of the Batman Deposit was based on economic considerations. Section 7.0 of the 1995 BKK feasibility study detailed the decrease in drill hole density with depth. At the time of that study, there were 593 holes in the assay database of which 531 were used in the construction of the MRT block model. Reserve Services Group ("RSG") reported that the drilling density in the Central area oxide and transition zone ore was generally 25 m by 25 m. The spacing was wider on the periphery of the ore envelope. The drilling density in the Central area of the primary ore ranged from 50 m by 50 m, but decreased to 50 m by 100 m and greater at depth.

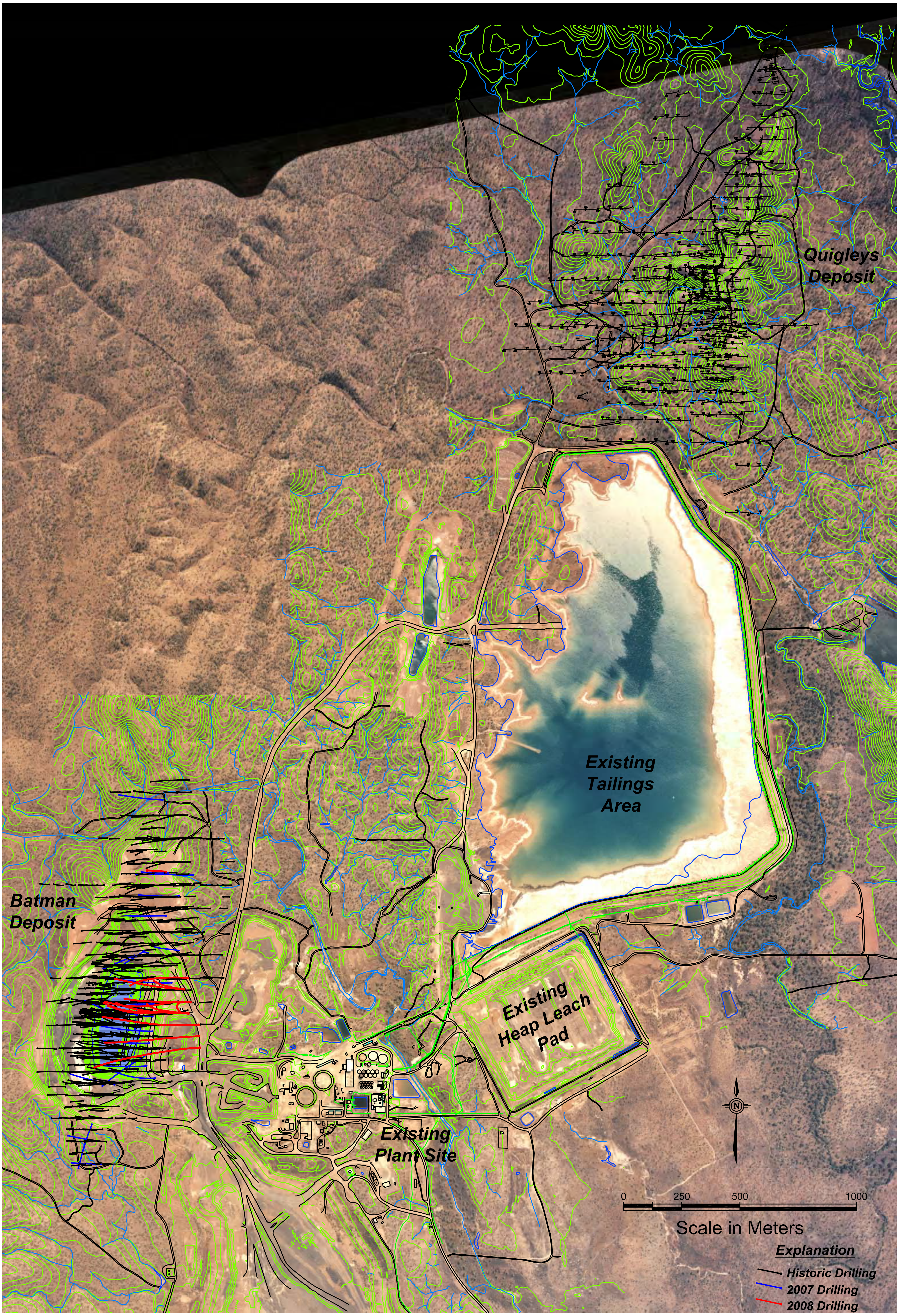
At the time of The Winters Company's ("TWC") site visit in 1997, the drill hole database numbered 730 holes. It is not known if any holes were excluded from the Pegasus exploration models. Most of the new drilling that had been added since the 1994 MRT model was relatively shallow. TWC reviewed PGA's 50-meter drill sections through the Batman Deposit and saw that there was a marked decrease in drill hole spacing below 1000 RL (the model has had constant 1000 m added to it in order to prevent elevations below 0 (sea level) and have been denoted as RL for relative elevation) and another sharp break below 900 RL. The drill hole spacing in the south of 1000 N on the 954 RL bench plan approached 80 m by 80 m. Pegasus was able to get around 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 drill holes. Most of the holes in the assay database are inclined to the west to capture the vein set which strikes N10° to 20°E, dips east, and which dominates the mineralized envelope. This orientation is the obvious choice to most geologists since these veins are by far the most abundant. Ormsby (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 drill hole database for the 1994 MRT model because their assay results appeared to be too low compared to other hole orientations. If vertical hole orientations were actually underestimating the gold content during exploration drilling, the vertical and often wet blastholes, 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 drill hole locations for the Quigleys Deposit.



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: Fig6-1.dwg	**Figure 6-1**
	Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a	**Drillhole Location Map**
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	**Batman & Quigleys Deposits**

TABLE 6-3: SUMMARY OF QUIGLEYS EXPLORATION DATABASE VISTA GOLD CORP. – MT TODD GOLD PROJECT June 2009			
Drillholes	**Gold Assays (approx 1m)**	**Copper Assays (approx 1m)**	**Lithologic Codes**
632	49,178	41,673	51,205

Snowden completed a statistical study of the Quigleys drill hole 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.

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 holes had averaged assays five percent to six percent higher than RVC holes; for that reason, MRT elected to exclude RVC holes from the drill hole database for grade estimation of the Central area of the Batman Deposit.

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 GGC, 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 holes. The procedure involved sawing the NQ core lengthwise in half. HQ core was quartered. RVC samples were riffle split on site and a 3- to 4-kg sample was sent to the laboratory for analyses. PAH stated that they actually witnessed the sample preparation process at a number of steps and concured with the methods in use; however, PAH also noted that they would prefer that the sample cuts following the ring grinding process be conducted with a splitter rather than a scoop. While free gold is not a problem in this deposit, the potential does exist for segregation based upon particle hardness, which could bias assay results.

Pegasus (and Zapopan NL, 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.

Billiton conducted an audit/analysis of the data set available in 1992, which resulted in a number of recommendations. Generally, factoring of any kind, particularly upward, can be a source of problems and is not recommended practice. The four percent adjustment applied to a portion of the pre-1989 data set is unlikely to introduce a significant problem. Similarly, averages of multiple samples were placed into the assay field designated AU_PREF, which is also a potential source of error, as it creates a set of samples whose variance will be somewhat lower than the single-assay population. Again, the number of samples subjected to averaging is less than one in ten, so the net effects are negligible.

While the concerns mentioned thus far are relatively minor, It was PAH's feeling that a more detailed examination of the assay set would be in order. The first concern focused on the integrity of the AU_PREF assays, which were calculated from a number of methods depending upon date drilled and the existence of check assays. PAH ran regressions and correlations on AU_PREF against the primary and repeat assays of the Batman Deposit and noted that their data set contained 39 percent more samples than the feasibility dataset, most of which have been prepared under the more stringent and repeatable guidelines as specified by Pegasus and others.

The results indicated that at higher grades, the AU_PREF assay differed by less than one percent (on average) from the primary and repeat assays. Agreement with the primary assay was within one percent over the entire range, which, indicates that AU_PREF, even with the averaged data, does not materially differ from the source assays. The average difference between the regressed grade and AU_PREF becomes larger at lower grades, particularly at less than 0.5 g au/t. This effect is probably due to detectability differences between the different labs and the mathematical effect of even small differences on low-grade samples.

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 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 meter. The minimum sample length is 0.1 m and the maximum sample length is 5 m. 137 samples are less than 1 meter and 65 samples are over one meter 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 hole were recorded.

6.4.2 Check Assays

Extensive check assaying was carried out on the exploration data. Approximately five percent of all RVC rejects were sent as duplicates and duplicate pulps were analyzed for 2.5 percent 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 GGC based on recommendations of consultants. It is Tetra Tech's opinion that the assay database used in the creation of the current independent resource estimation exercise is acceptable and meets industry standards for accuracy and reliability.

6.4.3 Security

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 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 Gold Australia Pty Ltd. 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 designed 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 percent 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 percent 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 percent 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 percent 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 percent 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.

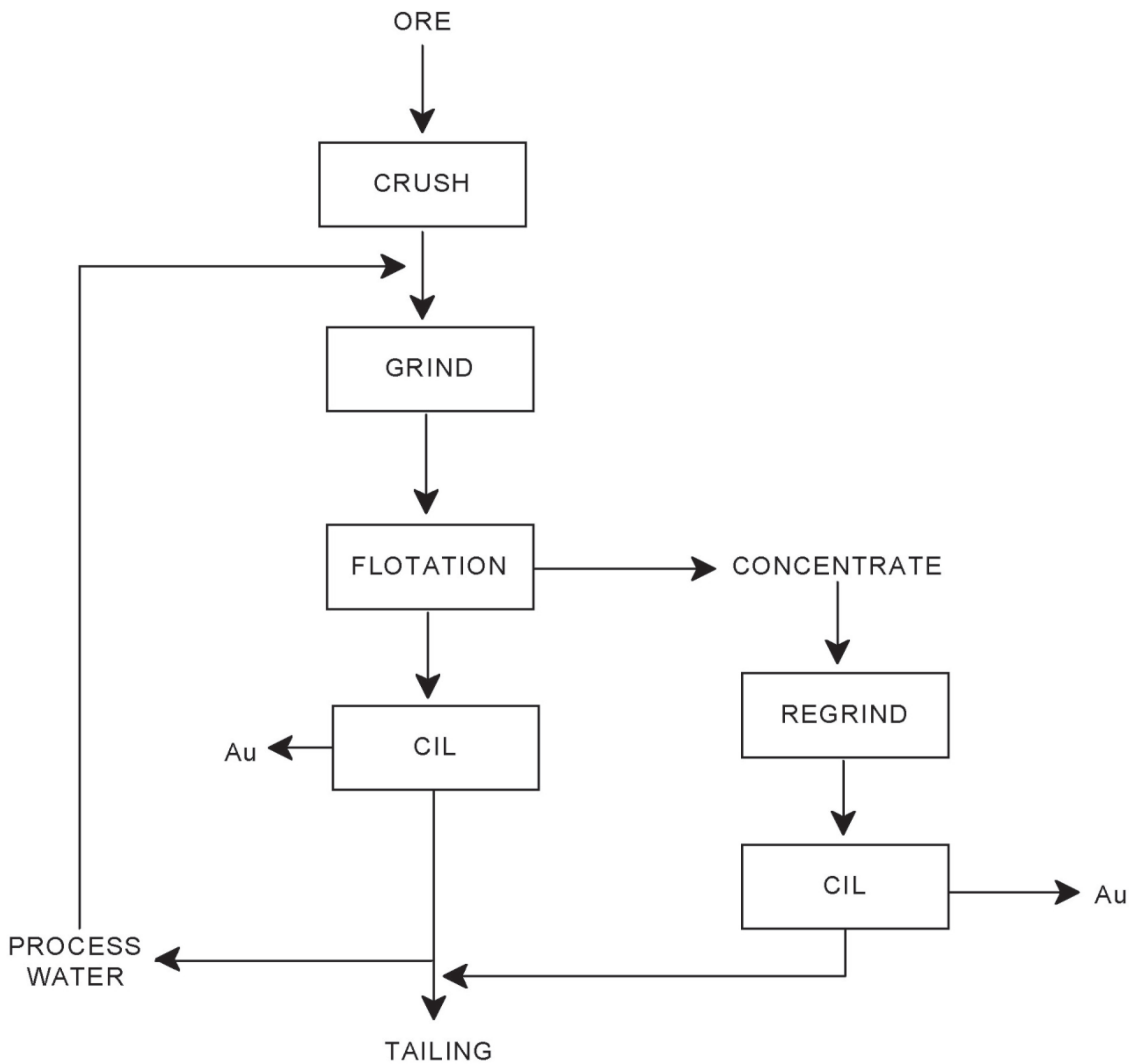
6.6 Technical Problems with Historical Process Flowsheet

Besides the collapse in the gold price, there were several technical problems with the design flowsheet. These technical problems have been documented by plant engineers, The Winters Company, 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, the tonnage was projected to be 8 million tonnes per year 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 million tonnes per year 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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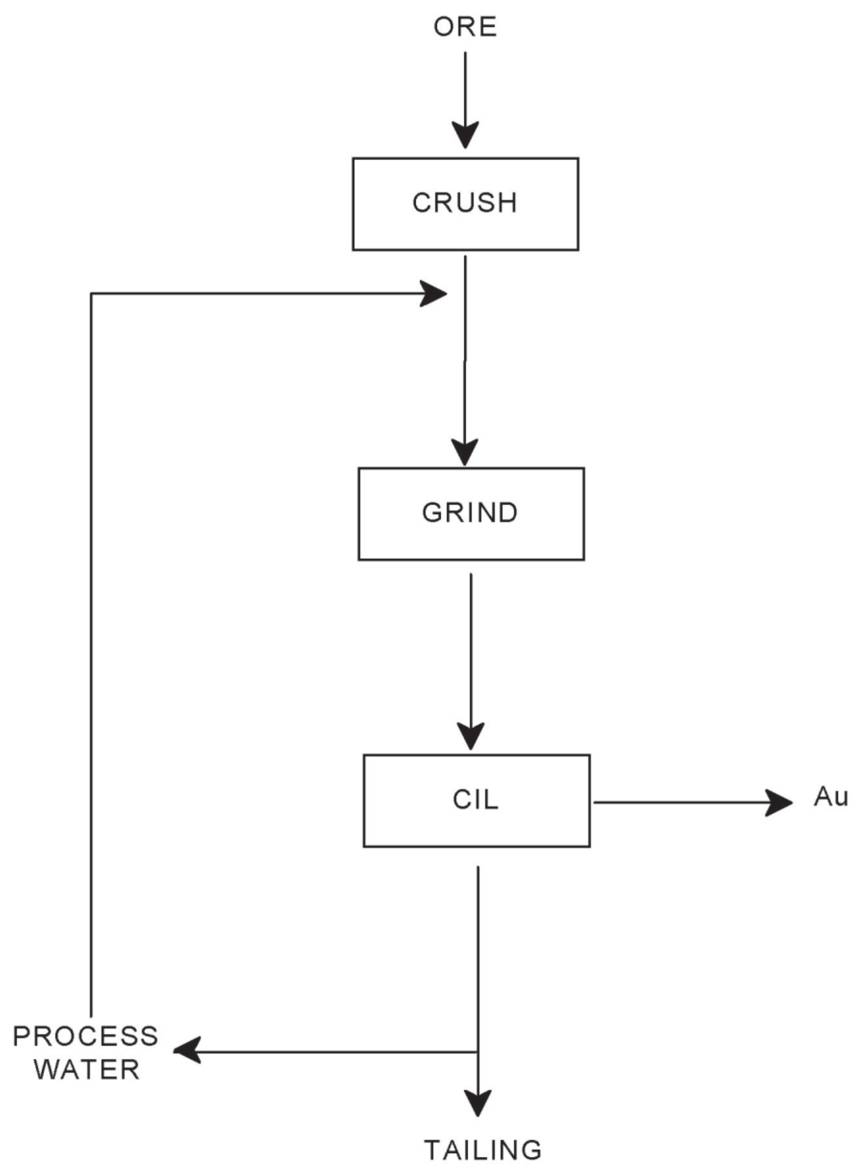
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Vista Gold Corp.***	Fig6-2.dwg	**Figure 6-2** **Plant Process Flowsheet for Mt. Todd Project (as designed)**
	Project: Mt. Todd Gold Project	Project Number: 114-310829	
	Project Location: Northern Territory, Australia	Date of Issue: 03/05/2008	



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.*	Fig6-3.dwg	**Figure 6-3** **Modified Plant Process Flowsheet for Mt. Todd Project**
	Project: Mt. Todd Gold Project	Project Number: 114-310829	
	Project Location: Northern Territory, Australia	Date of Issue: 03/05/2008	

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.

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

6.6.2 Grinding Circuit

The SAG mill/ ball mill / crusher (ABC circuit) would have been a better selection of the comminution circuit rather than the four-stage crushing/ball milling circuit. The circuit was tested, but not implemented in the final flowsheet for reasons discussed in the previous section.

6.6.3 Flotation Circuit

The flotation circuit was supposed to recover 60 to 70 percent of the gold in a bulk sulfide concentrate which was seven percent of the feed material. The flotation circuit recovered ± 1% of the weight of material and less than 50 percent 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 percent of copper recovery at a concentrate grade of +10% Cu. Approximately 45 percent 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 percent was achieved in some of the later tests.

6.6.4 CIL of Flotation Concentrate and Tailings

A portion of the copper was depressed with cyanide with the recycle 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 percent of gold recovery in the circuit.

7.0 GEOLOGICAL SETTING

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^{o}, dipping at 40 to 60^{o} 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
VISTA GOLD CORP. – MT TODD GOLD PROJECT
June 2009

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

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: Fig7-1.cdr	**Figure 7-1**
	Project: Mt. Todd PreFeasibility Study	Project Number: 114-310912a	**General Geologic Map**
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	

8.0 DEPOSIT TYPE

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 chanalization 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 chanalization 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 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 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.

9.1 Batman Deposit

9.1.1 Local Mineralization Controls

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

Sulfide minerals associated with the gold mineralization are pyrite, pyrrhotite and lesser amounts of 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.

9.1.2 North-South Trending Corridor

The north-south trending mineralization occurs in all rock units and is most dominant in the shales and greywackes designated SHGW23. Inspection of grade control and exploration data, drill logs, diamond core and the pit has shown that the north-south trending mineralization can be divided into 3 major zones based on veining and jointing intensity.

9.1.3 Core Complex

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

9.1.4 Hanging Wall Zone

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

9.1.5 Footwall Zone

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

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

9.1.6 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.

9.2 Quigleys Deposit

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

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 mineralisation with depth. Some adjacent holes were also noted with significant variation in the interpreted position of the shear zone, and some of the discrepancies appeared to have been resolved on the basis of selection of the highest gold grade. While the above method may result in a valid starting point for geological interpretation, the selection of such a narrow high grade zone is overly restrictive for interpretation of mineralization continuity and will require additional work prior to estimating any resources.

It was further thought that while the shear might be readily identified in diamond drill holes, 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.

10.0 EXPLORATION

Vista exploration staff conducted a surface exploration program, including prospecting, rock sampling and GPS surveying of drill hole collars and grid pickets on the Mt. Todd Exploration Licenses from April to July, 2008. Equipment and personnel were mobilized from the Mt. Todd Mine site. The work was conducted by geologists and field technicians.

During the 2008 field season, the exploration effort was focused on four areas: Red Kangaroo Dreaming ("RKD"), Mt. Todd mine site area, Tablelands area and Wolfram Hill. All prospects can be accessed from the Mt. Todd mine site easily via existing roads. A total of 216 rock samples were collected from all areas as presented in Table 10-1. These prospect areas were chosen for further exploration as they were along strike (or proximal) of a mineralized northeast regional trend which hosts the Batman Pit and numerous gold prospects.

TABLE 10-1: 2008 ROCK SAMPLES
VISTA GOLD CORP. – MT TODD GOLD PROJECT
June 2009

Prospect	Samples Collected
Red Kangaroo Dreaming (RKD)	145
Mt. Todd Mine Site Area	52
Tablelands Area	6
Wolfram Hill Area	13
Total Samples	**216**

RKD was explored by the previous operator (Pegasus: 38 RC holes, 58 RAB holes). Mineralization was defined along a south trending 575 meter strike length. The area sampled during the 2008 program is west and south of the main RKD mineralized zone. The rock sampling was conducted to confirm both historical gold anomalies and soil anomalies from the 2007 Vista soil sampling program. At RKD, 145 samples were collected and submitted for analysis.

Prospecting and rock sampling was conducted at the Mt. Todd mine site to locate mineralization proximal to Batman pit. Approximately 52 samples were collected and submitted for analysis. The area sampled includes the area south of the waste dump and heap leach pad. The sampled area contains historical soil and rock chip Au anomalies that have seen limited exploration.

In the Wolfram Hill area, 13 samples were collected and submitted for analysis. There are numerous historical gold anomalies in the Wolfram Hill area that have seen limited exploration. The area that was sampled includes historical shafts and adits from previous tungsten mining operations.

Limited sampling at Tablelands area, 33 km northeast of the Batman pit (14 km northeast of RKD), comprised only six samples. Previous drilling by past operators returned a near surface assay of 36 g/T Au as well as other anomalous values.

All observations and sampling are recorded as "stations" which have UTM coordinates that are located in the field with a GPS unit.

An ICP multi-element suite was utilized to analyze the rock samples from RKD, Mt. Todd mine site area, Tablelands area and Wolfram Hill prospect by ALS Chemex Labs in Adelaide, South

Australia. The ICP analysis consist of a multi-element suite that reports analyses for base and precious metals, pathfinder elements for these commodities, as well as elements useful for mapping bedrock geology.

Concurrent with the rock sampling, from April to July 2008, drill hole collar locations and grid pickets were surveyed at Tablelands prospects using a GPS unit. Accurate drill hole locations has enabled the compilation of an accurate database for further drill planning and geological interpretation.

10.1 Results

Approximately 1100 m due west of the RKD prospect, a 600 meter long arsenic soil anomaly was prospected and sampled during the 2008 exploration program. Historical rock samples have assayed up to 17.37 g Au/t within the anomaly. During the program, a topographic ridge corresponding within the southern portion of the anomaly was explored. The ridge was sampled along 500 m with 41 samples collected. Of the samples collected almost half (46 percent) were over 0.3 g Au/t (ranging from 0.3 to 2.36 Au/t). No known drilling has been conducted on the anomaly and the mineralized ridge, although historical drill holes are collared 500 m west and 200 m south of the current target. Further field work is recommended including mapping, rock sampling and further soil sampling to define the anomaly and develop a drill target.

At the Wolfram Hill prospect, the 2008 rock sampling located anomalous gold, silver, copper, and tungsten anomalies including one sample which assayed 2.33 g Au/t, 738 g Ag/t, 37.8 %Cu and 0.21 %W. Only preliminary work was conducted in 2008; further work is warranted due to the significant gold, silver and copper values that were delineated in 2008 and by previous operators. It should also be noted that other historic tungsten occurrences, similar to the Wolfram Hill prospect, in the Pine Creek Orogen, also have significant enrichment of tantalum (it is currently unclear if the Wolfram Hill prospect has been explored for or historic samples have been analyzed for tantalum). Tantalum mineralization is present in a number of deposit styles including pegmatites and polymetallic veins of which both are found at the Wolfram Hill prospect.

Preliminary reconnaissance exploration was completed at the Tablelands prospect and additional work is recommended to follow up anomalous gold mineralization identified by previous operators.

South of the waste dump at the Mt. Todd mine site, a spot gold anomaly of 1.2 g Au/t confirms historical gold anomalies of 1.99 to 14.2 g Au/t. All three samples occur along a 200 meter strike length which trends north-south. The area sampled south of the heap leach pad also had isolated spot gold anomalies up to 2.29 g Au/t. Further work is required and recommended to locate and further refine known areas of gold mineralization proximal to the Mt. Todd mine site.

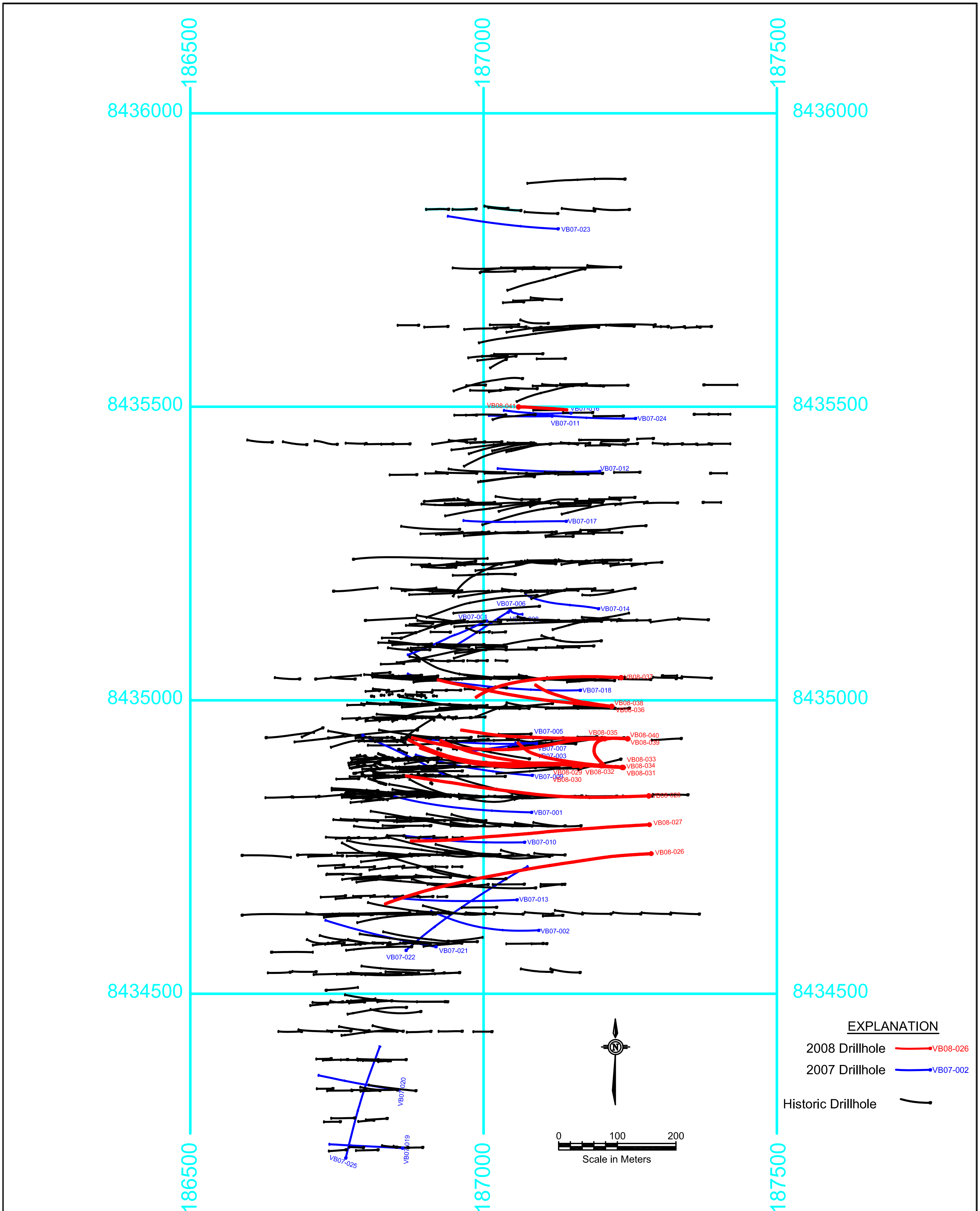
11.0 DRILLING

The 2008 Vista exploration program at the Batman deposit consisted of 16 diamond core drill holes containing some 9,037.4 m that targeted both infill definitional drilling and stepout drilling. TABLE 11-1 contains information of the 16 drill holes completed. A total of 7,367 assays were submitted from the program to the ALS Chemex for analyses. Core holes VB08-029 and VB08-033 were terminated early due to poor ground conditions.

TABLE 11-1: 2008 EXPLORATION DRILLHOLE SUMMARY
VISTA GOLD CORP. – MT TODD GOLD PROJECT
June 2009

Hole ID	Northing	Easting	Elevation (m above msl)	Bearing (degrees)	Dip (degrees)	Total Depth (m)
VB08-026	8434739.0	187386.1	144.9	267.2	49.2	700.5
VB08-027	8434788.0	187282.8	146.0	266.6	51.7	661.3
VB08-028	8434837.0	187282.0	146.4	268.1	52.9	647.8
VB08-029	8434888.0	187166.0	146.0	266.3	59.1	26.8
VB08-030	8434890.0	187165.9	146.3	275.1	59.6	599.1
VB08-031	8434886.0	187236.4	146.3	273.0	60.6	640.6
VB08-032	8434888.0	187201.0	146.4	273.0	58.2	632.7
VB08-033	8434886.0	187237.0	146.3	278.2	72.7	42.0
VB08-034	8434886.0	187238.1	146.3	274.7	73.2	750.0
VB08-035	8434934.0	187206.5	141.8	268.6	59.8	678.0
VB08-036	8434990.0	187218.3	143.3	274.1	60.0	657.1
VB08-037	8435039.0	187234.6	153.2	272.5	60.5	655.1
VB08-038	8434990.0	187218.7	143.3	278.3	76.3	730.7
VB08-039	8434934.0	187245.4	147.3	272.4	59.5	615.3
VB08-040	8434934.0	187246.1	147.3	274.7	73.7	700.0
VB08-041	8435500.0	187059.7	171.3	88.6	75.4	300.4

FIGURE 11-1 is a plan map that details the locations of the drill holes completed as part of the 2008 exploration program.



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TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Vista Gold Corp.*	Fig11-1.dwg	**Figure 11-1** **Drillhole Location Map** **Batman Deposit**
	Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	

12.0 SAMPLING METHOD AND APPROACH

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

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

13.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Vista and Tetra Tech developed an assay protocol for the analyses of the 2008 exploration drill core and for validation of the historic assays.

13.1 Sample Preparation

The diamond drilling program was conducted under the supervision of the Geologic Staff which was composed of a Chief Geologist, several contract expatriate geologists, and a core handling/cutting crew. The core handling crew was casual 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 1 meter. When this process was completed, the core was moved into the core cutting/storage area where it was lined out for sampling. The core was laid out for the following procedures:

- One-meter intervals were marked out on the core by a member of the geologic staff;
- Geotechnical logging was done in accordance with the instructions received from SRK;
- Geologic logging was then done by a member of the geologic staff. Assay intervals were selected at this time and a cut line marked on the core. The standard sample interval was one-meter. During the early part of the program some flexibility was allowed for portions of the core that were not expected to return significant values based on visual inspection. These portions of the core were sampled in two-meter intervals. This was discontinued when numerous > 1 ppm assays were received from the 2 meter intervals;
- Blind sample numbers were then assigned and sample tickets prepared. Duplicate sample tickets were placed in the core tray at the appropriate locations; and
- Each core tray was photographed and restacked on pallets pending sample cutting.

The core is then cut using diamond saws with each interval placed in marked plastic bags. At this time, the standards and blanks were also placed in plastic bags for inclusion in the shipment. When a sequence of 5 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 then placed in crates for shipping with 100 samples per crate (20 shipping bags). The crates were secured with padlocks and numbered globe seals as soon as they were loaded. The secured crates were stacked outside the core shed until picked up for transport.

13.2 Sample Analyses

After the samples were prepared, a split of the pulp was shipped directly to the ALS Chemex laboratory located in Perth for analysis.

ALS Chemex

31 Denninup Way
Malaga
Perth, Western Australia
Australia, 6090

The ALS Chemex sample preparation facility also prepared splits of the designated pulps and coarse rejects for cross laboratory checks. Genalysis was selected as the secondary laboratory to do the QA/QC checks. When a batch of samples had been prepared, the selected pulps and coarse rejects were shipped via TNT to the Genalysis sample preparation facility if Adelaide for the cross laboratory check work.

Genalysis

11 Senna Road
Wingfield South Australia 5013

ALS Chemex sent Vista an e-mail list of samples transmitted to Genalysis when they were shipped. When this notification was received by Vista, sample transmittals were prepared and e-mailed to Genalysis.

When the additional sample preparation work was completed, the Genalysis sample preparation facility in Adelaide shipped the pulps to their laboratory in Perth for the analytical work.

Genalysis

15 Davison Street
Maddington Western Australia 6109

13.3 Sample Security

ALS Chemex was selected as the primary laboratory for all further preparation and analysis. The closest ALS Chemex facility with the capability of preparing the samples to the desired specifications was their sample preparation facility located in Adelaide. A series of padlocks were purchased for the sample crates and keys to these padlocks were sent to the sample preparation facility. ALS Chemex was instructed to notify Vista immediately if a crate of samples arrived without the padlocks or if the globe seals were missing or showed evidence of tampering.

ALS Chemex

Unit 1, Burma Road
Pooraka
Adelaide, South Australia
Australia, 5095

Sample shipments were scheduled for approximately once a week. The sealed crates were picked up on site by the transport company for road transport to the preparation facility. A chain-of-custody note was prepared and signed by both the shipping company and the geologist supervising the loading. These con notes were attached to the sample inventory and filed in the geologist office on site.

When the shipment left site, sample transmittals were prepared and e-mailed to ALS Chemex. 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.

14.0 DATA VERIFICATION

14.1 Drill Core and Geologic Logs

As stated earlier in this report, the Mt. Todd Project has an excellent drill hole database comprised of drill core, photographs of the drill core, assay certificates and results, and geologic logs. The meticulous preservation of the drill core and associated "hard copies" of the data are a testament to the originators of the project and the subsequent companies that have looked at the project. All data are 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. Other than the "normal" types of errors inherent in a project this size, (i.e. mislabeled intervals, number transpositions, etc.), which were corrected prior to Tetra Tech's resource estimation, it is Tetra Tech's opinion that the databases and associated data are of a "high quality" in nature.

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

14.2 Topography

The topographic map of the project area was delivered electronically in an AutoCAD® compatible format and is dated December 1999. The surveyed drill hole collar coordinates agree well with the topographic map; it is Tetra Tech's opinion that the current topographic map is accurate and fairly represents the topography of the project area. In addition, it is suitable for the development of the geologic models, resource estimates, and potentially mineable resources.

14.3 Verification of Analytical Data

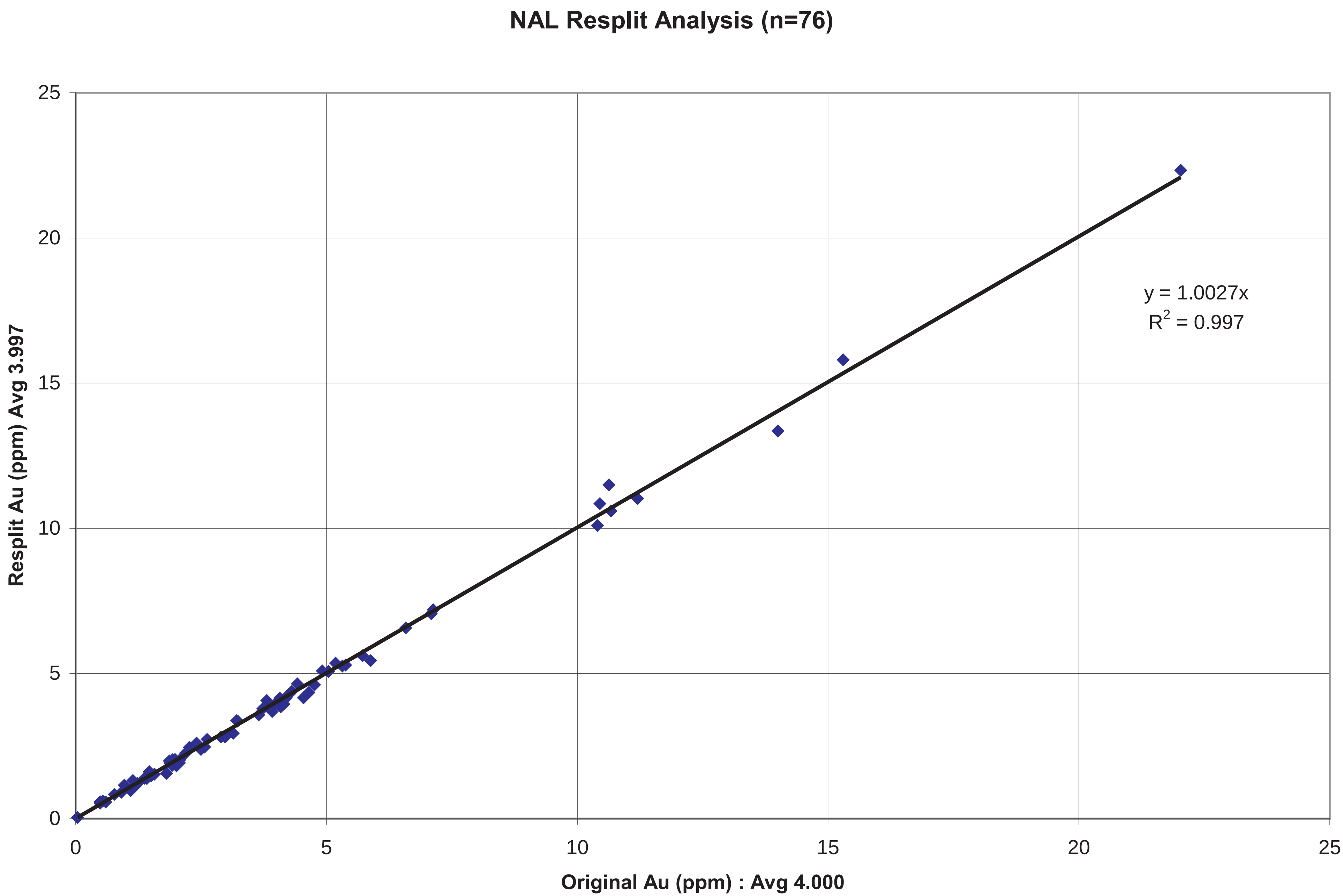
As part of the 2007 exploration program, Vista embarked on a program to both verify the historic assay results and ensure that any future analytical work meets all current NI 43-101 standards for reporting of mineral resources. This program consisted of two components; re-assaying of a portion of the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

Vista completed a multi-phase program to evaluate the accuracy of gold assays generated by North Australian Labs (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 1-meter 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. 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 14-1, 14-2, and 14-3 detail the results of the analytical check program that was completed on the 2007 exploration drill holes. The program was designed to check both internal laboratory accuracy and inter-laboratory accuracy. NAL was the primary laboratory for completion of the sample analyses. ALS Chemex in Sydney, Australia performed the inter-laboratory analyses. As can be seen from the plots, the correlation coefficient was 99.7 percent for the resplits of original assays, 99.2 percent for pulp repeats, and 98.6 percent for inter-laboratory analyses, respectively.

Vista continued their verification program as part of the 2008 exploration program.



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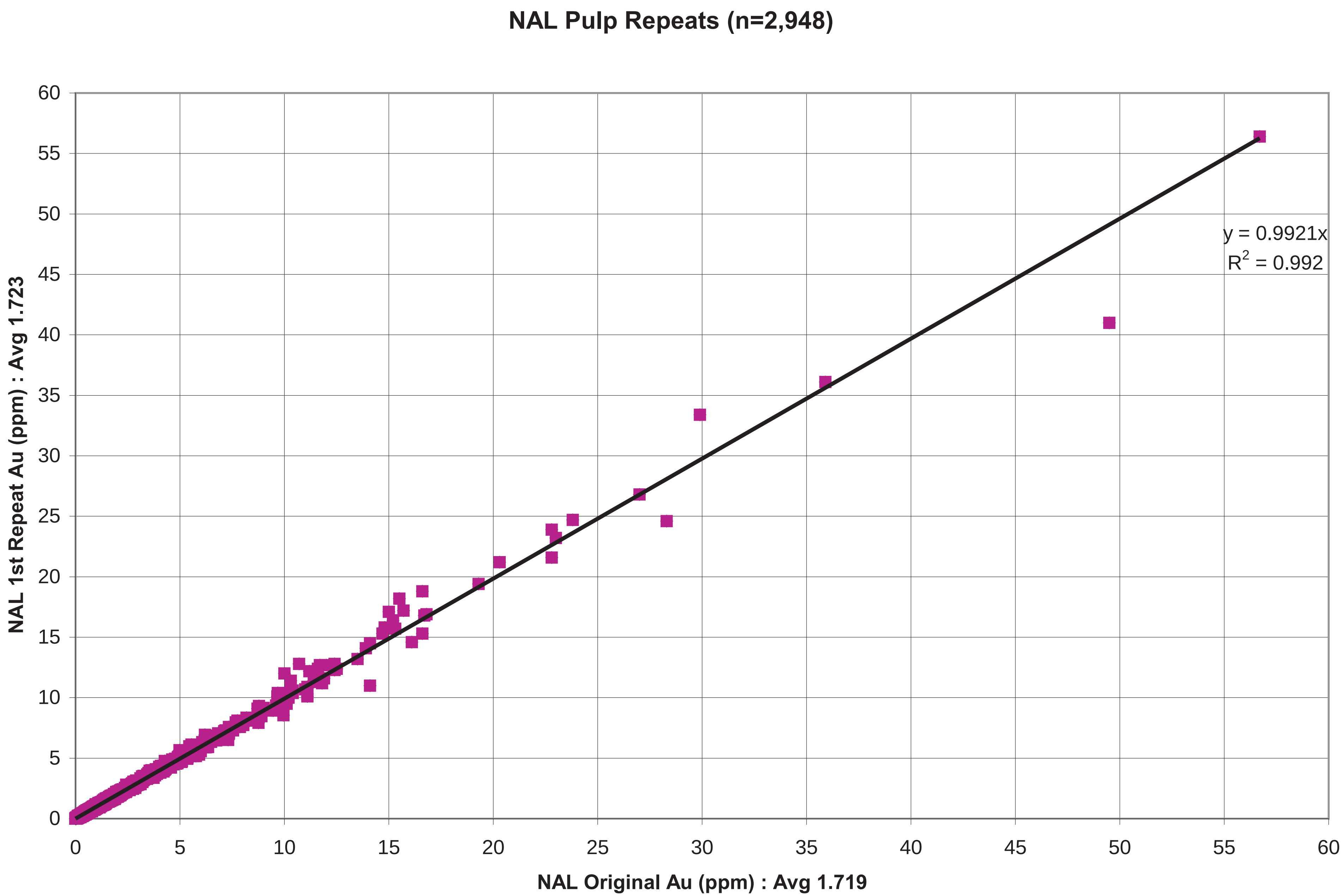
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Vista Gold Corp.***	Fig14-1.dwg	**Figure 14-1** **NAL Resplit Analysis (n=76)**
	Project: Mt. Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	



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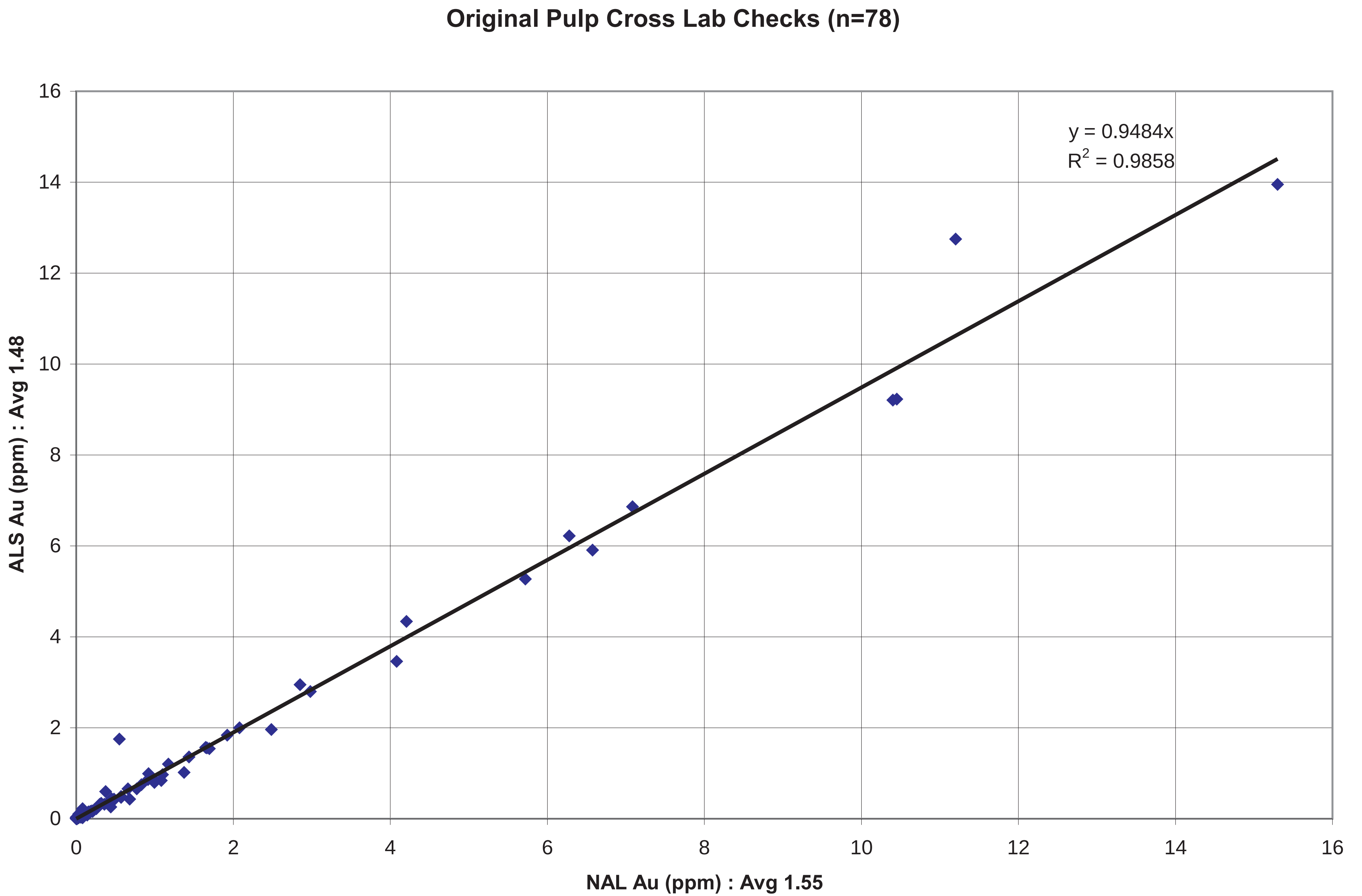
Project: Mt. Todd PreFeasibility Study	Project Number: 114-310912a
Project Location: Northern Territory, Australia	Date of Issue: Sept/2010

Figure 14-2
NAL Pulp Repeats (n=2,948)



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: Fig14-3.dwg	**Figure 14-3** **Original Pulp Cross Lab Checks (n=78)**
	Project: Mt. Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	

15.0 ADJACENT PROPERTIES

There are two major structural trends in the area (see FIGURE 15-1) that control most of the mineralization in the district. The northeast trending Cullen-Australus Corridor extends northeast and controls the deposits in the Pine Creek area including East Brilliant (Au), Saunders Rush (Au), Aston Hill (Au), etc. The Batman-Driffield trend within the tenements is northeast and is clearly defined by combined Landsat-Spot-aeromagnetic linear zones. There is a flexure in this trend around the Mountain View area that is associated with the Granitic Intrusive. The linear trends swing northwest in this area and define another mineralized linear zone linking Wandie-Moline and which is sub parallel to the Pine Creek linear.

Mineralization in the tenement blocks consists mainly of gold, tin, tungsten, with minor copper, lead, and zinc shows at Mountain View, Silver Spray, Tableland and Mt Diamond. Gold is usually associated with quartz veins and with chalcopyrite, arsenopyrite, pyrite, pyrrhotite and at Batman, minor bismuth and bismuthinite. At Batman, mineralization occurs as stockworks and sheeted quartz-sulfide veins. In other areas such as Quigleys, better grade mineralization is related to distinct shear zones that can have surrounding stockworks.

15.1 Yinberrie-EL 9733

Previous work defined two gold prospects. At Anomaly One, RC drilling by Billiton returned peak gold intercepts of 5 m of 2.93 g Au/t and 33 m of 1.21 g Au/t (including 6 m @ 2.54 g Au/t). Pegasus drill tested Anomaly One with 16 RC holes, for 1599 m on four sections between 10200N to 10700N. Intersections were from 2 to 8 m wide, grades from 1.05 to 3.14 g Au/t in strongly hornfelsed metasediments.

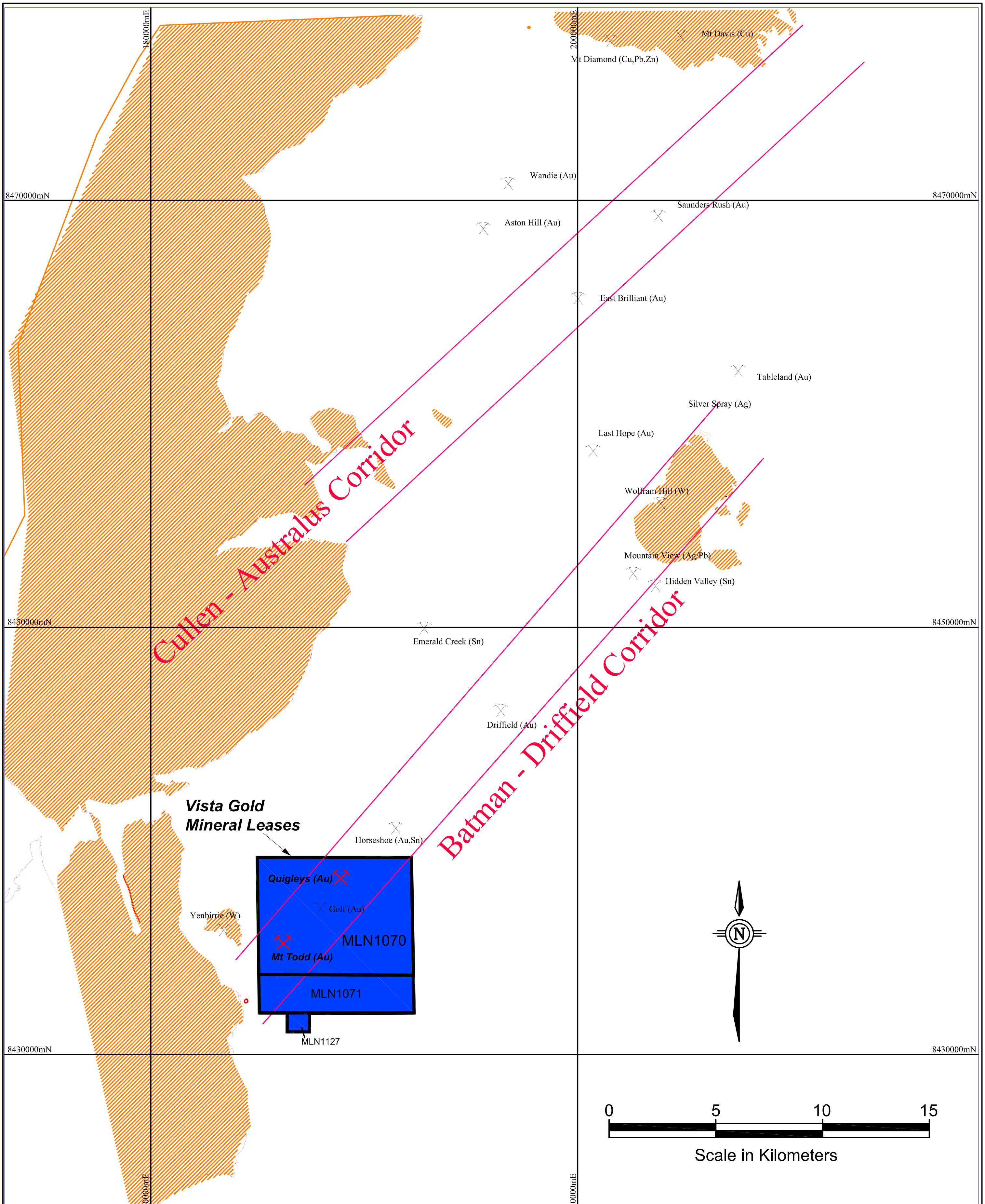
15.2 Horseshoe - EL 9735

This area was previously held as EL 7635 and Mineral Claims N1918 to N1923 and N3676 to N3683 (inclusive). Billiton work defined two significant gold anomalies: Central, at the northern end, now held under BJV tenement SEL9679, and Horseshoe at the south. At Central the best RC drill result was 9 m @ 4.2 g Au/t while 15 m @ 1.8 g Au/t gold at Horseshoe was drilled. The Pegasus work performed over 5 years downgraded the Central Prospect. RC drilling at Horseshoe, based on detailed mapping, indicates the prospect consists of a number of thin high-grade shears with minimal stockwork mineralization in foot and hanging wall.

15.3 Driffield-EL 9734

Previous mining at Driffield produced about 5,300toz of gold. Alluvial gold has also been worked on the EL and there are numerous small tin workings. Systematic exploration work carried out over previous years was collated, assessed and followed up. One diamond and sixty-six RC holes at six prospects were drilled by Pegasus for 4794 m at the Driffield Mining Center. Results indicated narrow lodes are only present. A further eleven RC holes were drilled at the Emerald Creek Prospect (670 m). No significant results were recorded.

Other prospects tested included Driffield North, Driffield West, Golden Slipper, and Driffield South. Results of five drill holes at Driffield North were disappointing. At Driffield West, nine RC holes were weakly anomalous, the best being DWRC 001 from 12 m, a length of 21 m @ 0.46 g Au/t; and from 45 m, 6 m @ 0.62 g Au/t. RAB drilling at Golden Slipper returned poor results and, while the bulk of rock chips at Driffield South were disappointing, some significant anomalies (+100 g Au/t) were recorded.



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Prepared for: *Vista Gold Corp.*	File Name: Fig15-1.dwg
Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a
Project Location: Northern Territory, Australia	Date of Issue: Sept/2010

Figure 15-1
Structural Trends with Mines & Prospects

While 1997 results failed to locate a significant deposit, exploration is incomplete and other anomalies remain to be evaluated and drill tested.

15.4 Barnjarn - SEL 9679

This tenement is a large block of ground (353 sub-blocks totaling 1,136 sq.km). Compilation of previous exploration data defined targets at Australis (flanks Mt Davis), Wandie/Saunders Rush/Brilliant, Everest, and Triple Bull. Further anomalies were defined at six other areas. Rock chip sampling by Pegasus at eight areas returned results from 0.76 to 24.3 g Au/t gold in fourteen samples. Soil sampling at nine prospects outlined anomalous zones. Preliminary RAB drilling was carried out at Everest, RKD extensions and GT prospects with inconclusive results. At RKD, 38 RC holes were drilled which intersected 1 to 4 m of mineralisation, grading between 1.3 and 14.3 g Au/t Au. An airborne magnetic survey at 100 m spacing at 60 m mean terrain clearance was flown, and GLS and remote sensing studies completed. A total of 65 anomalies were defined by geochemical and/or structural means. A small resource has been interpreted at RKD and drilling at Mountain View, Cullen and Highway was proposed.

15.5 Summary

The Mt. Todd region, and particularly the Batman style of mineralization, is one of sheeted veins that develop into a broad two-to-three dimensional stockwork. The grade of the > 200 million mineralized tonnes averages a little less than 1 g Au/t **(Historical Pegasus estimate, not NI43-101 compliant (circa 1997)),** and is associated with low grade copper, mostly as chalcopyrite.

At Cadia Hill in New South Wales, the mineralisation is similarly a sheeted vein, two to three dimensional stockwork grading around 0.9 g Au/t, associated with chalcopyrite grading < 0.2% copper. Exploration at Cadia was vigorously prosecuted and extremely persistent in testing of deeper combined magnetic/geochemical anomalies. This ultimately resulted in discovery, at depth, of the Ridgeway deposit (over 26 million tonnes at > 3 g Au/t and > 1% copper) **(Historical estimate, not NI43-101 compliant**).

Ridgeway is hosted by rocks similar to Cadia Hill, but there is a distinct increase in the quantity of mineralising fluid. Quartz veining with chalcopyrite-gold mineralization increases very significantly in proportion to the hosting altered, but unmineralized granitioid. It indicates an area of more forceful injection of fluids and an area of greater structural preparation. The Mt. Todd region has a large endowment of gold.

Whatever the source of the fluids that caused the Mt. Todd mineralization, it is the view of others that there is a high probability that somewhere in the ground currently under lease, may be a far more significant moderate to high grade economic deposit.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

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 Gold Corp. (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 deposit 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 communition 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.

16.1 Historical Review of Conceptual Process Flowsheet

RDi reviewed historical metallurgical testwork for the Mt. Todd 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 percent 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 percent Cu. The concentrate would contain approximately 50 percent 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 Gold. 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/t Au, 448 ppm Cu, and 1.43 percent S_{Total}. Based on sequential copper analyses, the copper present in the composite consisted of three percent oxide copper, 63 percent secondary copper and 34 percent 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 percent of gold and 90 percent 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 percent 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 percent of the gold extracted 84 percent of the gold in the tailing. The limited open-circuit testwork indicated that the proposed conceptual process flowsheet should work for the deposit.

16.2 Metallurgical Testwork

Vista Gold conducted the first of the two 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/t Au, 447 ppm Cu and 0.92 percent S_{Total}. The metallurgical testwork indicated that gold recovery into the rougher flotation concentrate was ± 80 percent at a primary grind of P_{80} of 200 mesh. Copper in the rougher concentrate could not be upgraded to provide concentrate assaying ± 20 percent Cu. The best results were ± 6 percent 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/t Au 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, summarized in TABLE 16-1, indicated that historical core had copper predominantly as secondary copper which is known to be a major consumer of cyanide. 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 percent 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 16-1: ASSAYS OF VARIOUS COMPOSITE SAMPLES
VISTA GOLD CORP. – MT TODD GOLD PROJECT
June 2009

Parameter	Historical Core	2007 Drilling	2008 Drilling
Au, g/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 16-1 was determined. The results found in TABLE 16-2 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 percent 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 Gold Corporation[4]. This testing included SAG Mill Comminution (SMC), Bond Rod Mill Work Index (BRMWI), Bond Ball Mill Work Index (BBMWI), Bond Abrasion Index (BAI) 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 Services Pty Ltd. undertook a technical evaluation of the various comminution circuits based on the testwork undertaken by JK Tech Pty Ltd.[5]. 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 percent reduction in energy requirements over the conventional SABC circuit.

TABLE 16-2: ENERGY REQUIREMENTS FOR DIFFERENT PROCESS FLOWSHEETS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
June 2009

	Process	
	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 16-1.

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 16-3, 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 82 percent of the contained gold with the proposed process flowsheet.

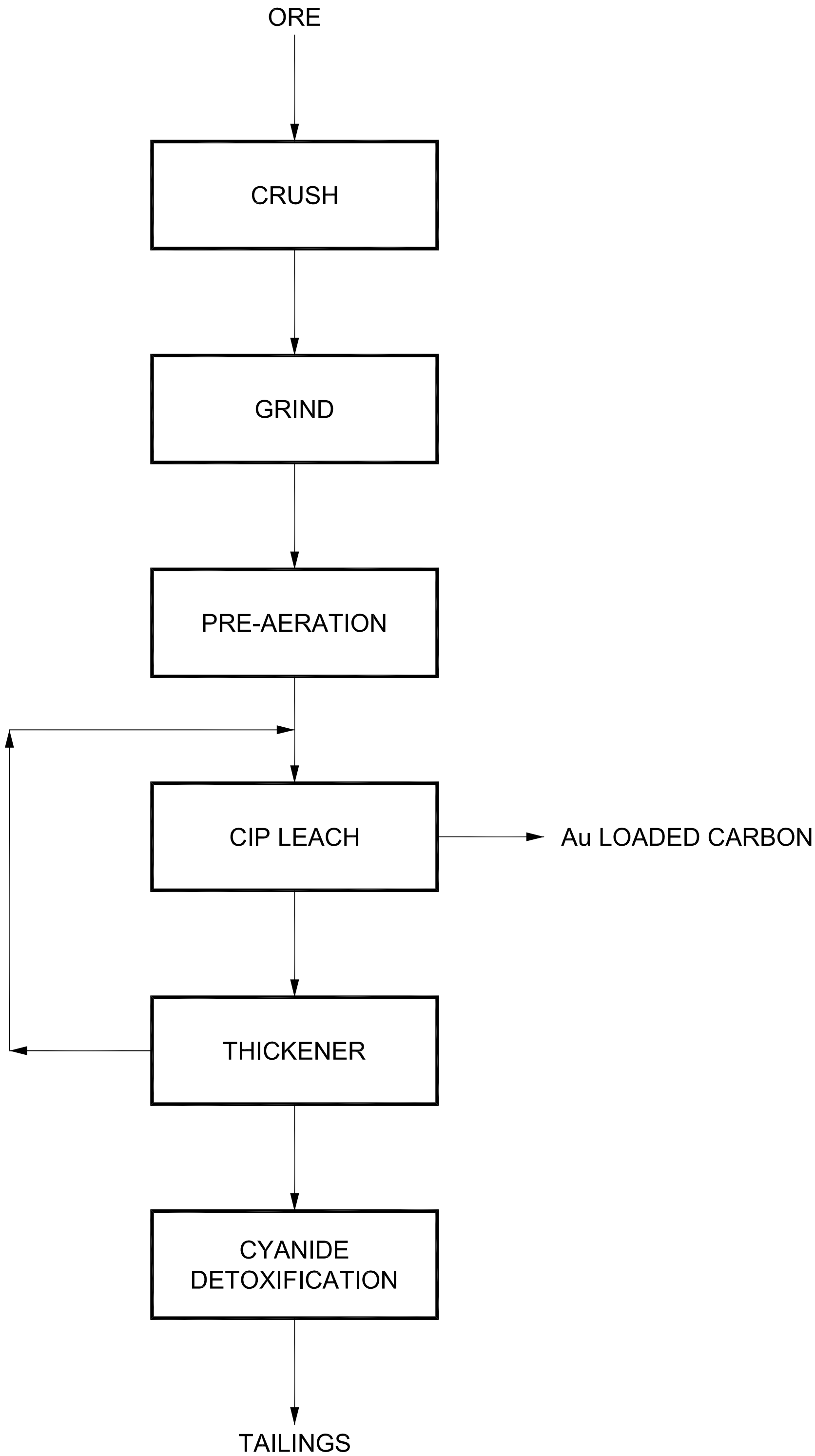
RDi provided cyanide leach residue to Pocock Industrial, Inc. to develop data for design of thickening and filtration equipment for the project. The testwork undertaken included flocculant screening tests, conventional and dynamic thickening tests, viscosity tests and vacuum filtration tests to size horizontal belt filters[6]. 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/Mtpd with 70 percent 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 percent underflow solids.
- For paste thickening (74 to 75 percent 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 percent).

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[7]. The SO_2/air process produced less than 50 ppm WAD cyanide following the detoxification process using 2.3 grams of SMBS per gram of total cyanide.



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TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Vista Gold Corp.***	Fig16-1.dwg	**Figure 16-1** **Leach Process Flowsheet**
	Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	

TABLE 16-3: LEACH TEST RESULTS (P_{80}=100 MESH)
VISTA GOLD CORP. – MT TODD GOLD PROJECT
June 2009

Test No.	Cyanide Maintain/ Decay	Leach Time, Hours	Extraction %		Residue g/t Au	Cal. Head g/t Au	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

Note: 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.

REFERENCES:

1. Metallurgical Review of Mt. Todd Project: Progress Report No. 1, RDi report dated May 19, 2006.

2. Preliminary Metallurgical Testing of Mt. Todd Ore: Progress Report No. 2, RDi Report dated May 9, 2007.

3. Metallurgical Testing of Mt. Todd Samples, RDi Report dated July 29, 2009.

4. Comminution Test Report on Five Samples from Mt. Todd Mine, JK Tech. Pty. Ltd., June to August 2009.

5. JKSimMet Circuit Simulations for the 11 mt Vista Gold Mt. Todd Plant, Ausenco Report dated August 19, 2009.

6. Flocculant Screening, Gravity Sedimentation, Pulp Rheology and Vacuum Filtration Studies for Vista Gold Mt. Todd Project, Pocock Industrial Inc. Report dated October 2009.

7. Mt. Todd Project Report of Tailings Characterization Test Work, KCA Report dated May 6, 2010.

17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

The following sections summarize the thought processes, procedures, and results of Tetra Tech's independent estimate of the contained gold resources of the Batman and Quigleys Deposits. Only the Batman and Quigleys' deposits currently have classified resource estimates. APPENDIX A provides detailed information on the resource estimation process, parameters, methodology utilized, and verification checks.

17.1 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 meter sample. Based on this work, the bulk densities applied to the resource model are presented in TABLE 17-1.

TABLE 17-1: SUMMARY OF BATMAN SG DIAMOND CORE DATA BY OXIDATION STATE
VISTA GOLD CORP. – MT TODD GOLD PROJECT
March 2008

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

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 and are presented in TABLE 17-2. Results show that the average moisture content is less than one percent and the average SG 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 17-2: BATMAN PIT SAMPLE SG DATA
VISTA GOLD CORP. – MT TODD GOLD PROJECT
March 2008

	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

17.2 Quigleys Deposit Drill Hole and Density Data

The Quigleys Deposit is approximately 3.5 kilometers 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 drill holes, with the majority reaching a depth of 100m at a 60 degree dip; oriented 83 degrees azimuth. Assays were taken at a nominal one meter interval. Geologic interpretation in section produced wireframes modeling thin ore zones dipping west. Material inside the wire frames has been given a code of 1. Outside the ore zones, the material has been given a code of 9999.

Zone 1 gold grades range from .001 to 21.75 g/t., averaging 0.703 g/t. Zone 9999 gold grades range from 0.001 to 11.318, with an average of 0.148 g/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 ore 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 1-m to 2-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 percent shear and 85 percent stockwork and weighting the density values accordingly. TABLE 17-3 contains the SG data assigned to the Quigleys area according to oxidation state.

TABLE 17-3: QUIGLEYS DEPOSIT SG DATA
VISTA GOLD CORP. – MT TODD GOLD PROJECT
March 2008

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.

17.3 Drillhole Data

An Access database set up in Gemcom has been recreated from the old exploration database. Tables for the grade control database have been inserted into this database.

17.3.1 Batman Exploration Database

TABLE 17-4 is a summary of the Batman exploration database that formed the basis of the resource estimation of that deposit.

TABLE 17-4: SUMMARY OF BATMAN EXPLORATION DATABASE
VISTA GOLD CORP. – MT TODD GOLD PROJECT
OCTOBER 2010

Drill Hole Statistics						
	Northing (m)	**Easting (m)**	**Elevation (m)**	**Azimuth**	**Dip**	**Depth**
Minimum	8,434,220.0	186,588.4	0.0	0.0	43.0	0.0
Maximum	8,435,888.0	187,388.7	223.5	294.0	90.0	570.0
Average	8,434,989.3	186,985.7	169.3	240.3	61.4	147.8
Range	1,668.0	800.3	223.5	294.0	47.0	570.5
Cumulative Drill Hole Statistics						
Total Count	759					
Total Length (m)	112,198.7					
Assay Length (m)	1 (approx)					
Drill Hole Grade Statistics						
Label	**Number**	**Average**	**Std. Dev.**	**Min.**	**Max**	**Missing**
Au (GPT)	106,012	0.5867	1.223	0.001	55.37	1,427
Cu (%)	20,062	0.0406	0.06147	0.001	2.40	87,377

The pre-2007 exploration database consisted of 743 drill holes, 226 diamond holes and 517 percussion holes. A total of 97,810 samples existed within that exploration database. Diamond core is 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 have been 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 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 a question. Billiton reports suggest that different laboratories along with the orientation of drill holes 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 have been found containing 80 percent of the original assay data. Inspection of these data has shown 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 have been used to fix this problem.

After going through all the logs and laboratory assays, the data have now been corrected and reloaded into the database. Codes have been allocated, with below detection assays given a grade of 0.005, which is half the detection limit of 0.01 and missing samples 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 drill hole database. A separate table has also been created for the multi-element data.

The existing lithology tables in the database are split into two tables, Extra and More (containing lithology, mineralization, oxidation structural data etc.).

In 2008 an additional sixteen (16) core holes were drilled. Gold was analyzed along with thirty-three (33) elements and added to the database. In addition, pulps from thirteen (13) of the pre-2007 holes were analyzed for the same suite of multi-elements.

17.3.2 Quigleys Exploration Database

TABLE 17-5 details the Quigleys exploration database.

TABLE 17-5: SUMMARY OF QUIGLEYS EXPLORATION DATABASE
VISTA GOLD CORP. – MT TODD GOLD PROJECT
OCTOBER 2010

Drill Hole Statistics

	Northing (m)	Easting (m)	Elevation (m)	Azimuth	Dip	Depth
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 Drill Hole Statistics

Total Count	631
Total Length (m)	57,605.8
Assay Length (m)	1 (approx)

Drill Hole Grade Statistics

Label	Number	Average	Std. Dev.	Min.	Max	Missing
Au (GPT)	52,152	0.2445	0.8764	0	36.00	82
Cu (%)	40,437	0.0105	0.0305	0	2.98	11,897

17.4 Batman Block Model Parameters

TABLE 17-6 details the physical limits of the Batman deposit block model utilized in the estimation of mineral resources.

TABLE 17-6: BLOCK MODEL* PHYSICAL PARAMETERS – BATMAN DEPOSIT
VISTA GOLD CORP. – MT TODD GOLD PROJECT
February 2009

Direction	Minimum	Maximum	Block size	#Blocks
x-dir	186,492 mE	187,548 mE	12m	84
y-dir	8,434,188 mN	8,435,952 mN	12m	146
z-dir	-994 m	224m	6	203

* Model changed from previous Tetra Tech estimates to reflect the new 2008 drill hole locations and depths.

17.5 Quigleys Block Model Parameters

Quigleys' block model parameters are shown in TABLE 17-7. The model consists of 37,082 blocks within the modeled ore zones (blocks within the modeled ore grade zones are coded as 1). Each of the blocks is 250 m^3 (5x25x2m) with a defined density of 2.77 (692.5 tonnes).

TABLE 17-7: BLOCK MODEL* PHYSICAL PARAMETERS – QUIGLEYS DEPOSIT
VISTA GOLD CORP. – MT TODD GOLD PROJECT
OCTOBER 2010

Direction	Minimum	Maximum	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

17.6 Mineral Resource Estimate

At the present time, resources have only been estimated for the Batman and Quigleys deposits. Tetra Tech created three-dimensional computerized geologic and grade models of the Batman and Quigleys deposits.

The geologic model of the Batman and Quigleys deposits was created by GGC and audited by Tetra Tech. 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. GGC 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 GGC geologic model accurately portrays 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 GGC in the geologic model. Gold grades were estimated into the individual blocks of the model by ordinary, whole-block kriging.

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 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 17-8.

TABLE 17-8: RESOURCE CLASSIFICATION CRITERIA
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

BATMAN (March 2008 & February 2009)			
Category	**Search Range & Kriging Variance**	**No. of Sectors/ Max Pts per DH**	**Min Pts**
Measured	Core Complex: 60 m & KV < 0.30	4/3	4
Indicated	Core Complex: 150 m search & KV >= 0.30 and <0.55	4/2	2
Indicated	Outside Core Complex: 50 m search & KV <0.45	4/3	8
Inferred	Core Complex: 150 m & KV >0.55	4/3	2
Inferred	Outside Core Complex: 150 m & KV <0.45	4/3	3
QUIGLEYS (October 2010)			
Category	**Search Range & Kriging Variance**	**No. of Sectors/ Max Pts per DH**	**Min Pts**
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 & < 0.335 Zone 9999 < 25 m	4/3	3

The classification was accomplished by a combination of kriging variance, number of points used in the estimate, and number of sectors used. TABLES 17-9 and 17-10 detail the results of the classification. Copper, lead, zinc, and silver quantities and grades are presented using the gold cutoff grades and classification. All of the resources quoted are contained on Vista's mineral leases.

TABLES 17-9 and 17-10 detail the estimated in-place resources by classification and by cutoff grade for the Batman and Quigleys Deposits respectively. All of the resources quoted are contained on Vista's mineral leases. The Reserve Case cutoff for the resource reporting is 0.4 g Au/t and is bolded in the table. This cutoff value was determined in the June 11, 2009 "Mt Todd Gold Project Updated Preliminary Economic Assessment Report" using then-current gold price and cost assumptions. The estimate of in-place resources remains consistent with those reported in the 2099 Report.

TABLE 17-9: BATMAN DEPOSIT CLASSIFIED GOLD RESOURCES
VISTA GOLD CORP. – MT TODD GOLD PROJECT
February 2009

Cutoff Grade g Au/tonne	Tonnes (x1000)	Average Grade g Au/tonne	Total Au Ounces (x1000)
MEASURED			
2.00	1,977	2.38	151
1.75	3,676	2.14	253
1.50	6,469	1.91	398
1.25	10,163	1.71	560
1.00	16,119	1.49	774
0.90	19,764	1.39	885
0.80	24,262	1.29	1,007
0.70	29,616	1.19	1,136
0.60	36,700	1.09	1,284
0.50	44,645	0.99	1,424
0.40	**52,919**	**0.91**	**1,543**
INDICATED			
2.00	3,238	2.49	259
1.75	5,773	2.21	410
1.50	10,140	1.95	637
1.25	17,532	1.70	961
1.00	30,873	1.45	1,437
0.90	39,308	1.34	1,694
0.80	50,410	1.23	1,996
0.70	64,371	1.13	2,332
0.60	82,412	1.02	2,707
0.50	105,936	0.92	3,121
0.40	**138,020**	**0.81**	**3,581**

MEASURED + INDICATED (1)			
2.00	5,215	2.45	410
1.75	9,449	2.18	663
1.50	16,609	1.94	1,035
1.25	27,695	1.71	1,521
1.00	46,992	1.46	2,210
0.90	59,072	1.36	2,578
0.80	74,672	1.25	3,003
0.70	93,987	1.15	3,468
0.60	119,112	1.04	3,991
0.50	150,581	0.94	4,545
0.40	**190,939**	**0.84**	**5,125**

NOTE (1): The sum of measured and indicated resources as reported under NI 43-101 is equivalent to mineralized material under SEC Industry Guide 7.

INFERRED RESOURCES			
Cutoff Grade g Au/tonne	**Tonnes (x1000)**	**Average Grade g Au/tonne**	**Total Au Ounces (x1000)**
2.00	2,058	2.76	183
1.75	3,056	2.47	242
1.50	4,808	2.16	333
1.25	7,936	1.84	470
1.00	14,280	1.52	696
0.90	18,878	1.38	836
0.80	25,593	1.24	1,018
0.70	35,885	1.10	1,266
0.60	48,503	0.98	1,529
0.50	66,725	0.86	1,849
0.40	**94,008**	**0.74**	**2,244**

TABLE 17-10: QUIGLEYS DEPOSIT CLASSIFIED GOLD RESOURCES
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Cutoff Grade g Au/tonne	Tonnes (x1000)	Average Grade g Au/tonne	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 (1)			
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**

NOTE (1): The sum of measured and indicated resources as reported under NI 43-101 is equivalent to mineralized material under SEC Industry Guide 7.

INFERRED RESOURCES			
Cutoff Grade g Au/tonne	**Tonnes (x1000)**	**Average Grade g Au/tonne**	**Total Au Ounces (x1000)**
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**

17.7 Mineral Reserves

As of the date of this report, only the Batman Deposit contains CIM definable mineral reserves. The Quigleys Deposit contains no CIM definable mineral reserves. Mineral reserves for the Batman Deposit are presented in SECTION 18.0 of this report.

18.0 PIT DESIGN AND MINERAL RESERVE ESTIMATE

At the present time, the Mt. Todd gold project mineral resource model used for the mining design study is documented in the technical report entitled "*Mt. Todd Gold Project Updated Preliminary Economic Assessment Report Northern Territory, Australia*" (June 11, 2009).

18.1 Geotechnical Data

An existing pit at Mt. Todd was excavated during the period from 1992 to 1997. This excavation reached a depth of approximately 130 m and was terminated at the end of the first ore phase, at which time the second phase had been essentially stripped. Water (pumped in from RP1) currently fills the pit to a depth of approximately 80 m, leaving only the slopes of the second phase exposed.

The pit has been standing for eleven years with little evidence of slope deterioration during this period, except in the upper 50 m of weathered materials where small failure scarps can be seen locally. Within the exposed pit wall there are local sections of the slope that are defined by geologic structure (bedding and/or joints), particularly on the eastern walls, but for the most part the slopes are standing as they were excavated and the bench faces reflect the equipment utilized for excavation.

18.1.1 Pit Wall Design

With today's technology, the design of pit slopes is based on a review of geologic conditions that might limit the stable slope angle. These conditions include geologic structures, rock strength, and groundwater. If no limiting conditions are found during the investigation, the designer usually falls back to some sort of "fail-safe" recommendation.

For all but the weakest rock (as long as geologic conditions don't change over spatial distance), a slope that will stand over a nominal height, say 10 m, will also stand over a considerably greater height (several tens of meters) at the same angle. In practice, however, we usually leave residual benches in the slope profile to "catch" rockfalls, hoping to protect men and equipment working at lower elevations. For the most part, rockfall is the result of careless excavation practice and can easily be minimized if the operators attend to good blasting and excavating practice during mining. With rockfall minimized, the need for catch benches is minimized, and benches can be safely stacked to improve the inter-ramp slope angle by as much as 15 degrees. There is generally a significant economic benefit to this and it more than covers the slightly increased mining cost that results from the improved practice.

Given the discussion above, it is apparent that the key parameter in pit slope design is the bench face angle, or the angle from the horizontal at which the bench face will stand in a stable fashion. This angle will either directly reflect the structural conditions within the rock mass; i.e., bedding, foliation, faults and joints, or the method of excavation; i.e., rope shovels, hydraulic excavators, backhoes, etc., as well as the blasting practice employed.

With an existing pit available for inspection, the determination of bench face angles and the governing structural conditions becomes a simple matter.

18.1.2 Geologic Structures

Bedding in the host rock metasediments is the single pervasive structural condition of concern. Through the pit area, bedding strikes consistently at 325 degrees (N35W) and dips southwesterly between 40 and 60 degrees. In the northeast corner of the present pit, bench faces are locally determined by bedding. Elsewhere along the east wall, bedding, in combination with northwesterly dipping joints, forms adversely oriented wedges which define the bench face angle. These structural conditions determine the geometry of the benches along the east wall, which are standing typically around 50 degrees but are locally flatter than that. For design, bench faces on the east wall should not be considered to stand at angles steeper than 50 degrees. Careful excavation should minimize rockfall, enabling inter-ramp slopes of around 40 degrees or slightly steeper.

Elsewhere around the pit, limiting conditions are rarely in evidence and most of the structures dip away from the pit. Bench faces are typically at 65 degrees or greater and often as steep as 80 degrees. There is no reason that these slopes shouldn't be planned at 70 degrees, with inter-ramp slopes in the 55- to 60-degree range. Diligent excavation practice will be required to minimize rockfall.

18.1.3 Rock Strength

As it stands within the ground, rock is under stress: gravitational assuredly, but most likely tectonic as well. As a general rule, the horizontal stress is about 1.5 times the vertical stress near the earth's surface. The effect of excavating an open pit is fundamentally to relieve this stress through unloading. The horizontal stress realigns around the excavation while the vertical stress is reduced. The only significant part of the pit in which stress levels increase is the region of the toe. Elsewhere, as the stress level reduces, simple elasticity considerations dictate a tendency for the slopes to move upward and toward the excavation. This trend is most noticeable at the pit crest and diminishes both with distance behind the slope and at depth within the pit. This general observation largely determines the behavior of the pit walls as excavation proceeds, including the development of surficial instabilities.

The metasediments at Mt. Todd are unusually strong: compressive strength is typically greater than 100 MPa (about 14,500 psi), but does drop to perhaps 70 MPa in local units. However, the stress levels to be generated in the toe area of the proposed approximately 500-meter-deep pit should not exceed 10 or 20 MPa, so failure of the rock materials is not likely.

18.1.4 Groundwater

The groundwater regime at Mt. Todd is poorly defined at present. But the rock has a very low porosity and water will be largely confined to and controlled by fracture systems within the rock mass. Permeability should be sufficient to encourage natural drainage towards the excavation and thereby reduce the influence of water pressures on wall stability. The affect of groundwater can be ignored for this stage of the project study.

18.1.5 Pit Slope Recommendations

To summarize, the limiting factors on slope performance at Mt. Todd are geologic structures; primarily bedding, but jointing as well to a lesser degree, and these are relevant only to the east pit wall. Rock strength and groundwater do not appear to be significant concerns at this time.

Bench faces on the east wall should be designed at 50 degrees, with inter-ramp slopes not to exceed 40 degrees. Elsewhere, bench faces can be designed at 70 degrees, with inter-ramp slopes in the 55 to 60 degree range. Apart from the east wall, these are arbitrary designations considering good, but not unusual, operating practices. Improvement may prove possible once experience is gained and slope behavior is better understood.

The remaining portions of Section 18 (sections 18.2 through 18.4) have been taken from *"MDA Pre-Feasibility Mine Study, Mt. Todd, Northern Territory, Australia" (January 17, 2011)* **with only minor changes for consistent formatting and terminology purposes (see Appendix B for complete report).**

18.2 Reserve Case Pit Optimization

Potentially mineable pit shapes were evaluated using a Lerchs-Grossman ("LG") analysis performed with the GEMS® Whittle pit optimization software and the Mt. Todd mineral resource model. The optimization is an iterative process with initial parameters coming from the Mt. Todd June 11th, 2009 PEA. The final parameters incorporate mining costs developed during this study. The optimization runs used only Measured and Indicated material for processing. All Inferred material was considered as waste. The parameters assumed for the LG analyses are summarized in TABLE 18-1.

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.

18.2.1 Economic Parameters

Mining, processing, tailings construction, tailings reclamation, waste dump rehabilitation, and general and administrative ("G&A") costs were based on the previous Mt. Todd prefeasibility study. The Au price of $1,000 per was rounded down from the 36-month trailing average gold price for the end of 2010. Based on prices published by Kitco.com, the 36-month trailing average gold price at the end of 2010 was approximately $1,022 per ounce. Table 18-1 contains the parameters used in the Lerch-Grossman Analyses for the Reserve Case.

TABLE 18-1: RESERVE CASE PARAMETERS FOR LERCHS-GROSSMAN ANALYSES
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Overall Pit Slopes	33° from pit centered azimuth ranging 10° – 150° 55° from pit centered azimuth ranging 150° – 10°
Gold Price	US$1000 per toz Au
Gold Recovery	82 percent
Mining Cost	US$1.40 per tonne mined
Processing Cost	US$7.60 per tonne processed
Tailings Construction	$1.00 per tonne processed
Tailings Reclamation	$1.14 per tonne processed
Waste Dump Rehabilitation	$0.12 per tonne waste
General and Administrative Cost	US$0.60 per tonne processed

18.2.2 Slope Parameters

Slope parameters were based on studies provided by Tetra Tech (Appendix C). These recommended slopes were reduced to account for ramps required for equipment access. For pit optimization, slopes were divided into two sectors based on bearing to the slope wall as follows:

Bearing	Overall Slope
10° to 150°	33°
150° to 10°	55°

18.2.3 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 $400 to $1,200 per ounce Au in increments of $25 per ounce Au in order to analyze the deposit's sensitivity to gold prices. Results for $50 per ounce increments are shown in TABLE 18-2.

TABLE 18-2: WHITTLE PIT OPTIMIZATION RESULTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2010

Gold Price ($US)	Material Processed			Waste Tonnes (1000's)	Total Tonnes (1000's)	Strip Ratio (W:O)
	Tonnes (1000's)	Au (gm/tone)	Gold toz (1000's)			
400	5,199	1.62	272	6,603	11,802	1.27
450	7,670	1.54	380	11,850	19,519	1.55
500	10,800	1.45	502	18,346	29,147	1.70
550	15,236	1.35	661	27,018	42,255	1.77
600	25,827	1.22	1,013	52,697	78,524	2.04
650	49,609	1.11	1.764	116,013	165,622	2.34
700	71,654	1.05	2,422	180,150	251,804	2.51
750	91,854	1.00	2,958	227,012	318,866	2.47
800	104,254	0.96	3,229	243,106	347,360	2.33
850	122,385	0.93	3,660	289,939	412,325	2.37
900	139,856	0.89	4,013	316,976	456,833	2.27
950	152,325	0.87	4,259	341,206	493,531	2.24
1,000	161,851	0.85	4,405	342,139	503,990	2.11
1,050	175,505	0.82	4,648	367,678	543,183	2.09
1,100	190,001	0.80	4,879	387,345	577,347	2.04
1,150	197,245	0.79	4,982	392,203	589,448	1.99
1,200	208,986	0.76	5,126	390,470	599,456	1.87

18.2.4 Pit-Shell Selection for Ultimate Pit Limit

The $1,000 per ounce Au Whittle pit shell was used as a guide for the ultimate pit design.

18.2.5 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. The internal pits or phases within the ultimate pit were designed to enhance the project by providing higher-value material to the process plant earlier in the mine life. Phase 1 and phase 2 pit designs remain unchanged from the previous preliminary feasibility work. Phase 3 was designed to the ultimate pit limit on the south, while phase 4 (the final pit phase) is used to achieve the ultimate pit in the north.

18.2.6 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 the equipment anticipated to be used in mining. In areas where the material is consistently ore or waste so that dilution is not an issue, benches may be mined in 12-meter heights. Mining of waste is anticipated to be conducted by using a larger mining fleet. A smaller fleet is anticipated to be used for construction and mining on six-meter benches. Benches using 12-meter heights should primarily be mined with the larger fleet. Should the smaller fleet be used for 12-meter benches, some dozing of material may be required.

18.2.7 Pit Slopes

Slope parameters were based on geotechnical studies provided by Tetra Tech which recommend inter-ramp slopes of 40° on the east and 55° to 60° for the western portions of the pit. The eastern wall is considered to be those projected outward between 10° and 150° azimuths. The remaining walls were considered west.

MDA used this recommendation to create slope parameters that include catch benches. For this purpose, bench-face angles were assumed to be 50° and 70° for east and west, respectively, with catch benches placed every 24 m in height. Catch-bench widths were rounded to the nearest half meter, and it was determined that catch-bench widths of 8.5 and 8 m for east and west walls, respectively, would provide the required inner-ramp angles. The back-calculated inner-ramp angles are 39.96° and 55.11° for east and west walls, respectively. Tetra Tech's recommendation for inner-ramp angles up to 60° for west walls was rejected as it would only provide a catch bench of 5.1 m in width for every 24 m in height. MDA believes that this is too narrow to be effective in catching rock that may fall from the crest of the catch bench. TABLE 18-3 shows the slope parameters used for pit design.

TABLE 18-3: PIT DESIGN SLOPE PARAMETERS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

	East Side	West Side	Units
Height Between Catch Benches	24	24	Meters
Inner-Ramp Angle	40	55	Degrees
Bench Face Angle	50	70	Degrees
Catch Bench Width	8.5	8	Meters

18.2.8 Haulage Roads

Ramps were designed to have a maximum centerline gradient of 10 percent. In areas where the ramps may curve along the outside of the pit, the inside gradient may be up to 11 or 12 percent 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 both CAT 789C and 785C trucks, which have operating widths of 7.67 and 6.64 m, respectively. For haul roads inside of the pit, a single safety berm on the pit side of the roadway will be required to be at least half the height of the largest vehicle tire that uses the road. MDA assumes that safety berms can be created at a 1.5 horizontal to 1 vertical slope using run-of-mine material. A flat top on the berms of 0.33 m is assumed, and a berm height of 1.82 m provides half of the truck tire heights plus 10% for 789C trucks. The 10 percent addition is used to ensure that the berm height exceeds half of the truck tire height in all cases. The resulting base width of safety berms is 5.78 m.

Haul-road designs inside of pits where only one safety berm is required are designed to be 30 m wide for two-way traffic. Subtracting berm widths, this provides 3.2 and 3.4 times the widths of 789C and 785C trucks, respectively, for running width.

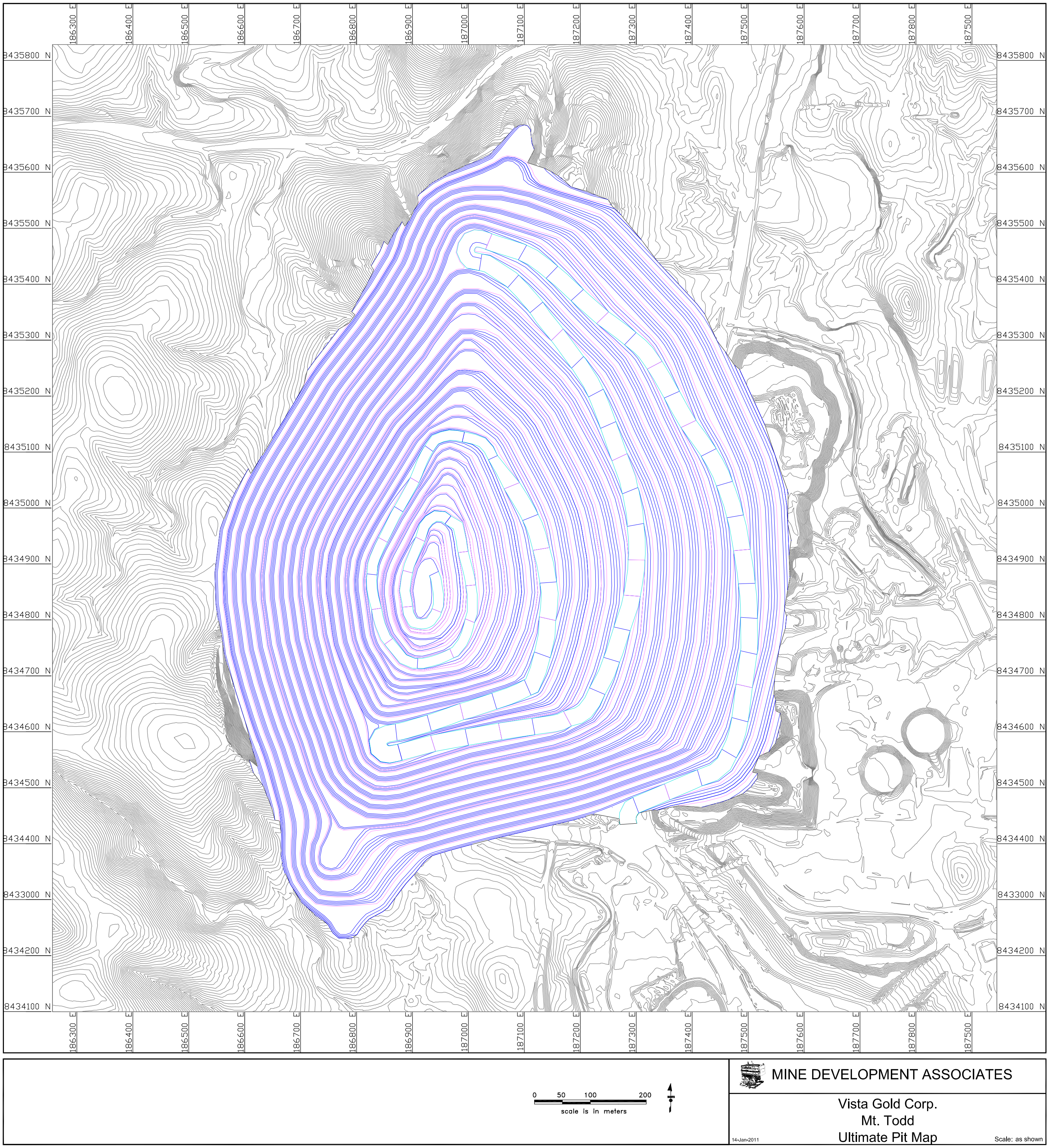
In lower portions of the pits where haulage requirements allow use of one way traffic, haul roads are designed to have a width of 18 m. This provides 1.6 and 1.9 times the widths of 789C and 785C trucks, respectively, for running width.

Haul roads outside of pit designs have been designed to be 38 m to account for an additional safety berm.

18.2.9 Ultimate Pit

As discussed in previous sections, the $1,000 Whittle pit shell was used for guidance when designing the ultimate pit. 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 two switchbacks in the ultimate pit design and the lower portion of the pit spirals counter clockwise to achieve the ultimate pit design.

FIGURE 18-1 shows the Base case ultimate pit design, and resulting reserves are shown in TABLE 18-5.



(from MDA 2011)

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TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Vista Gold Corp.***	Fig18-1.dwg	**Figure 18-1** **Pre-Feasibility (10.6 Mtpy) Case Ultimate Pit Design**
	Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: Jan/2011	

18.2.10 Pit Phasing

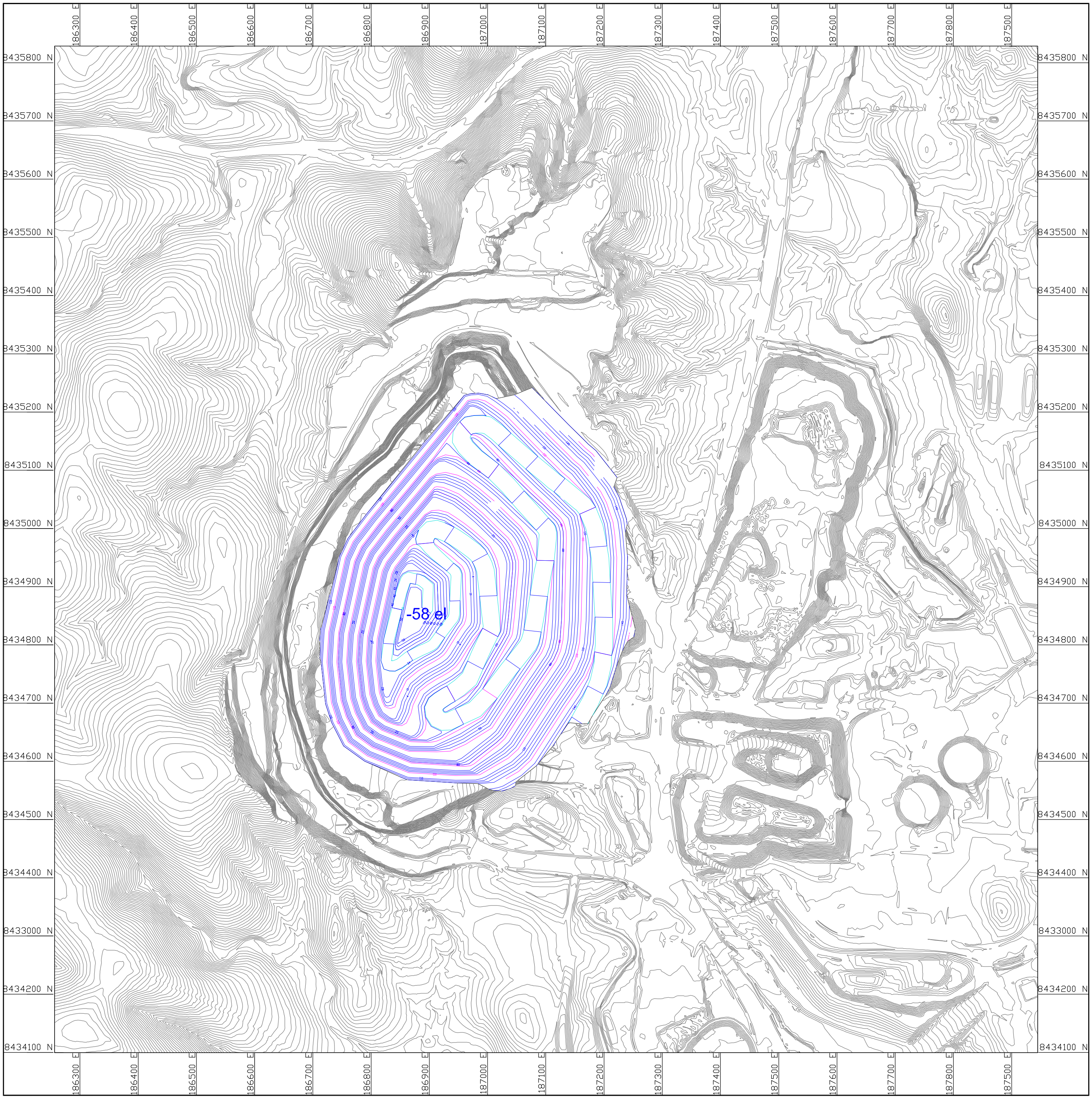
Pit phases were created to improve the project's NPV by mining higher-value material in the initial years while providing sufficient ore feed to the mill and maintaining access for people and equipment. The first two pit phases were designed during the previous preliminary prefeasibility study. As the previous preliminary prefeasibility ultimate pit design was smaller than the current ultimate pit, it was deemed that these pit phases were appropriate for the current study.

The final pit was designed using two additional pit phases to achieve the ultimate pit. The third phase was designed to achieve the ultimate pit in the south, while the fourth phase completed the ultimate pit design in the north.

FIGURES 18-2, 18-3, and 18-4 show phases 1, 2 and 3 pit designs, respectively. Resulting reserves for each of the phases are shown in TABLE 18-4. Bench reserves for phase 1, phase 2, and phase 3 are shown in TABLE 18-5. The combined ultimate pit bench reserves are shown in TABLE 18-6

18.2.11 Cutoff Grade

Based on the economic parameters and $1,000 per ounce Au, the break-even cutoff grade is calculated at 0.45 g Au/t, and the internal cutoff grade is calculated at 0.40 g Au/t. An additional cutoff grade of 0.55 g Au/t was used to delineate low-grade and medium-grade ore for scheduling. During scheduling, low-grade ore was stockpiled when appropriate and processed during lean times or at the end of the mine life.



0 50 100 200
scale is in meters

MINE DEVELOPMENT ASSOCIATES

Vista Gold Corp.
Mt. Todd
Phase 1 Pit Design

14-Jan-2011

Scale: as shown

(from MDA 2011)

Issued by:

TETRA TECH
350 Indiana Street, Suite 500
Golden, Colorado 80401
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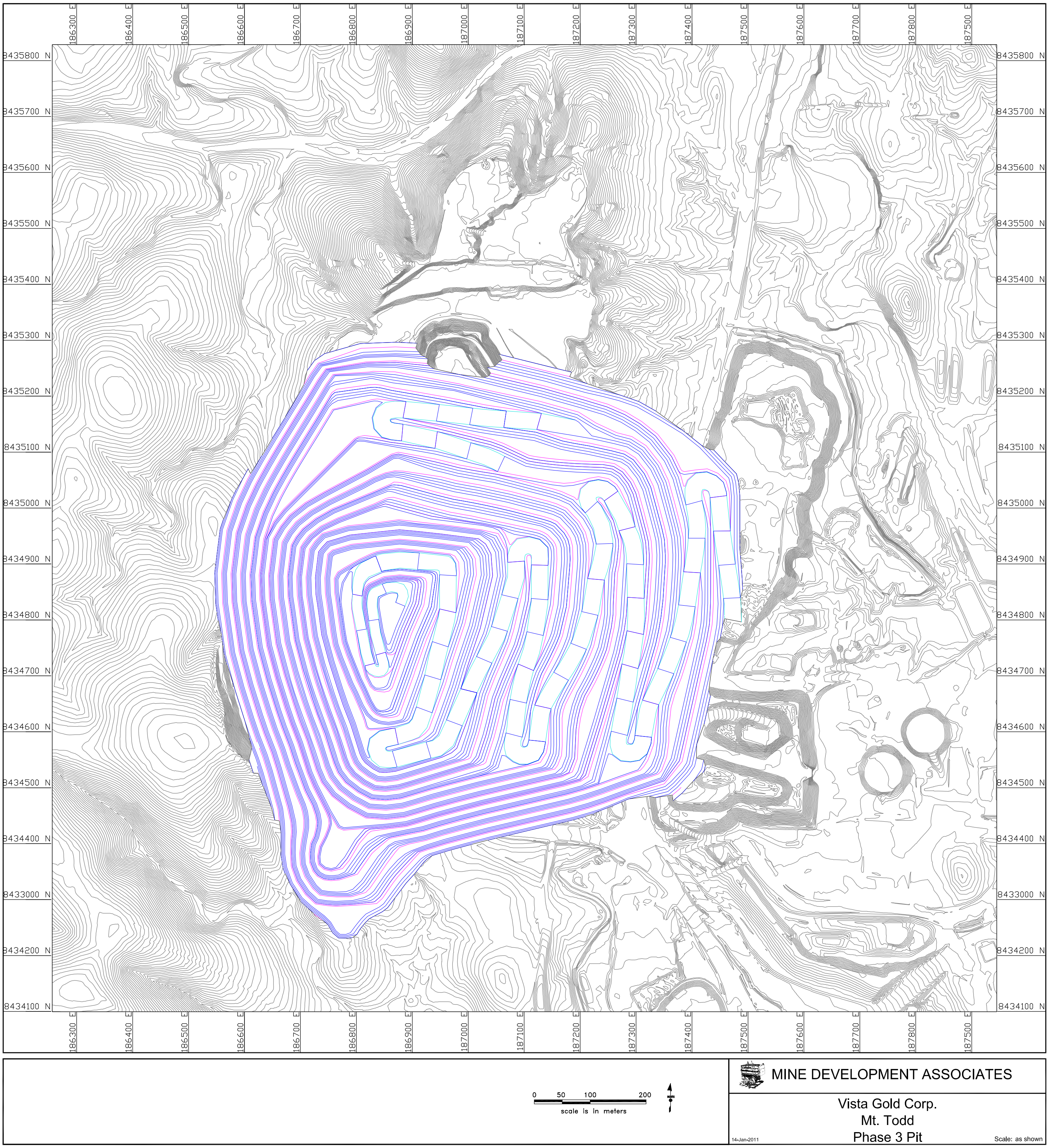
Prepared for:	File Name:
Vista Gold Corp.	Fig18-2.dwg
Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a
Project Location: Northern Territory, Australia	Date of Issue: Jan/2011

Figure 18-2
Pre-Feasibility (10.6 Mtpy) Case
Phase 1 Pit Design

(from MDA 2011)

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.***	Fig18-3.dwg	**Figure 18-3** **Pre-Feasibility (10.6 Mtpy) Case** **Phase 2 Pit Design**
	Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: Jan/2011	



(from MDA 2011)

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.***	Fig18-4.dwg	**Figure 18-4** **Pre-Feasibility (10.6 Mtpy) Case** **Phase 3 Pit Design**
	Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: Jan/2011	

18.2.12 Dilution

The resource model with block sizes of 12m by 12m by 6m was used to estimate reserves. The model was estimated based on the block size, and this model was used to define the ultimate pit limit and estimate Proven and Probable reserves. MDA considers the 12m by 12m by 6m block size to be reasonable for mining the deposit and believes that this represents an appropriate amount of dilution for statement of reserves.

18.3 Reserves and Resources

The Reserve Case has been used to define Proven and Probable reserves. By CIM standards, this requires completion of at least a pre-feasibility-level study. The following sections discuss the reserves and resources reported for the Reserve Case.

18.3.1 Reserve Case Reserves

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 standards. The NI 43-101 standards rely on the CIM Definition Standards on Mineral Resources and Mineral Reserves adopted by the CIM council. CIM standards define Proven and Probable Reserves as:

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.

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.

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.

TABLE 18-4 reports the Proven and Probable Reserves based on the Reserve Case pit design. These 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.

18.3.2 Bench Reserves

Proven and Probable bench reserves have been estimated for each phase and are shown in TABLE 18-5 The total Proven and Probable reserves by bench are shown in TABLE 18-6. Due to rounding issues in reporting, these do not add up exactly to the reserves reported in TABLE 18-5; however the differences are inconsequential.

TABLE 18-4: PROVEN AND PROBABLE RESERVES BY PHASE *
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Phase	Proven Reserves			Probable Reserves			Proven and Probable			Waste K Tonnes	Total K Tonnes	Strip Ratio
	K Tonnes	g Au/t	K toz Au	K Tonnes	g Au/t	K toz Au	K Tonnes	g Au/t	K toz Au			
1	11,640	1.11	416	11,183	0.93	336	22,823	1.03	752	17,682	40,505	0.77
2	7,731	0.84	208	17,009	0.81	444	24,740	0.82	653	45,223	69,963	1.83
3	7,734	0.84	208	15,652	0.82	414	23,386	0.83	623	84,814	108,200	3.63
4	21,855	0.85	598	57,070	0.81	1,486	78,925	0.82	2,084	123,760	202,685	1.57
Total	48,961	0.91	1,431	100,913	0.83	2,681	149,875	0.85	4,112	271,480	421,354	1.81

* **Reserves are reported using a cutoff grade of 0.40 g Au/t**

TABLE 18-5: PROVEN AND PROBABLE BENCH RESERVES BY PHASE *
VISTA GOLD CORP. – MT TODD
October 2010

	Phase 1						Phase 2						Phase 3						Phase 4					
	Proven & Probable Reserves			Waste	Total	Strip	Proven & Probable Reserves			Waste	Total	Strip	Proven & Probable Reserves			Waste	Total	Strip	Proven & Probable Reserves			Waste	Total	Strip
Bench	K Tonnes	g Au/t	K Oz Au	Tonnes	Tonnes	Ratio	K Tonnes	g Au/t	K Oz Au	Tonnes	Tonnes	Ratio	K Tonnes	g Au/t	K Oz Au	Tonnes	Tonnes	Ratio	K Tonnes	g Au/t	K Oz Au	Tonnes	Tonnes	Ratio
194	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		-	-	-	-	-	
188	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		-	-	-	-	-	
182	-	-	-	-	-		-	-	-	-	-		-	-	-	1	1	NA	-	-	-	24	24	NA
176	-	-	-	-	-		-	-	-	-	-		-	-	-	1	1	NA	-	-	-	165	165	NA
170	-	-	-	-	-		-	-	-	-	-		-	-	-	10	10	NA	126	0.62	3	294	420	2.33
164	-	-	-	-	-		-	-	-	-	-		-	-	-	153	153	NA	277	0.60	5	764	1,041	2.75
158	-	-	-	43	43	NA	-	-	-	178	178	NA	-	-	-	541	541	NA	475	0.60	9	1,138	1,613	2.40
152	8	0.44	0	163	171	20.34	57	0.63	1	457	514	8.01	28	0.48	0	1,017	1,045	36.65	727	0.61	14	1,278	2,005	1.76
146	11	0.59	0	323	335	28.66	108	0.63	2	830	938	7.72	69	0.55	1	2,000	2,069	28.93	730	0.60	14	1,940	2,670	2.66
140	19	0.54	0	580	599	30.78	104	0.64	2	1,547	1,652	14.85	92	0.56	2	2,959	3,052	31.99	738	0.59	14	2,398	3,136	3.25
134	34	0.59	1	687	720	20.38	130	0.61	3	1,746	1,876	13.43	127	0.56	2	3,257	3,384	25.72	740	0.59	14	2,862	3,602	3.87
128	35	0.58	1	711	747	20.23	136	0.61	3	1,800	1,936	13.23	115	0.59	2	3,039	3,154	26.48	723	0.58	13	2,853	3,576	3.95
122	47	0.53	1	894	941	18.97	236	0.63	5	2,154	2,389	9.14	162	0.61	3	3,273	3,435	20.16	794	0.59	15	3,052	3,846	3.85
116	136	0.58	3	1,564	1,700	11.50	281	0.61	6	2,257	2,538	8.03	180	0.60	3	3,231	3,411	17.96	817	0.59	16	3,055	3,872	3.74
110	794	0.72	18	1,321	2,114	1.66	369	0.63	8	2,211	2,579	6.00	209	0.59	4	3,227	3,436	15.44	831	0.61	16	3,150	3,981	3.79
104	794	0.74	19	1,060	1,853	1.33	378	0.63	8	2,137	2,515	5.65	252	0.60	5	3,029	3,281	12.04	860	0.60	17	3,063	3,923	3.56
98	914	0.75	22	1,014	1,927	1.11	409	0.61	8	2,046	2,456	5.00	263	0.60	5	3,009	3,271	11.45	887	0.59	17	3,035	3,922	3.42
92	980	0.77	24	1,002	1,982	1.02	409	0.60	8	2,032	2,441	4.96	274	0.59	5	2,958	3,232	10.80	881	0.59	17	3,020	3,902	3.43
86	984	0.80	25	987	1,971	1.00	436	0.60	8	1,982	2,417	4.55	265	0.63	5	2,924	3,189	11.03	865	0.59	16	3,032	3,897	3.51
80	972	0.82	25	776	1,748	0.80	505	0.60	10	1,831	2,336	3.63	255	0.62	5	2,768	3,023	10.86	912	0.59	17	2,895	3,806	3.17
74	923	0.85	25	789	1,712	0.85	536	0.63	11	1,819	2,355	3.39	230	0.62	5	2,698	2,929	11.73	929	0.59	18	2,916	3,846	3.14
68	933	0.87	26	769	1,702	0.82	560	0.64	12	1,800	2,360	3.21	219	0.63	4	2,608	2,828	11.89	923	0.58	17	2,955	3,878	3.20
62	955	0.90	28	756	1,710	0.79	493	0.64	10	1,771	2,264	3.59	223	0.63	5	2,564	2,787	11.50	919	0.58	17	2,983	3,903	3.25
56	841	0.96	26	616	1,456	0.73	577	0.65	12	1,529	2,106	2.65	226	0.65	5	2,491	2,717	11.01	942	0.60	18	2,891	3,833	3.07
50	831	0.98	26	581	1,413	0.70	622	0.64	13	1,407	2,030	2.26	253	0.64	5	2,281	2,534	9.01	960	0.59	18	3,049	4,009	3.18
44	818	1.03	27	794	1,612	0.97	680	0.62	13	1,316	1,996	1.94	215	0.63	4	2,200	2,416	10.21	993	0.60	19	3,030	4,023	3.05
38	1,206	1.07	42	642	1,848	0.53	677	0.64	14	1,300	1,977	1.92	218	0.67	5	2,146	2,364	9.83	1,037	0.60	20	2,979	4,017	2.87
32	1,216	1.05	41	426	1,642	0.35	687	0.65	14	1,127	1,814	1.64	239	0.69	5	1,917	2,156	8.03	1,050	0.62	21	2,998	4,048	2.86
26	1,177	1.08	41	359	1,536	0.30	656	0.65	14	1,125	1,781	1.72	257	0.68	6	1,837	2,095	7.14	1,121	0.62	22	2,943	4,064	2.62
20	1,154	1.09	40	247	1,400	0.21	673	0.67	14	1,130	1,802	1.68	278	0.68	6	1,768	2,045	6.37	1,132	0.62	23	2,913	4,045	2.57
14	1,125	1.08	39	175	1,300	0.16	661	0.67	14	1,120	1,781	1.70	305	0.66	6	1,711	2,016	5.62	1,127	0.64	23	2,886	4,013	2.56
8	945	1.12	34	98	1,043	0.10	749	0.71	17	960	1,709	1.28	335	0.68	7	1,594	1,930	4.75	1,118	0.65	23	2,742	3,859	2.45
2	869	1.16	32	77	945	0.09	811	0.72	19	866	1,678	1.07	346	0.68	8	1,552	1,899	4.48	1,095	0.67	24	2,738	3,833	2.50
-4	773	1.22	30	79	852	0.10	826	0.74	20	819	1,644	0.99	352	0.68	8	1,554	1,907	4.41	1,144	0.67	25	2,633	3,777	2.30
-10	716	1.23	28	57	773	0.08	842	0.76	21	768	1,611	0.91	393	0.69	9	1,486	1,879	3.78	1,161	0.69	26	2,582	3,743	2.22
-16	562	1.32	24	26	589	0.05	905	0.80	23	572	1,477	0.63	431	0.70	10	1,380	1,811	3.20	1,199	0.71	27	2,399	3,598	2.00
-22	467	1.43	21	26	493	0.06	948	0.83	25	491	1,438	0.52	464	0.70	10	1,358	1,823	2.93	1,197	0.72	28	2,357	3,554	1.97
-28	412	1.46	19	22	433	0.05	921	0.85	25	452	1,374	0.49	475	0.71	11	1,308	1,783	2.75	1,292	0.72	30	2,262	3,554	1.75
-34	357	1.52	17	10	366	0.03	933	0.86	26	392	1,326	0.42	506	0.67	11	1,266	1,772	2.50	1,325	0.74	32	2,194	3,519	1.66
-40	290	1.55	14	5	294	0.02	857	0.90	25	290	1,146	0.34	507	0.69	11	1,175	1,682	2.32	1,323	0.77	33	2,051	3,374	1.55
-46	227	1.68	12	2	230	0.01	847	0.95	26	266	1,113	0.31	521	0.71	12	1,127	1,649	2.16	1,340	0.78	34	2,007	3,347	1.50
-52	175	1.74	10	2	177	0.01	852	0.97	27	201	1,053	0.24	544	0.73	13	1,093	1,637	2.01	1,347	0.79	34	1,968	3,315	1.46
-58	115	1.88	7	-	115	-	823	1.02	27	187	1,010	0.23	599	0.72	14	1,035	1,634	1.73	1,339	0.81	35	1,917	3,256	1.43
-64	10	2.06	1	-	10	-	725	1.15	27	98	823	0.14	667	0.79	17	896	1,563	1.34	1,345	0.82	35	1,805	3,150	1.34
-70	-	-	-	-	-		661	1.19	25	69	730	0.11	717	0.78	18	845	1,562	1.18	1,394	0.83	37	1,711	3,105	1.23
-76	-	-	-	-	-		596	1.25	24	62	658	0.10	731	0.79	19	807	1,538	1.11	1,383	0.87	38	1,674	3,057	1.21
-82	-	-	-	-	-		512	1.31	22	38	550	0.07	797	0.82	21	749	1,546	0.94	1,343	0.87	38	1,666	3,009	1.24
-88	-	-	-	-	-		395	1.30	17	19	414	0.05	797	0.88	23	619	1,416	0.78	1,370	0.86	38	1,494	2,863	1.09
-94	-	-	-	-	-		330	1.27	13	14	345	0.04	840	0.90	24	551	1,392	0.66	1,363	0.86	38	1,456	2,819	1.07
-100	-	-	-	-	-		294	1.28	12	5	299	0.02	847	0.92	25	489	1,336	0.58	1,319	0.87	37	1,455	2,774	1.10
-106	-	-	-	-	-		237	1.29	10	2	239	0.01	862	0.95	26	450	1,312	0.52	1,314	0.87	37	1,405	2,719	1.07
-112	-	-	-	-	-		177	1.25	7	-	177	-	785	0.98	25	343	1,128	0.44	1,279	0.89	37	1,275	2,555	1.00
-118	-	-	-	-	-		120	1.21	5	-	120	-	761	1.02	25	298	1,059	0.39	1,247	0.89	36	1,286	2,533	1.03
-124	-	-	-	-	-		-	-	-	-	-		828	1.04	28	262	1,090	0.32	1,272	0.87	36	1,214	2,486	0.95
-130	-	-	-	-	-		-	-	-	-	-		771	1.08	27	243	1,014	0.31	1,261	0.87	35	1,174	2,435	0.93
-136	-	-	-	-	-		-	-	-	-	-		650	1.12	23	167	817	0.26	1,311	0.86	36	979	2,290	0.75
-142	-	-	-	-	-		-	-	-	-	-		580	1.15	21	145	725	0.25	1,367	0.87	38	896	2,263	0.66
-148	-	-	-	-	-		-	-	-	-	-		507	1.11	18	140	647	0.28	1,442	0.89	41	777	2,219	0.54
-154	-	-	-	-	-		-	-	-	-	-		440	1.13	16	128	568	0.29	1,458	0.89	42	723	2,180	0.50
-160	-	-	-	-	-		-	-	-	-	-		339	1.08	12	77	416	0.23	1,447	0.93	43	571	2,019	0.39
-166	-	-	-	-	-		-	-	-	-	-		298	1.04	10	29	327	0.10	1,453	0.95	44	545	1,999	0.38
-172	-	-	-	-	-		-	-	-	-	-		252	0.97	8	15	267	0.06	1,378	0.97	43	546	1,924	0.40
-178	-	-	-	-	-		-	-	-	-	-		194	0.90	6	10	203	0.05	1,383	1.00	44	500	1,883	0.36
-184	-	-	-	-	-		-	-	-	-	-		156	0.91	5	-	156	-	1,294	1.00	42	369	1,663	0.29
-190	-	-	-	-	-		-	-	-	-	-		115	0.95	4	2	117	0.02	1,272	1.02	42	342	1,615	0.27
-196	-	-	-	-	-		-	-	-	-	-		22	0.66	0	-	22	-	1,302	1.02	43	314	1,616	0.24
-202	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		1,223	1.05	41	328	1,551	0.27
-208	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		1,145	1.05	39	185	1,330	0.16
-214	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		1,086	1.05	37	167	1,253	0.15
-220	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		1,008	1.10	36	146	1,154	0.15
-226	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		964	1.13	35	116	1,080	0.12
-232	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		817	1.19	31	69	886	0.08
-238	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		755	1.23	30	54	810	0.07
-244	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		695	1.23	28	46	741	0.07
-250	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		627	1.27	26	38	665	0.06
-256	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		507	1.36	22	9	515	0.02
-262	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		442	1.40	20	2	444	0.01
-268	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		382	1.43	18	4	386	0.01
-274	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		318	1.41	14	5	323	0.01
-280	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		214	1.42	10	2	216	0.01
-286	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		167	1.44	8	-	167	-
-292	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		118	1.45	5	0	118	0.00
-298	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		65	1.16	2	0	65	0.01
-304	-	-	-	-	-		-	-	-	-	-		-	-	-	-	-		-	-	-	-	-	
Total	22,823	1.03	752	17,682	40,505	0.77	24,740	0.82	653	45,223	69,963	1.83	23,386	0.83	623	84,814	108,200	3.63	78,925	0.82	2,084	123,760	202,685	1.57

* Totals may not match exactly to reported reserves due to rounding issues

TABLE 18-6: TOTAL PROVEN AND PROBABLE RESERVES *
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

	Total of All Phases					
	Proven & Probable Reserves			Waste	Total	Strip
Bench	K Tonnes	g Au/t	K Oz Au	Tonnes	Tonnes	Ratio
194	-	-	-	-	-	
188	-	-	-	-	-	
182	-	-	-	25	25	NA
176	-	-	-	166	166	NA
170	126	0.62	3	304	431	2.41
164	277	0.60	5	917	1,195	3.31
158	475	0.60	9	1,901	2,375	4.00
152	820	0.61	16	2,915	3,734	3.56
146	918	0.60	18	5,094	6,012	5.55
140	954	0.59	18	7,485	8,438	7.85
134	1,030	0.59	20	8,552	9,583	8.30
128	1,009	0.58	19	8,404	9,413	8.33
122	1,238	0.60	24	9,373	10,611	7.57
116	1,414	0.60	27	10,107	11,521	7.15
110	2,202	0.65	46	9,908	12,111	4.50
104	2,284	0.66	48	9,288	11,572	4.07
98	2,472	0.65	52	9,104	11,576	3.68
92	2,545	0.66	54	9,011	11,556	3.54
86	2,550	0.67	55	8,925	11,475	3.50
80	2,644	0.68	57	8,270	10,914	3.13
74	2,619	0.69	58	8,222	10,841	3.14
68	2,635	0.70	59	8,131	10,767	3.09
62	2,590	0.71	59	8,074	10,664	3.12
56	2,586	0.73	61	7,527	10,112	2.91
50	2,666	0.73	62	7,319	9,985	2.74
44	2,707	0.74	64	7,340	10,047	2.71
38	3,139	0.80	80	7,067	10,206	2.25
32	3,191	0.80	82	6,468	9,659	2.03
26	3,212	0.80	83	6,264	9,476	1.95
20	3,235	0.80	83	6,057	9,293	1.87
14	3,217	0.80	83	5,892	9,109	1.83
8	3,148	0.81	82	5,393	8,541	1.71
2	3,121	0.82	83	5,233	8,355	1.68
-4	3,095	0.83	83	5,085	8,180	1.64
-10	3,112	0.83	83	4,894	8,006	1.57
-16	3,097	0.84	84	4,377	7,474	1.41
-22	3,075	0.86	85	4,233	7,308	1.38
-28	3,101	0.85	85	4,044	7,145	1.30
-34	3,121	0.86	86	3,862	6,983	1.24
-40	2,976	0.87	83	3,521	6,497	1.18
-46	2,936	0.89	84	3,402	6,338	1.16
-52	2,917	0.89	83	3,265	6,182	1.12
-58	2,876	0.89	83	3,138	6,014	1.09
-64	2,747	0.91	80	2,799	5,546	1.02
-70	2,772	0.90	80	2,626	5,398	0.95
-76	2,710	0.93	81	2,544	5,253	0.94
-82	2,652	0.94	80	2,454	5,105	0.93
-88	2,562	0.94	77	2,132	4,693	0.83
-94	2,533	0.92	75	2,022	4,555	0.80
-100	2,460	0.93	74	1,948	4,409	0.79
-106	2,413	0.94	73	1,857	4,270	0.77
-112	2,241	0.95	68	1,618	3,859	0.72
-118	2,128	0.96	65	1,583	3,711	0.74
-124	2,100	0.94	63	1,476	3,576	0.70
-130	2,032	0.95	62	1,417	3,450	0.70
-136	1,961	0.95	60	1,147	3,107	0.58
-142	1,947	0.95	60	1,041	2,988	0.53
-148	1,949	0.95	59	917	2,866	0.47
-154	1,898	0.95	58	851	2,749	0.45
-160	1,786	0.96	55	649	2,435	0.36
-166	1,752	0.96	54	574	2,326	0.33
-172	1,630	0.97	51	561	2,190	0.34
-178	1,577	0.99	50	509	2,086	0.32
-184	1,449	0.99	46	369	1,819	0.25
-190	1,387	1.01	45	345	1,732	0.25
-196	1,324	1.02	43	314	1,638	0.24
-202	1,223	1.05	41	328	1,551	0.27
-208	1,145	1.05	39	185	1,330	0.16
-214	1,086	1.05	37	167	1,253	0.15
-220	1,008	1.10	36	146	1,154	0.15
-226	964	1.13	35	116	1,080	0.12
-232	817	1.19	31	69	886	0.08
-238	755	1.23	30	54	810	0.07
-244	695	1.23	28	46	741	0.07
-250	627	1.27	26	38	665	0.06
-256	507	1.36	22	9	515	0.02
-262	442	1.40	20	2	444	0.01
-268	382	1.43	18	4	386	0.01
-274	318	1.41	14	5	323	0.01
-280	214	1.42	10	2	216	0.01
-286	167	1.44	8	-	167	-
-292	118	1.45	5	0	118	0.00
-298	65	1.16	2	0	65	0.01
-304	-	-	-	-	-	
Total	149,875	0.85	4,112	271,480	421,354	1.81

* Totals may not match exactly to reported reserves due to rounding issues

18.3.3 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 drill holes.*
>
> *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.*

The Inferred resource inside of the pit designs totals 6,330,000 tonnes at an average grade of 0.66 g Au/t. This inferred material contains 134,600 ounces of gold.

19.0 OTHER RELEVANT DATA AND INFORMATION

The following portions of Section 19 (sections 19.1 through 19.3) have been taken from *"MDA Pre-Feasibility Mine Study, Mt. Todd, Northern Territory, Australia" (August 31, 2010)* **with only minor changes for consistent formatting and terminology purposes (see Appendix B for complete report).**

19.1 Mine Operations

19.1.1 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.

19.1.2 Mine Waste Facilities

Mine-waste facilities have been designed to permanently contain the waste material associated with reserves in the pit. These facilities are an extension to existing waste dumps at site. The ultimate design incorporates an angle of repose slope of 1.5 vertical to 1 horizontal with catch benches of 30 m on 20-meter lifts (see FIGURE 19-1 for Site Layout).

During the mine life, as opportunity arises, slopes will be dozed to a slope of 2.5 to 1 with 10 meter benches left to arrest any water runoff. The final reclamation will have a slope of 3.0 to 1 or flatter.

The current waste facility is approximately 24 m high located to the southeast of the pit. The ultimate dump design is 140 to 170 m above the original topography. The base of the dump to the south was designed to have a minimum offset of 50 m from the existing RP1 waste water storage facility. To the east, the design is bound by the process facility and the Batman divide. This maintains potential drainage within the basin that feeds into the RP1 waste water management site. The west side of the waste dump is bound by exploration potential.

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 north around the crushing area. This will be placed early in the mine life to allow for road traffic and stockpile area.

A 40 percent swell factor and an average specific gravity of 2.67 (bank) has have been assumed for volume calculations. The total waste dump design will contain approximately 214.0 million tonnes of waste material, with an additional 5.6 million tonnes being used to flatten areas around the crushing plant and the stockpile area.

In addition to these two areas, waste will also be placed as part of the construction for the tailings storage facility one (TSF1) and the tailings storage facility two (TSF2), and suitable waste has been planned for use in capping of the TSF1. Total tonnage required for TSF1 and TSF2 construction is 4.0 million tonnes and 54.0 million tonnes, respectively. Material to be used for capping of TSF1 totals 7.0 million tonnes.

The total waste storage capacity is 273.5 million tonnes. This is one percent more than the capacity required. This is a slim margin and may be a risk to the reserves should the realized swell factor be greater than that assumed or should model reconciliation create additional waste volume that requires containment. MDA considers this risk to reserves to be minimal as the swell factor used is most likely conservative because it does not include compaction of the material as it is placed.

TABLE 19-1 shows the planned capacities for waste storage. Yearly mine plan maps show the growth of the waste dump and are provided in Appendix A of the MDA report. The MDA report is provided herewith as Appendix B to this PFS report. These maps also illustrate concurrent reclamation during mining operations along with yearly estimated pit positions.

TABLE 19-1: WASTE STORAGE CAPACITY
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

	K Tonnes
Ultimate Waste Dump Design	214,010
Base for Crusher and Stockpile	5,582
TSF1 Construction	3,980
TSF2 Construction	43,000
TSF1 Capping	6,972
Total Waste Storage Capacity	273,544

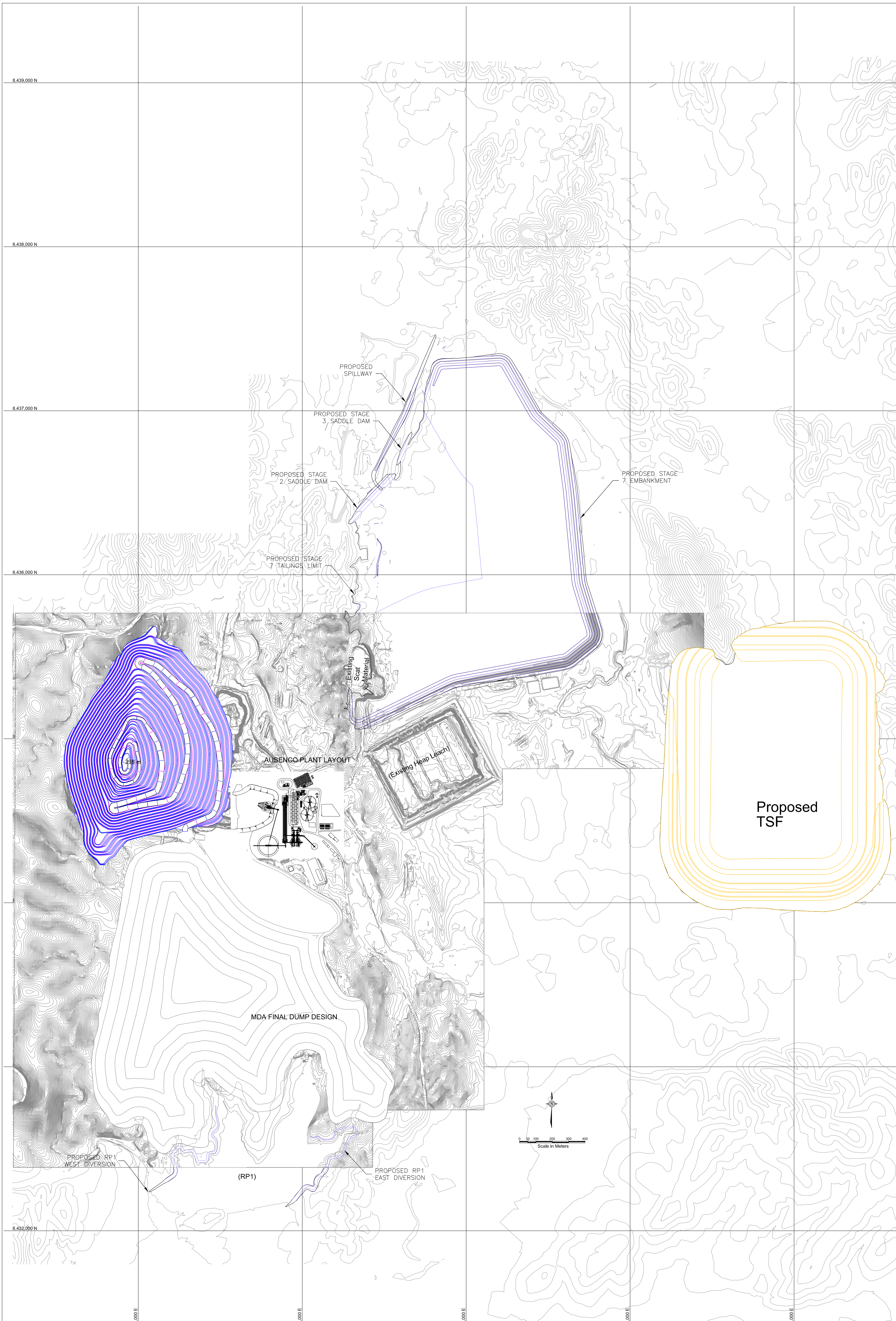
19.1.3 Mine Production Schedule

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 the number of trucks and shovels required to achieve annual production of 10.65 million tonnes of ore per year and maintain stripping requirements. TABLE 19-2 shows the mine-production schedule. Ore material is broken down into three categories: high-grade ore consists of Proven and Probable reserves above a 1.0 g Au/t cutoff grade; medium-grade ore is Proven and Probable reserves above a cutoff of 0.55 g Au/t but below a cutoff grade of 1.0 g Au/t cutoff; and low-grade ore that is above a cutoff of 0.40 g Au/t but below a cutoff grade of 0.55 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. The growth of the stockpile is shown in the yearly pit position maps in Appendix A of the MDA report (see Appendix B of this PFS report). 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 this schedule, three stockpiles are assumed: High-grade ore stockpile for high-grade ore; medium-grade stockpile 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. Thus, the yearly mine plan maps in Appendix A of the MDA report (see Appendix B of this PFS report) only show a single stockpile and it is assumed that the mine will maintain separate stockpiles during pre-stripping in year -1. During the life of mine, 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 life of mine. The maximum stockpile balance through the life of mine is estimated to be 9.2 million tonnes, which occurs near the end of the mine life. The total stockpile capacity is estimated to be 10.1 million tonnes. Re-handling of stockpiled material will be done using a loader and trucks to haul ore to the crusher. TABLE 19-3 shows the material that is re-handled, and TABLE 19-4 shows the resulting stockpile balances for the end of each year.

Table 19-5 shows the mine ore that is sent to the crusher, which is a combination of ore shipped directly from the mine and ore that is reclaimed from stockpiles.



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: Fig19-1.dwg	**Figure 19-1**
	Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a	**General Facilities Layout Map**
	Project Location: Northern Territory, Australia	Date of Issue: 01/26/2011	

TABLE 19-2: ANNUAL MINE PRODUCTION SCHEDULE
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

			Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Total
Total Mine Production	Mine to Crusher	Tonnes	-	10,035	10,650	10,650	10,650	6,066	8,175	10,650	10,650	8,568	10,650	10,650	10,650	10,650	2,213	-	130,906
		g Au/t	-	0.94	1.02	0.95	0.95	0.71	0.67	0.87	0.77	0.67	0.86	0.92	1.04	1.13	1.40	-	0.91
		Au Ozs	-	303	350	325	326	139	176	298	264	184	296	314	358	386	99	-	3,817
	Mine to Low-Grade Stockpile	Tonnes	469	2,175	2,934	1,347	-	134	-	2,548	508	422	2,976	1,452	2,729	660	-	-	18,354
		g Au/t	0.47	0.47	0.47	0.47	-	0.48	-	0.47	0.47	0.48	0.47	0.47	0.47	0.48	-	-	0.47
		Au Ozs	7	33	44	20	-	2	-	39	8	6	45	22	42	10	-	-	278
	Mine to Medium-Grade Stockpile	Tonnes	487	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	487
		g Au/t	0.72	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.72
		Au Ozs	11	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	11
	Mine to High-Grade Stockpile	Tonnes	128	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	128
		g Au/t	1.32	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.32
		Au Ozs	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5
	Total Ore Mined	Tonnes	1,084	12,210	13,584	11,997	10,650	6,200	8,175	13,198	11,158	8,990	13,626	12,102	13,379	11,310	2,213	-	149,875
		g Au/t	0.68	0.86	0.90	0.90	0.95	0.71	0.67	0.79	0.76	0.66	0.78	0.86	0.93	1.09	1.40	-	0.85
		Au Ozs	24	336	394	346	326	141	176	337	272	190	341	336	399	396	99	-	4,112
	Mine to Waste Dump	Tonnes	4,002	22,965	24,348	24,060	25,218	22,029	14,841	24,662	17,738	9,455	20,386	325	5,672	1,805	22	-	217,528
	Mine Waste to Tsf1	Tonnes	2,285	-	700	340	360	295	-	-	-	-	-	-	-	-	-	-	3,980
	Mine Waste to Tsf2	Tonnes	-	-	-	-	-	5,500	10,200	-	-	13,200	-	13,833	267	-	-	-	43,000
	Mine Waste to Tsf1 Capping	Tonnes	-	-	-	-	-	-	-	-	6,972	-	-	-	-	-	-	-	6,972
	Total Waste	Tonnes	6,287	22,965	25,048	24,400	25,578	27,824	25,041	24,662	24,710	22,655	20,386	14,158	5,940	1,805	22	-	271,480
	Total Mined	Tonnes	7,370	35,175	38,632	36,397	36,228	34,024	33,216	37,860	35,868	31,644	34,012	26,259	19,318	13,115	2,235	-	421,354
		Strip Ratio	5.80	1.88	1.84	2.03	2.40	4.49	3.06	1.87	2.21	2.52	1.50	1.17	0.44	0.16	0.01		1.81

TABLE 19-3: ANNUAL ORE RE-HANDLE SCHEDULE
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

		Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Total
Low-Grade Stockpile	K Tonnes	-	-	-	-	-	4,584	2,475	-	-	2,082	-	-	-	-	8,437	775	18,354
Rehandle	g Au/t	-	-	-	-	-	0.47	0.47	-	-	0.48	-	-	-	-	0.47	0.47	0.47
	K Au Ozs	-	-	-	-	-	69	37	-	-	32	-	-	-	-	128	12	278
Medium-Grade Stockpile	Tonnes	-	487	-	-	-	-	-	-	-	-	-	-	-	-	-	-	487
Rehandle	g Au/t	-	0.72	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.72
	Au Ozs	-	11	-	-	-	-	-	-	-	-	-	-	-	-	-	-	11
High-Grade Stockpile	Tonnes	-	128	-	-	-	-	-	-	-	-	-	-	-	-	-	-	128
Rehandle	g Au/t	-	1.32	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.32
	Au Ozs	-	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5
Total Stockpile	Tonnes	-	615	-	-	-	4,584	2,475	-	-	2,082	-	-	-	-	8,437	775	18,969
Rehandle	g Au/t	-	0.85	-	-	-	0.47	0.47	-	-	0.48	-	-	-	-	0.47	0.47	0.48
	Au Ozs	-	17	-	-	-	69	37	-	-	32	-	-	-	-	128	12	295

TABLE 19-4: ANNUAL STOCKPILE BALANCE
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

		Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15
Low-Grade Stockpile	Tonnes	469	2,644	5,578	6,925	6,925	2,475	-	2,548	3,055	1,395	4,372	5,823	8,552	9,212	775	-
Balance	g Au/t	0.47	0.47	0.47	0.47	0.47	0.47	-	0.47	0.47	0.47	0.47	0.47	0.47	0.47	0.47	-
	Au Ozs	7	40	84	104	104	37	-	39	46	21	66	88	130	140	12	-
Medium-Grade Stockpile	Tonnes	487	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Balance	g Au/t	0.72	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Au Ozs	11	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
High-Grade Stockpile	Tonnes	128	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Balance	g Au/t	1.32	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Au Ozs	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Stockpile	Tonnes	1,084	2,644	5,578	6,925	6,925	2,475	-	2,548	3,055	1,395	4,372	5,823	8,552	9,212	775	-
Balance	g Au/t	0.68	0.47	0.47	0.47	0.47	0.47	-	0.47	0.47	0.47	0.47	0.47	0.47	0.47	0.47	-
	Au Ozs	24	40	84	104	104	37	-	39	46	21	66	88	130	140	12	-

TABLE 19-5: ANNUAL ORE DELIVERY TO THE MILL CRUSHER
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

		Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Total
Pit to Mill	Tonnes	-	10,035	10,650	10,650	10,650	6,066	8,175	10,650	10,650	8,568	10,650	10,650	10,650	10,650	2,213	-	130,906
	g Au/t	-	0.94	1.02	0.95	0.95	0.71	0.67	0.87	0.77	0.67	0.86	0.92	1.04	1.13	1.40	-	0.91
	Au Ozs	-	303	350	325	326	139	176	298	264	184	296	314	358	386	99	-	3,817
Stockpiles to Mill	Tonnes	-	615	-	-	-	4,584	2,475	-	-	2,082	-	-	-	-	8,437	775	18,969
	g Au/t	-	0.85	-	-	-	0.47	0.47	-	-	0.48	-	-	-	-	0.47	0.47	0.48
	Au Ozs	-	17	-	-	-	69	37	-	-	32	-	-	-	-	128	12	295
Total to Mill	Tonnes	-	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	775	149,875
	g Au/t	-	0.93	1.02	0.95	0.95	0.61	0.62	0.87	0.77	0.63	0.86	0.92	1.04	1.13	0.66	0.47	0.85
	Au Ozs	-	320	350	325	326	208	213	298	264	215	296	314	358	386	228	12	4,112

19.1.4 Equipment Selection and Productivities

Mt. Todd has been planned as an open-pit mine using large haul trucks, hydraulic shovels, and front-end loading equipment. Primary mine production is achieved using two Hitachi EX3600 hydraulic shovels along with CAT 789C haul trucks. A shovel bucket size of 21 m^3 is assumed, though final equipment selection may differ. The CAT 789C haul trucks have a rated payload of 180 tonnes. This equipment is used primarily for the movement of waste material, though ore mining is planned using the equipment as ore is encountered.

Secondary mine production is achieved using a CAT 992 loader and smaller CAT 785C trucks. The 992 loader is assumed to have a 12 cubic meter bucket, and the CAT 785C trucks have a rated payload of 140 tonnes. The loader and smaller trucks are used primarily to move ore from the pit to the crusher and for reclamation of ore from stockpiles. Some waste production from the 992 loader and 785C trucks is anticipated as well.

TABLE 19-6 shows the maximum shovel productivity estimate based on scheduled time, availability, and truck and material parameters. This maximum productivity would require that the presentation of trucks is always available; however, that is not always the case.

Truck productivity is based on truck-cycle times from the pit to predetermined destinations. The destinations include the crusher, ore stockpiles, waste dumps, and tailings facilities. Because the planned waste dump is so large, it was divided into smaller volumes, and cycle times were calculated to each of the smaller dumps. In all, the dump was divided into a total of 21 parts. The cycle times were calculated by bench for each mining phase and used to calculate the truck hours required to move ore and waste. During scheduling, the truck hours and loading-unit hours were used as a mining constraint to ensure that available hours were not exceeded for either trucks or loading equipment. TABLE 19-7 shows the estimated loading and hauling fleet requirements.

TABLE 19-6: MAXIMUM LOADER PRODUCTIVITY ESTIMATE
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

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

Daily Schedule		
Shifts per Day	shift/day	2
Hours per Shift	hr/shift	12
Theoretical 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		**21 cm Hyd 140 T Trks**	**12 cm FEL 140 T Trks**	**21 cm Hyd 180 T Trks**
Shovel Mech. Avail.	%	85%	85%	85%
Operating Efficiency	%	83%	83%	83%
Bucket Capacity	cym	18	12	21
Bucket Fill Factor	%	95%	95%	95%
Avg. Cycle Time	sec	34	50	34
Truck Parameters				
Truck Mech. Avail.	%	85%	85%	85%
Operating Efficiency	%	83%	83%	83%
Volume Capacity	cym	78	78	105
Tonnage Capacity	lt (Wet)	136	136	180
Truck Spot Time	sec	24	24	24

Shovel Productivity		**21 cm Hyd 140 T Trks**	**12 cm FEL 140 T Trks**	**21 cm Hyd 180 T Trks**
Effective Bucket Capacity	cyd	17.10	11.40	19.95
Tonnes per Pass - Wet	lst (Wet)	33.0	22.0	38.5
Tonnes per Pass - Dry	lst (Dry)	30.5	20.4	35.6
Theoretical Passes - Vol	passes	4.56	6.84	5.26
Theoretical Passes - Wt	passes	4.12	6.19	4.68
Actual Passes Used	passes	4.0	6.0	5.0
Truck Tonnage - Wet	wmt/load	132	132	180
Truck Tonnage - Dry	dmt/load	122	122	167
Truck Capacity Utilized - Vol	%	88%	88%	89%
Truck Capacity Utilized - Wt	%	97%	97%	100%
Load Time	min	2.67	5.40	3.23
Theoretical Productivity	dst/hr	2,748	1,357	3,093
Tonnes per Operating Hour	dst/hr	2,280	1,130	2,570
Tonnes per Day	dst/day	40,700	20,200	45,900
Potential - 355 day year	t/year	14,448,500	7,171,000	16,294,500

TABLE 19-7: ANNUAL LOAD AND HAUL EQUIPMENT REQUIREMENTS

VISTA GOLD CORP. – MT TODD GOLD PROJECT

January 2011

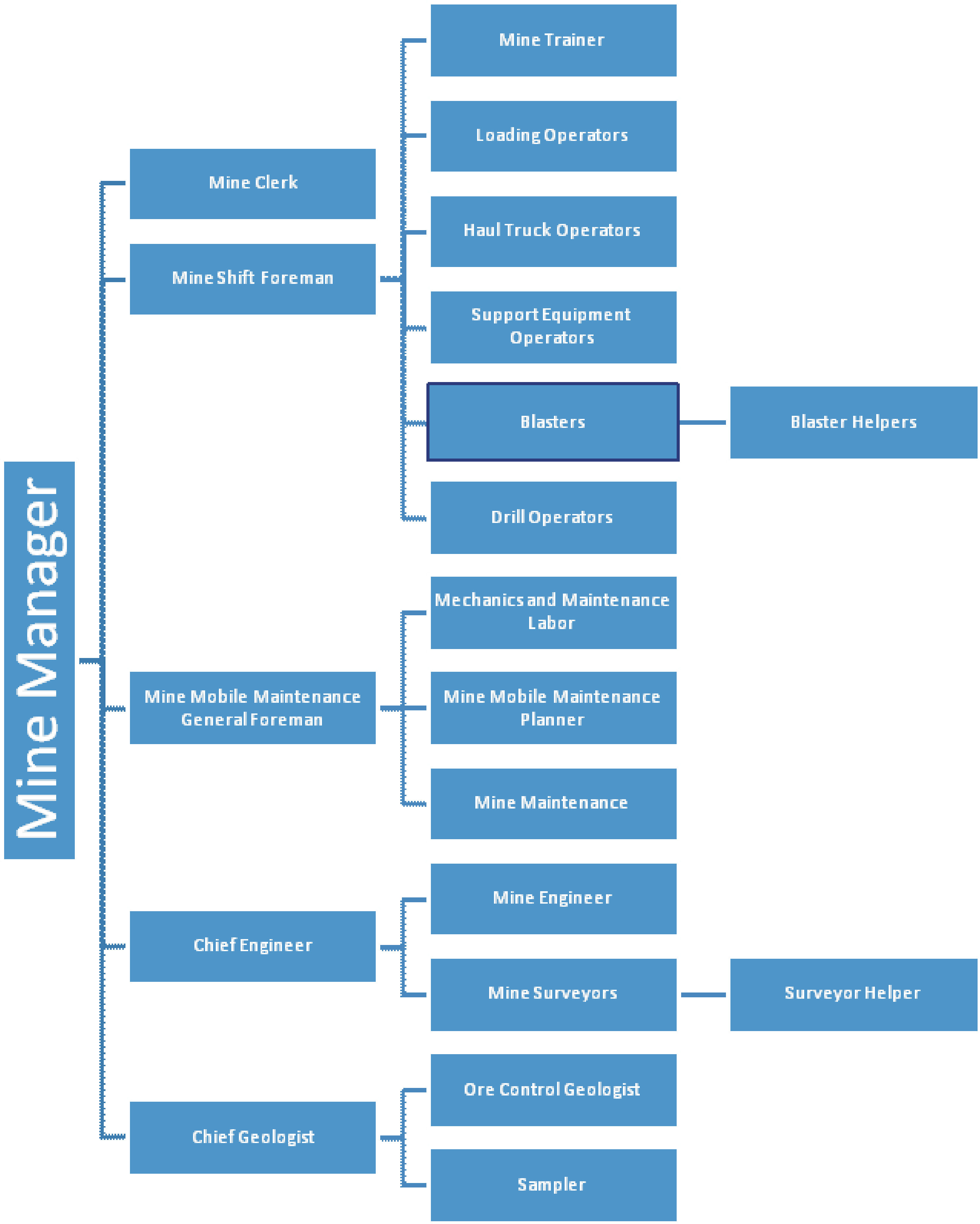
Group	Item	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15
	Total Tonnes Moved	7,370,445	35,789,795	38,631,766	36,396,945	36,227,623	38,608,061	35,691,332	37,859,883	35,867,985	33,726,545	34,012,094	26,259,486	19,318,441	13,115,445	10,671,698	775,387
Mine Schedule – Standby Time	Days per Period	245	366	365	365	365	366	365	365	365	366	365	365	365	366	365	365
	Holidays per Period	5	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9
	Weather Delays	4	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
	Days per Week	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7
	Shifts per Day	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
	Hrs per Shift	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12
	Scheduled Hrs / Period	5,664	8,424	8,400	8,400	8,400	8,424	8,400	8,400	8,400	8,424	8,400	8,400	8,400	8,424	8,400	8,400
Mine Schedule – Delays / Efficiency	Lunch Breaks	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
	Shift Startup / Shutdown	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
	Breaks	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
	Safety / Training Hrs/Shift	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25
	Misc - Blast & Move	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Operator Hours after Misc	4,838	7,196	7,175	7,175	7,175	7,196	7,175	7,175	7,175	7,196	7,175	7,175	7,175	7,196	7,175	7,175
	Gross Operator Efficiency	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
Total Cat785 Fleet	Average Number of Trucks	-	3	4	4	4	4	4	4	4	4	4	4	-	-	-	-
	Truck Fleet Availability	0%	89%	88%	87%	86%	86%	85%	85%	85%	85%	85%	85%	0%	0%	0%	0%
	Available Truck Operating Hrs	-	19,212	25,256	24,969	24,682	24,753	24,395	24,395	24,395	24,465	24,395	24,395	-	-	-	-
	Productive Truck Hrs Used	-	15,568	20,752	20,451	20,383	15,510	20,145	20,148	20,149	20,205	20,145	20,147	-	-	-	-
	Operating Efficiency	0%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	0%	0%	0%	0%
	Truck Operating Hrs Used	-	18,757	25,002	24,640	24,557	18,687	24,271	24,274	24,277	24,343	24,271	24,273	-	-	-	-
	Remaining Operating Hrs	-	455	254	329	125	6,066	124	121	118	121	124	122	-	-	-	-
	Use of Operating Hours	0%	98%	99%	99%	99%	75%	99%	100%	100%	100%	99%	100%	0%	0%	0%	0%
Total Cat789 Fleet	Average Number of Trucks	9	12	13	13	13	16	16	16	16	16	16	16	15	11	3	3
	Truck Fleet Availability	90%	90%	89%	88%	87%	87%	86%	86%	86%	86%	86%	86%	86%	86%	88%	88%
	Available Truck Operating Hrs	39,188	77,711	83,015	82,082	81,149	99,874	98,656	98,656	98,656	98,938	98,441	98,226	92,127	67,926	18,942	18,942
	Productive Truck Hrs Used	12,012	52,978	67,244	67,466	66,610	80,596	80,796	76,792	70,913	80,338	81,303	80,946	65,892	46,428	14,286	553
	Operating Efficiency	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%
	Truck Operating Hrs Used	14,472	63,828	81,017	81,284	80,253	97,103	97,345	92,521	85,438	96,793	97,955	97,526	79,388	55,938	17,211	667
	Remaining Operating Hrs	24,716	13,883	1,998	798	897	2,770	1,311	6,136	13,218	2,145	486	700	12,739	11,988	1,731	18,275
	Use of Operating Hours	37%	82%	98%	99%	99%	97%	99%	94%	87%	98%	100%	99%	86%	82%	91%	4%
Total Loader Fleet	Excavator Availability	0%	90%	89%	88%	87%	86%	85%	85%	85%	85%	85%	85%	0%	0%	0%	0%
	Available Excav Operating Hrs	-	6,476	6,386	6,314	6,242	6,188	6,099	6,099	6,099	6,116	6,099	6,099	-	-	-	-
	Productive Excav Hrs Used	-	4,443	5,071	4,597	4,489	5,131	4,996	4,808	3,312	4,691	3,503	2,667	-	-	-	-
	Operating Efficiency	0%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	0%	0%	0%	0%
	Excav Operating Hrs Used	-	4,937	5,698	5,223	5,159	5,966	5,877	5,657	3,897	5,519	4,121	3,138	-	-	-	-
	Remaining Operating Hrs	-	1,539	688	1,091	1,083	222	221	442	2,202	597	1,977	2,961	-	-	-	-
	Use of Operating Hours	0%	76%	89%	83%	83%	96%	96%	93%	64%	90%	68%	51%	0%	0%	0%	0%
Total Excavator Fleet	Excavator Availability	90%	90%	89%	88%	87%	86%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
	Available Excav Operating Hrs	8,708	12,921	12,738	12,590	12,413	12,304	12,198	12,198	12,198	12,232	12,198	12,198	12,198	6,116	6,099	6,099
	Productive Excav Hrs Used	2,376	9,594	10,237	9,724	9,717	10,203	9,322	10,103	10,115	8,822	9,433	7,299	6,227	4,228	3,440	250
	Operating Efficiency	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%
	Excav Operating Hrs Used	2,640	10,689	11,534	11,086	11,229	11,934	10,967	11,886	11,900	10,379	11,098	8,587	7,326	4,974	4,047	294
	Remaining Operating Hrs	6,069	2,232	1,204	1,504	1,183	370	1,230	311	298	1,853	1,100	3,610	4,871	1,142	2,052	5,805
	Use of Operating Hours	30%	83%	91%	88%	90%	97%	90%	97%	98%	85%	91%	70%	60%	81%	66%	5%

19.1.5 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 are based on the people required for supervision and support of mine production (including maintenance). The mine-staff organizational chart is shown in FIGURE 19-2. The estimated number of mine personnel required to execute the mine plan is shown in TABLE 19-8.

Salaries for each position were estimated based information received from Tetra Tech. Salaries include an allowance for benefits at a rate of 25 percent of the base salary for each position. The salaries used are shown in TABLE 19-9 presented in both Australian and US dollars. The extended cost for labor by year is shown in thousands of US dollars in TABLE 19-10. 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 was allocated to construction of tailings facilities and reclamation for capping of TSF1. These costs are reflected in the capital and reclamation costs and are not included in TABLE 19-10.

All dollars are presented in US dollars unless otherwise noted.



(from MDA 2011)

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.***	Fig19-2.cdr	**Figure 19-2** **Mine Organizational Chart**
	Project: Mt. Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	

TABLE 19-8: MINE PERSONNEL REQUIREMENTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Mine Overhead	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15
Mine Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Mine Clerk	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Mine Shift Foremen	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-
Mine Trainer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Blaster	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	-
Blaster's Helper	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	-
Mine Production																
Loading Operators	6	12	12	12	12	12	12	12	12	12	12	8	6	4	4	4
Haul Truck Operators	36	60	68	68	68	80	80	80	80	80	80	80	56	40	12	12
Drill Operators	4	16	16	16	16	16	16	16	16	16	16	12	10	8	4	-
Support Equipment Operators	12	20	20	20	20	20	20	20	20	20	20	20	16	12	12	4
Total Mine Operating	71	121	129	129	129	141	141	141	141	141	141	133	101	77	45	20
Mine Maintenance																
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Light Vehicle Mechanics	1	2	2	2	2	2	2	2	1	1	1	1	1	1	1	-
Mobile Equipment Mechanics	22	39	43	43	43	47	47	47	47	47	47	44	33	24	11	9
Mobile Equipment Welders	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	-
Mobile Equipment Servicemen	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Tiremen	1	2	2	2	2	2	2	2	1	1	1	1	1	1	1	-
Shop Laborers	2	4	4	4	4	4	4	4	2	2	2	2	2	2	2	-
Maintenance Planner	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Service, Fuel, & Lube	4	8	8	8	8	8	8	8	4	4	4	4	4	4	4	-
Total Mine Maintenance	36	61	65	65	65	69	69	69	61	61	61	58	47	38	25	9
Engineering																
Chief Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Mine Surveyors	1	2	2	2	2	2	2	2	2	2	2	2	1	1	1	-
Surveyor Helper	1	2	2	2	2	2	2	2	2	2	2	2	1	1	1	-
Mine Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Total Engineering	4	6	6	6	6	6	6	6	6	6	6	6	4	4	4	-
Mine Geology																
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Ore Control Geologist	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	-
Sampler	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	-
Total Geology	5	5	5	5	5	5	5	5	5	5	5	5	3	3	3	-
Total Mine Operations Workforce																
Mine Operations	71	121	129	129	129	141	141	141	141	141	141	133	101	77	45	20
Mine Maintenance	36	61	65	65	65	69	69	69	61	61	61	58	47	38	25	9
Engineering	4	6	6	6	6	6	6	6	6	6	6	6	4	4	4	-
Geology	5	5	5	5	5	5	5	5	5	5	5	5	3	3	3	-
Total	116	193	205	205	205	221	221	221	213	213	213	202	155	122	77	29

TABLE 19-9: MINE PERSONNEL SALARY RATES
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

	AU $/Year			US $/Year		
Mine Overhead	Labor Rates	Benefits 25%	Total Rate	Labor Rates	Benefits 25%	Total Rate
Mine Manager	$ 210,000	$ 52,500	$ 262,500	$ 178,500	$ 44,600	$ 223,100
Mine Clerk	$ 60,000	$ 15,000	$ 75,000	$ 51,000	$ 12,800	$ 63,800
Mine Shift Foremen	$ 100,000	$ 25,000	$ 125,000	$ 85,000	$ 21,300	$ 106,300
Mine Trainer	$ 80,000	$ 20,000	$ 100,000	$ 68,000	$ 17,000	$ 85,000
Blaster	$ 90,000	$ 22,500	$ 112,500	$ 76,500	$ 19,100	$ 95,600
Blaster's Helper	$ 75,000	$ 18,750	$ 93,750	$ 63,800	$ 15,900	$ 79,700
Mine Production						
Loading Operators	$ 90,000	$ 22,500	$ 112,500	$ 76,500	$ 19,100	$ 95,600
Haul Truck Operators	$ 80,000	$ 20,000	$ 100,000	$ 68,000	$ 17,000	$ 85,000
Drill Operators	$ 90,000	$ 22,500	$ 112,500	$ 76,500	$ 19,100	$ 95,600
Support Equipment Operators	$ 80,000	$ 20,000	$ 100,000	$ 68,000	$ 17,000	$ 85,000
Mine Maintenance						
Maintenance Superintendent	$ 150,000	$ 37,500	$ 187,500	$ 127,500	$ 31,900	$ 159,400
Light Vehicle Mechanics	$ 75,000	$ 18,750	$ 93,750	$ 63,800	$ 15,900	$ 79,700
Mobile Equipment Mechanics	$ 90,000	$ 22,500	$ 112,500	$ 76,500	$ 19,100	$ 95,600
Mobile Equipment Welders	$ 75,000	$ 18,750	$ 93,750	$ 63,800	$ 15,900	$ 79,700
Mobile Equipment Servicemen	$ 75,000	$ 18,750	$ 93,750	$ 63,800	$ 15,900	$ 79,700
Tiremen	$ 75,000	$ 18,750	$ 93,750	$ 63,800	$ 15,900	$ 79,700
Shop Laborers	$ 65,000	$ 16,250	$ 81,250	$ 55,300	$ 13,800	$ 69,100
Maintenance Planner	$ 75,000	$ 18,750	$ 93,750	$ 63,800	$ 15,900	$ 79,700
Service, Fuel, & Lube	$ 75,000	$ 18,750	$ 93,750	$ 63,800	$ 15,900	$ 79,700
Engineering						
Chief Engineer	$ 150,000	$ 37,500	$ 187,500	$ 127,500	$ 31,900	$ 159,400
Mine Surveyors	$ 100,000	$ 25,000	$ 125,000	$ 85,000	$ 21,300	$ 106,300
Surveyor Helper	$ 65,000	$ 16,250	$ 81,250	$ 55,300	$ 13,800	$ 69,100
Mine Engineer	$ 125,000	$ 31,250	$ 156,250	$ 106,300	$ 26,600	$ 132,900
Mine Geology						
Chief Geologist	$ 150,000	$ 37,500	$ 187,500	$ 127,500	$ 31,900	$ 159,400
Ore Control Geologist	$ 100,000	$ 25,000	$ 125,000	$ 85,000	$ 21,300	$ 106,300
Sampler	$ 75,000	$ 18,750	$ 93,750	$ 63,800	$ 15,900	$ 79,700

TABLE 19-10: MINE ANNUAL PERSONNEL COSTS (000)
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Mine Overhead	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Total
Mine Manager	$ 134	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 38	$ -	$ 3,072
Mine Clerk	$ 38	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 11	$ -	$ 879
Mine Shift Foremen	$ 255	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 72	$ -	$ 5,855
Mine Trainer	$ 51	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 14	$ -	$ 1,170
Blaster	$ 115	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 33	$ -	$ 2,633
Blaster's Helper	$ 96	$ 319	$ 310	$ 314	$ 314	$ 232	$ 162	$ 319	$ 220	$ 106	$ 319	$ 50	$ 312	$ 319	$ 54	$ -	$ 3,445
Mine Production																	
Loading Operators	$ 169	$ 1,147	$ 1,116	$ 1,131	$ 1,130	$ 857	$ 624	$ 1,147	$ 816	$ 436	$ 1,147	$ 119	$ 561	$ 382	$ 382	$ 28	$ 11,193
Haul Truck Operators	$ 685	$ 5,100	$ 5,601	$ 5,701	$ 5,695	$ 5,099	$ 3,750	$ 6,800	$ 4,853	$ 2,096	$ 6,800	$ 388	$ 4,629	$ 3,400	$ 1,020	$ 74	$ 61,691
Drill Operators	$ 108	$ 1,530	$ 1,488	$ 1,508	$ 1,507	$ 1,142	$ 830	$ 1,530	$ 1,087	$ 579	$ 1,530	$ 249	$ 936	$ 765	$ 65	$ -	$ 14,853
Support Equipment Operators	$ 387	$ 1,700	$ 1,664	$ 1,681	$ 1,680	$ 1,362	$ 1,090	$ 1,700	$ 1,314	$ 871	$ 1,700	$ 653	$ 1,337	$ 1,020	$ 1,020	$ 510	$ 19,690
Total Mine Operating	$ 2,038	$ 10,784	$ 11,168	$ 11,324	$ 11,314	$ 9,679	$ 7,444	$ 12,484	$ 9,277	$ 5,077	$ 12,484	$ 2,448	$ 8,763	$ 6,874	$ 2,710	$ 612	$ 124,480
Mine Maintenance																	
Maintenance Superintendent	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 27	$ -	$ 2,195
Light Vehicle Mechanics	$ 48	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 14	$ -	$ 1,655
Mobile Equipment Mechanics	$ 1,262	$ 3,728	$ 4,111	$ 4,111	$ 4,111	$ 4,493	$ 4,493	$ 4,493	$ 4,493	$ 4,493	$ 4,493	$ 4,206	$ 3,155	$ 2,294	$ 893	$ 144	$ 54,975
Mobile Equipment Welders	$ 143	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 173	$ -	$ 3,425
Mobile Equipment Servicemen	$ 48	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ -	$ 1,164
Tiremen	$ 48	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 14	$ -	$ 1,655
Shop Laborers	$ 83	$ 276	$ 276	$ 276	$ 276	$ 276	$ 276	$ 276	$ 138	$ 138	$ 138	$ 138	$ 138	$ 138	$ 24	$ -	$ 2,870
Maintenance Planner	$ 48	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 14	$ -	$ 1,098
Service, Fuel, & Lube	$ 191	$ 638	$ 638	$ 638	$ 638	$ 638	$ 638	$ 638	$ 319	$ 319	$ 319	$ 319	$ 319	$ 319	$ 54	$ -	$ 6,622
Total Mine Maintenance	$ 1,967	$ 5,519	$ 5,902	$ 5,902	$ 5,902	$ 6,284	$ 6,284	$ 6,284	$ 5,668	$ 5,668	$ 5,668	$ 5,381	$ 4,329	$ 3,469	$ 1,291	$ 144	$ 75,659
Engineering																	
Chief Engineer	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 27	$ -	$ 2,195
Mine Surveyors	$ 64	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 106	$ 106	$ 18	$ -	$ 2,633
Surveyor Helper	$ 41	$ 138	$ 138	$ 138	$ 138	$ 138	$ 138	$ 138	$ 138	$ 138	$ 138	$ 138	$ 69	$ 69	$ 12	$ -	$ 1,712
Mine Engineer	$ 80	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 23	$ -	$ 1,830
Total Engineering	$ 281	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 468	$ 468	$ 80	$ -	$ 8,370
Mine Geology																	
Chief Geologist	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 27	$ -	$ 2,195
Ore Control Geologist	$ 128	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 106	$ 106	$ 18	$ -	$ 2,697
Sampler	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ 80	$ 14	$ -	$ 2,022
Total Geology	$ 319	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 345	$ 345	$ 59	$ -	$ 6,914
Total Mine Operations Workforce																	
Mine Operations	$ 2,038	$ 10,784	$ 11,168	$ 11,324	$ 11,314	$ 9,679	$ 7,444	$ 12,484	$ 9,277	$ 5,077	$ 12,484	$ 2,448	$ 8,763	$ 6,874	$ 2,710	$ 612	$ 124,480
Mine Maintenance	$ 1,967	$ 5,519	$ 5,902	$ 5,902	$ 5,902	$ 6,284	$ 6,284	$ 6,284	$ 5,668	$ 5,668	$ 5,668	$ 5,381	$ 4,329	$ 3,469	$ 1,291	$ 144	$ 75,659
Engineering	$ 281	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 468	$ 468	$ 80	$ -	$ 8,370
Geology	$ 319	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 345	$ 345	$ 59	$ -	$ 6,914
Total	$ 4,604	$ 17,478	$ 18,244	$ 18,400	$ 18,390	$ 17,137	$ 14,902	$ 19,942	$ 16,119	$ 11,919	$ 19,326	$ 9,003	$ 13,905	$ 11,156	$ 4,140	$ 756	$ 215,423

19.2 Estimated Mine Capital Costs

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 19-11 shows the estimated mine capital requirements by year. Total life-of-mine capital is $134.8 million plus contingency. The initial mine capital is estimated to be $72.2 million (total of year -2 and year -1). This does not include pre-stripping capital of $9.4 million based on the mine operating cost for year -1 and tailings construction costs of $4.2 million. Sustaining capital is estimated to be $62.5 million for mining operations. Tailings construction sustaining capital is estimated to be $150.4 million, which is the cost of mining and haulage of material to TSF1 and TSF2 for construction purposes. Details for the capital expenditure estimates are given in the following sections. No contingency is included in the mine capital costs above.

TABLE 19-11: MINE ANNUAL CAPITAL COSTS (000)
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Total
Primary Mining Equipment																	
Atlas Copco PV235	$ 5,670	$ 5,670	$ -	$ -	$ 2,835	$ 2,835	$ 2,835	$ -	$ -	$ 2,835	$ 2,835	$ 2,835	$ -	$ -	$ -	$ -	$ 28,350
21cm Hyd. Shovel	$ 15,392	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 15,392
12cm FEL	$ -	$ 1,540	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,540
180t Haul Truck	$ 25,421	$ 8,474	$ 2,825	$ -	$ -	$ 8,474	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 45,194
140t Haul Truck	$ -	$ 6,237	$ 2,079	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 8,316
Support Equipment																	
300 Kw Dozer (D9)	$ 1,671	$ -	$ -	$ -	$ -	$ 1,671	$ -	$ -	$ -	$ -	$ 1,671	$ -	$ -	$ -	$ -	$ -	$ 5,013
230 Kw Dozer (D8)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
4.9 m Motor Grader (16H)	$ 727	$ -	$ -	$ -	$ -	$ 727	$ -	$ -	$ -	$ -	$ 727	$ -	$ -	$ -	$ -	$ -	$ 2,181
Water Truck - 45,000 Liter	$ 724	$ -	$ -	$ -	$ -	$ 724	$ -	$ -	$ -	$ -	$ 724	$ -	$ -	$ -	$ -	$ -	$ 2,172
RTD Dozer (834H)	$ 818	$ -	$ -	$ -	$ -	$ 818	$ -	$ -	$ -	$ -	$ 818	$ -	$ -	$ -	$ -	$ -	$ 2,454
Rock Breaker - Impact Hammer	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Backhoe/Loader (1.5 cu m)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Pit Pumps	$ 68	$ -	$ -	$ -	$ -	$ 68	$ -	$ -	$ -	$ -	$ 68	$ -	$ -	$ -	$ -	$ -	$ 204
36 ton Crane	$ 330	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 330
2 cm excavator	$ 178	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 178
Low Boy	$ 1,231	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,231
Flatbed	$ 52	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 52
Blasting																	
Sanding/Stemming Truck	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Explosives Truck	$ 187	$ -	$ -	$ -	$ -	$ 187	$ -	$ -	$ -	$ -	$ 187	$ -	$ -	$ -	$ -	$ -	$ 561
Skid Loader	$ 39	$ -	$ -	$ -	$ -	$ 39	$ -	$ -	$ -	$ -	$ 39	$ -	$ -	$ -	$ -	$ -	$ 117
Mine Maintenance																	
Lube/Fuel Truck	$ 193	$ -	$ -	$ -	$ -	$ 193	$ -	$ -	$ -	$ -	$ 193	$ -	$ -	$ -	$ -	$ -	$ 579
Mechanics Truck	$ 187	$ -	$ -	$ -	$ -	$ 187	$ -	$ -	$ -	$ -	$ 187	$ -	$ -	$ -	$ -	$ -	$ 561
Forklift	$ 137	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 137
Other Mine Capital																	
Light Plant	$ 54	$ -	$ 27	$ -	$ 27	$ -	$ 27	$ -	$ 27	$ -	$ 27	$ -	$ 27	$ -	$ -	$ -	$ 216
ANFO Storage Bins	$ 77	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 77
Powder Magazines	$ 9	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 9
Cap Magazine	$ 6	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 6
Mobile Radios	$ 27	$ 9	$ 2	$ -	$ 1	$ 9	$ 1	$ -	$ -	$ 1	$ 6	$ 1	$ -	$ -	$ -	$ -	$ 57
Shop Equipment	$ 263	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 263
Engineering & Office Equipment	$ 150	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 150
Water Storage (Dust Suppression)	$ 98	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 98
Base Radio & GPS Stations	$ 105	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 105
Unspecified Miscellaneous Equipment	$ 105	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 105
Fuel Facilities	$ 250	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 250
Shop Building	$ 1,500	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,500
Access Roads - Haul Roads - Site Work	$ 100	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 100
Ambulance & Fire Equipment	$ 150	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 150
Light Vehicles	$ 468	$ -	$ -	$ -	$ 386	$ -	$ -	$ -	$ 386	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,240
Total Mining Capital	$ 56,387	$ 21,930	$ 4,933	$ -	$ 3,249	$ 15,932	$ 2,863	$ -	$ 413	$ 2,836	$ 7,482	$ 2,836	$ 27	$ -	$ -	$ -	$ 118,888
Pre-Mining Cost	$ 9,394	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 9,394
Tailings Construction Cost	$ 4,193	$ -	$ 1,057	$ 496	$ 527	$ 9,485	$ 17,240	$ -	$ -	$ 24,818	$ -	$ 30,127	$ 614	$ -	$ -	$ -	$ 88,555
Tails Reclamation Cost (Capping Material)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 10,714	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 10,714
Total Capitalized Mining Costs	$ 13,587	$ -	$ 1,057	$ 496	$ 527	$ 9,485	$ 17,240	$ -	$ 10,714	$ 24,818	$ -	$ 30,127	$ 614	$ -	$ -	$ -	$ 108,663

19.2.1 Major Mining Equipment

Capital for major mining equipment is shown in TABLE 19-11 and discussed in the following subsections.

19.2.2 Drilling and Blasting

Drilling equipment capital is based on equipment quotations for a total of four Atlas Copco Pit Viper 235 blast-hole drills. Two of the drills will be purchased at the start of mining in year -1 with an additional two drills purchased in year 1 at a cost of $2,835,000 each (including shipping and commissioning). Replacement drills are purchased in years 4, 5, 6, 9, 10, and 11. As these drills are replaced, some of the older units may be used sparingly to augment the fleet production requirements as needed. The cost of the drills was provided by EMG LLC.

Blasting operations require the use of a truck (ANFO truck) to deliver bulk explosive to the hole ($187,000 US) and a skid loader ($39,000 US) to help with stemming of holes. Additional blasting capital includes ANFO/Emulsion storage bins ($77,000 US), powder magazines ($9,000 US), and a cap magazine ($6,000 US). Sustaining capital includes replacement of the ANFO truck and the skid loader in years 5 and 10.

19.2.3 Loading

Capital costs for loading equipment have been quoted by EMG LLC and include two Hitachi EX3600 hydraulic shovels and one Caterpillar 992 Loader. The two hydraulic shovels would be purchased in year -1 at the start of mining at an estimated cost of $7,696,000 each (including shipping and commissioning). The Caterpillar 992 loader would be purchased in year one at a cost of $1,540,000.

19.2.4 Haulage

Both 180-tonne and 140-tonne capacity trucks are used in the production schedule. The 180-tonne trucks were quoted by EMG LLC as Caterpillar 789C trucks. A total of 13 trucks would be purchased starting with nine in year -1, three in year 1, and one in year 3. The cost of the 789C trucks is estimated to be $2,824,500 each. Sustaining capital includes an additional three trucks that are to be purchased in year 5 as haulage requirements increase due to depth of mining and the extra distance required due to tailings construction requirements at TSF2.

The 140-tonne trucks have been quoted using Caterpillar 785C trucks. A total of three trucks is to be purchased in year 1 with an additional truck to be purchased in year 2 at an estimated cost of $2,079,000 each, based on quotations received from EMG LLC.

19.2.5 Mine Support

Capital estimates for mine support equipment include freight and erection. The support equipment considers initial support equipment to be purchased in year -1 as follows:

- Two Caterpillar D9 track dozers ($835,120 each quoted by EMG LLC);
- One Caterpillar 16H motor grader ($726,600 quoted by EMG LLC);
- One Caterpillar 773B with a 45K liter water truck ($723,060 quoted by EMG LLC);
- One Caterpillar 834H rubber tire dozer ($817,400 quoted by EMG LLC);
- One 36 tonne capacity crane ($329,600 quoted by EMG LLC);
- One Caterpillar 321DL excavator ($177,345 quoted by EMG LLC);

- One low-boy trailer complete with a used 60t haul truck to tow it ($1,230,900);
- One flatbed truck ($51,450);
- Two pit pumps ($33,690 each);
- One rock breaker to be attached to the 321DL excavator as needed ($30,975); and
- Four light plants ($13,423).

19.2.6 Mine Maintenance

Capital for mine maintenance equipment includes a fuel/lube truck ($192,610) and a mechanics truck ($187,000). Shop facilities are estimated at $1,500,000, and an additional $262,500 is included for shop equipment.

19.2.7 Mine Facilities

Mine facility capital includes $250,000 for fuel facilities and $100,000 for access and haul roads.

19.2.8 Light Vehicles

Capital for light vehicles is shown in Table 19-12.

TABLE 19-12:MINE LIGHT VEHICLE INITIAL CAPITAL
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Mine Department	Vehicle Type	Quantity	Unit Cost	Ext. Cost
Mine Superintendent	3/4 ton 4wd Pickup	1	33,250	33,250
Shift Foreman	4wd Pickup	2	33,250	66,500
Trainer	4wd Pickup	1	27,550	27,550
Blasting	4wd Pickup	1	31,350	31,350
Blasting	1 ton 4wd Pickup	1	27,550	27,550
Crew Vans	3/4 ton Passenger Van	2	33,250	66,500
Engineering				
Chief Engineer	4wd Pickup	1	33,250	33,250
Survey	4wd Pickup	1	33,250	33,250
Geology				
Chief Geologist	4wd Pickup	1	33,250	33,250
Ore Control	4wd Pickup	1	27,550	27,550
Mine Maintenance				
Maintenance Superintendent	4wd Pickup	1	33,250	33,250
Mechanics / Labor	4wd Pickup	2	27,550	55,100
Total		15		468,350

19.2.9 Other Mine Capital

Other miscellaneous capital includes mobile radios for mobile equipment ($1,000 per unit), ambulance and fire equipment ($150,000), engineering and office equipment ($150,000 US), water storage for dust suppression ($98,000), and other unspecified miscellaneous equipment ($105,000).

19.3 Estimated Mine Operating Costs

Annual mine operating costs have been built up based on estimated personnel requirements and equipment hourly costs. Table 19-13 summarizes the annual mine operating costs. The costs are provided based on functionality (mine general services, mine maintenance, engineering, geology, drilling, blasting, loading, hauling, and support).

Table 19-13 summarizes the total average mining cost is estimated to be $1.69/t after allocation of capital costs and excluding tonnage mined as capital. The overall mining cost per tonne without capital allocation is $1.63/t mined.

The following subsections describe the operating cost estimate by functionality.

19.3.1 Drilling Costs

The average life-of-mine drilling cost is estimated to be $0.17/t mined or $0.16/t mined before allocation of drilling costs for pre-stripping, tailings construction, and capping material for TSF1. This includes maintenance labor allocated to drill maintenance. Drilling operating costs are provided in TABLE 19-14 before allocations.

19.3.2 Blasting Costs

The average life-of-mine blasting cost is estimated to be $0.31/t mined both before and after allocation of blasting costs for pre-stripping, tailings construction, and capping material for TSF1. Blasting costs before allocations are provided in TABLE 19-15.

19.3.3 Loading Costs

The average life-of-mine loading cost is estimated to be $0.18/t moved or $0.172/t moved before allocation of loading costs for pre-stripping, tailings construction, and capping material for TSF1. This includes the re-handle of ore from stockpiles at the end of the mine life and maintenance labor allocated to loader and shovel maintenance. Loading costs before allocations are provided in TABLE 19-16.

19.3.4 Haulage Costs

The average life-of-mine haulage cost is estimated to be $0.79/t mined or $0.779/t moved before allocation of haulage costs for pre-stripping, tailings construction, and capping material for TSF1. This includes re-handle of stockpiled ore at the end of the mine life and maintenance labor allocated to truck maintenance. Haulage costs are provided in TABLE 19-17 before allocations.

19.3.5 Mine Support Costs

Mine-support costs include the operation of all of the mine-support equipment. The average life-of-mine support cost is estimated to be $0.12/t mined or $0.121/t moved before allocation of support costs for pre-stripping, tailings construction, and capping material for TSF1. This includes support during re-handling of stockpiled ore at the end of the mine life and maintenance labor allocated to drill maintenance. Mine-support costs are provided in TABLE 19-18 before allocation.

19.3.6 Mine Maintenance Costs

The average life-of-mine mine-maintenance cost is estimated to be $0.03/t mined or $0.04/t moved before allocation of support costs for pre-stripping, tailings construction, and capping material for TSF1. Mine-maintenance costs for personnel and shop supplies are provided in

TABLE 19-19 before allocations. Note that the maintenance wages for mechanics has been included in the operating cost for equipment. Thus, the maintenance costs provided in TABLE 19-19 do not include the labor directly attributed to equipment maintenance.

19.3.7 Mine General Services Costs

The average life-of-mine general services cost is estimated to be \$0.08/t mined or \$0.064/t moved before allocation of support costs for pre-stripping. Mine general services costs are provided in TABLE 19-20 before allocation and include costs for mine supervision, engineering, geology, light vehicles, and supplies.

TABLE 19-13: ANNUAL MINE OPERATING COSTS ($000)
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Mine Operating Cost	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Total
Mine General Service	K USD	$ -	$ 912	$ 912	$ 912	$ 912	$ 912	$ 912	$ 912	$ 912	$ 912	$ 912	$ 912	$ 863	$ 863	$ 147	$ -	$ 11,902
Mine Maintenance	K USD	$ -	$ 1,674	$ 1,674	$ 1,674	$ 1,674	$ 1,674	$ 1,674	$ 1,674	$ 1,057	$ 1,058	$ 1,057	$ 1,057	$ 1,049	$ 1,049	$ 179	$ -	$ 18,222
Engineering	K USD	$ -	$ 668	$ 668	$ 668	$ 668	$ 668	$ 668	$ 668	$ 668	$ 668	$ 668	$ 668	$ 492	$ 492	$ 84	$ -	$ 8,414
Geology	K USD	$ -	$ 556	$ 556	$ 556	$ 556	$ 556	$ 556	$ 556	$ 556	$ 556	$ 556	$ 556	$ 370	$ 370	$ 63	$ -	$ 6,920
Drilling	K USD	$ -	$ 5,748	$ 5,978	$ 5,814	$ 5,794	$ 4,675	$ 3,849	$ 6,013	$ 4,686	$ 3,147	$ 5,633	$ 2,032	$ 3,278	$ 2,520	$ 332	$ -	$ 59,498
Blasting	K USD	$ -	$ 10,746	$ 11,528	$ 10,994	$ 10,939	$ 8,641	$ 7,056	$ 11,518	$ 8,818	$ 5,672	$ 10,412	$ 3,815	$ 5,981	$ 4,403	$ 750	$ -	$ 111,275
Loading	K USD	$ -	$ 6,117	$ 6,087	$ 6,051	$ 5,821	$ 5,133	$ 4,256	$ 6,105	$ 4,709	$ 3,555	$ 5,878	$ 2,486	$ 4,432	$ 2,121	$ 1,847	$ 129	$ 64,725
Hauling	K USD	$ -	$ 20,561	$ 24,720	$ 25,028	$ 24,826	$ 23,611	$ 20,043	$ 28,524	$ 21,787	$ 14,960	$ 29,440	$ 10,622	$ 20,126	$ 14,520	$ 4,559	$ 221	$ 283,547
Mine Support	K USD	$ -	$ 3,901	$ 3,825	$ 3,859	$ 3,857	$ 3,237	$ 2,699	$ 3,896	$ 3,138	$ 2,274	$ 3,896	$ 1,844	$ 3,134	$ 2,746	$ 1,786	$ 795	$ 44,886
Total Mine Cost	K USD	$ -	$ 50,882	$ 55,947	$ 55,555	$ 55,046	$ 49,107	$ 41,713	$ 59,865	$ 46,330	$ 32,800	$ 58,451	$ 23,991	$ 39,725	$ 29,086	$ 9,747	$ 1,145	$ 609,389
Capitalized Pre-Stripping Cost	K USD	$ 9,394	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 9,394
Mine Cost per Tonne Mined																		
Mine General Service	$/t	$ -	$ 0.03	$ 0.02	$ 0.03	$ 0.03	$ 0.03	$ 0.04	$ 0.02	$ 0.03	$ 0.05	$ 0.03	$ 0.07	$ 0.05	$ 0.07	$ 0.07	$ -	$ 0.03
Mine Maintenance	$/t	$ -	$ 0.05	$ 0.04	$ 0.05	$ 0.05	$ 0.06	$ 0.07	$ 0.04	$ 0.04	$ 0.06	$ 0.03	$ 0.09	$ 0.06	$ 0.08	$ 0.08	$ -	$ 0.05
Engineering	$/t	$ -	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ 0.02	$ 0.02	$ 0.04	$ 0.02	$ 0.05	$ 0.03	$ 0.04	$ 0.04	$ -	$ 0.02
Geology	$/t	$ -	$ 0.02	$ 0.01	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.01	$ 0.02	$ 0.03	$ 0.02	$ 0.04	$ 0.02	$ 0.03	$ 0.03	$ -	$ 0.02
Drilling	$/t	$ -	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.17	$ 0.17	$ 0.16	$ 0.16	$ 0.17	$ 0.17	$ 0.16	$ 0.17	$ 0.19	$ 0.15	$ -	$ 0.17
Blasting	$/t	$ -	$ 0.31	$ 0.30	$ 0.30	$ 0.30	$ 0.31	$ 0.31	$ 0.30	$ 0.31	$ 0.31	$ 0.31	$ 0.31	$ 0.31	$ 0.34	$ 0.34	$ -	$ 0.31
Loading	$/t	$ -	$ 0.17	$ 0.16	$ 0.17	$ 0.16	$ 0.18	$ 0.18	$ 0.16	$ 0.16	$ 0.19	$ 0.17	$ 0.20	$ 0.23	$ 0.16	$ 0.83	$ -	$ 0.18
Hauling	$/t	$ -	$ 0.58	$ 0.65	$ 0.69	$ 0.69	$ 0.84	$ 0.87	$ 0.75	$ 0.75	$ 0.81	$ 0.87	$ 0.85	$ 1.06	$ 1.11	$ 2.04	$ -	$ 0.79
Mine Support	$/t	$ -	$ 0.11	$ 0.10	$ 0.11	$ 0.11	$ 0.11	$ 0.12	$ 0.10	$ 0.11	$ 0.12	$ 0.11	$ 0.15	$ 0.16	$ 0.21	$ 0.80	$ -	$ 0.12
Total Mine Cost	$/t	$ -	$ 1.45	$ 1.47	$ 1.54	$ 1.53	$ 1.74	$ 1.81	$ 1.58	$ 1.60	$ 1.78	$ 1.72	$ 1.93	$ 2.09	$ 2.22	$ 4.36	$ -	$ 1.69

TABLE 19-14: ANNUAL DRILLING OPERATING COSTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Drill Requirements	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Total
Number of Holes - Ore	holes	2,212	24,916	27,719	24,482	21,732	12,651	16,682	26,931	22,768	18,344	27,806	24,695	27,301	23,080	4,516	-	305,835
Drill Meters - Ore	meters	17,251	194,346	216,212	190,956	169,513	98,678	130,118	210,066	177,593	143,086	216,885	192,618	212,946	180,020	35,226	-	2,385,515
Production Hours - Ore	hrs	630	7,092	7,890	6,969	6,186	3,601	4,748	7,666	6,481	5,222	7,915	7,029	7,771	6,570	1,286	-	87,057
Operational Efficiency - Ore	%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	0%	
Operating Hrs - Ore	hrs	740.67	8,344.04	9,282.84	8,198.51	7,277.86	4,236.62	5,586.47	9,018.94	7,624.76	6,143.26	9,311.75	8,269.85	9,142.64	7,728.98	1,512.41	-	102,420
Number of Holes - Waste	holes	12,828	46,862	51,113	49,790	52,194	56,778	51,100	50,326	50,424	46,229	41,600	28,891	12,121	3,684	44	-	553,984
Drill Meters - Waste	meters	100,062	365,522	398,680	388,365	407,113	442,872	398,577	392,540	393,309	360,588	324,476	225,347	94,540	28,735	346	-	4,321,072
Production Hours - Waste	hrs	3,652	13,339	14,549	14,173	14,857	16,162	14,546	14,325	14,353	13,159	11,841	8,224	3,450	1,049	13	-	157,693
Operational Efficiency - Waste	%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	0%	
Operating Hrs - Waste	hrs	4,296.06	15,693.31	17,116.90	16,674.03	17,478.97	19,014.25	17,112.48	16,853.32	16,886.30	15,481.45	13,931.05	9,675.06	4,058.98	1,233.70	14.85	-	185,521
Number of Holes - Total	holes	15,040	71,778	78,832	74,272	73,926	69,429	67,781	77,257	73,193	64,574	69,405	53,585	39,421	26,763	4,561	-	859,819
Drill Meters - Total	meters	117,313	559,868	614,892	579,321	576,626	541,550	528,695	602,606	570,902	503,674	541,362	417,965	307,487	208,755	35,572	-	6,706,587
Production Hours - Total	hrs	4,281	20,432	22,440	21,142	21,043	19,763	19,294	21,991	20,834	18,381	19,756	15,253	11,221	7,618	1,298	-	244,749
Operating Hrs - Total	hrs	5,037	24,037	26,400	24,873	24,757	23,251	22,699	25,872	24,511	21,625	23,243	17,945	13,202	8,963	1,527	-	287,940
Drill Availability	%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	0%	
Required Drills - Calculated	#	1.22	3.86	4.26	4.01	3.99	3.74	3.66	4.17	3.95	3.48	3.75	2.89	2.13	1.44	0.25	-	
Required Drills - Rounded	#	2	4	4	4	4	4	4	4	4	4	4	3	3	2	1	0	
Cumulative Hours - Drill #1	hrs	5,037	11,046	17,646	23,864	30,053	35,866	41,541	48,009	54,137	59,543	65,354	71,335	75,736	80,217	81,744	-	
Cumulative Hours - Drill #2	hrs		6,009	12,609	18,827	25,017	30,829	36,504	42,972	49,100	54,506	60,317						
Cumulative Hours - Drill #3	hrs		6,009	12,609	18,827	25,017	30,829	36,504	42,972	49,100								
Operating Costs																		
Fuel Consumption (KL)	KL	355	1,692	1,859	1,751	1,743	1,637	1,598	1,822	1,726	1,522	1,636	1,263	929	631	108	-	20,272
Fuel Cost	K USD	$ 198.58	$ 947.71	$1,040.85	$ 980.64	$ 976.07	$ 916.70	$ 894.94	$1,020.05	$ 966.38	$ 852.59	$ 916.38	$ 707.50	$ 520.49	$ 353.37	$ 60.21	$ -	$ 11,352
Lube & Oil	K USD	$ 58.33	$ 278.35	$ 305.71	$ 288.02	$ 286.68	$ 269.25	$ 262.85	$ 299.60	$ 283.84	$ 250.41	$ 269.15	$ 207.80	$ 152.87	$ 103.79	$ 17.69	$ -	$ 3,334
Undercarriage	K USD	$ 25.18	$ 120.19	$ 132.00	$ 124.36	$ 123.78	$ 116.25	$ 113.49	$ 129.36	$ 122.56	$ 108.12	$ 116.21	$ 89.72	$ 66.01	$ 44.81	$ 7.64	$ -	$ 1,440
Drill Bits & Steel	K USD	$ 262.33	$1,251.95	$1,374.99	$1,295.44	$1,289.42	$1,210.98	$1,182.24	$1,347.51	$1,276.62	$1,126.29	$1,210.56	$ 934.63	$ 687.58	$ 466.81	$ 79.54	$ -	$ 14,997
Total Consumables	K USD	$ 544.42	$2,598.19	$2,853.54	$2,688.47	$2,675.96	$2,513.18	$2,453.53	$2,796.53	$2,649.40	$2,337.41	$2,512.31	$1,939.66	$1,426.96	$ 968.77	$ 165.08	$ -	$ 31,123
Parts / MARC Cost	K USD	$ 182.53	$ 871.11	$ 956.73	$ 901.38	$ 897.19	$ 842.61	$ 822.61	$ 937.61	$ 888.28	$ 783.68	$ 842.32	$ 650.32	$ 478.43	$ 324.81	$ 55.35	$ -	$ 10,435
Maintenance Labor	K USD	$ 162.54	$ 748.90	$ 748.90	$ 748.90	$ 748.90	$ 748.90	$ 748.90	$ 748.90	$ 748.90	$ 748.90	$ 748.90	$ 557.70	$ 462.10	$ 462.10	$ 46.16	$ -	$ 9,180
Total Maintenance Allocation	K USD	$ 345.07	$1,620.01	$1,705.63	$1,650.28	$1,646.09	$1,591.51	$1,571.51	$1,686.51	$1,637.18	$1,532.58	$1,591.22	$1,208.02	$ 940.53	$ 786.91	$ 101.51	$ -	$ 19,615
Operator Wages & Burden	K USD	$ 229.44	$1,529.60	$1,529.60	$1,529.60	$1,529.60	$1,529.60	$1,529.60	$1,529.60	$1,529.60	$1,529.60	$1,529.60	$1,147.20	$ 956.00	$ 764.80	$ 65.16	$ -	$ 18,459
Total Drilling Cost	K USD	$1,118.93	$5,747.81	$6,088.77	$5,868.35	$5,851.65	$5,634.29	$5,554.64	$6,012.64	$5,816.18	$5,399.59	$5,633.13	$4,294.89	$3,323.49	$2,520.48	$ 331.75	$ -	$ 69,197
Drilling Cost per Tonne Mined by Item																		
Fuel Cost	$/t	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ -	$ 0.03
Lube & Oil	$/t	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ -	$ 0.01
Undercarriage	$/t	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ -	$ 0.00
Drill Bits & Steel	$/t	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ -	$ 0.04
Total Consumables	$/t	$ 0.07	$ 0.07	$ 0.07	$ 0.07	$ 0.07	$ 0.07	$ 0.07	$ 0.07	$ 0.07	$ 0.07	$ 0.07	$ 0.07	$ 0.07	$ 0.07	$ 0.07	$ -	$ 0.07
Parts / MARC Cost	$/t	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ -	$ 0.02
Maintenance Labor	$/t	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.04	$ 0.02	$ -	$ 0.02
Total Maintenance Allocation	$/t	$ 0.05	$ 0.05	$ 0.04	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.04	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.06	$ 0.05	$ -	$ 0.05
Operator Wages & Burden	$/t	$ 0.03	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.05	$ 0.04	$ 0.04	$ 0.05	$ 0.04	$ 0.04	$ 0.05	$ 0.06	$ 0.03	$ -	$ 0.04
Total Drilling Cost	$/t	$ 0.15	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.17	$ 0.17	$ 0.16	$ 0.16	$ 0.17	$ 0.17	$ 0.16	$ 0.17	$ 0.19	$ 0.15	$ -	$ 0.16

TABLE 19-15: MINE ANNUAL BLASTING OPERATING COST
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Ore Blasting	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Total
Holes Loaded	holes	2,212	24,916	27,719	24,482	21,732	12,651	16,682	26,931	22,768	18,344	27,806	24,695	27,301	23,080	4,516	-	305,835
Loaded Meters	meters	9,142	102,987	114,574	101,190	89,827	52,291	68,951	111,317	94,109	75,824	114,931	102,071	112,843	95,395	18,667	-	1,264,119
AN Used	tonnes	263	2,960	3,293	2,908	2,582	1,503	1,982	3,199	2,705	2,179	3,303	2,934	3,243	2,742	536	-	36,331
ANFO Cost	K USD	$ 280	$ 3,152	$ 3,507	$ 3,097	$ 2,749	$ 1,601	$ 2,111	$ 3,407	$ 2,881	$ 2,321	$ 3,518	$ 3,124	$ 3,454	$ 2,920	$ 571	$ -	$ 38,693
Fuel Used (KL)	KL	13	145	161	143	127	74	97	157	133	107	162	144	159	134	26	-	1,781
Fuel Cost (000's)	K USD	$ 7	$ 81	$ 90	$ 80	$ 71	$ 41	$ 54	$ 88	$ 74	$ 60	$ 91	$ 81	$ 89	$ 75	$ 15	$ -	$ 997
Blasting Accessory	K USD	$ 25	$ 278	$ 309	$ 273	$ 242	$ 141	$ 186	$ 300	$ 254	$ 204	$ 310	$ 275	$ 304	$ 257	$ 50	$ -	$ 3,407
Blasting Consumables - Ore	K USD	$ 312	$ 3,511	$ 3,906	$ 3,450	$ 3,062	$ 1,783	$ 2,351	$ 3,795	$ 3,208	$ 2,585	$ 3,918	$ 3,480	$ 3,847	$ 3,252	$ 636	$ -	$ 43,097
Waste Blasting																		
Holes Loaded	holes	12,828	46,862	51,113	49,790	52,194	56,778	51,100	50,326	50,424	46,229	41,600	28,891	12,121	3,684	44	-	553,984
Loaded Meters	meters	53,024	193,695	211,266	205,800	215,735	234,684	211,212	208,013	208,420	191,081	171,945	119,415	50,098	15,227	183	-	2,289,799
AN Used (Mt)	tonnes	1,524	5,567	6,072	5,915	6,200	6,745	6,070	5,978	5,990	5,492	4,942	3,432	1,440	438	5	-	65,810
AN Cost (000's)	K USD	$ 1,623	$ 5,929	$ 6,467	$ 6,299	$ 6,603	$ 7,183	$ 6,465	$ 6,367	$ 6,379	$ 5,849	$ 5,263	$ 3,655	$ 1,533	$ 466	$ 6	$ -	$ 70,088
Fuel Used (KL)	KL	75	273	298	290	304	331	298	293	294	269	242	168	71	21	0	-	3,226
Fuel Cost (000's)	K USD	$ 42	$ 153	$ 167	$ 162	$ 170	$ 185	$ 167	$ 164	$ 164	$ 151	$ 136	$ 94	$ 40	$ 12	$ 0	$ -	$ 1,807
Blasting Accessory	K USD	$ 143	$ 522	$ 569	$ 555	$ 581	$ 633	$ 569	$ 561	$ 562	$ 515	$ 463	$ 322	$ 135	$ 41	$ 0	$ -	$ 6,171
Blasting Consumables - Waste	K USD	$ 1,808	$ 6,604	$ 7,203	$ 7,016	$ 7,355	$ 8,001	$ 7,201	$ 7,092	$ 7,106	$ 6,514	$ 5,862	$ 4,071	$ 1,708	$ 519	$ 6	$ -	$ 78,066
Total																		
Holes Loaded	holes	15,040	71,778	78,832	74,272	73,926	69,429	67,781	77,257	73,193	64,574	69,405	53,585	39,421	26,763	4,561	-	859,819
Loaded Meters	meters	62,166	296,682	325,840	306,991	305,562	286,975	280,163	319,330	302,529	266,904	286,875	221,486	162,942	110,622	18,850	-	3,553,918
AN Used	tonnes	1,787	8,527	9,365	8,823	8,782	8,248	8,052	9,178	8,695	7,671	8,245	6,366	4,683	3,179	542	-	102,142
AN Cost	K USD	$ 1,903	$ 9,081	$ 9,974	$ 9,397	$ 9,353	$ 8,784	$ 8,575	$ 9,774	$ 9,260	$ 8,170	$ 8,781	$ 6,779	$ 4,987	$ 3,386	$ 577	$ -	$ 108,781
Fuel Used	KL	88	418	459	433	430	404	395	450	426	376	404	312	230	156	27	-	5,007
Fuel Cost	K USD	$ 49	$ 234	$ 257	$ 242	$ 241	$ 226	$ 221	$ 252	$ 239	$ 211	$ 226	$ 175	$ 129	$ 87	$ 15	$ -	$ 2,804
Blasting Accessory	K USD	$ 168	$ 800	$ 878	$ 827	$ 824	$ 773	$ 755	$ 861	$ 815	$ 719	$ 773	$ 597	$ 439	$ 298	$ 51	$ -	$ 9,578
Total Blasting Consumables	K USD	$ 2,119	$ 10,115	$ 11,109	$ 10,466	$ 10,417	$ 9,784	$ 9,552	$ 10,887	$ 10,314	$ 9,100	$ 9,780	$ 7,551	$ 5,555	$ 3,771	$ 643	$ -	$ 121,163
Wages & Salaries																		
Blaster	K USD	114.72	191.2	191.2	191.2	191.2	191.2	191.2	191.2	191.2	191.2	191.2	191.2	191.2	191.2	32.58073	0	$ 2,633
Blaster's Helper	K USD	191.28	318.8	318.8	318.8	318.8	318.8	318.8	318.8	318.8	318.8	318.8	318.8	318.8	318.8	54.32394	0	$ 4,390
Summary																		
Consumables	K USD	$ 2,119	$ 10,115	$ 11,109	$ 10,466	$ 10,417	$ 9,784	$ 9,552	$ 10,887	$ 10,314	$ 9,100	$ 9,780	$ 7,551	$ 5,555	$ 3,771	$ 643	$ -	$ 121,163
Labor	K USD	$ 306	$ 510	$ 510	$ 510	$ 510	$ 510	$ 510	$ 510	$ 510	$ 510	$ 510	$ 510	$ 510	$ 510	$ 87	$ -	$ 7,023
Equipment	K USD	$ 58	$ 97	$ 97	$ 97	$ 97	$ 97	$ 97	$ 97	$ 97	$ 97	$ 97	$ -	$ -	$ 97	$ 16	$ -	$ 1,137
Outside Services	K USD	$ 15	$ 25	$ 25	$ 25	$ 25	$ 25	$ 25	$ 25	$ 25	$ 25	$ 25			$ 25	$ 4	$ -	$ 294
Total	K USD	$ 2,498	$ 10,746	$ 11,740	$ 11,098	$ 11,049	$ 10,415	$ 10,183	$ 11,518	$ 10,946	$ 9,731	$ 10,412	$ 8,061	$ 6,065	$ 4,403	$ 750	$ -	$ 129,617
Cost per Ton																		
Consumables	$/t	$ 0.29	$ 0.29	$ 0.29	$ 0.29	$ 0.29	$ 0.29	$ 0.29	$ 0.29	$ 0.29	$ 0.29	$ 0.29	$ 0.29	$ 0.29	$ 0.29	$ 0.29	$ -	$ 0.29
Labor	$/t	$ 0.04	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.02	$ 0.01	$ 0.01	$ 0.02	$ 0.01	$ 0.02	$ 0.03	$ 0.04	$ 0.04	$ -	$ 0.02
Equipment	$/t	$ 0.01	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ -	$ -	$ 0.01	$ 0.01	$ -	$ 0.00
Outside Services	$/t	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ -	$ -	$ 0.00	$ 0.00	$ -	$ 0.00
Total	$/t	$ 0.34	$ 0.31	$ 0.30	$ 0.30	$ 0.30	$ 0.31	$ 0.31	$ 0.30	$ 0.31	$ 0.31	$ 0.31	$ 0.31	$ 0.31	$ 0.34	$ 0.34	$ -	$ 0.31

TABLE 19-16: ANNUAL LOADING OPERATING COST
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

3600 Excavators	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Total
Fuel Consumption (KL)	KL	674	3,176	3,148	3,131	2,941	3,103	3,075	3,072	3,075	3,084	3,076	3,076	3,075	1,270	1,033	75	40,084
Fuel Cost	K USD	$ 377	$ 1,778	$ 1,763	$ 1,753	$ 1,647	$ 1,737	$ 1,722	$ 1,720	$ 1,722	$ 1,727	$ 1,722	$ 1,722	$ 1,722	$ 711	$ 579	$ 42	$ 22,447
Lube & Oil	K USD	$ 74	$ 348	$ 345	$ 343	$ 323	$ 340	$ 337	$ 337	$ 337	$ 338	$ 337	$ 337	$ 337	$ 139	$ 113	$ 8	$ 4,398
Undercarriage	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Wear Items & GET	K USD	$ 28	$ 132	$ 131	$ 131	$ 123	$ 129	$ 128	$ 128	$ 128	$ 129	$ 128	$ 128	$ 128	$ 53	$ 43	$ 3	$ 1,671
Total Consumables	K USD	$ 479	$ 2,259	$ 2,240	$ 2,227	$ 2,092	$ 2,207	$ 2,188	$ 2,185	$ 2,188	$ 2,194	$ 2,188	$ 2,188	$ 2,188	$ 903	$ 735	$ 53	$ 28,515
Parts / MARC Cost	K USD	$ 299	$ 1,411	$ 1,399	$ 1,391	$ 1,307	$ 1,378	$ 1,366	$ 1,365	$ 1,366	$ 1,370	$ 1,366	$ 1,366	$ 1,366	$ 564	$ 459	$ 33	$ 17,807
Total Maint. Allocation (no labor)	K USD	$ 779	$ 3,670	$ 3,638	$ 3,618	$ 3,399	$ 3,586	$ 3,554	$ 3,550	$ 3,554	$ 3,564	$ 3,554	$ 3,554	$ 3,554	$ 1,467	$ 1,194	$ 87	$ 46,322
992 Loaders																		
Fuel Consumption (KL)	KL	-	503	580	532	525	608	599	576	397	562	420	320	-	-	-	-	5,622
Fuel Cost	K USD	$ -	$ 282	$ 325	$ 298	$ 294	$ 340	$ 335	$ 323	$ 222	$ 315	$ 235	$ 179	$ -	$ -	$ -	$ -	$ 3,148
Lube & Oil	K USD	$ -	$ 66	$ 77	$ 70	$ 69	$ 80	$ 79	$ 76	$ 52	$ 74	$ 55	$ 42	$ -	$ -	$ -	$ -	$ 743
Tires	K USD	$ -	$ 229	$ 264	$ 242	$ 239	$ 276	$ 272	$ 262	$ 180	$ 255	$ 191	$ 145	$ -	$ -	$ -	$ -	$ 2,555
Wear Items & GET	K USD	$ -	$ 6	$ 7	$ 6	$ 6	$ 7	$ 7	$ 7	$ 5	$ 7	$ 5	$ 4	$ -	$ -	$ -	$ -	$ 68
Total Consumables	K USD	$ -	$ 583	$ 672	$ 616	$ 609	$ 704	$ 694	$ 668	$ 460	$ 651	$ 486	$ 370	$ -	$ -	$ -	$ -	$ 6,514
Parts / MARC Cost	K USD	$ -	$ 159	$ 184	$ 168	$ 166	$ 192	$ 190	$ 182	$ 126	$ 178	$ 133	$ 101	$ -	$ -	$ -	$ -	$ 1,780
Total Maint. Allocation (no labor)	K USD	$ -	$ 742	$ 856	$ 785	$ 775	$ 897	$ 883	$ 850	$ 586	$ 829	$ 619	$ 472	$ -	$ -	$ -	$ -	$ 8,294
Maintenance Labor	K USD	$ 220	$ 558	$ 558	$ 558	$ 558	$ 558	$ 558	$ 558	$ 558	$ 558	$ 558	$ 462	$ 367	$ 271	$ 271	$ 14	$ 7,181
Operator Wages & Burden	K USD	$ 344	$ 1,147	$ 1,147	$ 1,147	$ 1,147	$ 1,147	$ 1,147	$ 1,147	$ 1,147	$ 1,147	$ 1,147	$ 765	$ 574	$ 382	$ 382	$ 28	$ 13,947
Total Loading Cost	K USD	$ 1,343	$ 6,117	$ 6,199	$ 6,108	$ 5,879	$ 6,187	$ 6,142	$ 6,105	$ 5,845	$ 6,098	$ 5,878	$ 5,253	$ 4,494	$ 2,121	$ 1,847	$ 129	$ 75,745
Loading Cost per Tonne Moved by Item																		
Fuel Cost	$/t	$ 0.051	$ 0.059	$ 0.054	$ 0.056	$ 0.054	$ 0.061	$ 0.062	$ 0.054	$ 0.054	$ 0.065	$ 0.058	$ 0.072	$ 0.089	$ 0.054	$ 0.259	$ -	$ 0.061
Lube & Oil	$/t	$ 0.010	$ 0.012	$ 0.011	$ 0.011	$ 0.011	$ 0.012	$ 0.013	$ 0.011	$ 0.011	$ 0.013	$ 0.012	$ 0.014	$ 0.017	$ 0.011	$ 0.051	$ -	$ 0.012
Tires / Under Carriage	$/t	$ -	$ 0.006	$ 0.007	$ 0.007	$ 0.007	$ 0.008	$ 0.008	$ 0.007	$ 0.005	$ 0.008	$ 0.006	$ 0.006	$ -	$ -	$ -	$ -	$ 0.006
Wear Items & GET	$/t	$ 0.004	$ 0.004	$ 0.004	$ 0.004	$ 0.004	$ 0.004	$ 0.004	$ 0.004	$ 0.004	$ 0.004	$ 0.004	$ 0.005	$ 0.007	$ 0.004	$ 0.019	$ -	$ 0.004
Total Consumables	$/t	$ 0.065	$ 0.081	$ 0.075	$ 0.078	$ 0.075	$ 0.086	$ 0.087	$ 0.075	$ 0.074	$ 0.090	$ 0.079	$ 0.097	$ 0.113	$ 0.069	$ 0.329	$ -	$ 0.083
Parts / MARC Cost	$/t	$ 0.041	$ 0.045	$ 0.041	$ 0.043	$ 0.041	$ 0.046	$ 0.047	$ 0.041	$ 0.042	$ 0.049	$ 0.044	$ 0.056	$ 0.071	$ 0.043	$ 0.205	$ -	$ 0.046
Maintenance Labor	$/t	$ 0.030	$ 0.016	$ 0.014	$ 0.015	$ 0.015	$ 0.016	$ 0.017	$ 0.015	$ 0.016	$ 0.018	$ 0.016	$ 0.018	$ 0.019	$ 0.021	$ 0.121	$ -	$ 0.017
Operator Wages & Burden	$/t	$ 0.047	$ 0.033	$ 0.030	$ 0.032	$ 0.032	$ 0.034	$ 0.035	$ 0.030	$ 0.032	$ 0.036	$ 0.034	$ 0.029	$ 0.030	$ 0.029	$ 0.171	$ -	$ 0.033
Total Loading Cost	$/t	$ 0.182	$ 0.174	$ 0.160	$ 0.168	$ 0.162	$ 0.182	$ 0.185	$ 0.161	$ 0.163	$ 0.193	$ 0.173	$ 0.200	$ 0.233	$ 0.162	$ 0.827	$ -	$ 0.180
Cost per Tonne Moved	$/t	$ 0.182	$ 0.171	$ 0.160	$ 0.168	$ 0.162	$ 0.160	$ 0.172	$ 0.161	$ 0.163	$ 0.181	$ 0.173	$ 0.200	$ 0.233	$ 0.162	$ 0.173	$ 0.166	$ 0.172

TABLE 19-17: ANNUAL HAULAGE OPERATING COST
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Haulage Cost - CAT 785 Fleet	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Total
Fuel Consumption (KL)	KL	-	1,501	2,000	1,971	1,965	1,495	1,942	1,942	1,942	1,947	1,942	1,942	-	-	-	-	20,588
Fuel Cost	K USD	$ -	$ 840	$ 1,120	$ 1,104	$ 1,100	$ 837	$ 1,087	$ 1,087	$ 1,088	$ 1,091	$ 1,087	$ 1,087	$ -	$ -	$ -	$ -	$ 11,529
Lube & Oil	K USD	-	263	351	346	345	262	341	341	341	342	341	341	-	-	-	-	$ 3,611
Tires	K USD	-	681	908	895	892	679	882	882	882	884	882	882	-	-	-	-	$ 9,349
Wear Items & GET	K USD	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	$ -
Total Consumables	K USD	$ -	$ 1,785	$ 2,379	$ 2,345	$ 2,337	$ 1,778	$ 2,310	$ 2,310	$ 2,310	$ 2,316	$ 2,310	$ 2,310	$ -	$ -	$ -	$ -	$ 24,489
Parts / MARC Cost	K USD	-	460	614	605	603	459	596	596	596	598	596	596	-	-	-	-	$ 6,318
Total Maint. Allocation (no labor)	K USD	-	2,245	2,993	2,950	2,940	2,237	2,905	2,906	2,906	2,914	2,905	2,906	-	-	-	-	$ 30,807
Haulage Cost - CAT 789 Fleet																		
Fuel Consumption (KL)	KL	1,592	7,021	8,912	8,941	8,828	10,681	10,708	10,177	9,398	10,647	10,775	10,728	8,733	6,153	1,893	73	125,261
Fuel Cost	K USD	$ 891	$ 3,932	$ 4,991	$ 5,007	$ 4,944	$ 5,982	$ 5,996	$ 5,699	$ 5,263	$ 5,962	$ 6,034	$ 6,008	$ 4,890	$ 3,446	$ 1,060	$ 41	$ 70,146
Lube & Oil	K USD	247	1,091	1,385	1,389	1,372	1,659	1,664	1,581	1,460	1,654	1,674	1,667	1,357	956	294	11	$ 19,461
Tires	K USD	838	3,695	4,690	4,706	4,646	5,622	5,636	5,356	4,946	5,604	5,671	5,646	4,596	3,238	996	39	$ 65,926
Wear Items & GET	K USD	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	$ -
Total Consumables	K USD	$ 1,977	$ 8,718	$ 11,066	$ 11,102	$ 10,961	$ 13,263	$ 13,296	$ 12,637	$ 11,669	$ 13,220	$ 13,379	$ 13,320	$ 10,843	$ 7,640	$ 2,351	$ 91	$ 155,533
Parts / MARC Cost	K USD	396	2,044	2,594	2,603	2,570	3,109	3,117	2,962	2,736	3,099	3,136	3,123	2,542	1,791	551	21	$ 36,394
Total Maint. Allocation (no labor)	K USD	2,372	10,762	13,660	13,705	13,531	16,372	16,413	15,599	14,405	16,320	16,516	16,443	13,385	9,431	2,902	112	$ 191,928
Maintenance Labor	K USD	$ 956	$ 2,454	$ 2,836	$ 2,836	$ 2,836	$ 3,219	$ 3,219	$ 3,219	$ 3,219	$ 3,219	$ 3,219	$ 3,219	$ 2,358	$ 1,689	$ 637	$ 35	$ 39,168
Operator Wages & Burden	K USD	$ 1,836	$ 5,100	$ 5,780	$ 5,780	$ 5,780	$ 6,800	$ 6,800	$ 6,800	$ 6,800	$ 6,800	$ 6,800	$ 6,800	$ 4,760	$ 3,400	$ 1,020	$ 74	$ 81,130
Total Haulage Cost	K USD	$ 5,165	$ 20,561	$ 25,269	$ 25,271	$ 25,087	$ 28,627	$ 29,337	$ 28,524	$ 27,330	$ 29,252	$ 29,440	$ 29,367	$ 20,503	$ 14,520	$ 4,559	$ 221	$ 343,033
Haulage Cost per Tonne Moved by Item																		
Fuel Cost	$/t	$ 0.12	$ 0.14	$ 0.16	$ 0.17	$ 0.17	$ 0.20	$ 0.21	$ 0.18	$ 0.18	$ 0.22	$ 0.21	$ 0.27	$ 0.25	$ 0.26	$ 0.47	$ -	$ 0.19
Lube & Oil	$/t	$ 0.03	$ 0.04	$ 0.04	$ 0.05	$ 0.05	$ 0.06	$ 0.06	$ 0.05	$ 0.05	$ 0.06	$ 0.06	$ 0.08	$ 0.07	$ 0.07	$ 0.13	$ -	$ 0.05
Tires	$/t	$ 0.11	$ 0.12	$ 0.14	$ 0.15	$ 0.15	$ 0.19	$ 0.20	$ 0.16	$ 0.16	$ 0.21	$ 0.19	$ 0.25	$ 0.24	$ 0.25	$ 0.45	$ -	$ 0.18
Total Consumables	$/t	$ 0.27	$ 0.30	$ 0.35	$ 0.37	$ 0.37	$ 0.44	$ 0.47	$ 0.39	$ 0.39	$ 0.49	$ 0.46	$ 0.60	$ 0.56	$ 0.58	$ 1.05	$ -	$ 0.43
Parts / MARC Cost	$/t	$ 0.05	$ 0.07	$ 0.08	$ 0.09	$ 0.09	$ 0.10	$ 0.11	$ 0.09	$ 0.09	$ 0.12	$ 0.11	$ 0.14	$ 0.13	$ 0.14	$ 0.25	$ -	$ 0.10
Maintenance Labor	$/t	$ 0.13	$ 0.07	$ 0.07	$ 0.08	$ 0.08	$ 0.09	$ 0.10	$ 0.09	$ 0.09	$ 0.10	$ 0.09	$ 0.12	$ 0.12	$ 0.13	$ 0.29	$ -	$ 0.09
Total Maintenance Allocation	$/t	$ 0.45	$ 0.44	$ 0.50	$ 0.54	$ 0.53	$ 0.64	$ 0.68	$ 0.57	$ 0.57	$ 0.71	$ 0.67	$ 0.86	$ 0.81	$ 0.85	$ 1.58	$ -	$ 0.62
Operator Wages & Burden	$/t	$ 0.25	$ 0.14	$ 0.15	$ 0.16	$ 0.16	$ 0.20	$ 0.20	$ 0.18	$ 0.19	$ 0.21	$ 0.20	$ 0.26	$ 0.25	$ 0.26	$ 0.46	$ -	$ 0.19
Total Haulage Cost	$/t	$ 0.70	$ 0.58	$ 0.65	$ 0.69	$ 0.69	$ 0.84	$ 0.88	$ 0.75	$ 0.76	$ 0.92	$ 0.87	$ 1.12	$ 1.06	$ 1.11	$ 2.04	$ -	$ 0.81
Cost per Tonne Moved	$/t	$ 0.701	$ 0.574	$ 0.654	$ 0.694	$ 0.692	$ 0.741	$ 0.822	$ 0.753	$ 0.762	$ 0.867	$ 0.866	$ 1.118	$ 1.061	$ 1.107	$ 0.427	$ 0.286	$ 0.779

TABLE 19-18: ANNUAL MINE SUPPORT OPERATING COSTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Mine Support Labor Costs	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Total
Mine Support Wages	K USD	$ 612	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,360	$ 1,020	$ 1,020	$ 510	$ 23,222
Mine Support Maint. Labor	K USD	$ 115	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 191	$ 191	$ 96	$ 4,034
Total Mine Support Costs																		
Consumables	K USD	$ 765	$ 1,507	$ 1,503	$ 1,503	$ 1,503	$ 1,507	$ 1,503	$ 1,503	$ 1,503	$ 1,507	$ 1,503	$ 1,503	$ 1,205	$ 1,208	$ 452	$ 149	$ 20,322
Parts / MARC Cost	K USD	$ 207	$ 407	$ 406	$ 406	$ 406	$ 407	$ 406	$ 406	$ 406	$ 407	$ 406	$ 406	$ 326	$ 327	$ 122	$ 40	$ 5,493
Operating Labor	K USD	$ 612	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,360	$ 1,020	$ 1,020	$ 510	$ 23,222
Maintenance Labor	K USD	$ 115	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 191	$ 191	$ 96	$ 4,034
Total Costs	K USD	$ 1,698	$ 3,901	$ 3,896	$ 3,896	$ 3,896	$ 3,901	$ 3,896	$ 3,896	$ 3,896	$ 3,901	$ 3,896	$ 3,896	$ 3,178	$ 2,746	$ 1,786	$ 795	$ 53,072
Cost per tonne																		
Consumables	$/t	$ 0.10	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.05	$ 0.04	$ 0.04	$ 0.05	$ 0.04	$ 0.06	$ 0.06	$ 0.09	$ 0.20	$ -	$ 0.05
Maintenance Allocations	$/t	$ 0.03	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.02	$ 0.02	$ 0.02	$ 0.05	$ -	$ 0.01
Operating Labor	$/t	$ 0.08	$ 0.05	$ 0.04	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.04	$ 0.05	$ 0.05	$ 0.05	$ 0.06	$ 0.07	$ 0.08	$ 0.46	$ -	$ 0.06
Maintenance Labor	$/t	$ 0.02	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.09	$ -	$ 0.01
Total Costs	$/t	$ 0.23	$ 0.11	$ 0.10	$ 0.11	$ 0.11	$ 0.11	$ 0.12	$ 0.10	$ 0.11	$ 0.12	$ 0.11	$ 0.15	$ 0.16	$ 0.21	$ 0.80	$ -	$ 0.13
Cost per Tonne Moved	$/t	$ 0.230	$ 0.109	$ 0.101	$ 0.107	$ 0.108	$ 0.101	$ 0.109	$ 0.103	$ 0.109	$ 0.116	$ 0.115	$ 0.148	$ 0.164	$ 0.209	$ 0.167	$ 1.025	$ 0.121

TABLE 19-19: ANNUAL MINE MAINTENANCE COSTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Janaury 20110

Total Equipment Costs	**Units**	**Yr -1**	**Yr 1**	**Yr 2**	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yr 8**	**Yr 9**	**Yr 10**	**Yr 11**	**Yr 12**	**Yr 13**	**Yr 14**	**Yr 15**	**Total**
Consumables	K USD	$ 38	$ 95	$ 95	$ 95	$ 95	$ 95	$ 95	$ 95	$ 95	$ 95	$ 95	$ 95	$ 95	$ 95	$ 16	$ -	$ 1,293
Parts / MARC Cost	K USD	$ 13	$ 32	$ 32	$ 32	$ 32	$ 32	$ 32	$ 32	$ 32	$ 32	$ 32	$ 32	$ 32	$ 32	$ 5	$ -	$ 438
Total Equipment Costs	K USD	$ 51	$ 128	$ 127	$ 127	$ 127	$ 128	$ 127	$ 127	$ 127	$ 128	$ 127	$ 127	$ 127	$ 128	$ 22	$ -	$ 1,731
Wages & Sallaries																		
Supervision	K USD	95.64	159.4	159.4	159.4	159.4	159.4	159.4	159.4	159.4	159.4	159.4	159.4	159.4	159.4	27.16197	0	$ 2,195
Planners	K USD	47.82	79.7	79.7	79.7	79.7	79.7	79.7	79.7	79.7	79.7	79.7	79.7	79.7	79.7	13.58098	0	$ 1,098
Hourly Personnel	K USD	369.84	1232.8	1232.8	1232.8	1232.8	1232.8	1232.8	1232.8	616.4	616.4	616.4	616.4	616.4	616.4	105.0354	0	$ 12,803
Total	K USD	513.3	1471.9	1471.9	1471.9	1471.9	1471.9	1471.9	1471.9	855.5	855.5	855.5	855.5	855.5	855.5	145.7783	0	$ 16,095
Other Costs																		
Supplies	K USD	$ 30	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 9	$ -	$ 689
Light Vehicles	K USD	$ 10	$ 24	$ 24	$ 24	$ 24	$ 24	$ 24	$ 24	$ 24	$ 24	$ 24	$ 24	$ 16	$ 16	$ 3	$ -	$ 311
Total	K USD	$ 40	$ 74	$ 74	$ 74	$ 74	$ 74	$ 74	$ 74	$ 74	$ 74	$ 74	$ 74	$ 66	$ 66	$ 11	$ -	$ 1,000
Total Mine Maintenance	K USD	$ 604	$ 1,674	$ 1,674	$ 1,674	$ 1,674	$ 1,674	$ 1,674	$ 1,674	$ 1,057	$ 1,058	$ 1,057	$ 1,057	$ 1,049	$ 1,049	$ 179	$ -	$ 18,826
Cost per Tonne	$/t	$ 0.08	$ 0.05	$ 0.04	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.04	$ 0.03	$ 0.03	$ 0.03	$ 0.04	$ 0.05	$ 0.08	$ 0.08	$ -	$ 0.04
Cost per Tonne Moved	$/t	$ 0.082	$ 0.047	$ 0.043	$ 0.046	$ 0.046	$ 0.043	$ 0.047	$ 0.044	$ 0.029	$ 0.031	$ 0.031	$ 0.040	$ 0.054	$ 0.080	$ 0.017	$ -	$ 0.043

TABLE 19-20: ANNUAL MINE GENERAL SERVICES COST
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Wages & Sallary	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Total
Mine General Services																		
Supervision	K USD	$ 389	$ 648	$ 648	$ 648	$ 648	$ 648	$ 648	$ 648	$ 648	$ 648	$ 648	$ 648	$ 648	$ 648	$ 110	$ -	$ 8,927
Clerical	K USD	$ 38	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 11	$ -	$ 879
Training	K USD	$ 51	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 14	$ -	$ 1,170
Total	K USD	$ 478	$ 797	$ 797	$ 797	$ 797	$ 797	$ 797	$ 797	$ 797	$ 797	$ 797	$ 797	$ 797	$ 797	$ 136	$ -	$ 10,976
Engineering																		
Supervision	K USD	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 27	$ -	$ 2,195
Sallaried Personnel	K USD	$ 80	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 23	$ -	$ 1,830
Hourly Personnel	K USD	$ 105	$ 351	$ 351	$ 351	$ 351	$ 351	$ 351	$ 351	$ 351	$ 351	$ 351	$ 351	$ 175	$ 175	$ 30	$ -	$ 4,345
Total	K USD	$ 281	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 468	$ 468	$ 80	$ -	$ 8,370
Mine Geology																		
Supervision	K USD	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 27	$ -	$ 2,195
Sallaried Personnel	K USD	$ 128	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 106	$ 106	$ 18	$ -	$ 2,697
Hourly Personnel	K USD	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ 80	$ 14	$ -	$ 2,022
Total	K USD	$ 319	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 345	$ 345	$ 59	$ -	$ 6,914
Supplies & Other																		
Mine General Services	K USD	$ 30	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 9	$ -	$ 689
Mine Light Vehicle	K USD	$ 26	$ 65	$ 64	$ 64	$ 64	$ 65	$ 64	$ 64	$ 64	$ 65	$ 64	$ 64	$ 16	$ 16	$ 3	$ -	$ 771
Engineering Supplies	K USD	$ 9	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 3	$ -	$ 207
Engineering Light Vehicle	K USD	$ 4	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 2	$ -	$ 131
Geology Supplies	K USD	$ 9	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 3	$ -	$ 207
Geology Light Vehicle	K USD	$ 4	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 2	$ -	$ 131
Total	K USD	$ 82	$ 164	$ 164	$ 164	$ 164	$ 164	$ 164	$ 164	$ 164	$ 164	$ 164	$ 164	$ 115	$ 116	$ 20	$ -	$ 2,135
Total Mine Other Costs	K USD	$ 1,159	$ 2,136	$ 2,135	$ 2,135	$ 2,135	$ 2,136	$ 2,135	$ 2,135	$ 2,135	$ 2,136	$ 2,135	$ 2,135	$ 1,726	$ 1,726	$ 294	$ -	$ 28,395
Cost per Tonnes	$/t	$ 0.16	$ 0.06	$ 0.06	$ 0.06	$ 0.06	$ 0.06	$ 0.06	$ 0.06	$ 0.06	$ 0.07	$ 0.06	$ 0.08	$ 0.09	$ 0.13	$ 0.13	$ -	$ 0.07
Cost per Tonne Moved	$/t	$ 0.157	$ 0.060	$ 0.055	$ 0.059	$ 0.059	$ 0.055	$ 0.060	$ 0.056	$ 0.060	$ 0.063	$ 0.063	$ 0.081	$ 0.089	$ 0.132	$ 0.028	$ -	$ 0.064

19.4 Limestone Quarry and Lime Production

Limestone is currently commercially produced near Katherine by quarrying the Katherine limestone beds. The Mt. Todd operation plans to ensure a supply of economic lime is available for use in the processing and water treatment areas of the operation. A limestone quarrying operation will be developed by mining a nearby outcrop of the Katherine Limestone and a lime kiln plant will be established at the quarry to convert the limestone into lime.

This small (300 – 500 tpd) limestone quarry/ lime kiln operation will consist of a small conventional open pit mining operation utilizing a drill and blast, loader and truck operation feeding a jaw crusher and screening plant. The resulting screen plant product will be fed into a 150 tpd vertical lime kiln fired by natural gas. The kiln will produce the necessary quantity of lime needed for the gold processing and waste water treatment needs.

19.5 Power Supply

The following portion of Section 19 (section 19.5) has been taken from *"Power Engineers Inc. Mt. Todd Power Station - Phase 3 Pre-Feasibility" (September 30, 2010)* with only minor changes for consistent formatting and terminology purposes (see Appendix G-1 for complete report).

The report provides detailed discussion of the generation equipment options available for onsite electrical supply to meet the power requirements of the re-commissioned Mt. Todd Gold Mine in the NT of Australia operated by the Vista Gold Corporation.

The objective of this report is to compare equipment selections from leading vendors capable of meeting the site electrical power demand with consideration for surplus power exported to the local utility grid. Equipment options include a single gas turbine generator (GTG) or a group of reciprocating gas engine generator sets. This report also provides a brief overview of the Australian wholesale energy market and opportunities for surplus power sales to the NT utility grid to give a background of how a power sales agreement with the local utility, Power and Water Corporation (PWC), may be structured.

19.5.1 Summary

Project costs are analyzed over a 13-year period to correspond with the approximate life of the proposed Mt. Todd mining project. The generating equipment is expected to have a 35 percent salvage value estimated from published pricing for similarly used equipment. At the request of Vista, project costs are calculated without annual pricing index escalation or interest payments.

19.5.2 Conclusions and Recommendations

This study provides a preliminary budgetary outline for capital and operating costs to produce onsite power generation for the scale of process indicated by Vista Gold Corp. at the Mt. Todd Mine with limited information about the site. It is apparent that there is demand for the surplus power that could be available for export to the local utility grid. Without a well developed wholesale electricity market a bilateral contract with Power and Water Corporation for export power will be necessary in addition to applicable generation and environmental permitting considerations beyond those directly related to the Vista Gold Corp. mining process.

It is not expected that a secondary wholesale market will develop for another electricity retailer at the Mt. Todd site within the design life of the Mt. Todd project. The geographic isolation of the site has so far made it impractical and uneconomical to connect the Darwin-Katherine distribution network to other parts of the Australian electric utility grid.

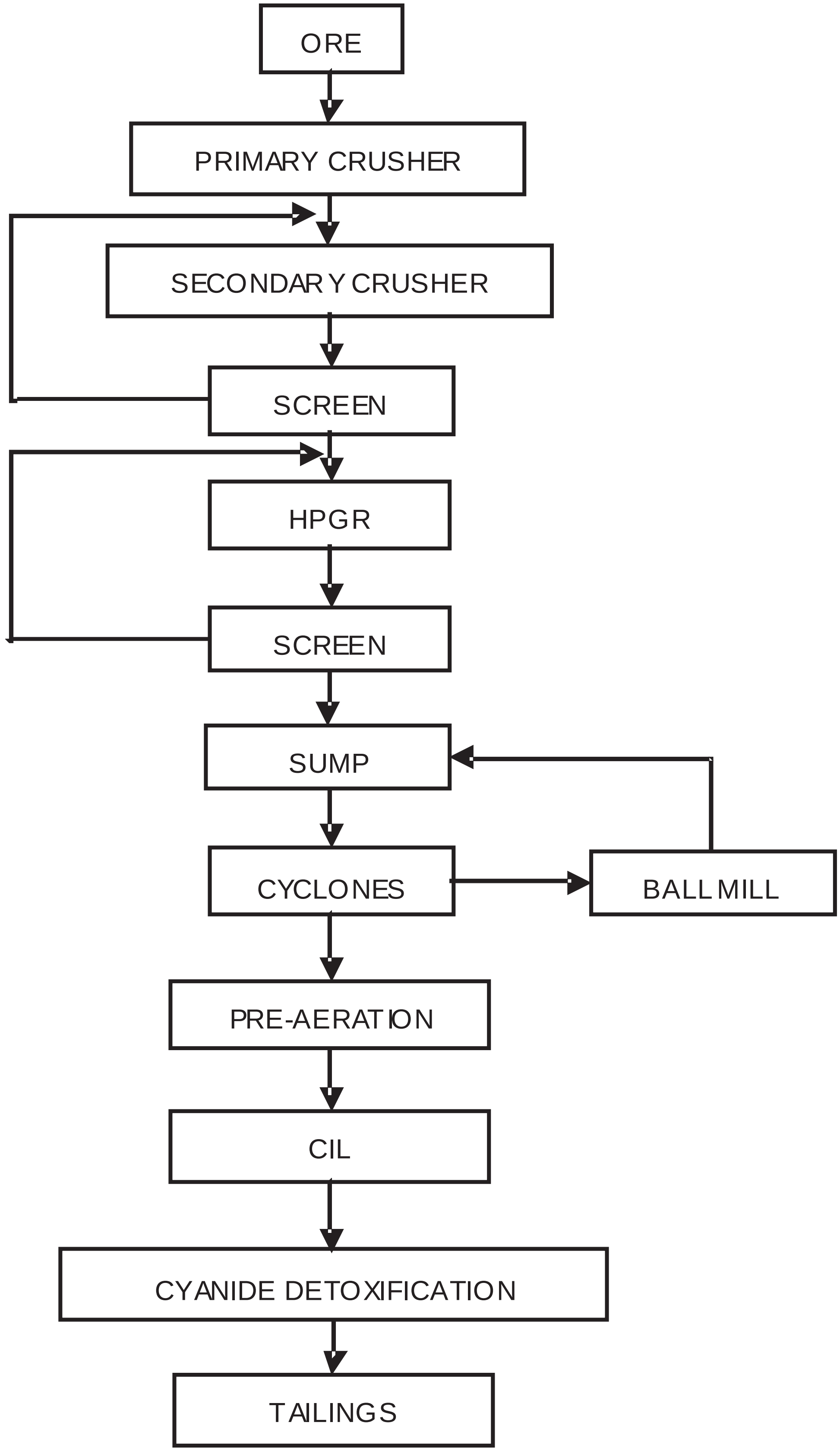
It will be necessary to determine how much surplus power PWC is willing to purchase in a long term supply contract before making a final selection on a preferred generation equipment supplier and model. When site and utility grid demands are better understood, a final price can be negotiated with equipment suppliers for the equipment that best suits the needs of the project.

The Rolls Royce Trent 60 is the only single aero-derivative gas turbine that can meet the entire site load and has reserve capacity to support a future expansion or export to the utility grid. The option with two LM2500+'s has the lowest initial cost but a much higher heat rate and fuel costs than all other options considered in this study. If continuous power supply is required, the Wartsila 20V34SG reciprocating engines are estimated to provide the lowest overall 13-year project costs.

19.6 Process Operations

The process flowsheet shown in FIGURE 19-3 was developed from work performed at RDi of Wheat Ridge, Colorado, USA. The direct leach scenario consists of crushing, grinding, classification, pre-aeration, leaching via CIL, and cyanide detoxification before final tailing deposition.

TABLE 19-21 details the key design criteria for the processing plant as interpreted by Ausenco Services Pty Ltd who provided process plant design, engineering, and cost estimation services for the Prefeasibility Study.



(Ausenco Services Pty. Ltd. Report 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: Fig19-3.cdr	**Figure 19-3 Block Flow Diagram Modified Leach Flowsheet**
	Project: Mt. Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	

TABLE 19-21: KEY PROCESS PLANT DESIGN CRITERIA
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Description	Unit	Value	Source
Nominal plant throughput	Mtpy	11	TetraTech
Primary crusher availability	%	75	Ausenco
HPGR availability	%	88	Ausenco
Grinding and CIL availability	%	88	Ausenco
Nominal plant feed rate	t/h	1,427	Ausenco
Comminution characteristics			
DWI	kWh/m^3	12.7	test work
RWI	kWh/t	22.6	test work
BWI	kWh/t	24.0	test work
Ai		0.135	test work
Head grade			
gold	g/t	0.853	TetraTech
copper (total)	g/t	519	TetraTech
copper (acid soluble)	g/t	24	TetraTech
copper (cyanide soluble)	g/t	65	TetraTech
Primary grind size P_{80}	µm	150	TetraTech
	mesh	100	TetraTech
Pre-leach thickening			
thickener flux	$t/m^2/h$	1.5	Ausenco
thickener underflow density	% solids w/w	55	TetraTech
Pre-aeration			
residence time, min	h	4	test work
CIL			
leach feed density	% solids w/w	50	TetraTech
leach time	h	4.1	Ausenco
gold extraction	%	82.0	RDi
gold recovery	%	80.5	RDi
adsorption time	h	20.7	Ausenco
gold solution loss target	mg/L	0.01	Ausenco
Desorption–gold room			
Elution circuit type		acid wash cold CN wash Split AARL	TetraTech TetraTech Ausenco
Number of parallel trains		2	Ausenco
batch size	t	12	Ausenco
strip frequency	#/week	6	Ausenco
Cyanide Detoxification			
method		Air–SO_2	Ausenco
residence time	h	2	Ausenco
residual CN_{wad} target level	ppm	<50	Ausenco

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Detail process design criteria and mass balances developed over the course of this study are included in Appendix E.

19.6.1 Plant Design Basis

The Reserve Case process plant as shown in FIGURE 19-4 was designed to treat 11 Mtpy of ore (~30 Ktpd or ~1,427 tph). This discussion is based on the Ausenco December 2010 report reference elsewhere and in Appendix E.

The Simplified Process Schematic shown in FIGURE 19-4 is similar to most common leach circuits with a few exceptions as noted below:

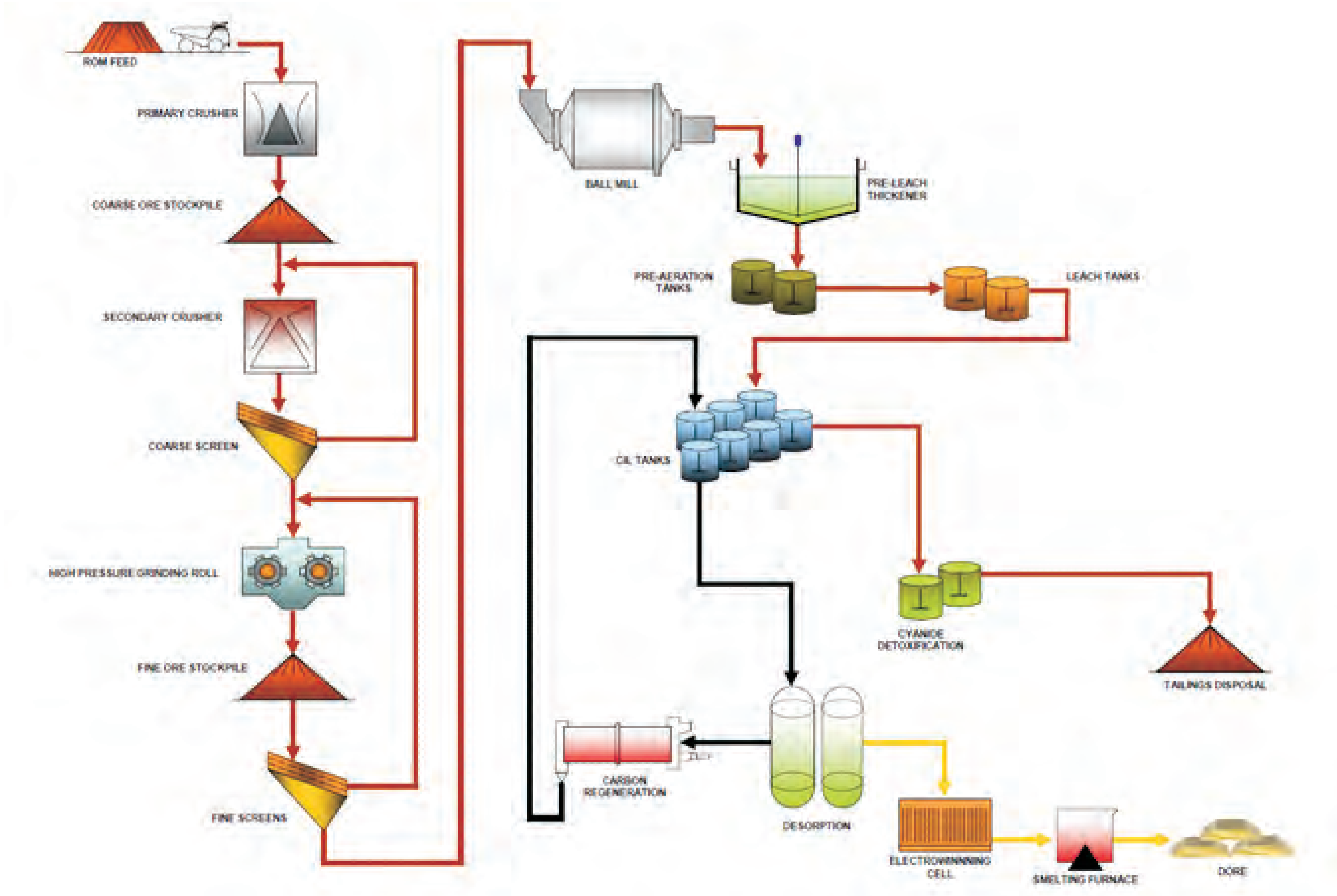
- HPGR will be used to prepare feed for the grinding mill replacing tertiary and quaternary crushing or SAG (or rod mill) grinding. HPGR technology is appropriate for use on hard ores as will be encountered at Mt. Todd. Mt. Todd ores are very hard in comparison to other ores as demonstrated by Bond work indices and JK SMC studies.
- Pre-aeration was included to reduce cyanide consumption during agitated leach operations by passivating pyrrhotite and secondary copper sulphide minerals to provide P_{80} 150 microns (P_{80} 100 mesh) feed. This was shown to be effective during the metallurgical testwork program.
- Gold desorption from carbon using a split Anglo American Research Laboratory (AARL) elution is planned to improve the circuit water balance.

As shown in the process schematic, ore is crushed and ground to the optimal leach size. It is expected that the design grind size as determined by the testwork will be optimized at the feasibility level on ore samples covering resource variables likely to be encountered during mining. Surge capacity is provided after the gyratory and after the HPGR crusher to provide a consistent feed rate to the tertiary screens before the grinding mill.

A single stage ball mill in closed circuit with hydrocyclones is designed to provide P_{80} 150um feed to pre-aeration and leaching. Mill feed enters the grind circuit in the cyclone feed sump where it mixes with process water and grinding mill discharge.

A pre-leach thickener provides surge capacity between the grinding mill and the pre-aeration circuit.

Pre-aeration before the CIL circuit reduces cyanide consumption during leach. CIL follows pre-aeration. Cyanide is added to the slurry at a pH of 10.5 or higher. Ultimately, carbon is added to adsorb the gold solubilized during leaching. Reactivated carbon, supplemented with fresh carbon as necessary, is added to the final tank in the circuit. The carbon is advanced countercurrent to slurry flow through interstage screens, the highest activity carbon always contacting the lowest gold grade solutions. A summary of the residence times and tank volumetric requirements is presented in TABLE 19-22.



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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: Fig19-4.dwg	**Figure 19-4** **Simplified Process Schematic**
	Project: Mt. Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	

TABLE 19-22: SUMMARY OF PRE-AERATION AND LEACH RESIDENCE TIMES AND TANK DETAILS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2010

Criterion		Pre-aeration	Leach	Adsorption
Residence Time - design criteria	h	4	4	20
Required volume	m^3	7123	8213	41064
Tank diameter	m	16.0	17.0	18.3
Tank height	m	18.9	18.9	18.9
Tank volume	m^3	3800	4290	4971
Number of tanks	#	2	2	9
Total volume	m^3	7600	8580	44740
Residence Time – estimated actual	h	4.3	4.2	21.8

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Loaded carbon is separated by screen from the CIL tank located immediately after the leach tanks and is transferred to a carbon strip vessel. Prior to gold stripping, a cold cyanide strip employing one bed volume of cold cyanide/caustic solution is used to strip copper cyanide ions loaded on the carbon during the CIL process.

Gold is subsequently removed from the carbon using a modified Anglo American Research Laboratories (AARL) carbon strip, also known as a split AARL, in which ten bed volumes hot cyanide/caustic solution are circulated though the strip vessel. The latter half of the total strip solution volume (i.e., five bed volumes) from the previous strip is heated and pumped from the eluant tank through the carbon strip vessel therein removing or “stripping” the gold from the carbon as it passes through the carbon bed. Pregnant solution is stored in a pregnant solution tank in preparation for electrowinning. An additional five bed volumes of fresh water are then used at temperature for the second half of the strip and are saved to a tank to be used for the first half of the next strip cycle. This has the effect of producing high grade pregnant liquor. A summary of the carbon strip circuit design criteria, as provided by Ausenco, is presented in TABLE 19-23.

TABLE 19-23: ELUTION AND REGENERATION DESIGN CRITERIA
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Criterion	Unit	Design Value
Parallel Circuits		2
Strip size	t	12
Strips per week per circuit		5
Acid wash		
type		Cold HCl
acid concentration to column	% w/w	3
bed volume	BV	0.67
water wash	BV	4
Cold Cyanide Wash		
cyanide strength	% w/w	3
caustic soda strength	% w/w	3
wash volume	BV	1
Elution		
type		Split AARL
elution rate	BV/h	2
elution temp	^{o}C	120–130
cyanide strength to column	% w/w	3
caustic soda strength to elution	% w/w	3
bed volume to starter tank	BV	5
bed volume to eluate tank	BV	5
Carbon reactivation		
kiln type		Horizontal rotating drum
kiln feed rate	kg/h	952
kiln utilization	%	75

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Electrowinning may commence immediately once the solution level in the pregnant solution tank is sufficient to cover the electrowinning feed pump intakes. Pregnant solution is circulated from the pregnant solution tank through electrowinning cells wherein gold is electrochemically plated onto stainless steel wool. Pregnant solution, once through the electrowinning cells, is returned to the pregnant solution tank. Circulation and plating typically take several hours, the process being complete once the gold grades are below economic levels. Once depleted in gold tenor, the barren solution is reintroduced into the leach circuit at the head of the CIL section to recover any residual gold that was not electrowon.

Gold adhering mildly to the stainless steel cathodes in the electrowinning cells is washed under high pressure from the cathodes into the bottom of the cell and transferred into a vacuum pan filter. Solids from the filter are further transferred into a drying oven and dried. The dried gold sludge is then transferred into an induction furnace, fluxed, smelted and poured into doré bars.

Ancillary operations to carbon loading and stripping include carbon acid washing and carbon stripping. Carbon is regularly washed in a mild (three percent) cold hydrochloric acid wash which removes carbonates that may have built up on the carbon during the CIL process. Acid washing is done before stripping to present the cyanide strip circuit with clean carbon, the pores in the carbon being free of the carbonate constituents that may hinder the strip process.

Carbon activity is reduced after carbon use in the CIL circuit. Carbon reactivation at high temperature in a reactivation kiln is performed after carbon stripping. The reactivation process burns off any contaminant organics and reopens the pores of the carbon increasing its activity to near that of fresh carbon. The reactivated carbon is screened to remove carbon fines and is ultimately reintroduced into the CIL circuit with fresh carbon make-up in the last CIL tank.

CIL plant tailing will be directed to a cyanide detoxification circuit in which the cyanide is reduced / eliminated by the SO_2 – Air process. The design criteria to be applied are presented in TABLE 19-24.

TABLE 19-24: CYANIDE DETOXIFICATION DESIGN CRITERIA
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Criterion	Unit	Design Value
Method		Air / SO_2
SO_2 source		Sodium Metabisulfite Solution
CN_{WAD} in feed	ppm	174
CN_{WAD} target in tails	ppm	<50
Residence time	min	120
No of reactors		2
SMBS dosage rate	g/g CN	2.3
Copper source		ore
Lime dosage rate	g/g SO_2	0.6
Oxygen demand	g/g CN	2.23
Oxygen source		air

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19.7 Reserve Case Process Capital Costs

Process capital costs were established at the prefeasibility level by Ausenco in their 17 December 2010 report entitled Mt. Todd Gold Project Engineering and Cost Study - 11MT/y Option Study a copy of which is included as Appendix E with this PFS report.

The estimated total plant capital cost, including tailings storage facilities, contingency, EPCM, and process indirects is $655,258,000.

Key design parameters used in the Ausenco capital cost estimate include the following:

Plant throughput, Mtpy	11
Plant feedrate, t/h	1,427
Head grade, g Au/t	0.853
Primary grind, P_{80} µm	150
Primary crusher availability, %	75
HPGR availability, %	88
Grinding & CIL availability, %	88

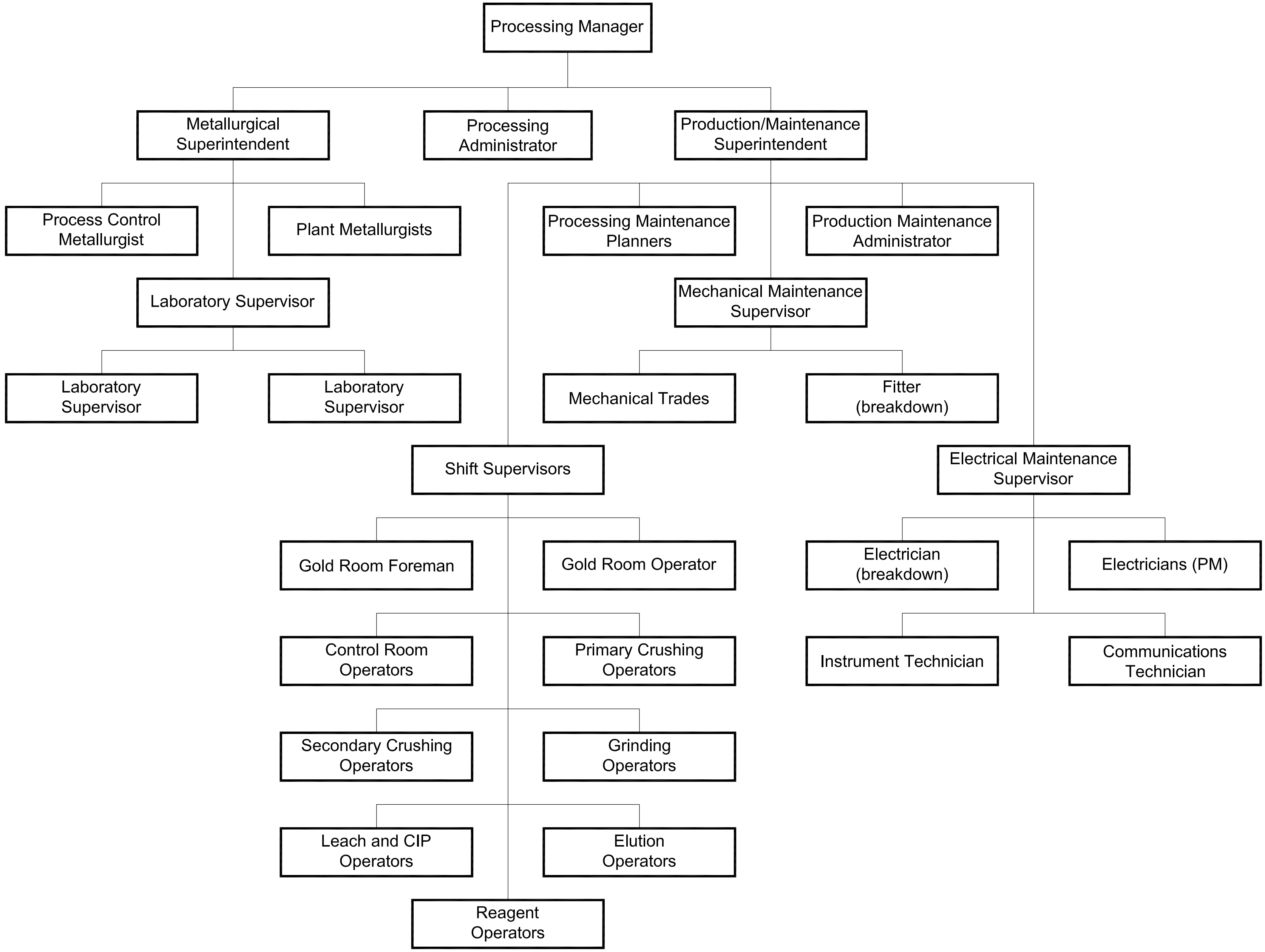
19.8 Reserve Case Process Operating Costs

Plant operating costs were established at the prefeasibility level by Ausenco in their 17 December 2010 report entitled Mt. Todd Gold Project Engineering and Cost Study - 11MT/y Option Study. FIGURE 19-5 presents the process plant labor organization.

The unit cost for ore processing estimated by Ausenco is $6.68/t (primary crushing through tailings pumping). This equates to an annualized process operating cost of ~$72 million at an annualized feedrate of ~11Mtpy.

Key assumptions used as a basis for the operating cost estimate include the following:

- Operating costs have a base date of Q4 2010
- No Contingency is applied
- Estimate has exclusions as listed below (see Ausenco Report, Appendix E)
- Owners Costs
- Mining Costs
- Administration Costs
- Contract labor and equipment, except where noted
- Insurance, shipping costs, umpire assay and refining charges for bullion
- Accuracy provisions
- Project insurances
- Corporate overhead charges
- Licenses, land use, water abstraction fees or other such charges
- Financing costs
- Royalties, taxes, goods and services tax (‘GST’) or similar imposts
- Expenditures classified as capital, sustaining capital
- Tailings management
- Water treatment costs
- Reclamation and closure costs



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	Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: 01/27/2011	

Figure 19-5
Process Plant Labor Organization

19.9 Capital and Operating Cost Summary

The following tables and associated pie charts provide a breakdown of the capital and operating costs for the proposed mine operations.

- Mine capital costs are based on mid-2010 costs.
- Process capital costs are based on late-2010 costs.

Initial Capital Costs of approximately $589.6 million, including contingency and EPCM occur during the 2 year construction period and the first year of production. The estimate assumes that the project will fund the full cost of all required infrastructure including a lime quarry / kiln / process facility, power plant, and all water treatment facilities necessary to support operations. Excess power will be sold to the local utility grid at an estimated rate of AUD$0.109/kwh for the duration of the project. Other capital including the cost of permitting, water treatment, tailings storage and reclamation are also included in the capital costs.

TABLE 19-25 is a summary of the original and sustaining capital expended during the project.

TABLE 19-25: SUMMARY OF INITIAL AND SUSTAINING CAPITAL COSTS
VISTA GOLD CORP. - MT. TODD GOLD PROJECT
January 2011

CAPITAL ($000'S)	LOM	INITIAL	SUSTAINING
MINE CAPITAL			
Primary:			
Open Pit Mine Equipment	98,792	46,483	52,309
Lime Operation Mine Equip	5,617	5,617	0
Sub-Total Primary	**104,409**	**52,100**	**52,309**
Ancillary:			
General Surface Mobil Equipment	18,596	8,404	10,191
Sub-Total Ancillary	**18,596**	**8,404**	**10,191**
Miscellaneous:			
Mine Office, Shop and Warehouse	2,268	2,268	0
Mining Development Supply and Labor Op Costs	9,394	9,394	0
Sub-Total Miscellaneous	**11,662**	**11,662**	**0**
TOTAL MINE CAPITAL (Before Contingency)	**134,667**	**72,166**	**62,500**
Mine Capital Contingency	**9,759**	**5,615**	**4,144**
PLANT CAPITAL			
Process Plant	269,243	269,243	0
Onsite Infrastructure	22,503	22,503	0
Mobile Equipment, Spares, First-Fills	11,223	11,223	0
Power Generating Station	37,678	37,678	0
Site Demolition	3,664	3,664	0
TAILING STORAGE FACILITIES CAPITAL			
Pre-production WTF + Tailings Management	4,777	4,777	0
TSF Fine Grading, Equipment, Piping, Drains	71,304	5,258	66,046
TSF Bulk Earthwork	88,555	4,193	84,362
TOTAL PLANT + TAILINGS STORAGE	**508,948**	**358,539**	**150,408**
INDIRECT PROCESS			
Temporary Construction Facilities	6,999	6,999	0
Commissioning	5,599	5,599	0
Total Indirect Process	**12,598**	**12,598**	**0**
TOTAL PLANT + TAILING + INDIRECT CAPITAL (Before Contingency)	**521,546**	**371,137**	**150,408**
Plant Capital Contingency	**60,208**	**51,202**	**9,006**
EPCM TOTAL (PLANT & TAILING)	**73,504**	**68,600**	**4,904**
OTHER CAPITAL			
Off-site Infrastructure / Accommodation Village	16,268	16,268	0
Excess Water Treatment Facility	17,985	0	17,985
Permitting	2,500	2,500	0
Recruit and Training	1,700	1,500	200
Lime Kiln/Processing	6,158	6,158	0
Total Other Capital	**44,611**	**26,426**	**18,185**
Other Capital Contingency	**6,692**	**3,964**	**2,728**
Total Contingency	**76,659**	**60,781**	**15,878**
TOTAL CAPITAL	**850,987**	**599,111**	**251,876**
TOTAL WORKING CAPITAL CHANGES	**102**	**(9,528)**	**9,630**
TOTAL CAPITAL + WORKING CAPITAL CHANGES	**851,088**	**589,583**	**261,506**

NOTE: Some rounding may occur due to truncation of the numbers.

TABLE 19-26 illustrates how process capital makes up nearly 50 percent of the total original capital costs. Mine capital is the next largest component making up approximately 20 percent of the total original capital cost.

TABLE 19-26: TOTAL INITIAL CAPITAL COSTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

	INITIAL CAPITAL (000)
MINE	72,166
PLANT	344,311
TAILINGS	14,228
INDIRECT PROCESS	12,598
EPCM	68,600
OTHER CAPITAL	26,426
TOTAL CONTINGENCY	60,781
OWNER RECLAMATION	
WORKING CAPITAL	(9,528)
	589,583

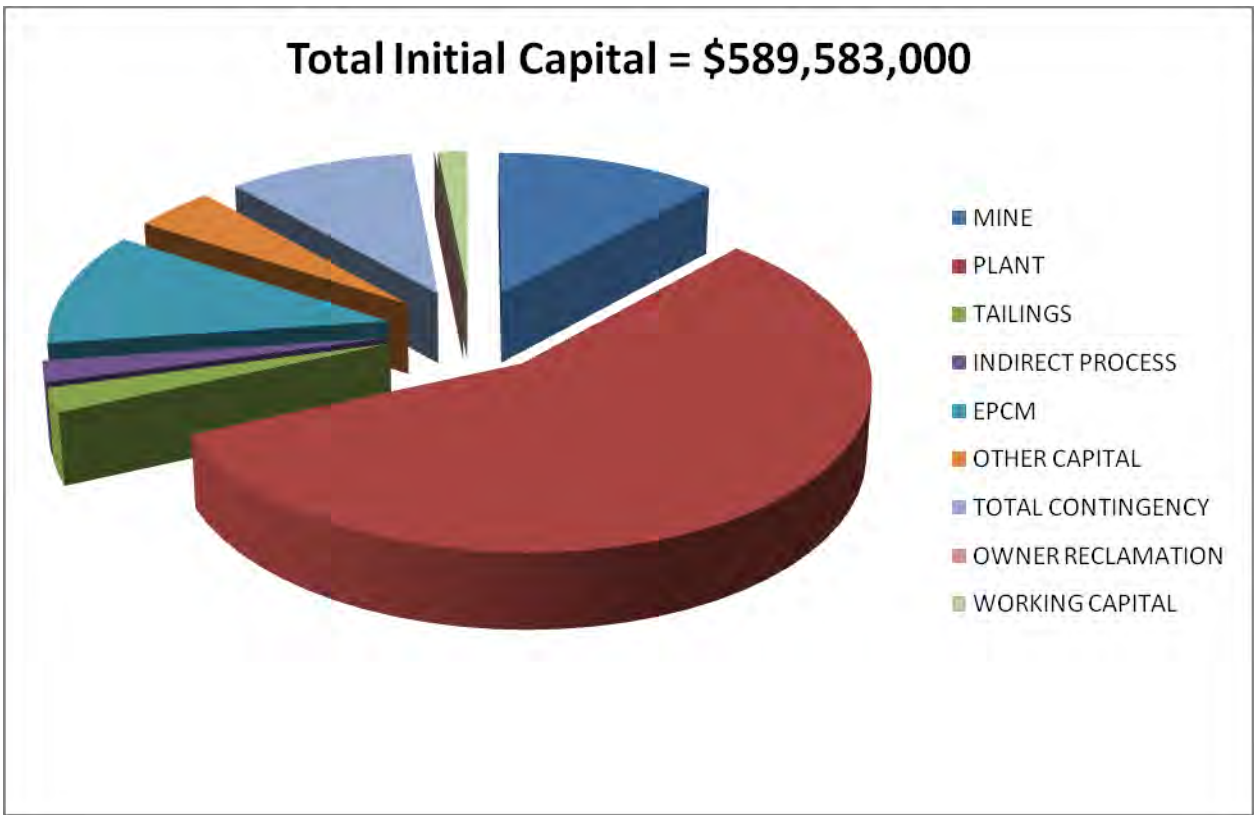


TABLE 19-27 illustrates the large portion of sustaining costs that will be dedicated to the TSF, tailings operations and the water treatment facilities.

TABLE 19-27: TOTAL SUSTAINING CAPITAL
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

	SUSTAINING CAPITAL (000)
MINE	62,500
PLANT	
TAILINGS	150,408
INDIRECT PROCESS	
EPCM	4,904
OTHER CAPITAL	18,185
TOTAL CONTINGENCY	15,878
WORKING CAPITAL	9,630
	261,505

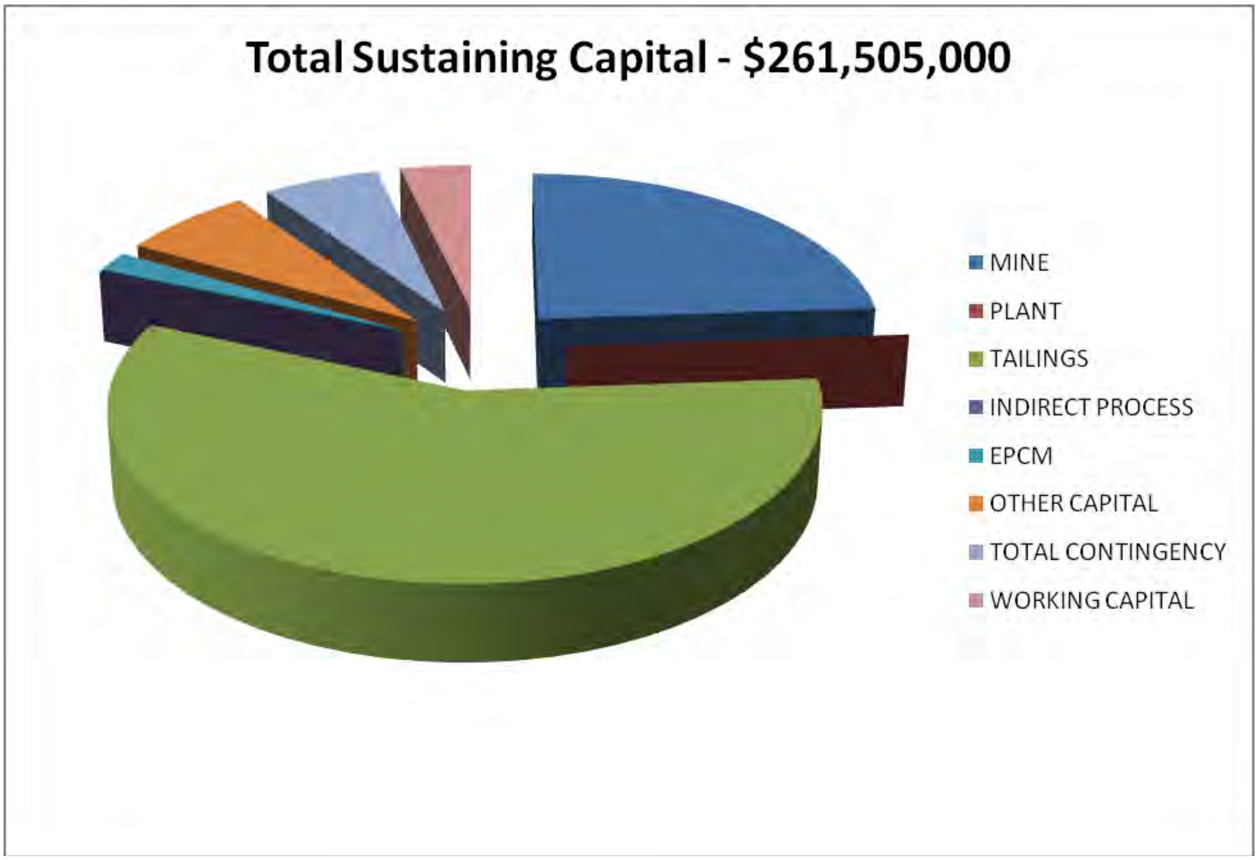


TABLE 19-28 illustrates that estimated initial capital costs for the process plant amounts to more than 50 percent of the total process and infrastructure startup capital.

TABLE 19-28:PROCESS INITIAL CAPITAL COSTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

	Total Plant INITIAL CAPITAL
Process Plant	269,243
Onsite Infrastructure	22,503
Mobile Equipment, Spares, First-Fills	11,223
Power Generating Station	37,678
Site Demolition	3,664
Pre-production WTF + Tails Op Costs	4,777
TSF Fine Grading, Equipment, Piping, Drains	5,258
TSF Bulk Earthwork	4,193
EPCM with AMAF	68,600
Plant Capital Contingency	51,202
Process inderects	12,598
	490,939

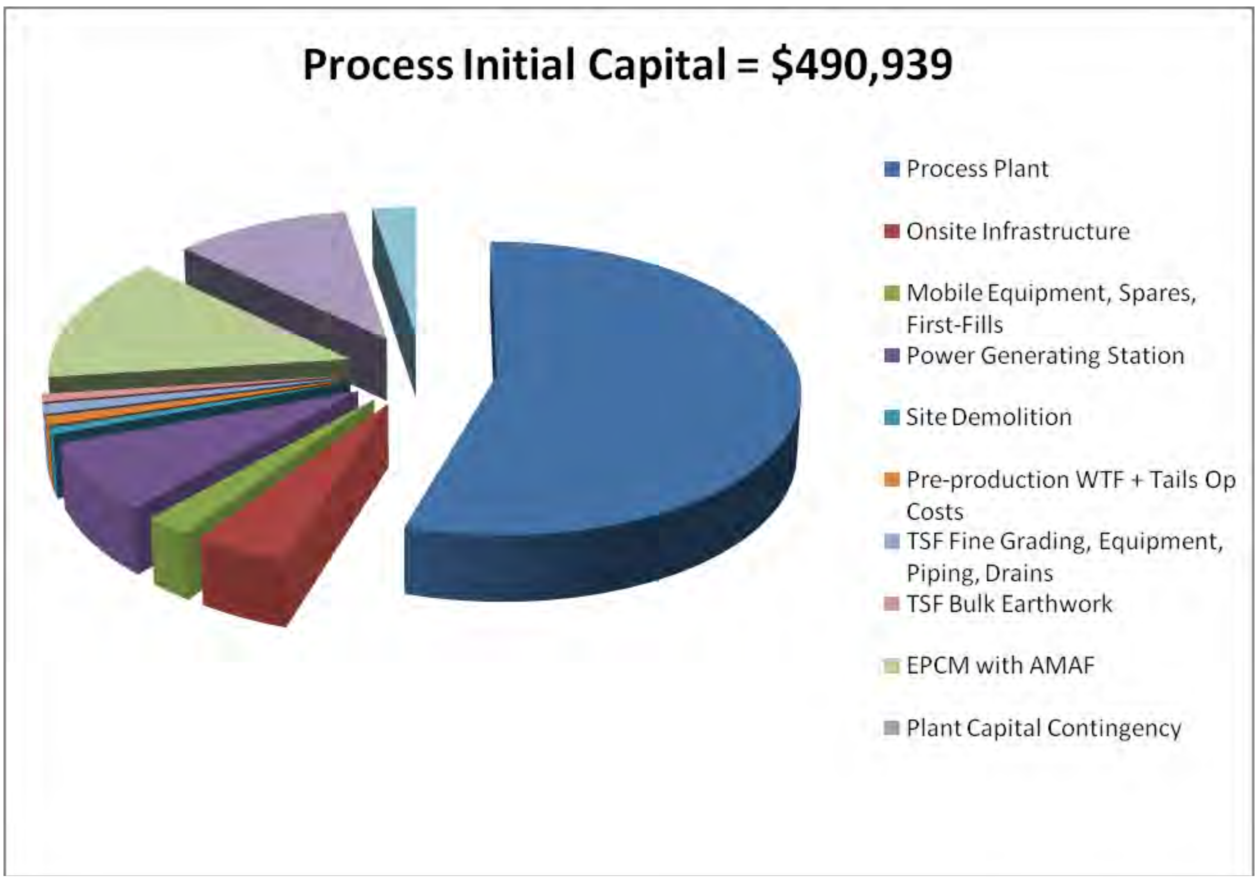


TABLE 19-29 illustrates the fact that the Primary Open Pit Mine Equipment comprises approximately 60% of the total mine initial capital costs.

TABLE 19-29: MINE INITIAL CAPITAL COSTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

	Total Mine INITIAL CAPITAL
Open Pit Mine Equipment	46,483
Lime Operation Mine Equip	5,617
General Surface Mobil Equipment	8,404
Mine Office, Shop and Warehouse	2,268
Mining Development Supply and Labor Op Costs	9,394
Mine Capital Contingency	5,615
	77,781

19.10 Environmental Considerations - Reclamation and Closure

Closure plans and strategies for each planned major facility at Mt. Todd and the mine-life water treatment system have been developed and are summarized in Section 5 and Appendix J. Tetra Tech estimated closure costs and mine-life water treatment cost. Mine-life water treatment cost estimates included water treatment cost during the pre-production and production phases of the project, and the closure and post-closure phases of the project. Closure and water treatment costs were estimated at a ± 25 percent level of accuracy based on the 10.65 Mtpy mine plan, the existing engineering and data presented in this PFS, stated assumptions and professional judgment.

The pre-feasibility level cost estimate for implementing the closure plan for the 10.65 Mtpy mine plan (excluding as summarized in Section 5.0 is $ 67,864,000. The PFS-level cost estimate for mine-life water treatment plan for the 10.65 Mtpy mine plan as summarized in Section 5.0 is $ 36,590,000. These estimated costs are summarized in TABLE 5-3.

19.11 Tailings Disposal

Previously, Tetra Tech evaluated twelve options for tailings disposal, including a dry stack facility, new TSF designs for both thickened and conventional tailings, and several raises to the existing TSF. Appendix K contains the tradeoff study. The 60 million tonne capacity raise to the existing TSF design (TSF1) was originally selected based on economic tradeoff studies and the relatively low cost per tonne of tailings stored. Since the project requires a total tailings storage capacity of 160 million tonnes, TSF2 is also required to provide the additional 100 million tonnes of tailings storage.

19.11.1 Existing Facility Raise – TSF1

The results of the tradeoff study indicated that the most cost effective option is the 60-million tonne raise to the existing TSF with thickened tailings. The cost will be lower than constructing a new facility because no liner or new water management systems will be required. Earthwork construction costs will be lower. Additionally, thickening the tailings will allow more tailings to be stored in a smaller impoundment, maximizing the available storage.

The existing TSF will be raised in a total of six separate stages. The Stage 2 embankment will be constructed mostly from waste rock fill using centerline construction methods and will have a core of low-permeability fill and a transition zone, each 3 m thick. All other stages will be constructed out of waste rock fill using upstream construction methods. Saddle dams will be constructed at Stages 2, 3, and 5 and will have the same zoned configuration as the Stage 2 raise. For both the main embankment raises and the saddle dams, the crest will be 8 m wide, with 2.5:1 (horizontal to vertical) downstream side slopes and 2:1 upstream side slopes. In addition to the embankment construction, each stage will require an emergency spillway to the northwest of the facility, raises to selected decant towers, extensions of the underdrains and toe drains, and tailings distribution header and spigot pipe assembly construction or relocation.

Several key assumptions were made in the TSF1 design and cost estimate:

- All decant towers, underdrains, and embankment toe drains installed during the Stage 1 construction are assumed to be in good condition and able to resume operation;
- The liner installed in the Return Water Pond and Water Polishing Pond is in good condition and the ponds can be used to store the flows from the toe drain and underdrain system;

- The tailings distribution header can be re-used for each stage through proper construction sequencing;
- Storage of new tailings, with different chemistry than the old tailings, can be placed in the facility without negatively impacting the reclaimed water;
- The raises to the existing facility, which are similar to those in the original design, can be permitted for construction through the NT of Australia government entities without significant changes to the containment system; and
- The thickened tailings can be pumped to their final spigot points, where they will be deposited and will consolidate to an in-place density of 1.6 tonnes per cubic meter.

The raises to the TSF and the estimated construction costs per stage are detailed in Appendix L and are summarized in TABLE 19-30 below.

TABLE 19-30:COST ESTIMATE BY STAGE FOR TSF1
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2010

TSF Stage	Embankment Elevation (m)	Construction Method	Storage Capacity (million tonnes)	Estimated Life of Stage (years)	Construction Cost (million $)
Stage 2	146.5	Centerline	23.9	2.1	9.5
Stage 3	149.0	Upstream	7.9	0.9	1.1
Stage 4	151.5	Upstream	8.1	0.8	0.8
Stage 5	154.0	Upstream	8.2	0.8	0.8
Stage 6	156.5	Upstream	8.3	0.8	0.8
Stage 7	158.0	Upstream	5.0	0.4	0.5
Total			61.4	5.8	13.5

19.11.2 New Facility – TSF2

The results of the tradeoff study indicated that the most cost effective option for a new TSF is the 100-million tonne new TSF using upstream raise methods with thickened tailings. Upstream construction methods are more cost effective than downstream methods for the following reasons:

- The footprint of the facility is smaller, yielding less costs required for clearing and grubbing, underdrains, liner, and overdrains;
- The quantity of waste rock required is much lower;
- The quantity of liner is much smaller; and
- The ultimate tailings area is smaller, yielding lower closure costs.

TSF2 will be constructed in a total of four stages. The embankment will be constructed mostly from waste rock fill using centerline construction methods and will have a one-meter wide filter zone on the upstream face. The crest will be 30 m wide, with 3:1 (horizontal to vertical) upstream and downstream slopes and a five-meter wide bench on the downstream crest at each stage. Including the benches, the downstream slope will be constructed at an overall slope of 3.2H:1V. In addition to the embankment construction, each stage will require an extension of the toe drains, liner, and underdrain and overdrain system.

Several key assumptions were made in the TSF2 design and cost estimate:

- A minimum of one meter of freeboard is required for the TSF at all stages of operation;
- The tailings can be distributed via gravity pipeline from a central distribution tank located at the high point on the crest and will consolidate to an in-place density of 1.6 tonnes per cubic meter;
- The tailings distribution header can be re-used for each stage through proper construction sequencing; and
- The design can be permitted for construction through the NT of Australia government entities without significant changes to the containment system.

The raises to the TSF and the estimated construction costs per stage are detailed in Appendix M and are summarized in TABLE 19-31 below.

TABLE 19-31: COST ESTIMATE BY STAGE FOR TSF2
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

TSF Stage	Embankment Elevation (m)	Construction Method	Storage Capacity (million tonnes)	Estimated Life of Stage (years)	Construction Cost (million $)
Stage 1	140.0	Native Ground	8.3	0.7	39.8
Stage 2	156.0	Upstream	31.0	3.1	26.3
Stage 3	172.0	Upstream	31.7	2.8	5.6
Stage 4	189.0	Upstream	29.3	2.8	1.7
Total			100.3	9.4	73.4

19.12 Cash Flow Analysis

The cash flow analysis developed for the mining, processing, tailings disposal and reclamation of the Mt. Todd Reserve Case 11 Mtpy scenario includes the following input parameters:

- Reserve Case Gold price of $1,000 per ounce, the current 3-year trailing average gold price.
- Metallurgical process recovery of 82 percent.
- An exchange rate US/AUD dollar of 0.85.

Unless specifically noted, all monetary values in the entire document are in US dollars.

19.12.1 Operating Costs

TABLE 19-32 details the mine operating costs by year for the 11 Mtpy Reserve Case.

The Reserve Case process operating cost range from $6.77 to $6.79 ore feed (including water treatment and tailings management costs) during the years of operation. The process plant operating costs by year are given in TABLE 19-33.

In addition to the above mine and process operating costs, Tetra Tech has assessed the following costs as part of the cash flow analyses:

- Open Pit Mine operating cost range from a high of $5.62/t ore occurring in year 7 to $2.73/t ore in year 13.
- G & A at $0.552 per tonne of ore processed
- Gold doré refining, transport and treatment charges are $4.50/toz Au

19.12.2 Reserve Case Results

TABLE 19-34 presents the cash flow summary for the Reserve Case production rate of 11 Mtpy (~30Ktpd, 365dpy), at a Au price of $1,000/toz Au, a US to Australian currency exchange rate of 0.85, and constant 2010 US dollars. Results for the Reserve Case scenario include a before tax net present value (NPV) of $385.336 million for the project evaluated at a 5 percent discount rate. Pretax Internal Rate of Return (IRR) is 13.9 percent. Capital and preproduction costs occur primarily in the two years prior to commencement of operations (Years -2 and -1); however, Year 1 also includes additional capital spending.

TABLE 19-32: MINE OPERATING COST SUMMARY (000)
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Ore Mined	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	775
Total mining costs	50,882	55,947	55,555	55,046	49,107	41,713	59,865	46,330	32,800	58,451	23,991	39,725	29,086	9,747	1,145
Mine Operating Cost / tonne	$4.78	$5.25	$5.22	$5.17	$4.61	$3.92	$5.62	$4.35	$3.08	$5.49	$2.25	$3.73	$2.73	$0.92	$1.48

TABLE 19-33: PROCESS OPERATING COST SUMMARY (000)
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Ore Processed	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	775
Total processing costs	72,159	72,109	72,120	72,080	72,169	72,200	72,366	72,286	72,277	72,213	72,213	72,201	72,019	72,068	5,535
Ore Processing Cost / tonne	$6.78	$6.77	$6.77	$6.77	$6.78	$6.78	$6.79	$6.79	$6.79	$6.78	$6.78	$6.78	$6.76	$6.77	$7.14

TABLE 19-34: MT TODD 10.65 MTPY RESERVE CASE, VISTA GOLD CORP - MT TODD GOLD PROJECT, January, 2011

Mt. Todd - 10.65Mtpa (28 January 2011)

PRETAX:	
IRR	13.9%
NPV0 (000'S)	$964,514
NPV5 (000'S)	$385,336
AVG ANNUAL CF (000's) PRODUCTION YEARS	$97,094
AVG ANNUAL CF (000's) LIFE OF MINE	$56,016
STRIPPING RATIO (WST:ORE)	1.81

AFTER-TAX:	
IRR	10.7%
NPV0 (000'S)	$584,562
NPV5 (000'S)	$184,312
AVG ANNUAL CF (000's) PRODUCTION YEAR	$71,764
AVG ANNUAL CF (000's) LIFE OF MINE	$41,403
PAYBACK PERIOD (YRS) FROM : START OF PRODUCTION	7.2
POST CLOSURE NET CASH FLOW:	$92,460

CAPITAL	
INITIAL CAPITAL (000'S)	$538,330
CONTINGENCY	$60,781
SUB-TOTAL	599,111
WORKING CAPITAL - YR -2 TO YR 1	(9,528)
INITIAL CAP, PRE-PROD DEV & WORKING CAP	$589,583
SUSTAINING CAPITAL (000'S)	235,998
CONTINGENCY	15,878
TOTAL SUSTAINING CAPITAL	251,876
WORKING CAPITAL - YR 2 TO YR 15	9,630
TOTAL MINE LIFE CAPITAL	$851,088

COSTS	
CASH OPER COST PER OUNCE	$520
TOTAL CASH COST PER OUNCE	$530
CAPITAL COST PER OUNCE	$231
TOTAL PRODUCTION COST PER OUNCE	$761
UNIT COSTS	
MINING COST ($/TONNE MINED)	$1.68
MINING COST ($/TONNE ORE)	$4.07
PROCESSING COST ($/TONNE ORE)	$6.847
G&A Cost ($/TONNE ORE)	$0.55
TOTAL OPERATING COSTS $/TONNE ORE	$11.47

PROJECT PRODUCTION SCHEDULE / GOLD GRADES AND CONTENT

		Total LOM	Project Year -2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
MINE																													
ORE TONNAGE TO CRUSHER (000's)	ore tonnes	149,875			10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	775										
ORE GRADE	g Au/tonne	0.853			0.93	1.02	0.95	0.95	0.61	0.62	0.87	0.77	0.63	0.86	0.92	1.04	1.13	0.66	0.47										
	toz Au/tonne	0.027			0.030	0.033	0.031	0.031	0.020	0.020	0.028	0.025	0.020	0.028	0.030	0.034	0.036	0.021	0.015										
CONTAINED GOLD	g Au	127,900,394			9,950,173	10,876,180	10,118,272	10,143,927	6,472,261	6,611,008	9,267,222	8,213,125	6,701,086	9,193,713	9,779,633	11,127,334	12,003,292	7,081,749	361,418										
	toz Au	4,112,090			319,905	349,677	325,310	326,135	208,088	212,549	297,948	264,058	215,445	295,585	314,422	357,752	385,914	227,683	11,620										
WASTE TONNAGE MINED (000's)	waste tonnes	271,480		6,287	22,965	25,048	24,400	25,578	27,824	25,041	24,662	24,710	22,655	20,386	14,158	5,940	1,805	22											
CAPITALIZED TONS (included in total material mined)	kt	57,954		6,287		700	340	360	5,795	10,200		6,972	13,200		13,833	267													
TOTAL MATERIAL MINED	total tonnes	421,354		6,287	33,615	35,698	35,050	36,228	38,474	35,691	35,312	35,360	33,304	31,036	24,808	16,590	12,455	10,672	775										
STRIPPING RATIO	waste : ore	1.8			2.16	2.35	2.29	2.40	2.61	2.35	2.32	2.32	2.13	1.91	1.33	0.56	0.17	0.00											
MILL																													
ORE TONNAGE TO MILL (000's)	ore tonnes	149,875			10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	775										
MILL FEED GRADE	g Au/tonne	0.853			0.93	1.02	0.95	0.95	0.61	0.62	0.87	0.77	0.63	0.86	0.92	1.04	1.13	0.66	0.47										
	toz Au/tonne	0.027			0.030	0.033	0.031	0.031	0.020	0.020	0.028	0.025	0.020	0.028	0.030	0.034	0.036	0.021	0.015										
CONTAINED GOLD	g Au	127,900,394			9,950,173	10,876,180	10,118,272	10,143,927	6,472,261	6,611,008	9,267,222	8,213,125	6,701,086	9,193,713	9,779,633	11,127,334	12,003,292	7,081,749	361,418										
	toz Au	4,112,090			319,905	349,677	325,310	326,135	208,088	212,549	297,948	264,058	215,445	295,585	314,422	357,752	385,914	227,683	11,620										
MILL RECOVERY @ 82%	% recovery of Au	82%			82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%										
GOLD RECOVERED	g Au	104,878,323			8,159,142	8,918,467	8,296,983	8,318,020	5,307,254	5,421,027	7,599,122	6,734,763	5,494,890	7,538,845	8,019,299	9,124,414	9,842,699	5,807,034	296,362										
	toz Au	3,371,914			262,322	286,735	266,754	267,430	170,632	174,290	244,317	216,527	176,665	242,379	257,826	293,357	316,450	186,700	9,528										
REFINERY																													
PAYABLE GOLD TO REFINERY	g Au	104,878,323			8,159,142	8,918,467	8,296,983	8,318,020	5,307,254	5,421,027	7,599,122	6,734,763	5,494,890	7,538,845	8,019,299	9,124,414	9,842,699	5,807,034	296,362										
	toz Au	3,371,914			262,322	286,735	266,754	267,430	170,632	174,290	244,317	216,527	176,665	242,379	257,826	293,357	316,450	186,700	9,528										

		Total LOM	Project Year -2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
GOLD PRICE	$/oz	$1,000			$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000										
WASTE TONNES	000's	271,480		6,287	22,965	25,048	24,400	25,578	27,824	25,041	24,662	24,710	22,655	20,386	14,158	5,940	1,805	22											
TONNES ORE TO MILL	000's	149,875			10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	775										
STRIPPING RATIO	waste:ore	1.81			2.16	2.35	2.29	2.40	2.61	2.35	2.32	2.32	2.13	1.91	1.33	0.56	0.17	0.00											
OUNCES PAYABLE	toz Au.	3,371,914			262,322	286,735	266,754	267,430	170,632	174,290	244,317	216,527	176,665	242,379	257,826	293,357	316,450	186,700	9,528										
GOLD GRADE	g/tonne	0.853			0.934	1.021	0.950	0.952	0.608	0.621	0.870	0.771	0.629	0.863	0.918	1.045	1.127	0.665	0.466				-	-	-	-	-	-	
GROSS GOLD SALES	$000's	$3,371,914			$262,322	$286,735	$266,754	$267,430	$170,632	$174,290	$244,317	$216,527	$176,665	$242,379	$257,826	$293,357	$316,450	$186,700	$9,528										
RENTAL INCOME/POWER INCOME	$000's	$208,312			$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$5,145	$16,159	$16,256	$16,265	$16,478	$16,478	$16,478	$16,490	$16,531		
GROSS REVENUE	$000's	$3,580,225			$267,467	$291,880	$271,899	$272,575	$175,777	$179,435	$249,462	$221,672	$181,810	$247,524	$262,971	$298,501	$321,595	$191,845	$14,673	$16,159	$16,256	$16,265	$16,478	$16,478	$16,478	$16,490	$16,531		
LESS REFINING, TRANS. & TREATMENT	$000's	15,174			1,180	1,290	1,200	1,203	768	784	1,099	974	795	1,091	1,160	1,320	1,424	840	43										
REVENUE FROM SALES	$000's	3,565,052			266,287	290,590	270,698	271,372	175,009	178,650	248,363	220,698	181,015	246,433	261,811	297,181	320,171	191,005	14,630	16,159	16,256	16,265	16,478	16,478	16,478	16,490	16,531		
LESS ROYALTY JAAC	$000's	33,719			2,623	2,867	2,668	2,674	1,706	1,743	2,443	2,165	1,767	2,424	2,578	2,934	3,164	1,867	95										
NET REVENUE	$000's	$3,531,333			$263,664	$287,722	$268,031	$268,697	$173,303	$176,908	$245,920	$218,533	$179,248	$244,010	$259,233	$294,248	$317,006	$189,138	$14,535	$16,159	$16,256	$16,265	$16,478	$16,478	$16,478	$16,490	$16,531		
NET REVENUE AFTER PRODUCTION	$131,138																												

		Total LOM	Project Year -2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
OPERATING COSTS																													
MINE	$000's	609,389			50,882	55,947	55,555	55,046	49,107	41,713	59,865	46,330	32,800	58,451	23,991	39,725	29,086	9,747	1,145										
MILL	$000's	1,026,251	2,291	3,254	72,159	72,109	72,120	72,080	72,169	72,200	72,366	72,286	72,277	72,213	72,213	72,201	72,019	72,068	5,535	944	838	830	377	377	377	364	317	268	
G&A	$000's	82,786		5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	548										
RECLAMATION	$000's	67,864			2,560	161	526	124	511	393	4,114	17,190	3,406	1,149	1,378		278	34	2,056	10,478	10,166	10,755	385	385	385	385	385	658	
TOTAL OPERATING COSTS		$1,786,290	$2,291	$8,737	$131,084	$133,699	$133,683	$132,731	$127,268	$119,788	$141,827	$141,289	$113,966	$137,295	$103,064	$117,408	$106,866	$87,331	$9,284	$11,423	$11,004	$11,585	$763	$763	$763	$749	$702	$927	
MILL OPERATING COSTS AFTER PRODUCTION	4,693																												
RECLAMATION COSTS AFTER PRODCTION	33,985																												
OPERATING MARGIN	$000's	$1,745,043	($2,291)	($8,737)	$132,579	$154,023	$134,348	$135,966	$46,034	$57,119	$104,092	$77,244	$65,282	$106,714	$156,169	$176,839	$210,140	$101,807	$5,251	$4,737	$5,253	$4,681	$15,716	$15,716	$15,716	$15,741	$15,829	($927)	
CAPITAL COSTS																													
MINE EQUIPMENT	$000's	134,667		72,166	21,930	4,933		3,249	15,932	2,863		413	2,836	7,482	2,836	27													
PLANT EQUIPMENT & CONSTRUCTION	$000's	361,686	30,779	330,906			0	(0)																					
TSF Fine Grading, Equipment, Piping, Drains	$000's	71,304		5,258		505	247	267	252	192	34,980	23,192		4,940			1,472												
TSF Bulk Earthwork	$000's	88,555		4,193		1,057	496	527	9,485	17,240			24,818		30,127	614													
OTHER/CONTINGENCY/EPCM	$000's	194,774	15,279	140,528	1,942	376	62	270	1,322	194	8,745	3,259	779	2,074	142	4	7,620			133	426	1,260	555					9,804	
SUB-TOTAL	$000's	$850,987	$46,059	$553,052	$23,872	$6,871	$804	$4,312	$26,991	$20,488	$43,725	$26,864	$28,433	$14,496	$33,104	$645	$9,091			$133	$426	$1,260	$555					$9,804	
SALVAGE VALUE	$000's	(70,559)																	(57,372)									(13,187)	
TOTAL CAPITAL	$000's	$780,427	$46,059	$553,052	$23,872	$6,871	$804	$4,312	$26,991	$20,488	$43,725	$26,864	$28,433	$14,496	$33,104	$645	$9,091		($57,372)	$133	$426	$1,260	$555					($3,383)	
CHANGES TO WORKING CAPITAL	$000's	102	2	3,635	(13,164)	2,533	(148)	94	(89)	578	(840)	810	787	(1,685)	122	(1,215)	1,177	291	6,553	585	15	7	(9)	(4)		(0)	(0)	68	
PRE-TAX CASH FLOWS	$000's	$964,514	($48,352)	($565,423)	$121,872	$144,620	$133,692	$131,560	$19,132	$36,053	$61,208	$49,570	$36,062	$93,903	$122,943	$177,410	$199,872	$101,516	$56,071	$4,019	$4,812	$3,413	$15,169	$15,719	$15,716	$15,741	$15,830	$2,388	
CUMM. PRE-TAX CASH FLOWS	$000's	$964,514	($48,352)	($613,775)	($491,903)	($347,283)	($213,591)	($82,031)	($62,899)	($26,846)	$34,362	$83,931	$119,993	$213,896	$336,839	$514,249	$714,121	$815,637	$871,707	$875,726	$880,538	$883,951	$899,120	$914,839	$930,555	$946,296	$962,126	$964,514	
DD&A	$000's	850,987	9,212	117,199	121,974	123,348	123,509	115,159	14,071	13,394	20,765	25,977	30,801	27,362	29,886	21,270	17,715	12,028	9,129	2,535	2,491	1,486	475	475	448	363	111	9,804	
PROFIT BEFORE TAX	$000's	894,056	(11,503)	(126,949)	7,752	25,087	5,720	15,095	26,278	38,370	81,754	62,763	32,524	75,502	123,665	152,898	190,697	88,094	(1,947)	12,680	12,927	13,949	15,626	15,626	15,653	15,763	16,104	(10,073)	
INCOME TAX - Australian & Northern Territories	$000's	379,952								4,163	35,909	21,802	14,417	34,160	54,809	67,695	83,912	38,747			326	958	4,572	4,572	4,580	4,613	4,715		
NET INCOME AFTER TAXES	$000's	$514,105	($11,503)	($126,949)	$7,752	$25,087	$5,720	$15,095	$26,278	$34,207	$45,845	$40,960	$18,107	$41,342	$68,856	$85,202	$106,786	$49,346	($1,947)	$12,680	$12,601	$12,991	$11,054	$11,054	$11,073	$11,150	$11,388	($10,073)	
AFTER-TAX CASH FLOW	$000's	$584,562	($48,352)	($565,423)	$121,872	$144,620	$133,692	$131,560	$19,132	$31,890	$25,299	$27,767	$21,645	$59,743	$68,133	$109,715	$115,960	$62,768	$56,071	$4,019	$4,486	$2,455	$10,597	$11,147	$11,135	$11,128	$11,114	$2,388	
CUMM. AFTER-TAX CASH FLOW	$000's	$584,562	($48,352)	($613,775)	($491,903)	($347,283)	($213,591)	($82,031)	($62,899)	($31,009)	($5,710)	$22,057	$43,702	$103,446	$171,579	$281,293	$397,254	$460,022	$516,093	$520,111	$524,598	$527,053	$537,650	$548,797	$559,932	$571,060	$582,174	$584,562	

19.12.3 Sensitivities Deviating from the Reserve Case

TABLE 19-35 presents a sensitivity analysis in which the price of Au is increased to $1,350/toz Au and the US to Australian dollar exchange rate is increased to 1.0; with all other inputs being held constant. Using these values for the price and currency exchange rate parameters results in a before tax NPV of $944.470 million for the project, again at a five percent discount rate. Pretax IRR for this scenario increases to 23.2 percent from 13.9 percent.

TABLE 19-36 presents a sensitivity analysis in which the price of Au is decreased to $950/toz Au and the US to Australian dollar exchange rate remains at the base rate of 0.85; with all other inputs being held constant. Using these values for the price and currency exchange rate parameters results in a before tax NPV of $274.047 million for the project, again at a 5 percent discount rate. Pretax IRR for this scenario decreases to 11.5 percent from 13.9 percent.

TABLE 19-37 summarizes the sensitivity of Net Present Value (NPV) of the projected cash flows to variations in Reserve Case gold price, operating cost, capital cost and US/AUD exchange rate. Note both pretax and after tax results are shown. Further note that the US/AUD exchange rate is reset to 1.0 for all situations in which a gold price of $1,200 or higher is used. Results indicate that the Mt. Todd project, as modeled by the Reserve Case, is robust and capable of maintaining profitability even in times of low gold price, higher operating costs, or in the advent of a higher capital cost.

TABLE 19-35: MT TODD 10.65 MTPY SENSITIVITY TO THE RESERVE CASE ($1,350/TOZ AU PRICE AND 1:1 CURRENCY EXCHANGE RATE), VISTA GOLD CORP - MT TODD GOLD PROJECT, January, 2011

Mt. Todd - 10.65Mtpa (28 January 2011)

PRETAX:	
IRR	23.2%
NPV0 (000'S)	$1,860,112
NPV5 (000'S)	$944,470
AVG ANNUAL CF (000's) PRODUCTION YEARS	$157,610
AVG ANNUAL CF (000's) LIFE OF MINE	$90,929
STRIPPING RATIO (WST:ORE)	1.81

AFTER-TAX:	
IRR	16.6%
NPV0 (000'S)	$1,059,338
NPV5 (000'S)	$475,309
AVG ANNUAL CF (000's) PRODUCTION YEAR	$104,226
AVG ANNUAL CF (000's) LIFE OF MINE	$60,130
PAYBACK PERIOD (YRS) FROM : START OF PRODUCTION	3.8
POST CLOSURE NET CASH FLOW:	$116,352

CAPITAL	
INITIAL CAPITAL (000'S)	$616,107
CONTINGENCY	$70,015
SUB-TOTAL	686,121
WORKING CAPITAL - YR -2 TO YR 1	(10,165)
INITIAL CAP, PRE-PROD DEV & WORKING CAP	$675,957
SUSTAINING CAPITAL (000'S)	235,998
CONTINGENCY	15,878
TOTAL SUSTAINING CAPITAL	251,876
WORKING CAPITAL - YR 2 TO YR 15	10,291
TOTAL MINE LIFE CAPITAL	$938,124

COSTS	
CASH OPER COST PER OUNCE	$587
TOTAL CASH COST PER OUNCE	$600
CAPITAL COST PER OUNCE	$256
TOTAL PRODUCTION COST PER OUNCE	$856
UNIT COSTS	
MINING COST ($/TONNE MINED)	$1.78
MINING COST ($/TONNE ORE)	$4.31
PROCESSING COST ($/TONNE ORE)	$8.026
G&A Cost ($/TONNE ORE)	$0.65
TOTAL OPERATING COSTS $/TONNE ORE	$12.99

PROJECT PRODUCTION SCHEDULE / GOLD GRADES AND CONTENT

		Total LOM	Project Year -2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
MINE																													
ORE TONNAGE TO CRUSHER (000's)	ore tonnes	149,875			10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	775										
ORE GRADE	g Au/tonne	0.853			0.93	1.02	0.95	0.95	0.61	0.62	0.87	0.77	0.63	0.86	0.92	1.04	1.13	0.66	0.47										
	toz Au/tonne	0.027			0.030	0.033	0.031	0.031	0.020	0.020	0.028	0.025	0.020	0.028	0.030	0.034	0.036	0.021	0.015										
CONTAINED GOLD	g Au	127,900,394			9,950,173	10,876,180	10,118,272	10,143,927	6,472,261	6,611,008	9,267,222	8,213,125	6,701,086	9,193,713	9,779,633	11,127,334	12,003,292	7,081,749	361,418										
	toz Au	4,112,090			319,905	349,677	325,310	326,135	208,088	212,549	297,948	264,058	215,445	295,585	314,422	357,752	385,914	227,683	11,620										
WASTE TONNAGE MINED (000's)	waste tonnes	271,480		6,287	22,965	25,048	24,400	25,578	27,824	25,041	24,662	24,710	22,655	20,386	14,158	5,940	1,805	22											
CAPITALIZED TONS (included in total material mined)	kt	57,954		6,287		700	340	360	5,795	10,200		6,972	13,200		13,833	267													
TOTAL MATERIAL MINED	total tonnes	421,354		6,287	33,615	35,698	35,050	36,228	38,474	35,691	35,312	35,360	33,304	31,036	24,808	16,590	12,455	10,672	775										
STRIPPING RATIO	waste : ore	1.8			2.16	2.35	2.29	2.40	2.61	2.35	2.32	2.32	2.13	1.91	1.33	0.56	0.17	0.00											
MILL																													
ORE TONNAGE TO MILL (000's)	ore tonnes	149,875			10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	775										
MILL FEED GRADE	g Au/tonne	0.853			0.93	1.02	0.95	0.95	0.61	0.62	0.87	0.77	0.63	0.86	0.92	1.04	1.13	0.66	0.47										
	toz Au/tonne	0.027			0.030	0.033	0.031	0.031	0.020	0.020	0.028	0.025	0.020	0.028	0.030	0.034	0.036	0.021	0.015										
CONTAINED GOLD	g Au	127,900,394			9,950,173	10,876,180	10,118,272	10,143,927	6,472,261	6,611,008	9,267,222	8,213,125	6,701,086	9,193,713	9,779,633	11,127,334	12,003,292	7,081,749	361,418										
	toz Au	4,112,090			319,905	349,677	325,310	326,135	208,088	212,549	297,948	264,058	215,445	295,585	314,422	357,752	385,914	227,683	11,620										
MILL RECOVERY @ 82%	% recovery of Au	82%			82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%										
GOLD RECOVERED	g Au	104,878,323			8,159,142	8,918,467	8,296,983	8,318,020	5,307,254	5,421,027	7,599,122	6,734,763	5,494,890	7,538,845	8,019,299	9,124,414	9,842,699	5,807,034	296,362										
	toz Au	3,371,914			262,322	286,735	266,754	267,430	170,632	174,290	244,317	216,527	176,665	242,379	257,826	293,357	316,450	186,700	9,528										
REFINERY																													
PAYABLE GOLD TO REFINERY	g Au	104,878,323			8,159,142	8,918,467	8,296,983	8,318,020	5,307,254	5,421,027	7,599,122	6,734,763	5,494,890	7,538,845	8,019,299	9,124,414	9,842,699	5,807,034	296,362										
	toz Au	3,371,914			262,322	286,735	266,754	267,430	170,632	174,290	244,317	216,527	176,665	242,379	257,826	293,357	316,450	186,700	9,528										

		Total LOM	Project Year -2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
GOLD PRICE	$/oz	$1,350			$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350	$1,350										
WASTE TONNES	000's	271,480		6,287	22,965	25,048	24,400	25,578	27,824	25,041	24,662	24,710	22,655	20,386	14,158	5,940	1,805	22											
TONNES ORE TO MILL	000's	149,875			10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	775										
STRIPPING RATIO	waste:ore	1.81			2.16	2.35	2.29	2.40	2.61	2.35	2.32	2.32	2.13	1.91	1.33	0.56	0.17	0.00											
OUNCES PAYABLE	toz Au.	3,371,914			262,322	286,735	266,754	267,430	170,632	174,290	244,317	216,527	176,665	242,379	257,826	293,357	316,450	186,700	9,528										
GOLD GRADE	g/tonne	0.853			0.934	1.021	0.950	0.952	0.608	0.621	0.870	0.771	0.629	0.863	0.918	1.045	1.127	0.665	0.466				-	-	-	-	-	-	
GROSS GOLD SALES	$000's	$4,552,084			$354,135	$387,092	$360,118	$361,031	$230,353	$235,291	$329,828	$292,312	$238,497	$327,212	$348,065	$396,031	$427,207	$252,045	$12,863										
RENTAL INCOME/POWER INCOME	$000's	$245,819			$6,053	$6,053	$6,053	$6,053	$6,053	$6,053	$6,053	$6,053	$6,053	$6,053	$6,053	$6,053	$6,053	$6,053	$6,053	$19,146	$19,243	$19,252	$19,465	$19,465	$19,465	$19,477	$19,518		
GROSS REVENUE	$000's	$4,797,903			$360,188	$393,145	$366,171	$367,084	$236,406	$241,344	$335,881	$298,365	$244,550	$333,265	$354,118	$402,084	$433,260	$258,098	$18,916	$19,146	$19,243	$19,252	$19,465	$19,465	$19,465	$19,477	$19,518		
LESS REFINING, TRANS. & TREATMENT	$000's	15,174			1,180	1,290	1,200	1,203	768	784	1,099	974	795	1,091	1,160	1,320	1,424	840	43										
REVENUE FROM SALES	$000's	4,782,730			359,007	391,855	364,970	365,880	235,638	240,560	334,781	297,390	243,755	332,174	352,958	400,764	431,836	257,258	18,873	19,146	19,243	19,252	19,465	19,465	19,465	19,477	19,518		
LESS ROYALTY JAAC	$000's	45,521			3,541	3,871	3,601	3,610	2,304	2,353	3,298	2,923	2,385	3,272	3,481	3,960	4,272	2,520	129										
NET REVENUE		$4,737,209			$355,466	$387,984	$361,369	$362,270	$233,334	$238,207	$331,483	$294,467	$241,370	$328,902	$349,477	$396,803	$427,564	$254,737	$18,744	$19,146	$19,243	$19,252	$19,465	$19,465	$19,465	$19,477	$19,518		
NET REVENUE AFTER PRODUCTION	$155,030																												

		Total LOM	Project Year -2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
OPERATING COSTS																													
MINE	$000's	646,592			53,967	59,166	58,802	58,291	52,131	44,342	63,384	49,175	34,904	61,862	25,580	42,179	31,055	10,478	1,278										
MILL	$000's	1,202,879	2,291	3,254	84,710	84,660	84,671	84,631	84,720	84,751	84,917	84,837	84,828	84,764	84,764	84,752	84,570	84,619	6,449	944	838	830	377	377	377	364	317	268	
G&A	$000's	97,395		6,450	6,450	6,450	6,450	6,450	6,450	6,450	6,450	6,450	6,450	6,450	6,450	6,450	6,450	6,450	645										
RECLAMATION	$000's	67,864			2,560	161	526	124	511	393	4,114	17,190	3,406	1,149	1,378		278	34	2,056	10,478	10,166	10,755	385	385	385	385	385	658	
TOTAL OPERATING COSTS	$000's	$2,014,731	$2,291	$9,704	$147,687	$150,437	$150,448	$149,495	$143,811	$135,937	$158,865	$157,652	$129,588	$154,224	$118,171	$133,381	$122,353	$101,580	$10,428	$11,423	$11,004	$11,585	$763	$763	$763	$749	$702	$927	
MILL OPERATING COSTS AFTER PRODUCTION	4,693																												
RECLAMATION COSTS AFTER PRODCTION	33,985																												
OPERATING MARGIN	$000's	$2,722,478	($2,291)	($9,704)	$207,779	$237,547	$210,921	$212,775	$89,524	$102,270	$172,618	$136,816	$111,782	$174,678	$231,306	$263,423	$305,210	$153,157	$8,316	$7,723	$8,239	$7,667	$18,702	$18,702	$18,702	$18,728	$18,816	($927)	
CAPITAL COSTS																													
MINE EQUIPMENT	$000's	134,802		72,302	21,930	4,933		3,249	15,932	2,863		413	2,836	7,482	2,836	27													
PLANT EQUIPMENT & CONSTRUCTION	$000's	424,670	35,368	389,302			0	(0)																					
TSF Fine Grading, Equipment, Piping, Drains	$000's	71,304		5,258		505	247	267	252	192	34,980	23,192		4,940			1,472												
TSF Bulk Earthwork	$000's	88,555		4,193		1,057	496	527	9,485	17,240			24,818		30,127	614													
OTHER/CONTINGENCY/EPCM	$000's	218,666	17,342	162,357	1,942	376	62	270	1,322	194	8,745	3,259	779	2,074	142	4	7,620			133	426	1,260	555					9,804	
SUB-TOTAL	$000's	$937,997	$52,710	$633,411	$23,872	$6,871	$804	$4,312	$26,991	$20,488	$43,725	$26,864	$28,433	$14,496	$33,104	$645	$9,091			$133	$426	$1,260	$555					$9,804	
SALVAGE VALUE	$000's	(75,757)																	(60,243)									(15,515)	
TOTAL CAPITAL	$000's	$862,240	$52,710	$633,411	$23,872	$6,871	$804	$4,312	$26,991	$20,488	$43,725	$26,864	$28,433	$14,496	$33,104	$645	$9,091		($60,243)	$133	$426	$1,260	$555					($5,710)	
CHANGES TO WORKING CAPITAL	$000's	126	39	4,097	(14,300)	2,603	(205)	96	(367)	589	(639)	731	673	(1,496)	166	(1,113)	1,243	(82)	7,337	679	15	7	(9)	(4)		(0)	(0)	68	
PRE-TAX CASH FLOWS	$000's	$1,860,112	($55,040)	($647,213)	$198,208	$228,073	$210,321	$208,366	$62,900	$81,193	$129,532	$109,221	$82,677	$161,677	$198,035	$263,891	$294,876	$153,239	$61,222	$6,910	$7,798	$6,400	$18,155	$18,706	$18,702	$18,728	$18,816	$4,715	
CUMM. PRE-TAX CASH FLOWS	$000's	$1,860,112	($55,040)	($702,252)	($504,045)	($275,971)	($65,650)	$142,717	$205,616	$286,809	$416,341	$525,563	$608,239	$769,916	$967,952	$1,231,843	$1,526,719	$1,679,958	$1,741,180	$1,748,091	$1,755,889	$1,762,289	$1,780,445	$1,799,150	$1,817,853	$1,836,580	$1,855,396	$1,860,112	
DD&A	$000's	852,967	9,212	117,298	122,073	123,447	123,608	115,258	14,170	13,493	20,864	26,076	30,900	27,461	29,985	21,369	17,814	12,127	9,228	2,634	2,590	1,684	475	475	448	363	111	9,804	
PROFIT BEFORE TAX	$000's	1,869,511	(11,503)	(128,015)	82,852	108,511	82,193	91,805	69,668	83,422	150,181	122,235	78,925	143,366	198,703	239,382	285,669	139,345	1,019	15,568	15,814	16,738	18,613	18,613	18,639	18,750	19,090	(10,073)	
INCOME TAX - Australian & Northern Territories	$000's	800,774				26,090	37,535	42,025	32,273	38,180	65,890	47,843	34,706	63,893	87,699	105,621	125,572	61,171		1,253	1,695	1,795	5,468	5,468	5,476	5,509	5,611		
NET INCOME AFTER TAXES	$000's	$1,068,737	($11,503)	($128,015)	$82,852	$82,421	$44,658	$49,780	$37,395	$45,243	$84,291	$74,392	$44,219	$79,473	$111,004	$133,761	$160,097	$78,174	$1,019	$14,314	$14,120	$14,943	$13,145	$13,145	$13,163	$13,241	$13,479	($10,073)	
AFTER-TAX CASH FLOW	$000's	$1,059,338	($55,040)	($647,213)	$198,208	$201,983	$172,786	$166,342	$30,626	$43,014	$63,642	$61,378	$47,971	$97,784	$110,336	$158,270	$169,304	$92,068	$61,222	$5,657	$6,104	$4,605	$12,687	$13,238	$13,226	$13,218	$13,205	$4,715	
CUMM. AFTER-TAX CASH FLOW	$000's	$1,059,338	($55,040)	($702,252)	($504,045)	($302,061)	($129,275)	$37,066	$67,693	$110,706	$174,349	$235,727	$283,697	$381,482	$491,818	$650,088	$819,392	$911,460	$972,683	$978,340	$984,443	$989,048	$1,001,736	$1,014,974	$1,028,200	$1,041,418	$1,054,623	$1,059,338	

TABLE 19-36: MT TODD 10.65 MTPY SENSITIVITY TO THE RESERVE CASE ($950/TOZ AU PRICE AND 0.85:1 CURRENCY EXCHANGE RATE), VISTA GOLD CORP - MT TODD GOLD PROJECT, January, 2011

Mt. Todd - 10.65Mtpa (28 January 2011)

PRETAX:	
IRR	11.5%
NPV0 (000'S)	$797,604
NPV5 (000'S)	$274,047
AVG ANNUAL CF (000's) PRODUCTION YEARS	$86,826
AVG ANNUAL CF (000's) LIFE OF MINE	$50,092
STRIPPING RATIO (WST:ORE)	1.81

AFTER-TAX:	
IRR	8.9%
NPV0 (000'S)	$490,272
NPV5 (000'S)	$116,729
AVG ANNUAL CF (000's) PRODUCTION YEAR	$66,337
AVG ANNUAL CF (000's) LIFE OF MINE	$38,271
PAYBACK PERIOD (YRS) FROM : START OF PRODUCTION	9.3
POST CLOSURE NET CASH FLOW:	$92,460

CAPITAL	
INITIAL CAPITAL (000'S)	$538,330
CONTINGENCY	$60,781
SUB-TOTAL	599,111
WORKING CAPITAL - YR -2 TO YR 1	(9,636)
INITIAL CAP, PRE-PROD DEV & WORKING CAP	$589,475
SUSTAINING CAPITAL (000'S)	235,998
CONTINGENCY	15,878
TOTAL SUSTAINING CAPITAL	251,876
WORKING CAPITAL - YR 2 TO YR 15	9,737
TOTAL MINE LIFE CAPITAL	$851,088

COSTS	
CASH OPER COST PER OUNCE	$520
TOTAL CASH COST PER OUNCE	$529
CAPITAL COST PER OUNCE	$231
TOTAL PRODUCTION COST PER OUNCE	$760
UNIT COSTS	
MINING COST ($/TONNE MINED)	$1.68
MINING COST ($/TONNE ORE)	$4.07
PROCESSING COST ($/TONNE ORE)	$6.847
G&A Cost ($/TONNE ORE)	$0.55
TOTAL OPERATING COSTS $/TONNE ORE	$11.47

PROJECT PRODUCTION SCHEDULE / GOLD GRADES AND CONTENT

MINE		Total LOM	Project Year -2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
ORE TONNAGE TO CRUSHER (000's)	ore tonnes	149,875			10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	775										
ORE GRADE	g Au/tonne	0.853			0.93	1.02	0.95	0.95	0.61	0.62	0.87	0.77	0.63	0.86	0.92	1.04	1.13	0.66	0.47										
	toz Au/tonne	0.027			0.030	0.033	0.031	0.031	0.020	0.020	0.028	0.025	0.020	0.028	0.030	0.034	0.036	0.021	0.015										
CONTAINED GOLD	g Au	127,900,394			9,950,173	10,876,180	10,118,272	10,143,927	6,472,261	6,611,008	9,267,222	8,213,125	6,701,086	9,193,713	9,779,633	11,127,334	12,003,292	7,081,749	361,418										
	toz Au	4,112,090			319,905	349,677	325,310	326,135	208,088	212,549	297,948	264,058	215,445	295,585	314,422	357,752	385,914	227,683	11,620										
WASTE TONNAGE MINED (000's)	waste tonnes	271,480		6,287	22,965	25,048	24,400	25,578	27,824	25,041	24,662	24,710	22,655	20,386	14,158	5,940	1,805	22											
CAPITALIZED TONS (included in total material mined)	kt	57,954		6,287		700	340	360	5,795	10,200		6,972	13,200		13,833	267													
TOTAL MATERIAL MINED	total tonnes	421,354		6,287	33,615	35,698	35,050	36,228	38,474	35,691	35,312	35,360	33,304	31,036	24,808	16,590	12,455	10,672	775										
STRIPPING RATIO	waste : ore	1.8			2.16	2.35	2.29	2.40	2.61	2.35	2.32	2.32	2.13	1.91	1.33	0.56	0.17	0.00											
MILL																													
ORE TONNAGE TO MILL (000's)	ore tonnes	149,875			10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	775										
MILL FEED GRADE	g Au/tonne	0.853			0.93	1.02	0.95	0.95	0.61	0.62	0.87	0.77	0.63	0.86	0.92	1.04	1.13	0.66	0.47										
	toz Au/tonne	0.027			0.030	0.033	0.031	0.031	0.020	0.020	0.028	0.025	0.020	0.028	0.030	0.034	0.036	0.021	0.015										
CONTAINED GOLD	g Au	127,900,394			9,950,173	10,876,180	10,118,272	10,143,927	6,472,261	6,611,008	9,267,222	8,213,125	6,701,086	9,193,713	9,779,633	11,127,334	12,003,292	7,081,749	361,418										
	toz Au	4,112,090			319,905	349,677	325,310	326,135	208,088	212,549	297,948	264,058	215,445	295,585	314,422	357,752	385,914	227,683	11,620										
MILL RECOVERY @ 82%	% recovery of Au	82%			82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%										
GOLD RECOVERED	g Au	104,878,323			8,159,142	8,918,467	8,296,983	8,318,020	5,307,254	5,421,027	7,599,122	6,734,763	5,494,890	7,538,845	8,019,299	9,124,414	9,842,699	5,807,034	296,362										
	toz Au	3,371,914			262,322	286,735	266,754	267,430	170,632	174,290	244,317	216,527	176,665	242,379	257,826	293,357	316,450	186,700	9,528										
REFINERY																													
PAYABLE GOLD TO REFINERY	g Au	104,878,323			8,159,142	8,918,467	8,296,983	8,318,020	5,307,254	5,421,027	7,599,122	6,734,763	5,494,890	7,538,845	8,019,299	9,124,414	9,842,699	5,807,034	296,362										
	toz Au	3,371,914			262,322	286,735	266,754	267,430	170,632	174,290	244,317	216,527	176,665	242,379	257,826	293,357	316,450	186,700	9,528										

		Total LOM	Project Year -2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
GOLD PRICE	$/oz	$950			$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950										
WASTE TONNES	000's	271,480		6,287	22,965	25,048	24,400	25,578	27,824	25,041	24,662	24,710	22,655	20,386	14,158	5,940	1,805	22											
TONNES ORE TO MILL	000's	149,875			10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	775										
STRIPPING RATIO	waste:ore	1.81			2.16	2.35	2.29	2.40	2.61	2.35	2.32	2.32	2.13	1.91	1.33	0.56	0.17	0.00											
OUNCES PAYABLE	toz Au.	3,371,914			262,322	286,735	266,754	267,430	170,632	174,290	244,317	216,527	176,665	242,379	257,826	293,357	316,450	186,700	9,528										
GOLD GRADE	g/tonne	0.853			0.934	1.021	0.950	0.952	0.608	0.621	0.870	0.771	0.629	0.863	0.918	1.045	1.127	0.665	0.466				-	-	-	-	-	-	
GROSS GOLD SALES	**$000's**	**$3,203,318**			**$249,206**	**$272,398**	**$253,416**	**$254,059**	**$162,100**	**$165,575**	**$232,101**	**$205,701**	**$167,831**	**$230,260**	**$244,935**	**$278,689**	**$300,627**	**$177,365**	**$9,052**										
RENTAL INCOME/POWER INCOME	**$000's**	**$208,312**			**$5,145**	**$5,145**	**$5,145**	**$5,145**	**$5,145**	**$5,145**	**$5,145**	**$5,145**	**$5,145**	**$5,145**	**$5,145**	**$5,145**	**$5,145**	**$5,145**	**$5,145**	**$16,159**	**$16,256**	**$16,265**	**$16,478**	**$16,478**	**$16,478**	**$16,490**	**$16,531**		
GROSS REVENUE	**$000's**	**$3,411,630**			**$254,351**	**$277,543**	**$258,561**	**$259,204**	**$167,245**	**$170,720**	**$237,246**	**$210,846**	**$172,976**	**$235,405**	**$250,080**	**$283,834**	**$305,772**	**$182,510**	**$14,197**	**$16,159**	**$16,256**	**$16,265**	**$16,478**	**$16,478**	**$16,478**	**$16,490**	**$16,531**		
LESS REFINING, TRANS. & TREATMENT	$000's	15,174			1,180	1,290	1,200	1,203	768	784	1,099	974	795	1,091	1,160	1,320	1,424	840	43										
REVENUE FROM SALES	$000's	3,396,456			253,171	276,253	257,361	258,000	166,477	169,936	236,147	209,872	172,181	234,315	248,920	282,513	304,348	181,670	14,154	16,159	16,256	16,265	16,478	16,478	16,478	16,490	16,531		
LESS ROYALTY JAAC	$000's	32,033			2,492	2,724	2,534	2,541	1,621	1,656	2,321	2,057	1,678	2,303	2,449	2,787	3,006	1,774	91										
NET REVENUE		**$3,364,423**			**$250,679**	**$273,529**	**$254,827**	**$255,460**	**$164,856**	**$168,280**	**$233,826**	**$207,815**	**$170,503**	**$232,012**	**$246,470**	**$279,727**	**$301,342**	**$179,896**	**$14,063**	**$16,159**	**$16,256**	**$16,265**	**$16,478**	**$16,478**	**$16,478**	**$16,490**	**$16,531**		
NET REVENUE AFTER PRODUCTION	$131,138																												

		Total LOM	Project Year -2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
OPERATING COSTS																													
MINE	$000's	609,389			50,882	55,947	55,555	55,046	49,107	41,713	59,865	46,330	32,800	58,451	23,991	39,725	29,086	9,747	1,145										
MILL	$000's	1,026,251	2,291	3,254	72,159	72,109	72,120	72,080	72,169	72,200	72,366	72,286	72,277	72,213	72,213	72,201	72,019	72,068	5,535	944	838	830	377	377	377	364	317	268	
G&A	$000's	82,786		5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	5,483	548										
RECLAMATION	$000's	67,864			2,560	161	526	124	511	393	4,114	17,190	3,406	1,149	1,378		278	34	2,056	10,478	10,166	10,755	385	385	385	385	385	658	
TOTAL OPERATING COSTS	$000's	$1,786,290	$2,291	$8,737	$131,084	$133,699	$133,683	$132,731	$127,268	$119,788	$141,827	$141,289	$113,966	$137,295	$103,064	$117,408	$106,866	$87,331	$9,284	$11,423	$11,004	$11,585	$763	$763	$763	$749	$702	$927	
MILL OPERATING COSTS AFTER PRODUCTION	4,693																												
RECLAMATION COSTS AFTER PRODCTION	33,985																												
OPERATING MARGIN	**$000's**	**$1,578,133**	**($2,291)**	**($8,737)**	**$119,594**	**$139,830**	**$121,144**	**$122,728**	**$37,588**	**$48,492**	**$91,999**	**$66,526**	**$56,537**	**$94,717**	**$143,406**	**$162,318**	**$194,476**	**$92,565**	**$4,780**	**$4,737**	**$5,253**	**$4,681**	**$15,716**	**$15,716**	**$15,716**	**$15,741**	**$15,829**	**($927)**	
CAPITAL COSTS																													
MINE EQUIPMENT	$000's	134,667		72,166	21,930	4,933		3,249	15,932	2,863		413	2,836	7,482	2,836	27													
PLANT EQUIPMENT & CONSTRUCTION	$000's	361,686	30,779	330,906			0	(0)																					
TSF Fine Grading, Equipment, Piping, Drains	$000's	71,304		5,258		505	247	267	252	192	34,980	23,192		4,940			1,472												
TSF Bulk Earthwork	$000's	88,555		4,193		1,057	496	527	9,485	17,240			24,818		30,127	614													
OTHER/CONTINGENCY/EPCM	$000's	194,774	15,279	140,528	1,942	376	62	270	1,322	194	8,745	3,259	779	2,074	142	4	7,620			133	426	1,260	555					9,804	
SUB-TOTAL	$000's	$850,987	$46,059	$553,052	$23,872	$6,871	$804	$4,312	$26,991	$20,488	$43,725	$26,864	$28,433	$14,496	$33,104	$645	$9,091			$133	$426	$1,260	$555					$9,804	
SALVAGE VALUE	$000's	(70,559)																	(57,372)									(13,187)	
TOTAL CAPITAL	$000's	$780,427	$46,059	$553,052	$23,872	$6,871	$804	$4,312	$26,991	$20,488	$43,725	$26,864	$28,433	$14,496	$33,104	$645	$9,091		($57,372)	$133	$426	$1,260	$555					($3,383)	
CHANGES TO WORKING CAPITAL	$000's	102	2	3,635	(13,272)	2,523	(139)	94	(49)	577	(869)	822	804	(1,712)	115	(1,230)	1,167	345	6,625	589	15	7	(9)	(4)		(0)	(0)	68	
PRE-TAX CASH FLOWS	$000's	$797,604	($48,352)	($565,423)	$108,995	$130,437	$120,479	$118,322	$10,646	$27,427	$49,143	$38,840	$27,301	$81,932	$110,187	$162,903	$184,217	$92,221	$55,526	$4,015	$4,812	$3,413	$15,169	$15,719	$15,716	$15,741	$15,830	$2,388	
CUMM. PRE-TAX CASH FLOWS	$000's	$797,604	($48,352)	($613,775)	($504,780)	($374,343)	($253,864)	($135,542)	($124,896)	($97,469)	($48,326)	($9,486)	$17,815	$99,747	$209,934	$372,837	$557,054	$649,275	$704,801	$708,816	$713,628	$717,041	$732,210	$747,930	$763,645	$779,387	$795,216	$797,604	
DD&A	$000's	850,987	9,212	117,199	121,974	123,348	123,509	115,159	14,071	13,394	20,765	25,977	30,801	27,362	29,886	21,270	17,715	12,028	9,129	2,535	2,491	1,486	475	475	448	363	111	9,804	
PROFIT BEFORE TAX	$000's	727,147	(11,503)	(126,949)	(5,233)	10,894	(7,484)	1,858	17,832	29,743	69,661	52,044	23,779	63,504	110,903	138,377	175,033	78,852	(2,419)	12,680	12,927	13,949	15,626	15,626	15,653	15,763	16,104	(10,073)	
INCOME TAX - Australian & Northern Territories	$000's	307,333									4,401	17,086	10,569	28,881	49,194	61,306	77,019	34,681			184	958	4,572	4,572	4,580	4,613	4,715		
NET INCOME AFTER TAXES	$000's	$419,814	($11,503)	($126,949)	($5,233)	$10,894	($7,484)	$1,858	$17,832	$29,743	$65,260	$34,958	$13,210	$34,624	$61,709	$77,071	$98,014	$44,171	($2,419)	$12,680	$12,743	$12,991	$11,054	$11,054	$11,073	$11,150	$11,388	($10,073)	
AFTER-TAX CASH FLOW	**$000's**	**$490,272**	**($48,352)**	**($565,423)**	**$108,995**	**$130,437**	**$120,479**	**$118,322**	**$10,646**	**$27,427**	**$44,742**	**$21,754**	**$16,732**	**$53,052**	**$60,993**	**$101,597**	**$107,198**	**$57,540**	**$55,526**	**$4,015**	**$4,628**	**$2,455**	**$10,597**	**$11,147**	**$11,135**	**$11,128**	**$11,114**	**$2,388**	
CUMM. AFTER-TAX CASH FLOW	**$000's**	**$490,272**	**($48,352)**	**($613,775)**	**($504,780)**	**($374,343)**	**($253,864)**	**($135,542)**	**($124,896)**	**($97,469)**	**($52,727)**	**($30,973)**	**($14,241)**	**$38,811**	**$99,803**	**$201,401**	**$308,599**	**$366,138**	**$421,665**	**$425,679**	**$430,307**	**$432,762**	**$443,359**	**$454,506**	**$465,641**	**$476,769**	**$487,883**	**$490,272**	

TABLE 19-37: MT TODD 11MTPY SENSITIVITY ANALYSES - MT TODD GOLD PROJECT, January, 2011

PRETAX SENSITIVITY ANALYSES - MT. TODD GOLD PROJECT

Net Present Value Calculations ($000s)
Gold Price Sensitivity

PRICE ($/oz)	IRR	NPV(0)	NPV(5)
$ 850	6.4%	$463,785	$51,470
$ 900	9.0%	$630,695	$162,759
$ 950	11.5%	$797,604	$274,047
$ 1,000	13.9%	$964,514	$385,336
$ 1,050	16.3%	$1,131,424	$496,625
$ 1,100	18.7%	$1,298,334	$607,914
$ 1,150	21.0%	$1,465,243	$719,202
$ 1,200	17.1%	$1,359,383	$610,603
$ 1,250	19.2%	$1,526,292	$721,892
$ 1,300	21.2%	$1,693,202	$833,181
$ 1,350	23.2%	$1,860,112	$944,470
$ 1,400	25.2%	$2,027,022	$1,055,758
$ 1,450	27.2%	$2,193,931	$1,167,047
$ 1,500	29.1%	$2,360,841	$1,278,336
$ 1,550	31.1%	$2,527,751	$1,389,625
$ 1,600	33.0%	$2,694,661	$1,500,914
$ 1,650	34.9%	$2,861,570	$1,612,202
$ 1,700	36.8%	$3,028,480	$1,723,491
$ 1,750	38.7%	$3,195,390	$1,834,780
$ 1,800	40.6%	$3,362,299	$1,946,069
$ 1,850	42.5%	$3,529,209	$2,057,357
$ 1,900	44.4%	$3,696,119	$2,168,646
$ 1,950	46.2%	$3,863,029	$2,279,935
$ 2,000	48.1%	$4,029,938	$2,391,224

Net Present Value Calculations ($000s)
Operating Cost Sensitivity, Au@ $1,000

	IRR	NPV(0)	NPV(5)
+20%	8.6%	$618,950	$150,854
+10%	11.3%	$791,732	$268,095
0%	13.9%	$964,514	$385,336
-10%	16.5%	$1,137,296	$502,577
-20%	19.1%	$1,310,078	$619,819

Net Present Value Calculations ($000s)
Capital Cost Sensitivity, Au @ $1,000

	IRR	NPV(0)	NPV(5)
+20%	10.0%	$810,307	$240,261
+10%	11.8%	$887,411	$312,798
0%	13.9%	$964,514	$385,336
-10%	16.5%	$1,041,618	$457,874
-20%	19.6%	$1,118,721	$530,412

Net Present Value Calculations ($000s)
Exchange Rate Sensitivity

US : AU	IRR	NPV(0)	NPV(5)
0.75 : 1.00	18.3%	$1,146,361	$531,928
0.80 : 1.00	16.0%	$1,055,438	$458,632
0.85 : 1.00	13.9%	$964,514	$385,336
0.90 : 1.00	12.0%	$873,591	$312,040
0.95 : 1.00	10.2%	$782,667	$238,744
1.00 : 1.00	8.5%	$691,744	$165,448
1.05 : 1.00	6.9%	$600,820	$92,152
1.10 : 1.00	5.4%	$509,897	$18,856
1.15 : 1.00	4.0%	$418,973	($54,440)
1.20 : 1.00	2.7%	$328,050	($127,735)
1.25 : 1.00	1.4%	$237,126	($201,031)

AFTERTAX SENSITIVITY ANALYSES - MT. TODD GOLD PROJECT

Net Present Value Calculations ($000s)
Gold Price Sensitivity

PRICE ($/oz)	IRR	NPV(0)	NPV(5)
$ 850	5.1%	$300,104	($26,103)
$ 900	7.0%	$394,470	$46,248
$ 950	8.9%	$490,272	$116,729
$ 1,000	10.7%	$584,562	$184,312
$ 1,050	12.5%	$678,853	$250,607
$ 1,100	14.3%	$773,144	$315,748
$ 1,150	15.9%	$867,435	$380,140
$ 1,200	12.4%	$777,849	$284,528
$ 1,250	13.8%	$872,267	$348,856
$ 1,300	15.2%	$965,803	$412,083
$ 1,350	16.6%	$1,059,338	$475,309
$ 1,400	17.9%	$1,152,874	$538,536
$ 1,450	19.3%	$1,246,409	$601,762
$ 1,500	20.6%	$1,339,945	$664,986
$ 1,550	21.9%	$1,433,480	$728,209
$ 1,600	23.2%	$1,527,898	$791,478
$ 1,650	24.5%	$1,621,433	$853,804
$ 1,700	25.7%	$1,714,969	$916,131
$ 1,750	26.9%	$1,808,504	$978,458
$ 1,800	28.1%	$1,902,040	$1,040,784
$ 1,850	29.4%	$1,995,575	$1,103,111
$ 1,900	30.5%	$2,089,111	$1,165,438
$ 1,950	31.7%	$2,182,646	$1,227,764
$ 2,000	32.9%	$2,276,182	$1,290,091

Net Present Value Calculations ($000s)
Operating Cost Sensitivity, Au@ $1,000

	IRR	NPV(0)	NPV(5)
+20%	6.7%	$386,920	$37,223
+10%	8.8%	$486,327	$112,559
0%	10.7%	$584,562	$184,312
-10%	12.7%	$682,251	$254,214
-20%	14.5%	$779,939	$322,939

Net Present Value Calculations ($000s)
Capital Cost Sensitivity, Au @ $1,000

	IRR	NPV(0)	NPV(5)
+20%	6.7%	$431,265	$39,765
+10%	8.6%	$507,914	$112,038
0%	10.7%	$584,562	$184,312
-10%	13.3%	$661,211	$256,586
-20%	16.5%	$737,860	$328,859

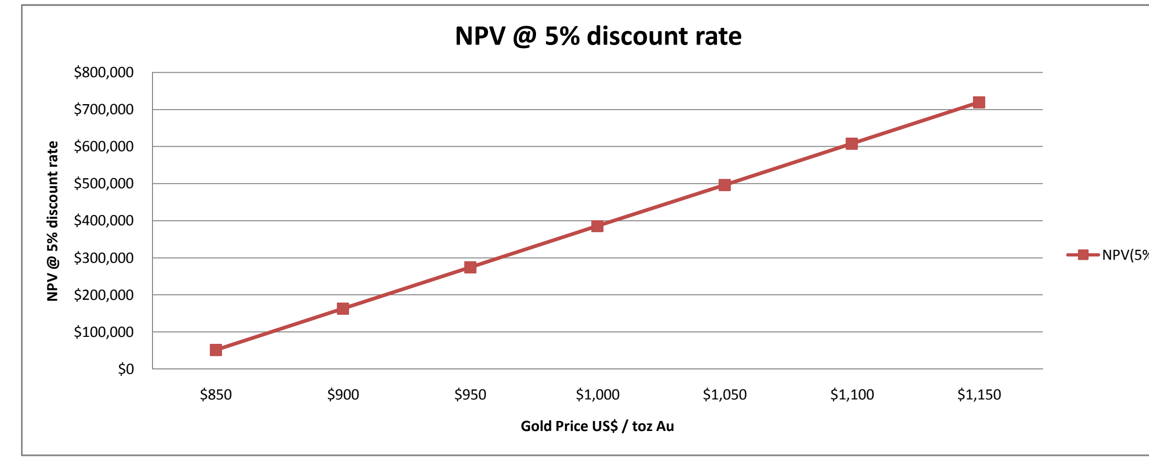
Net Present Value Calculations ($000s)
Exchange Rate Sensitivity

US : AU	IRR	NPV(0)	NPV(5)
0.75 : 1.00	14.5%	$707,568	$292,911
0.80 : 1.00	12.5%	$645,730	$238,535
0.85 : 1.00	10.7%	$584,562	$184,312
0.90 : 1.00	9.1%	$522,597	$129,231
0.95 : 1.00	7.6%	$461,388	$74,511
1.00 : 1.00	6.2%	$399,296	$18,960
1.05 : 1.00	5.0%	$336,194	($37,886)
1.10 : 1.00	3.8%	$273,890	($94,970)
1.15 : 1.00	2.8%	$212,511	($152,104)
1.20 : 1.00	1.9%	$150,080	($210,006)
1.25 : 1.00	1.1%	$88,658	($267,819)

Note: All sensitivities run at gold prices of $1200 / toz Au and greater use a 1.00 : 1.00, US to AUS exchange rate.

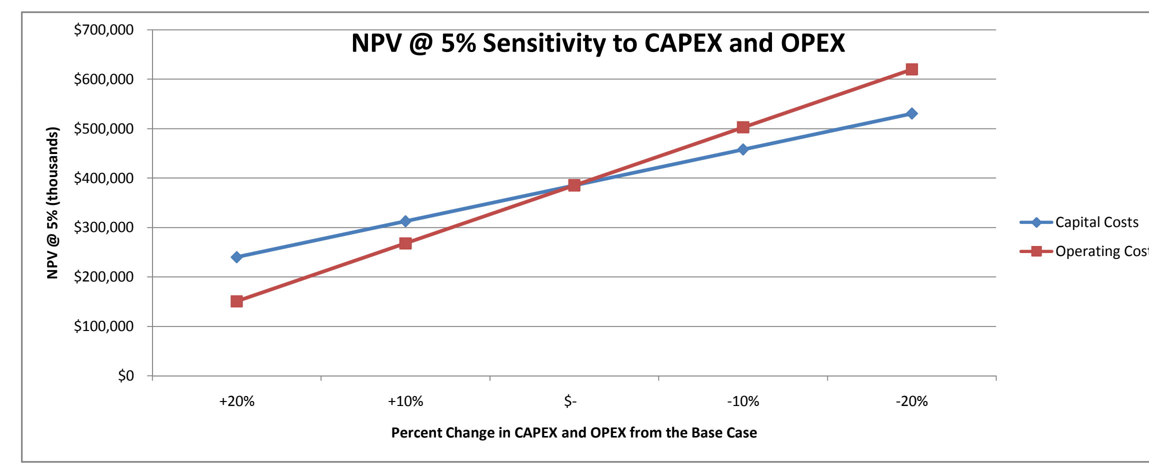
Sensitivity of Pretax Net Present Value @ 5% Discount Rate to Gold Price (thousands)

PRICE ($/oz)	$ 850	$ 900	$ 950	$ 1,000	$ 1,050	$ 1,100	$ 1,150
NPV(5%)	$51,470	$162,759	$274,047	$385,336	$496,625	$607,914	$719,202



Sensitivity of Pretax Net Present Value @5% to CAPEX and OPEX Costs

	+20%	+10%	$ -	-10%	-20%
Capital Costs	$240,261	$312,798	$385,336	$457,874	$530,412
Operating Costs	$ 150,854	$ 268,095	$ 385,336	$ 502,577	$ 619,819



20.0 INTERPRETATION AND CONCLUSIONS

20.1 Interpretation

It is Tetra Tech's opinion that all of the current Vista work meets and/or exceeds the current CIM standards for reporting of mineral resources. Any historic work that does not meet current standards has either been replaced with new data by Vista as part of their ongoing exploration program and/or has been identified within the body of this report. The work completed prior to Vista, was been completed by well-qualified technical professionals, reputable mining companies, and independent third-party contractors and laboratories according to standards that meet most of today's requirements; however, all of the Vista work completed meets and/or exceeds all of the current requirements.

The results of the 2008 Vista exploration and development programs continue to provide strong support that the current geologic model and resource estimates are indicative of the mineralization present at Mt. Todd. In addition, the 2008 exploration program has identified an additional "sympathetic" structure and mineralization east of the main Batman mineralized zone. This new resource area will have significant impact on the waste to ore ratios because it represents previously undefined mineralization as opposed to mineralization that changes from waste to ore due to changing gold prices. The 2008 Batman exploration program was designed to complete four main objectives:

1) Confirmation of the existing geologic and grade model at depth;
2) Confirmation of the previous assaying programs and grades in the assay database;
3) Development of additional definition in the short-range portion of the variogram; and
4) Development of additional measured and indicated mineral resources.

All of these objectives were met and/or exceeded. The results of the 2008 exploration program added approximately 197,000 ounces of gold to the measured resource class and approximately 2,032,000 ounces to the indicated resource class at a 0.4 g Au/t cutoff grade. Measured and indicated resources now account for approximately 70 percent of the known resources at the Batman deposit. Approximately 713,000 ounces of gold were added to the inferred resource class as compared to the March 2008 inferred resource estimate.

Utilizing the above project advances, Tetra Tech, on behalf of Vista, has completed this prefeasibility study and the results of this prefeasibility study continue to show that the project is capable of producing positive economic results and therefore; should continued to be advanced through full feasibility.

20.2 Conclusions

Vista's exploration and development work on the Mt. Todd Gold Project and specifically the Batman and Quigleys deposits continue to provide strong justification for additional expenditures and efforts to develop a new mine at this site. The positive results of this study clearly demonstrate the potential robustness of several different development scenarios.

Exploration Leases

A significant portion of the exploration leases is yet to be systematically explored and evaluated. The broad structural and geologic trends that host the Batman, Quigleys, and Golf Tollis deposits may well host other deposits. Much of what Vista has learned from more detailed exploration of the Batman deposit has yet to be applied to these other areas and therefore, these areas remain highly prospective.

21.0 RECOMMENDATIONS

Based on Tetra Tech's review of the database, previous studies and work products, and as an outgrowth of the recent mineral resource modeling, PEA, and this PFS, Tetra Tech recommends that the project be advanced to a Feasibility Study and detailed engineering in support of the construction of a mine and process facility at the Project. The work programs suggested below involve optimizations typical of a project at this stage of development and in no way reflect material issues to the Project.

21.1 Recommended Work Programs

21.1.1 Resources

Vista's 2008 exploration program on the Batman Deposit provided answers to three major questions; improvement of the short range portion of the gold variogram, infill drilling for improvement in the quantity of measured and indicated resources, and confirmation of the work completed by previous owners/operators. With this in mind, the following recommendations are made for future exploration programs:

- Additional exploration drilling, as the deposit is still open to the north, south, and at depth.
- The 2007 and 2008 exploration drill hole programs have identified what appear to be parallel and/or sub-parallel structures to the east of the main core complex. Additional exploration and definition of these structures is warranted.
- Completion of additional geologic and geotechnical mapping to increase the understanding of the larger system.
- Advance the Batman deposit through feasibility studies in order to advance the project to a development decision.

Quigleys and Golf-Tollis Deposits

The Quigleys and Golf-Tollis deposits appear to be more structurally controlled than Batman with the mineralization occurring in narrower bands. Because of this, additional work will need to be undertaken in order to develop a more accurate geologic model and mineralization controls. Tetra Tech proposes that the following items be considered when preparing the work plan:

- Surface mapping and subsequent re-interpretation of the footwall contact to the shear zone mineralization are recommended. Any additional structural complexity that results should, where appropriate, be used to refine the mineralized envelope upon which modeling updates are based.
- Optimization of the resource provides a focus to define areas requiring further investigation or infill drilling. Due to the high degree of variability in the deposit, infill drilling is best targeted at key areas of geological complexity.
- A model should be developed for the area outside the shear zone. This will require separation of areas of mineralization from unmineralized areas using suitable envelope constraints.
- The cause of an apparent bias between some of the old and new RC drilling should be confirmed to validate the inclusion of all samples in resource calculations.

21.1.2 Mining

Tetra Tech recommends that the following areas be upgraded with additional study to a Bankable Feasibility Document level of development. This work would include:

- Geotechnical and drilling and laboratory testing (analysis to include geotechnical logging results);
- Refinement of cut off grade;
- New pit designs with scheduled haul road movement designs;
- Monthly mine plans for the first two years;
- Quarterly plans for years three and four;
- Annual mine plans through life-of-mine;
- Designed / Scheduled Waste rock facilities for life-of-mine ultimate foot print;
- Mine production schedule with accounted material movement;
- Refined Mine equipment requirements;
- Refined Manpower requirements; and
- Quoted CAPEX and OPEX costing

21.1.3 Metallurgy

Tt, RDi, and Ausenco recommend additional metallurgical testwork and process studies in working toward the feasibility stage of development to validate key metallurgical information, explore possible process improvements, and to reduce process risk.

- Process testwork is proposed on samples representing different rock/ore types within the resource to include extremes in grade, hardness, and associative mineralogy. Such work should be performed for all deposit areas that may ultimately become minable reserves. Several advanced techniques are available through which to perform such work.
- Ore variability testing for the whole ore flowsheet (i.e., transition ore, oxide zone), including ore grade variation and blending should be conducted. Of specific interest in addition to gold leaching and recovery is the copper constituent and potential for deleterious copper loading on the activated carbon, potentially beyond current circuit design capacity.
- Several commercial scale HPGR applications have begun operation in the past 18 months. Undoubtedly, manufactures and the mining industry have learned from these efforts. A study to benchmark the commercial operations against the envisioned application at Mt. Todd including specific energy requirements, circuit design, and wear/maintenance issues is recommended.
- Efforts to optimize the crushing and grinding circuit in general should be continued considering that comminution in total defines a major proportion of both the project capital and operating costs.
- Development of improved blasting techniques to safely produce the fine feed for the crushing circuit has the potential to reduce comminution costs. With regard to comminution, as crushing is more efficient than grinding, so is blasting more efficient than crushing.

- Use of the grind thickener as a precursor to the preaeration unit operation should be optimized. Often the residence time inherent with a grind thickener allows an opportunity for significant geochemical precursor reactions to occur or be in place before the actual preaeration step. This is a logical step in addition to optimization of the entire pre-aeration process so as to minimize the consumption of lime.
- Additional metallurgical testwork should include optimization of oxygen and cyanide concentration in the CIL circuit. Such tests would consider leaching under conditions of decay versus maintenance of NaCN concentration and oxygen content. Further, whole ore leach (WOL) tests should be performed using material crushed by HPGR (without grinding) to investigate if there is potential for a simplified process.
- Confirmation CIL extraction testwork using site water and cyanide destructed tailing water should be conducted as a continuation of the metallurgical testwork.
- Carbon loading and stripping tests should be performed.
- Detoxification process studies on CIL tailings should be performed to investigate different commercial approaches, reagent consumption, and overall effectiveness of such processes on the different ore types that might be encountered at Mt. Todd.
- Slurry rheology tests should be conducted as a component of the metallurgical testwork program as the project moves into the feasibility phase of development. This should include testwork on the thickening of ground material before pre-aeration and the thickening of cyanide destructed leach residues. Such tests will also give information pertinent to slurry pumping, pipelines and the selection and design / layout thereof.
- The ore(s) should be tested for mercury and, if found in significant quantities, provisions should be made in the process flowsheet for mercury capture and condensation.
- The size of the coarse ore storage facility should be studied to determine optimum capacity. Appropriately sized storage will assist in preventing mine delay when the crusher is down and, conversely, crusher delay when the mine is down. Coarse ore storage capacity is tied directly to the mobility of equipment in the pit and the flexibility of the mine plan to switch production from ore to waste.
- Elemental tests of the fuels to be used in the kiln should be performed so as to ensure the selection of the best material for the kiln shell. Some fuels are higher in specific elemental constituents detrimental to specific metals and alloys.

21.1.4 Tailings and Geotechnical Design

The following studies and investigations are recommended for further design reports for both proposed facilities:

- The material properties used in the geotechnical modeling must be confirmed through additional drilling, laboratory testing, and tailings testing;
- The geotechnical modeling must be updated and expanded to include a consolidation analysis, a liquefaction analysis, and deposition modeling;
- A seismic hazard analysis should be performed even though the site is in a relatively inactive seismic area;
- The TSF water balances must be updated to optimize the size of the water pool to accommodate the water requirements of the processing facilities; and

- Engineering analyses should be performed to verify the liner and drain systems can support loads from the tailings and embankment.

For TSF1, the following site-specific studies and investigations should be performed for further design reports:

- The spillway design must be revised to account for any required change in cross-section at different stages of the impoundment; and
- The condition of the existing toe drains, underdrains, and decant towers must be investigated to confirm their operation when tailings deposition resumes.

Further design reports for TSF2 should include the studies listed above, as well as a tradeoff study to determine the feasibility of using a sidehill decant structure instead of a floating barge pump for water reclaim.

21.1.5 Environmental, Permitting, and Reclamation

Environmental Baseline Studies

It is anticipated that additional studies will be needed to further assess environmental baseline conditions to support feasibility-level design, permitting and closure planning of the Mt. Todd Project. Key baseline studies that will need additional information are summarized below. Feasibility-level environmental baseline studies are estimated to cost between $1.8 to $3.2 million. Tetra Tech recommends that this area be given additional study as part of the full feasibility study in order to determine the actual costs for each study.

Hydrogeology

Groundwater at the site is not well characterized at the current time although monitoring wells are present at the site. A detailed assessment and ultimate development of the project will require hydrogeologic investigations to fully characterize the existing groundwater conditions. Characterization is necessary to:

- Provide input to a refined site water balance model;
- Develop a project water management plan, including potential dewatering requirements; and
- Establish a defensible groundwater monitoring program.

The investigations will need to include acquisition and compilation of all significant hydrologic and related information for development of a hydrogeologic characterization, development of a hydrogeologic conceptual model and, if warranted, completion of a pit groundwater inflow analysis.

Data Compilation and Site-wide Hydrogeologic Characterization. A thorough site hydrogeologic characterization will be foundational to understand groundwater flow and solute-transport processes at the site, and possible impacts to the aquifer from mine operation. The characterization will be developed by compilation of existing groundwater and surface water data from approximately 11 monitoring boreholes (MWH, 2006a), the existing pit lake, and from the waste rock dump, heap leach pad, TSF, and other site facilities. Additional data will be collected as necessary. The following information should be assembled for this characterization:

- Regional and deposit scale subsurface geology including distribution of lithologies, alteration (e.g. silicification), sulfide mineralization, and structures (faults, fracture zones, penetrative jointing and cleavage);
- Potentiometric information from groundwater monitoring wells. Additional wells will be required to provide adequate up-gradient background characterization, particularly north of the project site and immediately down-gradient of certain facilities (e.g. Batman Pit, Golf Tollis Pits);

Estimated hydraulic conductivity (likely from slug testing of monitoring wells) or packer testing of any pit-area fracture/fault zones;

Data characterizing the site hydrologic conditions will require synthesis into a comprehensive site hydrogeologic conceptual model to demonstrate:

- The site potentiometric surface and its relationship to the hydrostratigraphy from which groundwater flow direction and gradients may be defined;
- Key geologic controls on the distribution of groundwater;
- Aquifer recharge areas which may potentially include mine pits, TSF, leach pad, water supply reservoir, waste rock dump, and retention ponds;
- Aquifer discharge areas including seepage into down-gradient sections of various creeks, and the Edith River, and extraction wells (if any);
- Definition of hydrogeologic boundary conditions for the site; and

A report detailing the conceptual model should include representative maps, vertical hydrostratigraphic sections, hydrographs, and tables demonstrating and describing hydrogeologic site conditions. The conceptual model will form the basis for potential analytical or numerical modeling, if warranted, to quantitatively assess in greater detail specific hydrologic parameters and components of the site hydrogeologic system. The report description may be a foundational component to future permit documents.

Batman Pit Hydrology. The Mt. Todd hydrogeologic investigations will also serve to improve understanding of the hydrology of expanded pits as deeper reserves are developed. If in development of the hydrogeologic conceptual model, it becomes evident that significant groundwater management may be necessary during mining, analytical solution modeling will need to be completed to assess inflow rates. The modeling should be designed to interactively evaluate and refine dewatering option(s) which may include a combination of approaches such as:

- Perimeter wells;
- Pit wall horizontal drains;
- Pit sumps;
- Grout curtains to impede inflow; and
- Drainage portals pending potential underground mine development.

Modeling may also be used to estimate post-mining pit inflow and ultimate lake levels. Inflow rates, together with wall rock mineralogy and chemistry will be critical parameters to estimate long-term pit lake chemistry. Additional field tests may be required to refine selected designs and allow cost estimation for implementation.

The estimated cost for the hydrogeology program is $500,000 to 750,000 depending on the number of wells that will need to be installed.

Geochemical Characterization and Waste Management Planning

Additional ABA testing is recommended to establish that waste rock to be generated is adequately represented by the characterization program. Analysis of an additional 150 samples (50 from each rock unit) should be sufficient for feasibility-level characterization.

The three waste rock samples currently undergoing accelerated weathering using the standardized humidity cell test procedures should be continued for at least three months (total of one year of data) to obtain stable solute concentrations and to evaluate the longer-term potential of fully oxygenated waste rock to generate/consume acid and produce metal-laden leachate.

A subset of waste rock samples should be subjected to NAG testing (complete oxidation with 15 percent hydrogen peroxide) to evaluate the technique as a field method for waste rock segregation.

Additional tailings samples should be subjected to static testing to confirm the preliminary findings to date. Humidity cell testing should also be initiated to investigate long-term metal leaching and potential to generate acid.

The estimated cost for continuation of the geochemical characterization program is approximately $75,000 to 100,000.

A Waste Rock Management Plan should be developed as part of the feasibility study to specify how waste rock is to be handled to minimize the potential for ARD/ML and maximize the use of non-PAG waste rock for closure. A Tailings Management Plan should also be developed as part of the feasibility study to specify how tailings are to be handled to minimize ARD/ML, and facilitate closure and rapid dewatering and consolidation of tailings.

Development of these plans is estimated to cost between $150,000 and 300,000.

Soils

Soils are a limited resource throughout the Mt. Todd site and additional information will need to be gathered to verify that sufficient quantities will be available for closure of proposed and existing facilities. The adequacy of available soils and growth media for supporting plant growth and suitability for use as liner/cap material needs to be evaluated. A soil resource survey for the Mt. Todd site was identified as a priority study in the BRS Report (DRDPIFR, 2008b). Recommended studies are detailed below under the Additional Closure Requirements.

Cultural/Archeological

Compiling and mapping the previous investigations into archaeological and historical assessments undertaken at the site can be conducted in less than a month and is estimated to cost between $7,500 and $10,000.

Biological Resources

The discussion of recommended baseline studies related to biologic resources is separated into subsections including wildlife, vegetation, and aquatic and benthic studies.

Wildlife. The absence of sensitive species or habitats within the Mt. Todd Project area should be confirmed with additional surveys and habitat mapping to ensure all data is current, including the present distribution of Gouldian finch. While other wildlife species were evaluated in support of the 1992 Draft EIS completed prior to initial development, key indicator species should be reassessed at the same time sensitive species are surveyed to fully characterize the existing use and habitat value at the site. Additional baseline studies can be completed within 6 months to 1 year, depending upon the time of year started, at an estimated cost of $80,000 to $100,000.

Vegetation. A comprehensive study should be conducted to determine habitats and provide further characterization in terms of species richness and abundance, productivity, and plant cover as well as to develop current vegetation community mapping. Special emphasis should be placed on describing characteristics to support reclamation/closure plans and the potential occurrence of endangered, vulnerable or otherwise sensitive species and communities. The estimated cost for this study is between $50,000 and $75,000.

Aquatic and Benthic. It is anticipated that DoR will continue to conduct regular sampling of freshwater fish and macroinvertebrates in the Edith and Fergusson Rivers, and in the Stow Creek. It is not anticipated that additional work beyond the scope of these ongoing studies will be required. In support of permitting efforts, all data collected to date should be reviewed and summarized. The estimated cost for data compilation is between $5,000 and $7,500 and is anticipated to take less than 1 month.

Water Treatment

Based on the goal of the partial dewatering of the Batman Pit by approximately planning year -1 to permit in-pit preparation activities (lay backs) prior to the initiation of mining while meeting the WDL and Edith River water quality–based effluent standards, Tetra Tech recommends that Vista complete the tasks as follows:

- As soon as possible gain approval from the NT Government to permit effluent releases from the existing Water Treatment Plant (WTP) and proposed Water Treatment Plant (New WTP) to Batman Creek or other appropriate discharge location that comply with the requirements of the former of revised Waste Discharge License;
- Initiate dialog with the NT Government to determine if they intend to apply additional numeric standards for sulfate, arsenic and other oxyanions to the WDL or water quality-based effluent standards for the Edith River and its tributaries.
- Construct run-on diversion(s) to achieve, at a minimum, the performance criteria as follows:
  - Divert approximately 70 percent of the surface runoff from the RP 7 catchment area between planning years -2 and -1;
  - Divert approximately 22 percent of the surface runoff from the RP1 catchment area between planning years -2 and -1; and

  - Divert approximately 15 percent of the surface runoff from the RP1 catchment area between planning year -2 through post-closure.

- By planning year -2 commission the following facilities:
  - New WTP with a minimum ARD/ML treatment capacity of approximate 1000 m3/hour;
  - LLDPE-lined (or equivalent) equalization pond (storage capacity ~ 74,000 m^3) for the mixing and temporary storage of ARD/ML from various on-site sources prior to treatment in the New WTP and for the storage of ARD/ML in case of system upset (i.e. ARD/ML flow surge due to extreme storm events or shutdown of the New WTP);
  - LLDPE-lined (or equivalent) sludge disposal cell (Mine-life storage capacity ~ 75,000 m^3) for the disposal of water treatment sludge produced by the New WTP.

Tetra Tech has prepared recommendations for addressing water treatment data gaps at Mt. Todd that should be filled as part of the full feasibility study. These are as follows:

- Mine-life ARD/ML water quality and quantity and thus the mine-life ARD/ML treatment and sludge disposal design and capacity requirements require additional study. Therefore, Tetra Tech recommends that a process water treatment and sludge management study be considered prior to the feasibility study to determine the following:
  - New WTP requirements, system capacity, and optimal location;
  - Post-neutralization/clarification water quality;
  - Optimal reagents;
  - Reagent consumption;
  - Sludge volumes, type, density, consolidation (settleability), handling, and disposal location and facility design;
  - Optimization of New WTP operations necessary to accommodate declining flow volumes and potentially increasing acidity and TDS over the mine-life;
  - The size and precise volume and design necessary to contain ARD/ML prior to treatment; and
  - Regulatory classification of sludge (i.e., solid or hazardous waste).
- The optimal design to convey process water to treatment and sludge to a disposal facility requires additional study. Therefore, as part of the feasibility study the process water and sludge conveyance system requirements necessary for continuous treatment of ARD/ML year-round should be determined to define site-wide pipeline and pumping system requirements and costs and risks minimizations.
- The results of the studies identified in Item 1 and 2 immediately above should be coupled with existing and future water treatment needs, costs, constraints, and benefits to determine the optimal New WTP capacity, design and location.
- Inventory all existing water management facilities (e.g. ponds, pumps, pipelines, WTP inflow pipes, lime silo, utility installation, offices) to determine overall system arrangement, facility capacity, operation and maintenance status, remaining functional life. Determine the cost and benefits and risks associated with the integration of these facilities into the proposed water management and treatment system.

The estimated cost for feasibility-level water treatment studies is between $500,000 and $1,000,000 and is anticipated to take approximately one year.

Water Quality

Water quality data collection from monitoring wells and surface water sampling sites should continue but an expanded analyte suite including major ions and regulated constituents (e.g., trigger values, recreational, drinking water guidelines) should be implemented. This information can be used to further assess the extent of surface water and groundwater contamination and form a basis for future site performance monitoring.

Additional Closure Recommendations

The following information is needed to progress closure planning to the full feasibility level. The recommended work should be performed strategically so that decisions about closure can be made sequentially and at the appropriate phase of the project. The work items that are recommended for completion as part of the feasibility study are as follows:

- Waste and cover material hydraulic properties characterization and analysis;
- Tailings trafficability testing;
- Improvement of the watershed hydrologic data collection system to enable an update of precipitation-yield characteristics of the site;
- Site-wide soils, closure cover, and reclamation material inventory and characterization to identify material sources, properties and balance; and
- Erosion and sediment control analysis.

Waste and Cover Material Hydraulic Properties Characterization and Analysis

The hydraulic properties of waste rock, tailings and potential cover materials require additional characterization as part of the feasibility study. These results should be used to improve:

- Waste facility and site-wide water balance prediction; and
- Evaluation of closure cover design alternative and performance.

Additional samples of waste rock, tailings, and potential cover materials should be collected and analyzed to determine particle size distribution. These particle size distribution data should be compared with available computational databases (e.g. Soilvision) to estimate variably-saturated hydraulic properties (soil water characteristic curves - SWCC, saturated and unsaturated permeability). The SWCC describes the water content of a material as a function of soil suction, or negative pore-water pressure. The particle size analyses and database query results should be used to select a wide range of samples for further empirical characterization of their saturated and unsaturated hydraulic properties.

Tetra Tech recommends that saturated hydraulic conductivity and SWCC of waste rock, tailings and potential sources of soil cover materials be tested.

Samples should be collected as follows:

1. Waste Rock - Fifteen to twenty five waste rock samples, each with a mass of 50-kg, should be selected to represent the majority of the rock mass lithology anticipated to be deposited in WRDs. Samples should be collected from shallow trenches excavated in the existing waste rock facilities.
2. Tailings - Ten paired tailings material cores should be collected along a transect from the deposition zone to the far side of the impoundment or supernatant pond, as practicable.

The cores should be collected using core barrels with clear plastic liners so that stratigraphy can be readily assessed. Cores should be collected to a minimum depth of 3 m. One of the paired cores should be used to visually assess stratigraphy. Areas of distinct sandy characteristic should be identified and evaluated for vertical continuity, with the goal of determining if there are large (e.g., greater than 0.5-m in depth) intervals composed solely of sandy material. Material from intervals of interest will be sampled and submitted to a laboratory for analysis (discussed below).

The second paired core will be sealed to prevent atmospheric oxygen from entering and archived for possible future chemical analysis, depending on whether the particle size analysis indicates a significant possibility that ARD generation could be an issue.

3. Cover Material - Fifteen to twenty five samples of potential cover material sources, each with a mass of 50-kg, should be selected to represent the range of possible cover materials. Samples should be collected from shallow trenches in areas that are representative of the majority of cover material by mass.

Particle size distributions should be determined using the sieve and hydrometer method, in accordance with American Society for Testing and Materials (ASTM) D 422. Material classification should be conducted according to ASTM D 2487. Results will include percentages of cobbles, sand, silt and clay, and the material classification. Saturated hydraulic conductivity tests are most often completed using a triaxial permeameter. A falling head permeameter is more appropriate for coarse textured materials or for the determination of the saturated hydraulic conductivity of cover material following placement. SWCC test are most often completed using a conventional or modified pressure plate apparatus.

Results of the field characterization should be incorporated into hydrologic models (e.g. GOLDSIM, VADOSE/W, SEEP/W, SOILCOVER, H-SAT, etc.) used to simulate the long-term water balance of tailings, waste rock facilities including the amount of meteoric water that infiltrates through closure covers. Detailed deterministic models of waste facility and cover designs alternatives should be developed using probabilistic analysis of precipitation that represent the wet, average and dry year conditions.

The estimated cost to assess the hydraulic properties of waste rock, tailings and potential cover materials for data compilation is between $150,000 to 300,000.

Tailings Trafficability Testing

The minimum cover that will be needed to bridge the thixotropic tailings located on the impounded surface of the TSF and the trafficability and stability of saturated and dewatered slimes requires study and should be investigated to adequately define capping techniques and the quantity of cover needed to successfully reclaim the TSF.

The estimated cost to study the tailings trafficability is $30,000 to 50,000.

Design Storm Events and Watershed Characterization

The design of operational and closure storm water management systems depend in part on the accurate assessment of watershed characteristics, design storm magnitude, and the rainfall-runoff relationships of the catchment basins contributing flow to Mt. Todd. This information is critical for the design of operational and closure storm water management systems and the understanding of WRD, TSF1, TSF2 and pit recharge and infilling characteristics and the potential to generate ARD/ML.

Tetra Tech recommends that additional precipitation, streamflow, and watershed data be collected to improve the understanding of the magnitude of storm events and the precipitation-watershed yield relationship of the catchment basins contributing flow to Mt. Todd.

- Precipitation Data – To improve the correlation between onsite and offsite precipitation data, install additional meteorological gages on-site that collect, at a minimum, precipitation data on an hourly basis.
- Streamflow Gage Data – To check the validity of design storm events calculated for the PFS, evaluate existing streamflow gages onsite and select new streamflow gage locations to improve the current data set. Collect all streamflow data from now until closure to provide additional information to improve the design and engineering for closure storm water systems.
- Hydrologic Soil Type, Native Geology and Other Land Use Information - Develop site-specific curve numbers (runoff coefficients) for the catchment basins contributing flow to Mt. Todd. This includes the development of runoff coefficients for waste rock, tailings, pit walls and closure covers.
- Visual Observation of Existing Structures during Large Storm Events – Conduct visual assessment of existing drainage structure performance during large rainfall events to allow greater understanding of the adequacy of Mt. Todd’s current storm water system design. Near failure scenarios observed at existing drainage structures should correlate roughly to the structure’s design storm event as defined by on-site meteorological stations. Events that are readily controlled allow for the empirical quantification of lower magnitude storms.
- Initiate gauging of runoff/seepage rates from the WRD, HLP, and TSF1 (and TSF2). Develop a relationship of runoff/seepage from each facility with precipitation.

The estimated cost for these studies is between $150,000 and 300,000.

Reclamation Material Inventory and Characterization

Tetra Tech recommends that site-wide inventories be conducted to identify reclamation materials. We recommend inventories of the following materials:

- Non-PAG waste rock and other waste materials on site;
- Clay and low-permeability soils;
- Undisturbed or slightly disturbed soils, stockpiled soils, and regolith;
- Durable rock rip rap and gravels;
- Acid-resistant drain rock; and
- Organic wastes and amendments, etc.

These inventories should be followed by field-tests to determine the materials suitability for the anticipated uses. The potential sources of closure materials a Mt. Todd include, but are not limited to:

- Production of waste covers, riprap, drain and low-permeability clay materials excavated from the pit during mining;
- Production of waste covers, riprap, drain and low-permeability clay materials excavated from the borrow areas;

- Production of organic soil amendments developed by composting organic waste such as feedlot manure, crop stubble, biosolids, wood waste from logging operations, etc.;
- Uncontaminated fill material in materials storage yards, roads, and ancillary facilities;
- Uncontaminated material excavated for creation of the WRD, RP1 and TSF diversions; and
- Soil salvage from the footprint of TSF2 (and the expansion of the WRD and Batman Pit).

Inventories should define the location, volume, properties, uniformity, retrievability, and where necessary, acid-resistance of all potential sources of reclamation materials on or immediately adjacent to the site. Due to the significant cost associated with the excavation, processing (if necessary), transportation and distribution of these reclamation materials, Vista should evaluate approximate haul distance and road grades between each potential closure material source and major closure areas. This process will eliminate some potential sources from further consideration.

When the properties, volume and viability of closure material sources are determined based on site inventories, material balance and costs should be developed and the results be integrated into the closure planning process. The suitability of many of the existing on-site sources of durable rock riprap and gravels, acid-resistant drain rock, low-permeability clays, and other material have already been evaluated by Vista and others. However, the size of these inventories will likely need to be expanded to address the volumes of materials needed for closure.

Standard test references should be used to guide the analysis to assess the suitability of potential sources of durable rock riprap and gravels, acid-resistant drain rock, low-permeability clays, and other materials (e.g., ASTM). Based on an initial assessment of materials contained in each potential cover source, representative material samples should be collected and the following material properties should be determined as appropriate for the intend use of the material.

Physical Parameters

- Particle size distribution (dry sieve and hydrometer for < 2mm fraction);
- Atterberg limits;
- Specific gravity;
- Compaction curve (i.e. Proctor curve);
- Saturated hydraulic conductivity;
- Consolidation - saturated hydraulic conductivity tests; and,
- Soil water characteristic curve (moisture release curves) tests.

Chemical Parameters

- pH (saturated paste and KCl);
- Electrical Conductivity (saturated paste extract);
- Bulk Density;
- Organic Carbon;
- Sodium absorption ratio;
- Cation (Anion) Exchange Capacity;

- Total Nitrogen;
- Nitrate-Nitrogen;
- Available Phosphorus;
- Soluble cations (K, Ca, Mg, Na);
- Exchangeable Bases (K, Ca, Mg, Na Fe, Mn, and Ti) and Aluminum; and
- Acid Base Accounting (additional analysis may be necessary if NNP < + 20 Tons $CaCO_3$ equivalent/1000 tonne material or a neutralization potential ratio (NPR < 2).

Inventories and chemical and physical characterization can be completed relatively quickly (i.e., ~6 months) at an estimated cost of $50,000 to $60,000.

Waste and Cover Material Erosion and Sedimentation Analysis

The erosion from tailings, waste rock, ancillary facility and closure covers should be evaluated to:

- Predict soil loss from facilities during operations and following closure;
- Develop and evaluate erosion and sediment control options; and,
- Predict the rate and magnitude of sediment loads to operational and closure storm water drainage systems (ponds, channels, sumps, etc.).

Vegetation monitoring data should be collected for the existing (and future) reclamation test plots. These data, and data from the characterization of waste and cover hydraulic properties should be used as inputs to empirical or process-based erosion and sedimentation prediction models (RUSLE, Water Erosion Prediction Project – WEPP, Erodibility Index Method, SEDCAD, and others) for the evaluation of facility drainage designs, sediment management plans and erosions and sediment control alternatives.

The estimated cost for these studies is between $50,000 and 100,000.

21.2 Planned Work Commitments

Vista, based on the above recommendations and their own work commitments, have developed a proposed work program to be completed during the next 12 to 18 months in order to advance the Batman deposit through completion of a feasibility study. This program is detailed in TABLE 21-1. As with these types of programs, some of the specific work items are dependent on the results of earlier items, and it is expected that some adjustments to the program will be made based on initial results. It is Tetra Tech's opinion that the proposed program is designed to address the most significant issues detailed in the recommendations above, is logical in its approach and well thought out, and is representative of the level of financial commitment necessary to complete the proposed work.

TABLE 21-1: PROPOSED WORK PLAN AND BUDGET
VISTA GOLD CORP. – MT TODD GOLD PROJECT
January 2011

Description	Estimated Cost (Millions of $)
Batman Deposit Development Drilling	2.0 to 3.5
Exploration on Mineral Leases	0.5 to 1.0
Exploration on Exploration Leases	1.0 to 2.0
Permitting and Baseline Studies	2.5 to 3.8
Metallurgical Testing and Feasibility Study	4.0 to 6.0
TOTAL	**9.0 to 15.5**

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23.0 DATE AND SIGNATURE PAGE

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CERTIFICATE of AUTHOR

I, John W. Rozelle, P.G., do hereby certify that:

1. I am currently employed by Tetra Tech MM, Inc. at:

   350 Indiana Street
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   Golden, Colorado 80401

2. I graduated with a degree in Geology (BA) from the State University of New York at Plattsburg, New York, in 1976. In addition, I graduated from the Colorado School of Mines, Golden, Colorado, with a graduate degree in Geochemistry (M.Sc.) in 1978.
3. I am a Member of the American Institute of Professional Geologists (CPG-07216), a registered Geologist in the State of Wyoming (PG-337), a member of Society for Mining, Metallurgy, and Exploration, Inc. (SME) and the Society of Economic Geologists.
4. I have worked as a geologist for a total of twenty-nine years since my graduation from university; as a graduate student, as an employee of a major mining company, and as a consultant for more than 25 years.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.
6. I am responsible for the preparation of the technical report titled "10.65 Mtpy *PRELIMINARY FEASIBILITY STUDY – NI 43-101 TECHNCIAL REPORT - MT TODD GOLD PROJECT – NORTHERN TERRITORY, AUSTRALIA.*" and dated 28th January 2011 (the "Preliminary Feasibility Report"). I visited the subject property on June 20, 2005, June 12-14, 2008, and November 10-12, 2008.
7. I have either supervised the data collection, preparation, and analysis and/or personally completed an independent review and analysis of the data and written information contained in this Technical Report.
8. 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 and Preliminary Economic Assessment Reports.

9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
10. I do not hold, nor do I expect to receive, any securities or any other interest in any corporate entity, private or public, with interests in the properties that are the subject of this report or in the properties themselves, nor do I have any business relationship with any such entity apart from a professional consulting relationship with the issuer, nor to the best of my knowledge do I have any interest in any securities of any corporate entity with property within a 2-km distance of any of the subject properties.
11. I have read National Instrument 43-101 and Form 43-101F, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Preliminary Feasibility Report with any stock exchanges or other regulatory authority and any publication by them, including electronic publication in the public company files on the websites accessible by the public, of the Preliminary Feasibility Report.

Dated this 28th Day of January 2011.

John W. Rozelle, P.G.

Digitally signed by John W. Rozelle, P.G.
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Signature of Qualified Person

John W. Rozelle .

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CERTIFICATE of AUTHOR

I, Stephen Krajewski., do hereby certify that:

1. I am currently employed by Tetra Tech MM, Inc. at:

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2. I graduated with a degree in Geography (BS) in 1968, a degree in geology (MS) in 1971, and a degree in Earth Science Education (Ed.D) in 1977, all from The Pennsylvania State University.
3. I am a Member of the American Institute of Professional Geologists (certification CPG-04739 since June, 1980), a member in good standing of the Society for Mining, Metallurgy, and Exploration, Inc. (SME), the American Institute of Professional Geologists, the American Association of Petroleum Geologists, and the Rocky Mountain Association of Geologists.
4. I have worked as a geologist for a total of forty-six years since my graduation from university; as a graduate student, as an employee of mining, environmental, oil & gas, and consulting companies; and for government agencies and academic institutions. This work has been completed throughout the United States and internationally.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.
6. I am responsible for the preparation of portions of SECTION 17.0 of the technical report titled "10.65 Mtpy *PRELIMINARY FEASIBILITY STUDY – NI 43-101 TECHCNICAL REPORT - MT TODD GOLD PROJECT – NORTHERN TERRITORY, AUSTRALIA.*" and dated 28th January 2011 (the "Preliminary Feasibility Report"). I have not visited the property.
7. I have either supervised the data collection, preparation, and analysis and/or personally completed an independent review and analysis of the data and written information contained in portions of SECTION 17 of this Technical Report.
8. I have had prior involvement with Vista Gold Corp. on the property that is the subject of this Technical Report. My involvement has been on creation and updating of the three-dimensional geologic models of the Mt. Todd and Quigleys deposits.

9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
10. I do not hold, nor do I expect to receive, any securities or any other interest in any corporate entity, private or public, with interests in the properties that are the subject of this report or in the properties themselves, nor do I have any business relationship with any such entity apart from a professional consulting relationship with the issuer, nor to the best of my knowledge do I have any interest in any securities of any corporate entity with property within a 2-km distance of any of the subject properties.
11. I have read National Instrument 43-101 and Form 43-101F, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Preliminary Feasibility Report with any stock exchanges or other regulatory authority and any publication by them, including electronic publication in the public company files on the websites accessible by the public, of the Preliminary Feasibility Report.

Dated this 28th Day of January 2011.

"**Stephen A. Krajewski**" .
Signature of Qualified Person

Stephen A., Krajewski .

Print name of Qualified Person

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CERTIFICATE OF AUTHOR

I, Edwin C. Lips, do hereby certify that:

1. I am a Sr. Mining Engineer of:
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   USA

2. I graduated from Montana Tech, Butte, Montana with a degree in Mining Engineering (BS) in 1982.

3. I am a registered Professional Engineer (Mining) in the State of Arizona (47670), and a member of the Society of Mining, Metallurgy, and Exploration (SME).

4. I have practiced my profession as a mining engineer continuously since graduation for a 6. 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.

5. I am responsible for and prepared, or contributed to, sections 1.0, 18.0, and 19.0, and 21.0 of the report titled "10.65 Mtpy *PRELIMINARY FEASIBILITY STUDY – NI 43-101 TECHNCIAL REPORT - MT TODD GOLD PROJECT – NORTHERN TERRITORY, AUSTRALIA.*" and dated 28th January 2011 (the "Preliminary Feasibility Report").

6. I have not had prior involvement with the property that is the subject of the Technical Report.

7. To the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

8. I am independent of the issuer applying all of the tests of Section 1.4 of National Instrument 43-101.

9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and that form.

10. I consent to the filing of the Preliminary Feasibility Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Preliminary Feasibility Report.

Dated this 28th Day of January 2011.

“Edwin C Lips”
Signature of Qualified Person

Edwin C. Lips
Print name of Qualified Person

D. Erik Spiller, QP

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CERTIFICATE of AUTHOR

I, D. Erik Spiller, do hereby certify that:

1. I am currently employed by Tetra Tech MM, Inc. at:

   350 Indiana Street
   Suite 500
   Golden, Colorado 80401

2. I graduated with a B.S. diploma in Metallurgical Engineering from the Colorado School of Mines, Golden Colorado, in 1970.
3. I am a Qualified Professional (QP) member of the Mining and Metallurgical Society of America (MMSA). In addition, I am a Registered (QP) member of Society for Mining, Metallurgy, and Exploration, Inc. (SME).
4. I have worked as a metallurgical engineer in the mineral resource industry for more than 40 years. During this career I held responsible positions in process research, process development, engineering, and senior management. In addition, I have served as an Adjunct instructor (20 years) and as an appointed Research Professor (4 years) in the Metallurgical and Materials Engineering Department at the Colorado School of Mines where I lecture in mineral beneficiation and direct graduate students conducting metallurgical research in my area of expertise.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.
6. I am responsible for the preparation of portions of the technical report titled "10.65 Mtpy *PRELIMINARY FEASIBILITY STUDY – NI 43-101 TECHNCIAL REPORT - MT TODD GOLD PROJECT – NORTHERN TERRITORY, AUSTRALIA.*" and dated 28th January 2011 (the "Preliminary Feasibility Report"). I have not visited the property.
7. I have either supervised the data collection, preparation, and analysis and/or personally completed an independent review and analysis of the data and written information contained in portions of SECTIONS 16, 19, and 21 of this Technical Report.
8. I have not had prior involvement with Vista Gold Corp. or previous owners on the property that is the subject of this Technical Report.

9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
10. I do not hold, nor do I expect to receive, any securities or any other interest in any corporate entity, private or public, with interests in the properties that are the subject of this report or in the properties themselves, nor do I have any business relationship with any such entity apart from a professional consulting relationship with the issuer, nor to the best of my knowledge do I have any interest in any securities of any corporate entity with property within a 2-km distance of any of the subject properties.
11. I have read National Instrument 43-101 and Form 43-101F, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Preliminary Feasibility Report with any stock exchanges or other regulatory authority and any publication by them, including electronic publication in the public company files on the websites accessible by the public, of the Preliminary Feasibility Report.

Dated this 28th Day of January 2011

"D. Erik Spiller"
Signature of Qualified Person

D. Erik Spiller .

Print name of Qualified Person

MINE DEVELOPMENT ASSOCIATES
MINE ENGINEERING SERVICES

CERTIFICATE OF AUTHOR
Thomas Dyer, P. E.

I, Thomas Dyer, P. E., do hereby certify that I am currently employed as Senior Engineer by Mine Development Associates, Inc., 210 South Rock Blvd., Reno, Nevada 89502 and:

1. 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 14 years since graduation.

2. I am registered as a Professional Engineer – Mining in the State of Nevada (# 15729). I am also a Registered Member of SME (# 4029995RM) in good standing.

3. I have read the definition of "qualified person" set out in National Instrument 43-101 ("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. I am independent of Vista Gold Corp. and all their subsidiaries as defined in Section 1.4 of NI 43-101 and in Section 3.5 of the Companion Policy to NI 43-101.

4. I am responsible for the preparation of the Pit Design and Reserves sections 18.2 through 18.3 and sections 19.1 through 19.3 of this report titled "10.65 Mtpy Preliminary Feasibility Study NI 43-101 Technical Report Mt Todd Gold Project Northern Territory, Australia". I have not visited the site.

5. 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).

6. To the best of my knowledge, information and belief, those sections for which I am responsible contain all the scientific and technical information that is required to be disclosed to make this technical report not misleading.7. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

8. I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 28th day of January 2011.

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CERTIFICATE of AUTHOR

I, Deepak Malhotra do hereby certify that:

1. I am currently employed as President by:

   Resource Development Inc. (RDi)
   11475 W. I-70 Frontage Road North
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2. I graduated with an M.S. in Metallurgical Engineering and a PhD in Mineral Economics from the Colorado School of Mines in 1974 and 1978, respectively.
3. I am a Member of the Society for Mining, Metallurgy, and Exploration, Inc. (SME) and the Canadian Institute of Mining (CIM).
4. 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.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.
6. I am responsible for the preparation of SECTION 16.0 of the report titled "10.65 Mtpy *PRELIMINARY FEASIBILITY STUDY – NI 43-101 TECHNCIAL REPORT - MT TODD GOLD PROJECT – NORTHERN TERRITORY, AUSTRALIA.*" and dated 28th January 2011 (the "the "Preliminary Feasibility Report").
7. I have either supervised the data collection, preparation, and analysis and/or personally completed an independent review and analysis of the data and written information contained in this Technical Report.
8. 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 and Preliminary Economic Assessment Reports.
9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
10. I do not hold, nor do I expect to receive, any securities or any other interest in any corporate entity, private or public, with interests in the properties that are the subject of this report or in the properties themselves, nor do I have any business relationship with any such entity apart from a professional consulting relationship with the issuer, nor to

the best of my knowledge do I have any interest in any securities of any corporate entity with property within a 2-km distance of any of the subject properties.

11. I have read National Instrument 43-101 and Form 43-101F, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Preliminary Feasibility Report with any stock exchanges or other regulatory authority and any publication by them, including electronic publication in the public company files on the websites accessible by the public, of the Preliminary Feasibility Report.

Dated this 28th Day of January 2011.

"Deepak Malhotra"
Signature of Qualified Person

"Deepak Malhotra" .

Print name of Qualified Person

24.0 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

Tetra Tech is unaware of any additional information, technical reports, and/or documents that would result in any changes to the information presented in this PFS Technical Report.

25.0 ILLUSTRATIONS

All of the illustrations used in the preparation of this report appear in each of their respective sections.

APPENDIX A
RESOURCE ESTIMATION OF THE BATMAN AND QUIGLEYS DESPOSITS BY TETRA TECH, 2009 - 2010

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APPENDIX C
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APPENDIX E
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APPENDIX G
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APPENDIX I
MT. TODD PROJECT AREA
WATER MANAGEMENT UPDATE
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APPENDIX J
MT. TODD PROJECT – PROPOSED RECLAMATION AND CLOSURE PLAN BY TETRA TECH, 2011

APPENDIX K
TAILINGS STORAGE FACILITY TRADEOFF STUDY
BY TETRA TECH, 2011

APPENDIX L
TAILINGS DESIGN STUDY
BY TETRA TECH, 2010

APPENDIX M
NEW TAILINGS STORAGE FACILITY (TSF2)
BY TETRA TECH, 2011