MT. TODD GOLD PROJECT PREFEASIBILITY STUDY

Northern Territory, Australia

Prepared for
VISTA GOLD CORP.

October 1, 2010

VISTA GOLD CORP.

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

Northern Territory, Australia

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1.0 SUMMARY

Tetra Tech, Inc. ("Tt") was commissioned by Vista Gold Corp. ("Vista") in September 2009 to prepare a NI 43-101 compliant Preliminary Feasibility Study (PFS) for the Mt Todd Gold Project (the "Project") located in Northern Territory ("NT"), Australia. On March 1, 2006, Vista purchased the Mt Todd property, and the acquisition was completed on June 16, 2006, when the mineral leases transferred to Vista and funds were released from escrow. 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.

1.1 Location

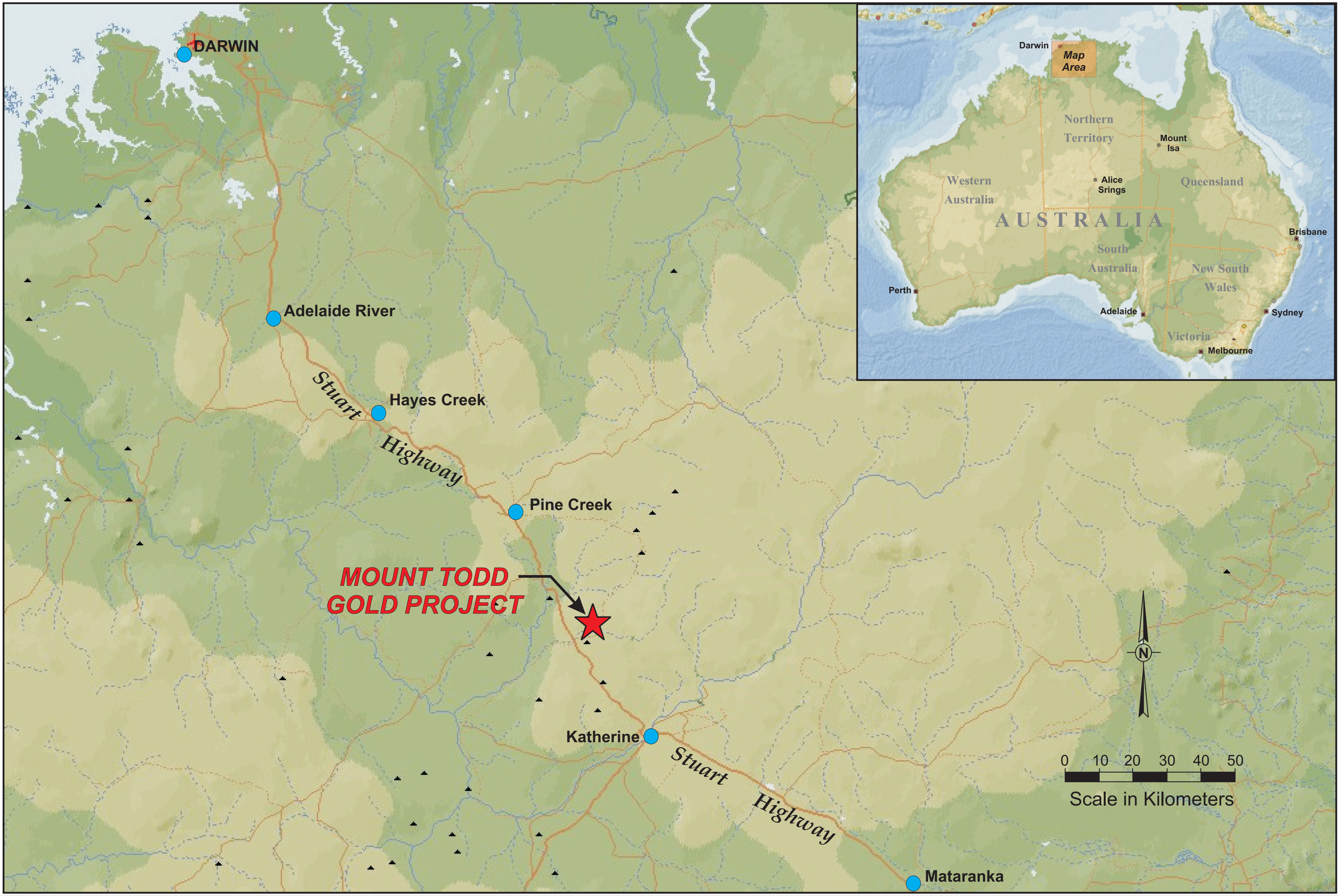
The Mt Todd Project is located 56 kilometers (km) by road northwest of Katherine, and approximately 250 km southeast of Darwin in NT, Australia (FIGURE 1-1). Access to the property is via high quality, two-lane paved roads from the Stuart Highway, the main arterial within the territory.

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 processing 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. The improper processing techniques are also currently being reviewed and revised. A brief write up of this work is presented in Section 16.0 of this report. It is Tt's opinion that this information is very important when examining the Mt Todd Gold Project as envisaged by Vista.



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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 US$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 EL 25670 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 Territory 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 Territory. 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 Territory 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.

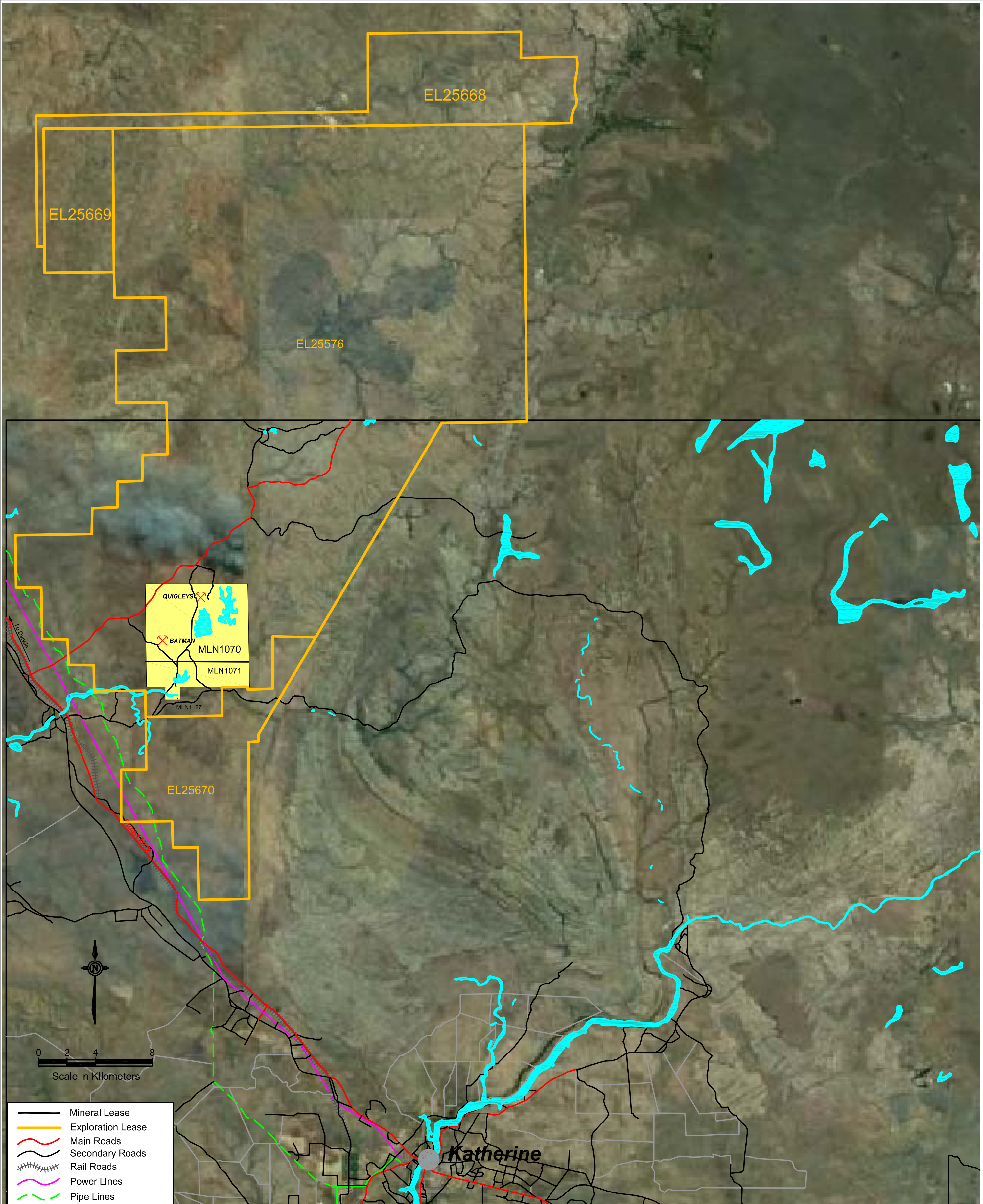
1.4 Geology

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, posses limited drillhole and other geologic information, but do not have investigations by Vista. Tt 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.

Tt 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 Tt'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 Tt 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 only using data from this zone. The second pass is for the material outside of the main core complex only using 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 Base Case cutoff for the resource reporting is 0.4 g Au/t and is bolded in the table. This cutoff value was determined according to a three-year average gold price of US$950, 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			
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 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 Tt 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 drillhole 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 "stepout" drilling is still needed.

Another feature that came to light from the 2007 and 2008 exploration-drilling program is the potential existence of a 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 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. Tt proposes that the following items be considered when preparing the work plan:

1) 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;
2) 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;
3) 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
4) 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 the resource calculation.

1.6 Base Case Mine Plan and Mineral Reserves

Potentially mineable pit shapes were evaluated using a Lerchs-Grossman ("LG") analyses 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 1-5 (all prices and costs are reported in first quarter 2010 US dollars).

TABLE 1-5: BASE CASE PARAMETERS FOR LERCHS-GROSSMAN ANALYSES
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Overall Pit Slopes	33° from pit centered azimuth ranging 10° – 150° 55° from pit centered azimuth ranging 150° – 10°
Gold Price (1)	US$950 per oz Au
Gold Recovery	82 percent
Mining Cost	US$1.75 per tonne mined
Processing Cost	US$7.80 per tonne processed
General and Administrative Cost	US$0.84 per tonne processed

The Base 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. For the Mt. Todd Base case, the ultimate pit was further constrained by assuming that the total tailings facility capacity is approximately 60 million tonnes. This constraint was provided by Vista and is based on land, environmental, and capital constraints.

Using the Base 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. Ore mining is planned to use primarily 12 cubic meter Front-end Loaders and 140 tonne haul trucks. using the equipment as ore is encountered.

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. The design includes smoothed pit walls, haulage ramps, benches, and pit access. After the ultimate pit was designed, pit phases were created to improve the project NPV by mining higher-value material in the initial years. The $500 per ounce Au pit was chosen for a guide to design phase 1, and the $605 per ounce Au pit was used to guide the design of phase 2. Phase 3 was defined by the remaining volume to achieve the ultimate pit.

TABLE 1-6 summarizes mineable reserves resulting from the base case ultimate pit.

TABLE 1-6: CLASSIFICATION OF BASE CASE MINEABLE RESERVES
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

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	24,458	1.09	854			
Probable	35,592	1.02	1,173			
Proven + Probable	60,050	1.05	2,027	142,524	202,574	2.37

** Elevated cutoff grades were used to constrain the total reserve tonnes to remaining tailings capacity while maximizing return. In most areas, a cutoff of 0.55 g Au/t was used. Select benches in the first two phases of mining used a cutoff of 0.60 g Au/t.*

The Base Case production schedule for this PFS assumes a 6.77 million-tonne-per-year ore production rate, resulting in a 9-year operating life, as shown in TABLE 1-7.

TABLE 1-7: BASE CASE PRODUCTION SCHEDULE
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Year	"Ore" Tonnes (x 1000)	Avg. Grade (g Au/tonne)	Waste Tonnes (x 1000)	Stripping Ratio (W:O)
PP1	538	0.87	5,583	10.38
1	7,107	1.06	27,898	3.93
2	8,880	1.11	25,095	2.83
3	9,921	1.06	23,519	2.37
4	6,770	0.96	29,948	4.42
5	6,789	0.98	17,573	2.59
6	6,770	0.91	8,717	1.29
7	6,770	1.08	3,685	0.54
8	6,327	1.23	504	0.08
9	178	1.22	2	0.01
Total	**60,050**	**1.05**	**142,524**	**2.37**

** Elevated cutoff grades were used to constrain the total reserve tonnes to remaining tailings capacity while maximizing return. In most areas, a cutoff of 0.55 g Au/t was used. Select benches in the first two phases of mining used a cutoff of 0.60 g Au/t.*

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 report 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 Northern Territory, Australia operated by the Vista Gold Corporation. The site electrical power demands are a fixed constant operating load estimated at 34MW 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 10 year operating plant life without annual pricing index. Fuel costs are based on a rate of $5.75 (AUS) per gigajoule. Calculated 10 year project life costs are estimated $0.0775 to 0.0863 (AUS) per kilowatt-hour for the 34MW site demand compared to the commercially purchased electricity rate of $0.1636 per kWh (adjusted for demand) for the same time period.

The GE LM6000PF provides the lowest initial capital cost per kilowatt with a single unit. If there is higher demand for export power the Rolls Royce gas turbine has the most reserve capacity. If continuous power supply is required, the Wartsila 20V34SG reciprocating engines are estimated to provide the lowest overall 10 year project costs.

1.9 Processing and Process Flowsheet

The Mt Todd gold recovery process evolved both historically and through studies commissioned by Vista Gold at RDi. The evolved process uses proven technologies to recover 82% of the contained gold by CIL leaching. For purposes of this prefeasibility, an ore feed grade of 1.08 g/t and an Ausenco adjusted plant feed rate of 844 t/h (nominally 18,500 mtpd or 6.8Mt/y) was used.

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

1.10 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.

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 Mining Management Plan (MMP) for resumption of mining operations. A decision on the appropriate permitting route will be initiated by submission of the MMP or Notice of Intent (NOI) to the Department of Regional Development, Primary Industry, Fisheries and Resources (DRDIPFR).

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).

Water Management

Current and historic evidence indicates that Mt Todd waste rock, ore and tailings that contain sulfides are capable of generating acid and metal laden leachates (ARD/ML). ARD/ML currently occurs in the waste rock dump and associated pond (RP1), the lean ore stockpile and associated pond (RP2), exposed pit walls (RP3), the heap leach facility, 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 which allows controlled discharges from RP1 to the Edith River during high flow events. The impacted water is sufficiently diluted to ensure downstream compliance with established copper criteria which in turn dilutes other regulated constituents to acceptable levels. Improvements to the water management system reduced uncontrolled discharges during the 2006/2007 wet season as compared to the previous reporting periods.

In August 2009, Vista commissioned a water treatment plant (WTP) to treat ARD/ML water at a capacity of 193 m^3/hr. Pilot studies showed that lime treatment removed 98% of the cadmium, 98.8% of aluminum and greater than 99% 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.

Tailings Disposal

A tailings disposal tradeoff study was completed in early 2010 in order to explore several different types of tailings disposal, such as a dry stack facility, heap leach pads, new tailings storage facility (TSF) designs, and several raises to the existing TSF. The 60 million tonne capacity raise to the existing TSF design was selected based on economic tradeoff studies and the relatively low cost per tonne of tailings stored.

The raise to the existing TSF was adapted from the MWH design completed in 2006, with some modifications to accommodate the projected capacity of the facility. The raise 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 to vertical) 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, underdrains, 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.

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. Required 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.

Reclamation and Closure

The major and immediate environmental challenges for Mt Todd are the management of acid rock drainage and metal-laden leachates (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, controlled discharge to the Edith River during major flow events, active water treatment, pumping excess water to the Batman Pit, 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 and design for, and implement effective plans for:

- Year-round collection, containment and treatment 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 (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 metaliferous 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 stormwater 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 6.77 Million tonnes per year (Mtpy) mine plan are as follows:

- Batman Pit;
- Batman Pit Lake (RP3);
- Waste Rock Dump (WRD);
- WRD Pond (RP1) and pumping system;

- Tailings Storage Facility (TSF);
- TSF Pond (RP7);
- Processing Plant and Operations Area;
- Processing Plan Runoff Pond (RP5) and pumping system;
- Heap Leach Pad (HLP);
- HLP Pond and Moat (HLP Ponds) 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 TSF and RP1;
- New WTP;
- Equalization Pond;
- Sludge Disposal Cell;
- TSF Operational (and Closure) Spillway;
- TSF Moat;
- WRD Seepage Collection System;
- Clay Borrow Area; and
- Anaerobic treatment wetlands (or equivalent passive/semi-passive water treatment system).

The PFCP includes descriptions, approximate dimensions and performance criteria for proposed facilities and are provided elsewhere in this report. 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 Waste Discharge License (WDL) and applicable Edith River water quality-based effluent standards and; 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 cost were estimated at a ± 25 percent level of accuracy based on the following:

- 6.77 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 9 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 prefeasibility-level cost estimates for implementing the closure and mine-life water treatment plans are US$64,938,000 and US$31,203,000, respectively.

TABLE 1-8: 6.77 MTPY CLOSURE AND MINE-LIFE WATER TREATMENT COST ESTIMATE SUMMARY
VISTA GOLD CORP. – MT TODD PROJECT
October 2010

Area	Cost (US$)[1]
Tailings Storage Facility	$ 18,050,000
Heap	$ 2,585,000
Processing Plant And Pad Area	$ 8,813,000
Batman Pit	$ 99,000
Waste Rock Dump	$ 16,702,000
WRD Retention Pond	$ 525,000
Low Grade Ore Stockpile	$ 256,000
Mine Roads	$ 5,787,000
Clay Borrow Area[2]	$ 1,243,000
Sludge And Equalization Pond Closure[2]	$ 286,000
Total Direct Closure Cost	**$ 54,344,000**
Mobilization/Demobilization (Assume On-Site Mining Equipment Fleet Used)	$ 0-
Incidentals (Communication, Misc. Supplies, Etc.) = 0.5 % Of Total Direct Cost.	$ 345,000
Haul Road Maintenance During Closure = 1 % Of Total Direct Cost.	$ 689,000
Construction Quality Assurance = $200,000 /Year During 3 Year Closure Period	$ 600,000
Engineering Re-Design = 2 % Of Total Direct Cost.	$ 1,378,000
Contingency = 8 % Of Total Direct Cost.	$ 5,514,000
Total Indirect Cost [3]	**$ 8,526,000**
Annual Site Maintenance and Monitoring For 6 Years Post Closure	$ 2,068,000
Total Closure Costs	**$ 64,938,000**

Water Treatment System Facility/Component	Cost (US$)
Active Water Treatment And Sludge Disposal System Construction[2]	$ 5,799,000
Passive Water Treatment System	$ 8,780,000
Total Direct Water Treatment Construction Cost	$ 14,569,000
Prep-Production Period (Years -2 and -1) Water Treatment O&M, Reagent and Pumping[4]	$ 8,102,000
Production Period (Years 1 through 9) Water Treatment O&M, Reagent and Pumping[4]	$ 4,201,000
Closure Production Period (Years 10 through 12) Water Treatment O&M, Reagent and Pumping[4]	$ 2,733,000
Post-Closure Production Period (Years 13 through 18) Water Treatment O&M, Reagent and Pumping[4]	$ 1,588,000
Total Mine-Life Water Treatment O&M, Reagent and Pumping[3]	$ 16,624,000
Total Mine-Life Water Treatment Costs	**$ 31,203,000**

[1] Cost rounded to nearest $1,000 in current US$.
2Following submittal of the PFS projected cash flow for the project, Tetra Tech determined the size of the Equalization Pond, Sludge Disposal Cell and Clay Borrow Area were erroneous calculated. As such the published cash flow is based on oversized water treatment and closure facilities. The direct costs for the construction and closure of these facilities should decrease by approximately US$2.3 million. Indirect costs (e.g. re-design, quality assurance, contingency, post-closure maintenance) should decrease by approximately US$0.27 million. According to Tetra Tech's stated assumptions and the limitations of our analysis, the size of the Equalization Pond, Sludge Disposal Cell and Clay Borrow Area reported here are correct. However, to maintain consistency with the published cash flow, the costs to construct and close these facilities were not corrected.
[3]Includes indirect costs associated with the construction of Water Treatment System
[4]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.

As part of the larger production sensitivity analysis presented in this report, the 10.6 Mtpy scenario included a comparison of the closure and water treatment plans and costs to the 6.77 Mtpy mine plan. For this sensitivity analysis, the closure and water treatment plans and strategies present in the 6.77 Mtpy were replicated and proportionally scaled according to the quantities (e.g. facility dimension, material/fluid volumes, surface areas, disturbance footprints) estimated for the 10.6 Mtpy mine plan.

As summarized in TABLE 1-9 the scoping-level cost estimates for implementing the closure and mine-life water treatment plans for the 10.6 Mtpy mine plan are US$120,564,000 and US$51,364,000, respectively.

TABLE 1-9: 10.6 MTPY SCOPING-LEVEL CLOSURE AND MINE-LIFE WATER TREATMENT COST ESTIMATE SUMMARY
VISTA GOLD CORP. – MT TODD PROJECT
October 2010

Area	Cost (US$)[1]
Tailings Storage Facility	$ 18,049,000
New Tailings Storage Facility	$ 21,973,000
Heap	$ 2,585,000
Processing Plant And Pad Area	$ 11,419,000
Batman Pit	$ 174,000
Waste Rock Dump	$ 31,785,000
WRD Retention Pond	$ 300,000
Low Grade Ore Stockpile	$ 256,000
Mine Roads	$ 8,680,000
Clay Borrow Area	$ 1,717,000
Sludge And Equalization Pond Closure	$ 1,365,000
Total Direct Closure Cost	**$ 98,303,000**
Mobilization/Demobilization (Assume On-Site Mining Equipment Fleet Used)	$ 0-
Incidentals (Communication, Misc. Supplies, Etc.) = 0.5 % Of Total Direct Cost.	$ 619,000
Haul Road Maintenance During Closure = 1 % Of Total Direct Cost.	$ 1,239,000
Construction Quality Assurance = $400,000 /Year During 3 Year Closure Period	$ 1,200,000
Engineering Re-Design = 2 % Of Total Direct Cost.	$ 2,478,000
Contingency = 8 % Of Total Direct Cost.	$ 9,911,000
Total Indirect Cost [2]	$ 15,447,000
Annual Site Maintenance and Monitoring For 11 Years Post Closure	$ 6,814,000
Total Closure Cost	**$ 120,564,000**
Water Treatment System Facility/Component	**Cost (US$)**
Active Water Treatment And Sludge Disposal System Construction	$ 7,705,000
Passive Water Treatment System	$ 17,872,000
Total Direct Water Treatment Construction Cost	$ 25,577,000
Prep-Production Period (Years -2 and -1) Water Treatment O&M, Reagent and Pumping[3]	$ 8,075,000
Production Period (Years 1 through 13) Water Treatment O&M, Reagent and Pumping[3]	$ 8,415,000
Closure Production Period (Years 14 through 16) Water Treatment O&M, Reagent and Pumping[3]	$ 4,290,000
Post-Closure Production Period (Years 17 through 27) Water Treatment O&M, Reagent and Pumping[3]	$ 5,007,000
Total Mine-Life Water Treatment O&M, Reagent and Pumping[3]	$ 25,787,000
Total Mine-Life Water Treatment Costs	**$ 51,364,000**

[1] Cost rounded to nearest $1,000 in current US$.
[2] Includes indirect costs associated with the construction of Water Treatment System
[3] Includes Plant O& M, Lime, and Water and Sludge Pumping

1.11 Economic Evaluation

The financial results presented in this prefeasibility study have been developed co-operatively between Vista, Tt, and other consultants. The financial results are presented in constant dollars with the mill capital being from the last quarter of 2009 and the mine capital from the second quarter of 2010. A five percent (5%) discount rate has been applied to the financial analysis. Besides the Base Case, sensitivity analysis was completed that varied the Base Case gold price, varied the production rate, and then varied the gold price for the larger production rate scenario. Unless otherwise noted, an US/AUD conversion rate of 0.85 was used.

1.11.1 Base Case

The Base Case project entails mining 60,050,000 of ore tonnes over a 9 year period. The scenario requires that 6.77Mtpy ore be mined and processed assuming $950/toz Au, and an exchange rate of 0.85 US/AUD dollars, and metallurgical recoveries of 82%.

1.11.2 Capital Costs

Estimated capital expenditures for the life-of-mine Base Case are estimated to be US$508.945 million; this being a combination of US$475.964 million start-up capital and US$32.981 million sustaining capital. Mine capital estimates are current as of mid-2010. Process capital estimates are current as of late 2009. TABLE 1-10 provides a summary of the project capital over the life of the proposed operation.

1.11.3 Mine Operating Costs

Mine operating costs are higher in the earlier years due to the higher strip ratios necessary for ore production. A summary of the mine operating costs are presented in TABLE 1-11 for the 6.77 Mtpy base case.

TABLE 1-11: MINE OPERATING COST SUMMARY (000)
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Year	1	2	3	4	5	6	7	8	9
Ore Tonnes Mined	6,789	6,770	6,770	6,770	6,770	6,770	6,770	6,770	5,852
Total Mining Costs	$45,817	$46,149	$45,657	$49,084	$40,046	$28,884	$23,189	$17,699	$6,398
Mine Operating Costs ($/tonne ore)	$6.75	$6.82	$6.74	$7.25	$5.90	$4.27	$3.43	$2.61	$1.09

1.11.4 Process Operating Costs

The Base Case process operating cost range from $7.35 to $7.38/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 1-12.

TABLE 1-10: SUMMARY OF TOTAL PROJECT CAPITAL COSTS(000) VISTA GOLD CORP. – MT TODD GOLD PROJECT October 2010	
Cost Center	**US$ (000)**
Summary of Initial Capital Costs	
Mine	
Primary Open Pit Mine Equipment	62,754
Lime Operation Mine Equipment	5,617
Ancillary General Surface Mobil Equipment	13,588
Mine Office, Shop, and Warehouse	768
Mining Development Supply and Labor Op Costs	11,323
Mine Sub-total	**$94,050**
Process	
Process Plant	155,763
Onsite Infrastructure	12,819
Offsite Infrastructure	15,147
Mobile Equipment, Spares, First-Fills	8,393
Power Generating Station	31,519
Process Sub-total	**$223,641**
Tailings	
Water Treatment Facility & Tailings Operation Costs	20,226
Tailings Storage Facility	11,910
Tailings Sub-total	**$32,136**
Indirects	
Temporary Construction Facilities	4,208
Accommodations	3,585
EPCM (Contractor's Cost)	30,843
Commissioning	3,503
Indirects Sub-total	**$42,139**
Other Capital	
Water Treatment Facility	14,659
Site Demolition	850
EPCM Contractor Fee	3,994
Permitting	500
Recruiting and Training	1,700
Lime Kiln/processing	6,158
Other Capital Sub-total	**$27,861**
Contingency	56,137
Salvage Value	-59,567
Owner/Reclamation (no contingency)	57,735
Summary of Sustaining Capital Costs	
General Surface Mine Equipment	5,086
Mine Capital Contingency	763
Water Treatment Facility (WTF)	8,780
Pre-production WTF & Tailings Operation Costs	11,269
Tailings Storage Facility	3,544
Process Plant Contingency	2,222
Other Capital Contingency	1,317
Sustaining Capital Starting in Year 2	**$32,981**

TABLE 1-12: ADR PROCESS OPERATING COST SUMMARY (000)
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Year	1	2	3	4	5	6	7	8	9
Ore Tonnes Mined	6,789	6,770	6,770	6,770	6,770	6,770	6,770	6,770	5,852
Total Processing Costs	$49,969	$49,789	$49,829	$49,789	$50,088	$49,893	$49,848	$49,848	$43,194
Ore Processing Costs ($/tonne ore)	$7.36	$7.35	$7.36	$7.35	$7.38	$7.37	$7.36	$7.36	$7.38

Included in the process operating costs is a general and administrative (G & A) cost of US$0.569 per tonne ore processed. Gold dore' refining, transport and treatment charges are US$4.50/toz Au, but are included separately in the cash flow analyses.

1.11.5 Cash Flow Analyses

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

Cash flow analyses at US$950/toz Au and a US/AUD exchange rate of 0.85 results in a project pretax NPV of $210.175 million and a pre-tax Internal Rate of Return of 14.9% and a post-tax rate of return of 9.8% evaluated at a 5 percent discount rate. TABLE 1-13 is the cash flow associated with the base case scenario.

Since the development and publishing of the cash flow statement in the August 18, 2010 press release, Tt has become aware of two financial items that could impact the results presented. They are:

- Approximately US$1.5 million for a mine shop was not accounted for in the cash flow presented.
- Approximately US$2.56 million of environmental related costs were over-estimated due to a calculation error for pond sizing.

Clearly, the net result is that the cash flow has slightly over-estimated the capital costs for the project by approximately US$1.06 million. Given that the PFS is a plus/minus 25 percent estimate and this difference represents less than a one percent change total capital costs, it is Tt's opinion that these omissions are not material to the results presented.

1.11.6 Sensitivity Gold Price Sensitivities

Gold Price sensitivity analyses were performed on the Base Case reflecting gold prices from US$850 to US$1,150 in increments of US$50. A graph showing the results of these sensitivities is shown below.

TABLE 1-13: Mt. Todd 6.77 Mtpy Base Case Cash Flow Analysis (US$950/toz Au Price)

PRETAX:	
IRR	14.9%
NPV0 (000'S)	$472,840
NPV5 (000'S)	$210,175
AVG ANNUAL CF (000's) PRODUCTION YEARS	$75,099
AVG ANNUAL CF (000's) LIFE OF MINE	$41,305
STRIPPING RATIO (WST:ORE)	2.37

AFTER-TAX:	
IRR	9.8%
NPV0 (000'S)	$252,782
NPV5 (000'S)	$71,208
AVG ANNUAL CF (000's) PRODUCTION YEARS	$55,094
AVG ANNUAL CF (000's) LIFE OF MINE	$30,302
PAYBACK PERIOD (YRS) FROM : START OF PRODUCTION	5.4
POST CLOSURE NET CASH FLOW:	$72,995

CAPITAL	
INITIAL CAPITAL (000'S)	$391,148
CONTINGENCY	$50,335
SUB-TOTAL	441,483
WORKING CAPITAL - 2011 - 2013	(7,012)
INITIAL CAP, PRE-PROD DEV & WORKING CAP	$434,471
SUSTAINING CAPITAL (000'S)	28,679
CONTINGENCY	4,302
TOTAL SUSTAINING CAPITAL	32,981
WORKING CAPITAL - 2014 - 2030	7,244
TOTAL MINE LIFE CAPITAL	$474,697

COSTS	
CASH OPER COST PER OUNCE	$476
TOTAL CASH COST PER OUNCE	$487
CAPITAL COST PER OUNCE	$250
TOTAL PRODUCTION COST PER OUNCE	$737
UNIT COSTS	
MINING COST ($/TONNE MINED)	$1.50
MINING COST ($/TONNE ORE)	$5.04
PROCESSING COST ($/TONNE ORE)	$7.44
G&A Cost ($/TONNE ORE)	$0.64
TOTAL OPERATING COSTS $/TONNE ORE	$13.12

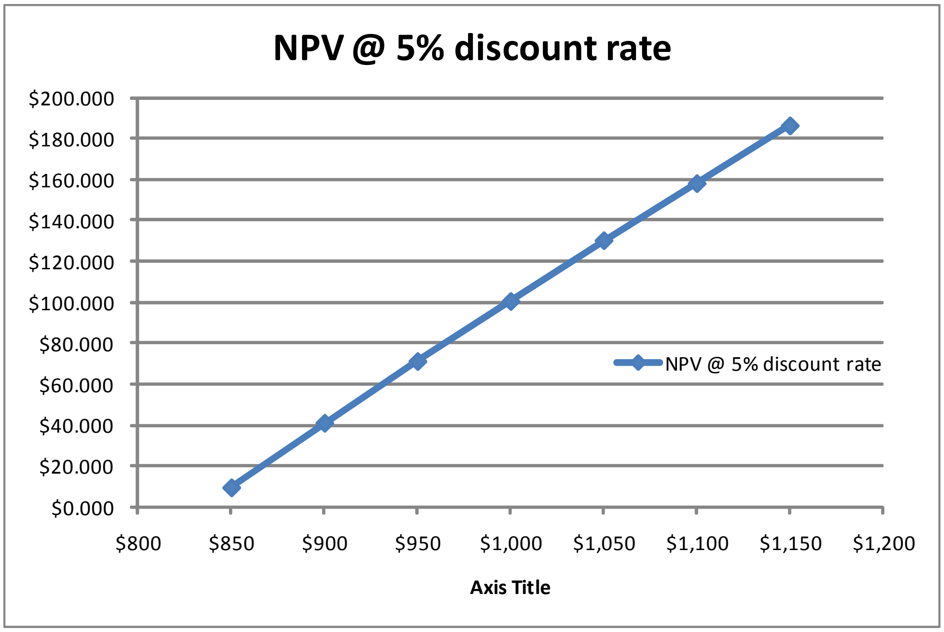
PROJECT PRODUCTION SCHEDULE / GOLD GRADES AND CONTENT

			Project Year										
MINE			-2	-1	1	2	3	4	5	6	7	8	9
ORE TONNAGE TO CRUSHER (000's)	ore tonnes	60,050			6,789	6,770	6,770	6,770	6,789	6,770	6,770	6,770	5,852
ORE GRADE	g Au/tonne	1.045			1.09	1.22	1.22	0.97	0.98	0.91	1.08	1.20	0.75
	toz Au/tonne	0.034			0.035	0.039	0.039	0.031	0.031	0.029	0.035	0.039	0.024
CONTAINED GOLD	g Au	63,027,314			7,381,696	8,271,100	8,228,457	6,535,370	6,621,247	6,148,847	7,342,132	8,114,680	4,383,787
	toz Au	2,026,374			237,327	265,922	264,551	210,117	212,878	197,690	236,055	260,893	140,942
WASTE TONNAGE MINED (000's)	waste tonnes	142,524		5,583	27,898	25,095	23,519	29,948	17,573	8,717	3,685	504	2
TOTAL MATERIAL MINED	total tonnes	202,573		5,583	34,687	31,865	30,289	36,718	24,362	15,487	10,455	7,274	5,855
STRIPPING RATIO	waste : ore	2.4			4.11	3.71	3.47	4.42	2.59	1.29	0.54	0.07	0.00
MILL													
ORE TONNAGE TO MILL (000's)	ore tonnes	60,050			6,789	6,770	6,770	6,770	6,789	6,770	6,770	6,770	5,852
MILL FEED GRADE	g Au/tonne	1.045			1.09	1.22	1.22	0.97	0.98	0.91	1.08	1.20	0.75
	toz Au/tonne	0.034			0.035	0.039	0.039	0.031	0.031	0.029	0.035	0.039	0.024
CONTAINED GOLD	g Au	63,027,314			7,381,696	8,271,100	8,228,457	6,535,370	6,621,247	6,148,847	7,342,132	8,114,680	4,383,787
	toz Au	2,026,374			237,327	265,922	264,551	210,117	212,878	197,690	236,055	260,893	140,942
MILL RECOVERY @ 82%	% recovery of Au	82%			82%	82%	82%	82%	82%	82%	82%	82%	82%
GOLD RECOVERED	g Au	51,682,398			6,052,990	6,782,302	6,747,335	5,359,003	5,429,422	5,042,055	6,020,548	6,654,038	3,594,705
	toz Au	1,661,626			194,608	218,056	216,932	172,296	174,560	162,106	193,565	213,932	115,572
REFINERY													
PAYABLE GOLD TO REFINERY	g Au	51,682,398			6,052,990	6,782,302	6,747,335	5,359,003	5,429,422	5,042,055	6,020,548	6,654,038	3,594,705
	toz Au	1,661,626			194,608	218,056	216,932	172,296	174,560	162,106	193,565	213,932	115,572

		Total LOM	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
GOLD PRICE	$/oz	$950			$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950
WASTE TONNES	000's	142,524		5,583	27,898	25,095	23,519	29,948	17,573	8,717	3,685	504	2									
TONNES ORE TO MILL	000's	60,050			6,789	6,770	6,770	6,770	6,789	6,770	6,770	6,770	5,852									
STRIPPING RATIO	waste:ore	2.37			4.11	3.71	3.47	4.42	2.59	1.29	0.54	0.07	0.00									
OUNCES PAYABLE	toz Au.	1,661,626			194,608	218,056	216,932	172,296	174,560	162,106	193,565	213,932	115,572									
GOLD GRADE	g/tonne	1.050			1.087	1.222	1.215	0.965	0.975	0.908	1.085	1.199	0.749									
GROSS GOLD SALES	$000's	$1,578,545			$184,878	$207,153	$206,085	$163,681	$165,832	$154,000	$183,887	$203,236	$109,794									
RENTAL INCOME/POWER INCOME	$000's	$178,401			$5,132	$5,132	$5,132	$5,132	$5,132	$5,132	$5,132	$5,132	$5,132	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690
GROSS REVENUE	$000's	$1,756,947			$190,010	$212,286	$211,218	$168,814	$170,964	$159,133	$189,019	$208,368	$114,926	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690
LESS REFINING, TRANS. & TREATMENT	$000's	7,477			876	981	976	775	786	729	871	963	520									
REVENUE FROM SALES	$000's	1,749,469			189,134	211,304	210,241	168,038	170,179	158,403	188,148	207,405	114,406	14,690	14,690	14,690	14,690	14,690	14,690	14,690	14,690	14,690
LESS ROYALTY	$000's	15,785			1,849	2,072	2,061	1,637	1,658	1,540	1,839	2,032	1,098									
NET REVENUE		$1,733,684			$187,286	$209,233	$208,181	$166,401	$168,521	$156,863	$186,309	$205,373	$113,308	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690
NET REVENUE AFTER PRODUCTION	$132,209																					
OPERATING COSTS																						
MINE	$000's	302,923			45,817	46,149	45,657	49,084	40,046	28,884	23,189	17,699	6,398									
MILL	$000's	446,567			49,969	49,789	49,829	49,789	50,088	49,893	49,848	49,848	43,194	957	886	890	268	268	268	268	268	248
G&A	$000's	38,489		3,849	3,849	3,849	3,849	3,849	3,849	3,849	3,849	3,849	3,849									
RECLAMATION	$000's	57,735			2,585			220	36					14,949	35,413	2,501	338	338	338	338	338	338
TOTAL OPERATING COSTS	$000's	$845,715		$3,849	$102,220	$99,787	$99,335	$102,943	$94,019	$82,625	$76,886	$71,395	$53,441	$15,906	$36,300	$3,391	$607	$607	$607	$607	$606	$586
MILL OPERATING COSTS AFTER PRODUCTION	4,321																					
RECLAMATION COSTS AFTER PRODCTION	54,894																					
OPERATING MARGIN	$000's	$887,969		($3,849)	$85,066	$109,445	$108,846	$63,459	$74,502	$74,238	$109,423	$133,978	$59,867	($1,216)	($21,610)	$11,299	$14,083	$14,083	$14,083	$14,083	$14,084	$14,104
CAPITAL COSTS																						
MINE EQUIPMENT	$000's	94,051		64,210	24,755	27		413	4,619	27												
PLANT EQUIPMENT & CONSTRUCTION	$000's	297,915	29,491	252,756	856	810	1,796	1,528	974	1,628	1,592	1,311	853	957	886	890	268	268	268	268	268	248
OTHER/CONTINGENCY	$000's	82,498	12,737	55,082	1,597	126	269	291	839	248	239	197	128	143	133	133	40	40	40	40	40	10,135
SUB-TOTAL	$000's	$474,464	$42,228	$372,048	$27,208	$963	$2,065	$2,232	$6,432	$1,903	$1,831	$1,508	$981	$1,100	$1,019	$1,023	$308	$308	$308	$308	$308	$10,382
SALVAGE VALUE	$000's	(59,567)												(43,937)								(15,630)
TOTAL CAPITAL	$000's	$414,897	$42,228	$372,048	$27,208	$963	$2,065	$2,232	$6,432	$1,903	$1,831	$1,508	$981	($42,837)	$1,019	$1,023	$308	$308	$308	$308	$308	($5,248)
CHANGES TO WORKING CAPITAL	$000's	232	294	2,457	(9,762)	2,522	51	(697)	879	872	756	532	641	4,066	(2,585)	(0)	67				0	140
PRE-TAX CASH FLOWS	$000's	$472,840	($42,522)	($378,353)	$67,620	$105,961	$106,730	$61,924	$67,190	$71,463	$106,836	$131,938	$58,245	$37,555	($20,045)	$10,276	$13,708	$13,775	$13,775	$13,775	$13,776	$19,211
CUMM. PRE-TAX CASH FLOWS	$000's	$472,840	($42,522)	($420,875)	($353,255)	($247,294)	($140,564)	($78,640)	($11,450)	$60,013	$166,849	$298,787	$357,032	$394,587	$374,542	$384,819	$398,527	$412,302	$426,077	$439,852	$453,628	$472,840
DD&A	$000's	474,464	8,446	80,464	85,905	86,098	86,511	78,512	9,332	4,271	4,445	4,333	4,083	1,884	1,708	1,546	1,306	1,172	1,013	871	728	11,838
PROFIT BEFORE TAX	$000's	413,505	(8,446)	(85,904)	(8,141)	14,266	13,702	(25,298)	58,263	65,554	101,999	127,571	54,115	11,849	12,096	12,254	13,116	13,250	13,409	13,551	13,695	2,605
INCOME TAX - Australian & Northern Territories	$000's	163,583							660	17,761	37,278	49,280	34,925	8,400				2,707	3,921	3,964	4,007	680
NET PROFIT	$000's	$249,922	($8,446)	($85,904)	($8,141)	$14,266	$13,702	($25,298)	$57,602	$47,793	$64,721	$78,292	$19,190	$3,449	$12,096	$12,254	$13,116	$10,543	$9,488	$9,587	$9,688	$1,925
AFTER-TAX CASH FLOW	$000's	$252,782	($42,522)	($378,353)	$67,620	$105,961	$106,730	$61,924	$66,530	$32,174	$47,123	$57,837	$27,229	$37,555	($20,045)	$10,276	$13,708	$11,068	$9,854	$9,811	$9,769	$18,531
CUMM. AFTER-TAX CASH FLOW	$000's	$252,782	($42,522)	($420,875)	($353,255)	($247,294)	($140,564)	($78,640)	($12,110)	$20,064	$67,187	$125,024	$152,253	$189,808	$169,764	$180,040	$193,748	$204,816	$214,670	$224,481	$234,251	$252,782

Sensitivity of Net Present Value a @5% discount rate to Gold Price ($millions)

Gold Price US$ / toz Au	$850	$900	$950	$1,000	$1,050	$1,100	$1,150
NPV @ 5% discount rate	$9.364	$40.937	$71.127	$100.497	$130.247	$158.192	$186.429

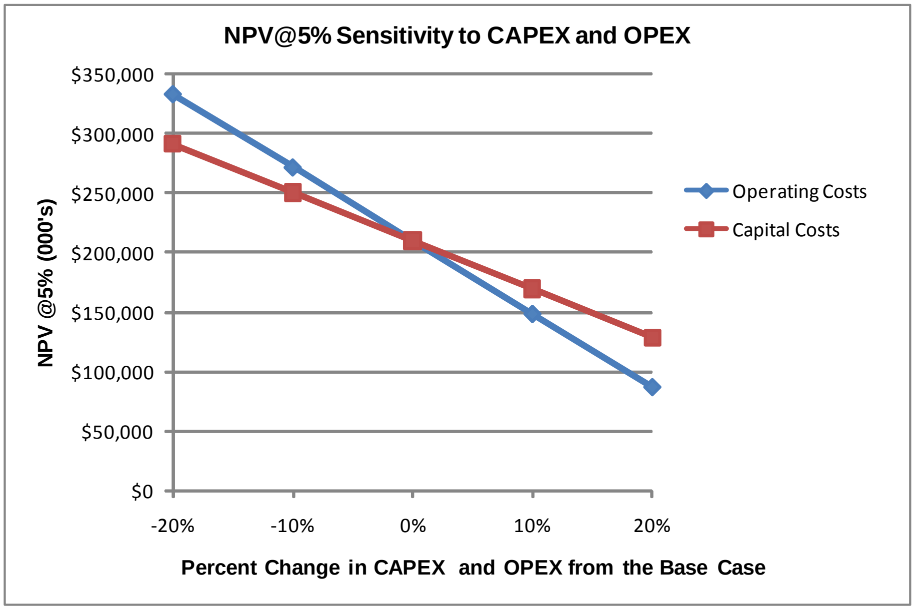


1.11.7 Capital and Operating Cost Sensitivities

Capital and Operating sensitivity analyses were performed on the Base Case reflecting mutually exclusive increases and decreases of 10% and 20% for both. A graph showing the results of these sensitivities is shown below.

Sensitivity of Net Present Value a @5% discount rate to OPEX and CAPEX ($000s)

Parameter	Parameter Change	-20%	-10%	0%	10%	20%
	Operating Costs	$332,861	$271,502	$210,144	$148,785	$87,426
	Capital Costs	$291,005	$250,574	$210,144	$169,713	$129,282



1.11.8 Sensitivities Deviating from the Base Case

Sensitivity analysis performed on the Base Case scenario at a gold price of $1,200/toz Au and 0.90 US/AUD exchange rate yielded an NPV of US$487.188 million at a 5 percent discount rate.

Two sensitivities were run on a 10.6 Mtpy case in which 139,175,000 tons are mined over a 13 year period. The first 10.6 Mtpy sensitivity considered a gold price of $950/toz Au and 0.85 US/AUD exchange rate. The analysis resulted in an NPV of US$154.511 million at a 5 percent discount rate.

A second 10.6 Mtpy sensitivity considered a gold price of $1,200/toz Au and 0.90 US/AUD exchange rate. The analysis resulted in an NPV of US$631.013 million at a 5 percent discount rate.

Note that although the economics resulting from the 10.6 Mtpy scenarios are lower than those for the 6.77 Mtpy case, nearly twice as much gold is produced (3,195,553 toz Au vs. 1,661,626 toz Au) by undertaking the larger project.

1.12 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 and progress the project through full feasibility. In addition to the Batman and Quigleys deposits, other known deposits/areas that warrant addition exploration include:.

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 and therefore, these areas remain highly prospective.

1.13 Recommendations

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

1.13.1 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 drillhole 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 waste characterization analyses. Initial waste characterization tests have provided a basis for the environmental considerations of this report; however, additional tests 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. Tt 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, is 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 has 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.

1.13.2 Metallurgy/Process Engineering

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 high pressure grinding roll (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 Mount 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.
- Consider a vibrating grizzly before the MP800 standard cone crushers to scalp undersize from the standard cone feed thereby reducing total tonnes passing through these crushers.
- 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 in detail.
- 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 Mount 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 thickened before pre-aeration and the thickening and filtration 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.
- A conceptual study of alternative processes may be appropriate in consideration of the copper contained in the ore, to confirm the selected flowsheet is the most cost-effective. Processes that may be considered include copper-selective flotation and subsequent cyanidation of flotation tailings, ammonia-cyanide leach, processes for recovery of copper and cyanide such as MNR, SART, Sceresini, ion-exchange resins or solvents. Options for cyanide detoxification should be addressed as part of this study.

1.13.3 Water Treatment

The following water treatment studies are recommended:

- Obtain NT Government approval to permit effluent releases from the existing Water Treatment Plant (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 Waste Rock Dump 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 and pilot treatment plant testing and regulatory review) to define, optimize and cost:
    - Water treatment plant capacity, design and location.
    - Impacted water collection, conveyance and storage.
    - Site-wide pipeline and pumping system requirements.
    - Sludge conveyance, storage and containment.
- Inventory all existing water management facilities to determine overall system arrangement, facility capacity, operation and maintenance status, remaining functional life.
- Develop stage-storage relationships for all water storage ponds.

1.13.4 Closure

The following closure studies are recommended:

- Complete a waste and cover material hydraulic properties analysis.

- Complete a tailings trafficability study.
- 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.
- Complete a waste and closure cover erosion and sediment control study.

1.13.5 Tailings Storage Recommendations

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

- 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 upon to include re-evaluation of the material properties, a liquefaction analysis, and deposition modeling;
- The inputs to the TSF water balance such as phreatic levels within the impoundment and the capacity of the raw water supply reservoir must be confirmed;
- The TSF water balance must be updated and expanded to include the Return Water Pond and Water Polishing Pond, as well as to optimize the size of the water pool to accommodate the water requirements of the processing facilities;
- The spillway design must be revised to account for any required change in cross-section at different stages of the impoundment;
- The condition of the existing toe drains, underdrains, and decant towers must be investigated to confirm their operation when tailings deposition resumes; and
- A pipe-crushing analysis should be performed on the existing installed pipes to ensure that the pipes will not crush under the additional loading by the tailings and the embankment raises.

1.14 Limitations

Tt 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 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

Tetra Tech, Inc. (“Tt”) was commissioned by Vista Gold Corp. ("Vista") in September 2009 to prepare a NI 43-101 compliant Preliminary Feasibility Study (PFS) for the Mt Todd Gold Project (the "Project") located in Northern Territory (“NT”), Australia. On March 1, 2006, Vista purchased the Mt Todd property, and the acquisition was completed on June 16, 2006, when the mineral leases transferred to Vista and funds were released from escrow. 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.

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 Tt. 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
John W. Rozelle, P.G.	Tetra Tech, Inc.	Overall Study QP
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
Mr. Erik Spiller, Dr. Deepak Malhotra	Tetra Tech, Inc. Resource Development Inc.	Section 16: Mineral Processing
Dr. Richard Jolk, P.E.	Tetra Tech, Inc.	Section 19: Mineral Economics

Neither Tt 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. Tt 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 Tt 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 include exploration, geology, and assay work completed by Vista as part of their 2007 and 2008 exploration program. Based on these additional data, Tt re-estimated the capital and operating costs, 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 October 1, 2010.

2.4 Units

All dollars are presented in US dollars unless otherwise noted. For the purpose of this report the exchange rates are US$0.75 = CDN$1.00 and US$0.85 = A$1.00 expect 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
oz Ag/t = troy ounces silver per short ton (oz/ton)
oz 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 = 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

Tt 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 government of the Northern Territory of Australia, 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. Tt 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. Tt 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

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 30 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).

Tenements

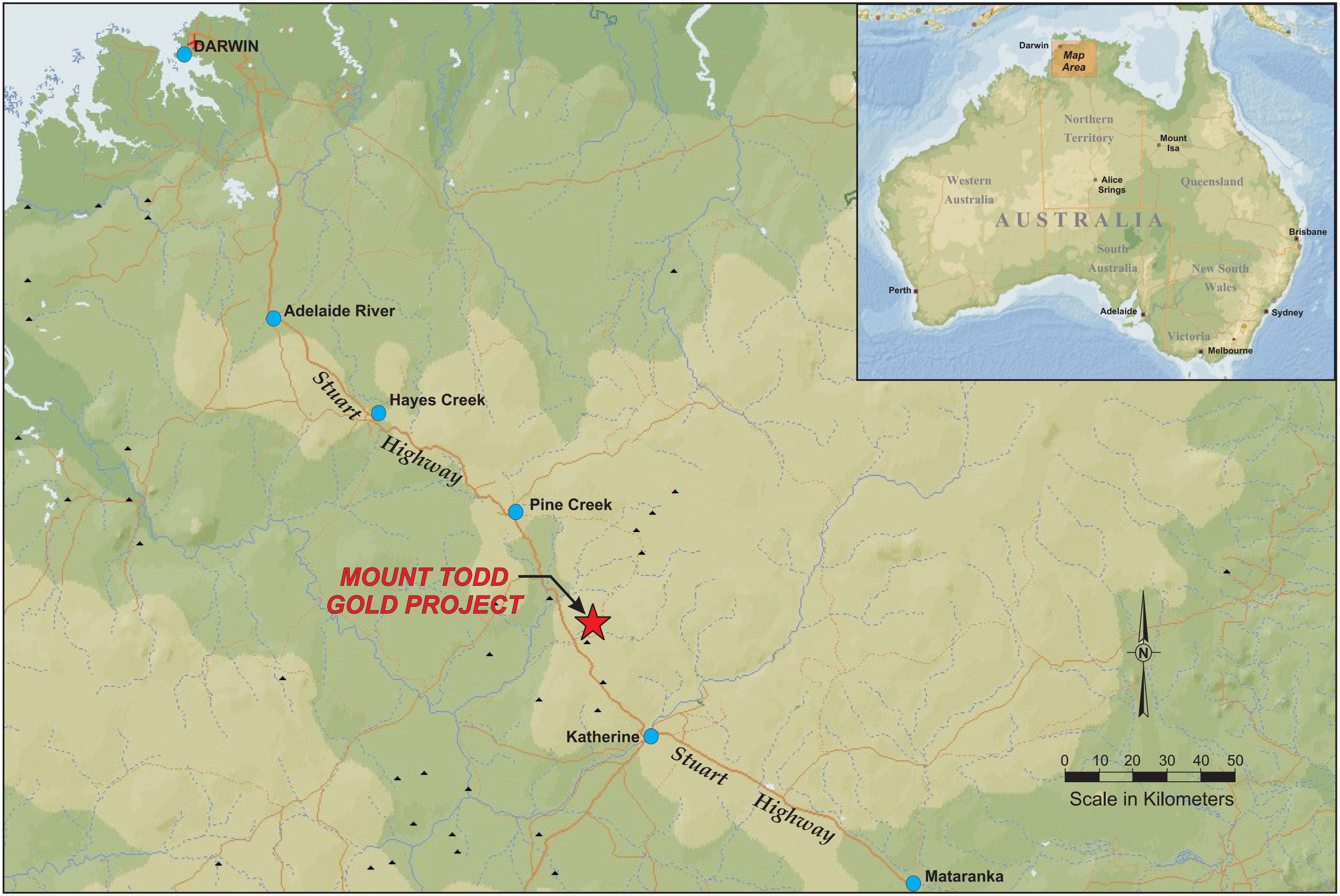
The concession consists of three individual mineral leases ("ML"), MLN1070, MLN1071, and MLN1127 comprising some 5,365.27 hectares. In addition, Vista controls exploration leases ("EL") EL25668, EL25669, EL25576, and EL 25670 comprising approximately 117,632 hectares. FIGURE 4-2 illustrates the general location of the tenements and the relative position of the two primary mineral deposits: Batman and Quigleys.

Lease and Royalty Structure

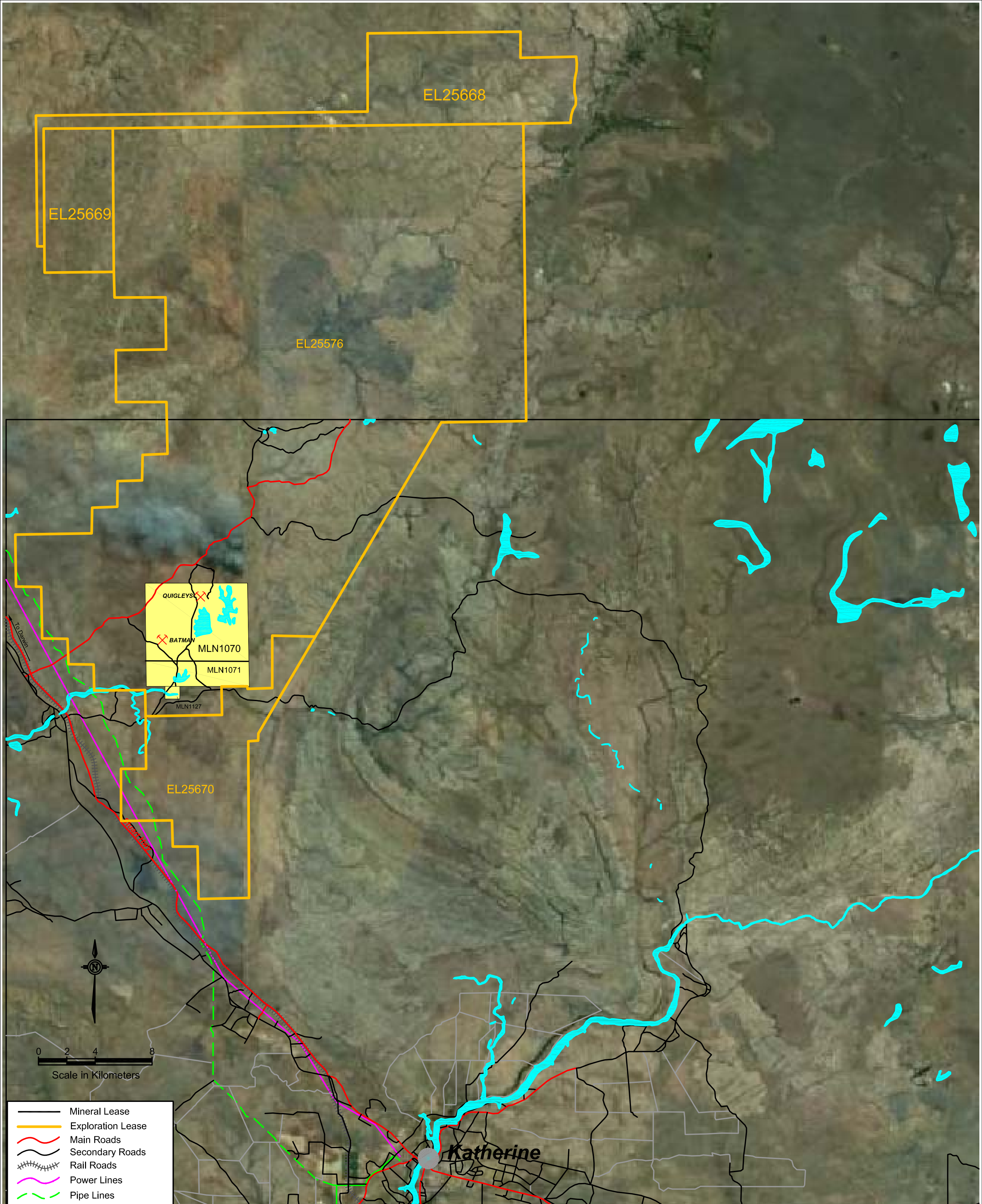
The agreement with the Territory is for an initial term of five years commencing January 1, 2006, with an extension of five years at Vista's option and three additional years possible at option of the Territory. During the first five-year term in accordance with the conditions of the agreement, Vista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

Vista paid the Territory's costs of management and operation of the Mt Todd site up to a maximum of A$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 Territory 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 Territory 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 Jawoyn Association Aboriginal Corporation (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 A$5,000 per month in return for consulting with respect to Aboriginal, cultural and heritage issues.



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



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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 Territory and the JAAC. Additionally, Vista will offer the JAAC the opportunity for joint venture participation in the operation on a 90% Vista / 10% 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 1% of the annual value of production with an annual minimum of A$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 4-2). 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 a Mining Management Plan (MMP) for resumption of mining operations.

The first step in formal mine permitting is the submission of a Notice of Intent ("NOI") or MMP to the NT government. These documents 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 MMP will be assessed against the following NOI requirements:

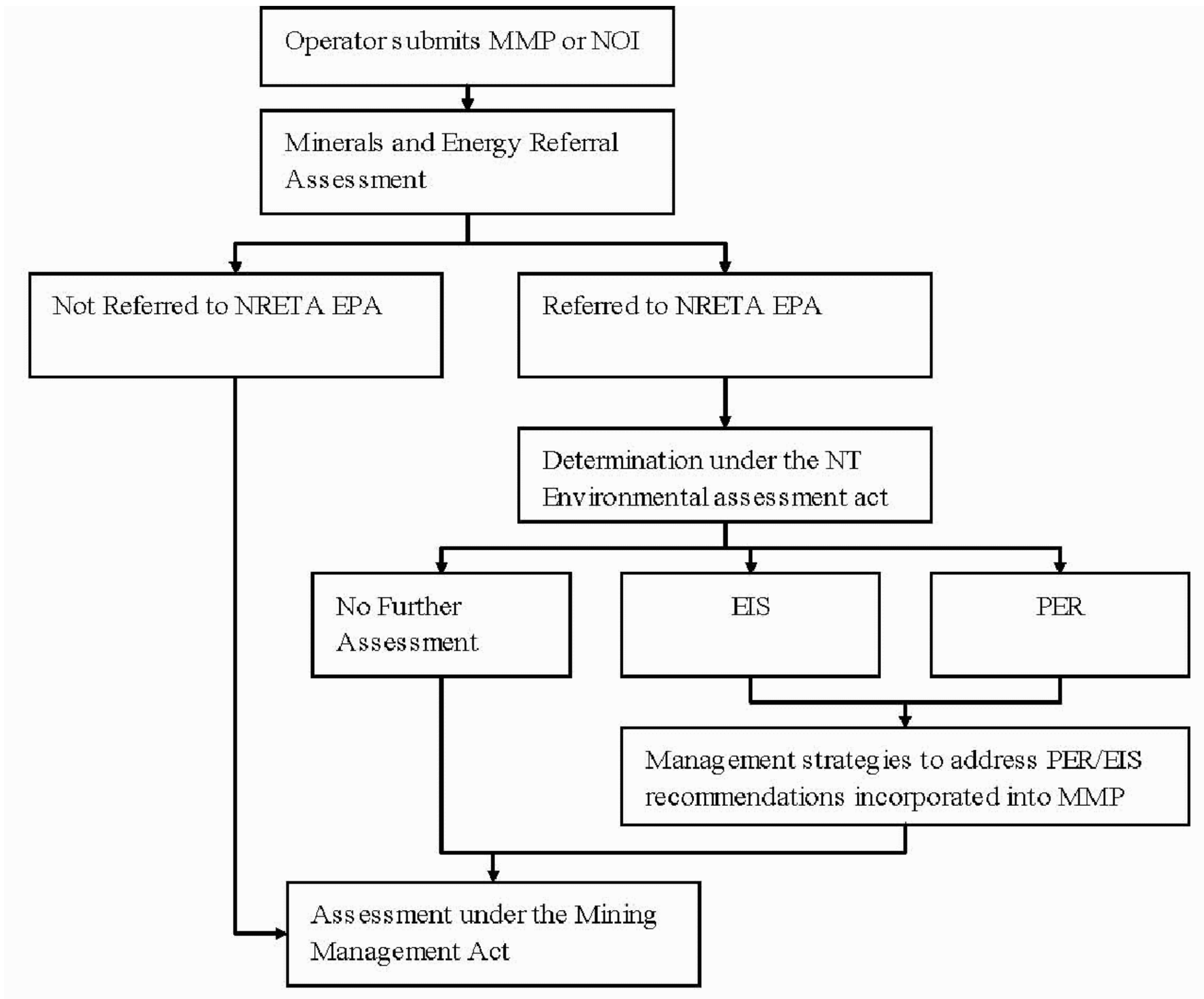
- Description of mining activities;
- Description of the existing environment;
- Safety, health and environmental issues relevant to the mining activities and the management system to be implemented;
- Description of current and proposed mine workings and infrastructure; and
- A plan and costing of closure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

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 (NRETA) for assessment under the NT Environmental Assessment Act as detailed in FIGURE 5-1 (DRDPIFR, 2008a). If the DoR recommends referral, NRETA will advise on the requirement for either a Public Environmental Report (PER) or an Environmental Impact Statement (EIS).

The guidelines provided by NRETA 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.

The estimated costs and timing of the three possible paths associated with the environmental assessment process are provided in TABLE 5-1. These costs are based on estimates provided by Gustavson (2006) and have been updated assuming an 18% increase in costs since 2006 (Engineering News Record, 2006). An allocation of US$300,000 for permitting the Mt Todd Project has been included in the project capital costs. This estimate assumes the permitting process will include an EIS; however, it is unclear at this time whether DoR will refer the project.



Extracted from DRDPIFR (2008) Environmental Assessment of Mining Proposals Guide. Department of Regional Development, Primary Industry, Fisheries and Resources.

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TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Mt TODD ENVIRONMENTAL ASSESSMENT PROCESS			A REVISION
	Project: Mt TODD GOLD PROJECT PREFEASIBILITY STUDY	Project no.: 310912A	DRAWING/FIGURE FIGURE 5–1	
	Location: NORTHERN TERRITORY, AUSTRALIA	Date:		

TABLE 5-1: ESTIMATED MINE DEVELOPMENT PERMITTING COSTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Task	Time[1]	Cost ($)[2]
Case 1: Assessment under the Mining Management Act (not referred to NRETA)		
Mining Management Plan or Notice of Intent	1 month	$30,000
Total	1 month	$30,000
Case 2: Referred to NRETA, Public Environmental Review Required		
Mining Management Plan or Notice of Intent	1 month	$30,000
Public Environmental Report	3 – 4 months	$118,000 - $212,000
Total	4 – 5 months	$148,000 - $242,000
Case 3: Referred to NRETA, Environmental Impact Statement Required		
Mining Management Plan or Notice of Intent	1 month	$30,000
Environmental Impact Statement	3 – 6 months	$177,000 - $295,000
Total	4 – 7 months	$207,000 - $325,000

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 (Chadwick T&T Pty LTD, 2009):

- Waste Discharge License 135 (EPA Northern Territory, 2005);
- Mt Todd Environmental Management Services – Report 1: Environmental Assessment (MWH, 2006a);
- Mt Todd Environmental Management Services – Report 2: Water Management (MWH, 2006b);
- Mt Todd Gold Project Preliminary Economic Assessment (Gustavson, 2006);
- Environmental Management Plan (Vista, 2007a);
- Mt Todd Waste Discharge License Report, 2006 – 2007 (Vista, 2007b);
- Mt. Todd Water Treatment Plant Commissioning Report (Vista, 2009);
- Mount Todd Blueprint Rehabilitation Strategy (BRS) Report (DRDPIFR, 2008b);
- Mt Todd Strategic Rehabilitation Reference Group: Status Update Papers in lieu of Meeting 11 (DRDPIFR, 2008c); and
- Mt Todd Mine Site Status Report, April 2008 to October 2008 (Vista, 2008).

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. 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% 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.

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.

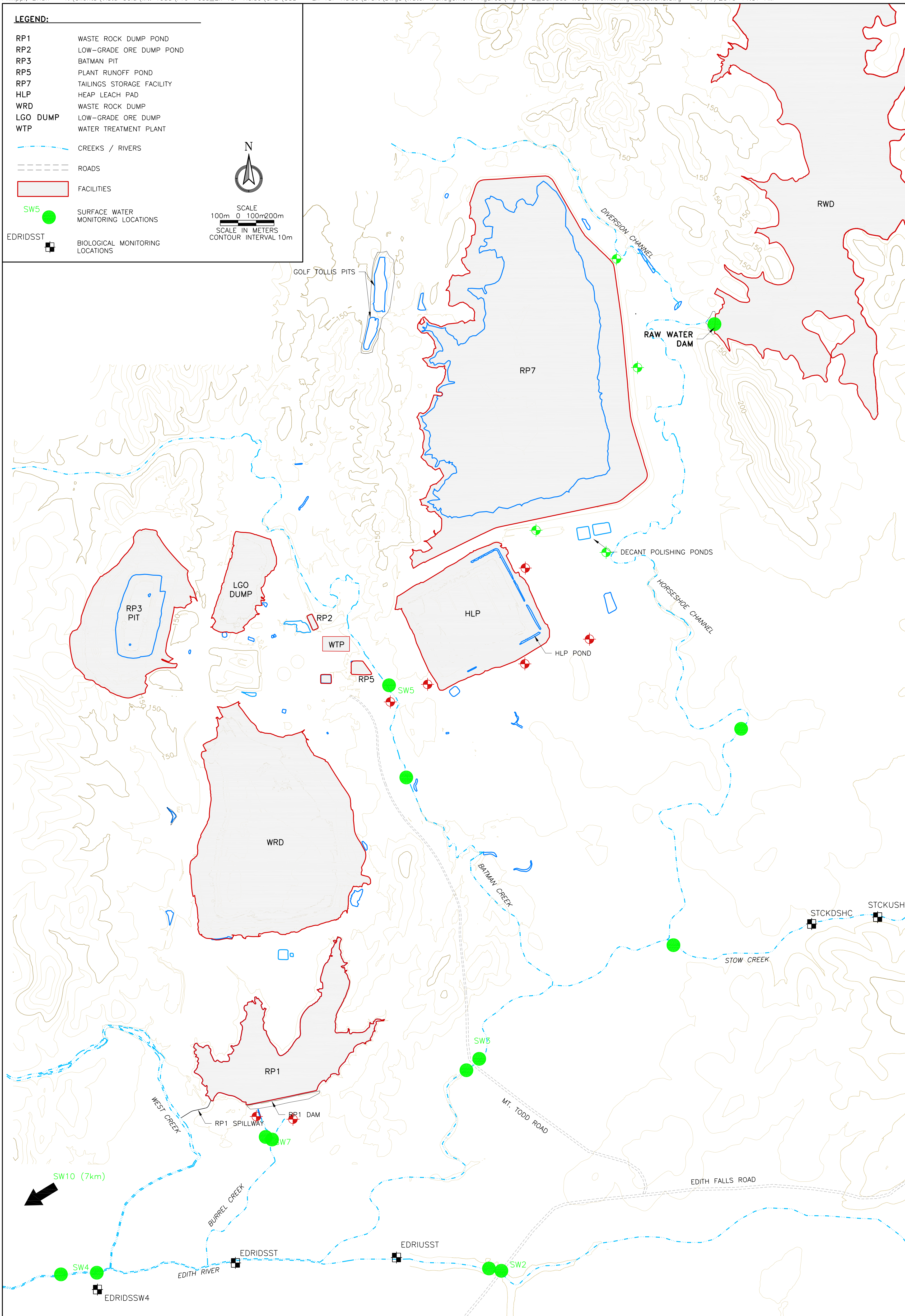
Updated Water Balance

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.

For the PFS, the Goldsim water balance model was updated to estimate water treatment requirements during all phases of the project including current, start-up, operations, closure and post-closure phases (Appendix I) as three separate models. The start-up/operations model was used to size a new water treatment plant with sufficient capacity to partially dewater Batman Pit by approximately February 2012 to permit in-pit preparation activities (lay backs) prior to the initiation of mining.

Several additional improvements to the water balance include:

- Revised climatological data including a new synthetic precipitation and evaporation data series;
- The catchment basins, sub-basins, and flow paths for the waste rock dump and pond (RP1) and the tailings storage facility (RP7) were delineated;
- Diversions were incorporated around RP1 and RP7, as appropriate; and
- Updated stage/storage-area curve for RP7.



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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

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 Waste Discharge License (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 Northern Territory, 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 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.

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

Analyte		($\mu g L^{-1}$) 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. Filterable copper concentrations at SW2 from 2004-2007 ranged from 0.27 to 3.1 µg/L with a median value of 0.4 µg /L. This criterion was breached several times in each of the previous four wet seasons (2002-2006) with the 2005/2006 wet season exceedences being partially attributed to delays in installation of the water management infrastructure. As a result of the water management strategy implemented by Vista, from November 2006 through October 2007 uncontrolled discharge from RP1 occurred on four days compared to 17 days over the previous reporting period. Five exceedences in filterable copper concentrations were observed at SW10 during the 2006/2007 wet season (FIGURE 5-3). 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.

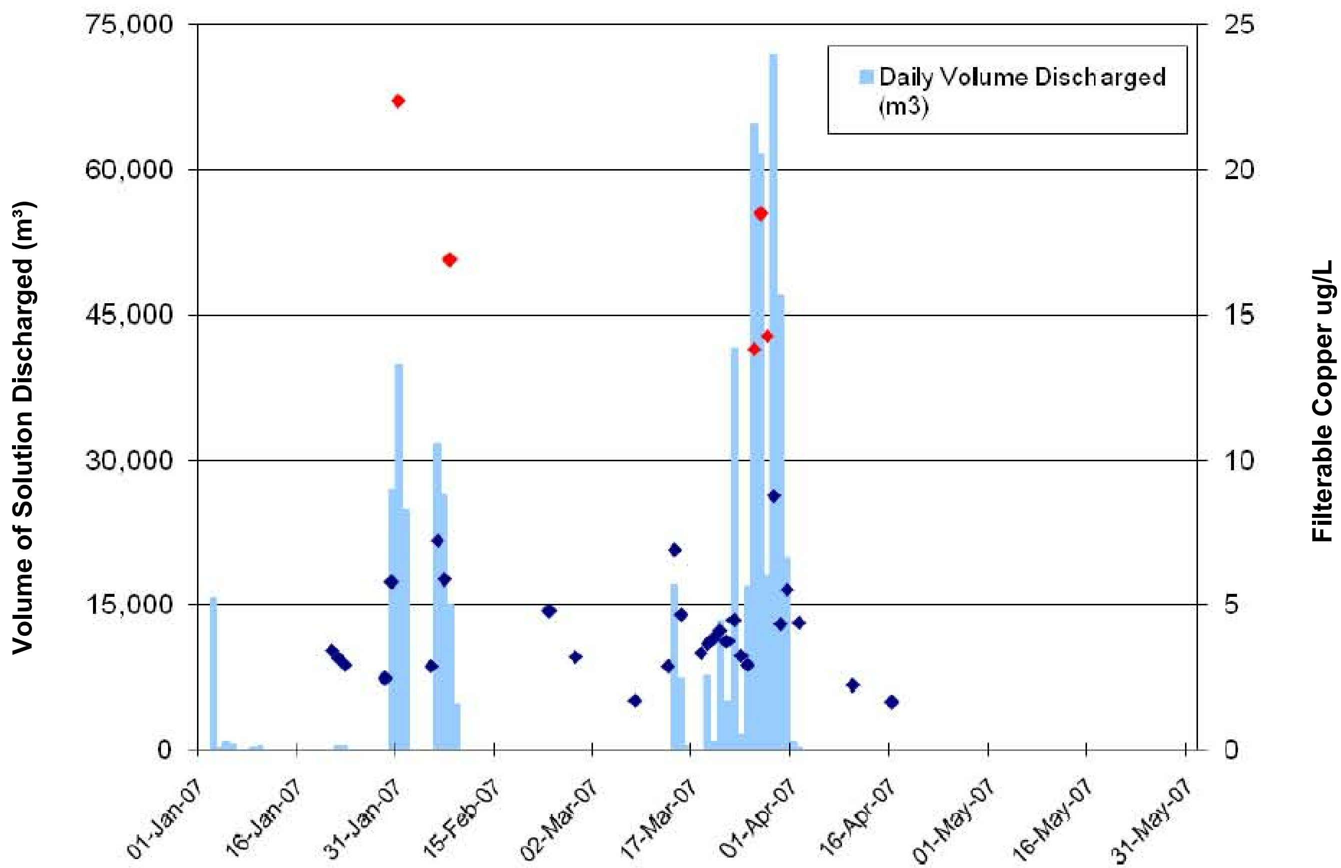
In August 2009, Vista commissioned the water treatment plant (WTP) to treat water from RP1 and RP3 using milk-of-lime at a capacity of 193 m^3/hr (Vista, 2009). Lime treatment removed 98% of the cadmium, 98.8% of aluminum and greater than 99% of the copper and zinc in RP1 water (TABLE 5-3). The sulfate 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), flows 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. Thee groundwater monitoring network is limited to the immediate vicinity of RP1, RP7, and the heap leach facility. 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).

From the Annual Mt Todd Project Waste Discharge License Report



*Red dots represent sample with filterable copper concentrations above the regulatory limit and blue dots represent samples concentrations below the cut off.

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	Project: Mt TODD GOLD PROJECT PREFEASIBILITY STUDY; Project no.: 310912A	DRAWING/FIGURE FIGURE 5-3
	Location: NORTHERN TERRITORY, AUSTRALIA; Date:	

Philippi, Brian - T:\CLIENTS\VISTA GOLD\MT. TODD\PRE-FEAS_EX TSF RAISE\CAD\C3D - EX TSF RAISE\DRAFT\DWGS\WATER MANAGEMENT FIGURES\FIG-5-3.DWG - Friday, October 01, 2010 9:29:51 AM

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

5.4.4 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 Northwest Territory 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. 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 waste rock dump 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. As a result the project shutdown without implementation of closure or reclamation activities, 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. % is comparatively low with interbedded and shale samples containing 0.51 and 0.31 wt. %, 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. % 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 ≤ 11 kg $CaCO_3$/tonne rock. A neutralizing 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% 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. % 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. % 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 Environment Protection and Biodiversity Conservation Act (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 3 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) and Vista formed the Mt Todd Strategic Rehabilitation Reference Group 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. 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).

5.4.5 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);
- Waste rock dump retention pond (RP1);
- The tailings storage facility (TSF also referred to as RP7);
- The heap leach facility;
- 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. In preparation for the 2006/07 wet season, 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 water treatment plant (WTP) is being used to raise the pH and reduce metals concentrations in water from RP1 and RP3 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 hydrogeological 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 verify that sufficient quantities of soil 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.

5.4.6 Reclamation and Closure

Tt was retained by Vista to develop a pre-feasibility closure plan (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 Waste Rock Dump Pond (RP1), the Low Grade Ore Stockpile Pond (RP2), the Batman Pit Lake (RP3), the Process Plant Runoff Pond (RP5), the Tailings Storage Facility Pond (RP7), and the Heap Leach 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 a Water Treatment Plant (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 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-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 metaliferous 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 6.77 Mtpy mine plan are as follows:

- Batman Pit (RP3);
- WRD;
- RP1 and pumping system,
- TSF (RP7);
- Processing 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).

All facilities identified above, as well as the major creeks, rivers and existing water monitoring locations in the vicinity of Mt Todd are shown on PFS FIGURE 5-1.

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

- Run-on diversions up-gradient of the TSF and RP1;

- New WTP;
- Equalization Pond;
- Sludge Disposal Cell;
- TSF Operational (and Closure) Spillway;
- TSF Moat;
- WRD Seepage Collection System;
- Clay Borrow Area; and
- Anaerobic treatment wetlands (or equivalent passive/semi-passive water treatment system).

This 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 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;
- Protect 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 a preliminary conceptual water balance model for the Batman Pit (See Appendix I), a hydrologic sink should be maintained passively during the post-closure phase. Therefore, active pit dewatering and treatment of pit water was 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 to primarily impede human access to pit and also reduce the inflow of surface water to the pit.

Waste Rock Dump

The dimensions of the WRD provided in Appendix I do not reflect the most recent dump designs provided to Tt by Mine Development Associates (MDA). The changes to the WRD design were not provided to Tt in time to revise our analysis. MDA's revised dump designs should not materially change the WRD closure plan provided below and may include a smaller disturbance footprint.

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 maximized the use of non-PAG waste rock for closure. The WRD will be constructed at a 2.5H:1V overall slope. Closure grading will include pushing waste rock to attain an overall slope of 3H:1V and to construct the stormwater management system. 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.

Prior to WRD grading a seepage collection system (e.g. French Drain, cutoff wall, grout curtain) will be constructed along the down-gradient toe of WRD and subsequently covered with waste rock from grading activities. Seepage water stored in the sump will be pumped to the New WTP for treatment. The seepage collection system is anticipated to separate seepage from storm water collected off the capped surfaces of the WRD. Tt estimates that seepage from the WRD will be ARD/ML and continue following closure. Therefore, ARD/ML collected by WRD seepage collection system will initially be pumped to the New WTP for treatment prior to release. The estimated mine-life rate-of-seepage from the WRD is provided in Appendix I. The WRD seepage collection system will be maintained until it is feasible to treat this and other ARD/ML on-site using a passive treatment system (see discussion below).

A store and release cover will be installed on the graded surface 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 metaliferous 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. To the degree possible, store and release covers will be installed concurrently during construction when portions of the WRD reach final grade. 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.

Prior to cover-placement subgrade materials 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 0.6 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, Processing Plant and Pad Area, and mine roads. Closure of the TSF, Heap, and Sludge Disposal Cell will include a 1.0 meter-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 Appendix J.

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. 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. 0.6 meter-thick) and Heap (1.0 meter-thick) was increased to 5 percent. Appendix J 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, Heap and TSF 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 TSF (See Appendix I) and as a basis for the development of mine-life water treatment system requirements and costs (See discussion below).

Tailings Disposal Facility

Approximately 60 Mt of slurry tailings will be disposed of in the TSF. Thickened tailings (65 percent solids - by weight) will be produced at the Processing Plant and disposed of in the TSF as a slurry. The particle size of the tailings is approximately 80 percent passing #100 mesh. Tt anticipates the majority of the impounded surface of the TSF as closure will be primarily composed of thixotropic tailings 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. To bridge thixotropic tailings, a one-meter thick cover composed of non-PAG waste rock will be installed on the impounded surface of the TSF to allow equipment access for the installation of a 1 meter-thick store and release cover (i.e. blended non-PAG waste rock-clay cover). The net flux of annual precipitation through the 2 meter-thick TSF cover was estimated to be zero.

Surface runoff from the impounded and capped surface of the TSF will be conveyed to the TSF pool. The operational spillway will be modified at closure to safely convey surface runoff from the TSF pool to Horseshoe Creek. Plans will be developed for the treatment of TSF surface discharge prior to release to Horseshoe Creek if there is unacceptable risk that water quality-based or WDL standards will be exceeded.

On the outside slopes of the main TSF dam and saddle dams a one meter-thick blended non-PAG waste rock-clay cover will be installed. Prior to cover placement on the TSF dam faces, 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. To the degree possible, store and release covers will be installed concurrently during construction when portions of the TSF dams reach final grade.

The TSF decant pipes will be plugged with concrete and a seepage collection moat or a series of moats will be constructed at closure to collect seepage from the toe of the TSF. The bottom and down-gradient interior side slope of moat will be lined with LLDPE. The Return Water Pond or Polishing Pond will be modified to receive seepage from TSF moat via gravity and the TSF foundation drains.

Tt estimates the tailings placed in the TSF will be PAG and seepage from the TSF will be ARD/ML. Tt also assumed that the operation of the Processing Plant during the production phase will use all excess water from the TSF. As such, TSF seepage will only be collected and treated prior to release during the closure and post-closure phases of the mine-life. The estimated rate of draindown and steady-state seepage from the TSF are provided in Appendix I. The Return Water Pond or Polishing Pond and the moat will be maintain until it is feasible to treat this and other ARD/ML on-site using a passive treatment system (see discussion below).

Processing Plant and Pad Area

A new processing plant will be built for renewed mining (Appendix D). Once ore processing ceases, the Processing Plant will be decommissioned, decontaminated, demolished and any reusable equipment and materials will be salvaged and resold. The current operating assumption is that Processing 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. Material that cannot be treated in situ will be excavated to the extent of soil contamination and disposed of in the WRD, TSF, on-site solid waste landfill or an off-site facility that is certified to accept and dispose of, for example, petroleum-and solvent-contaminated soil. Concrete foundations, walls and bridges and other non-reactive, non-combustive, non-corrosive and non-hazardous demolish waste will be broken up and either:

- Placed in the WRD; and/or
- Buried in-place or backfilled against cut banks and highwalls throughout the Processing Plant and Pad Area, as well as other areas that will be reclaimed at Mt Todd.

Surface and large shallow pipes will be removed and pipes at depth will be plugged with concrete or other suitable materials.

The Process Plant Area will be graded to blend into the surround topography and drain towards Batman Creek. The Processing Plant Area and Pad will be covered with a 0.6 meter-thick blended non-PAG waste rock-clay cover as described previously. Prior to cover placement subgrade materials will be deep ripped to reduce compaction and prevent slippage of the cover. Storm water drainage, erosion, and sediment controls will be designed and constructed to minimize erosion and channel scour and the area will be revegetated.

The New WTP, Equalization Pond and Sludge disposal cell will be left in place, up-graded if necessary and used to treat ARD/ML during the closure and post-closure phases. These facilities will be closed as discussed below when it is feasible to treat ARD/ML in passive treatment systems. RP4, RP5 and RP6 will be closed as described below if it is determined they are no longer needed for water storage or containment of surface runoff from the reclaimed Processing Plant and Pad Area. We assume that the Processing Plant and Pad Area would no longer be a source of ARD/ML following closure.

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, Tt assumes the Weak Acid Dissociable (WAD) Cyanide concentration of Heap effluent and pore water meets applicable standards. Therefore, Tt 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 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. Tt 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 maintain 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 Stockpile

The LGO Stockpile will no longer be needed for mineral processing in year five of the production phase. Ore will be removed from the stockpile pad. While not confirmed by test results, the material below the LGO Stockpile is assumed to be PAG. This assumption must be verified prior to closure of the LGO Stockpile.

Residual ore will be removed from the stockpile area. Tt assumed this area will not be graded. 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 0.6 meter-thick blended non-PAG waste rock-clay cover as described previously. 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. It is assumed that RP2 will be closed as described below during the closure phase and that the LGO Stockpile will no longer be a source 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

Prior to the initiation of New WTP construction the Equalization Pond will be constructed for mixing of ARD/ML from various on-site sources prior to treatment and temporarily store ARD/ML in case of system upset (i.e. ARD/ML surge due to extreme storm events or shutdown of the New WTP). In the event of a system failure or shut-down for maintenance of the New WTP, the Equalization Pond would provide up to 3 days of ARD/ML storage at an inflow rate equal to the treatment capacity of the New WTP or 1000m^3/hour. As previously discussed a seepage collection system which includes foundation drains and a small sump will be constructed at the toe of the WRD. The foundation drains will be covered during grading. The sump will temporarily store seepage from the WRD. The seepage collected in the sump will be pumped to the New WTP for treatment prior to release. These and the existing ponds at Mt Todd will be maintained for the collection of seepage, storm water and ARD/ML until long-term quality of water collected by the WRD seepage collection system meets applicable standards, flows to collection system cease or alternative passive water treatment system are installed. 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 (Appendix I).

Tt anticipates that RP2 will remain during the production phase following closure of the LGO Stockpile. Prior to the closure of the Heap during the first year of the production phase, the Heap Leach Ponds will also be modified. All other existing ponds as well as the Equalization Pond are anticipated to remain through closure. During the closure phase the Return Water Pond or Polishing Pond will be modified to receive seepage from TSF moat via gravity and the TSF foundation drains.

During the closure phase all residual water in RP1 will be pumped to the New WTP for treatment. Following this, the 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. In the post closure phase, RP4, RP5 and RP6 will be phased out depending on the need for surface storage capacity. The remaining ponds post-closure (i.e. Heap Leach Ponds, WRD sump, and Return Water Pond or Polishing Pond, Equalization Pond) may be incorporated into the passive water treatment system or used as backup water storage in case treatment upset occurs.

To decommission and close ponds, residual standing water in the pond will be pumped to the New WTP. Prior to decommissioning and regrading, the accumulated sediments in the base of all pond and foundation materials will be tested to determine their chemical characteristics. Acidic, PAG and metaliferous materials, will be disposed of in the on-site landfill, treated *in situ* or buried in place if it can be demonstrated that this disposal method will not adversely affect ground or surface water. If present, liners will be cut and folded in place. The pond berms will be pushed towards the pond void until the area no longer impounds water. The top 0.6 meter of graded material will meet the criteria immediately above and have the physical and chemical properties to support plant growth. Storm water drainage, erosion, and sediment controls will be designed and constructed to minimize erosion and channel scour and the area will be revegetated.

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

Base 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, Tt 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 would be graded and reclaimed (on an interim of final basis) or remain open until closure activities are completed on site.

Water Treatment

There are two fundamental approaches to water treatment over the remainder of the mine-life. The first phase will include active water treatment in the New WTP. This phase will be initiated during pre-production and will continue to until ARD/ML flow and water quality properties are conducive to treatment in a passive/semi-passive water treatment system. We anticipated this will be approximately year 6 of the post-closure phase. The second phase will be initiated post-closure and include passive water treatment in an anaerobic wetland (or equivalent passive treatment system). 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 existing WTP is located adjacent to the Primary Crusher near the LGO Stockpile. 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 predicted volume of water in the Batman Pit as of May 2010 is approximately 10.0 million m^3);
- At a minimum, maintain pit water elevation 5 meters below the lowest planned pit operation anticipated during each year of pit preparation and production;
- Maintain the pit in a dewater 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 RP 7 at an elevation less than or equal to 136 meters 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 Tt's methodologies and the estimates of hydrated lime consumption and sludge production during the 20 year period when the New WTP is anticipated to be in operation. As discussed below, following year six of the post-closure phase a passive water treatment system would be installed to treat ARD/ML at Mt Todd.

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 55 million m^3 of ARD/ML will be treated in the New WTP. This is estimated to consume approximately 17,000 m^3 of hydrated lime and produce 75,000 m^3 of sludge. Approximately 18 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 dewater of the Batmen Pit during the pre-production phase, treatments rates are approximately 9 million m^3/year, compared to the 2 million m^3/year treated during the production phase. At closure, treatment rates increase to approximately 4 million m^3/year. This is largely attributed to dewatering RP 1 to permit the removal of sediments and breaching of the RP1 Dam. Treatment rates fall off to approximately 1 million m^3/year during the post-closure period. Therefore, excess treatment capacity may exist during the production phase and potentially throughout the remaining life of the New WTP.

These conclusions and design recommendations are based on preliminary estimates and must be confirmed based on detailed hydraulic investigations and Vista's design 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).

Tt estimates that in year 6 of the post-closure phase, the flow-weighted average acidity of all sources may be on the order of 300 to 400 mg/L Non-CO_2 Acidity (as $CaCO_3$ mg/L) when total post-closure flows ~ ≤ 120 m^3/hour. At or near this time period during the post-closure phase, an anaerobic wetland or successive alkalinity producing systems (SAPS) should be suitable for passive treatment of ARD/ML at Mt Todd. The estimated area of the anaerobic wetlands necessary to treat ARD/ML given the flow and quality above would be approximately 6 hectares. This was based on ARD/ML treatment to 0.0 mg/L Non-CO_2 Acidity (as $CaCO_3$ mg/L) and a anaerobic treatment efficiency of 16.4 g acidity/m^2/day (See discussion in Appendix J).

Estimating flows and water quality 20 years in the future is wrought with uncertainty and the potential introduction of 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.

Closure Cost Estimate

Tt 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 cost were estimated at a ± 25 percent level of accuracy based on the following:

- 6.77 million tonnes/year (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;
- Tt's recent mine closure and water treatment costing experience; and
- Best professional judgment.

Tt used a 2010 estimate for the demolition of a 40,000 tonnes/day (TPD) Silver-Zinc Mill (US$12.3 Million) processing plant as the basis for estimating the cost to demolish the Processing 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 pre-feasibility level cost estimate for implementing this closure plan is US$64,938,000. As summarized in TABLE 5-4 this cost estimate includes closure of the sludge disposal cell, the equalization the pond clay borrow pit, contingency, engineering re-design, construction quality assurance, 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 US$31,203,000. This cost estimate included 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: 6.77 MTPY CLOSURE AND MINE-LIFE WATER TREATMENT COST ESTIMATE SUMMARY
VISTA GOLD CORP. – MT TODD PROJECT
October 2010

Area	**Cost (US$)**[1]
Tailings Storage Facility	$ 18,050,000
Heap	$ 2,585,000
Processing Plant And Pad Area	$ 8,813,000
Batman Pit	$ 99,000
Waste Rock Dump	$ 16,702,000
WRD Retention Pond	$ 525,000
Low Grade Ore Stockpile	$ 256,000
Mine Roads	$ 5,787,000
Clay Borrow Area[2]	$ 1,243,000
Sludge And Equalization Pond Closure[2]	$ 286,000
Total Direct Closure Cost	**$ 54,344,000**
Mobilization/Demobilization (Assume On-Site Mining Equipment Fleet Used)	$ 0-
Incidentals (Communication, Misc. Supplies, Etc.) = 0.5 % Of Total Direct Cost.	$ 345,000
Haul Road Maintenance During Closure = 1 % Of Total Direct Cost.	$ 689,000
Construction Quality Assurance = $200,000 /Year During 3 Year Closure Period	$ 600,000
Engineering Re-Design = 2 % Of Total Direct Cost.	$ 1,378,000
Contingency = 8 % Of Total Direct Cost.	$ 5,514,000
Total Indirect Cost [3]	**$ 8,526,000**
Annual Site Maintenance and Monitoring For 6 Years Post Closure	$ 2,068,000
Total Closure Costs	**$ 64,938,000**
Water Treatment System Facility/Component	**Cost (US$)**
Active Water Treatment And Sludge Disposal System Construction[2]	$ 5,799,000
Passive Water Treatment System	$ 8,780,000
Total Direct Water Treatment Construction Cost	$ 14,569,000
Prep-Production Period (Years -2 and -1) Water Treatment O&M, Reagent and Pumping[4]	$ 8,102,000
Production Period (Years 1 through 9) Water Treatment O&M, Reagent and Pumping[3]	$ 4,201,000
Closure Production Period (Years 10 through 12) Water Treatment O&M, Reagent and Pumping[4]	$ 2,733,000
Post-Closure Production Period (Years 13 through 18) Water Treatment O&M, Reagent and Pumping[4]	$ 1,588,000
Total Mine-Life Water Treatment O&M, Reagent and Pumping[4]	$ 16,624,000
Total Mine-Life Water Treatment Costs	**$ 31,203,000**

[1] Cost rounded to nearest $1,000 in current US$.
[2]Following submittal of the PFS projected cash flow for the project, Tetra Tech determined the size of the Equalization Pond, Sludge Disposal Cell and Clay Borrow Area were erroneous calculated. As such the published cash flow is based on oversized water treatment and closure facilities. The direct costs for the construction and closure of these facilities should decrease by approximately US$2.3 million. Indirect costs (e.g. re-design, quality assurance, contingency, post-closure maintenance) should decrease by approximately US$0.27 million. According to Tetra Tech's stated assumptions and the limitations of our analysis, the size of the Equalization Pond, Sludge Disposal Cell and Clay Borrow Area reported here are correct. However, to maintain consistency with the published cash flow, the costs to construct and close these facilities were not corrected.

[3]Includes indirect costs associated with the construction of Water Treatment System
[4]Includes Plant O& M, Lime, and Water and Sludge Pumping

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 WRD, TSF, HLP 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 WRD, TSF, HLP 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 impounded surface of the TSF is adequate to bridge thixotropic tailings to permit the installation of the 1m-thick store and release cover on the TSF.
- The operational TSF spillway 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.
- Grading mine waste, and cut and fill slopes to a maximum 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.
- Leached ore in the HLP will remain within the current HDPE-lined area following grading to following the attainment of 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 Line will be disposed of in an adequately designed and operated on-site disposal facility.
- Concrete foundations, walls and bridges and other non reactive, non combustive, non corrosive and non hazardous demolish waste will be broken up and either:

- Placed in the WRD; and/or
- Buried in-place or backfilled against cut banks and highwalls throughout the Processing 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.
- Processing Plant demolition cost estimates include the following assumptions:
  - Salvage value will equal the removal cost for all Processing 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-base effluent standards established for the Edith River as currently approved) from the existing WTP and New WTP to Batman Creek.
- Sulfate, arsenic and other oxyanion numeric standards 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 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 years -2 through post-closure.
- By 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; and
  - LLDPE-lined (or equivalent) sludge disposal cell for the dispose of water treatment sludge produced by the New WTP.
- 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 TSF is a closed hydrologic system.
- TSF foundation drains will be operational and maintained during closure and post-closure.
- Operation of the Processing Plant during the production period will use all excess water from the TSF.
- Adverse impacts to ground water from previous and planned mining and processing activities at Mt Todd do and will not occur. As such groundwater remediation due to ARD/ML contamination from Mt Todd is not necessary.
- The RP2 and Processing Plant and Pad Area will no longer be a source of ARD/ML immediately following closure.
- 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.
- Reagent-grade lime will be produced at or adjacent to Mt Todd for the prices noted in Tetra Tech cost estimate in quantities sufficient to meet mine-life water treatment requirements (Approximately 17,000 tonnes of reagent grade hydrated lime).
- Existing pumps and 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 ≥ 50 percent.
- Passive water treatment systems (most likely anaerobic wetland or Successive Alkalinity Producing System - SAPS) may be activated when total post-closure ARD/ML flows are 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.

Scoping-Level Closure Cost Estimate

As part of the larger production sensitivity analysis presented in this report, the 10.6 Mtpy scenario included a comparison of the closure and water treatment plans and costs to the 6.77 Mtpy mine plan. For this sensitivity analysis, the closure and water treatment plans and strategies present in the 6.77 Mtpy were replicated and proportionally scaled according to the quantities (e.g. facility dimension, material/fluid volumes, surface areas, disturbance footprints) estimated for the 10.6 Mtpy mine plan.

The production phase and thus active water treatment during production, increase from 9 years to 13 years. The period of active water treatment during the post closure phase increased from 6 years to 11 years due to the assumed increase in ARD/ML from the New TSF and the total volume of ARD/ML treated in the New WTP. Closure schedules were also modified to address the concurrent closure of the existing TSF during the production period. In addition, cover material quantities and the quantity of other materials need for closure (e.g. rip rap, clay, liners) increased especially due to the increased disturbance created by the New TSF.

As summarized in TABLE 5-5 the scoping-level cost estimate for implementing the closure plan for the 10.6 Mtpy mine plan is US$120,564,000. TABLE 5-5 also includes the scoping-level cost estimate for implementing the mine-life water treatment plan for 10.6 Mtpy mine plan and is US$51,364,000. These estimated costs include the same basic costing approaches and assumptions used to estimate the cost to implement the closure and mine-life water treatment for the 6.77 Mtpy mine plan.

TABLE 5-5: 10.6 MTPY SCOPING-LEVEL CLOSURE AND MINE-LIFE WATER TREATMENT COST ESTIMATE SUMMARY
VISTA GOLD CORP. – MT TODD PROJECT
October 2010

Area	**Cost (US$)**[1]
Tailings Storage Facility	$ 18,049,000
New Tailings Storage Facility	$ 21,973,000
Heap	$ 2,585,000
Processing Plant And Pad Area	$ 11,419,000
Batman Pit	$ 174,000
Waste Rock Dump	$ 31,785,000
WRD Retention Pond	$ 300,000
Low Grade Ore Stockpile	$ 256,000
Mine Roads	$ 8,680,000
Clay Borrow Area	$ 1,717,000
Sludge And Equalization Pond Closure	$ 1,365,000
Total Direct Closure Cost	**$ 98,303,000**
Mobilization/Demobilization (Assume On-Site Mining Equipment Fleet Used)	$ 0-
Incidentals (Communication, Misc. Supplies, Etc.) = 0.5 % Of Total Direct Cost.	$ 619,000
Haul Road Maintenance During Closure = 1 % Of Total Direct Cost.	$ 1,239,000
Construction Quality Assurance = $400,000 /Year During 3 Year Closure Period	$ 1,200,000
Engineering Re-Design = 2 % Of Total Direct Cost.	$ 2,478,000
Contingency = 8 % Of Total Direct Cost.	$ 9,911,000
Total Indirect Cost [2]	$ 15,447,000
Annual Site Maintenance and Monitoring For 11 Years Post Closure	$ 6,814,000
Total Closure Cost	**$ 120,564,000**
Water Treatment System Facility/Component	**Cost (US$)**
Active Water Treatment And Sludge Disposal System Construction	$ 7,705,000
Passive Water Treatment System	$ 17,872,000
Total Direct Water Treatment Construction Cost	$ 25,577,000
Prep-Production Period (Years -2 and -1) Water Treatment O&M, Reagent and Pumping[3]	$ 8,075,000
Production Period (Years 1 through 13) Water Treatment O&M, Reagent and Pumping[3]	$ 8,415,000
Closure Production Period (Years 14 through 16) Water Treatment O&M, Reagent and Pumping[3]	$ 4,290,000
Post-Closure Production Period (Years 17 through 27) Water Treatment O&M, Reagent and Pumping[3]	$ 5,007,000
Total Mine-Life Water Treatment O&M, Reagent and Pumping[3]	$ 25,787,000
Total Mine-Life Water Treatment Costs	**$ 51,364,000**

[1] Cost rounded to nearest $1,000 in current US$.
[2] Includes indirect costs associated with the construction of Water Treatment System
[3] Includes Plant O& M, Lime, and Water and Sludge Pumping

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 million tonne per annum heap leach plant designed to recover 90,000 ounces per annum 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 annum 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 - oz	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 million tonnes per annum 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 $A369 ($US266) per ounce. Capital cost for Phase II was estimated at $A207.8 million.

The Pegasus Board approved the project on 17 August 1995 and awarded an EPCM contract to BKK in October 1995. Commissioning commenced in November 1996. Final capital cost to complete the project was $A232 million (US$181 million).

Design capacity was never achieved due to inadequacies in the crushing circuit. An annualized throughput rate of just under 7 million tonnes per annum 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 $US400 in early 1996 to below $US300 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 14 November 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 2%, Multiplex Resources 93%, and Pegasus Gold Australia 5%. The joint venture appointed General Gold as mine operator, which contributed the operating plan in exchange for a 50% share of the net cash flow generated by the project, after allowing for acquisition costs and environmental sinking fund contributions. General Gold operated the mine from March 1999 to July 2000.

6.1 History of Previous Exploration

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

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

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

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

Follow-up work concentrated on alluvial tin and, later, auriferous reefs. Backhoe trenching, costeaning, and ground follow-up were the favored mode of exploration. Two diamond drillholes were drilled at Quigleys 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 drillhole program, with an aggregate meterage of 676.5 m, to test the gold content of Quigleys Reef over a strike length of 800 m. Following this program 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 US$200 million

Australia Pty Ltd.	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.

6.2 Historic Drilling

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

Batman Deposit

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

Drillhole Density and Orientation

Pegasus was aware of the problem of drillhole 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 drillhole 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 meters by 25 meters. 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 meters by 50 meters, but decreased to 50 meters by 100 meters and greater at depth.

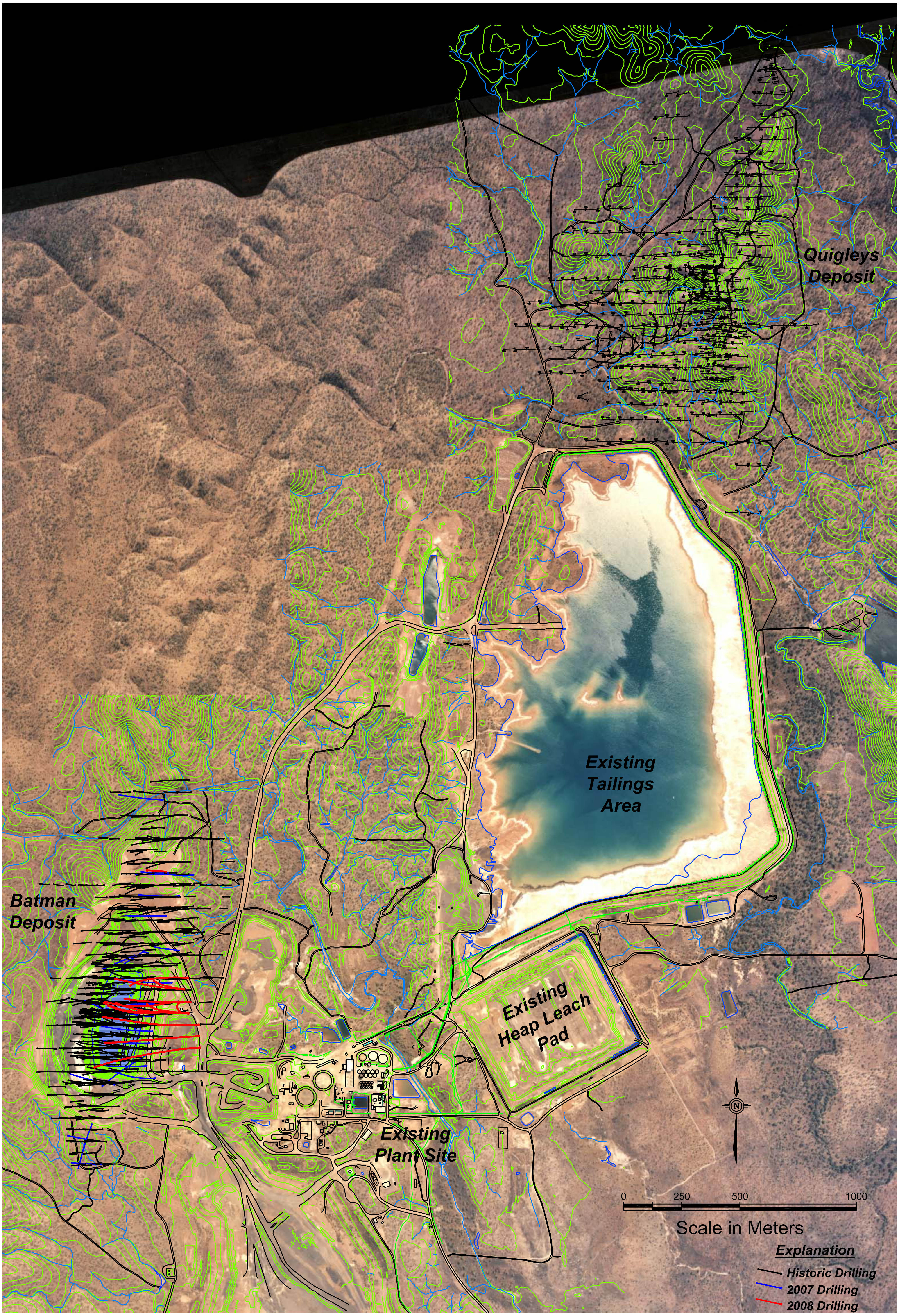
At the time of The Winters Company's ("TWC") site visit in 1997, the drillhole database numbered 730 holes. It is not known if any holes were excluded from the Pegasus exploration models. Most of the new drilling that had been added since the 1994 MRT model was relatively shallow. TWC reviewed PGA's 50-meter drill sections through the Batman Deposit and saw that there was a marked decrease in drillhole spacing below 1000 RL (the model has had constant 1000 meters 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 drillhole spacing in the south of 1000 N on the 954 RL bench plan approached 80 meters by 80 meters. 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 meters.

Another potential problem related to drilling is the preferred orientation of the drillholes. 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 drillhole 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.

Quigleys

TABLE 6-3 details the Quigleys exploration database as of the time of this report. FIGURE 6-1 also shows the drillhole locations for the Quigleys Deposit.



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TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	**Vista Gold Corp.**	Fig6-1.dwg	**Figure 6-1**
	Project: Mt Todd 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 drillhole database in order to bias test it. A comparison of historic and recent data by Snowden suggested that a bias might exist. Further study concluded that a bias is not apparent where all drilling is oriented in a similar direction (and not clustered). This suggests the inclusion of assay data from all phases of drilling is reasonable. The March 2008 report entitled “Mt Todd Gold Project, Gold Resource Update” contains additional information regarding the Snowden findings.

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 5% to 6% higher than RVC holes; for that reason, MRT elected to exclude RVC holes from the drillhole database for grade estimation of the Central area of the Batman Deposit.

Since the property is currently not operating, Tt 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 Tt’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 4% 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% 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 1% (on average) from the primary and repeat assays. Agreement with the primary assay was within 1% 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.

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 meters and the maximum sample length is 5 meters. 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.

Check Assays

Extensive check assaying was carried out on the exploration data. Approximately 5% of all RVC rejects were sent as duplicates and duplicate pulps were analyzed for 2.5% of all DDH intervals. Duplicate halves of 130 core intervals were analyzed as well. Overall, Mt Todd's check assay work is systematic and acceptable. The feasibility study showed that the precision of field duplicates of RVC samples is poor and that high errors exist in the database. The 1995 feasibility study stressed that because of the problems with the RVC assays, the RVC and OP assays should be kept in a separate database from the DDH assays. However, since that time, the majority of the identified assaying issues have been corrected by GGC based on recommendations of consultants. It is Tt'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.

Security

Tt 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 P80 of 2.6 mm. The primary crusher was a gyratory followed by secondary cone crushers in closed circuit. Barmacs 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 P80 of 150 microns.*
>
> ***Flotation:*** *Cyclone overflow was sent to the flotation circuit where a bulk concentrate was supposed to recover 7% of the feed with 65% to 70% of the gold.*
>
> ***CIL of Tailing:*** *The flotation tailing was leached in carbon-in-leach circuit. The leach residue was sent to the tailings pond. Approximately 60% of the gold in the flotation tailings was supposed to be recovered in the CIL circuit.*

CIL of Flotation Concentrate: *The flotation concentrate was reground in Tower mills to 15 microns and subjected to cyanide leaching to recover the bulk of the gold in this product (94.5% of the flotation concentrate). The leach residue was sent to the tailings pond.*

Process Recycle: *The process water was recycled to the milling circuit from the tailings pond. The overall gold recovery was projected to be 83.8% for the proposed circuit. However, during the initial phase of plant optimization, problems were encountered with high levels of cyanide in the recycled process water which, when returned to the mill, caused depression of pyrite and much lower recoveries to the flotation concentrate. As a result, the flotation plant was shut down and the ground ore was directly sent to the CIL circuit. The modified process flowsheet is given in FIGURE 6-3. Without the flotation circuit, the CIL plant recovered 72% to 75% of the gold.*

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

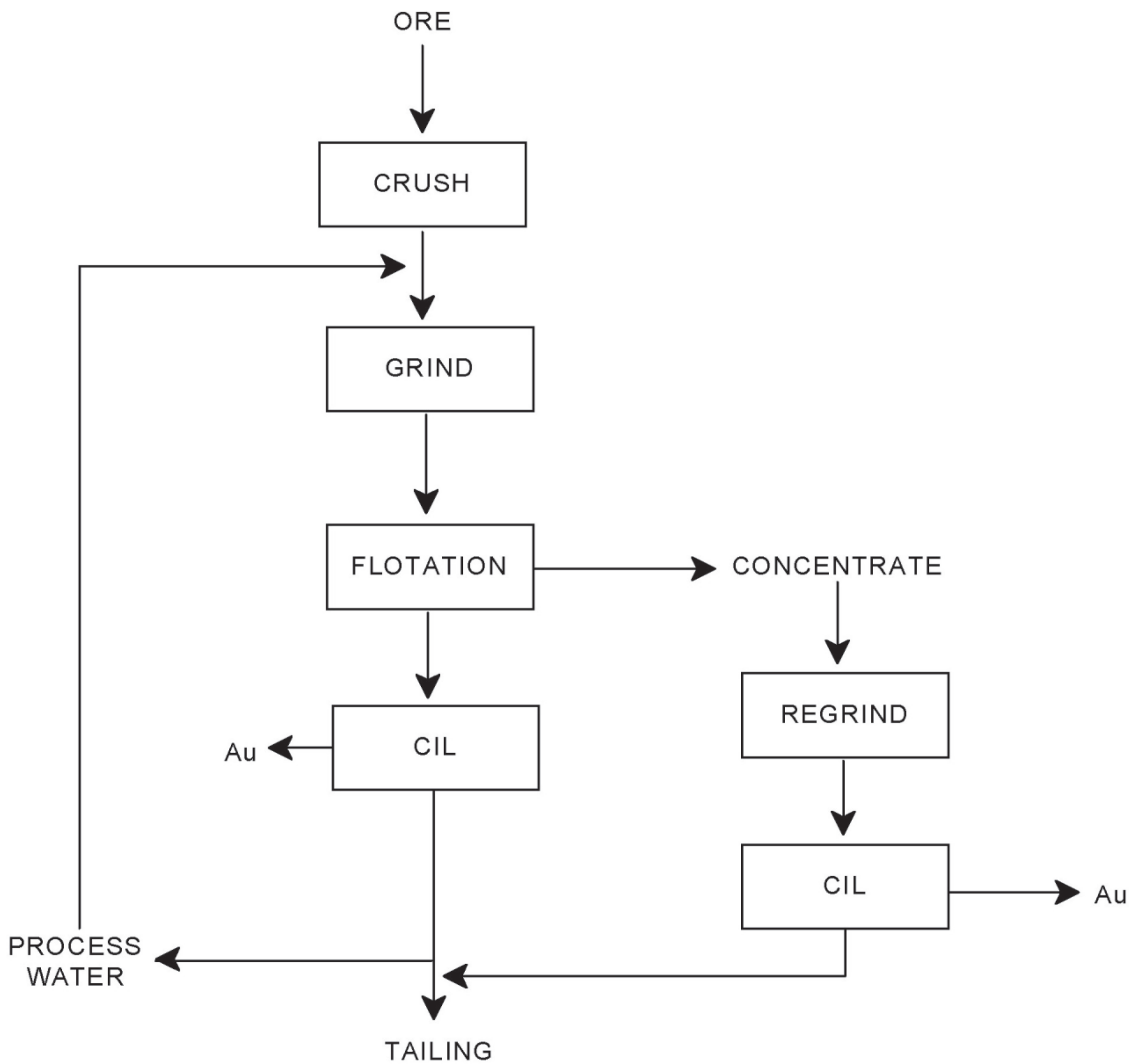
6.6 Technical Problems with Historical Process Flowsheet

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.

Crushing

The four-stage crushing circuit was supposed to produce a product with P80 of 2.6 mm. Also, the tonnage was projected to be 8 million tonnes per annum. The actual product achieved in the plant had a P80 of 3.2 to 3.5 mm and the circuit could handle a maximum of 7 million tonnes per annum. 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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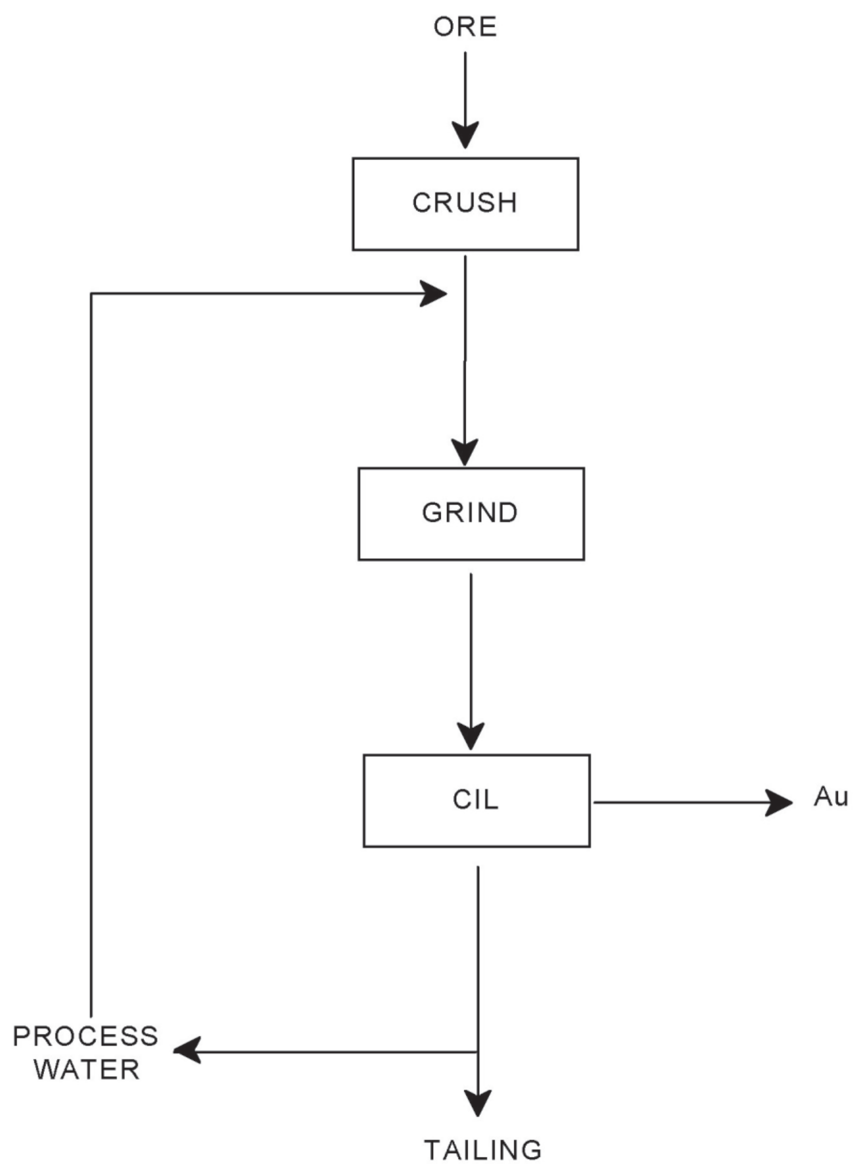
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Vista Gold Corp.***	Fig6-2.dwg	**Figure 6-2** **Plant Process Flowsheet for Mt. Todd Project (as designed)**
	Project: Mt. Todd Gold Project	Project Number: 114-310829	
	Project Location: Northern Territory, Australia	Date of Issue: 03/05/2008	



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Prepared for:	File Name:
Vista Gold Corp.	Fig6-3.dwg
Project: Mt. Todd Gold Project	Project Number: 114-310829
Project Location: Northern Territory, Australia	Date of Issue: 03/05/2008

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

The following problems were encountered with the crushing circuit:

- The mechanical availability of the Barmac 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.

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.

Flotation Circuit

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

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

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

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% 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.

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

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

0 100 200 300 400 500 km

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TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Vista Gold Corp.*	Fig7-1.cdr	**Figure 7-1 General Geologic Map**
	Project: Mt. Todd PreFeasibility Study	Project Number: 114-310912a	
	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

Local Mineralization Controls

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

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

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

North-South Trending Corridor

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

Core Complex

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

Hanging Wall Zone

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

Footwall Zone

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

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

Bedding Parallel Mineralization

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

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 drillholes, interpretation in RC drilling, and in particular later interpretation from previously omitted RC holes, must invoke a degree of uncertainty in the interpretation.

The conclusion was that, while the shear zone was identifiable on a broad scale, the local variation was difficult to map with confidence and therefore difficult to estimate with any degree of certainty at this time.

10.0 EXPLORATION

Vista exploration staff conducted a surface exploration program, including prospecting, rock sampling and GPS surveying of drillhole 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 (see table below). 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	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 6 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, drillhole collar locations and grid pickets were surveyed at Tablelands prospects using a GPS unit. Accurate drillhole locations has enabled the compilation of an accurate database for further drill planning and geological interpretation.

RESULTS

Approximately 1100 meters 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 meters with 41 samples collected. Of the samples collected almost half (46%) 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 drillholes are collared 500 meters west and 200 meters 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.

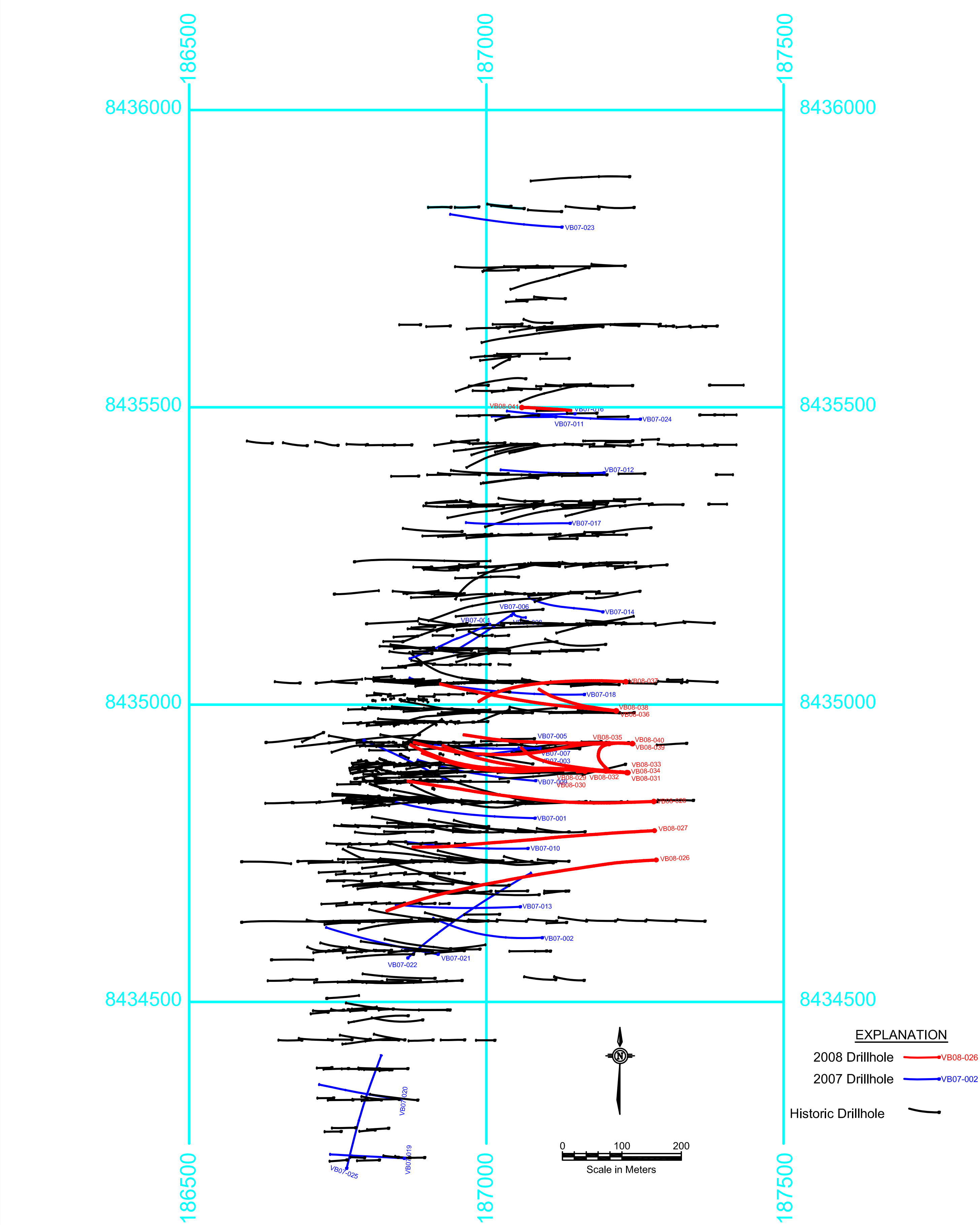
11.0 DRILLING

The 2008 Vista exploration program at the Batman deposit consisted of 16 diamond core drillholes containing some 9,037.4 meters that targeted both infill definitional drilling and stepout drilling. TABLE 11-1 contains information of the 16 drillholes 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 drillholes completed as part of the 2008 exploration program.



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Prepared for: *Vista Gold Corp.*	**File Name:** Fig11-1.dwg
Project: Mt Todd PreFeasibility Study	**Project Number:** 114-310912a
Project Location: Northern Territory, Australia	**Date of Issue:** Sept/2010

Figure 11-1
Drillhole Location Map
Batman Deposit

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 Tt 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 drillhole 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 Tt’s resource estimation, it is Tt’s opinion that the databases and associated data are of a “high quality” in nature.

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

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 drillhole collar coordinates agree well with the topographic map; it is Tt’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

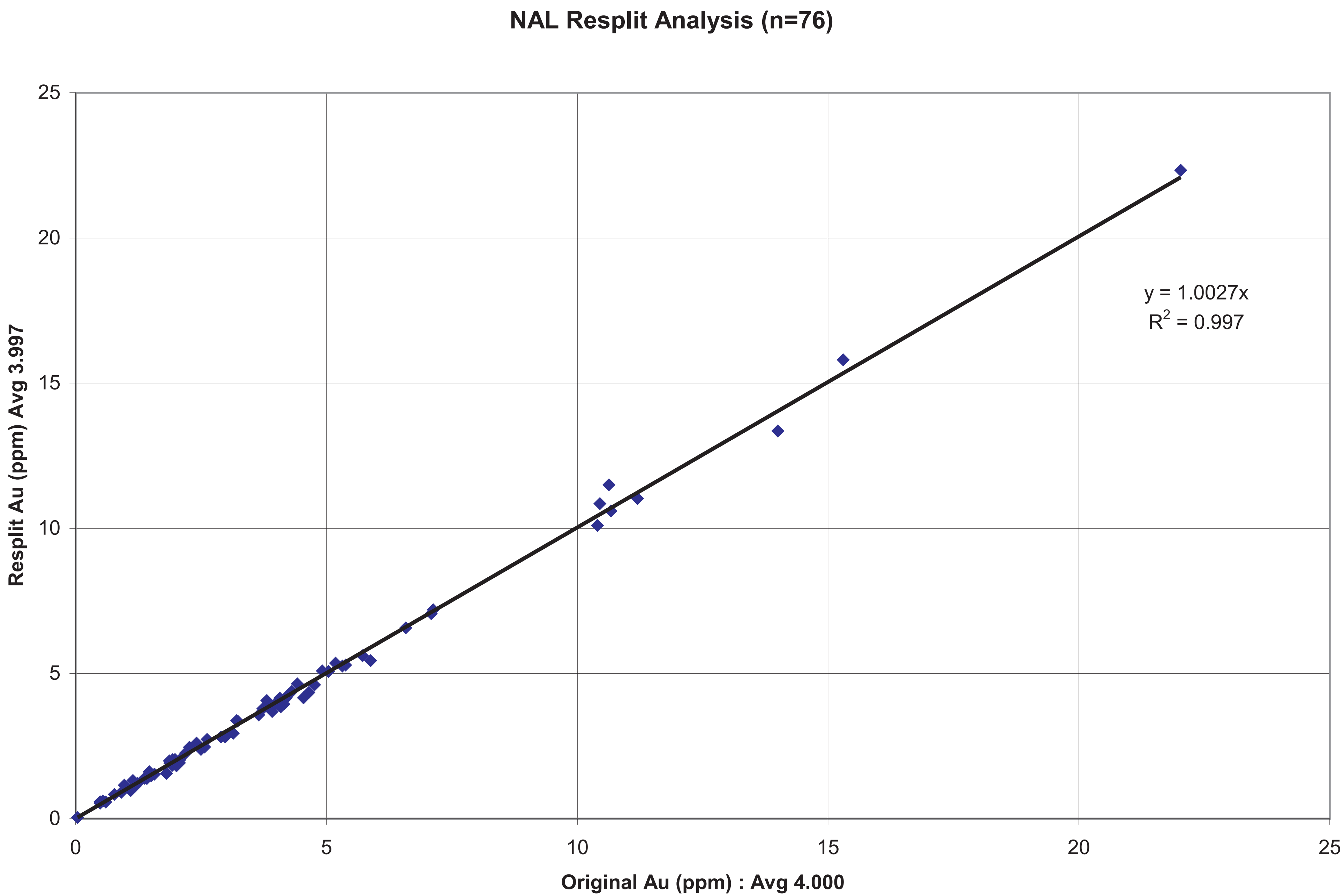
As part of the 2007 exploration program, Vista embarked on a program to both verify the historic assay results and ensure that any future analytical work meets all current NI 43-101 standards for reporting of mineral resources. This program consisted of two components; re-assaying of a portion of the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

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 14-1 detail the results of the analytical check program that was completed on the 2007 exploration drillholes. The program was designed to check both internal laboratory accuracy and inter-laboratory accuracy. NAL was the primary laboratory for completion of the sample analyses. ALS Chemex in Sydney, Australia performed the inter-laboratory analyses. As can be seen from the plats, the correlation coefficient for was 99.7% for the resplits of original assays, 99.2% for pulp repeats, and 98.6% for inter-laboratory analyses, respectively.

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



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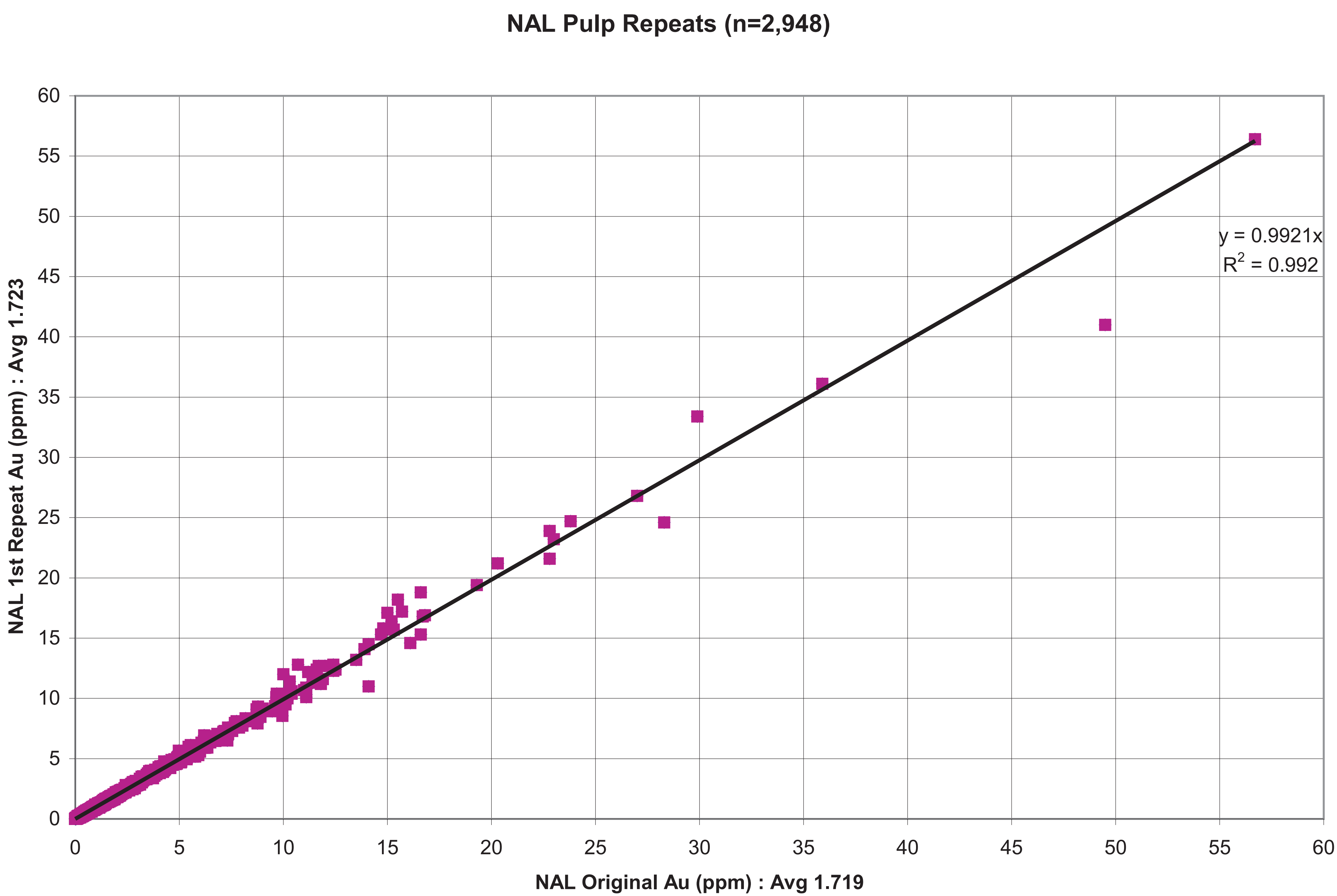
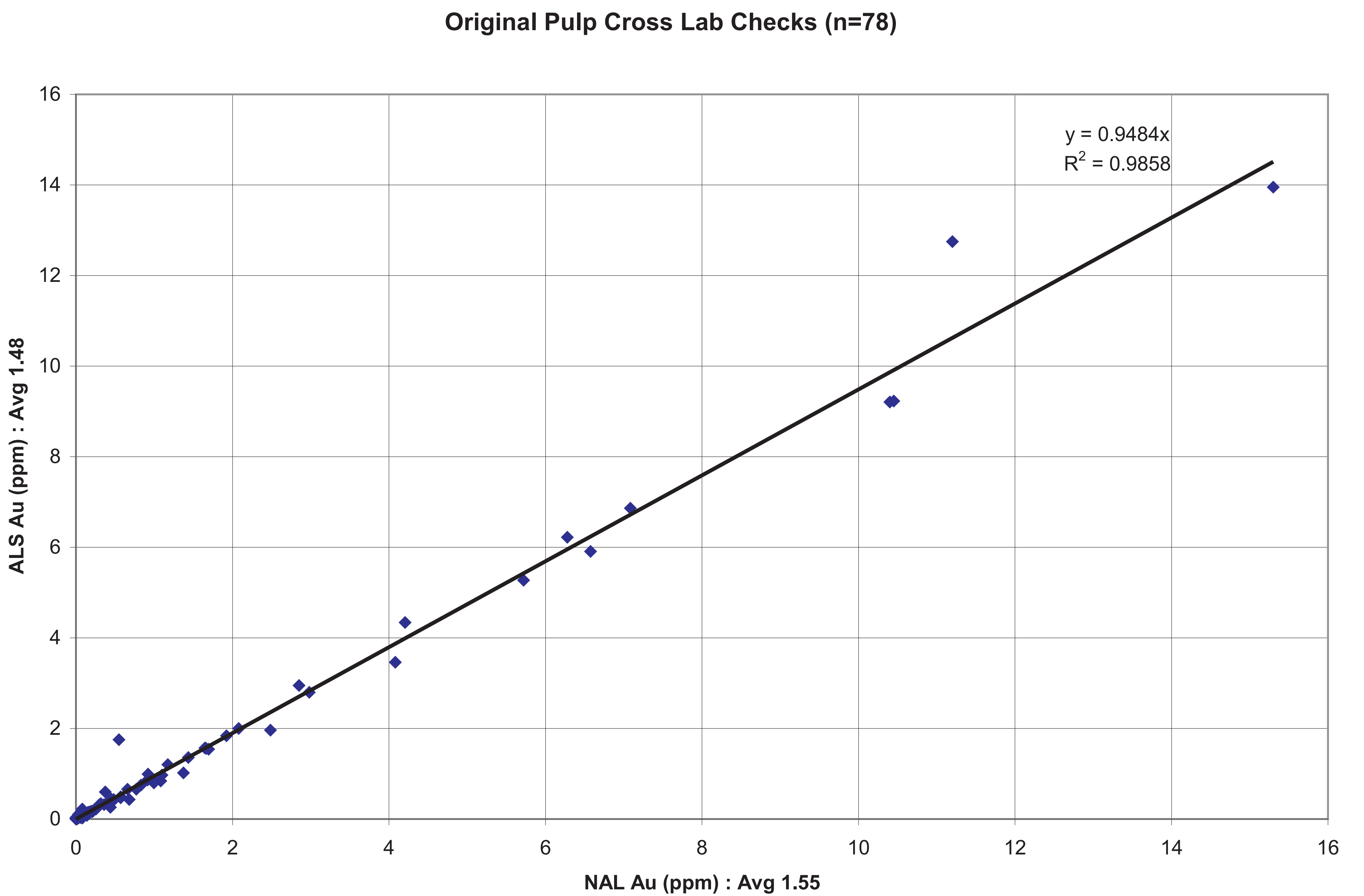
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Vista Gold Corp.***	Fig14-1.dwg	**Figure 14-1** **NAL Resplit Analysis (n=76)**
	Project: Mt. Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date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



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: Fig14-2.cdr	**Figure 14-2 NAL Pulp Repeats (n=2,948)**
	Project: Mt. Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: Sept/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: 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.

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.

Horseshoe - EL 9735

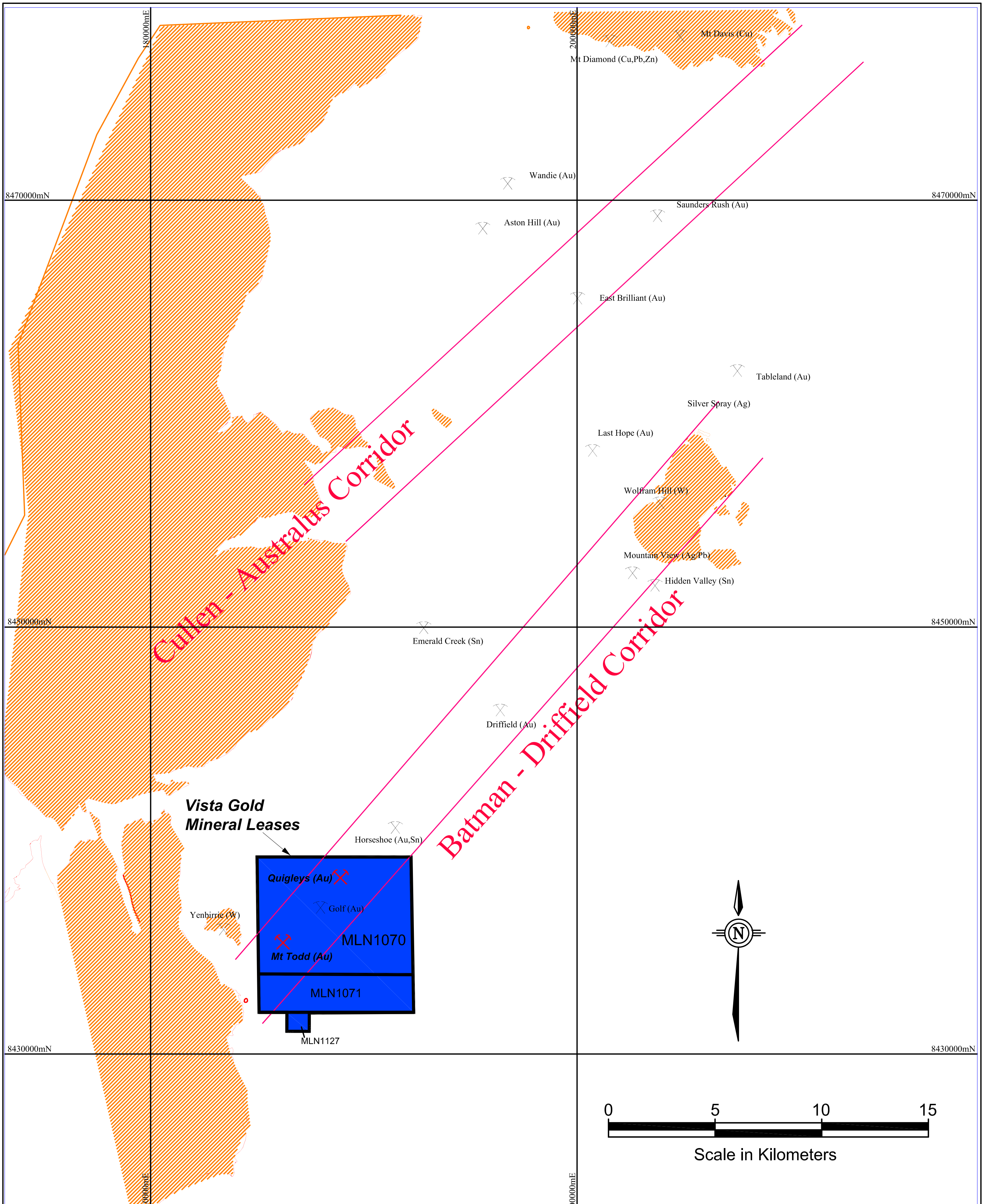
This area was previously held as EL 7635 and Mineral Claims N1918 to N1923 and N3676 to N3683 (inclusive). Billiton work defined two significant gold anomalies: Central, at the northern end, now held under BJV tenement SEL9679, and Horseshoe at the south. At Central the best RC drill result was 9 m @ 4.2 g Au/t Au 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.

Driffield-EL 9734

Previous mining at Driffield produced about 5,300 oz 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 drillholes 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.

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



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TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Vista Gold Corp.*	Fig15-1.dwg	**Figure 15-1** **Structural Trends with Mines & Prospects**
	Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: Sept/2010	

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.

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 Resource Development Inc., (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. Test work has also been undertaken at several other testing facilities including; Krupp Polsius Research Center Germany, JK Tech Pty. Ltd. Australia, Pocock Industrial, Inc. Utah, and Kappas, Cassidy and Associates Nevada. The extensive metallurgical test work 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.
- 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 improves 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% or more of the gold content in the ore would be reground and selectively floated to recover copper and gold in a cleaner concentrate which would assay over 20% Cu. The concentrate would contain approximately 50% of the gold and would be sold to a smelter. Cleaner tailings would be cyanide leached in the CIL circuit. Leach residue would be subjected to cyanide destruction and the sulfides would be sent to a separate tailings pond. The tailings pond would be constantly monitored to ensure that acid is not generated.

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

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% S_{Total}. The metallurgical testwork indicated that gold recovery into the rougher flotation concentrate was ± 80% at a primary grind of P_{80} of 200 mesh. Copper in the rougher concentrate could not be upgraded to provide concentrate assaying ± 20% Cu. The best results were ± 6% Cu using the same test procedure as employed for earlier core testing (2006).

Similar metallurgical results were obtained on a composite using 2008 core samples. This composite assayed 0.89 g/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.

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

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 5% of the remaining resources at the mine. Over 95% of the resources were typical of ore encountered in 2007 and 2008 drilling. Hence, copper may not be as important an issue as indicated by a review of the historical processing challenges encountered by earlier operators.

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. Power requirements dropped from 33.70 kwh/t to 18.11 kwh/t. The reduction in energy consumption was ± 25% when HPGRs were incorporated into the circuit. JK Tech Pty Ltd. conducted comminution tests on five samples of drill core from Mt. Todd Mine for Vista 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 High Pressure Grinding (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 Bateman deposit. This circuit would have 23% 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.

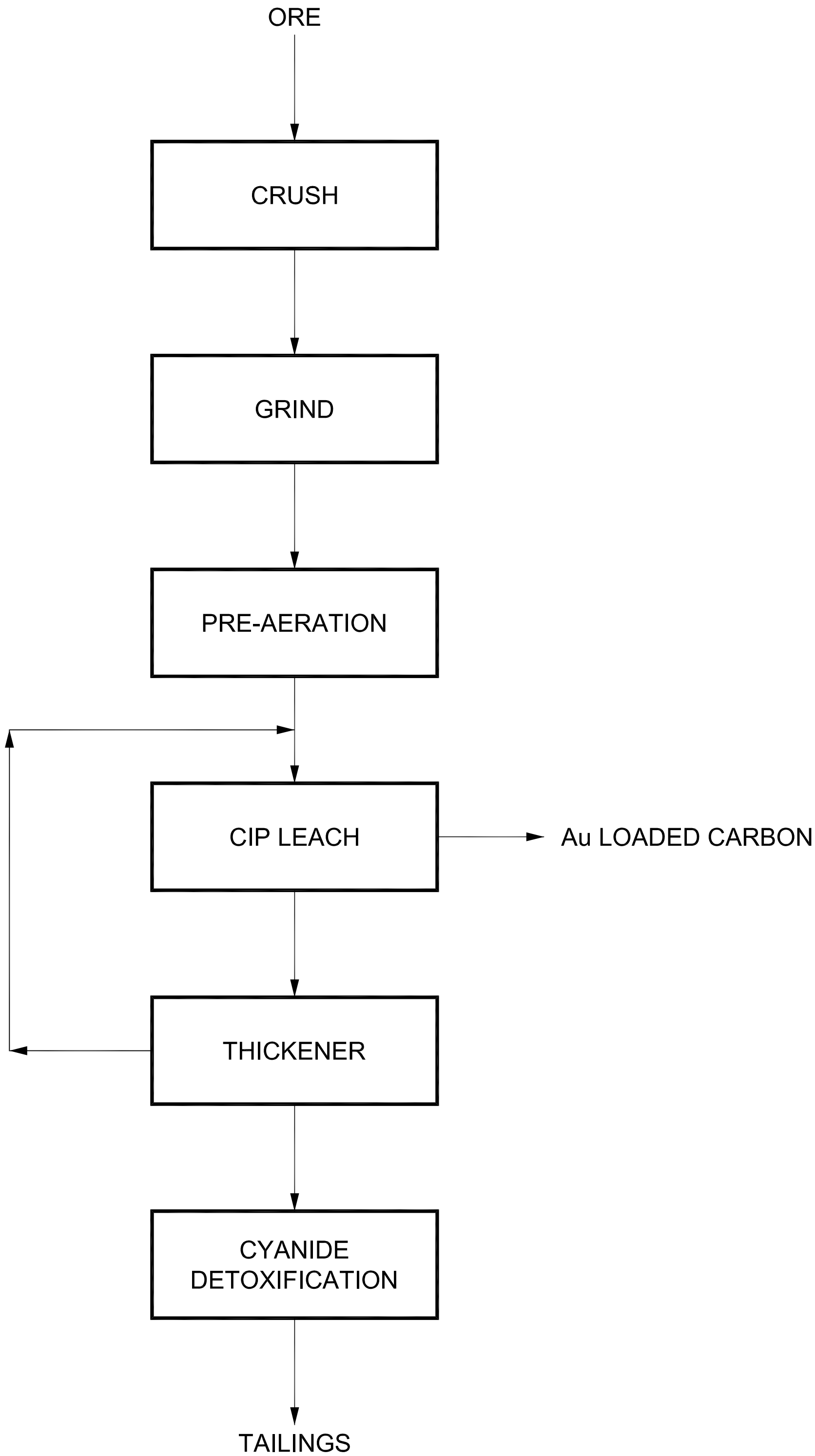
The Mt. Todd project can be expected to recover 82% of the contained gold with the proposed process flowsheet.

Resource Development Inc., (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% underflow solids using 10-15 g/mt of flocculant.
- The design basis for a high rate thickener was determined to be 7.33 m^3/m^2hr of feed loading with maximum 70% underflow solids.
- For paste thickening (74-75% solids), the recommended design basis net feed loading was determined to be 7.3 to 8.3 m^3/m^2hr.
- The horizontal belt filtration rate ranged from 65.88 to 1076 dry kg/m^2hr depending on the moisture content of the filter cake (i.e., 15% to 18%).

Kappes, Cassiday and Associates undertook limited tailing characterization testwork which included detoxification of leached tailings followed by characterization and environmental testing of the detoxified tailings[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 Mount 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 Tt'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 1% 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 Density Data

The Quigleys Deposit is approximately 3.5 kilometers northeast of the Bateman deposit. The deposit is not as deep as the Bateman Deposit. It reaches a maximum depth of approximately 200 meters. 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% shear and 85% 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 drillholes, 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 drillholes have impacted on the grade returned from the laboratory and factors to counter this have been applied in the calculation of the Au Preferred field.

MicroModel® files have been found containing 80% 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 dill 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 Tt estimates to reflect the new 2008 drillhole 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 cubic meters (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. Tt 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 Tt. 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 Tt's opinion that the GGC geologic model accurately portrays the geologic environment of the Batman Deposit.

Tt 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 Tt'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 base case cutoff for the resource reporting is 0.4 g Au/t and is bolded in the table. This cutoff value was determined according to a three-year average gold price of US$750, a three-year average exchange rate of A$1.35 = US$1.00, and accompanying parameters as presented in TABLE 1-4 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 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 meters 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 meters, 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 meters 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 meters, 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" (August 31, 2010)* **with only minor changes for consistent formatting and terminology purposes (see Appendix B for complete report).**

18.2 Base Case Pit Optimization

Pit optimization was done using Gemcom's Whittle software to define pit limits with input for economic and slope parameters. The optimization is an iterative process with initial parameters coming from the Mt. Todd PEA. The final parameters incorporate mining costs developed during this study, as well as geotechnical, metallurgical, and processing inputs provided by other consultants.

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

Varying gold prices were used to evaluate the sensitivity of the deposit to the price of gold as well as to develop a strategy for optimizing project cash flow. To achieve cash flow optimization, mining phases or push backs were developed using the guidance of Whittle pit shells at lower gold prices. For the Mt. Todd PF case, the ultimate pit was further constrained by assuming that the total tailings facility capacity is approximately 60 million tonnes. This constraint was provided by Vista and is based on land, environmental, and capital constraints.

18.2.1 Economic Parameters

Mining, processing, and general and administrative (G&A) costs were based on previous work done by Vista's staff. The Base case used an initial gold price of $900 per ounce for economic evaluation, which was determined based on averaging gold prices over the past 36 months at the time the study was started. Prior to completion of the study, the gold price was increased to $950 per ounce of gold.

TABLE 18-1: BASE CASE ECONOMIC PARAMETERS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Gold Price	$950 per Au oz – PF Case
Gold Recovery	82%
Mining Cost	$1.75 per tonne
Processing Cost	$7.80 / tonne processed
General Administration Costs	$0.84 / tonne processed

18.2.2 Slope Parameters

Slope parameters were based on studies provided by Tt (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 for the PF case were completed using prices of $400 to $850 per ounce Au in $5 per ounce Au increments, providing good incremental detail for analyzing elevated gold grade cutoffs (discussed later). An additional pit optimization run was made from $400 to $1,200 per ounce Au with increments of $25 per ounce Au in order to analyze the deposit's sensitivity to higher 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
October 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 oz (1000's)			
400	6,389	1.53	314	7,409	13,798	1.16
450	9,467	1.43	436	13,127	22,594	1.39
500	14,478	1.32	616	22,847	37,325	1.58
550	21,083	1.22	827	34,292	55,375	1.63
600	31,897	1.11	1,142	53,501	85,397	1.68
650	61,873	1.01	2,008	19,353	181,225	1.93
700	89,331	0.95	2,726	177,302	266,633	1.98
750	110,551	0.91	3,245	222,601	333,152	2.01
800	128,736	0.88	3,641	256,125	384,861	1.99
850	149,545	0.84	4,040	284,735	434,280	1.90
900	162,981	0.82	4,298	308,642	471,623	1.89
950	177,872	0.79	4,539	322,396	500,268	1.81
1,000	186,119	0.78	4,688	339,805	525,924	1.83
1,050	200,982	0.76	4,914	356,962	557,944	1.78
1,100	212,897	0.74	5,098	376,827	589,724	1.77
1,150	225,008	0.73	5,248	381,661	606,669	1.70
1,200	234,281	0.71	5,383	399,210	633,491	1.70

18.2.4 Pit-Shell Selection for Ultimate Pit Limit

The Whittle pit shells were used to determine ultimate pit limits and internal pit-phase designs. While the choice of pit shells for the Base-case ultimate pit limit was straight forward, the Base-case ultimate pit determination required additional analysis. The following subsections describe the selection of Whittle pit shells used for ultimate pit limits.

Pit-Shell Selection for the Base Case Ultimate Pit

Due to the assumed limited tailings capacity, the ultimate pit limit for the Base case was based on optimizing the discounted cash flow or net present value ("NPV") generated from mining 60 million tonnes of ore. Whittle was used to generate NPV's for each potential pit shell using the base case parameters. The pits that would mine closest to 60 million tonnes of material were selected, and straight line interpolation was done to infer an NPV for the base case parameters.

Subsequent runs in Whittle were made, elevating the minimum cutoff grade from 0.40 g Au/t to 0.75 g Au/t in increments of 0.05 g Au/t. Within each of these runs, the NPV for a 60 million tonne pit was interpolated from the pits that were closest to 60 million tonnes. These NPV values were compared with the base case value to determine a relative NPV improvement achieved by increasing the cutoff grade. As more tonnage would be required to be mined using the higher cutoff grades, additional capital was added to represent the required addition of trucks and shovels. Using this approach, pits using a 0.55 g Au/t cutoff grade were observed to increase the NPV by $48.8 million over the base case, while considering the maximum tailings capacity of 60 million tonnes. The results of this analysis are shown in TABLE 18-3.

TABLE 18-3: RELATIVE NPV IMPROVEMENTS OF PITS CONTAINING 60 MILLION PROCESSED TONNES
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Cutoff	Material Processed			Waste Tonnes	Total Tonnes	Strip Ratio	Relative NPV Improvement ($)
	K tonnes	G Au/t	K ozs				
Base	60,000	0.89	1,708	58,825	118,825	0.98	-
0.40	60,000	0.91	1,754	67,521	127,521	1.12	13.52
0.45	60,000	0.95	1,833	84,065	144,065	1.40	33.73
0.50	60,000	0.99	1,912	103,069	163,069	1.72	47.95
0.55	60,000	1.04	2,001	129,232	189,232	2.15	48.82
0.60	60,000	1.08	2,077	156,638	216,638	2.61	43.05
0.65	60,000	1.13	2,170	195,468	255,468	3.26	30.39
0.70	60,000	1.17	2,258	240,933	300,933	4.02	(3.65)
0.75	60,000	1.22	2,348	299,277	359,277	4.99	(50.12)

Using this methodology, Whittle pits relating to gold prices of $660 and $665 per ounce Au were selected as guides in the development of the ultimate pit design. Using a 0.55 g Au/t cutoff grade, the $660 per ounce Au pit contains 58.7 million tonnes of Measured and Indicated resources at an average grade of 1.04 g Au/t and contains 1.96 million ounces Au. In comparison, the $665 per ounce Au pit contains 65 million tonnes of Measured and Indicated resources at an average grade of 1.03 g Au/t and contains 2.16 million ounces Au.

Pit-Shell Selection for the Sensitivity Case Ultimate Pit

The $850 per ounce Au Whittle pit shell was used as the Sensitivity case ultimate pit limit. Larger pits were examined; however the pit shells start to encroach on the current crusher location. The $850 per ounce pit allows approximately 70 meters of room near the crusher. MDA suggests that future studies be done to evaluate the cost of relocating the crusher to allow mining of additional resources.

18.2.5 Pit Designs

Detailed pit design was completed for the Base case, including an ultimate pit and two internal pit phases (see FIGURES 18-1, 18-2, and 18-3). 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 processing plant earlier in the mine life. The pit-design parameters and resulting pit designs for the Base case are discussed in following sections.

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 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. However, this will be dependent on equipment sizing and may require additional dozers to push material to loading units.

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 meters 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 meters 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 meters in width for every 24 meters 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 18-4 shows the slope parameters used for pit design.

TABLE 18-4: PIT DESIGN SLOPE PARAMETERS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

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

Ramp width was determined as a function of the largest truck width to be used in mine planning. Mine plans use both CAT 789C and 785C trucks, which have operating widths of 7.67 and 6.64 meters, 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 and sufficient moisture. A flat top on the berms of 0.33 meters is assumed, and berm heights of 1.82 and 1.63 meters would provide half of the truck tire heights plus 10% for 789C and 785C trucks, respectively. The resulting base width of safety berms is 5.78 and 5.23 meters for 789C and 785C trucks, respectively.

Haul-road designs inside of pits where only one safety berm is required are designed to be 30 meters 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 meters. This provides 1.6 and 1.9 times the widths of 789C and 785C trucks, respectively, for running width.

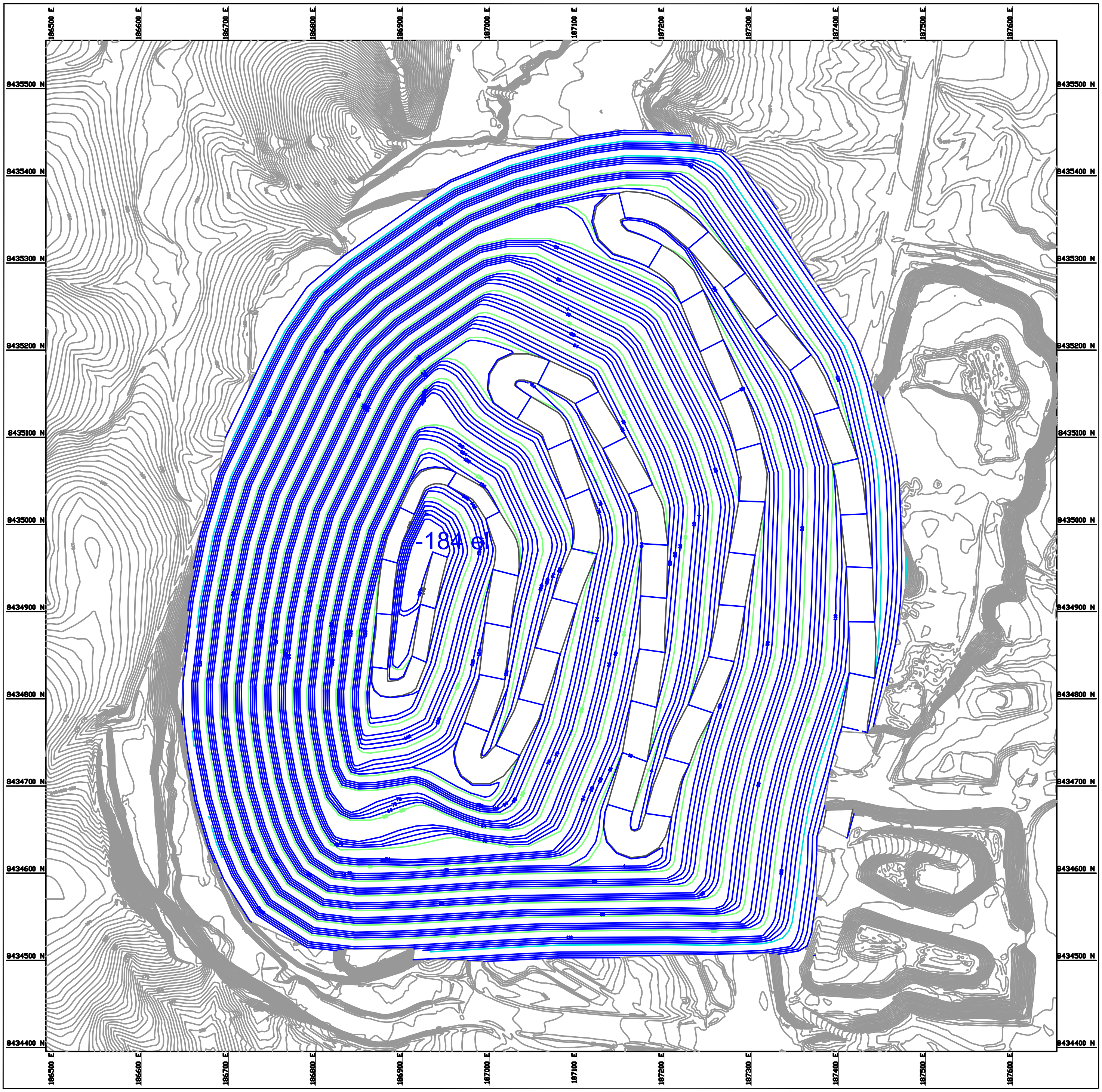
Haul roads outside of pit designs have been designed to be 38 meters to account for an additional safety berm.

18.2.9 Ultimate Pit

The ultimate pit was designed to target 60 million tonnes of Measured and Indicated resources above a 0.55 g Au/t cutoff grade due to the constraint of tailings capacity. As discussed in previous sections, the $650 and $660 per ounce Au pits were used for guidance when designing the ultimate pit. This became an iterative process, and as the ultimate pit exceeded the 60 million tonne target by approximately 2 million tonnes, cutoff grades were adjusted in select benches to achieve the overall capacity.

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 and helps to follow the dip of the deposit on the west side of the pit. In all, there are four switchbacks in the ultimate pit design.

FIGURE 18-1 shows the Base case ultimate pit design, and resulting reserves are shown in TABLE 18-6.



(from MDA 2010)

MINE DEVELOPMENT ASSOCIATES

Vista Gold Corp
Mt. Todd
Prefeasibility Case: Ultimate Pit

23-Jul-2010

Scale: as shown

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.*	Fig18-1.dwg	**Figure 18-1**
	Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a	**PreFeasibility (6.77 mtpy) Case**
	Project Location: Northern Territory, Australia	Date of Issue: 09/20/2010	**Ultimate Pit**

18.2.10 Pit Phasing

Pit phases were created to improve the project NPV by mining higher-value material in the initial years. Three criteria were used to establish the best pit-phasing strategy. First, the Whittle analysis tools were used to determine the NPV of mining individual Whittle pits as separate phases and comparing them with the NPV of mining the ultimate pit as a single pit phase. This showed that the NPV improvement was marginal up through the $600 per ounce Au pit.

Secondly, total tonnes of available resource were compared in each of the pits, with the goal of finding incremental pits that approximate one third of the ultimate pit reserves. This would allow for mining three phases that would still maintain large enough areas for safe operation of equipment.

The third criterion was to ensure that the first phase would have sufficient grade to create a better payback on capital used in developing the project.

Based on the criteria above, the $500 per ounce Au pit was chosen for a guide to design phase 1, and the $605 per ounce Au pit was used to guide the design of phase 2. Phase 3 was defined by the remaining volume to achieve the ultimate pit.

FIGURE 18-2 and FIGURE 18-3 show phase 1 and 2 pit designs, respectively. Resulting reserves for each of the phases are shown in TABLE 18-6. Bench reserves for phase 1, phase 2, and phase 3 are shown in TABLE 18-7. The combined ultimate pit bench reserves are shown in TABLE 18-8.

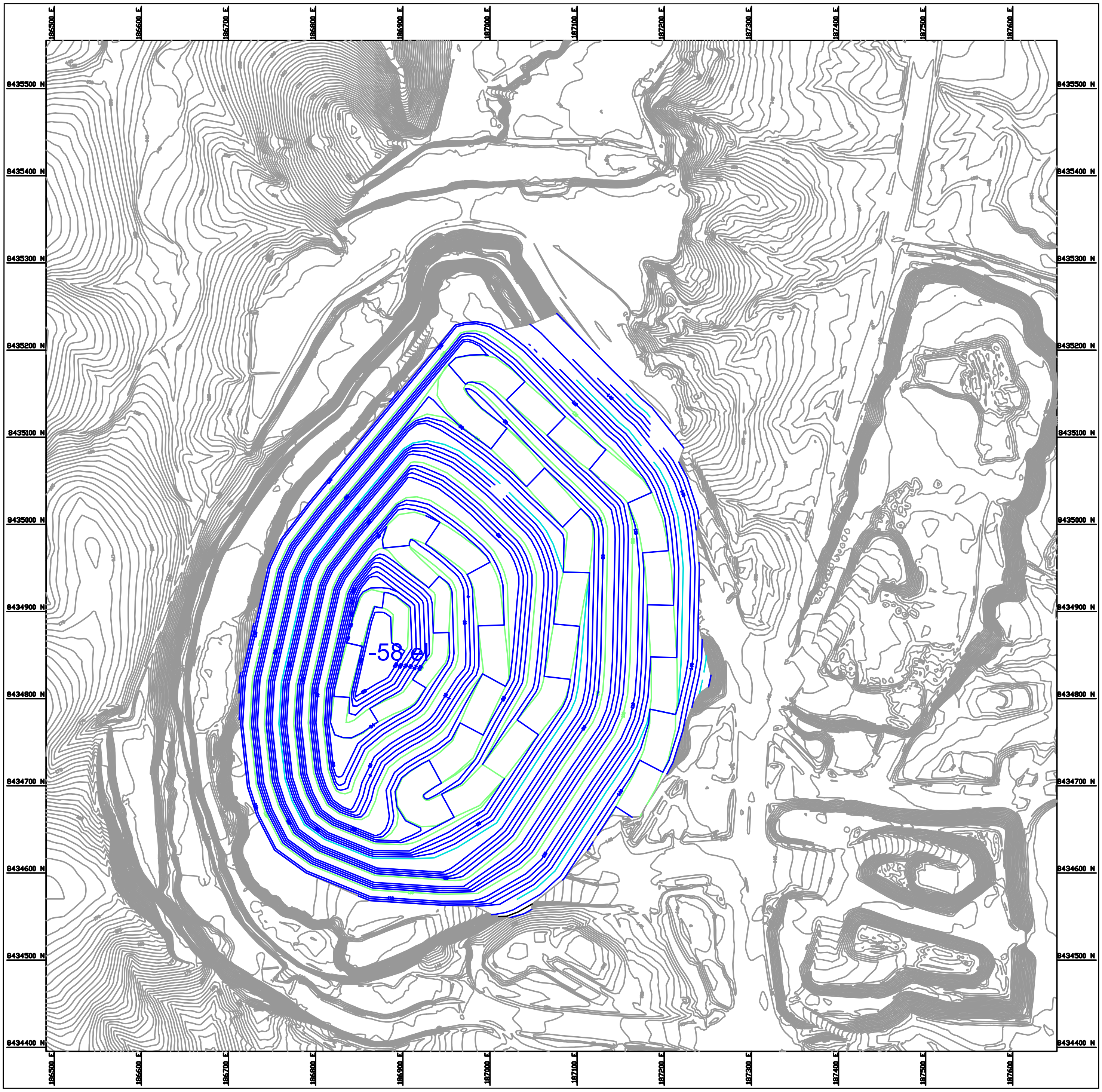
18.2.11 Cutoff Grade

Based on the economic parameters and $900 per ounce Au, the break-even cutoff grade is calculated at 0.45 g Au/t, and the internal cutoff grade is calculated at 0.37 g Au/t for the Base case. A 0.40 g Au/t cutoff was used to define resources inside of the Sensitivity case (10.69 Mtpy) pit designs as this is the cutoff used in the Mt. Todd PEA to define resources.

Due to tailings constraint and the discussion in the pit optimization section, the Base case cutoff grade is variable, with the lower-end optimal cutoff grade near 0.55 g Au/t. Once pit designs were completed, the cutoff grade was adjusted on a bench-by-bench basis with the goal of elevating the cutoff in the earlier years of mining. The actual cutoffs used to define reserves are shown in TABLE 18-5.

TABLE 18-5: RESERVE CUTOFF GRADES
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Phase	Bench		g Au/t Cutoff
	From	To	
1	158	110	0.55
	104	-58	0.60
2	158	14	0.60
	8	-118	0.55
3	170	-184	0.55



0 25 50 100
scale is in meters

(from MDA 2010)

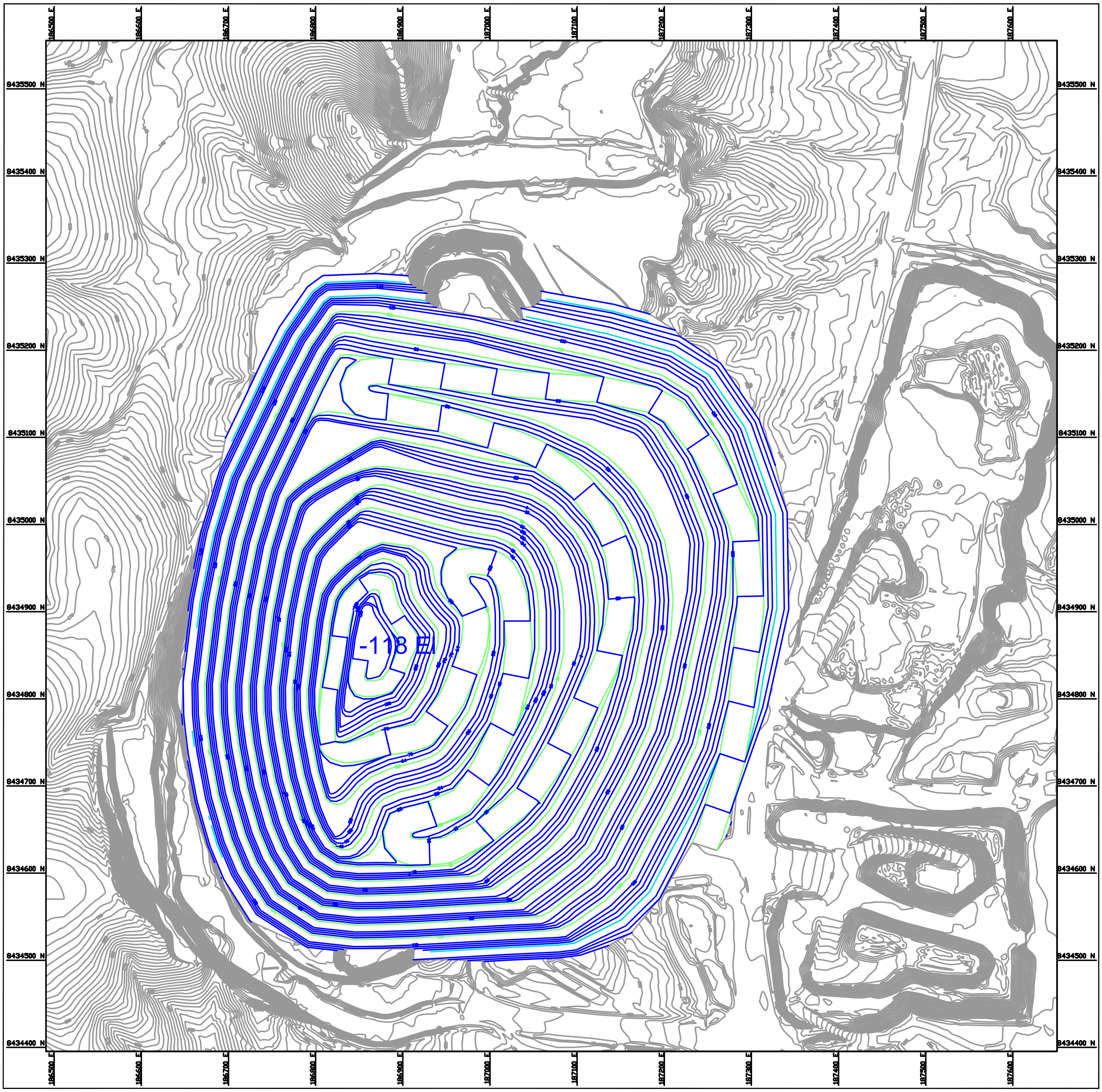
MINE DEVELOPMENT ASSOCIATES

Vista Gold Corp.
Mt. Todd
Prefeasibility Case: Phase 1 Pit

23-Jul-2010

Scale: as shown

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.***	Fig18-2.dwg	**Figure 18-2**
	Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a	**PreFeasibility (6.77 mtpy) Case**
	Project Location: Northern Territory, Australia	Date of Issue: 09/20/2010	**Phase 1 Pit**



0 25 50 100
scale is in meters

(from MDA 2010)

MINE DEVELOPMENT ASSOCIATES

Vista Gold Corp.
Mt. Todd
Prefeasibility Case: Phase 2 Pit

23-Jul-2010

Scale: as shown

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** **PreFeasibility (6.77 mtpy) Case** **Phase 2 Pit**
	Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: 09/20/2010	

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 Base 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 Sensitivity case (10.6 Mtpy) economics and pit design have not met the criteria to determine reserves. Thus, there are no reserves established for Mt. Todd beyond the Base case, and only resources are reported for the Sensitivity-case. The following sections discuss the reserves and resources reported for the Base case.

18.3.1 Base 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 18-6 reports the Proven and Probable reserves based on the Base case. 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 18-7 The total Proven and Probable reserves by bench are shown in. Due to rounding issues in reporting, these do not add up exactly to the reserves reported in TABLE 18-6; however the differences are inconsequential.

TABLE 18-6: PROVEN AND PROBOBLE RESERVES BY PHASE *
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Phase	Proven Reserves			Probably Reserves			Proven and Probable			Waste K Tonnes	Total K Tonnes	Strip Ratio
	K Tonnes	g Au/t	K oz Au	K Tonnes	g Au/t	K oz Au	K Tonnes	g Au/t	K oz Au			
1	8,655	1.27	354	6,039	1.19	231	14,694	1.24	585	19,331	34,025	1.32
2	5,404	1.01	176	10,676	1.01	347	16,080	1.01	523	49,687	65,767	3.09
3	10,399	0.97	824	18,877	0.98	595	29,276	0.98	919	73,506	102,782	2.51
Total	24,458	1.09	854	35,592	1.02	1,173	60,050	1.05	2,027	142,524	202,574	2.37

* Reserves use variable cutoff grades as described in TABLE 18-5

TABLE 18-7: PROVEN AND PROBABLE BENCH RESERVES BY PHASE *
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Phase 1 Proven and Probable Bench Reserves

Cutoff	Toe Elev	Total Ore			Waste	Total	Strip
g Au/t	Meters	K Tonnes	g Au/t	K Oz Au	K Tonnes	K Tonnes	Ratio
	170	-	-	-	-	-	
	164	-	-	-	-	-	
0.55	158	-	-	-	17	17	NA
0.55	152	-	-	-	137	137	NA
0.55	146	2	0.97	0	279	281	139.40
0.55	140	3	0.74	0	465	468	179.10
0.55	134	11	0.64	0	579	591	50.80
0.55	128	5	0.68	0	599	604	110.64
0.55	122	8	0.64	0	752	760	90.18
0.55	116	42	0.81	1	1,415	1,457	33.95
0.55	110	466	0.87	13	1,340	1,806	2.88
0.60	104	417	0.93	13	1,164	1,581	2.79
0.60	98	488	0.95	15	1,143	1,631	2.34
0.60	92	566	0.94	17	1,126	1,693	1.99
0.60	86	609	0.96	19	1,081	1,690	1.77
0.60	80	583	0.98	18	893	1,476	1.53
0.60	74	550	1.05	19	911	1,461	1.65
0.60	68	588	1.07	20	887	1,475	1.51
0.60	62	576	1.13	21	882	1,457	1.53
0.60	56	506	1.22	20	724	1,230	1.43
0.60	50	505	1.26	20	691	1,196	1.37
0.60	44	547	1.26	22	854	1,401	1.56
0.60	38	846	1.29	35	797	1,643	0.94
0.60	32	835	1.28	34	606	1,442	0.73
0.60	26	823	1.29	34	513	1,335	0.62
0.60	20	801	1.30	33	410	1,211	0.51
0.60	14	774	1.28	32	342	1,116	0.44
0.60	8	667	1.31	28	203	870	0.30
0.60	2	618	1.33	26	166	784	0.27
0.60	-4	565	1.38	25	130	696	0.23
0.60	-10	507	1.40	23	106	613	0.21
0.60	-16	382	1.54	19	58	440	0.15
0.60	-22	341	1.60	18	39	380	0.11
0.60	-28	308	1.60	16	13	321	0.04
0.60	-34	248	1.69	13	10	258	0.04
0.60	-40	193	1.68	10	5	199	0.03
0.60	-46	147	1.79	8	5	152	0.04
0.60	-52	103	1.91	6	1	104	0.01
0.60	-58	63	1.91	4	-	63	-
	-64	-	-	-	-	-	
	-70	-	-	-	-	-	
	-76	-	-	-	-	-	
	-82	-	-	-	-	-	
	-88	-	-	-	-	-	
	-94	-	-	-	-	-	
	-100	-	-	-	-	-	
	-106	-	-	-	-	-	
	-112	-	-	-	-	-	
	-118	-	-	-	-	-	
	-124	-	-	-	-	-	
	-130	-	-	-	-	-	
	-136	-	-	-	-	-	
	-142	-	-	-	-	-	
	-148	-	-	-	-	-	
	-154	-	-	-	-	-	
	-160	-	-	-	-	-	
	-166	-	-	-	-	-	
	-172	-	-	-	-	-	
	-178	-	-	-	-	-	
	-184	-	-	-	-	-	
	Total	14,694	1.24	585	19,342	34,036	1.32

Phase 2 Proven and Probable Bench Reserves

Cutoff	Toe Elev	Total Ore			Waste	Total	Strip
g Au/t	Meters	K Tonnes	g Au/t	K Oz Au	K Tonnes	K Tonnes	Ratio
	170	-	-	-	-	-	
	164	-	-	-	-	-	
0.60	158	-	-	-	176	176	NA
0.60	152	15	1.01	0	462	477	31.61
0.60	146	32	0.96	1	838	871	25.83
0.60	140	35	0.94	1	1,496	1,531	42.56
0.60	134	48	0.84	1	1,664	1,712	34.54
0.60	128	53	0.82	1	1,724	1,777	32.61
0.60	122	104	0.78	3	2,117	2,221	20.37
0.60	116	112	0.76	3	2,331	2,444	20.74
0.60	110	188	0.80	5	2,348	2,536	12.51
0.60	104	209	0.79	5	2,235	2,443	10.72
0.60	98	208	0.77	5	2,207	2,415	10.61
0.60	92	201	0.75	5	2,183	2,383	10.88
0.60	86	181	0.79	5	2,175	2,357	11.99
0.60	80	239	0.82	6	2,042	2,281	8.55
0.60	74	234	0.86	6	2,044	2,278	8.72
0.60	68	251	0.85	7	1,994	2,245	7.95
0.60	62	240	0.82	6	1,974	2,214	8.21
0.60	56	321	0.83	9	1,708	2,029	5.33
0.60	50	314	0.84	8	1,642	1,956	5.23
0.60	44	312	0.80	8	1,615	1,927	5.18
0.60	38	335	0.84	9	1,564	1,898	4.67
0.60	32	348	0.85	10	1,398	1,746	4.02
0.60	26	331	0.86	9	1,397	1,728	4.22
0.60	20	344	0.89	10	1,385	1,729	4.03
0.60	14	369	0.87	10	1,335	1,703	3.62
0.55	8	545	0.87	15	1,084	1,630	1.99
0.55	2	607	0.88	17	997	1,604	1.64
0.55	-4	653	0.88	18	928	1,581	1.42
0.55	-10	684	0.90	20	869	1,553	1.27
0.55	-16	708	0.95	22	698	1,406	0.99
0.55	-22	747	0.97	23	604	1,351	0.81
0.55	-28	710	1.01	23	566	1,276	0.80
0.55	-34	743	1.02	24	489	1,232	0.66
0.55	-40	693	1.06	24	357	1,049	0.52
0.55	-46	688	1.12	25	310	997	0.45
0.55	-52	698	1.13	25	249	947	0.36
0.55	-58	638	1.22	25	242	880	0.38
0.55	-64	561	1.31	24	115	676	0.21
0.55	-70	505	1.33	22	93	597	0.18
0.55	-76	450	1.38	20	70	520	0.16
0.55	-82	389	1.43	18	44	433	0.11
0.55	-88	291	1.38	13	14	305	0.05
0.55	-94	232	1.35	10	22	254	0.09
0.55	-100	201	1.32	8	3	204	0.02
0.55	-106	150	1.33	6	2	152	0.01
0.55	-112	107	1.28	4	2	109	0.01
0.55	-118	58	1.17	2	1	58	0.01
	-124	-	-	-	-	-	
	-130	-	-	-	-	-	
	-136	-	-	-	-	-	
	-142	-	-	-	-	-	
	-148	-	-	-	-	-	
	-154	-	-	-	-	-	
	-160	-	-	-	-	-	
	-166	-	-	-	-	-	
	-172	-	-	-	-	-	
	-178	-	-	-	-	-	
	-184	-	-	-	-	-	
	Total	16,080	1.01	523	49,811	65,890	3.10

Phase 3 Proven and Probable Bench Reserves

Cutoff	Toe Elev	Total Ore			Waste	Total	Strip
g Au/t	Meters	K Tonnes	g Au/t	K Oz Au	K Tonnes	K Tonnes	Ratio
0.55	170	-	-	-	6	6	NA
0.55	164	13	0.80	0	132	145	10.21
0.55	158	54	0.78	1	444	498	8.15
0.55	152	183	0.76	4	650	833	3.55
0.55	146	181	0.75	4	1,147	1,327	6.34
0.55	140	158	0.72	4	2,013	2,171	12.73
0.55	134	184	0.72	4	2,392	2,576	13.00
0.55	128	173	0.71	4	2,323	2,496	13.39
0.55	122	191	0.73	4	2,502	2,693	13.11
0.55	116	209	0.72	5	2,504	2,713	11.97
0.55	110	294	0.73	7	2,510	2,804	8.53
0.55	104	281	0.73	7	2,457	2,738	8.76
0.55	98	256	0.74	6	2,475	2,732	9.66
0.55	92	267	0.73	6	2,454	2,721	9.20
0.55	86	252	0.73	6	2,458	2,710	9.75
0.55	80	281	0.73	7	2,356	2,637	8.37
0.55	74	289	0.73	7	2,318	2,608	8.01
0.55	68	236	0.75	6	2,344	2,580	9.94
0.55	62	261	0.75	6	2,266	2,527	8.67
0.55	56	251	0.76	6	2,254	2,505	9.00
0.55	50	244	0.78	6	2,283	2,527	9.35
0.55	44	293	0.77	7	2,216	2,509	7.58
0.55	38	344	0.75	8	2,142	2,486	6.22
0.55	32	358	0.76	9	2,084	2,442	5.82
0.55	26	414	0.76	10	2,005	2,419	4.84
0.55	20	447	0.77	11	1,948	2,395	4.35
0.55	14	476	0.78	12	1,893	2,369	3.98
0.55	8	532	0.78	13	1,748	2,281	3.28
0.55	2	601	0.79	15	1,648	2,249	2.74
0.55	-4	624	0.79	16	1,586	2,210	2.54
0.55	-10	615	0.83	16	1,524	2,139	2.48
0.55	-16	704	0.84	19	1,382	2,087	1.96
0.55	-22	719	0.84	19	1,350	2,068	1.88
0.55	-28	756	0.85	21	1,314	2,070	1.74
0.55	-34	823	0.86	23	1,223	2,046	1.49
0.55	-40	847	0.89	24	1,065	1,912	1.26
0.55	-46	852	0.92	25	1,029	1,881	1.21
0.55	-52	860	0.94	26	994	1,853	1.16
0.55	-58	901	0.95	27	936	1,837	1.04
0.55	-64	937	0.99	30	805	1,743	0.86
0.55	-70	864	1.03	29	791	1,655	0.92
0.55	-76	931	1.05	31	686	1,617	0.74
0.55	-82	958	1.08	33	631	1,590	0.66
0.55	-88	949	1.10	33	478	1,427	0.50
0.55	-94	954	1.11	34	421	1,375	0.44
0.55	-100	960	1.14	35	363	1,323	0.38
0.55	-106	961	1.16	36	314	1,274	0.33
0.55	-112	857	1.18	32	196	1,054	0.23
0.55	-118	869	1.18	33	97	966	0.11
0.55	-124	853	1.18	32	81	934	0.09
0.55	-130	782	1.21	30	62	844	0.08
0.55	-136	630	1.28	26	15	645	0.02
0.55	-142	555	1.31	23	17	572	0.03
0.55	-148	481	1.31	20	16	497	0.03
0.55	-154	425	1.32	18	9	433	0.02
0.55	-160	296	1.33	13	6	302	0.02
0.55	-166	242	1.31	10	5	246	0.02
0.55	-172	169	1.27	7	1	170	0.00
0.55	-178	117	1.20	4	2	119	0.02
0.55	-184	62	1.17	2	-	62	-
	Total	29,276	0.98	918	73,371	102,647	2.51

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

TABLE 18-8: TOTAL PROVEN AND PROBABLE RESERVES *
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

	Total Proven and Probable Bench Reserves					
Toe Elevation	Total Ore			Waste	Total	Strip
m Above SL	K Tonnes	g Au/t	K Oz Au	K Tonnes	K Tonnes	Ratio
170	-	-	-	6	6	
164	13	0.80	0	132	145	10.21
158	54	0.78	1	637	691	11.69
152	197	0.78	5	1,248	1,446	6.32
146	215	0.78	5	2,264	2,479	10.51
140	196	0.76	5	3,974	4,169	20.29
134	244	0.74	6	4,635	4,878	19.03
128	232	0.73	5	4,646	4,877	20.05
122	303	0.74	7	5,371	5,674	17.72
116	363	0.74	9	6,251	6,614	17.21
110	948	0.81	25	6,198	7,145	6.54
104	907	0.84	24	5,856	6,762	6.46
98	953	0.85	26	5,825	6,778	6.11
92	1,034	0.85	28	5,763	6,797	5.58
86	1,043	0.87	29	5,714	6,756	5.48
80	1,104	0.88	31	5,290	6,394	4.79
74	1,074	0.92	32	5,273	6,347	4.91
68	1,075	0.95	33	5,224	6,299	4.86
62	1,077	0.97	33	5,122	6,199	4.75
56	1,077	1.00	35	4,686	5,763	4.35
50	1,063	1.03	35	4,616	5,679	4.34
44	1,151	1.01	38	4,685	5,836	4.07
38	1,525	1.07	52	4,502	6,027	2.95
32	1,541	1.06	53	4,089	5,630	2.65
26	1,568	1.06	53	3,915	5,482	2.50
20	1,592	1.06	54	3,743	5,335	2.35
14	1,619	1.04	54	3,570	5,188	2.21
8	1,745	1.01	57	3,035	4,780	1.74
2	1,825	1.00	59	2,811	4,636	1.54
-4	1,842	1.00	59	2,645	4,487	1.44
-10	1,805	1.02	59	2,499	4,305	1.38
-16	1,794	1.03	59	2,139	3,933	1.19
-22	1,807	1.04	60	1,993	3,799	1.10
-28	1,774	1.04	59	1,893	3,666	1.07
-34	1,814	1.04	60	1,722	3,536	0.95
-40	1,733	1.05	58	1,428	3,160	0.82
-46	1,686	1.08	58	1,344	3,031	0.80
-52	1,661	1.08	58	1,243	2,904	0.75
-58	1,602	1.10	56	1,178	2,780	0.74
-64	1,498	1.11	53	921	2,419	0.61
-70	1,368	1.14	50	884	2,252	0.65
-76	1,381	1.16	51	757	2,138	0.55
-82	1,348	1.18	51	675	2,023	0.50
-88	1,240	1.16	46	492	1,732	0.40
-94	1,186	1.16	44	443	1,629	0.37
-100	1,161	1.17	44	367	1,527	0.32
-106	1,111	1.19	42	316	1,426	0.28
-112	964	1.19	37	198	1,162	0.21
-118	927	1.18	35	97	1,024	0.11
-124	853	1.18	32	81	934	0.09
-130	782	1.21	30	62	844	0.08
-136	630	1.28	26	15	645	0.02
-142	555	1.31	23	17	572	0.03
-148	481	1.31	20	16	497	0.03
-154	425	1.32	18	9	433	0.02
-160	296	1.33	13	6	302	0.02
-166	242	1.31	10	5	246	0.02
-172	169	1.27	7	1	170	0.00
-178	117	1.20	4	2	119	0.02
-184	62	1.17	2	-	62	-
Total	60,050	1.05	2,026	142,524	202,573	2.37

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

18.3.3 In-pit Inferred Resources

For the Base case, 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 Base case pit contain a total of 162,000 tonnes of Inferred material above the variable cutoffs used for the Base case reserves (a cutoff of 0.55 g Au/t was used for most areas, and a 0.60 cutoff was used for select benches in phases one and two – *See Section 18.2.11*). The average grade of the Inferred material is 0.66 g Au/t.

18.4 Sensitivity Case – 10.6 Million TPY Mining Option

18.4.1 Pit Designs

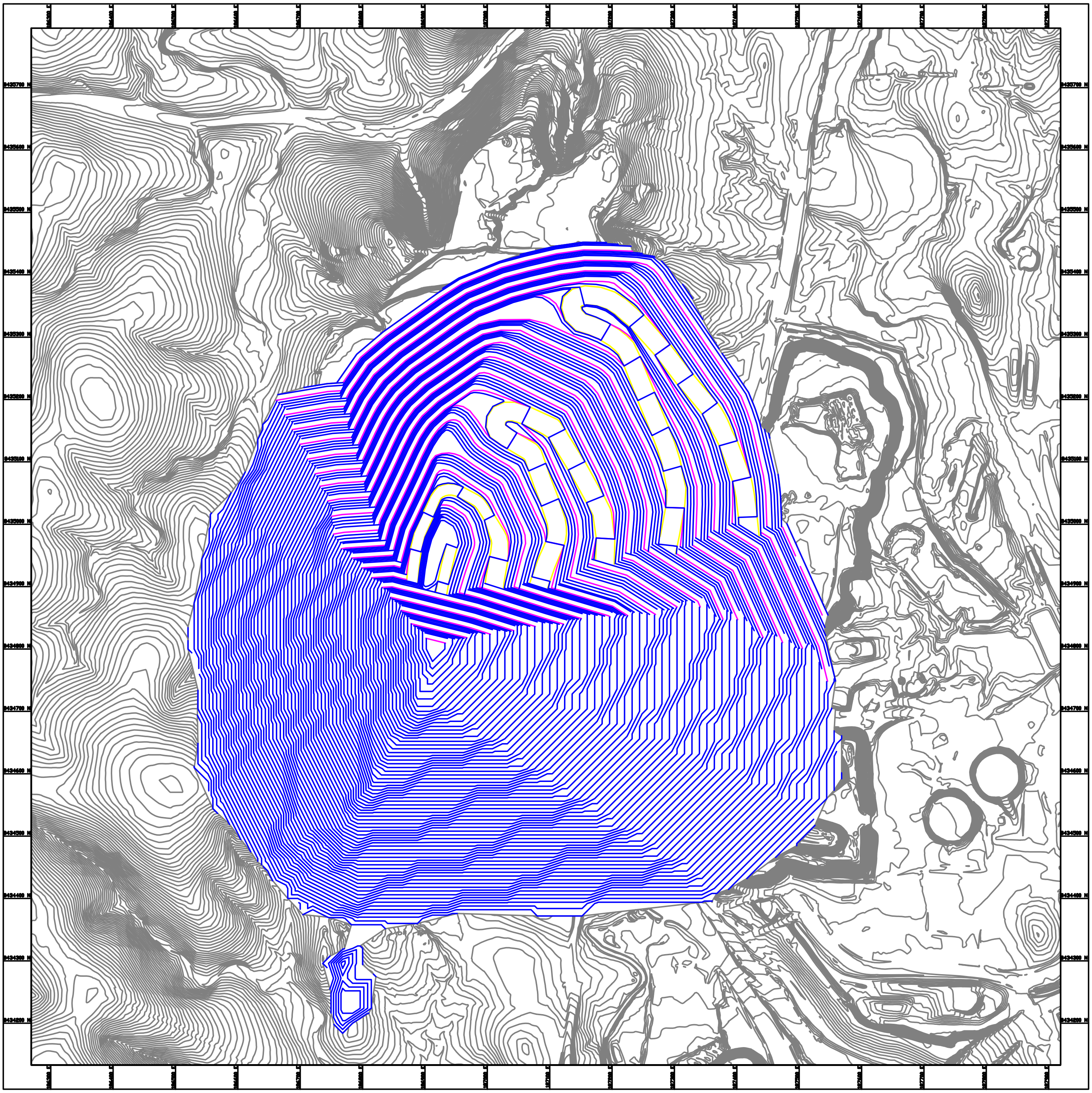
The detailed pit design was completed for the Sensitivity case pits used the two internal and the Base case ultimate pit as phases one through three. The ultimate pit volume was defined by the $850 per ounce Au Whittle pit without additional design work for access. As the slope angles for the Whittle pit shells used shallow angles to represent ramp access, MDA considers the use of the Whittle pit shell to be reasonable for a Preliminary Assessment of the resources within the Whittle pit shell. The pits used to define the resources for the Sensitivity case are discussed in further sections.

18.4.2 Sensitivity case Pit Designs

The Sensitivity case pit uses the Base case pit designs as phase one through three and then extend the mine life mining two additional pit phases to achieve the Sensitivity case ultimate pit limit. Thus, for the Sensitivity case, five pit phases have been defined. Sensitivity case phase one and phase two pits are shown in FIGURE 18-1 (Base case phase 1) and FIGURE 18-2 (Base case phase 2) respectively. The Sensitivity phase three pit is shown in FIGURE 18-3 is the Base case ultimate pit).

The Sensitivity case ultimate pit limit is defined by the $850 per ounce Au Whittle pit without additional design work for access. As the slope angles for the Whittle pit shells used shallow angles to represent ramp access, MDA considers the use of the Whittle pit shell to be reasonable for a Preliminary Assessment of these resources.

The incremental volume between the ultimate pit limit and the Base case ultimate pit was divided roughly into two equal remaining parts: east and west. This was done to reduce the total tonnes in each phase to manageable quantities for pit phases. The western portion was designated as phase four, and the eastern portion is phase five. The division used a sloped wall based on pit-design criteria. The resulting Sensitivity case phases four and five are shown in FIGURES 18-4 and, respectively.

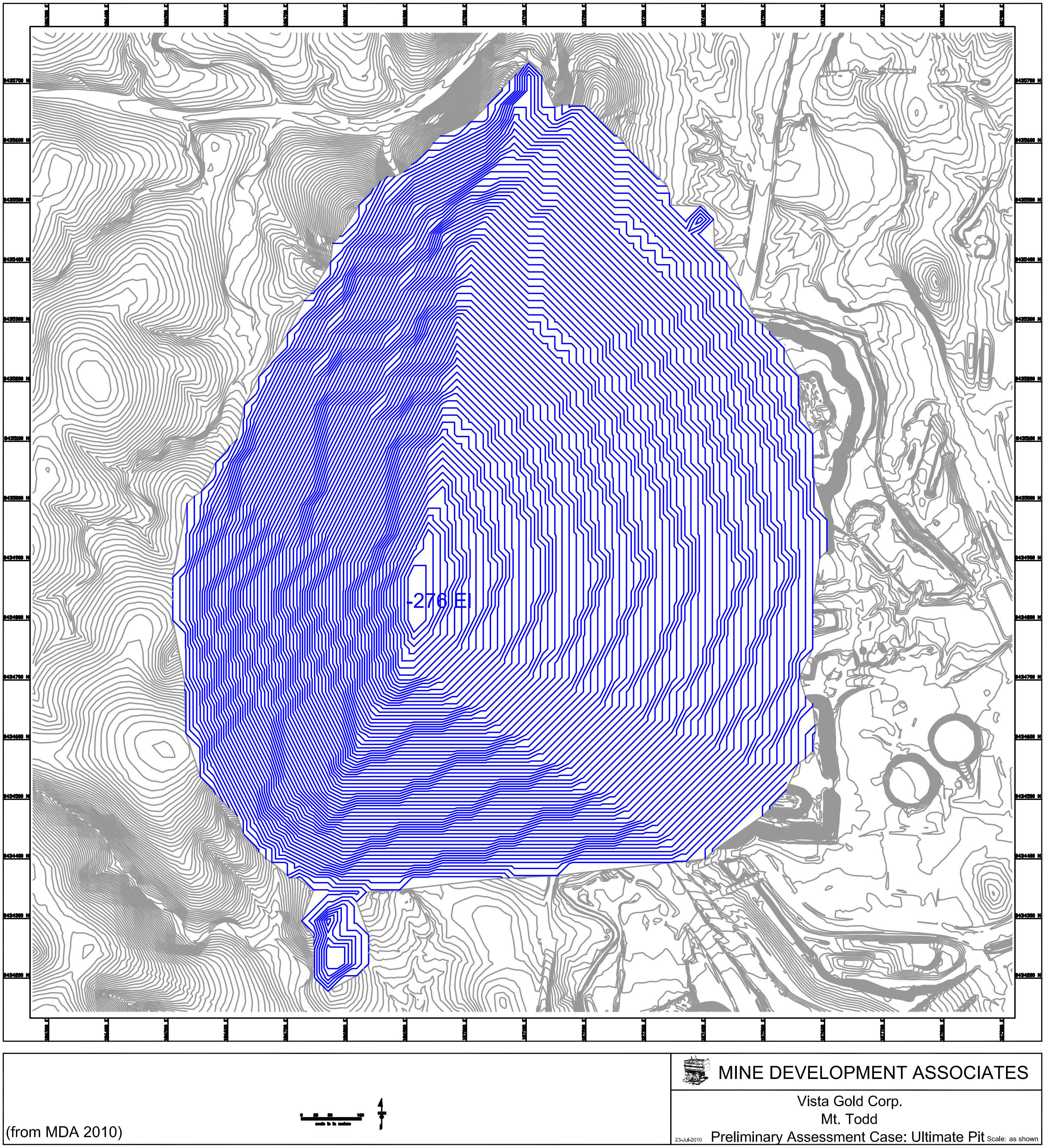


(from MDA 2010)

MINE DEVELOPMENT ASSOCIATES

Vista Gold Corp.
Mt. Todd
13-Sep-2010 Preliminary Assessment Case: Phase 4 Scale: as shown

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: Fig18-4.dwg	Figure 18-4
	Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a	**Preliminary Assessment**
	Project Location: Northern Territory, Australia	Date of Issue: 09/15/2010	**(10.9 mtpy) Case - Phase 4 Pit**



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TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Vista Gold Corp.*	Fig18-5.dwg	**Figure 18-5** **Preliminary Assessment (10.9 mtpy) Case - Ultimate Pit**
	Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: 09/15/2010	

18.4.3 Sensitivity case Resources

The Sensitivity case economics and pit designs have not been developed to the standards of a pre-feasibility study. As such, no reserves have been stated beyond the Base case. It is important to note that while a Sensitivity study may include Inferred resources, only Measured and Indicated resources were used for the Sensitivity case economic analysis. TABLE 18-9 shows the Measured and Indicated resources for the Sensitivity case based on a cutoff of 0.40 g Au/t. The Sensitivity resources reported in TABLE 18-9 are inclusive of the Proven and Probable reserves reported for the Base case.

18.4.4 In-pit Inferred Resources

For the Sensitivity-case, 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 Sensitivity case pits contain a total of 4,099,000 tonnes of Inferred material above a 0.40 g Au/t cutoff with an average grade of 0.71 g Au/t.

TABLE 18-9: SENSITIVITY CASE – MEASURED AND INDICATED RESERVES
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Phase	Measured Resources			Indicated Resources			Measured and Indicated			Waste K Tonnes	Total K Tonnes	Strip Ratio
	K Tonnes	g Au/t	K oz Au	K Tonnes	g Au/t	K oz Au	K Tonnes	g Au/t	K oz Au			
1	10,428	1.14	382	9,103	0.95	279	19,531	1.05	661	14,428	33,959	0.74
2	7,800	0.85	214	16,891	0.82	444	24,691	0.83	658	40,929	65,620	1.66
3	13,029	0.87	364	29,163	0.80	749	42,192	0.82	1,113	60,301	102,493	1.43
4	5,151	0.90	149	15,160	0.94	456	20,311	0.93	605	99,430	119,741	4.90
5	10,147	0.85	276	22,304	0.82	584	32,451	0.82	860	67,282	99,733	2.06
Total	46,555	0.93	1,385	92,621	0.84	2,512	139,176	0.87	3,897	282,370	421,546	2.03

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. Both the Base and Sensitivity cases considered only open-pit mining methods.

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 (see FIGURE 19-1 for Site Layout. During the mine life, it is assumed that the slopes would be dozed to a final reclamation slope dependent on specification through permitting. The cost for reclaiming the waste dumps has been estimated by Tetra Tech. The final dump will be sloped to promote runoff of precipitation.

The current waste facility is approximately 24 meters high located to the southeast of the pit. The ultimate dump design is approximately 70 meters above the original topography. Based on an assumed 1.4 swell factor and an average SG of 2.67 (bank), dumps have been designed to contain approximately 165 million tonnes of waste, which is 16% larger than required for the Base case.

Additional design work has been done to increase the dump height to approximately 100 meters above original topography. This will provide approximately 283.6 million tonnes of capacity, which is 2% greater than required for the Sensitivity case.

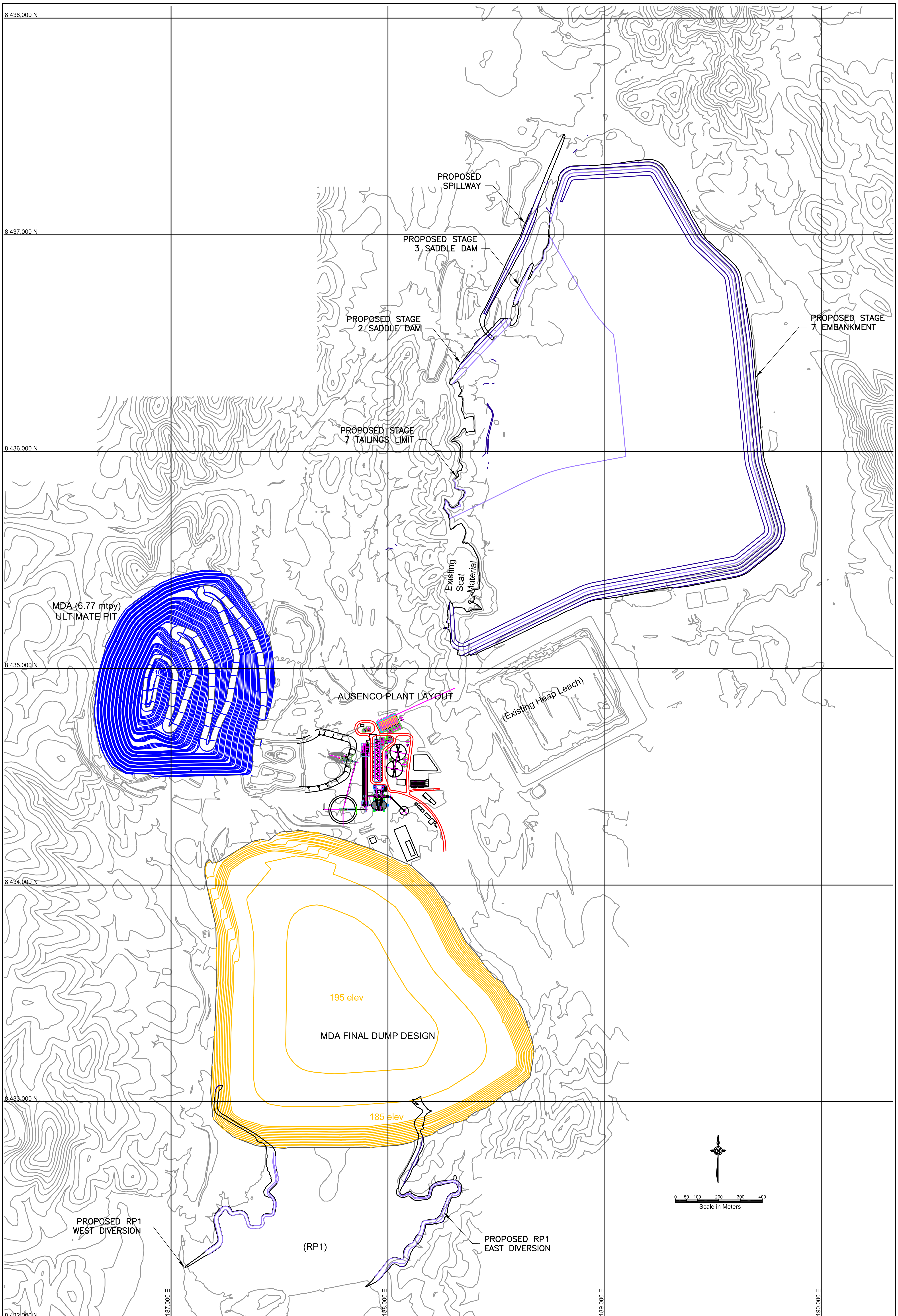
The waste dump annual positions are shown as they are used in Appendix A for the Base case.

19.1.3 Mine Production Schedule

Various production scenarios were investigated based on ore tonnage requirements. These scenarios include:

- 5.2 million tonnes per year;
- 6.7 million tonnes per year (Base Case);
- million tonnes per year; and
- 10.6 million tonnes per year (Sensitivity Case).

The 5.2, 6.7, and 7.5 million tonne per year scenarios used Measured and Indicated resources inside of the Base case pit designs. The 10.6 million tonnes per year scenario used Measured and Indicated resources inside of the Sensitivity case. Available truck and excavator hours were used to constrain the schedules for each scenario to ensure that the production is achievable and as realistic as possible. MDA provided mining costs and capital to Tetra Tech for economic evaluation for each of the scenarios. The 6.7 million tonne per year scenario was



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TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Vista Gold Corp.*	Fig19-1.dwg	Figure 19-1 **General Facilities Layout Map**
	Project: Mt Todd PreFeasibility Study	Project Number: 114-310912a	
	Project Location: Northern Territory, Australia	Date of Issue: 09/17/2010	

chosen as the basis for the pre-feasibility study based on meeting capital goals and Vista's corporate goals for the project; thus that scenario became the Base case. The 10.6 million tonnes per year scenario is the basis for the Sensitivity case production and costs. Both the Base and Sensitivity case production schedules are presented in the following sections.

19.1.4 Base Case 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 anticipated the trucks and shovels required to produce 6.7 million tonnes of ore per year and maintain stripping requirements. TABLE 19-1 shows the Base case mine-production schedule. Ore material is broken down into two categories: high-grade ore consists of Proven and Probable reserves that are equal to or above a 1.0 g Au/t cutoff grade; and medium-grade ore is Proven and Probable reserves below a 1.0 g Au/t cutoff (and above the lower-end reserve cutoff as shown in TABLE 18-5).

Ore material from the mine is to be sent from the pit directly to the crusher or to a mill ore stockpile. For the purpose of this schedule, two stockpiles are assumed: the high-grade ore stockpile for high-grade ore; and the medium-grade stockpile for longer-term storage of medium-grade ore. The stockpiles were used to increase the gold grades in the first years of mining as much as possible. During the life-of-mine, the medium-grade stockpile is allowed to grow to a maximum of over six million tonnes. Re-handling of stockpiled material will be done using a loader and trucks to haul ore to the crusher. TABLE 19-2 shows the material that is re-handled, and TABLE 19-3 shows the resulting stockpile balances for the end of each year.

TABLE 19-4 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.

Appendix C shows the annual pit positions for the production schedule.

19.1.5 Sensitivity Case Mine Production Schedule

The Sensitivity case mine-production schedule was developed to supply 10.6 million tonnes of ore per year to the mill while maintaining required waste stripping levels to a minimum where possible. The annual production estimate is shown in TABLE 19-5. Annual ore re-handle, stockpile balances, and ore delivery to the crusher are shown in TABLE 19-6, TABLE 19-7, and TABLE 19-8, respectively.

For the Sensitivity case, an additional stockpile was used for low-grade material between 0.40 and 0.55 g Au/t material. When mill ore requirements were satisfied within the schedule, this material was stockpiled as encountered and then fed through the crusher at the end of the mine life. The medium-grade stockpiles were used to store material from 0.55 to 1.00 g Au/t, and the high-grade stockpiles were used for storage of material equal to or greater than 1.00 g Au/t.

TABLE 19-1: BASE CASE ANNUAL MINE PRODUCTION SCHEDULE
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

			Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Total
Total Mine Production	Mined to Crusher	K Tonnes	-	6,628	6,770	6,770	6,770	6,789	6,770	6,770	6,327	178	-	53,772
		g Au/t	-	1.08	1.22	1.22	0.96	0.98	0.91	1.08	1.23	1.22	-	1.09
		K Au Ozs	-	231	266	265	210	213	198	236	250	7	-	1,876
	Mined to Medium Grade Stockpile	K Tonnes	421	479	2,110	3,151	-	-	-	-	-	-	-	6,161
		g Au/t	0.74	0.78	0.75	0.72	-	-	-	-	-	-	-	0.74
		K Au Ozs	10	12	51	73	-	-	-	-	-	-	-	146
	Mined to High Grade Stockpile	K Tonnes	117	-	-	-	-	-	-	-	-	-	-	117
		g Au/t	1.33	-	-	-	-	-	-	-	-	-	-	1.33
		K Au Ozs	5	-	-	-	-	-	-	-	-	-	-	5
	Total Ore Mined	K Tonnes	538	7,107	8,880	9,921	6,770	6,789	6,770	6,770	6,327	178	-	60,050
		g Au/t	0.87	1.06	1.11	1.06	0.96	0.98	0.91	1.08	1.23	1.22	-	1.05
		K Au Ozs	15	243	317	338	210	213	198	236	250	7	-	2,027
	Waste Mined	K Tonnes	5,583	27,898	25,095	23,519	29,948	17,573	8,717	3,685	504	2	-	142,524
	Total Mined	K Tonnes	6,121	35,005	33,975	33,440	36,718	24,362	15,487	10,455	6,831	180	-	202,574
		Strip Ratio	10.38	3.93	2.83	2.37	4.42	2.59	1.29	0.54	0.08	0.01		2.37

TABLE 19-2: BASE CASE ANNUAL ORE RE-HANDLE SCHEDULE
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Rehandle		Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Total
Medium Grade Stockpile to Crusher	K Tonnes	-	44	-	-	-	-	-	-	443	5,674	-	6,161
	g Au/t	-	0.71	-	-	-	-	-	-	0.77	0.73	-	0.74
	K Au Ozs	-	1	-	-	-	-	-	-	11	134	-	146
High Grade Stockpile to Crusher	K Tonnes	-	117	-	-	-	-	-	-	-	-	-	117
	g Au/t	-	1.33	-	-	-	-	-	-	-	-	-	1.33
	K Au Ozs	-	5	-	-	-	-	-	-	-	-	-	5
Total Rehandle to Crusher	K Tonnes	-	161	-	-	-	-	-	-	443	5,674	-	6,278
	g Au/t	-	1.16	-	-	-	-	-	-	0.77	0.73	-	0.75
	K Au Ozs	-	6	-	-	-	-	-	-	11	134	-	151

TABLE 19-3: BASE CASE ANNUAL STOCKPILE BALANCE
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Stockpile Balance		Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10
MG_StkPl	K Tonnes	421	856	2,966	6,117	6,117	6,117	6,117	6,117	5,674	-	-
	g Au/t	0.74	0.76	0.76	0.74	0.74	0.74	0.74	0.74	0.73	-	-
	K Au Ozs	10	21	72	145	145	145	145	145	134	-	-
HG_StkPl	K Tonnes	117	-	-	-	-	-	-	-	-	-	-
	g Au/t	1.33	-	-	-	-	-	-	-	-	-	-
	K Au Ozs	5	-	-	-	-	-	-	-	-	-	-
Total	K Tonnes	538	856	2,966	6,117	6,117	6,117	6,117	6,117	5,674	-	-
	g Au/t	0.87	0.76	0.76	0.74	0.74	0.74	0.74	0.74	0.73	-	-
	K Au Ozs	15	21	72	145	145	145	145	145	134	-	-

TABLE 19-4: BASE CASE ANNUAL ORE DELIVERY TO THE MILL CRUSHER
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Sent to Crusher		Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Total
Pit to Crusher	K Tonnes	-	6,628	6,770	6,770	6,770	6,789	6,770	6,770	6,327	178	-	53,772
	g Au/t	-	1.08	1.22	1.22	0.96	0.98	0.91	1.08	1.23	1.22	-	1.09
	K Au Ozs	-	231	266	265	210	213	198	236	250	7	-	1,876
Rehandle to Crusher	K Tonnes	-	160	-	-	-	-	-	-	443	5,674	-	6,277
	g Au/t	-	1.17	-	-	-	-	-	-	0.77	0.73	-	0.75
	K Au Ozs	-	6	-	-	-	-	-	-	11	134	-	151
Total to Crusher	K Tonnes	-	6,788	6,770	6,770	6,770	6,789	6,770	6,770	6,770	5,852	-	60,049
	g Au/t	-	1.09	1.22	1.22	0.96	0.98	0.91	1.08	1.20	0.75	-	1.05
	K Au Ozs	-	237	266	265	210	213	198	236	261	141	-	2,027

TABLE 19-5: SENSITIVITY CASE ANNUAL MINE PRODUCTION SCHEDULE
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

			Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Total
Total Mine Production	Mined to Crusher	K Tonnes	-	10,806	10,404	9,883	10,950	10,980	10,950	10,950	4,260	8,440	10,950	8,447	10,950	7,685	-	125,655
		g Au/t	-	0.89	1.18	0.80	0.94	0.69	0.87	1.13	0.85	0.82	0.98	0.68	0.83	1.12	-	0.91
		K Au Ozs	-	310	394	255	332	245	307	398	116	222	344	184	292	276	-	3,675
	LG_StkPl	Tonnes	359	692	4,246	2,064	1,182	-	2,126	1,064	-	-	-	-	-	-	-	11,733
		g Au/t	0.43	0.45	0.47	0.47	0.47	-	0.47	0.50	-	-	-	-	-	-	-	0.47
		Au Ozs	5	10	64	31	18	-	32	17	-	-	-	-	-	-	-	177
	Mined to Medium Grade Stockpile	K Tonnes	421	-	1,229	-	-	-	-	-	-	-	-	-	-	-	-	1,650
		g Au/t	0.74	-	0.73	-	-	-	-	-	-	-	-	-	-	-	-	0.74
		K Au Ozs	10	-	29	-	-	-	-	-	-	-	-	-	-	-	-	39
	Mined to High Grade Stockpile	K Tonnes	117	-	21	-	-	-	-	-	-	-	-	-	-	-	-	138
		g Au/t	1.33	-	1.48	-	-	-	-	-	-	-	-	-	-	-	-	1.35
		K Au Ozs	5	-	1	-	-	-	-	-	-	-	-	-	-	-	-	6
	Total Ore Mined	K Tonnes	897	11,498	15,900	11,947	12,132	10,980	13,076	12,014	4,260	8,440	10,950	8,447	10,950	7,685	-	139,176
		g Au/t	0.69	0.87	0.95	0.74	0.90	0.69	0.81	1.07	0.85	0.82	0.98	0.68	0.83	1.12	-	0.87
		K Au Ozs	20	320	488	286	350	245	339	415	116	222	344	184	292	276	-	3,897
	Waste Mined	K Tonnes	5,224	24,287	19,608	24,002	25,294	25,777	23,112	24,382	27,282	26,476	20,483	27,283	8,688	472	-	282,370
	Total Mined	K Tonnes	6,121	35,785	35,508	35,949	37,426	36,757	36,188	36,396	31,542	34,916	31,433	35,730	19,638	8,157	-	421,546
		Strip Ratio	5.82	2.11	1.23	2.01	2.08	2.35	1.77	2.03	6.40	3.14	1.87	3.23	0.79	0.06		2.03

TABLE 19-6: SENSITIVITY CASE ANNUAL ORE RE-HANDLE SCHEDULE
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Rehandle		Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Total
LG_StkPl	K Tonnes	-	-	-	-	-	-	-	-	6,690	2,540	-	2,503	-	-	-	11,733
	g Au/t	-	-	-	-	-	-	-	-	0.47	0.48	-	0.47	-	-	-	0.47
	K Au Ozs	-	-	-	-	-	-	-	-	101	39	-	38	-	-	-	178
Medium Grade	K Tonnes	-	58	525	1,067	-	-	-	-	-	-	-	-	-	-	-	1,650
Stockpile to	g Au/t	-	0.54	0.77	0.73	-	-	-	-	-	-	-	-	-	-	-	0.74
Crusher	K Au Ozs	-	1	13	25	-	-	-	-	-	-	-	-	-	-	-	39
High Grade	K Tonnes	-	117	21	-	-	-	-	-	-	-	-	-	-	-	-	138
Stockpile to	g Au/t	-	1.33	1.48	-	-	-	-	-	-	-	-	-	-	-	-	1.35
Crusher	K Au Ozs	-	5	1	-	-	-	-	-	-	-	-	-	-	-	-	6
Total Rehandle	K Tonnes	-	175	546	1,067	-	-	-	-	6,690	2,540	-	2,503	-	-	-	13,521
to Crusher	g Au/t	-	1.07	0.80	0.73	-	-	-	-	0.47	0.48	-	0.47	-	-	-	0.51
	K Au Ozs	-	6	14	25	-	-	-	-	101	39	-	38	-	-	-	223

TABLE 19-7: SENSITIVITY CASE ANNUAL STOCKPILE BALANCE
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Stockpile Balance		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
LG_StkPl	K Tonnes	359	1,051	5,297	7,361	8,543	8,543	10,669	11,733	5,043	2,503	2,503	-	-	-	-
	g Au/t	0.43	0.47	0.47	0.47	0.47	0.47	0.47	0.47	0.47	0.47	0.47	-	-	-	-
	K Au Ozs	5	16	80	111	129	129	162	178	77	38	38	-	-	-	-
MG_StkPl	K Tonnes	421	363	1,067	-	-	-	-	-	-	-	-	-	-	-	-
	g Au/t	0.74	0.69	0.73	-	-	-	-	-	-	-	-	-	-	-	-
	K Au Ozs	10	8	25	-	-	-	-	-	-	-	-	-	-	-	-
HG_StkPl	K Tonnes	117	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	g Au/t	1.33	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	K Au Ozs	5	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total	K Tonnes	897	1,414	6,364	7,361	8,543	8,543	10,669	11,733	5,043	2,503	2,503	-	-	-	-
	g Au/t	0.69	0.53	0.51	0.47	0.47	0.47	0.47	0.47	0.47	0.47	0.47	-	-	-	-
	K Au Ozs	20	24	105	111	129	129	162	178	77	38	38	-	-	-	-

TABLE 19-8: SENSITIVITY CASE ANNUAL ORE DELIVERY TO THE MILL CRUSHER
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Sent to Crusher		Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Total
Pit to	K Tonnes	-	10,806	10,404	9,883	10,950	10,980	10,950	10,950	4,260	8,440	10,950	8,447	10,950	7,685	-	125,655
Crusher	g Au/t	-	0.89	1.18	0.80	0.94	0.69	0.87	1.13	0.85	0.82	0.98	0.68	0.83	1.12	-	0.91
	K Au Ozs	-	310	394	255	332	245	307	398	116	222	344	184	292	276	-	3,675
Rehandle to	K Tonnes	-	174	546	1,067	-	-	-	-	6,690	2,540	-	2,503	-	-	-	13,520
Crusher	g Au/t	-	1.07	0.80	0.73	-	-	-	-	0.47	0.48	-	0.47	-	-	-	0.51
	K Au Ozs	-	6	14	25	-	-	-	-	101	39	-	38	-	-	-	223
Total to	K Tonnes	-	10,980	10,950	10,950	10,950	10,980	10,950	10,950	10,950	10,980	10,950	10,950	10,950	7,685	-	139,175
Crusher	g Au/t	-	0.90	1.16	0.80	0.94	0.69	0.87	1.13	0.62	0.74	0.98	0.63	0.83	1.12	-	0.87
	K Au Ozs	-	316	408	280	332	245	307	398	217	261	344	222	292	276	-	3,898

19.1.6 Equipment Selection and Productivities

Both the Base and Sensitivity cases have been planned as an open-pit mine using large haul trucks, hydraulic shovels, and front-end loading equipment. Primary production is achieved using two Hitachi Ex3600 hydraulic shovels along with CAT 789C haul trucks. A shovel bucket size of 21 cubic meters 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 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. Waste production from the 992 loader and 785C trucks is anticipated as well.

TABLE 19-9 shows the maximum shovel productivity estimate based on scheduled time, availability, and truck and material parameters. This maximum productivity would require that trucks are always available; however, that is not always the case.

Truck productivity is based on truck-cycle times from the pit to predetermined destinations. The destinations include the crusher, ore stockpiles, and waste dumps. 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. A total of 11 dumps was used. 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. The number of trucks used for the Base case is shown in TABLE 19-10 along with the total available and used operating hour estimates. TABLE 19-10 also includes the same information for the 992 loader and the Ex3600 shovel fleet.

Truck, loader, and shovel usage is shown for the Sensitivity case in TABLE 19-11.

TABLE 19-9: MAXIMUM LOADER PRODUCTIVITY ESTIMATE
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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 19-10:BASE CASE ANNUAL LOAD AND HAUL EQUIPMENT REQUIREMENTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

		Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10
	Total Tonnes Moved	6,120,320	35,166,177	33,974,500	33,439,878	36,717,877	24,361,878	15,486,544	10,455,361	7,273,625	5,854,799	-
Mine Schedule – Standby Time	Days per Period	245	366	365	365	365	366	365	365	365	366	
	Holidays per Period	5	9	9	9	9	9	9	9	9	9	
	Weather Delays	4	6	6	6	6	6	6	6	6	6	
	Days per Week	7	7	7	7	7	7	7	7	7	7	
	Shifts per Day	2	2	2	2	2	2	2	2	2	2	
	Hrs per Shift	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	
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	
	Shift Startup / Shutdown	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	
	Safety / Training Hrs/Shift	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	
	Gross Operator Efficiency	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	
Cat 785 Truck Fleet	Average Number of Trucks	-	3	3	2	3	2	-	-	1	2	-
	Truck Fleet Availability	0%	90%	89%	88%	87%	86%	0%	0%	85%	85%	0%
	Available Truck Operating Hrs	-	19,428	19,157	12,628	18,727	12,376	-	-	6,099	12,232	-
	Productive Truck Hrs Used	-	7,602	12,523	7,928	15,287	8,243	-	-	448	5,736	-
	Operating Efficiency	0%	83%	83%	83%	83%	83%	0%	0%	83%	83%	0%
	Truck Operating Hrs Used	-	9,159	15,088	9,551	18,419	9,932	-	-	540	6,911	-
	Remaining Operating Hrs	-	10,269	4,070	3,077	308	2,444	-	-	5,559	5,321	-
	Use of Operating Hours	0%	47%	79%	76%	98%	80%	0%	0%	9%	56%	0%
Cat 789 Truck Fleet	Average Number of Trucks	6	9	9	10	10	9	7	6	4	1	-
	Truck Fleet Availability	90%	89%	88%	87%	86%	85%	85%	85%	85%	85%	0%
	Available Truck Operating Hrs	26,125	57,636	56,826	62,423	61,705	55,046	42,691	36,593	24,395	6,116	-
	Productive Truck Hrs Used	7,499	43,961	44,756	48,138	50,486	43,566	35,096	25,778	18,021	532	-
	Operating Efficiency	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	0%
	Truck Operating Hrs Used	9,035	52,965	53,923	57,998	60,826	52,490	42,284	31,058	21,712	641	-
	Remaining Operating Hrs	17,091	4,671	2,903	4,425	879	2,556	407	5,534	2,683	5,476	-
	Use of Operating Hours	35%	92%	95%	93%	99%	95%	99%	85%	89%	10%	0%
Cat 992 Loader Fleet	Excavator Availability	0%	90%	89%	88%	87%	86%	0%	0%	85%	85%	0%
	Available Excav Operating Hrs	-	6,476	6,386	6,314	6,242	6,188	-	-	6,099	6,116	-
	Productive Excav Hrs Used	-	2,863	3,526	2,014	3,531	1,778	-	-	327	4,186	-
	Operating Efficiency	0%	83%	83%	83%	83%	83%	0%	0%	83%	83%	0%
	Excav Operating Hrs Used	-	3,182	3,961	2,288	4,059	2,067	-	-	385	4,925	-
	Remaining Operating Hrs	-	3,294	2,424	4,026	2,183	4,121	-	-	5,714	1,192	-
	Use of Operating Hours	0%	49%	62%	36%	65%	33%	0%	0%	6%	81%	0%
Ex3600 Shovel Fleet	Excavator Availability	90%	90%	89%	88%	87%	85%	85%	85%	85%	85%	0%
	Available Excav Operating Hrs	8,708	12,925	12,734	12,592	12,413	12,304	6,099	6,099	6,099	6,116	-
	Productive Excav Hrs Used	1,973	10,085	9,411	9,900	10,293	7,076	4,992	3,370	2,202	58	-
	Operating Efficiency	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	0%
	Excav Operating Hrs Used	2,192	11,229	10,610	11,283	11,900	8,298	5,873	3,965	2,590	69	-
	Remaining Operating Hrs	6,516	1,695	2,124	1,310	513	4,007	226	2,134	3,508	6,048	-
	Use of Operating Hours	25%	87%	83%	90%	96%	67%	96%	65%	42%	1%	0%

TABLE 19-11: SENSITIVITY CASE ANNUAL LOAD AND HAUL EQUIPMENT REQUIREMENTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

			Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14
		Total Tonnes Moved	6,120,320	35,958,475	36,053,031	37,016,659	37,426,106	36,756,953	36,187,979	36,395,628	38,232,354	37,456,333	31,432,586	38,232,688	19,638,270	8,157,109	-
Mine Schedule	Standby Time	Days per Period	245	366	365	365	365	366	365	365	365	366	365	365	365	366	365
		Holidays per Period	5	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
		Days per Week	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
		Hrs per Shift	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
	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
		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
		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
		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
		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
		Gross Operator Efficiency	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%
	Cat 785 Truck Fleet	Average Number of Trucks	-	3	3	3	3	3	3	3	3	3	3	3	-	-	-
		Truck Fleet Availability	0%	89%	87%	86%	85%	85%	85%	85%	85%	85%	85%	85%	0%	0%	0%
		Available Truck Operating Hrs	-	19,212	18,727	18,512	18,296	18,349	18,296	18,296	18,296	18,349	18,296	18,296	-	-	-
		Productive Truck Hrs Used	-	10,212	14,907	14,747	14,321	15,153	15,110	15,111	9,340	13,071	106	18,010	-	-	-
		Operating Efficiency	0%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	0%	0%	0%
		Truck Operating Hrs Used	-	12,303	17,960	17,767	17,254	18,257	18,205	18,206	11,253	15,748	128	21,699	-	-	-
		Remaining Operating Hrs	-	6,909	767	744	1,042	92	91	90	7,043	2,601	18,168	(3,402)	-	-	-
		Use of Operating Hours	0%	64%	96%	96%	94%	100%	100%	100%	62%	86%	1%	119%	0%	0%	0%
	Cat 789 Truck Fleet	Average Number of Trucks	3	9	10	10	10	11	12	13	14	14	13	13	11	6	-
		Truck Fleet Availability	90%	89%	87%	86%	85%	85%	86%	86%	86%	86%	86%	86%	85%	85%	0%
		Available Truck Operating Hrs	13,063	57,636	62,710	61,992	61,275	67,494	73,616	79,930	86,244	86,490	79,930	79,930	67,302	36,697	-
		Productive Truck Hrs Used	7,357	41,257	41,857	45,397	50,735	54,256	57,423	64,215	71,303	71,911	61,643	64,321	53,245	25,581	-
		Operating Efficiency	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	0%
		Truck Operating Hrs Used	8,864	49,707	50,430	54,695	61,127	65,368	69,185	77,368	85,907	86,639	74,268	77,495	64,151	30,821	-
		Remaining Operating Hrs	4,199	7,929	12,279	7,297	147	2,125	4,431	2,562	336	(150)	5,661	2,434	3,151	5,876	-
		Use of Operating Hours	68%	86%	80%	88%	100%	97%	94%	97%	100%	100%	93%	97%	95%	84%	0%
	Cat 992 Loader Fleet	Excavator Availability	0%	90%	89%	88%	87%	86%	85%	85%	85%	85%	85%	85%	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	-	3,562	4,351	4,567	4,053	3,766	3,549	3,702	5,057	5,072	41	5,057	-	-	-
		Operating Efficiency	0%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	0%	0%	0%
		Excav Operating Hrs Used	-	3,957	4,889	5,190	4,659	4,379	4,176	4,355	5,949	5,967	48	5,949	-	-	-
		Remaining Operating Hrs	-	2,519	1,497	1,124	1,583	1,809	1,923	1,744	150	149	6,051	150	-	-	-
		Use of Operating Hours	0%	61%	77%	82%	75%	71%	68%	71%	98%	98%	1%	98%	0%	0%	0%
	Ex3600 Shovel Fleet	Excavator Availability	90%	90%	89%	88%	87%	86%	85%	85%	85%	85%	85%	85%	85%	85%	0%
		Available Excav Operating Hrs	8,708	12,920	12,735	12,591	12,413	12,304	12,198	12,198	12,198	12,232	12,198	12,198	12,198	6,116	-
		Productive Excav Hrs Used	1,973	10,035	9,721	9,937	10,293	10,203	10,114	10,115	10,115	9,858	10,115	10,115	6,331	2,629	-
		Operating Efficiency	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	83%	0%
		Excav Operating Hrs Used	2,192	11,178	10,951	11,327	11,900	11,934	11,899	11,900	11,900	11,598	11,900	11,900	7,448	3,094	-
		Remaining Operating Hrs	6,516	1,742	1,784	1,264	513	371	298	298	298	635	298	298	4,750	3,023	-
		Use of Operating Hours	25%	87%	86%	90%	96%	97%	98%	98%	98%	95%	98%	98%	61%	51%	0%

19.1.7 Mine Personnel

Mine personnel estimates include both operating and mine staff personnel. Operating personnel is estimated as the number of people required to operate trucks, loading equipment, and support equipment to achieve the production schedules for both the Base and Sensitivity cases. Mine staff is based on the people required for supervision and support of mine production. The mine staff organizational chart is shown in FIGURE 19-1. The estimated number of mine personnel required to execute the Base case mine plan is shown in TABLE 19-12 and the Sensitivity case personnel requirements are shown in TABLE 19-13.

Salaries for each position were estimated based information received from Tetra Tech. Salaries include an allowance for benefits at a rate of 25% of the base salary for each position. The salaries used are shown in TABLE 19-14 presented in both Australian and US dollars. The extended cost for labor by year is shown in thousands of US dollars in TABLE 19-15 and TABLE 19-16 for the Base and Sensitivity cases, respectively. Note that mobile equipment labor costs are allocated to production equipment in the calculation of mining costs in later sections.

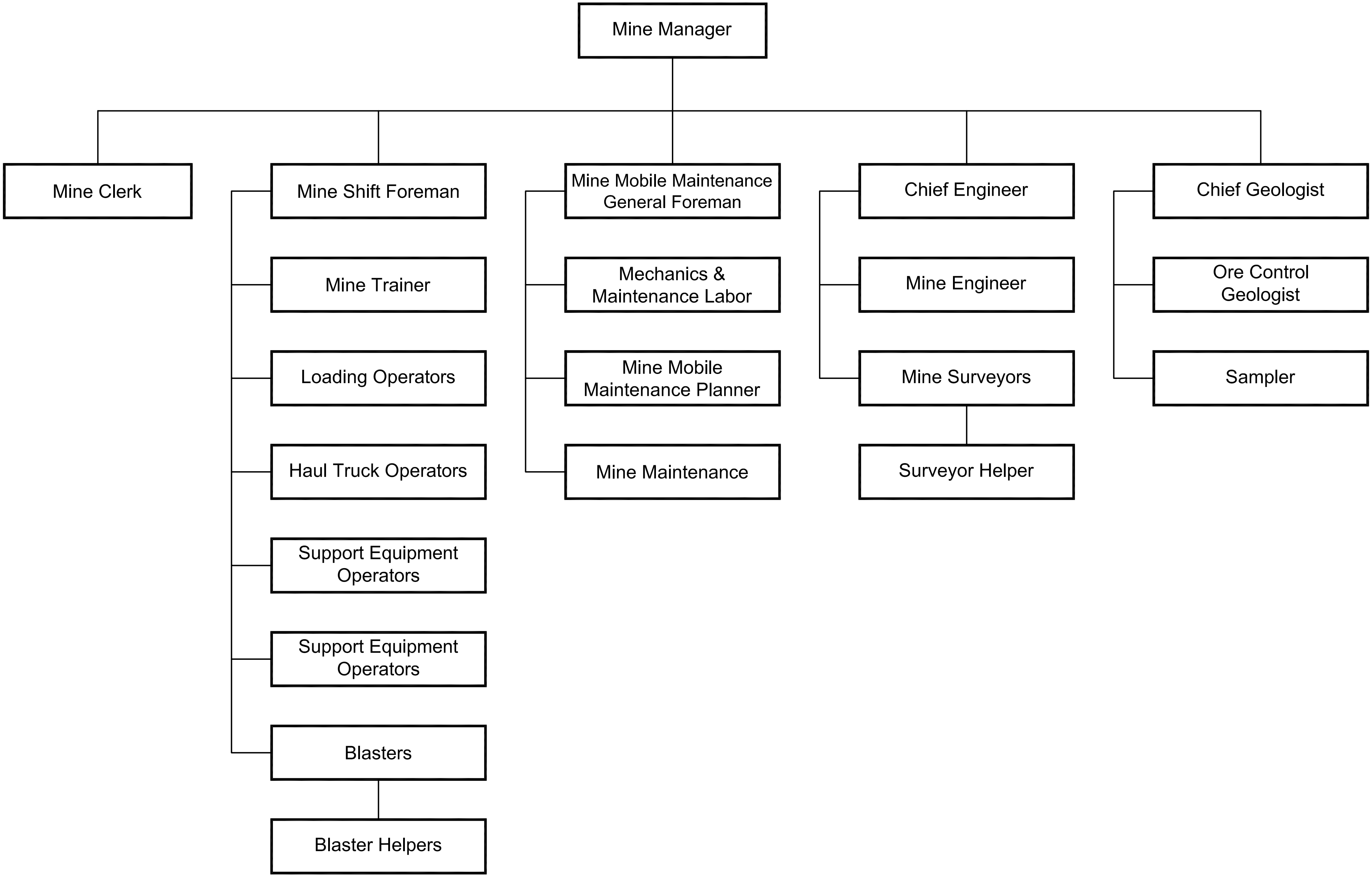

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Figure 19-2
Mine Organizational Chart

TABLE 19-12: BASE CASE MINE PERSONNEL REQUIREMENTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Mine Overhead	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10
Mine Manager	1	1	1	1	1	1	1	1	1	1	-
Mine Clerk	1	1	1	1	1	1	1	1	1	1	-
Mine Shift Foremen	4	4	4	4	4	4	4	4	4	4	-
Mine Trainer	1	1	1	1	1	1	1	1	1	1	-
Blaster	2	2	2	2	2	2	2	2	2	2	-
Blaster's Helper	4	4	4	4	4	4	4	4	4	4	-
Mine Production											
Loading Operators	8	10	12	12	12	10	6	4	4	4	-
Haul Truck Operators	24	48	48	48	52	44	28	24	20	12	-
Drill Operators	6	16	16	16	16	12	8	6	4	2	-
Support Equipment Operators	12	20	20	20	20	20	16	16	12	12	-
Total Mine Operating	63	107	109	109	113	99	71	63	53	43	-
Mine Maintenance											
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	-
Light Vehicle Mechanics	1	2	2	2	2	2	2	2	1	1	-
Mobile Equipment Mechanics	19	34	29	29	29	26	24	18	14	10	-
Mobile Equipment Welders	3	3	3	3	3	3	3	3	3	3	-
Mobile Equipment Servicemen	1	1	1	1	1	1	1	1	1	1	-
Tiremen	1	2	2	2	2	2	2	2	1	1	-
Shop Laborers	2	4	4	4	4	4	4	4	2	2	-
Maintenance Planner	1	1	1	1	1	1	1	1	1	1	-
Service, Fuel, & Lube	4	8	8	8	8	8	8	8	4	4	-
Total Mine Maintenance	33	56	51	51	51	48	46	40	28	24	-
Engineering											
Chief Engineer	1	1	1	1	1	1	1	1	1	1	-
Mine Surveyors	1	2	2	2	2	2	2	2	1	1	-
Surveyor Helper	1	2	2	2	2	2	2	2	1	1	-
Mine Engineer	1	1	1	1	1	1	1	1	1	1	-
Total Engineering	4	6	6	6	6	6	6	6	4	4	-
Mine Geology											
Chief Geologist	1	1	1	1	1	1	1	1	1	1	-
Ore Control Geologist	2	2	2	2	2	2	2	2	1	1	-
Sampler	2	2	2	2	2	2	2	2	1	1	-
Total Geology	5	5	5	5	5	5	5	5	3	3	-
Total Mine Operations Workforce											
Mine Operations	63	107	109	109	113	99	71	63	53	43	-
Mine Maintenance	33	56	51	51	51	48	46	40	28	24	-
Engineering	4	6	6	6	6	6	6	6	4	4	-
Geology	5	5	5	5	5	5	5	5	3	3	-
Total	105	174	171	171	175	158	128	114	88	74	-

TABLE 19-13: SENSITIVITY CASE MINE PERSONNEL REQUIREMENTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

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
Mine Manager	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	-
Mine Shift Foremen	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	-
Blaster	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	-
Mine Production															
Loading Operators	6	12	12	12	12	12	12	12	12	12	12	12	6	4	-
Haul Truck Operators	24	48	52	52	52	56	60	64	68	68	64	64	44	24	-
Drill Operators	5	16	16	16	17	17	17	17	15	15	15	15	9	4	-
Support Equipment Operators	12	20	20	20	20	20	20	20	20	20	20	20	16	12	-
Total Mine Operating	60	109	113	113	114	118	122	126	128	128	124	124	88	57	-
Mine Maintenance															
Maintenance Superintendent	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	-
Mobile Equipment Mechanics	17	35	29	29	29	29	29	41	42	42	40	40	28	16	-
Mobile Equipment Welders	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	-
Tiremen	1	2	2	2	2	2	2	2	1	1	1	1	1	1	-
Shop Laborers	2	4	4	4	4	4	4	4	2	2	2	2	2	2	-
Maintenance Planner	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	-
Total Mine Maintenance	31	57	51	51	51	51	51	63	56	56	54	54	42	30	-
Engineering															
Chief Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Mine Surveyors	1	2	2	2	2	2	2	2	1	1	1	1	1	1	-
Surveyor Helper	1	2	2	2	2	2	2	2	1	1	1	1	1	1	-
Mine Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-
Total Engineering	4	6	6	6	6	6	6	6	4	4	4	4	4	4	-
Mine Geology															
Chief Geologist	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	1	1	1	1	1	1	-
Sampler	2	2	2	2	2	2	2	2	1	1	1	1	1	1	-
Total Geology	5	5	5	5	5	5	5	5	3	3	3	3	3	3	-
Total Mine Operations Workforce															
Mine Operations	60	109	113	113	114	118	122	126	128	128	124	124	88	57	-
Mine Maintenance	31	57	51	51	51	51	51	63	56	56	54	54	42	30	-
Engineering	4	6	6	6	6	6	6	6	4	4	4	4	4	4	-
Geology	5	5	5	5	5	5	5	5	3	3	3	3	3	3	-
Total	100	177	175	175	176	180	184	200	191	191	185	185	137	94	-

TABLE 19-14:MINE PERSONNEL SALARY RATES
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

	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 19-15: BASE CASE MINE ANNUAL PERSONNEL COSTS
($000's USD)
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Mine Overhead	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Total
Mine Manager	$ 134	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 112	$ -	$ 2,030
Mine Clerk	$ 38	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 32	$ -	$ 581
Mine Shift Foremen	$ 255	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 213	$ -	$ 3,869
Mine Trainer	$ 51	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 43	$ -	$ 774
Blaster	$ 115	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 96	$ -	$ 1,740
Blaster's Helper	$ 191	$ 319	$ 319	$ 319	$ 319	$ 319	$ 319	$ 319	$ 319	$ 159	$ -	$ 2,901
Mine Production												
Loading Operators	$ 459	$ 956	$ 1,147	$ 1,147	$ 1,147	$ 956	$ 574	$ 382	$ 382	$ 191	$ -	$ 7,342
Haul Truck Operators	$ 1,224	$ 4,080	$ 4,080	$ 4,080	$ 4,420	$ 3,740	$ 2,380	$ 2,040	$ 1,700	$ 510	$ -	$ 28,254
Drill Operators	$ 344	$ 1,530	$ 1,530	$ 1,530	$ 1,530	$ 1,147	$ 765	$ 574	$ 382	$ 96	$ -	$ 9,426
Support Equipment Operators	$ 612	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,360	$ 1,360	$ 1,020	$ 510	$ -	$ 13,362
Total Mine Operating	$ 3,423	$ 9,573	$ 9,764	$ 9,764	$ 10,104	$ 8,850	$ 6,386	$ 5,663	$ 4,792	$ 1,960	$ -	$ 70,279
Mine Maintenance												
Maintenance Superintendent	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ -	$ 1,451
Light Vehicle Mechanics	$ 48	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ 40	$ -	$ 1,283
Mobile Equipment Mechanics	$ 1,090	$ 3,250	$ 2,772	$ 2,772	$ 2,772	$ 2,486	$ 2,294	$ 1,721	$ 1,338	$ 478	$ -	$ 20,975
Mobile Equipment Welders	$ 143	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 120	$ -	$ 2,176
Mobile Equipment Servicemen	$ 48	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 40	$ -	$ 725
Tiremen	$ 48	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ 40	$ -	$ 1,283
Shop Laborers	$ 83	$ 276	$ 276	$ 276	$ 276	$ 276	$ 276	$ 276	$ 138	$ 69	$ -	$ 2,225
Maintenance Planner	$ 48	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 40	$ -	$ 725
Service, Fuel, & Lube	$ 191	$ 638	$ 638	$ 638	$ 638	$ 638	$ 638	$ 638	$ 319	$ 159	$ -	$ 5,133
Total Mine Maintenance	$ 1,794	$ 5,041	$ 4,563	$ 4,563	$ 4,563	$ 4,276	$ 4,085	$ 3,512	$ 2,513	$ 1,065	$ -	$ 35,976
Engineering												
Chief Engineer	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ -	$ 1,451
Mine Surveyors	$ 64	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 106	$ 53	$ -	$ 1,711
Surveyor Helper	$ 41	$ 138	$ 138	$ 138	$ 138	$ 138	$ 138	$ 138	$ 69	$ 35	$ -	$ 1,113
Mine Engineer	$ 80	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 66	$ -	$ 1,209
Total Engineering	$ 281	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 468	$ 234	$ -	$ 5,484
Mine Geology												
Chief Geologist	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ -	$ 1,451
Ore Control Geologist	$ 128	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 106	$ 53	$ -	$ 1,775
Sampler	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ 40	$ -	$ 1,331
Total Geology	$ 319	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 345	$ 173	$ -	$ 4,557
Total Mine Operations Workforce												
Mine Operations	$ 3,423	$ 9,573	$ 9,764	$ 9,764	$ 10,104	$ 8,850	$ 6,386	$ 5,663	$ 4,792	$ 1,960	$ -	$ 70,279
Mine Maintenance	$ 1,794	$ 5,041	$ 4,563	$ 4,563	$ 4,563	$ 4,276	$ 4,085	$ 3,512	$ 2,513	$ 1,065	$ -	$ 35,976
Engineering	$ 281	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 468	$ 234	$ -	$ 5,484
Geology	$ 319	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 345	$ 173	$ -	$ 4,557
Total	$ 5,817	$ 15,788	$ 15,502	$ 15,502	$ 15,842	$ 14,301	$ 11,645	$ 10,349	$ 8,118	$ 3,432	$ -	$ 116,295

TABLE 19-16: SENSITIVITY CASE MINE ANNUAL PERSONNEL COSTS
($000's USD)
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

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	Total
Mine Manager	$ 134	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 223	$ 112	$ -	$ 2,923
Mine Clerk	$ 38	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 32	$ -	$ 836
Mine Shift Foremen	$ 255	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 425	$ 213	$ -	$ 5,570
Mine Trainer	$ 51	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 43	$ -	$ 1,114
Blaster	$ 115	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 191	$ 96	$ -	$ 2,505
Blaster's Helper	$ 191	$ 319	$ 319	$ 319	$ 319	$ 319	$ 319	$ 319	$ 319	$ 319	$ 319	$ 319	$ 319	$ 159	$ -	$ 4,176
Mine Production																
Loading Operators	$ 344	$ 1,147	$ 1,147	$ 1,147	$ 1,147	$ 1,147	$ 1,147	$ 1,147	$ 1,147	$ 1,147	$ 1,147	$ 1,147	$ 574	$ 191	$ -	$ 13,728
Haul Truck Operators	$ 1,224	$ 4,080	$ 4,420	$ 4,420	$ 4,420	$ 4,760	$ 5,100	$ 5,440	$ 5,780	$ 5,780	$ 5,440	$ 5,440	$ 3,740	$ 1,020	$ -	$ 61,064
Drill Operators	$ 287	$ 1,530	$ 1,530	$ 1,530	$ 1,625	$ 1,625	$ 1,625	$ 1,625	$ 1,434	$ 1,434	$ 1,434	$ 1,434	$ 860	$ 191	$ -	$ 18,164
Support Equipment Operators	$ 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	$ 510	$ -	$ 21,182
Total Mine Operating	$ 3,251	$ 9,764	$ 10,104	$ 10,104	$ 10,200	$ 10,540	$ 10,880	$ 11,220	$ 11,368	$ 11,368	$ 11,028	$ 11,028	$ 7,841	$ 2,566	$ -	$ 131,261
Mine Maintenance																
Maintenance Superintendent	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ -	$ 2,088
Light Vehicle Mechanics	$ 48	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ 80	$ 80	$ 80	$ 80	$ 40	$ -	$ 1,602
Mobile Equipment Mechanics	$ 975	$ 3,346	$ 2,772	$ 2,772	$ 2,772	$ 2,772	$ 2,772	$ 3,920	$ 4,015	$ 4,015	$ 3,824	$ 3,824	$ 2,677	$ 765	$ -	$ 41,223
Mobile Equipment Welders	$ 143	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 239	$ 120	$ -	$ 3,132
Mobile Equipment Servicemen	$ 48	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 40	$ -	$ 1,044
Tiremen	$ 48	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ 80	$ 80	$ 80	$ 80	$ 40	$ -	$ 1,602
Shop Laborers	$ 83	$ 276	$ 276	$ 276	$ 276	$ 276	$ 276	$ 276	$ 138	$ 138	$ 138	$ 138	$ 138	$ 69	$ -	$ 2,778
Maintenance Planner	$ 48	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 40	$ -	$ 1,044
Service, Fuel, & Lube	$ 191	$ 638	$ 638	$ 638	$ 638	$ 638	$ 638	$ 638	$ 319	$ 319	$ 319	$ 319	$ 319	$ 159	$ -	$ 6,408
Total Mine Maintenance	$ 1,680	$ 5,137	$ 4,563	$ 4,563	$ 4,563	$ 4,563	$ 4,563	$ 5,710	$ 5,190	$ 5,190	$ 4,998	$ 4,998	$ 3,851	$ 1,352	$ -	$ 60,921
Engineering																
Chief Engineer	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ -	$ 2,088
Mine Surveyors	$ 64	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 106	$ 106	$ 106	$ 106	$ 106	$ 53	$ -	$ 2,137
Surveyor Helper	$ 41	$ 138	$ 138	$ 138	$ 138	$ 138	$ 138	$ 138	$ 69	$ 69	$ 69	$ 69	$ 69	$ 35	$ -	$ 1,389
Mine Engineer	$ 80	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 66	$ -	$ 1,741
Total Engineering	$ 281	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 468	$ 468	$ 468	$ 468	$ 468	$ 234	$ -	$ 7,355
Mine Geology																
Chief Geologist	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ -	$ 2,088
Ore Control Geologist	$ 128	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 106	$ 106	$ 106	$ 106	$ 106	$ 53	$ -	$ 2,200
Sampler	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ 80	$ 80	$ 80	$ 80	$ 40	$ -	$ 1,650
Total Geology	$ 319	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 345	$ 345	$ 345	$ 345	$ 345	$ 173	$ -	$ 5,938
Total Mine Operations Workforce																
Mine Operations	$ 3,251	$ 9,764	$ 10,104	$ 10,104	$ 10,200	$ 10,540	$ 10,880	$ 11,220	$ 11,368	$ 11,368	$ 11,028	$ 11,028	$ 7,841	$ 2,566	$ -	$ 131,261
Mine Maintenance	$ 1,680	$ 5,137	$ 4,563	$ 4,563	$ 4,563	$ 4,563	$ 4,563	$ 5,710	$ 5,190	$ 5,190	$ 4,998	$ 4,998	$ 3,851	$ 1,352	$ -	$ 60,921
Engineering	$ 281	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 468	$ 468	$ 468	$ 468	$ 468	$ 234	$ -	$ 7,355
Geology	$ 319	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 345	$ 345	$ 345	$ 345	$ 345	$ 173	$ -	$ 5,938
Total	$ 5,530	$ 16,075	$ 15,842	$ 15,842	$ 15,937	$ 16,277	$ 16,617	$ 18,104	$ 17,371	$ 17,371	$ 16,840	$ 16,840	$ 12,505	$ 4,324	$ -	$ 205,475

19.2 Mine Capital Costs

The mine capital cost is estimated based on the quantity of equipment required to achieve the mine production and the costs for equipment from equipment procurement firms, estimation guides, and recent project data with which MDA has been involved. TABLE 19-17 shows the estimated Base case mine capital requirements by year. The Base case initial mine capital is estimated to be US$68.4 million (total of year -1 and year 1). This does not include pre-mining capital of US$11.3 million based on the mine operating cost for year -1. Sustaining capital for the Base case is estimated to be US$7.9 million. Details for the Base case capital are given in the following sections.

The Sensitivity case capital cost estimate is shown in TABLE 19-18.

TABLE 19-17: BASE CASE MINE ANNUAL CAPITAL COSTS
($000's USD)
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Primary Mining Equipment	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Total
Atlas Copco PV235	$ 5,670	$ 5,670	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 11,340
21cm Hyd. Shovel	$ 15,392	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 15,392
12cm FEL	$ -	$ 1,540	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,540
180t Haul Truck	$ 16,947	$ 8,474	$ -	$ 2,824	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 28,245
140t Haul Truck	$ -	$ 6,237	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 6,237
Support Equipment												
300 Kw Dozer (D9)	$ 1,671	$ -	$ -	$ -	$ -	$ 1,671	$ -	$ -	$ -	$ -	$ -	$ 3,342
4.9 m Motor Grader (16H)	$ 727	$ -	$ -	$ -	$ -	$ 727	$ -	$ -	$ -	$ -	$ -	$ 1,454
Water Truck - 45,000 Liter	$ 724	$ -	$ -	$ -	$ -	$ 724	$ -	$ -	$ -	$ -	$ -	$ 1,448
RTD Dozer (834H)	$ 818	$ -	$ -	$ -	$ -	$ 818	$ -	$ -	$ -	$ -	$ -	$ 1,636
Pit Pumps	$ 68	$ -	$ -	$ -	$ -	$ 68	$ -	$ -	$ -	$ -	$ -	$ 136
36 ton Crane	$ 330	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 330
Flatbed	$ 52	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 52
Blasting												
Explosives Truck	$ 187	$ -	$ -	$ -	$ -	$ 187	$ -	$ -	$ -	$ -	$ -	$ 374
Skid Loader	$ 39	$ -	$ -	$ -	$ -	$ 39	$ -	$ -	$ -	$ -	$ -	$ 78
Mine Maintenance												
Lube/Fuel Truck	$ 193	$ -	$ -	$ -	$ -	$ 193	$ -	$ -	$ -	$ -	$ -	$ 386
Mechanics Truck	$ 187	$ -	$ -	$ -	$ -	$ 187	$ -	$ -	$ -	$ -	$ -	$ 374
Forklift	$ 137	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 137
Other Mine Capital												
Light Plant	$ 54	$ -	$ 27	$ -	$ 27	$ -	$ 27	$ -	$ -	$ -	$ -	$ 135
ANFO Storage Bins	$ 77	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 77
Powder Magazines	$ 9	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 9
Cap Magazine	$ 6	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 6
Mobile Radios	$ 24	$ 9	$ -	$ 1	$ -	$ 5	$ -	$ -	$ -	$ -	$ -	$ 39
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	$ -	$ -	$ -	$ -	$ -	$ -	$ 854
Total Mining Capital	$ 46,501	$ 21,930	$ 27	$ 2,826	$ 413	$ 4,619	$ 27	$ -	$ -	$ -	$ -	$ 76,342

TABLE 19-18: SENSITIVITY CASE MINE ANNUAL CAPITAL COSTS
($000's USD)
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Total
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	$ 16,947	$ 11,298	$ -	$ -	$ -	$ 2,825	$ 2,825	$ 2,825	$ 2,825	$ -	$ -	$ -	$ -	$ -	$ -	$ 39,545
140t Haul Truck	$ -	$ 6,237	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 6,237
Support Equipment																
300 Kw Dozer (D9)	$ 1,671	$ -	$ -	$ -	$ -	$ 1,671	$ -	$ -	$ -	$ -	$ 1,671	$ -	$ -	$ -	$ -	$ 5,013
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
Pit Pumps	$ 68	$ -	$ -	$ -	$ -	$ 68	$ -	$ -	$ -	$ -	$ 68	$ -	$ -	$ -	$ -	$ 204
36 ton Crane	$ 330	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 330
Flatbed	$ 52	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 52
Blasting																
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	$ 24	$ 10	$ -	$ -	$ 1	$ 7	$ 2	$ 1	$ 1	$ 1	$ 6	$ 1	$ -	$ -	$ -	$ 54
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	$ 46,501	$ 24,755	$ 27	$ -	$ 3,249	$ 10,281	$ 5,689	$ 2,826	$ 3,239	$ 2,836	$ 7,482	$ 2,836	$ 27	$ -	$ -	$ 109,748

19.2.1 Base Case Major Mining Equipment

Capital for major mining equipment is shown in TABLE 19-17 and discussed in the following subsections. 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 US$2,835,000 each (including shipping and commissioning). The cost of the drills was provided by EMG LLC.

Blasting operations require the use of a truck to deliver bulk explosive to the hole (US$187,000) and a skid loader (US$38,016) to help with stemming of holes. Additional blasting capital includes ANFO/Emulsion storage bins (US$38,500), powder magazines (US$8,400), and a cap magazine (US$5,250).

19.2.2 Loading

Capital costs for loading equipment have been quoted by EMG LLC and includes 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 1 at a cost of $1,539,200.

19.2.3 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 10 trucks would be purchased starting with six in year -1, three in year 1, and one in year 3. The cost of the 789C trucks is estimated to be US$2,824,500 each.

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

19.2.4 Base Case 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 (US$835,120 each quoted by EMG LLC);
- One Caterpillar 16H motor grader (US$726,600 quoted by EMG LLC);
- One Caterpillar 773B with a 45K liter water truck (US$723,060 quoted by EMG LLC);
- One Caterpillar 834H rubber tire dozer (US$817,400 quoted by EMG LLC);
- One 36 tonne capacity crane (US$329,600 quoted by EMG LLC);
- One Caterpillar 321DL excavator (US$177,345 quoted by EMG LLC);
- One low-boy trailer complete with a used 60t haul truck to tow it (US$1,230,900);
- One flatbed truck US$51,450);
- Two pit pumps (US$33,690 each);
- One rock breaker to be attached to the 321DL excavator as needed (US$30,975); and

- Four light plants ($13,423 US).

19.2.5 Base Case Mine Maintenance

Capital for mine maintenance equipment includes a fuel/lube truck (US$192,610), mechanics truck (US$187,000), and a tire truck (US$137,000). Shop facilities are estimated at US$1,500,000 and an additional US$262,500 is included for shop equipment.

19.2.6 Base Case Mine Facilities

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

19.2.7 Base Case Light Vehicles

Capital for light vehicles is shown in TABLE 19-19.

TABLE 19-19: MINE LIGHT VEHICLE INITIAL CAPITAL (USD)
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

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

19.2.8 Base Case Other Mine Capital

Other miscellaneous capital includes mobile radios for mobile equipment (US$1,000 per unit), ambulance and fire equipment (US$150,000), engineering and office equipment (US$150,000), water storage for dust suppression (US$97,900), and other unspecified miscellaneous equipment (US$105,000).

19.3 Mine Operating Cost

Annual mine operating costs have been built based on estimated personnel requirements and equipment hourly costs. TABLE 19-20 and TABLE 19-21 summarize the annual mine operating costs for the Base and Sensitivity cases, respectively. The costs are provided based on functionality (mine general services, mine maintenance, engineering, geology, drilling, blasting, loading, hauling, and support). The total average mining cost is estimated to be $1.50/t and $1.41 for the Base and Sensitivity cases, respectively. Note that tables include the mining cost for year -1; however, this is capitalized in the cash flow estimate by Tetra Tech.

The following subsections describe the operating cost estimate for the Base case by functionality.

TABLE 19-20: BASE CASE ANNUAL MINE OPERATING COSTS ($000'S USD)
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Mined Production	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Total
Ore to Mill	k tonnes	-	6,628	6,770	6,770	6,770	6,789	6,770	6,770	6,327	178	-	53,772
Ore to Stkpl	k tonnes	537	479	2,110	3,151	-	-	-	-	-	-	-	6,278
Total Ore Mined	k tonnes	537	7,108	8,880	9,921	6,770	6,789	6,770	6,770	6,327	178	-	60,050
Rehandel	k tonnes	-	160	-	-	-	-	-	-	443	5,674	-	6,278
Waste to Dumps	k tonnes	5,583	27,898	25,095	23,519	29,948	17,573	8,717	3,685	504	2	-	142,524
Total Tonnes Mined	k tonnes	6,120	35,006	33,975	33,440	36,718	24,362	15,487	10,455	6,830	181	-	202,573
Total Tonnes Moved	k tonnes	6,120	35,166	33,975	33,440	36,718	24,362	15,487	10,455	7,274	5,855	-	208,851
Strip Ratio	w:o	10.39	3.93	2.83	2.37	4.42	2.59	1.29	0.54	0.08	0.01		2.37
Mining Costs													
Mine General Service	K USD	$ 534	$ 912	$ 912	$ 912	$ 912	$ 912	$ 912	$ 912	$ 912	$ 456	$ -	$ 8,283
Mine Maintenance	K USD	$ 604	$ 1,674	$ 1,674	$ 1,674	$ 1,674	$ 1,674	$ 1,674	$ 1,674	$ 1,057	$ 529	$ -	$ 13,906
Engineering	K USD	$ 294	$ 668	$ 668	$ 668	$ 668	$ 668	$ 668	$ 668	$ 492	$ 246	$ -	$ 5,707
Geology	K USD	$ 332	$ 556	$ 556	$ 556	$ 556	$ 556	$ 556	$ 556	$ 370	$ 185	$ -	$ 4,779
Drilling	K USD	$ 1,168	$ 5,731	$ 5,629	$ 5,577	$ 5,900	$ 4,108	$ 2,754	$ 1,971	$ 1,327	$ 201	$ -	$ 34,366
Blasting	K USD	$ 2,139	$ 10,698	$ 10,401	$ 10,247	$ 11,190	$ 7,637	$ 5,085	$ 3,638	$ 2,596	$ 368	$ -	$ 63,999
Loading	K USD	$ 1,383	$ 5,500	$ 5,938	$ 5,629	$ 5,911	$ 5,314	$ 2,673	$ 1,823	$ 1,475	$ 1,087	$ -	$ 36,732
Hauling	K USD	$ 3,374	$ 16,178	$ 16,475	$ 16,500	$ 18,378	$ 15,277	$ 11,007	$ 8,392	$ 6,349	$ 1,764	$ -	$ 113,695
Mine Support	K USD	$ 1,496	$ 3,901	$ 3,896	$ 3,896	$ 3,896	$ 3,901	$ 3,556	$ 3,556	$ 3,120	$ 1,563	$ -	$ 32,779
Total Mine Cost	K USD	$ 11,323	$ 45,817	$ 46,149	$ 45,657	$ 49,084	$ 40,046	$ 28,884	$ 23,189	$ 17,699	$ 6,398	$ -	$ 314,247
Mine Cost per Tonne Mined													
Mine General Service	$/t	$ 0.09	$ 0.03	$ 0.03	$ 0.03	$ 0.02	$ 0.04	$ 0.06	$ 0.09	$ 0.13	$ 2.52	$ -	$ 0.04
Mine Maintenance	$/t	$ 0.10	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.07	$ 0.11	$ 0.16	$ 0.15	$ 2.93	$ -	$ 0.07
Engineering	$/t	$ 0.05	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ 0.04	$ 0.06	$ 0.07	$ 1.36	$ -	$ 0.03
Geology	$/t	$ 0.05	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.04	$ 0.05	$ 0.05	$ 1.02	$ -	$ 0.02
Drilling	$/t	$ 0.19	$ 0.16	$ 0.17	$ 0.17	$ 0.16	$ 0.17	$ 0.18	$ 0.19	$ 0.19	$ 1.11	$ -	$ 0.17
Blasting	$/t	$ 0.35	$ 0.31	$ 0.31	$ 0.31	$ 0.30	$ 0.31	$ 0.33	$ 0.35	$ 0.38	$ 2.04	$ -	$ 0.32
Loading	$/t	$ 0.23	$ 0.16	$ 0.17	$ 0.17	$ 0.16	$ 0.22	$ 0.17	$ 0.17	$ 0.22	$ 6.02	$ -	$ 0.18
Hauling	$/t	$ 0.55	$ 0.46	$ 0.48	$ 0.49	$ 0.50	$ 0.63	$ 0.71	$ 0.80	$ 0.93	$ 9.76	$ -	$ 0.56
Mine Support	$/t	$ 0.24	$ 0.11	$ 0.11	$ 0.12	$ 0.11	$ 0.16	$ 0.23	$ 0.34	$ 0.46	$ 8.65	$ -	$ 0.16
Total Mine Cost	$/t	$ 1.85	$ 1.31	$ 1.36	$ 1.37	$ 1.34	$ 1.64	$ 1.87	$ 2.22	$ 2.59	$ 35.41	$ -	$ 1.55
Mine Cost per Tonne Moved													
Mine General Service	$/t	$ 0.09	$ 0.03	$ 0.03	$ 0.03	$ 0.02	$ 0.04	$ 0.06	$ 0.09	$ 0.13	$ 0.08	$ -	$ 0.04
Mine Maintenance	$/t	$ 0.10	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.07	$ 0.11	$ 0.16	$ 0.15	$ 0.09	$ -	$ 0.07
Engineering	$/t	$ 0.05	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ 0.04	$ 0.06	$ 0.07	$ 0.04	$ -	$ 0.03
Geology	$/t	$ 0.05	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.04	$ 0.05	$ 0.05	$ 0.03	$ -	$ 0.02
Drilling	$/t	$ 0.19	$ 0.16	$ 0.17	$ 0.17	$ 0.16	$ 0.17	$ 0.18	$ 0.19	$ 0.18	$ 0.03	$ -	$ 0.16
Blasting	$/t	$ 0.35	$ 0.30	$ 0.31	$ 0.31	$ 0.30	$ 0.31	$ 0.33	$ 0.35	$ 0.36	$ 0.06	$ -	$ 0.31
Loading	$/t	$ 0.23	$ 0.16	$ 0.17	$ 0.17	$ 0.16	$ 0.22	$ 0.17	$ 0.17	$ 0.20	$ 0.19	$ -	$ 0.18
Hauling	$/t	$ 0.55	$ 0.46	$ 0.48	$ 0.49	$ 0.50	$ 0.63	$ 0.71	$ 0.80	$ 0.87	$ 0.30	$ -	$ 0.54
Mine Support	$/t	$ 0.24	$ 0.11	$ 0.11	$ 0.12	$ 0.11	$ 0.16	$ 0.23	$ 0.34	$ 0.43	$ 0.27	$ -	$ 0.16
Total Mine Cost	$/t	$ 1.85	$ 1.30	$ 1.36	$ 1.37	$ 1.34	$ 1.64	$ 1.87	$ 2.22	$ 2.43	$ 1.09	$ -	$ 1.50

TABLE 19-21:SENSITIVITY CASE ANNUAL MINE OPERATING COSTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Mined Tonnes	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Total
Ore to Mill	k tonnes	-	10,806	10,404	9,883	10,950	10,980	10,950	10,950	4,260	8,440	10,950	8,447	10,950	7,685	-	125,655
Ore to Stkpl	k tonnes	897	692	5,495	2,064	1,182	-	2,126	1,064	-	-	-	-	-	-	-	13,520
Total Ore Mined	k tonnes	897	11,498	15,900	11,947	12,132	10,980	13,076	12,014	4,260	8,440	10,950	8,447	10,950	7,685	-	139,175
Rehandel	k tonnes	-	174	546	1,067	-	-	-	-	6,690	2,540	-	2,503	-	-	-	13,520
Waste to Dumps	k tonnes	5,224	24,287	19,608	24,002	25,294	25,777	23,112	24,382	27,282	26,476	20,483	27,283	8,688	472	-	282,369
Total Tonnes Mined	k tonnes	6,120	35,784	35,507	35,950	37,426	36,757	36,188	36,396	31,543	34,916	31,433	35,730	19,638	8,157	-	421,545
Total Tonnes Moved	k tonnes	6,120	35,958	36,053	37,017	37,426	36,757	36,188	36,396	38,232	37,456	31,433	38,233	19,638	8,157	-	435,064
Strip Ratio	w:o	5.83	2.11	1.23	2.01	2.08	2.35	1.77	2.03	6.40	3.14	1.87	3.23	0.79	0.06		2.03
Mining Costs																	
Mine General Service	K USD	$ 534	$ 912	$ 912	$ 912	$ 912	$ 912	$ 912	$ 912	$ 912	$ 912	$ 912	$ 912	$ 863	$ 432	$ -	$ 11,857
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	$ 525	$ -	$ 18,122
Engineering	K USD	$ 294	$ 668	$ 668	$ 668	$ 668	$ 668	$ 668	$ 668	$ 492	$ 492	$ 492	$ 492	$ 492	$ 246	$ -	$ 7,676
Geology	K USD	$ 332	$ 556	$ 556	$ 556	$ 556	$ 556	$ 556	$ 556	$ 370	$ 370	$ 370	$ 370	$ 370	$ 185	$ -	$ 6,260
Drilling	K USD	$ 1,053	$ 5,808	$ 5,781	$ 5,824	$ 6,065	$ 5,999	$ 5,943	$ 5,964	$ 5,198	$ 5,531	$ 5,188	$ 5,611	$ 3,259	$ 1,131	$ -	$ 68,357
Blasting	K USD	$ 2,139	$ 10,922	$ 10,842	$ 10,969	$ 11,394	$ 11,201	$ 11,038	$ 11,097	$ 9,702	$ 10,672	$ 9,670	$ 10,784	$ 6,157	$ 2,661	$ -	$ 129,249
Loading	K USD	$ 1,211	$ 5,907	$ 5,799	$ 6,102	$ 6,002	$ 5,949	$ 5,887	$ 5,914	$ 6,153	$ 6,166	$ 5,266	$ 6,153	$ 4,494	$ 1,239	$ -	$ 72,241
Hauling	K USD	$ 3,346	$ 16,005	$ 16,570	$ 17,266	$ 18,289	$ 19,465	$ 20,442	$ 23,309	$ 24,447	$ 25,109	$ 20,622	$ 23,748	$ 16,436	$ 6,774	$ -	$ 251,830
Mine Support	K USD	$ 1,496	$ 3,901	$ 3,896	$ 3,896	$ 3,896	$ 3,901	$ 3,896	$ 3,896	$ 3,896	$ 3,901	$ 3,896	$ 3,896	$ 3,178	$ 1,373	$ -	$ 48,915
Total Mine Cost	K USD	$ 11,008	$ 46,352	$ 46,696	$ 47,867	$ 49,455	$ 50,324	$ 51,014	$ 53,988	$ 52,228	$ 54,211	$ 47,473	$ 53,024	$ 36,300	$ 14,567	$ -	$ 614,508
Mine Cost per Tonne Mined																	
Mine General Service	$/t	$ 0.09	$ 0.03	$ 0.03	$ 0.03	$ 0.02	$ 0.02	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.04	$ 0.05	$ -	$ 0.03
Mine Maintenance	$/t	$ 0.10	$ 0.05	$ 0.05	$ 0.05	$ 0.04	$ 0.05	$ 0.05	$ 0.05	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.05	$ 0.06	$ -	$ 0.04
Engineering	$/t	$ 0.05	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.01	$ 0.02	$ 0.01	$ 0.03	$ 0.03	$ -	$ 0.02
Geology	$/t	$ 0.05	$ 0.02	$ 0.02	$ 0.02	$ 0.01	$ 0.02	$ 0.02	$ 0.02	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.02	$ 0.02	$ -	$ 0.01
Drilling	$/t	$ 0.17	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.17	$ 0.16	$ 0.17	$ 0.14	$ -	$ 0.16
Blasting	$/t	$ 0.35	$ 0.31	$ 0.31	$ 0.31	$ 0.30	$ 0.30	$ 0.31	$ 0.30	$ 0.31	$ 0.31	$ 0.31	$ 0.30	$ 0.31	$ 0.33	$ -	$ 0.31
Loading	$/t	$ 0.20	$ 0.17	$ 0.16	$ 0.17	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.20	$ 0.18	$ 0.17	$ 0.17	$ 0.23	$ 0.15	$ -	$ 0.17
Hauling	$/t	$ 0.55	$ 0.45	$ 0.47	$ 0.48	$ 0.49	$ 0.53	$ 0.56	$ 0.64	$ 0.78	$ 0.72	$ 0.66	$ 0.66	$ 0.84	$ 0.83	$ -	$ 0.60
Mine Support	$/t	$ 0.24	$ 0.11	$ 0.11	$ 0.11	$ 0.10	$ 0.11	$ 0.11	$ 0.11	$ 0.12	$ 0.11	$ 0.12	$ 0.11	$ 0.16	$ 0.17	$ -	$ 0.12
Total Mine Cost	$/t	$ 1.80	$ 1.30	$ 1.32	$ 1.33	$ 1.32	$ 1.37	$ 1.41	$ 1.48	$ 1.66	$ 1.55	$ 1.51	$ 1.48	$ 1.85	$ 1.79	$ -	$ 1.46
Mine Cost per Tonne Moved																	
Mine General Service	$/t	$ 0.09	$ 0.03	$ 0.03	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ 0.03	$ 0.02	$ 0.02	$ 0.03	$ 0.02	$ 0.04	$ 0.05	$ -	$ 0.03
Mine Maintenance	$/t	$ 0.10	$ 0.05	$ 0.05	$ 0.05	$ 0.04	$ 0.05	$ 0.05	$ 0.05	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.05	$ 0.06	$ -	$ 0.04
Engineering	$/t	$ 0.05	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.01	$ 0.01	$ 0.02	$ 0.01	$ 0.03	$ 0.03	$ -	$ 0.02
Geology	$/t	$ 0.05	$ 0.02	$ 0.02	$ 0.02	$ 0.01	$ 0.02	$ 0.02	$ 0.02	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.02	$ 0.02	$ -	$ 0.01
Drilling	$/t	$ 0.17	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.14	$ 0.15	$ 0.17	$ 0.15	$ 0.17	$ 0.14	$ -	$ 0.16
Blasting	$/t	$ 0.35	$ 0.30	$ 0.30	$ 0.30	$ 0.30	$ 0.30	$ 0.31	$ 0.30	$ 0.25	$ 0.28	$ 0.31	$ 0.28	$ 0.31	$ 0.33	$ -	$ 0.30
Loading	$/t	$ 0.20	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.16	$ 0.17	$ 0.16	$ 0.23	$ 0.15	$ -	$ 0.17
Hauling	$/t	$ 0.55	$ 0.45	$ 0.46	$ 0.47	$ 0.49	$ 0.53	$ 0.56	$ 0.64	$ 0.64	$ 0.67	$ 0.66	$ 0.62	$ 0.84	$ 0.83	$ -	$ 0.58
Mine Support	$/t	$ 0.24	$ 0.11	$ 0.11	$ 0.11	$ 0.10	$ 0.11	$ 0.11	$ 0.11	$ 0.10	$ 0.10	$ 0.12	$ 0.10	$ 0.16	$ 0.17	$ -	$ 0.11
Total Mine Cost	$/t	$ 1.80	$ 1.29	$ 1.30	$ 1.29	$ 1.32	$ 1.37	$ 1.41	$ 1.48	$ 1.37	$ 1.45	$ 1.51	$ 1.39	$ 1.85	$ 1.79	$ -	$ 1.41

19.3.1 Base Case Drilling Costs

Drilling operating costs are provided in TABLE 19-22. The average life-of-mine drilling cost is estimated to be $0.17/t mined. This includes maintenance labor allocated to drill maintenance.

TABLE 19-22:BASE CASE ANNUAL DRILLING OPERATING COSTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

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**	**Total**
Number of Holes - Ore	holes	1,096	14,504	18,120	20,245	13,815	13,853	13,815	13,815	12,911	364	-	122,538
Drill Meters - Ore	meters	8,550	113,132	141,339	157,911	107,757	108,052	107,757	107,757	100,703	2,838	-	955,795
Production Hours - Ore	hrs	312	4,129	5,158	5,763	3,932	3,943	3,932	3,932	3,675	104	-	34,881
Operational Efficiency - Ore	%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	0%	
Operating Hrs - Ore	hrs	367.10	4,857.22	6,068.24	6,779.72	4,626.42	4,639.10	4,626.42	4,626.42	4,323.60	121.83	-	41,036
Number of Holes - Waste	holes	11,393	56,929	51,208	47,993	61,112	35,860	17,787	7,520	1,028	5	-	290,835
Drill Meters - Waste	meters	88,865	444,047	399,425	374,343	476,672	279,710	138,739	58,659	8,016	38	-	2,268,514
Production Hours - Waste	hrs	3,243	16,205	14,577	13,661	17,396	10,208	5,063	2,141	293	1	-	82,787
Operational Efficiency - Waste	%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	0%	
Operating Hrs - Waste	hrs	3,815.34	19,064.71	17,148.88	16,072.04	20,465.43	12,009.05	5,956.60	2,518.45	344.15	1.64	-	97,396
Number of Holes - Total	holes	12,489	71,433	69,329	68,238	74,927	49,713	31,602	21,335	13,938	369	-	413,373
Drill Meters - Total	meters	97,416	557,180	540,763	532,254	584,429	387,762	246,495	166,415	108,719	2,876	-	3,224,309
Production Hours - Total	hrs	3,555	20,334	19,735	19,424	21,328	14,151	8,996	6,073	3,968	105	-	117,667
Operating Hrs - Total	hrs	4,182	23,922	23,217	22,852	25,092	16,648	10,583	7,145	4,668	123	-	138,432
Drill Availability	%	85%	85%	85%	85%	85%	85%	85%	85%	85%	85%	0%	
Required Drills - Calculated	#	1.02	3.85	3.74	3.68	4.04	2.68	1.71	1.15	0.75	0.02	-	
Required Drills - Rounded	#	2	4	4	4	4	3	2	2	1	1	0	
Cumulative Hours - Drill #1	hrs	4,182	10,163	15,967	21,680	27,953	33,502	38,794	42,366	47,034	47,158	-	
Cumulative Hours - Drill #2	hrs		5,980	11,785	17,498	23,771	29,320	34,612	38,184	42,852	42,975	-	
Cumulative Hours - Drill #3	hrs		5,980	11,785	17,498	23,771	29,320	34,612	38,184	42,852			
Operating Costs													
Fuel Consumption (KL)	KL	294	1,684	1,635	1,609	1,767	1,172	745	503	329	9	-	9,746
Fuel Cost	K USD	$ 164.90	$ 943.16	$ 915.37	$ 900.96	$ 989.28	$ 656.38	$ 417.25	$ 281.70	$ 184.03	$ 4.87	$ -	$ 5,458
Lube & Oil	K USD	$ 48.43	$ 277.02	$ 268.85	$ 264.62	$ 290.56	$ 192.79	$ 122.55	$ 82.74	$ 54.05	$ 1.43	$ -	$ 1,603
Undercarriage	K USD	$ 20.91	$ 119.61	$ 116.09	$ 114.26	$ 125.46	$ 83.24	$ 52.92	$ 35.72	$ 23.34	$ 0.62	$ -	$ 692
Drill Bits & Steel	K USD	$ 217.84	$ 1,245.93	$ 1,209.22	$ 1,190.20	$ 1,306.87	$ 867.09	$ 551.20	$ 372.13	$ 243.11	$ 6.43	$ -	$ 7,210
Total Consumables	K USD	$ 452.08	$ 2,585.72	$ 2,509.53	$ 2,470.04	$ 2,712.17	$ 1,799.49	$ 1,143.92	$ 772.29	$ 504.54	$ 13.35	$ -	$ 14,963
Parts / MARC Cost	K USD	$ 151.57	$ 866.93	$ 841.39	$ 828.15	$ 909.33	$ 603.33	$ 383.53	$ 258.93	$ 169.16	$ 4.47	$ -	$ 5,017
Maintenance Labor	K USD	$ 219.90	$ 748.90	$ 748.90	$ 748.90	$ 748.90	$ 557.70	$ 462.10	$ 366.50	$ 270.90	$ 87.65	$ -	$ 4,960
Total Maintenance Allocation	K USD	$ 371.47	$ 1,615.83	$ 1,590.29	$ 1,577.05	$ 1,658.23	$ 1,161.03	$ 845.63	$ 625.43	$ 440.06	$ 92.12	$ -	$ 9,977
Operator Wages & Burden	K USD	$ 344.16	$ 1,529.60	$ 1,529.60	$ 1,529.60	$ 1,529.60	$ 1,147.20	$ 764.80	$ 573.60	$ 382.40	$ 95.60	$ -	$ 9,426
Total Drilling Cost	K USD	$ 1,167.71	$ 5,731.15	$ 5,629.42	$ 5,576.69	$ 5,900.00	$ 4,107.72	$ 2,754.35	$ 1,971.32	$ 1,326.99	$ 201.07	$ -	$ 34,366
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
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
Undercarriage	$/t	$ 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
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
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
Maintenance Labor	$/t	$ 0.04	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.02	$ 0.03	$ 0.04	$ 0.04	$ 0.49	$ -	$ 0.02
Total Maintenance Allocation	$/t	$ 0.06	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.06	$ 0.06	$ 0.51	$ -	$ 0.05
Operator Wages & Burden	$/t	$ 0.06	$ 0.04	$ 0.05	$ 0.05	$ 0.04	$ 0.05	$ 0.05	$ 0.05	$ 0.06	$ 0.53	$ -	$ 0.05
Total Drilling Cost	$/t	$ 0.19	$ 0.16	$ 0.17	$ 0.17	$ 0.16	$ 0.17	$ 0.18	$ 0.19	$ 0.19	$ 1.11	$ -	$ 0.17

19.3.2 Base Case Blasting Costs

Blasting costs are provided in TABLE 19-23. The average life-of-mine blasting cost is estimated to be $0.32/t mined.

TABLE 19-23: BASE CASE MINE ANNUAL BLASTING OPERATING COST
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Ore Blasting Consumables	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Total
Holes Loaded	holes	1,096	14,504	18,120	20,245	13,815	13,853	13,815	13,815	12,911	364	-	122,538
Loaded Meters	meters	4,531	59,950	74,898	83,679	57,102	57,258	57,102	57,102	53,364	1,504	-	506,490
AN Used	tonnes	130	1,723	2,153	2,405	1,641	1,646	1,641	1,641	1,534	43	-	14,557
ANFO Cost	K USD	$ 139	$ 1,835	$ 2,293	$ 2,561	$ 1,748	$ 1,753	$ 1,748	$ 1,748	$ 1,633	$ 46	$ -	$ 15,503
Fuel Used (KL)	KL	6	84	106	118	80	81	80	80	75	2	-	714
Fuel Cost (000's)	K USD	$ 4	$ 47	$ 59	$ 66	$ 45	$ 45	$ 45	$ 45	$ 42	$ 1	$ -	$ 400
Blasting Accessory	K USD	$ 12	$ 162	$ 202	$ 226	$ 154	$ 154	$ 154	$ 154	$ 144	$ 4	$ -	$ 1,365
Blasting Consumables - Ore	K USD	$ 154	$ 2,044	$ 2,553	$ 2,853	$ 1,947	$ 1,952	$ 1,947	$ 1,947	$ 1,819	$ 51	$ -	$ 17,268
Waste Blasting Consumables													
Holes Loaded	holes	11,393	56,929	51,208	47,993	61,112	35,860	17,787	7,520	1,028	5	-	290,835
Loaded Meters	meters	47,091	235,307	211,661	198,370	252,596	148,222	73,520	31,084	4,248	20	-	1,202,118
AN Used (Mt)	tonnes	1,353	6,763	6,083	5,701	7,260	4,260	2,113	893	122	1	-	34,550
AN Cost (000's)	K USD	$ 1,441	$ 7,202	$ 6,479	$ 6,072	$ 7,732	$ 4,537	$ 2,250	$ 951	$ 130	$ 1	$ -	$ 36,795
Fuel Used (KL)	KL	66	332	298	279	356	209	104	44	6	0	-	1,694
Fuel Cost (000's)	K USD	$ 37	$ 186	$ 167	$ 157	$ 199	$ 117	$ 58	$ 25	$ 3	$ 0	$ -	$ 948
Blasting Accessory	K USD	$ 127	$ 634	$ 570	$ 535	$ 681	$ 399	$ 198	$ 84	$ 11	$ 0	$ -	$ 3,240
Blasting Consumables - Waste	K USD	$ 1,605	$ 8,022	$ 7,216	$ 6,763	$ 8,612	$ 5,053	$ 2,506	$ 1,060	$ 145	$ 1	$ -	$ 40,984
Total of Blasting Consumables													
Holes Loaded	holes	12,489	71,433	69,329	68,238	74,927	49,713	31,602	21,335	13,938	369	-	413,373
Loaded Meters	meters	51,622	295,258	286,558	282,049	309,697	205,481	130,621	88,186	57,612	1,524	-	1,708,608
AN Used	tonnes	1,484	8,486	8,236	8,106	8,901	5,906	3,754	2,535	1,656	44	-	49,106
AN Cost	K USD	$ 1,580	$ 9,037	$ 8,771	$ 8,633	$ 9,479	$ 6,289	$ 3,998	$ 2,699	$ 1,763	$ 47	$ -	$ 52,298
Fuel Used	KL	73	416	404	397	436	289	184	124	81	2	-	2,407
Fuel Cost	K USD	$ 41	$ 233	$ 226	$ 223	$ 244	$ 162	$ 103	$ 70	$ 45	$ 1	$ -	$ 1,348
Blasting Accessory	K USD	$ 139	$ 796	$ 772	$ 760	$ 835	$ 554	$ 352	$ 238	$ 155	$ 4	$ -	$ 4,605
Total Blasting Consumables	K USD	$ 1,760	$ 10,066	$ 9,770	$ 9,616	$ 10,558	$ 7,005	$ 4,453	$ 3,007	$ 1,964	$ 52	$ -	$ 58,251
Wages & Salaries													
Blaster	K USD	114.72	191.2	191.2	191.2	191.2	191.2	191.2	191.2	191.2	95.6	0	$ 1,740
Blaster's Helper	K USD	191.28	318.8	318.8	318.8	318.8	318.8	318.8	318.8	318.8	159.4	0	$ 2,901
Equipment Costs													
Consumables	K USD	$ 46	$ 77	$ 77	$ 77	$ 77	$ 77	$ 77	$ 77	$ 77	$ 39	$ -	$ 704
Maintenance Allocation	K USD	$ 12	$ 19	$ 19	$ 19	$ 19	$ 19	$ 19	$ 19	$ 19	$ 10	$ -	$ 175
Total	K USD	$ 58	$ 97	$ 97	$ 97	$ 97	$ 97	$ 97	$ 97	$ 97	$ 48	$ -	$ 879
Summary													
Consumables	K USD	$ 1,760	$ 10,066	$ 9,770	$ 9,616	$ 10,558	$ 7,005	$ 4,453	$ 3,007	$ 1,964	$ 52	$ -	$ 58,251
Labor	K USD	$ 306	$ 510	$ 510	$ 510	$ 510	$ 510	$ 510	$ 510	$ 510	$ 255	$ -	$ 4,641
Equipment	K USD	$ 58	$ 97	$ 97	$ 97	$ 97	$ 97	$ 97	$ 97	$ 97	$ 48	$ -	$ 879
Outside Services	K USD	$ 15	$ 25	$ 25	$ 25	$ 25	$ 25	$ 25	$ 25	$ 25	$ 13	$ -	$ 228
Total	K USD	$ 2,139	$ 10,698	$ 10,401	$ 10,247	$ 11,190	$ 7,637	$ 5,085	$ 3,638	$ 2,596	$ 368	$ -	$ 63,999
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
Labor	$/t	$ 0.05	$ 0.01	$ 0.02	$ 0.02	$ 0.01	$ 0.02	$ 0.03	$ 0.05	$ 0.07	$ 1.41	$ -	$ 0.02
Equipment	$/t	$ 0.01	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.01	$ 0.01	$ 0.01	$ 0.27	$ -	$ 0.00
Outside Services	$/t	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.00	$ 0.07	$ -	$ 0.00
Total	$/t	$ 0.35	$ 0.31	$ 0.31	$ 0.31	$ 0.30	$ 0.31	$ 0.33	$ 0.35	$ 0.38	$ 2.04	$ -	$ 0.32

19.3.3 Base Case Loading Costs

Loading costs are provided in TABLE 19-24. The average life-of-mine loading cost is estimated to be $0.176/t moved. 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.

TABLE 19-24: BASE CASE ANNUAL LOADING OPERATING COST
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

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	Total
Fuel Consumption (KL)	KL	560	3,118	3,148	3,098	3,112	3,103	1,499	1,012	661	17	-	19,328
Fuel Cost	K USD	$ 313	$ 1,746	$ 1,763	$ 1,735	$ 1,743	$ 1,737	$ 840	$ 567	$ 370	$ 10	$ -	$ 10,824
Lube & Oil	K USD	$ 61	$ 342	$ 345	$ 340	$ 341	$ 340	$ 164	$ 111	$ 73	$ 2	$ -	$ 2,120
Undercarriage	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Wear Items & GET	K USD	$ 23	$ 130	$ 131	$ 129	$ 130	$ 129	$ 62	$ 42	$ 28	$ 1	$ -	$ 806
Total Consumables	K USD	$ 398	$ 2,218	$ 2,240	$ 2,204	$ 2,214	$ 2,207	$ 1,066	$ 720	$ 470	$ 12	$ -	$ 13,750
Parts / MARC Cost	K USD	$ 249	$ 1,385	$ 1,399	$ 1,376	$ 1,383	$ 1,378	$ 666	$ 450	$ 294	$ 8	$ -	$ 8,586
Total Maint. Allocation (no labor)	K USD	$ 647	$ 3,603	$ 3,638	$ 3,580	$ 3,596	$ 3,586	$ 1,732	$ 1,170	$ 764	$ 20	$ -	$ 22,336
992 Loaders													
Fuel Consumption (KL)	KL	-	324	403	233	413	211	-	-	39	502	-	2,125
Fuel Cost	K USD	$ -	$ 181	$ 226	$ 131	$ 232	$ 118	$ -	$ -	$ 22	$ 281	$ -	$ 1,190
Lube & Oil	K USD	$ -	$ 43	$ 53	$ 31	$ 55	$ 28	$ -	$ -	$ 5	$ 66	$ -	$ 281
Tires	K USD	$ -	$ 147	$ 183	$ 106	$ 188	$ 96	$ -	$ -	$ 18	$ 228	$ -	$ 966
Wear Items & GET	K USD	$ -	$ 4	$ 5	$ 3	$ 5	$ 3	$ -	$ -	$ 0	$ 6	$ -	$ 26
Total Consumables	K USD	$ -	$ 376	$ 468	$ 270	$ 479	$ 244	$ -	$ -	$ 45	$ 581	$ -	$ 2,463
Parts / MARC Cost	K USD	$ -	$ 103	$ 128	$ 74	$ 131	$ 67	$ -	$ -	$ 12	$ 159	$ -	$ 673
Total Maint. Allocation (no labor)	K USD	$ -	$ 478	$ 595	$ 344	$ 610	$ 311	$ -	$ -	$ 58	$ 740	$ -	$ 3,136
Maintenance Labor	K USD	$ 277	$ 462	$ 558	$ 558	$ 558	$ 462	$ 367	$ 271	$ 271	$ 135	$ -	$ 3,918
Operator Wages & Burden	K USD	$ 459	$ 956	$ 1,147	$ 1,147	$ 1,147	$ 956	$ 574	$ 382	$ 382	$ 191	$ -	$ 7,342
Total Loading Cost	K USD	$ 1,383	$ 5,500	$ 5,938	$ 5,629	$ 5,911	$ 5,314	$ 2,673	$ 1,823	$ 1,475	$ 1,087	$ -	$ 36,732
Loading Cost per Tonne Moved by Item													
Fuel Cost	$/t	$ 0.051	$ 0.055	$ 0.059	$ 0.056	$ 0.054	$ 0.076	$ 0.054	$ 0.054	$ 0.057	$ 1.609	$ -	$ 0.059
Lube & Oil	$/t	$ 0.010	$ 0.011	$ 0.012	$ 0.011	$ 0.011	$ 0.015	$ 0.011	$ 0.011	$ 0.011	$ 0.378	$ -	$ 0.012
Tires / Under Carriage	$/t	$ -	$ 0.004	$ 0.005	$ 0.003	$ 0.005	$ 0.004	$ -	$ -	$ 0.003	$ 1.262	$ -	$ 0.005
Wear Items & GET	$/t	$ 0.004	$ 0.004	$ 0.004	$ 0.004	$ 0.004	$ 0.005	$ 0.004	$ 0.004	$ 0.004	$ 0.038	$ -	$ 0.004
Total Consumables	$/t	$ 0.065	$ 0.074	$ 0.080	$ 0.074	$ 0.073	$ 0.101	$ 0.069	$ 0.069	$ 0.076	$ 3.286	$ -	$ 0.080
Parts / MARC Cost	$/t	$ 0.041	$ 0.043	$ 0.045	$ 0.043	$ 0.041	$ 0.059	$ 0.043	$ 0.043	$ 0.045	$ 0.922	$ -	$ 0.046
Maintenance Labor	$/t	$ 0.045	$ 0.013	$ 0.016	$ 0.017	$ 0.015	$ 0.019	$ 0.024	$ 0.026	$ 0.040	$ 0.750	$ -	$ 0.019
Operator Wages & Burden	$/t	$ 0.075	$ 0.027	$ 0.034	$ 0.034	$ 0.031	$ 0.039	$ 0.037	$ 0.037	$ 0.056	$ 1.058	$ -	$ 0.036
Total Loading Cost	$/t	$ 0.226	$ 0.157	$ 0.175	$ 0.168	$ 0.161	$ 0.218	$ 0.173	$ 0.174	$ 0.216	$ 6.016	$ -	$ 0.181
Cost per Tonne Moved	$/t	$ 0.226	$ 0.156	$ 0.175	$ 0.168	$ 0.161	$ 0.218	$ 0.173	$ 0.174	$ 0.203	$ 0.186	$ -	$ 0.176

19.3.4 Base Case Haulage Costs

Haulage costs are provided in TABLE 19-25. The average life-of-mine haulage cost is estimated to be $0.544/t moved. This includes re-handle of stockpiled ore at the end of the mine life and maintenance labor allocated to truck maintenance.

TABLE 19-25: BASE CASE ANNUAL HAULAGE OPERATING COST
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

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	Total
Fuel Consumption (KL)	KL	-	733	1,207	764	1,473	795	-	-	43	553	-	5,568
Fuel Cost	K USD	$ -	$ 410	$ 676	$ 428	$ 825	$ 445	$ -	$ -	$ 24	$ 310	$ -	$ 3,118
Lube & Oil	K USD	-	128	212	134	258	139	-	-	8	97	-	$ 976
Tires	K USD	-	333	548	347	669	361	-	-	20	251	-	$ 2,528
Wear Items & GET	K USD	-	-	-	-	-	-	-	-	-	-	-	$ -
Total Consumables	K USD	$ -	$ 872	$ 1,436	$ 909	$ 1,753	$ 945	$ -	$ -	$ 51	$ 658	$ -	$ 6,623
Parts / MARC Cost	K USD	-	225	370	234	452	244	-	-	13	170	-	$ 1,709
Total Maint. Allocation (no labor)	K USD	-	1,096	1,806	1,143	2,205	1,189	-	-	65	827	-	$ 8,331
Haulage Cost - CAT 789 Fleet													
Fuel Consumption (KL)	KL	994	5,826	5,932	6,380	6,691	5,774	4,651	3,416	2,388	70	-	42,122
Fuel Cost	K USD	$ 557	$ 3,263	$ 3,322	$ 3,573	$ 3,747	$ 3,233	$ 2,605	$ 1,913	$ 1,337	$ 39	$ -	$ 23,589
Lube & Oil	K USD	154	905	922	991	1,040	897	723	531	371	11	-	$ 6,544
Tires	K USD	523	3,066	3,122	3,358	3,521	3,039	2,448	1,798	1,257	37	-	$ 22,169
Wear Items & GET	K USD	-	-	-	-	-	-	-	-	-	-	-	$ -
Total Consumables	K USD	$ 1,234	$ 7,234	$ 7,365	$ 7,922	$ 8,308	$ 7,169	$ 5,775	$ 4,242	$ 2,965	$ 87	$ -	$ 52,302
Parts / MARC Cost	K USD	247	1,696	1,727	1,857	1,948	1,681	1,354	994	695	21	-	$ 12,219
Total Maint. Allocation (no labor)	K USD	1,481	8,930	9,092	9,779	10,255	8,850	7,129	5,237	3,661	108	-	$ 64,521
Labor Costs													
Maintenance Labor	K USD	$ 669	$ 2,071	$ 1,498	$ 1,498	$ 1,498	$ 1,498	$ 1,498	$ 1,115	$ 924	$ 319	$ -	$ 12,588
Operator Wages & Burden	K USD	$ 1,224	$ 4,080	$ 4,080	$ 4,080	$ 4,420	$ 3,740	$ 2,380	$ 2,040	$ 1,700	$ 510	$ -	$ 28,254
Total Haulage Cost	K USD	$ 3,374	$ 16,178	$ 16,475	$ 16,500	$ 18,378	$ 15,277	$ 11,007	$ 8,392	$ 6,349	$ 1,764	$ -	$ 113,695
Haulage Cost per Tonne Mined													
Fuel Cost	$/t	$ 0.09	$ 0.10	$ 0.12	$ 0.12	$ 0.12	$ 0.15	$ 0.17	$ 0.18	$ 0.20	$ 1.93	$ -	$ 0.13
Lube & Oil	$/t	$ 0.03	$ 0.03	$ 0.03	$ 0.03	$ 0.04	$ 0.04	$ 0.05	$ 0.05	$ 0.06	$ 0.60	$ -	$ 0.04
Tires	$/t	$ 0.09	$ 0.10	$ 0.11	$ 0.11	$ 0.11	$ 0.14	$ 0.16	$ 0.17	$ 0.19	$ 1.59	$ -	$ 0.12
Wear Items & GET	$/t	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Consumables	$/t	$ 0.20	$ 0.23	$ 0.26	$ 0.26	$ 0.27	$ 0.33	$ 0.37	$ 0.41	$ 0.44	$ 4.12	$ -	$ 0.29
Parts / MARC Cost	$/t	$ 0.04	$ 0.05	$ 0.06	$ 0.06	$ 0.07	$ 0.08	$ 0.09	$ 0.10	$ 0.10	$ 1.05	$ -	$ 0.07
Maintenance Labor	$/t	$ 0.11	$ 0.06	$ 0.04	$ 0.04	$ 0.04	$ 0.06	$ 0.10	$ 0.11	$ 0.14	$ 1.76	$ -	$ 0.06
Total Maintenance Allocation	$/t	$ 0.35	$ 0.35	$ 0.36	$ 0.37	$ 0.38	$ 0.47	$ 0.56	$ 0.61	$ 0.68	$ 6.94	$ -	$ 0.42
Operator Wages & Burden	$/t	$ 0.20	$ 0.12	$ 0.12	$ 0.12	$ 0.12	$ 0.15	$ 0.15	$ 0.20	$ 0.25	$ 2.82	$ -	$ 0.14
Total Haulage Cost per Tonne Mined	$/t	$ 0.55	$ 0.46	$ 0.48	$ 0.49	$ 0.50	$ 0.63	$ 0.71	$ 0.80	$ 0.93	$ 9.76	$ -	$ 0.56
Total Haulage Cost per Tonne Moved	$/t	$ 0.551	$ 0.460	$ 0.485	$ 0.493	$ 0.501	$ 0.627	$ 0.711	$ 0.803	$ 0.873	$ 0.301	$ -	$ 0.544

19.3.5 Base Case Mine Support Costs

Mine support costs are provided in TABLE 19-26 and include the operation of all of the mine-support equipment. The average life-of-mine support cost is estimated to be $0.157/t moved. This includes support during re-handling of stockpiled ore at the end of the mine life and maintenance labor allocated to drill.

TABLE 19-26: BASE CASE ANNUAL MINE SUPPORT OPERATING COSTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

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	Total
Mine Support Wages	K USD	$ 612	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,360	$ 1,360	$ 1,020	$ 510	$ -	$ 13,362
Mine Support Maint. Labor	K USD	$ 115	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 191	$ 96	$ -	$ 2,409
Total Mine Support Costs													
Consumables	K USD	$ 605	$ 1,507	$ 1,503	$ 1,503	$ 1,503	$ 1,507	$ 1,503	$ 1,503	$ 1,503	$ 753	$ -	$ 13,389
Parts / MARC Cost	K USD	$ 164	$ 407	$ 406	$ 406	$ 406	$ 407	$ 406	$ 406	$ 406	$ 204	$ -	$ 3,619
Operating Labor	K USD	$ 612	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,700	$ 1,360	$ 1,360	$ 1,020	$ 510	$ -	$ 13,362
Maintenance Labor	K USD	$ 115	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 287	$ 191	$ 96	$ -	$ 2,409
Total Costs	K USD	$ 1,496	$ 3,901	$ 3,896	$ 3,896	$ 3,896	$ 3,901	$ 3,556	$ 3,556	$ 3,120	$ 1,563	$ -	$ 32,779
Cost per tonne													
Consumables	$/t	$ 0.10	$ 0.04	$ 0.04	$ 0.04	$ 0.04	$ 0.06	$ 0.10	$ 0.14	$ 0.22	$ 4.17	$ -	$ 0.07
Maintenance Allocations	$/t	$ 0.03	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.02	$ 0.03	$ 0.04	$ 0.06	$ 1.13	$ -	$ 0.02
Operating Labor	$/t	$ 0.10	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.07	$ 0.09	$ 0.13	$ 0.15	$ 2.82	$ -	$ 0.07
Maintenance Labor	$/t	$ 0.02	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.01	$ 0.02	$ 0.03	$ 0.03	$ 0.53	$ -	$ 0.01
Total Costs	$/t	$ 0.24	$ 0.11	$ 0.11	$ 0.12	$ 0.11	$ 0.16	$ 0.23	$ 0.34	$ 0.46	$ 8.65	$ -	$ 0.16
Cost per Tonne Moved	$/t	$ 0.244	$ 0.111	$ 0.115	$ 0.117	$ 0.106	$ 0.160	$ 0.230	$ 0.340	$ 0.429	$ 0.267	$ -	$ 0.157

19.3.6 Base Case Mine Maintenance Costs

Mine maintenance costs for personnel and shop supplies are provided in TABLE 19-27. Note that the maintenance wages for mechanics has been included in the operating cost for equipment. Thus, the maintenance costs provided in TABLE 19-27 do not include the labor directly attributed to equipment maintenance. The average life-of-mine mine-maintenance cost is estimated to be $0.067/t moved.

TABLE 19-27: BASE CASE ANNUAL MINE MAINTENANCE COSTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

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	Total
Consumables	K USD	$ 38	$ 95	$ 95	$ 95	$ 95	$ 95	$ 95	$ 95	$ 95	$ 48	$ -	$ 848
Parts / MARC Cost	K USD	$ 13	$ 32	$ 32	$ 32	$ 32	$ 32	$ 32	$ 32	$ 32	$ 16	$ -	$ 287
Total Equipment Costs	K USD	$ 51	$ 128	$ 127	$ 127	$ 127	$ 128	$ 127	$ 127	$ 127	$ 64	$ -	$ 1,135
Wages & Sallaries													
Supervision	K USD	95.64	159.4	159.4	159.4	159.4	159.4	159.4	159.4	159.4	79.7	0	$ 1,451
Planners	K USD	47.82	79.7	79.7	79.7	79.7	79.7	79.7	79.7	79.7	39.85	0	$ 725
Hourly Personnel	K USD	369.84	1232.8	1232.8	1232.8	1232.8	1232.8	1232.8	1232.8	616.4	308.2	0	$ 9,924
Total	K USD	513.3	1471.9	1471.9	1471.9	1471.9	1471.9	1471.9	1471.9	855.5	427.75	0	$ 12,100
Other Costs													
Supplies	K USD	$ 30	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 25	$ -	$ 455
Light Vehicles	K USD	$ 10	$ 24	$ 24	$ 24	$ 24	$ 24	$ 24	$ 24	$ 24	$ 12	$ -	$ 215
Total	K USD	$ 40	$ 74	$ 74	$ 74	$ 74	$ 74	$ 74	$ 74	$ 74	$ 37	$ -	$ 670
Total Mine Maintenance	K USD	$ 604	$ 1,674	$ 1,674	$ 1,674	$ 1,674	$ 1,674	$ 1,674	$ 1,674	$ 1,057	$ 529	$ -	$ 13,906
Cost per Tonne Mined	$/t	$ 0.10	$ 0.05	$ 0.05	$ 0.05	$ 0.05	$ 0.07	$ 0.11	$ 0.16	$ 0.15	$ 2.93	$ -	$ 0.07
Cost per Tonne Moved	$/t	$ 0.099	$ 0.048	$ 0.049	$ 0.050	$ 0.046	$ 0.069	$ 0.108	$ 0.160	$ 0.145	$ 0.090	$ -	$ 0.067

19.3.7 Base Case Mine General Services Costs

Mine general services costs are provided in TABLE 19.28 and include costs for mine supervision, engineering, geology, light vehicles, and supplies. The average life-of-mine general service cost is estimated to be $0.090/t moved.

TABLE 19-28: BASE CASE ANNUAL MINE GENERAL SERVICES COST
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

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	Total
Mine General Services													
Supervision	K USD	$ 389	$ 648	$ 648	$ 648	$ 648	$ 648	$ 648	$ 648	$ 648	$ 324	$ -	$ 5,900
Clerical	K USD	$ 38	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 64	$ 32	$ -	$ 581
Training	K USD	$ 51	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 85	$ 43	$ -	$ 774
Total	K USD	$ 478	$ 797	$ 797	$ 797	$ 797	$ 797	$ 797	$ 797	$ 797	$ 399	$ -	$ 7,254
Engineering													
Supervision	K USD	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ -	$ 1,451
Sallaried Personnel	K USD	$ 80	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 133	$ 66	$ -	$ 1,209
Hourly Personnel	K USD	$ 105	$ 351	$ 351	$ 351	$ 351	$ 351	$ 351	$ 351	$ 175	$ 88	$ -	$ 2,824
Total	K USD	$ 281	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 643	$ 468	$ 234	$ -	$ 5,484
Mine Geology													
Supervision	K USD	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ -	$ 1,451
Sallaried Personnel	K USD	$ 128	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 213	$ 106	$ 53	$ -	$ 1,775
Hourly Personnel	K USD	$ 96	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 159	$ 80	$ 40	$ -	$ 1,331
Total	K USD	$ 319	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 531	$ 345	$ 173	$ -	$ 4,557
Supplies & Other													
Mine General Services	K USD	$ 30	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 50	$ 25	$ -	$ 455
Mine Light Vehicle	K USD	$ 26	$ 65	$ 64	$ 64	$ 64	$ 65	$ 64	$ 64	$ 64	$ 32	$ -	$ 575
Engineering Supplies	K USD	$ 9	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 8	$ -	$ 137
Engineering Light Vehicle	K USD	$ 4	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 5	$ -	$ 86
Geology Supplies	K USD	$ 9	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 15	$ 8	$ -	$ 137
Geology Light Vehicle	K USD	$ 4	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 10	$ 5	$ -	$ 86
Total	K USD	$ 82	$ 164	$ 164	$ 164	$ 164	$ 164	$ 164	$ 164	$ 164	$ 82	$ -	$ 1,475
Total Mine Other Costs	K USD	$ 1,159	$ 2,136	$ 2,135	$ 2,135	$ 2,135	$ 2,136	$ 2,135	$ 2,135	$ 1,774	$ 887	$ -	$ 18,769
Cost per Tonne Mined	$/t	$ 0.19	$ 0.06	$ 0.06	$ 0.06	$ 0.06	$ 0.09	$ 0.14	$ 0.20	$ 0.26	$ 4.91	$ -	$ 0.09
Cost per Tonne Moved	$/t	$ 0.189	$ 0.061	$ 0.063	$ 0.064	$ 0.058	$ 0.088	$ 0.138	$ 0.204	$ 0.244	$ 0.152	$ -	$ 0.090

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 (150 – 300 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" (July 27, 2010)* **with only minor changes for consistent formatting and terminology purposes (see Appendix G-1 for complete report).**

The report 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 Northern Territory, 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 Northern Territory 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 Generation Option Selection

The site electrical power demands are a fixed constant operating load estimated at 34MW 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. Multiple reciprocating gas engines are available to more precisely meet the site power needs. They do not provide a reduced capital investment to meet the 34MW site requirement but have better fuel efficiency.

19.5.2 Pre-Feasibility Cost Analysis

The cost analysis for this study is based on a 10 year operating plant life without annual pricing index. Fuel costs are based from the draft proposal provided by the Power and Water Corporation, dated May 11, 2010 at a rate of $5.75 (AUS) per gigajoule. A construction cost model for each gas turbine was created using GT Pro software with auxiliary equipment budgets based on vendor quotes. Construction costs for the reciprocating engine sets were estimated based on cost per kilowatt installed. Calculated 10 year project life costs are estimated $0.0775 to 0.0863 (AUS) per kilowatt-hour for the 34MW site demand compared to the commercially purchased electricity rate of $0.1636 per kWh (adjusted for demand) for the same time period.

19.5.3 Conclusions and Recommendations

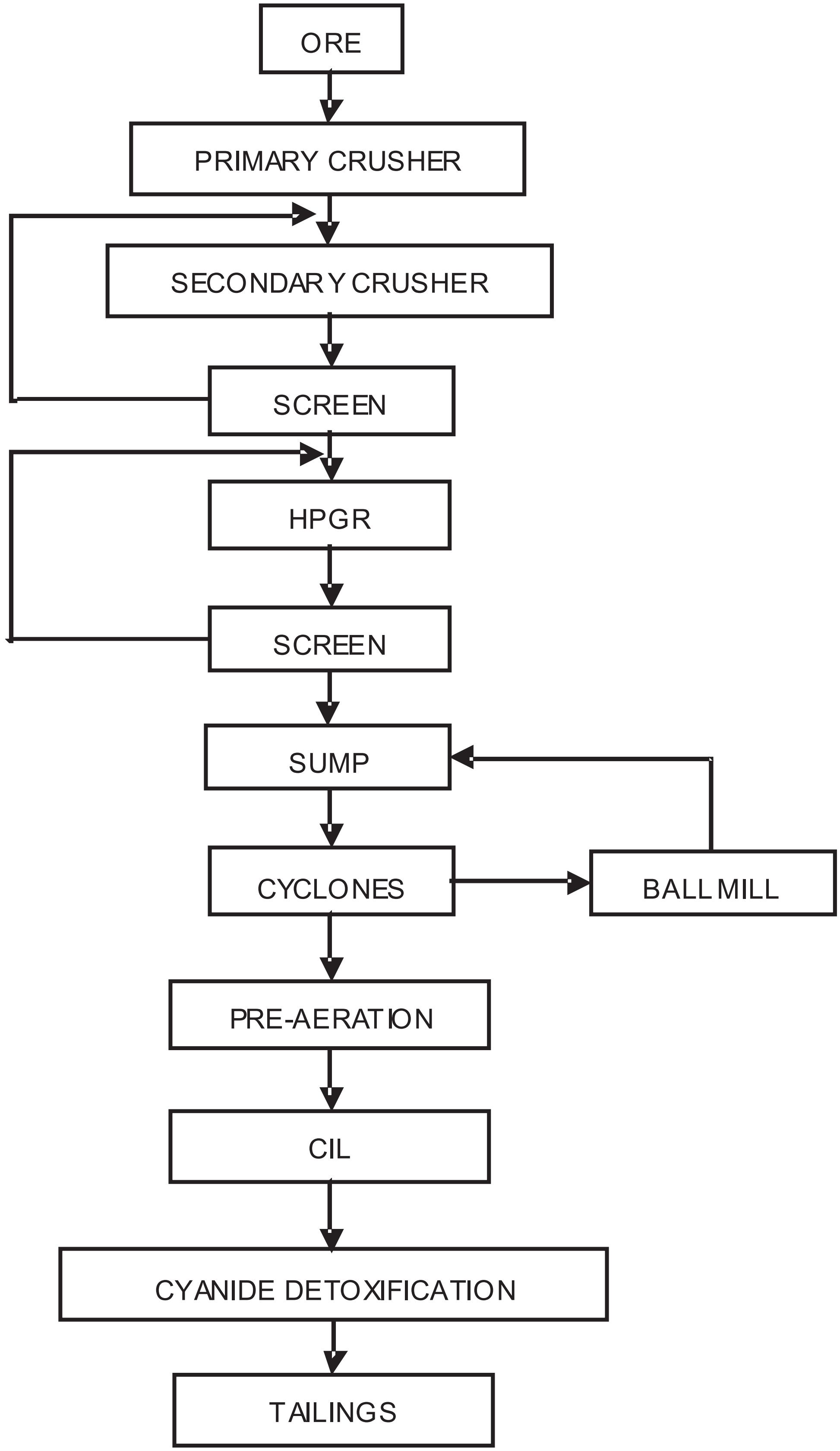
This study provides a preliminary budgetary outline for capital and operating costs to produce onsite power generation for the scale of process indicated by Vista Gold Corp. at the Mt. Todd Mine with limited information about the site. It is apparent that there is demand for the surplus power that could 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 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 GE LM6000PF provides the lowest initial capital cost per kilowatt with a single unit. If there is higher demand for export power the Rolls Royce gas turbine has the most reserve capacity. If continuous power supply is required, the Wartsila 20V34SG reciprocating engines are estimated to provide the lowest overall 10 year project costs.

19.6 Process Operations

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

TABLE 19-29 details the key design criteria applicable to processing flowsheet.



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TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***Vista Gold Corp.***	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 19-29: KEY PROCESS PLANT DESIGN CRITERIA VISTA GOLD CORP. – MT TODD GOLD PROJECT October 2010			
Description	**Unit**	**Value**	**Source**
Plant throughput	Mt/y	6.8	TetraTech
Primary crusher availability	%	75	Ausenco
HPGR availability	%	80	Ausenco
Grinding and CIL availability	%	92	Ausenco
Plant feed rate	t/h	844	calculated
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	1.08	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	5.3	Ausenco
gold extraction	%	82.0	RDi
gold recovery	%	80.5	RDi
adsorption time	h	18.8	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
batch size	t	12	Ausenco
strip frequency	#/week	9	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 base case process plant as shown in FIGURE 19-4 was designed to treat 6.8 Mtpy of ore (~18.5Ktpd or ~844tph). This discussion is based on the Ausenco September 2010 report reference elsewhere and in Appendix E.

The Simplified Process Schematic shown in FIGURE 19-4 is similar, when not identical; to most common leach circuits with a few exceptions as noted below:

- High pressure grinding rolls (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 secondary copper sulphide minerals. 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.

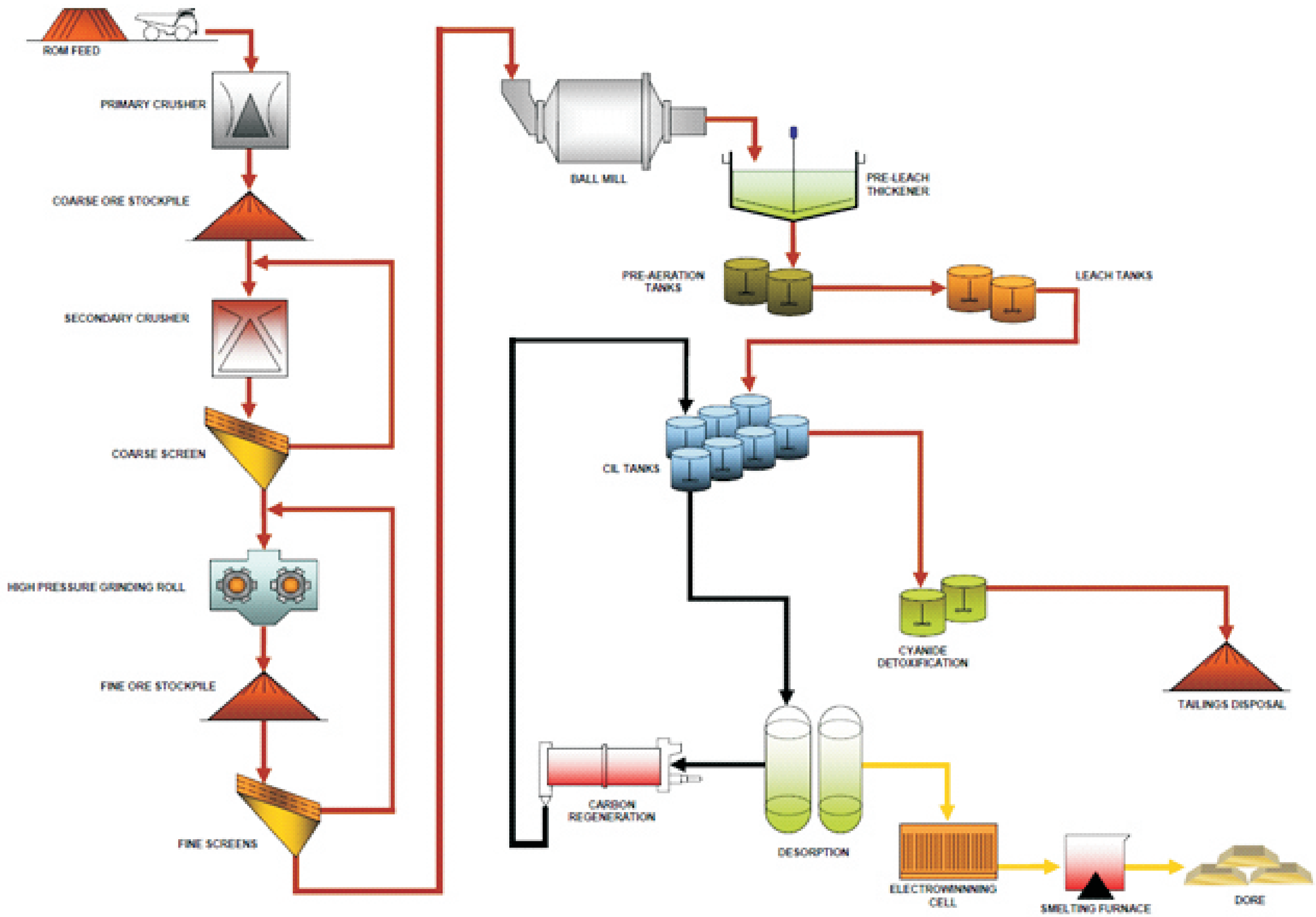
Standard equipment sizes were used to the greatest extent possible so that “off the shelf” equipment and parts could be used and would be available as needed.

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 solublized 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 19-30.



(Ausenco Services Pty. Ltd. Report 2010)

Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*Vista Gold Corp.*	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 19-30: SUMMARY OF PRE-AERATION AND LEACH RESIDENCE TIMES AND TANK DETAILS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Criterion		Pre-aeration	Leach	Adsorption
Residence Time – specified design	h	4	5.3	18.7
Required volume	m^3	4011	6128	21622
Tank diameter	m	15.8	15.8	15.8
Tank height	m	15.8	15.8	15.8
Tank volume	m^3	3098	3098	3098
Number of tanks	#	2	2	7
Total volume	m^3	6196	6196	21686
Residence Time - actual	h	6.2	5.4	18.8

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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 not only producing high grade pregnant liquor but also minimizing the amount of deleterious metals contained in the liquor that ultimately may affect the electrowinning process. A summary of the carbon strip circuit design criteria, as provided by Ausenco, is presented in TABLE 19-31.

TABLE 19-31: ELUTION AND REGENERATION DESIGN CRITERIA
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Criterion	Unit	Design Value
Strip size	t	12
Strips per day		1.3
Acid wash		
type		Cold HCl
acid concentration to column	% w/w	3
bed volume	BV	1
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	857
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 500 toz Au doré bars.

Ancillary operations to carbon loading and stripping include carbon acid washing and carbon stripping. Carbon is regularly washed in a mild (3%)/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 19-32.

TABLE 19-32: CYANIDE DETOXIFICATION DESIGN CRITERIA
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

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	h	2
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 Base Case Process Capital Costs

Process capital costs were established at the prefeasibility level by Ausenco in their July 29, 2009 report entitled Mount Todd Gold Project Engineering and Cost Study (6.8MT). Costs are presented in US dollars converted from Australian dollars (per referenced Ausenco report) at a factor of US$0.85 per AUD$1.00.

The estimated total process capital cost is US$155,762,500.

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

Plant throughput, Mt/y	6.8
Plant feedrate, t/h	844
Head grade, g Au/t	1.08
Primary grind, P80 µm	150
Primary crusher availability, %	75
HPGR availability, %	80
Grinding & CIL availability, %	92

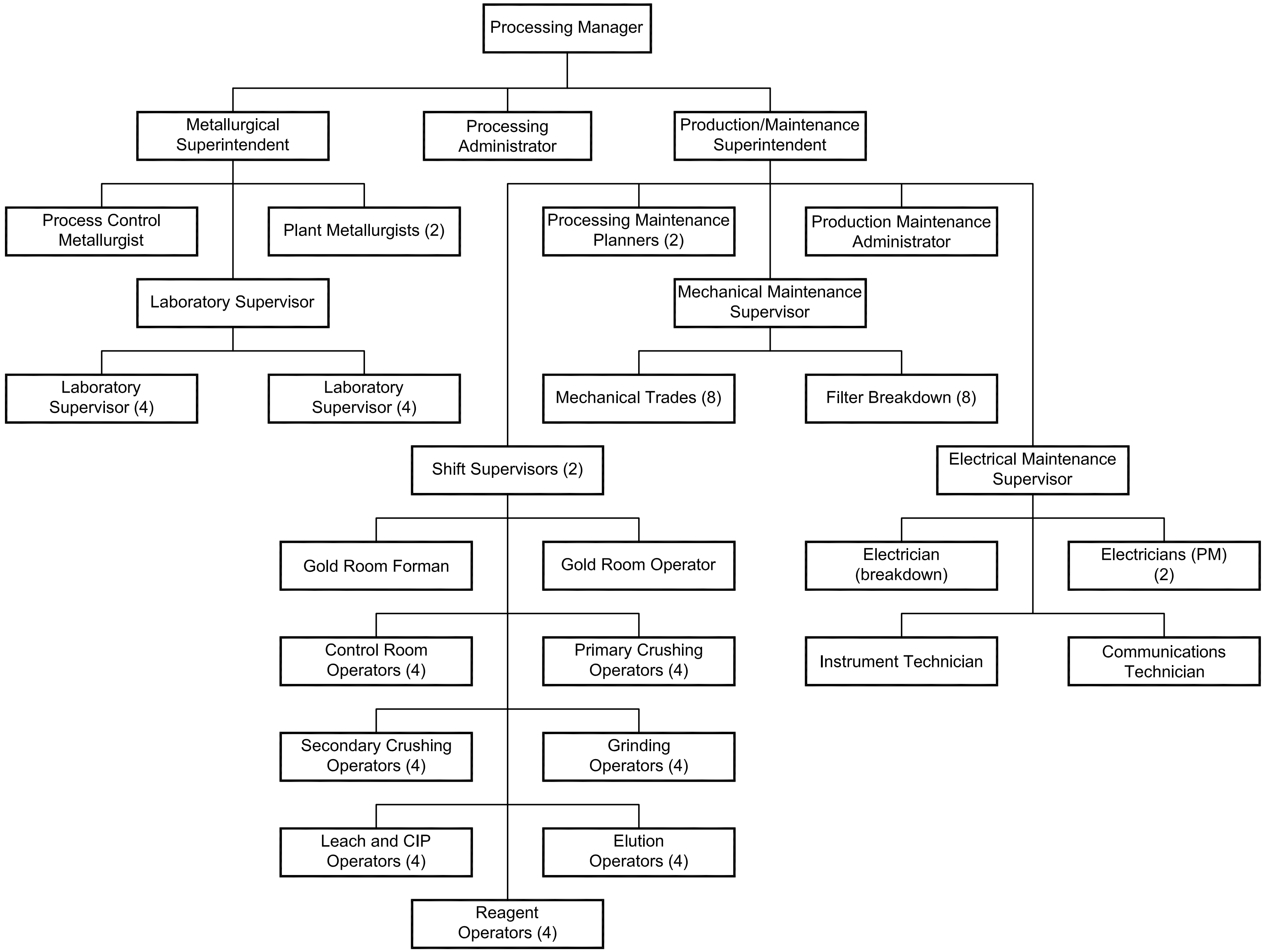
19.8 Base Case Process Operating Costs

Plant operating costs were established at the prefeasibility level by Ausenco in their September 2010 report entitled *Mount Todd Gold Project Engineering and Cost Study (6.8MT)*. Costs are presented in US dollars converted from Australian dollars (per referenced Ausenco report) at a factor of US$0.85 per AUD$1.00. FIGURE 19-5 presents the process plant labor organization.

The estimated unit cost for processing is US$7.85/t (feed to primary crusher) at annualized feedrate of 6.8Mt/y. This equates to an annualized process operating cost of US$53.42M.

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

- Operating costs have a base date of Q1 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
    - Rehabilitation and closure costs



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Project Location: Northern Territory, Australia
File Name: Fig19-5.dwg
Project Number: 114-310912a
Date of Issue: 10/01/2010

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-2009 costs.

Initial Capital Costs of US$475.964 million 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 village at a nominal rate 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 19-33 is a summary of the original and sustaining capital expended during the project.

TABLE 19-33:SUMMARY OF TOTAL PROJECT CAPITAL COSTS(000)
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Cost Center	**US$ (000)**
Summary of Initial Capital Costs	
Mine	
Primary Open Pit Mine Equipment	62,754
Lime Operation Mine Equipment	5,617
Ancillary General Surface Mobil Equipment	13,588
Mine Office, Shop, and Warehouse	768
Mining Development Supply and Labor Op Costs	11,323
Mine Sub-total	**$94,050**
Process	
Process Plant	155,763
Onsite Infrastructure	12,819
Offsite Infrastructure	15,147
Mobile Equipment, Spares, First-Fills	8,393
Power Generating Station	31,519
Process Sub-total	**$223,641**
Tailings	
Water Treatment Facility & Tailings Operation Costs	20,226
Tailings Storage Facility	11,910
Tailings Sub-total	**$32,136**
Indirects	
Temporary Construction Facilities	4,208
Accommodations	3,585
EPCM (Contractor's Cost)	30,843
Commissioning	3,503
Indirects Sub-total	**$42,139**
Other Capital	
Water Treatment Facility	14,659
Site Demolition	850
EPCM Contractor Fee	3,994
Permitting	500
Recruiting and Training	1,700
Lime Kiln/processing	6,158
Other Capital Sub-total	**$27,861**
Contingency	56,137
Salvage Value	-59,567
Owner/Reclamation (no contingency)	57,735
Summary of Sustaining Capital Costs	
General Surface Mine Equipment	5,086
Mine Capital Contingency	763
Water Treatment Facility (WTF)	8,780
Pre-production WTF & Tailings Operation Costs	11,269
Tailings Storage Facility	3,544
Process Plant Contingency	2,222
Other Capital Contingency	1,317
Sustaining Capital Starting in Year 2	**$32,981**

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

TABLE 19-34 illustrates how process capital makes up nearly 50% of the total original capital costs. Mine capital is the next largest component making up approximately 20% of the total original capital cost.

TABLE 19-34: TOTAL ORIGINAL CAPITAL COSTS
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

TOTAL ORIGINAL CAPITAL	
MINE	$94,051
PROCESS	$223,641
OTHER CAPITAL	$27,861
TAILINGS	$32,136
INDIRECTS	$42,139
CONTINGENCY	$56,137
	$475,964

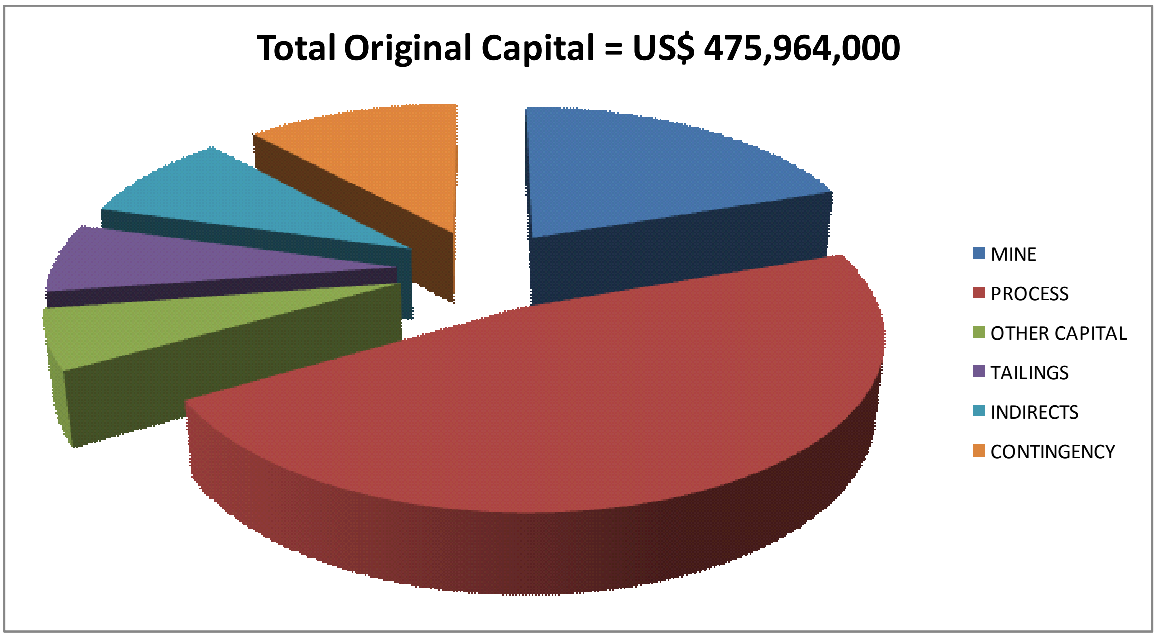


TABLE 19-35 illustrates the large portion of sustaining costs that will be dedicated to the tailings storage facility (TSF), tailings operations and the water treatment facilities.

TABLE 19-35: SUSTAINING CAPITAL STARTING IN YEAR 2
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

SUSTAINING CAPITAL STARTING YEAR 2	
General Surface Mobil Equipment	$5,086
Mine Capital Contingency	$763
Pre-production WTF + Tails Op Costs	$11,269
TSF	$3,544
Plant Contingency	$2,222
WTF	$8,780
Other Capital Contingency	$1,317
	$32,981

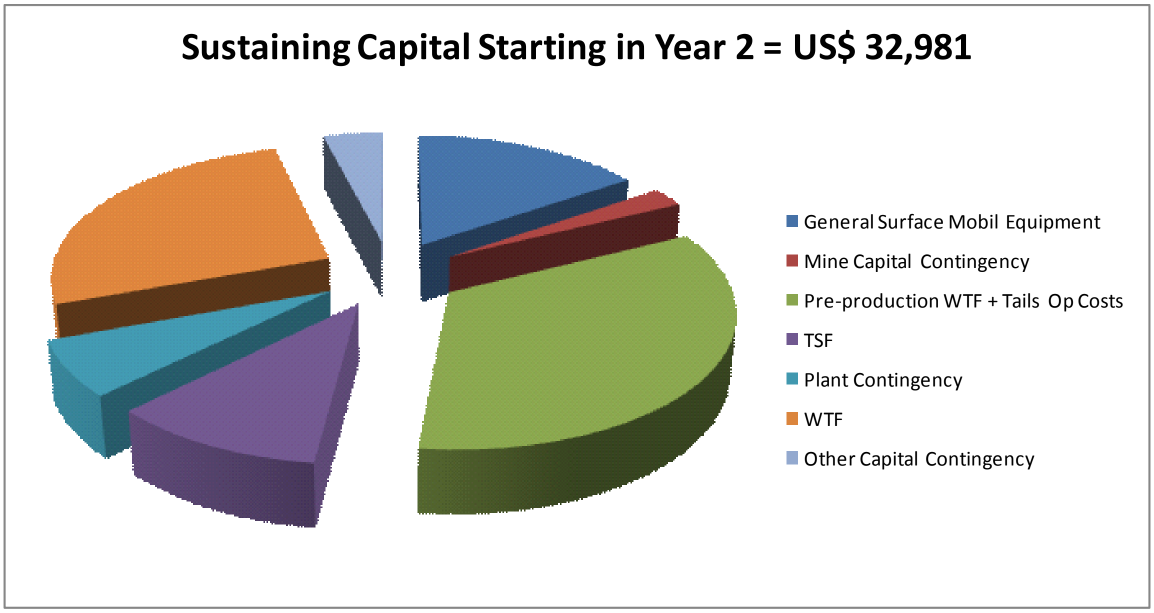


TABLE 19-36 illustrates the fact that initial process plant capital costs amount to more than two-third (3/3) of the total initial startup capital.

TABLE 19-36:ORIGINAL PROCESS CAPITAL,
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

PROCESS	
Process Plant	$155,763
Onsite Infrastructure	$12,819
Offsite Infrastructure	$15,147
Mobile Equipment, Spares, First-Fills	$8,393
Power Generating Station	$31,519
	$223,641

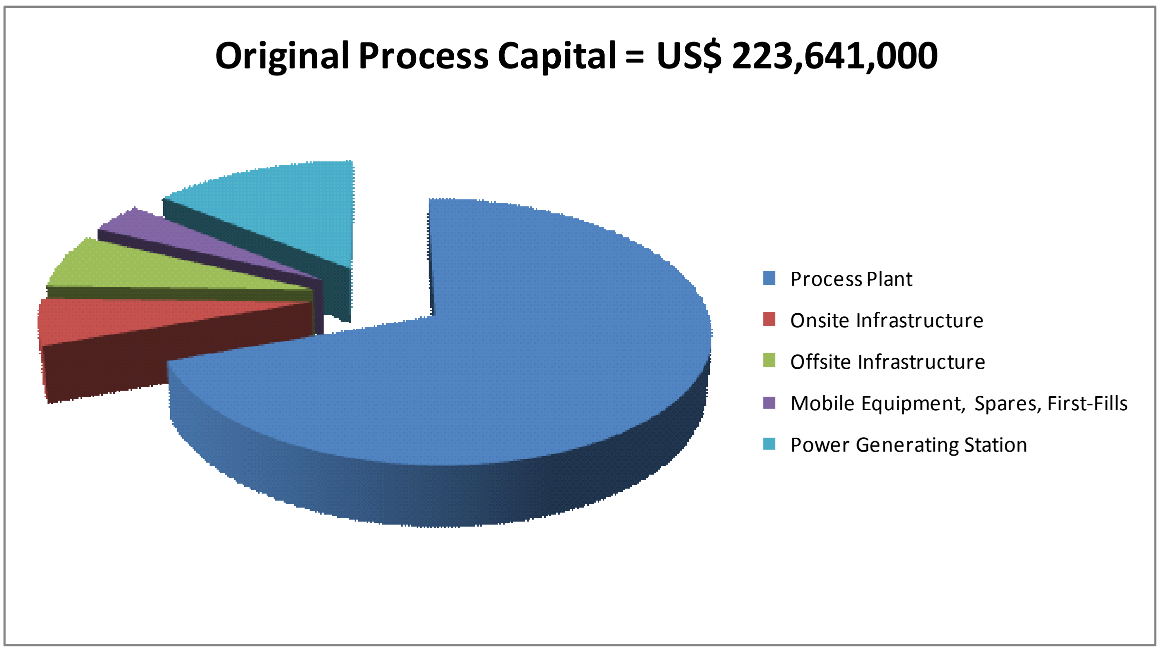


TABLE 19-37 illustrates the fact that the Primary Open Pit Mine Equipment comprise approximately two-thirds (2/3) of the total mine capital costs.

TABLE 19-37: MINE CAPITAL
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

MINE	
Primary Open Pit Mine Equipment	$62,754
Lime Operation Mine Equip	$5,617
Ancillary General Surface Mobil Equipment	$13,588
Mine Office, Shop and Warehouse	$768
Mining Development Supply and Labor Op Costs	$11,323
	$94,051

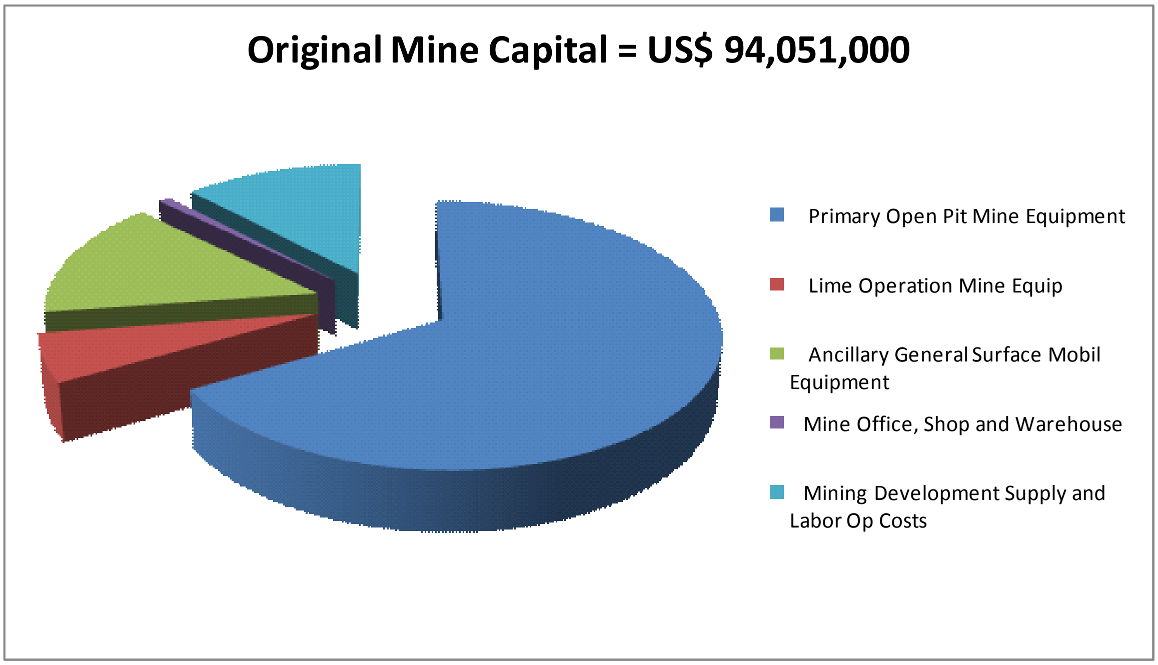
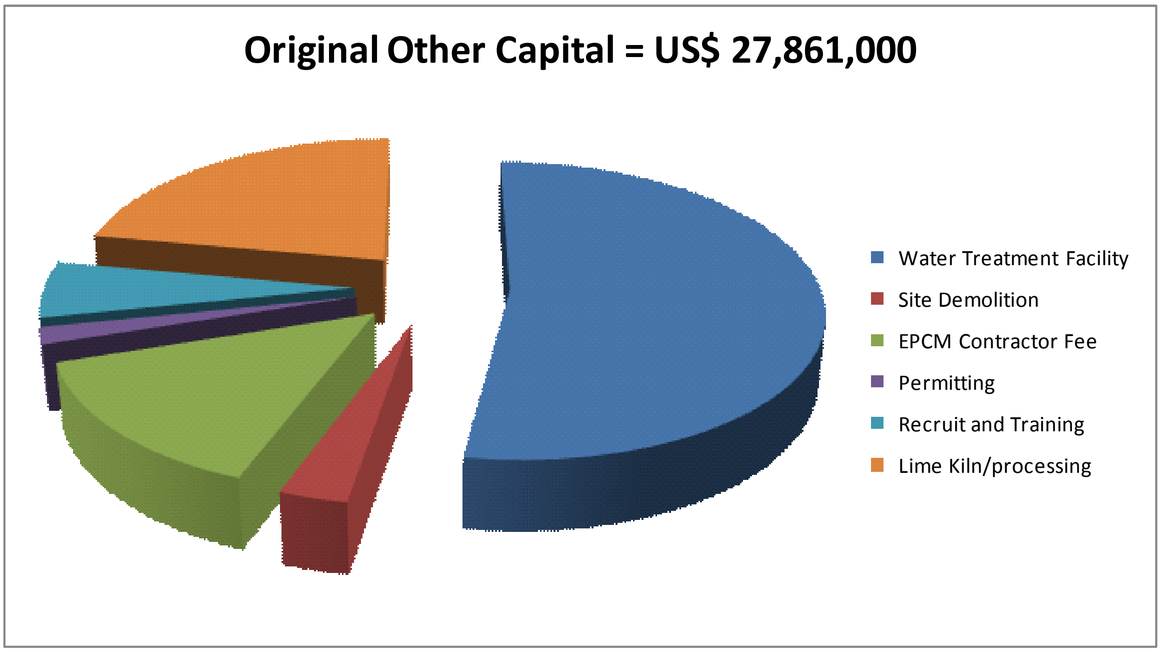


TABLE 19-38 illustrates the fact that the Water Treatment Facility (WTF) amounts to approximately (1/2) of the Other Capital expenditures for the planned mine operations.

TABLE 19-38: OTHER CAPITAL
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

OTHER CAPITAL	
Water Treatment Facility	$14,659
Site Demolition	$850
EPCM Contractor Fee	$3,994
Permitting	$500
Recruit and Training	$1,700
Lime Kiln/processing	$6,158
	$27,861



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. Tt 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 6.77 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 6.77 Mtpy mine plan as summarized in Section 5.0 is US$64,938,000. The pre-feasibility level cost estimate for mine-life water treatment plan for the 6.77 Mtpy mine plan as summarized in Section 5.0 is US$31,203,000. These estimated costs are summarized in TABLE 5-3.

A scoping-level sensitivity analysis was conducted and included an evaluation of closure and water treatment costs for the 10.6 Mtpy mine plan. For this sensitivity analysis, the closure and water treatment plans and strategies present in the 6.77 Mtpy were replicated and proportionally scaled according to the quantities (e.g. facility dimension, material/fluid volumes, surface areas, disturbance footprints) estimated for the 10.6 Mtpy mine plan. These estimated costs include the same basic costing approaches and assumptions used to estimate the cost to implement the closure and mine-life water treatment for the 6.77 Mtpy mine plan.

As summarized in TABLE 5-4 the scoping-level cost estimate for implementing the closure plan for 10.6 Mtpy mine plan is US$120,564,000. TABLE 5-4 also includes the scoping-level cost estimate for implementing the mine-life water treatment plan for the 10.6 Mtpy mine plan and is US$51,364,000.

19.11 Tailings Disposal

Previously, Tt evaluated twelve options for tailings disposal, including a dry stack facility, heap leach pads, new TSF designs, and several raises to the existing TSF. Appendix K contains the tradeoff study. The 60 million tonne capacity raise to the existing TSF design was selected based on economic tradeoff studies and the relatively low cost per tonne of tailings stored. The capital costs are much lower than constructing a new TSF based on the following factors:

- There is no liner cost associated with constructing raises to the existing unlined TSF;
- The existing underdrains, toe drains, and decant structures were assumed to be functional, yielding a minimal cost associated with raising or extending these structures to accommodate subsequent raises;
- The additional stages can be constructed using centerline or upstream raise construction methods, reducing the volume of waste rock required to construct the embankments; and
- The existing Return Water Pond and Water Polishing Pond were also assumed to be functional and no additional costs will be incurred to construct ponds to collect flows from the toe drain and underdrains.

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 three meters 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 eight meters 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 design of the raises to the existing TSF:

- 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 Northern Territory 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 19-39 below.

TABLE 19-39: COST ESTIMATE BY STAGE
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

TSF Stage	Embankment Elevation (m)	Construction Method	Storage Capacity (million tonnes)	Estimated Life of Stage (years)	Construction Cost (million $US)
Stage 2	146.5	Centerline	23.9	3.5	9.5
Stage 3	149.0	Upstream	7.9	1.2	1.1
Stage 4	151.5	Upstream	8.1	1.2	0.8
Stage 5	154.0	Upstream	8.2	1.2	0.8
Stage 6	156.5	Upstream	8.3	1.2	0.8
Stage 7	158.0	Upstream	5.0	0.7	0.5
Total			61.4	9	13.5

19.12 Cash Flow Analysis

The cash flow analysis developed for the mining, processing, tailings disposal and reclamation of the Mt Todd Base Case 6.77 Mtpy scenario includes the following input parameters:

- Base Case Gold price of US$950 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.
- Open Pit Mine operating cost range from a high of $7.25/t ore occurring in year 4, the year with the highest strip ratio, to $1.09/t ore in year 9.

19.12.1 Operating Costs

TABLE 19-40 details the mine operating costs by year for the 6.77 Mtpy base case.

TABLE 19-40: MINE OPERATING COST SUMMARY (000)
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Year	1	2	3	4	5	6	7	8	9
Ore Tonnes Mined	6,789	6,770	6,770	6,770	6,770	6,770	6,770	6,770	5,852
Total Mining Costs	$45,817	$46,149	$45,657	$49,084	$40,046	$28,884	$23,189	$17,699	$6,398
Mine Operating Costs ($/tonne ore)	$6.75	$6.82	$6.74	$7.25	$5.90	$4.27	$3.43	$2.61	$1.09

The Base Case process operating cost range from $7.35 to $7.38/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 19-41.

TABLE 19-41: ADR PROCESS OPERATING COST SUMMARY (000)
VISTA GOLD CORP. – MT TODD GOLD PROJECT
October 2010

Year	1	2	3	4	5	6	7	8	9
Ore Tonnes Mined	6,789	6,770	6,770	6,770	6,770	6,770	6,770	6,770	5,852
Total Processing Costs	$49,969	$49,789	$49,829	$49,789	$50,088	$49,893	$49,848	$49,848	$43,194
Ore Processing Costs ($/tonne ore)	$7.36	$7.35	$7.36	$7.35	$7.38	$7.37	$7.36	$7.36	$7.38

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

- G & A at US$0.569 per tonne of ore processed
- Gold dore' refining, transport and treatment charges are US$4.50/toz Au

19.12.2 Base Case Results

TABLE 19-42 presents the cash flow summary for the Base Case production rate of 6.77 Mtpy (18,549tpd, 365dpy), at a gold price of $950/toz Au, a US to Australian currency exchange rate of 0.85, and constant 2010 US dollars. Results for the Base Case scenario include a before tax net present value (NPV) of US$210.175 million for the project evaluated at a 5 percent discount rate. Pretax Internal Rate of Return (IRR) is 14.9%. 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.

Since the development and publishing of the cash flow statement in the August 18, 2010 press release, Tt has become aware of two financial items that could impact the results presented. They are:

- Approximately US$1.5 million for a mine shop was not accounted for in the cash flow presented.

- Approximately US$2.56 million of environmental related costs were over-estimated due to a calculation error for pond sizing.

Clearly, the net result is that the cash flow has slightly over-estimated the capital costs for the project by approximately US$1.06 million. Given that the PFS is a plus/minus 25 percent estimate and this difference represents less than a one percent change total capital costs, it is Tt's opinion that these omissions are not material to the results presented.

TABLE 19-42: Mt. Todd 6.77 Mtpy Base Case Cash Flow Analysis (US$950/toz Au Price)

PRETAX:	
IRR	14.9%
NPV0 (000'S)	$472,840
NPV5 (000'S)	$210,175
AVG ANNUAL CF (000's) PRODUCTION YEARS	$75,099
AVG ANNUAL CF (000's) LIFE OF MINE	$41,305
STRIPPING RATIO (WST:ORE)	2.37

AFTER-TAX:	
IRR	9.8%
NPV0 (000'S)	$252,782
NPV5 (000'S)	$71,208
AVG ANNUAL CF (000's) PRODUCTION YEARS	$55,094
AVG ANNUAL CF (000's) LIFE OF MINE	$30,302
PAYBACK PERIOD (YRS) FROM : START OF PRODUCTION	5.4
POST CLOSURE NET CASH FLOW:	$72,995

CAPITAL	
INITIAL CAPITAL (000'S)	$391,148
CONTINGENCY	$50,335
SUB-TOTAL	441,483
WORKING CAPITAL - 2011 - 2013	(7,012)
INITIAL CAP, PRE-PROD DEV & WORKING CAP	$434,471
SUSTAINING CAPITAL (000'S)	28,679
CONTINGENCY	4,302
TOTAL SUSTAINING CAPITAL	32,981
WORKING CAPITAL - 2014 - 2030	7,244
TOTAL MINE LIFE CAPITAL	$474,697

COSTS	
CASH OPER COST PER OUNCE	$476
TOTAL CASH COST PER OUNCE	$487
CAPITAL COST PER OUNCE	$250
TOTAL PRODUCTION COST PER OUNCE	$737

UNIT COSTS	
MINING COST ($/TONNE MINED)	$1.50
MINING COST ($/TONNE ORE)	$5.04
PROCESSING COST ($/TONNE ORE)	$7.44
G&A Cost ($/TONNE ORE)	$0.64
TOTAL OPERATING COSTS $/TONNE ORE	$13.12

PROJECT PRODUCTION SCHEDULE / GOLD GRADES AND CONTENT

			Project Year -2	-1	1	2	3	4	5	6	7	8	9
MINE													
ORE TONNAGE TO CRUSHER (000's)	ore tonnes	60,050			6,789	6,770	6,770	6,770	6,789	6,770	6,770	6,770	5,852
ORE GRADE	g Au/tonne	1.045			1.09	1.22	1.22	0.97	0.98	0.91	1.08	1.20	0.75
	toz Au/tonne	0.034			0.035	0.039	0.039	0.031	0.031	0.029	0.035	0.039	0.024
CONTAINED GOLD	g Au	63,027,314			7,381,696	8,271,100	8,228,457	6,535,370	6,621,247	6,148,847	7,342,132	8,114,680	4,383,787
	toz Au	2,026,374			237,327	265,922	264,551	210,117	212,878	197,690	236,055	260,893	140,942
WASTE TONNAGE MINED (000's)	waste tonnes	142,524		5,583	27,898	25,095	23,519	29,948	17,573	8,717	3,685	504	2
TOTAL MATERIAL MINED	total tonnes	202,573		5,583	34,687	31,865	30,289	36,718	24,362	15,487	10,455	7,274	5,855
STRIPPING RATIO	waste : ore	2.4			4.11	3.71	3.47	4.42	2.59	1.29	0.54	0.07	0.00
MILL													
ORE TONNAGE TO MILL (000's)	ore tonnes	60,050			6,789	6,770	6,770	6,770	6,789	6,770	6,770	6,770	5,852
MILL FEED GRADE	g Au/tonne	1.045			1.09	1.22	1.22	0.97	0.98	0.91	1.08	1.20	0.75
	toz Au/tonne	0.034			0.035	0.039	0.039	0.031	0.031	0.029	0.035	0.039	0.024
CONTAINED GOLD	g Au	63,027,314			7,381,696	8,271,100	8,228,457	6,535,370	6,621,247	6,148,847	7,342,132	8,114,680	4,383,787
	toz Au	2,026,374			237,327	265,922	264,551	210,117	212,878	197,690	236,055	260,893	140,942
MILL RECOVERY @ 82%	% recovery of Au	82%			82%	82%	82%	82%	82%	82%	82%	82%	82%
GOLD RECOVERED	g Au	51,682,398			6,052,990	6,782,302	6,747,335	5,359,003	5,429,422	5,042,055	6,020,548	6,654,038	3,594,705
	toz Au	1,661,626			194,608	218,056	216,932	172,296	174,560	162,106	193,565	213,932	115,572
REFINERY													
PAYABLE GOLD TO REFINERY	g Au	51,682,398			6,052,990	6,782,302	6,747,335	5,359,003	5,429,422	5,042,055	6,020,548	6,654,038	3,594,705
	toz Au	1,661,626			194,608	218,056	216,932	172,296	174,560	162,106	193,565	213,932	115,572

		Total LOM	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
GOLD PRICE	$/oz	$950			$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950
WASTE TONNES	000's	142,524		5,583	27,898	25,095	23,519	29,948	17,573	8,717	3,685	504	2									
TONNES ORE TO MILL	000's	60,050			6,789	6,770	6,770	6,770	6,789	6,770	6,770	6,770	5,852									
STRIPPING RATIO	waste:ore	2.37			4.11	3.71	3.47	4.42	2.59	1.29	0.54	0.07	0.00									
OUNCES PAYABLE	toz Au.	1,661,626			194,608	218,056	216,932	172,296	174,560	162,106	193,565	213,932	115,572									
GOLD GRADE	g/tonne	1.050			1.087	1.222	1.215	0.965	0.975	0.908	1.085	1.199	0.749									
GROSS GOLD SALES	$000's	$1,578,545			$184,878	$207,153	$206,085	$163,681	$165,832	$154,000	$183,887	$203,236	$109,794									
RENTAL INCOME/POWER INCOME	$000's	$178,401			$5,132	$5,132	$5,132	$5,132	$5,132	$5,132	$5,132	$5,132	$5,132	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690
GROSS REVENUE	$000's	$1,756,947			$190,010	$212,286	$211,218	$168,814	$170,964	$159,133	$189,019	$208,368	$114,926	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690
LESS REFINING, TRANS. & TREATMENT	$000's	7,477			876	981	976	775	786	729	871	963	520									
REVENUE FROM SALES	$000's	1,749,469			189,134	211,304	210,241	168,038	170,179	158,403	188,148	207,405	114,406	14,690	14,690	14,690	14,690	14,690	14,690	14,690	14,690	14,690
LESS ROYALTY	$000's	15,785			1,849	2,072	2,061	1,637	1,658	1,540	1,839	2,032	1,098									
NET REVENUE		$1,733,684			$187,286	$209,233	$208,181	$166,401	$168,521	$156,863	$186,309	$205,373	$113,308	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690	$14,690
NET REVENUE AFTER PRODUCTION	$132,209																					
OPERATING COSTS																						
MINE	$000's	302,923			45,817	46,149	45,657	49,084	40,046	28,884	23,189	17,699	6,398									
MILL	$000's	446,567			49,969	49,789	49,829	49,789	50,088	49,893	49,848	49,848	43,194	957	886	890	268	268	268	268	268	248
G&A	$000's	38,489		3,849	3,849	3,849	3,849	3,849	3,849	3,849	3,849	3,849	3,849									
RECLAMATION	$000's	57,735			2,585			220	36					14,949	35,413	2,501	338	338	338	338	338	338
TOTAL OPERATING COSTS	$000's	$845,715		$3,849	$102,220	$99,787	$99,335	$102,943	$94,019	$82,625	$76,886	$71,395	$53,441	$15,906	$36,300	$3,391	$607	$607	$607	$607	$606	$586
MILL OPERATING COSTS AFTER PRODUCTION	4,321																					
RECLAMATION COSTS AFTER PRODCTION	54,894																					
OPERATING MARGIN	$000's	$887,969		($3,849)	$85,066	$109,445	$108,846	$63,459	$74,502	$74,238	$109,423	$133,978	$59,867	($1,216)	($21,610)	$11,299	$14,083	$14,083	$14,083	$14,083	$14,084	$14,104
CAPITAL COSTS																						
MINE EQUIPMENT	$000's	94,051		64,210	24,755	27		413	4,619	27												
PLANT EQUIPMENT & CONSTRUCTION	$000's	297,915	29,491	252,756	856	810	1,796	1,528	974	1,628	1,592	1,311	853	957	886	890	268	268	268	268	268	248
OTHER/CONTINGENCY	$000's	82,498	12,737	55,082	1,597	126	269	291	839	248	239	197	128	143	133	133	40	40	40	40	40	10,135
SUB-TOTAL	$000's	$474,464	$42,228	$372,048	$27,208	$963	$2,065	$2,232	$6,432	$1,903	$1,831	$1,508	$981	$1,100	$1,019	$1,023	$308	$308	$308	$308	$308	$10,382
SALVAGE VALUE	$000's	(59,567)												(43,937)								(15,630)
TOTAL CAPITAL	$000's	$414,897	$42,228	$372,048	$27,208	$963	$2,065	$2,232	$6,432	$1,903	$1,831	$1,508	$981	($42,837)	$1,019	$1,023	$308	$308	$308	$308	$308	($5,248)
CHANGES TO WORKING CAPITAL	$000's	232	294	2,457	(9,762)	2,522	51	(697)	879	872	756	532	641	4,066	(2,585)	(0)	67				0	140
PRE-TAX CASH FLOWS	$000's	$472,840	($42,522)	($378,353)	$67,620	$105,961	$106,730	$61,924	$67,190	$71,463	$106,836	$131,938	$58,245	$37,555	($20,045)	$10,276	$13,708	$13,775	$13,775	$13,775	$13,776	$19,211
CUMM. PRE-TAX CASH FLOWS	$000's	$472,840	($42,522)	($420,875)	($353,255)	($247,294)	($140,564)	($78,640)	($11,450)	$60,013	$166,849	$298,787	$357,032	$394,587	$374,542	$384,819	$398,527	$412,302	$426,077	$439,852	$453,628	$472,840
DD&A	$000's	474,464	8,446	80,464	85,905	86,098	86,511	78,512	9,332	4,271	4,445	4,333	4,083	1,884	1,708	1,546	1,306	1,172	1,013	871	728	11,838
PROFIT BEFORE TAX	$000's	413,505	(8,446)	(85,904)	(8,141)	14,266	13,702	(25,298)	58,263	65,554	101,999	127,571	54,115	11,849	12,096	12,254	13,116	13,250	13,409	13,551	13,695	2,605
INCOME TAX - Australian & Northern Territories	$000's	163,583							660	17,761	37,278	49,280	34,925	8,400				2,707	3,921	3,964	4,007	680
NET PROFIT	$000's	$249,922	($8,446)	($85,904)	($8,141)	$14,266	$13,702	($25,298)	$57,602	$47,793	$64,721	$78,292	$19,190	$3,449	$12,096	$12,254	$13,116	$10,543	$9,488	$9,587	$9,688	$1,925
AFTER-TAX CASH FLOW	$000's	$252,782	($42,522)	($378,353)	$67,620	$105,961	$106,730	$61,924	$66,530	$32,174	$47,123	$57,837	$27,229	$37,555	($20,045)	$10,276	$13,708	$11,068	$9,854	$9,811	$9,769	$18,531
CUMM. AFTER-TAX CASH FLOW	$000's	$252,782	($42,522)	($420,875)	($353,255)	($247,294)	($140,564)	($78,640)	($12,110)	$20,064	$67,187	$125,024	$152,253	$189,808	$169,764	$180,040	$193,748	$204,816	$214,670	$224,481	$234,251	$252,782

19.12.3 Sensitivities Deviating from the Base Case

TABLE 19-43 presents a sensitivity of the 6.77 Mtpy Base Case in which the price of gold is increased to $1,200/toz Au and the US to Australian dollar exchange rate is increased to 0.90; 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 US$487.188 million for the project, again at a 5 percent discount rate. Pretax IRR for this scenario increases to 25.4% from 14.9%.

TABLE 19-44 presents a sensitivity that increases the ore production rate to 30Ktpd (10.6 Mtpy) at a gold price of $950/toz Au, a US to Australian currency exchange rate of 0.85, and constant 2010 US dollars. The life of this project increases from nine years to thirteen years due to a lower cutoff grade that result in a larger pit volume. Results for this Sensitivity Case scenario produce a before tax net present NPV of US$154.511 million for the project at a 5 percent discount rate. Pretax IRR for this scenario is 10.1%. Capital and preproduction costs, once again, occur primarily in the two years prior to commencement of operations (Years -2 and -1).

TABLE 19-45 presents sensitivity for the 30Ktpd Sensitivity Case above, in which the price of gold is increased to $1,200/toz Au and the US to Australian dollar exchange rate is increased to 0.90; with all other input parameters being held constant. Using these values for the price and currency exchange rate parameters results in a before tax NPV of US$631.013 million for the project, again at a 5 percent discount rate. Pretax IRR for this scenario increases to 21.0%

Tt believes that is important to note that although the economics resulting from the 30Ktpd scenarios are lower than those for the 18.5Ktpd case, nearly twice as much gold is produced (3,195,553 toz Au vs. 1,661,626 toz Au) by undertaking the larger project.

TABLE 19-46 summarizes the sensitivity of Net Present Value (NPV) of the projected cash flows to variations in Base 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 set at 0.90 for all situations in which a metal price of $1,200 or greater occurs. Results indicate that the Mt. Todd project, as modeled by the Base 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 19-43: Mt. Todd 6.77 Mtpy Base Case Sensitivity (US$1,200/toz Au Price)

PRETAX:	
IRR	25.4%
NPV0 (000'S)	$848,949
NPV5 (000'S)	$487,188
AVG ANNUAL CF (000's) PRODUCTION YEARS	$106,896
AVG ANNUAL CF (000's) LIFE OF MINE	$58,793
STRIPPING RATIO (WST:ORE)	2.37

AFTER-TAX:	
IRR	16.2%
NPV0 (000'S)	$418,574
NPV5 (000'S)	$198,966
AVG ANNUAL CF (000's) PRODUCTION YEARS	$67,771
AVG ANNUAL CF (000's) LIFE OF MINE	$37,274
PAYBACK PERIOD (YRS) FROM : START OF PRODUCTION	3.6
POST CLOSURE NET CASH FLOW:	$80,772

CAPITAL	
INITIAL CAPITAL (000'S)	$407,289
CONTINGENCY	$52,756
SUB-TOTAL	460,045
WORKING CAPITAL - 2011 - 2013	(6,917)
INITIAL CAP, PRE-PROD DEV & WORKING CAP	$453,128
SUSTAINING CAPITAL (000'S)	28,679
CONTINGENCY	4,302
TOTAL SUSTAINING CAPITAL	32,981
WORKING CAPITAL - 2014 - 2030	7,157
TOTAL MINE LIFE CAPITAL	$493,265

COSTS	
CASH OPER COST PER OUNCE	$493
TOTAL CASH COST PER OUNCE	$507
CAPITAL COST PER OUNCE	$260
TOTAL PRODUCTION COST PER OUNCE	$766
UNIT COSTS	
MINING COST ($/TONNE MINED)	$1.50
MINING COST ($/TONNE ORE)	$5.04
PROCESSING COST ($/TONNE ORE)	$7.86
G&A Cost ($/TONNE ORE)	$0.68
TOTAL OPERATING COSTS $/TONNE ORE	$13.59

PROJECT PRODUCTION SCHEDULE / GOLD GRADES AND CONTENT

			Project Year -2	-1	1	2	3	4	5	6	7	8	9
MINE													
ORE TONNAGE TO CRUSHER (000's)	ore tonnes	60,050			6,789	6,770	6,770	6,770	6,789	6,770	6,770	6,770	5,852
ORE GRADE	g Au/tonne	1.045			1.09	1.22	1.22	0.97	0.98	0.91	1.08	1.20	0.75
	toz Au/tonne	0.034			0.035	0.039	0.039	0.031	0.031	0.029	0.035	0.039	0.024
CONTAINED GOLD	g Au	63,027,314			7,381,696	8,271,100	8,228,457	6,535,370	6,621,247	6,148,847	7,342,132	8,114,680	4,383,787
	toz Au	2,026,374			237,327	265,922	264,551	210,117	212,878	197,690	236,055	260,893	140,942
WASTE TONNAGE MINED (000's)	waste tonnes	142,524		5,583	27,898	25,095	23,519	29,948	17,573	8,717	3,685	504	2
TOTAL MATERIAL MINED	total tonnes	202,573		5,583	34,687	31,865	30,289	36,718	24,362	15,487	10,455	7,274	5,855
STRIPPING RATIO	waste : ore	2.4			4.11	3.71	3.47	4.42	2.59	1.29	0.54	0.07	0.00
MILL													
ORE TONNAGE TO MILL (000's)	ore tonnes	60,050			6,789	6,770	6,770	6,770	6,789	6,770	6,770	6,770	5,852
MILL FEED GRADE	g Au/tonne	1.045			1.09	1.22	1.22	0.97	0.98	0.91	1.08	1.20	0.75
	toz Au/tonne	0.034			0.035	0.039	0.039	0.031	0.031	0.029	0.035	0.039	0.024
CONTAINED GOLD	g Au	63,027,314			7,381,696	8,271,100	8,228,457	6,535,370	6,621,247	6,148,847	7,342,132	8,114,680	4,383,787
	toz Au	2,026,374			237,327	265,922	264,551	210,117	212,878	197,690	236,055	260,893	140,942
MILL RECOVERY @ 82%	% recovery of Au	82%			82%	82%	82%	82%	82%	82%	82%	82%	82%
GOLD RECOVERED	g Au	51,682,398			6,052,990	6,782,302	6,747,335	5,359,003	5,429,422	5,042,055	6,020,548	6,654,038	3,594,705
	toz Au	1,661,626			194,608	218,056	216,932	172,296	174,560	162,106	193,565	213,932	115,572
REFINERY													
PAYABLE GOLD TO REFINERY	g Au	51,682,398			6,052,990	6,782,302	6,747,335	5,359,003	5,429,422	5,042,055	6,020,548	6,654,038	3,594,705
	toz Au	1,661,626			194,608	218,056	216,932	172,296	174,560	162,106	193,565	213,932	115,572

		Total LOM	-2	-1	1	2	3	4	5	6	7	8	9
GOLD PRICE	$/oz	$1,200			$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200
WASTE TONNES	000's	142,524		5,583	27,898	25,095	23,519	29,948	17,573	8,717	3,685	504	2
TONNES ORE TO MILL	000's	60,050			6,789	6,770	6,770	6,770	6,789	6,770	6,770	6,770	5,852
STRIPPING RATIO	waste:ore	2.37			4.11	3.71	3.47	4.42	2.59	1.29	0.54	0.07	0.00
OUNCES PAYABLE	toz Au.	1,661,626			194,608	218,056	216,932	172,296	174,560	162,106	193,565	213,932	115,572
GOLD GRADE	g/tonne	1.050			1.087	1.222	1.215	0.965	0.975	0.908	1.085	1.199	0.749
GROSS GOLD SALES	$000's	$1,993,952			$233,530	$261,667	$260,318	$206,755	$209,472	$194,527	$232,278	$256,719	$138,687
RENTAL INCOME/POWER INCOME	$000's	$187,837			$5,317	$5,317	$5,317	$5,317	$5,317	$5,317	$5,317	$5,317	$5,317
GROSS REVENUE	$000's	$2,181,789			$238,846	$266,984	$265,635	$212,072	$214,789	$199,844	$237,595	$262,035	$144,004
LESS REFINING, TRANS. & TREATMENT	$000's	7,477			876	981	976	775	786	729	871	963	520
REVENUE FROM SALES	$000's	2,174,311			237,971	266,003	264,659	211,296	214,003	199,114	236,724	261,073	143,484
LESS ROYALTY	$000's	19,940			2,335	2,617	2,603	2,068	2,095	1,945	2,323	2,567	1,387
NET REVENUE		$2,154,372			$235,635	$263,386	$262,055	$209,229	$211,908	$197,169	$234,401	$258,505	$142,097
NET REVENUE AFTER PRODUCTION	$139,986												
OPERATING COSTS													
MINE	$000's	302,923			45,817	46,149	45,657	49,084	40,046	28,884	23,189	17,699	6,398
MILL	$000's	472,123			52,858	52,670	52,710	52,671	52,977	52,774	52,729	52,729	45,684
G&A	$000's	40,753		4,075	4,075	4,075	4,075	4,075	4,075	4,075	4,075	4,075	4,075
RECLAMATION	$000's	57,735			2,585			220	36				
TOTAL OPERATING COSTS		$873,534		$4,075	$105,335	$102,895	$102,442	$106,050	$97,134	$85,733	$79,994	$74,503	$56,158
MILL OPERATING COSTS AFTER PRODUCTION	4,321												
RECLAMATION COSTS AFTER PRODCTION	54,894												
OPERATING MARGIN	$000's	$1,280,838		($4,075)	$130,300	$160,491	$159,613	$103,179	$114,774	$111,436	$154,407	$184,003	$85,939
CAPITAL COSTS													
MINE EQUIPMENT	$000's	94,096		64,255	24,755	27		413	4,619	27			
PLANT EQUIPMENT & CONSTRUCTION	$000's	313,726	30,983	267,074	856	810	1,796	1,528	974	1,628	1,592	1,311	853
OTHER/CONTINGENCY	$000's	85,204	13,018	57,507	1,597	126	269	291	839	248	239	197	128
SUB-TOTAL	$000's	$493,026	$44,001	$388,836	$27,208	$963	$2,065	$2,232	$6,432	$1,903	$1,831	$1,508	$981
SALVAGE VALUE	$000's	(61,377)											
TOTAL CAPITAL	$000's	$431,649	$44,001	$388,836	$27,208	$963	$2,065	$2,232	$6,432	$1,903	$1,831	$1,508	$981
CHANGES TO WORKING CAPITAL	$000's	240	306	2,563	(9,786)	2,667	49	(789)	883	848	821	574	477
PRE-TAX CASH FLOWS	$000's	$848,949	($44,307)	($395,474)	$112,878	$156,861	$157,499	$101,736	$107,459	$108,686	$151,755	$181,921	$84,481
CUMM. PRE-TAX CASH FLOWS	$000's	$848,949	($44,307)	($439,781)	($326,903)	($170,042)	($12,543)	$89,193	$196,652	$305,338	$457,093	$639,014	$723,494
DD&A	$000's	493,026	8,800	84,102	89,544	89,736	90,149	81,795	9,357	4,296	4,469	4,358	4,108
PROFIT BEFORE TAX	$000's	787,812	(8,800)	(89,769)	33,455	61,673	60,831	11,139	98,510	102,727	146,958	177,572	80,162
INCOME TAX - Australian & Northern Territories	$000's	316,239				3,912	19,139	16,274	29,269	43,705	54,692	69,251	49,749
NET PROFIT	$000's	$471,573	($8,800)	($89,769)	$33,455	$57,761	$41,693	($5,135)	$69,242	$59,022	$92,266	$108,320	$30,413
AFTER-TAX CASH FLOW	$000's	$418,574	($44,307)	($395,474)	$112,878	$149,961	$118,556	$90,839	$47,543	$47,764	$65,869	$78,706	$38,323
CUMM. AFTER-TAX CASH FLOW	$000's	$418,574	($44,307)	($439,781)	($326,903)	($176,942)	($58,386)	$32,453	$79,996	$127,760	$193,628	$272,334	$310,657

		10	11	12	13	14	15	16	17	18
GOLD PRICE	$/oz	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200
WASTE TONNES	000's									
TONNES ORE TO MILL	000's									
STRIPPING RATIO	waste:ore									
OUNCES PAYABLE	toz Au.									
GOLD GRADE	g/tonne									
GROSS GOLD SALES	$000's									
RENTAL INCOME/POWER INCOME	$000's	$15,554	$15,554	$15,554	$15,554	$15,554	$15,554	$15,554	$15,554	$15,554
GROSS REVENUE	$000's	$15,554	$15,554	$15,554	$15,554	$15,554	$15,554	$15,554	$15,554	$15,554
LESS REFINING, TRANS. & TREATMENT	$000's									
REVENUE FROM SALES	$000's	15,554	15,554	15,554	15,554	15,554	15,554	15,554	15,554	15,554
LESS ROYALTY	$000's									
NET REVENUE		$15,554	$15,554	$15,554	$15,554	$15,554	$15,554	$15,554	$15,554	$15,554
OPERATING COSTS										
MINE	$000's									
MILL	$000's	957	886	890	268	268	268	268	268	248
G&A	$000's									
RECLAMATION	$000's	14,949	35,413	2,501	338	338	338	338	338	338
TOTAL OPERATING COSTS		$15,906	$36,300	$3,391	$607	$607	$607	$607	$606	$586
OPERATING MARGIN	$000's	($352)	($20,746)	$12,163	$14,947	$14,947	$14,947	$14,947	$14,948	$14,968
CAPITAL COSTS										
MINE EQUIPMENT	$000's									
PLANT EQUIPMENT & CONSTRUCTION	$000's	957	886	890	268	268	268	268	268	248
OTHER/CONTINGENCY	$000's	143	133	133	40	40	40	40	40	10,135
SUB-TOTAL	$000's	$1,100	$1,019	$1,023	$308	$308	$308	$308	$308	$10,382
SALVAGE VALUE	$000's	(44,828)								(16,549)
TOTAL CAPITAL	$000's	($43,728)	$1,019	$1,023	$308	$308	$308	$308	$308	($6,167)
CHANGES TO WORKING CAPITAL	$000's	4,102	(2,681)	(0)	67				0	140
PRE-TAX CASH FLOWS	$000's	$39,274	($19,084)	$11,140	$14,572	$14,639	$14,639	$14,639	$14,640	$20,995
CUMM. PRE-TAX CASH FLOWS	$000's	$762,768	$743,684	$754,825	$769,397	$784,036	$798,675	$813,315	$827,955	$848,949
DD&A	$000's	1,909	1,732	1,571	1,331	1,196	1,038	896	753	11,887
PROFIT BEFORE TAX	$000's	12,688	12,935	13,093	13,955	14,090	14,248	14,390	14,534	3,419
INCOME TAX - Australian & Northern Territories	$000's	12,711				3,967	4,173	4,216	4,259	924
NET PROFIT	$000's	($22)	$12,935	$13,093	$13,955	$10,123	$10,075	$10,175	$10,275	$2,495
AFTER-TAX CASH FLOW	$000's	$39,274	($19,084)	$11,140	$14,572	$10,673	$10,466	$10,424	$10,382	$20,070
CUMM. AFTER-TAX CASH FLOW	$000's	$349,931	$330,847	$341,987	$356,560	$367,232	$377,699	$388,122	$398,504	$418,574

Table 19-44: 10.6 Mtpy Sensitivity Case (US$950/toz Au Price)

PRETAX:	
IRR	10.1%
NPV0 (000'S)	$513,231
NPV5 (000'S)	$154,511
AVG ANNUAL CF (000's) PRODUCTION YEARS	$34,775
AVG ANNUAL CF (000's) LIFE OF MINE	$25,662
STRIPPING RATIO (WST:ORE)	2.03

AFTER-TAX:	
IRR	6.2%
NPV0 (000'S)	$245,316
NPV5 (000'S)	$3,929
AVG ANNUAL CF (000's) PRODUCTION YEARS	$16,914
AVG ANNUAL CF (000's) LIFE OF MINE	$12,266
PAYBACK PERIOD (YRS) FROM : START OF PRODUCTION	22.3
POST CLOSURE NET CASH FLOW:	-$19,622

CAPITAL	
INITIAL CAPITAL (000'S)	$564,025
CONTINGENCY	$83,104
SUB-TOTAL	647,129
WORKING CAPITAL - 2011 - 2013	(9,886)
INITIAL CAP, PRE-PROD DEV & WORKING CAP	$637,244
SUSTAINING CAPITAL (000'S)	200,827
CONTINGENCY	30,124
TOTAL SUSTAINING CAPITAL	230,951
WORKING CAPITAL - 2014 - 2030	9,131
TOTAL MINE LIFE CAPITAL	$877,326

COSTS	
CASH OPER COST PER OUNCE	$525
TOTAL CASH COST PER OUNCE	$541
CAPITAL COST PER OUNCE	$252
TOTAL PRODUCTION COST PER OUNCE	$793

UNIT COSTS	
MINING COST ($/TONNE MINED)	$1.43
MINING COST ($/TONNE ORE)	$4.34
PROCESSING COST ($/TONNE ORE)	$7.14
G&A Cost ($/TONNE ORE)	$0.55
TOTAL OPERATING COSTS $/TONNE ORE	$12.02

PROJECT PRODUCTION SCHEDULE / GOLD GRADES AND CONTENT

			Project Year -2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
MINE																	
ORE TONNAGE TO CRUSHER (000's)	ore tonnes	139,175			10,980	10,950	10,950	10,950	10,980	10,950	10,950	10,950	10,980	10,950	10,950	10,950	7,685
ORE GRADE	g Au/tonne	0.871			0.90	1.16	0.80	0.94	0.69	0.87	1.13	0.62	0.74	0.98	0.63	0.83	1.12
	toz Au/tonne	0.028			0.029	0.037	0.026	0.030	0.022	0.028	0.036	0.020	0.024	0.031	0.020	0.027	0.036
CONTAINED GOLD	g Au	121,210,821			9,841,017	12,662,942	8,712,116	10,319,668	7,606,263	9,551,723	12,380,927	6,753,250	8,106,096	10,713,127	6,901,396	9,088,157	8,574,141
	toz Au	3,897,015			316,396	407,123	280,101	331,785	244,547	307,095	398,056	217,122	260,617	344,435	221,885	292,191	275,665
WASTE TONNAGE MINED (000's)	waste tonnes	282,369		5,224	24,287	19,608	24,002	25,294	25,777	23,112	24,382	27,282	26,476	20,483	27,283	8,688	472
TOTAL MATERIAL MINED (000's)	total tonnes	421,545		5,224	35,267	30,558	34,952	36,244	36,757	34,062	35,332	38,232	37,456	31,433	38,233	19,638	8,157
STRIPPING RATIO	waste : ore	2.0			2.21	1.79	2.19	2.31	2.35	2.11	2.23	2.49	2.41	1.87	2.49	0.79	0.06
MILL																	
ORE TONNAGE TO MILL (000's)	ore tonnes	139,175			10,980	10,950	10,950	10,950	10,980	10,950	10,950	10,950	10,980	10,950	10,950	10,950	7,685
MILL FEED GRADE	g Au/tonne	0.871			0.90	1.16	0.80	0.94	0.69	0.87	1.13	0.62	0.74	0.98	0.63	0.83	1.12
	toz Au/tonne	0.028			0.029	0.037	0.026	0.030	0.022	0.028	0.036	0.020	0.024	0.031	0.020	0.027	0.036
CONTAINED GOLD	g Au	121,210,821			9,841,017	12,662,942	8,712,116	10,319,668	7,606,263	9,551,723	12,380,927	6,753,250	8,106,096	10,713,127	6,901,396	9,088,157	8,574,141
	toz Au	3,897,015			316,396	407,123	280,101	331,785	244,547	307,095	398,056	217,122	260,617	344,435	221,885	292,191	275,665
MILL RECOVERY @ 82%	% recovery of Au	82%			82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%
GOLD RECOVERED	g Au	99,392,874			8,069,634	10,383,613	7,143,935	8,462,128	6,237,135	7,832,413	10,152,360	5,537,665	6,646,998	8,784,764	5,659,144	7,452,289	7,030,795
	toz Au	3,195,553			259,445	333,841	229,683	272,064	200,528	251,818	326,406	178,040	213,706	282,437	181,946	239,596	226,045
REFINERY																	
PAYABLE GOLD TO REFINERY	g Au	99,392,874			8,069,634	10,383,613	7,143,935	8,462,128	6,237,135	7,832,413	10,152,360	5,537,665	6,646,998	8,784,764	5,659,144	7,452,289	7,030,795
	toz Au	3,195,553			259,445	333,841	229,683	272,064	200,528	251,818	326,406	178,040	213,706	282,437	181,946	239,596	226,045

		Total LOM	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
GOLD PRICE	$/oz	$950			$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950	$950
WASTE TONNES	000's	282,369		5,224	24,287	19,608	24,002	25,294	25,777	23,112	24,382	27,282	26,476	20,483	27,283	8,688	472					
TONNES ORE TO MILL	000's	139,175			10,980	10,950	10,950	10,950	10,980	10,950	10,950	10,950	10,980	10,950	10,950	10,950	7,685					
STRIPPING RATIO	wst:ore	2.03			2.21	1.79	2.19	2.31	2.35	2.11	2.23	2.49	2.41	1.87	2.49	0.79	0.06					
OUNCES PAYABLE	ozs.	3,195,553			259,445	333,841	229,683	272,064	200,528	251,818	326,406	178,040	213,706	282,437	181,946	239,596	226,045					
GOLD GRADE	g/tonne	0.871			0.90	1.16	0.80	0.94	0.69	0.87	1.13	0.62	0.74	0.98	0.63	0.83	1.12					
GROSS GOLD SALES	$000's	$3,035,775			$246,472	$317,149	$218,199	$258,460	$190,502	$239,227	$310,085	$169,138	$203,021	$268,315	$172,848	$227,617	$214,743					
RENTAL INCOME/POWER INCOME	$000's	$112,205			$2,355	$2,355	$2,355	$2,355	$2,355	$2,355	$2,355	$2,355	$2,355	$2,355	$2,355	$2,355	$2,355	$16,318	$16,318	$16,318	$16,318	$16,318
GROSS REVENUE	$000's	$3,147,981			$248,827	$319,504	$220,553	$260,815	$192,857	$241,582	$312,440	$171,493	$205,375	$270,670	$175,203	$229,972	$217,098	$16,318	$16,318	$16,318	$16,318	$16,318
LESS REFINING, TRANS. & TREATMENT	$000's	14,380			1,168	1,502	1,034	1,224	902	1,133	1,469	801	962	1,271	819	1,078	1,017					
REVENUE FROM SALES	$000's	3,133,601			247,660	318,001	219,520	259,591	191,955	240,449	310,972	170,692	204,414	269,399	174,384	228,893	216,081	16,318	16,318	16,318	16,318	16,318
LESS ROYALTY	$000's	30,358			2,465	3,171	2,182	2,585	1,905	2,392	3,101	1,691	2,030	2,683	1,728	2,276	2,147					
NET REVENUE		$3,103,243			$245,195	$314,830	$217,338	$257,006	$190,050	$238,056	$307,871	$169,000	$202,384	$266,716	$172,656	$226,617	$213,933	$16,318	$16,318	$16,318	$16,318	$16,318
NET REVENUE AFTER PRODUCTION	$81,591																					
OPERATING COSTS																						
MINE	$000's	603,499			46,352	46,696	47,867	49,455	50,324	51,014	53,988	52,228	54,211	47,473	53,024	36,300	14,567					
MILL	$000's	993,034			77,532	77,273	77,316	77,290	77,674	77,401	77,584	77,560	77,823	77,465	77,463	77,460	54,562	1,517	1,398	1,375	553	3,787
G&A	$000's	76,755		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					
RECLAMATION	$000's	113,474			2,585			220	36		14,346	3,158	545					15,419	60,559	7,246	619	8,740
TOTAL OPERATING COSTS		$1,786,762		$5,483	$131,952	$129,452	$130,665	$132,447	$133,517	$133,898	$151,401	$138,428	$138,061	$130,421	$135,969	$119,242	$74,612	$16,936	$61,957	$8,621	$1,172	$12,528
MILL OPERATING COSTS AFTER PRODUCTION	8,630																					
RECLAMATION COSTS AFTER PRODCTION	92,583																					
OPERATING MARGIN	$000's	$1,316,481		($5,483)	$113,243	$185,378	$86,673	$124,559	$56,532	$104,158	$156,470	$30,572	$64,322	$136,294	$36,687	$107,375	$139,322	($618)	($45,638)	$7,697	$15,146	$3,791
CAPITAL COSTS																						
MINE EQUIPMENT	$000's	127,222		63,978	24,755	27		3,249	10,281	5,689	2,825	3,239	2,835	7,482	2,835	27						
PLANT EQUIPMENT & CONSTRUCTION	$000's	602,837	46,780	408,706		946	718	714	723	98,964			45,286									
OTHER/CONTINGENCY	$000's	148,021	20,757	78,209	3,943	146	108	594	1,651	15,698	11,641	486	7,218	1,122	425	4						6,018
SUB-TOTAL	$000's	$878,080	$67,538	$550,893	$28,698	$1,119	$825	$4,558	$12,655	$120,351	$14,466	$3,725	$55,339	$8,604	$3,260	$31						$6,018
SALVAGE VALUE	$000's	(74,076)																(58,446)				(15,630)
TOTAL CAPITAL	$000's	$804,004	$67,538	$550,893	$28,698	$1,119	$825	$4,558	$12,655	$120,351	$14,466	$3,725	$55,339	$8,604	$3,260	$31		($58,446)				($9,612)
CHANGES TO WORKING CAPITAL	$000's	(754)	470	3,505	(13,860)	809	1,426	220	61	(199)	530	3,885	(2,385)	(585)	(3,237)	264	3,896	5,447	5	(0)	(33)	(973)
PRE-TAX CASH FLOWS	$000's	$513,231	($68,007)	($559,881)	$98,406	$183,449	$84,421	$119,782	$43,816	($15,993)	$141,473	$22,963	$11,369	$128,275	$36,663	$107,081	$135,425	$52,381	($45,644)	$7,697	$15,179	$14,375
CUMM. PRE-TAX CASH FLOWS	$000's	$513,231	($68,007)	($627,888)	($529,483)	($346,033)	($261,612)	($141,831)	($98,014)	($114,008)	$27,466	$50,429	$61,797	$190,072	$226,736	$333,816	$469,242	$521,623	$475,979	$483,676	$498,856	$513,231
DD&A	$000's	878,080	14,554	120,964	126,704	126,927	127,092	113,450	11,561	29,892	32,561	33,141	43,297	41,386	17,968	15,238	14,493	3,425	1,705	1,053	889	1,779
PROFIT BEFORE TAX	$000's	438,401	(14,554)	(127,853)	(20,368)	50,225	(49,828)	1,573	35,113	65,098	128,744	(9,702)	11,488	86,447	8,427	86,851	122,633	11,376	13,216	13,891	14,876	10,752
INCOME TAX - Australian & Northern Territories	$000's	192,707									48,790		7,386	39,241	7,495	38,087	51,708					
NET PROFIT	$000's	$245,694	($14,554)	($127,853)	($20,368)	$50,225	($49,828)	$1,573	$35,113	$65,098	$79,953	($9,702)	$4,102	$47,206	$931	$48,764	$70,925	$11,376	$13,216	$13,891	$14,876	$10,752
AFTER-TAX CASH FLOW	$000's	$245,316	($68,007)	($559,881)	$98,406	$183,449	$84,421	$119,782	$43,816	($15,993)	$73,480	$22,963	$1,341	$73,650	$26,482	$54,072	$63,347	$52,381	($45,644)	$7,697	$15,179	$14,375
CUMM. AFTER-TAX CASH FLOW	$000's	$245,316	($68,007)	($627,888)	($529,483)	($346,033)	($261,612)	($141,831)	($98,014)	($114,008)	($40,528)	($17,565)	($16,224)	$57,426	$83,908	$137,980	$201,327	$253,708	$208,064	$215,762	$230,941	$245,316

Table 19-45: Mt. Todd 10.6 Mtpy Sensitivity Case (US$1,200/toz Au Price)

PRETAX:	
IRR	21.0%
NPV0 (000'S)	$1,219,633
NPV5 (000'S)	$631,013
AVG ANNUAL CF (000's) PRODUCTION YEARS	$81,551
AVG ANNUAL CF (000's) LIFE OF MINE	$60,982
STRIPPING RATIO (WST:ORE)	2.03

AFTER-TAX:	
IRR	13.0%
NPV0 (000'S)	$547,977
NPV5 (000'S)	$219,307
AVG ANNUAL CF (000's) PRODUCTION YEARS	$36,774
AVG ANNUAL CF (000's) LIFE OF MINE	$27,399
PAYBACK PERIOD (YRS) FROM : START OF PRODUCTION	3.8
POST CLOSURE NET CASH FLOW:	-$14,823

CAPITAL	
INITIAL CAPITAL (000'S)	$590,294
CONTINGENCY	$87,044
SUB-TOTAL	677,338
WORKING CAPITAL - 2011 - 2013	(9,852)
INITIAL CAP, PRE-PROD DEV & WORKING CAP	$667,486
SUSTAINING CAPITAL (000'S)	200,827
CONTINGENCY	30,124
TOTAL SUSTAINING CAPITAL	230,951
WORKING CAPITAL - 2014 - 2030	9,097
TOTAL MINE LIFE CAPITAL	$907,533

COSTS	
CASH OPER COST PER OUNCE	$545
TOTAL CASH COST PER OUNCE	$563
CAPITAL COST PER OUNCE	$260
TOTAL PRODUCTION COST PER OUNCE	$824
UNIT COSTS	
MINING COST ($/TONNE MINED)	$1.43
MINING COST ($/TONNE ORE)	$4.34
PROCESSING COST ($/TONNE ORE)	$7.54
G&A Cost ($/TONNE ORE)	$0.58
TOTAL OPERATING COSTS $/TONNE ORE	$12.46

PROJECT PRODUCTION SCHEDULE / GOLD GRADES AND CONTENT

			Project Year -2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
MINE																	
ORE TONNAGE TO CRUSHER (000's)	ore tonnes	139,175			10,980	10,950	10,950	10,950	10,980	10,950	10,950	10,950	10,980	10,950	10,950	10,950	7,685
ORE GRADE	g Au/tonne	0.871			0.90	1.16	0.80	0.94	0.69	0.87	1.13	0.62	0.74	0.98	0.63	0.83	1.12
	toz Au/tonne	0.028			0.029	0.037	0.026	0.030	0.022	0.028	0.036	0.020	0.024	0.031	0.020	0.027	0.036
CONTAINED GOLD	g Au	121,210,821			9,841,017	12,662,942	8,712,116	10,319,668	7,606,263	9,551,723	12,380,927	6,753,250	8,106,096	10,713,127	6,901,396	9,088,157	8,574,141
	toz Au	3,897,015			316,396	407,123	280,101	331,785	244,547	307,095	398,056	217,122	260,617	344,435	221,885	292,191	275,665
WASTE TONNAGE MINED (000's)	waste tonnes	282,369		5,224	24,287	19,608	24,002	25,294	25,777	23,112	24,382	27,282	26,476	20,483	27,283	8,688	472
TOTAL MATERIAL MINED (000's)	total tonnes	421,545		5,224	35,267	30,558	34,952	36,244	36,757	34,062	35,332	38,232	37,456	31,433	38,233	19,638	8,157
STRIPPING RATIO	waste : ore	2.0			2.21	1.79	2.19	2.31	2.35	2.11	2.23	2.49	2.41	1.87	2.49	0.79	0.06
MILL																	
ORE TONNAGE TO MILL (000's)	ore tonnes	139,175			10,980	10,950	10,950	10,950	10,980	10,950	10,950	10,950	10,980	10,950	10,950	10,950	7,685
MILL FEED GRADE	g Au/tonne	0.871			0.90	1.16	0.80	0.94	0.69	0.87	1.13	0.62	0.74	0.98	0.63	0.83	1.12
	toz Au/tonne	0.028			0.029	0.037	0.026	0.030	0.022	0.028	0.036	0.020	0.024	0.031	0.020	0.027	0.036
CONTAINED GOLD	g Au	121,210,821			9,841,017	12,662,942	8,712,116	10,319,668	7,606,263	9,551,723	12,380,927	6,753,250	8,106,096	10,713,127	6,901,396	9,088,157	8,574,141
	toz Au	3,897,015			316,396	407,123	280,101	331,785	244,547	307,095	398,056	217,122	260,617	344,435	221,885	292,191	275,665
MILL RECOVERY @ 82%	% recovery of Au	82%			82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%	82%
GOLD RECOVERED	g Au	99,392,874			8,069,634	10,383,613	7,143,935	8,462,128	6,237,135	7,832,413	10,152,360	5,537,665	6,646,998	8,784,764	5,659,144	7,452,289	7,030,795
	toz Au	3,195,553			259,445	333,841	229,683	272,064	200,528	251,818	326,406	178,040	213,706	282,437	181,946	239,596	226,045
REFINERY																	
PAYABLE GOLD TO REFINERY	g Au	99,392,874			8,069,634	10,383,613	7,143,935	8,462,128	6,237,135	7,832,413	10,152,360	5,537,665	6,646,998	8,784,764	5,659,144	7,452,289	7,030,795
	toz Au	3,195,553			259,445	333,841	229,683	272,064	200,528	251,818	326,406	178,040	213,706	282,437	181,946	239,596	226,045

		Total LOM	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
GOLD PRICE	$/oz	$1,200			$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200	$1,200
WASTE TONNES	000's	282,369		5,224	24,287	19,608	24,002	25,294	25,777	23,112	24,382	27,282	26,476	20,483	27,283	8,688	472					
TONNES ORE TO MILL	000's	139,175			10,980	10,950	10,950	10,950	10,980	10,950	10,950	10,950	10,980	10,950	10,950	10,950	7,685					
STRIPPING RATIO	wst:ore	2.03			2.21	1.79	2.19	2.31	2.35	2.11	2.23	2.49	2.41	1.87	2.49	0.79	0.06					
OUNCES PAYABLE	ozs.	3,195,553			259,445	333,841	229,683	272,064	200,528	251,818	326,406	178,040	213,706	282,437	181,946	239,596	226,045					
GOLD GRADE	g/tonne	0.871			0.90	1.16	0.80	0.94	0.69	0.87	1.13	0.62	0.74	0.98	0.63	0.83	1.12					
GROSS GOLD SALES	$000's	$3,834,663			$311,333	$400,609	$275,619	$326,476	$240,634	$302,181	$391,687	$213,648	$256,447	$338,924	$218,335	$287,516	$271,254					
RENTAL INCOME/POWER INCOME	$000's	$117,276			$2,376	$2,376	$2,376	$2,376	$2,376	$2,376	$2,376	$2,376	$2,376	$2,376	$2,376	$2,376	$2,376	$17,278	$17,278	$17,278	$17,278	$17,278
GROSS REVENUE	$000's	$3,951,940			$313,709	$402,985	$277,995	$328,852	$243,010	$304,557	$394,063	$216,024	$258,823	$341,300	$220,711	$289,892	$273,630	$17,278	$17,278	$17,278	$17,278	$17,278
LESS REFINING, TRANS. & TREATMENT	$000's	14,380			1,168	1,502	1,034	1,224	902	1,133	1,469	801	962	1,271	819	1,078	1,017					
REVENUE FROM SALES	$000's	3,937,560			312,542	401,482	276,961	327,628	242,108	303,424	392,594	215,223	257,861	340,029	219,892	288,813	272,613	17,278	17,278	17,278	17,278	17,278
LESS ROYALTY	$000's	38,347			3,113	4,006	2,756	3,265	2,406	3,022	3,917	2,136	2,564	3,389	2,183	2,875	2,713					
NET REVENUE		$3,899,213			$309,428	$397,476	$274,205	$324,363	$239,701	$300,402	$388,677	$213,086	$255,297	$336,639	$217,708	$285,938	$269,900	$17,278	$17,278	$17,278	$17,278	$17,278
NET REVENUE AFTER PRODUCTION	$86,391																					
OPERATING COSTS																						
MINE	$000's	603,499			46,352	46,696	47,867	49,455	50,324	51,014	53,988	52,228	54,211	47,473	53,024	36,300	14,567					
MILL	$000's	1,049,953			82,022	81,752	81,794	81,768	82,165	81,880	82,063	82,038	82,313	81,944	81,941	81,938	57,705	1,517	1,398	1,375	553	3,787
G&A	$000's	81,270		5,805	5,805	5,805	5,805	5,805	5,805	5,805	5,805	5,805	5,805	5,805	5,805	5,805	5,805					
RECLAMATION	$000's	113,474			2,585			220	36		14,346	3,158	545					15,419	60,559	7,246	619	8,740
TOTAL OPERATING COSTS		$1,848,197		$5,805	$136,765	$134,253	$135,466	$137,248	$138,330	$138,699	$156,202	$143,229	$142,874	$135,222	$140,770	$124,043	$78,077	$16,936	$61,957	$8,621	$1,172	$12,528
MILL OPERATING COSTS AFTER PRODUCTION	8,630																					
RECLAMATION COSTS AFTER PRODCTION	92,583																					
OPERATING MARGIN	$000's	$2,051,016		($5,805)	$172,664	$263,223	$138,739	$187,115	$101,371	$161,703	$232,475	$69,857	$112,422	$201,418	$76,938	$161,895	$191,823	$342	($44,679)	$8,657	$16,106	$4,751
CAPITAL COSTS																						
MINE EQUIPMENT	$000's	127,272		64,028	24,755	27		3,249	10,281	5,689	2,825	3,239	2,835	7,482	2,835	27						
PLANT EQUIPMENT & CONSTRUCTION	$000's	628,840	49,290	432,198		946	718	714	723	98,964			45,286									
OTHER/CONTINGENCY	$000's	152,177	21,382	81,741	3,943	146	108	594	1,651	15,698	11,641	486	7,218	1,122	425	4						6,018
SUB-TOTAL	$000's	$908,289	$70,672	$577,968	$28,698	$1,119	$825	$4,558	$12,655	$120,351	$14,466	$3,725	$55,339	$8,604	$3,260	$31						$6,018
SALVAGE VALUE	$000's	(76,150)																(59,601)				(16,549)
TOTAL CAPITAL	$000's	$832,139	$70,672	$577,968	$28,698	$1,119	$825	$4,558	$12,655	$120,351	$14,466	$3,725	$55,339	$8,604	$3,260	$31		($59,601)				($10,531)
CHANGES TO WORKING CAPITAL	$000's	(755)	492	3,674	(14,018)	809	1,426	220	61	(199)	530	3,885	(2,385)	(585)	(3,237)	264	3,887	5,422	5	(0)	(33)	(973)
PRE-TAX CASH FLOWS	$000's	$1,219,633	($71,164)	($587,447)	$157,983	$261,295	$136,488	$182,337	$88,655	$41,552	$217,479	$62,248	$59,469	$193,398	$76,915	$161,601	$187,936	$54,521	($44,684)	$8,657	$16,139	$16,255
CUMM. PRE-TAX CASH FLOWS	$000's	$1,219,633	($71,164)	($658,610)	($500,627)	($239,332)	($102,844)	$79,493	$168,148	$209,699	$427,178	$489,426	$548,895	$742,293	$819,208	$980,809	$1,168,745	$1,223,266	$1,178,582	$1,187,239	$1,203,379	$1,219,633
DD&A	$000's	908,289	15,181	126,849	132,588	132,812	132,977	118,708	11,614	29,944	32,613	33,193	43,350	41,439	18,020	15,290	14,546	3,478	1,757	1,105	942	1,883
PROFIT BEFORE TAX	$000's	1,142,728	(15,181)	(134,060)	33,167	122,185	(3,647)	58,871	79,899	122,590	204,697	29,530	59,535	151,518	48,626	141,319	175,082	12,283	14,124	14,798	15,784	11,607
INCOME TAX - Australian & Northern Territories	$000's	482,455				9,132	2,094	28,197	37,093	54,593	82,969	14,970	28,474	66,352	24,243	60,779	73,559					
NET PROFIT	$000's	$660,273	($15,181)	($134,060)	$33,167	$113,053	($5,741)	$30,674	$42,806	$67,997	$121,727	$14,560	$31,061	$85,166	$24,383	$80,540	$101,522	$12,283	$14,124	$14,798	$15,784	$11,607
AFTER-TAX CASH FLOW	$000's	$547,977	($71,164)	($587,447)	$157,983	$248,826	$133,870	$143,177	$37,040	($34,564)	$101,637	$41,605	$19,920	$100,821	$43,290	$76,825	$85,268	$54,521	($44,684)	$8,657	$16,139	$16,255
CUMM. AFTER-TAX CASH FLOW	$000's	$547,977	($71,164)	($658,610)	($500,627)	($251,801)	($117,931)	$25,246	$62,286	$27,722	$129,359	$170,964	$190,884	$291,705	$334,995	$411,820	$497,089	$551,610	$506,926	$515,583	$531,722	$547,977

TABLE 19-46: Pretax Sensitivity Analyses - Mt. Todd Gold Project

PRETAX SENSITIVITY ANALYSES - MT. TODD GOLD PROJECT

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

PRICE ($/oz)	IRR	NPV(0)	NPV(5)
$ 850	9.7%	$308,114	$85,910
$ 900	12.3%	$390,364	$148,027
$ 950	14.9%	$472,615	$210,144
$ 1,000	17.5%	$554,865	$272,260
$ 1,050	20.1%	$637,116	$334,377
$ 1,100	22.6%	$719,366	$396,494
$ 1,150	25.1%	$801,617	$458,611
$ 1,200	25.3%	$847,449	$485,688
$ 1,250	27.7%	$929,700	$547,804
$ 1,300	30.0%	$1,011,950	$609,921
$ 1,350	32.3%	$1,094,201	$672,038
$ 1,400	21.3%	$555,033	$305,160
$ 1,450	36.9%	$1,258,702	$796,271
$ 1,500	39.1%	$1,340,952	$858,388

Net Present Value Calculations ($000s)
Operating Cost Sensitivity, Au@ $950

	IRR	NPV(0)	NPV(5)
+20%	9.7%	$312,754	$87,426
+10%	12.3%	$392,684	$148,785
0%	14.9%	$472,615	$210,144
-10%	17.6%	$552,545	$271,502
-20%	20.2%	$632,475	$332,861

Net Present Value Calculations ($000s)
Capital Cost Sensitivity, Au @ $950

	IRR	NPV(0)	NPV(5)
+20%	10.7%	$395,503	$129,282
+10%	12.7%	$434,059	$169,713
0%	14.9%	$472,615	$210,144
-10%	17.6%	$511,170	$250,574
-20%	20.8%	$549,726	$291,005

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

US : AU	IRR	NPV(0)	NPV(5)
0.75 : 1.00	18.9%	$542,900	$277,287
0.80 : 1.00	16.8%	$507,757	$243,715
0.85 : 1.00	14.9%	$472,615	$210,144
0.90 : 1.00	13.2%	$437,472	$176,572
0.95 : 1.00	11.6%	$402,329	$143,001
1.00 : 1.00	10.1%	$367,187	$109,429
1.05 : 1.00	8.7%	$332,044	$75,858
1.10 : 1.00	7.4%	$296,901	$42,286
1.15 : 1.00	6.3%	$261,759	$8,715
1.20 : 1.00	5.2%	$226,616	($24,857)
1.25 : 1.00	4.2%	$191,474	($58,428)

AFTERTAX SENSITIVITY ANALYSES - MT. TODD GOLD PROJECT

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

PRICE ($/oz)	IRR	NPV(0)	NPV(5)
$ 850	6.5%	$181,735	$9,364
$ 900	8.1%	$217,531	$40,937
$ 950	9.8%	$252,490	$71,127
$ 1,000	11.4%	$287,598	$100,497
$ 1,050	13.0%	$323,975	$130,247
$ 1,100	14.5%	$359,190	$158,192
$ 1,150	16.0%	$394,987	$186,429
$ 1,200	16.1%	$417,949	$198,183
$ 1,250	17.4%	$452,220	$224,927
$ 1,300	18.7%	$486,491	$251,671
$ 1,350	20.0%	$520,762	$278,416
$ 1,400	34.6%	$1,176,451	$734,155
$ 1,450	22.5%	$589,304	$331,904
$ 1,500	23.8%	$623,852	$358,836

Net Present Value Calculations ($000s)
Operating Cost Sensitivity, Au@ $950

	IRR	NPV(0)	NPV(5)
+20%	6.5%	$183,424	$9,019
+10%	8.1%	$218,375	$40,815
0%	9.8%	$252,490	$71,127
-10%	11.4%	$286,754	$100,573
-20%	13.1%	$323,231	$131,082

Net Present Value Calculations ($000s)
Capital Cost Sensitivity, Au @ $950

	IRR	NPV(0)	NPV(5)
+20%	7.1%	$224,137	$23,553
+10%	8.3%	$238,732	$47,758
0%	9.8%	$252,490	$71,127
-10%	11.5%	$266,397	$93,805
-20%	13.5%	$281,021	$116,263

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

US : AU	IRR	NPV(0)	NPV(5)
0.75 : 1.00	12.2%	$275,745	$103,856
0.80 : 1.00	10.9%	$264,523	$87,788
0.85 : 1.00	9.8%	$252,490	$71,127
0.90 : 1.00	8.7%	$240,554	$53,916
0.95 : 1.00	7.7%	$227,774	$36,076
1.00 : 1.00	6.9%	$216,375	$18,502
1.05 : 1.00	6.1%	$205,074	$983
1.10 : 1.00	5.4%	$191,988	($18,050)
1.15 : 1.00	4.7%	$180,635	($36,331)
1.20 : 1.00	4.2%	$169,282	($54,613)
1.25 : 1.00	3.6%	$155,986	($74,612)

Note: All sensitivities run at gold prices of $1200 / toz Au and greater use a 0.90 : 1.00, US to AUS exchange rate.

20.0 INTERPRETATION AND CONCLUSIONS

20.1 Interpretation

It is Tt'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% 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, Tt, 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 form 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 Tt's review of the database, previous studies and work products, and as an outgrowth of the recent mineral resource modeling, PEA, and this PFS, Tt 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 drillhole 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.
- Advance the Quigleys deposit through feasibility in order to evaluate and optimize its potential impact on the early production years.

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. Tt 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 is 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.

21.1.2 Mining

Tt 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);
- Trade off Studies: Economy of scale (large scale mining methods);
- 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 high pressure grinding roll (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 Mount 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.
- Consider a vibrating grizzly before the MP800 standard cone crushers to scalp undersize from the standard cone feed thereby reducing total tonnes passing through these crushers.
- 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 in detail.
- 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 Mount 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 and filtration 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.
- A conceptual study of alternative processes may be appropriate in consideration of the copper contained in the ore, to confirm the selected flowsheet is the most cost-effective. Processes that may be considered include copper-selective flotation and subsequent cyanidation of flotation tailings, ammonia-cyanide leach, processes for recovery of copper and cyanide such as MNR, SART, Sceresini, ion-exchange resins or solvents. Options for cyanide detoxification should be addressed as part of this study.

21.1.4 Tailings and Geotechnical Design

- The material properties must be confirmed through a program of geotechnical drilling and laboratory testing;
- The geotechnical modeling must be updated based on any changes to material properties;
- A liquefaction analysis must be performed;
- Additional tailings testing and deposition modeling must be performed to verify the assumed in-place tailings density;
- The existing phreatic levels within the impoundment must be confirmed;
- A bathymetric survey should be performed on the raw water supply reservoir to develop a stage-storage relationship that can be used to validate assumptions made in the TSF water balance;
- The TSF water balance must be updated and expanded to include the Return Water Pond and Water Polishing Pond, as well as to optimize the size of the water pool to accommodate the water requirements of the processing facilities;
- The spillway design must be revised to account for any required change in cross-section at different stages of the impoundment;
- The condition of the existing toe drains, underdrains, and decant towers must be investigated to confirm their operation when tailings deposition resumes; and
- A pipe-crushing analysis should be performed on the existing installed pipes to ensure that the pipes will not crush under the additional loading by the tailings and the embankment raises.

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 US$1.8 to $3.2 million. Tt 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 US$500,000 to 750,000 depending on the number of wells that will need to be installed. Recent expansion of the site monitoring well network may eliminate some components of this program.

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% 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 US$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 for 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 US$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 US$7,500 and US$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 US$80,000 to US$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 US$50,000 and US$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 data should be reviewed and summarized. The estimated cost for data compilation is between US$5,000 and US$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, Tt 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

  - o 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:
  - o New WTP with a minimum ARD/ML treatment capacity of approximate 1000 m^3/hour;
  - o 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);
  - o LLDPE-lined (or equivalent) sludge disposal cell (Mine-life storage capacity ~ 75,000 m^3) for the dispose of water treatment sludge produced by the New WTP.

Tt 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, Tt recommends that a process water treatment and sludge management study (including bench-scale and pilot treatment plant testing and regulatory review) be considered prior to the feasibility study to determine the following:
  - o New WTP requirements, system capacity, and optimal location;
  - o Post-neutralization/clarification water quality;
  - o Optimal reagents;
  - o Reagent consumption;
  - o Sludge volumes, type, density, consolidation (settleability), handling, and disposal location and facility design;
  - o Optimization of New WTP operations necessary to accommodate declining flow volumes and potentially increasing acidity and TDS over the mine-life;
  - o The size and precise volume and design necessary to contain ARD/ML prior to treatment; and,
  - o 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.
- Based on land surveys, develop stage-storage relationships for all water storage ponds at Mt. Todd.

The estimated cost for feasibility-level water treatment studies is between US$500,000 and US$1,000,000 and is anticipated to take approximately 1 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.

21.1.5.1 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 characterized 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.

Tt 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. <u>Waste Rock</u> - 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. <u>Tailings</u> - 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 meters. 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 to that representative the range in 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 US$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 US$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. There currently is insufficient data to estimate the hydrologic regime at the Mt. Todd with a high degree of confidence. This information is critical for the design of operational and closure storm water management systems and the understanding of WRD, TSF and pit recharge and infilling characteristics and the potential to generate ARD/ML.

Tt 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 - Developed 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.

The estimated cost for these studies is between US$150,000 and 300,000.

Reclamation Material Inventory and Characterization

Tt 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 clays ;
- 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 and ancillary facilities; and,
- Uncontaminated material excavated for creation of the WRD, RP 1 and TSF diversions.

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
- Acid Base Accounting (additional analysis may be necessary if NNP < + 20 Tons $CaCO_3$ equivalent/1000 T Material or NP:AP < 3)

Inventories and chemical and physical characterization can be completed relatively quickly (i.e., ~6 months) at an estimated cost of US$50,000 to US$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 estimate cost for these studies is between US$50,000 and 100,000.

21.2 Planned Work Commitments

Vista, based on the above recommendations and their own work commitments, has developed a proposed work program to be completed during the next 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 Tt'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
October 2010

Description	Estimated Cost (Millions of US$)
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
Environmental, Permitting and Reclamation	1.5 to 3.0
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
   Suite 500
   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 "*PRELIMINARY FEASIBILITY STUDY – NI 43-101 TECHNCIAL REPORT - MT TODD GOLD PROJECT – NORTHERN TERRITORY, AUSTRALIA.*" and dated 01 October 2010 (the "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 1st Day of October, 2010.

John W. Rozelle, P.G.

Digitally signed by John W. Rozelle, P.G.
DN: cn=John W. Rozelle, P.G., o=Tetra Tech Inc., ou=Manager, Mineral Resource Division, email=john.rozelle@tetratech.com, c=US
Date: 2010.10.04 14:00:25 -06'00'

__________________________.
Signature of Qualified Person

"John W. Rozelle" .
Print name of Qualified Person

Stephen Krajewski, Ed. D., P.G.
Senior Geologist – Modeller -& GIS
Tetra Tech MM, Inc.
350 Indiana Street, Suite 500
Golden, Colorado 80401
Telephone: 303-217-5700
Facsimile: 303-217-5705
Email: steve.krajewski@tetratech.com

CERTIFICATE of AUTHOR

I, Stephen Krajewski., 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 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 “*PRELIMINARY FEASIBILITY STUDY – NI 43-101 TECHNCIAL REPORT - MT TODD GOLD PROJECT – NORTHERN TERRITORY, AUSTRALIA.*” and dated 01 October 2010 (the “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 1st Day of October 2010.

"**Stephen A. Krajewski**" .
Signature of Qualified Person

"Stephen A., Krajewski" .

Print name of Qualified Person

Edwin C. Lips, P.E.
Tetra Tech
350 Indiana Street, Suite 500
Golden, Colorado 80401
USA
Telephone: 303-217-5700
Email: ed.lips@tetratech.com

CERTIFICATE OF AUTHOR

I, Edwin C. Lips, do hereby certify that:

1. I am a Sr. Mining Engineer of:
   Tetra Tech
   350 Indiana Street, Suite 500
   Golden, Colorado 80401
   USA
2. This certificate relates to the "*PRELIMINARY FEASIBILITY STUDY – NI 43-101 TECHNCIAL REPORT - MT TODD GOLD PROJECT – NORTHERN TERRITORY, AUSTRALIA*" dated 01 October 2010.
3. I graduated from Montana Tech, Butte, Montana with a degree in Mining Engineering (BS) in 1982.
4. 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).
5. 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.
7. I am responsible for and prepared, or contributed to, sections 1.0, 18.0, and 19.0, and 21.0 of the report titled "*PRELIMINARY FEASIBILITY STUDY – NI 43-101 TECHNCIAL REPORT - MT TODD GOLD PROJECT – NORTHERN TERRITORY, AUSTRALIA.*" and dated 01 October 2010 (the "Preliminary Feasibility Report").
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. To the best of my knowledge, information and belief, the Preliminary Feasibility Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of the issuer applying all of the tests of Section 1.4 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Preliminary Feasibility Report has been prepared in compliance with that instrument and that form.
12. 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 1st Day of October 2010.

"Edwin C Lips"
Signature of Qualified Person

Edwin C. Lips
Print name of Qualified Person

D. Erik Spiller, QP
Principal Metallurgist
Tetra Tech MM, Inc.
350 Indiana Street, Suite 500
Golden, Colorado 80401
Telephone: 303-217-5700
Facsimile: 303-217-5705
Email: erik.spiller@tetratech.com

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 "*PRELIMINARY FEASIBILITY STUDY – NI 43-101 TECHNCIAL REPORT - MT TODD GOLD PROJECT – NORTHERN TERRITORY, AUSTRALIA.*" and dated 1 October 2010 (the "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 1st Day of October 2010.

___**"D Erik Spiller"**________**.**
Signature of Qualified Person

D. Erik Spiller .

Print name of Qualified Person

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 the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

4. I am responsible for the preparation of the Pit Design and Reserves sections 18.2 through 18.4 and sections 19.1 through 19.3 of this report titled "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 no prior involvement with the property.

6. 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 of which would make the Technical Report 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 1st day of October 2010.

"Thomas Dyer"

Thomas Dyer, P.E.

Print Name of Qualified Person

Deepak Malhotra
President
Resource Development Inc.
11475 W. I-70 Frontage Road North
Wheat Ridge, Colorado 80033
Telephone: 303-422-1176
Facsimile: 303-424-8580
Email: dmalhotra@aol.com

CERTIFICATE of AUTHOR

I, Deepak Malhotra do hereby certify that:

1. I am currently employed as President by:

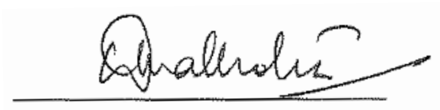
   Resource Development Inc. (RDi)
   11475 W. I-70 Frontage Road North
   Wheat Ridge, Colorado 80033

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 “*PRELIMINARY FEASIBILITY STUDY – NI 43-101 TECHNCIAL REPORT - MT TODD GOLD PROJECT – NORTHERN TERRITORY, AUSTRALIA.*” and dated 01 October 2010 (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 1st Day of October, 2010.



Signature of Qualified Person

"Deepak Malhotra" .

Print name of Qualified Person

24.0 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

Tt 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 G
MT TODD POWER STATION
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APPENDIX I
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APPENDIX J
MT TODD PROJECT – PROPOSED RECLAMATION AND CLOSURE PLAN BY TETRA TECH, 2010

APPENDIX K
TAILINGS STORAGE FACILITY TRADEOFF STUDY
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APPENDIX L
TAILINGS DESIGN STUDY
BY TETRA TECH, 2010