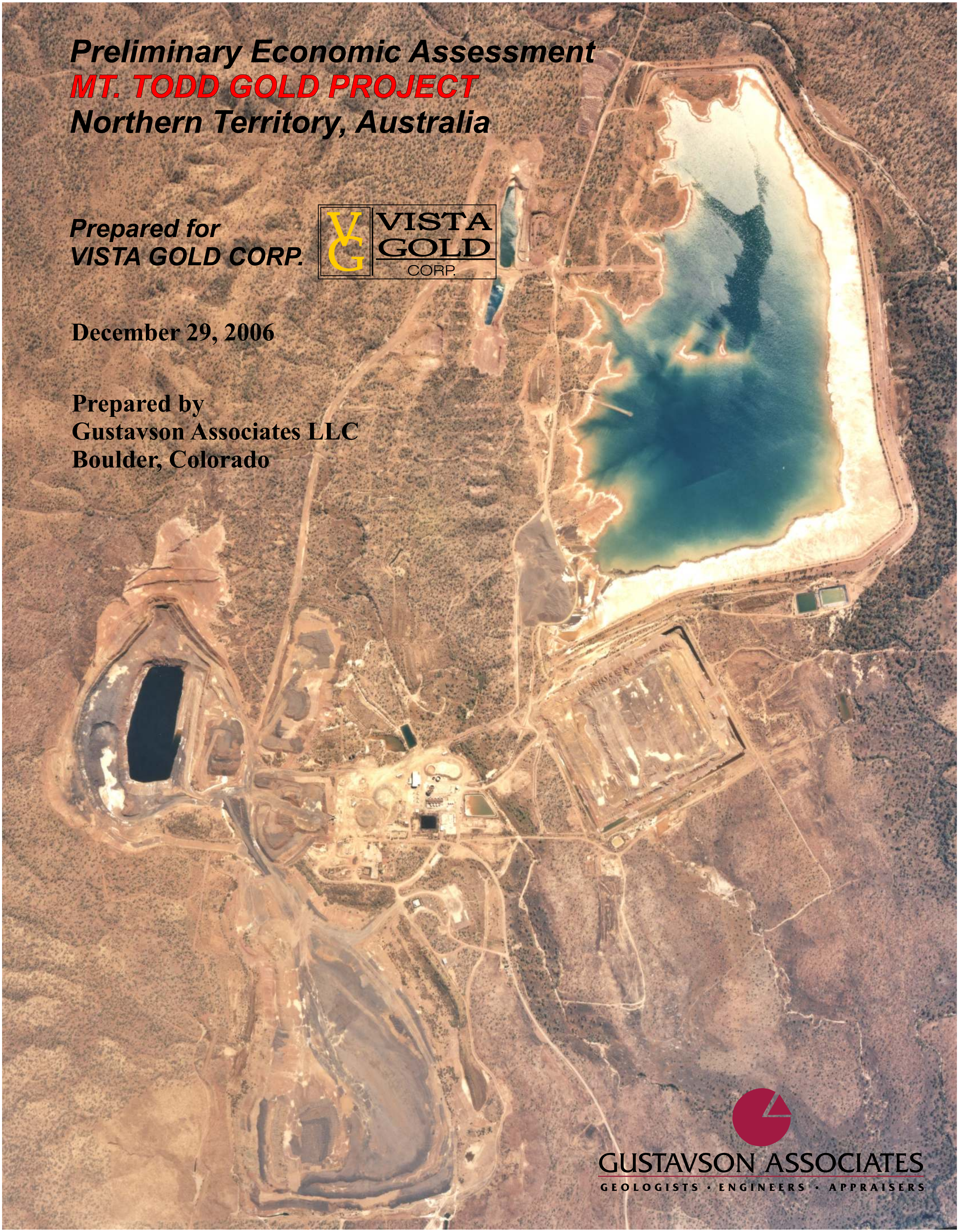
Preliminary Economic Assessment
MT. TODD GOLD PROJECT
Northern Territory, Australia



Prepared for
VISTA GOLD CORP.

VISTA GOLD CORP.

December 29, 2006

Prepared by
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Boulder, Colorado

GUSTAVSON ASSOCIATES
GEOLOGISTS • ENGINEERS • APPRAISERS

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1.0 SUMMARY

Gustavson Associates, LLC ("Gustavson") was commissioned by Vista Gold Corp. ("Vista") in August 2006 to prepare a Canadian National Instrument 43-101 (NI43-101) compliant Preliminary Economic Assessment Report on the Mt Todd Gold Project (the "Project") located in the 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. A NI43-101 Technical Report was completed on June 26, 2006. 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.

Location

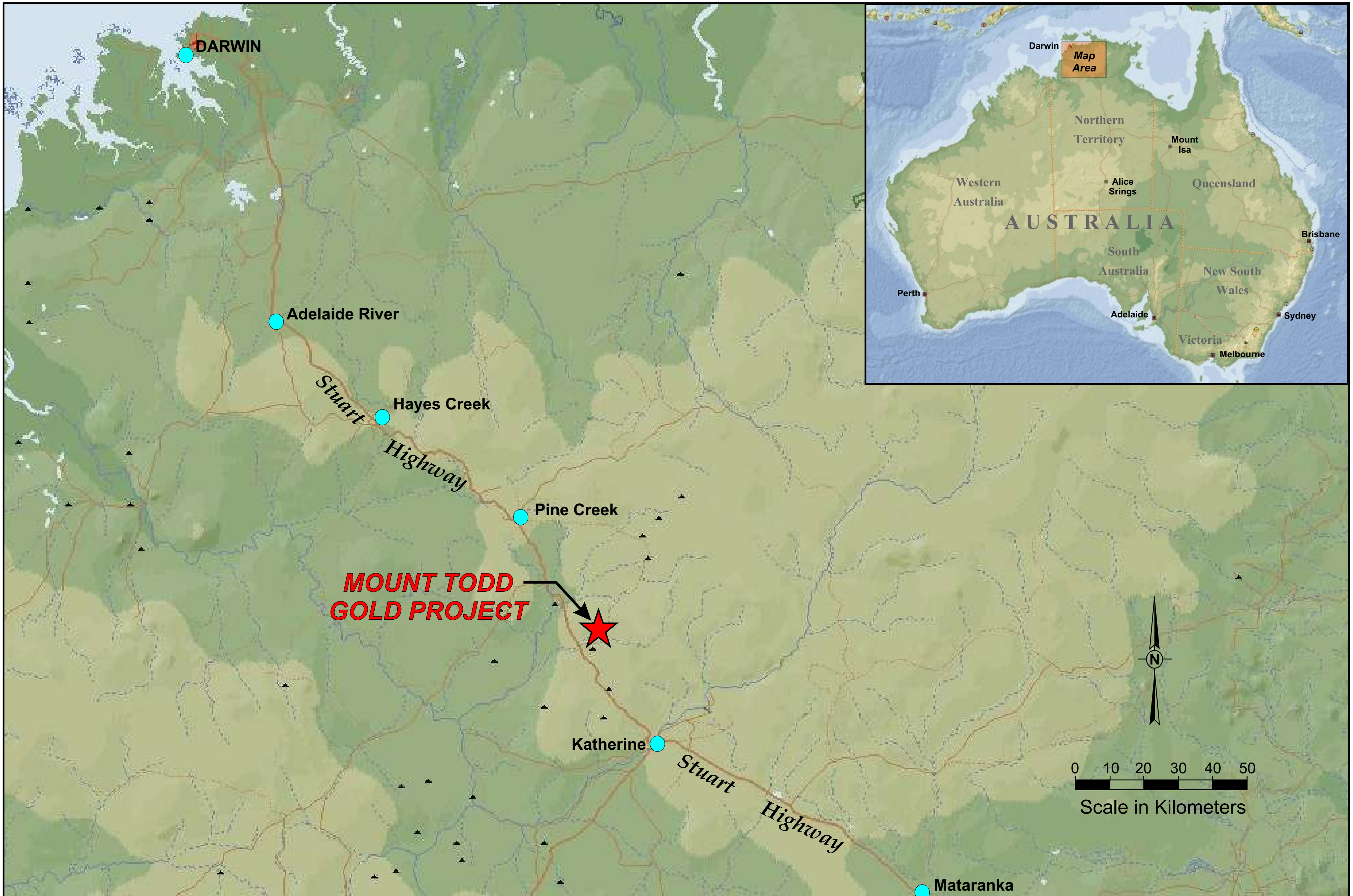
The Mt Todd Project is located 50 kilometers northwest of Katherine, and approximately 250 kilometers southeast of Darwin in the NT of Australia (FIGURE 1-1). Access to the property is via high quality, two-lane paved roads from the Stuart Highway, the main arterial within the territory.

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, 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 modelling techniques that "over-smoothed" the grades and poor sampling techniques of the blast holes. The improper modelling of the resource has been rectified in this report with the entire deposit being remodelled. 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 of this report. It is Gustavson's opinion that this information is very important when examining the Mt Todd Gold Project as envisaged by Vista.



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Date of Issue. 08/03/2006	Client Name. **VISTA GOLD CORP.**	FIGURE 1-1
Project No. VIS005	Project Name. Mt. Todd Gold Project	General Location Map
Drawing File Name. Fig1-1.cdr		Mount Todd Gold Project

TABLE 1-1 VISTA GOLD CORP. – MT TODD GOLD PROJECT Property History December 2006	
<u>1986</u> October 1986 – January 1987:	Conceptual Studies, Australia Gold PTY LTD (Billiton); Regional Screening; (Higgins), Ground Acquisition by Zapopan N.L.
<u>1987</u> 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)
<u>1988</u> 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.)
<u>1989</u> 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).
<u>1990</u> 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)
<u>1993 - 1997</u> 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.
<u>1999 - 2000</u> 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.
<u>2000 – 2006</u>	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.
<u>2006</u> March to Present	Vista Gold Corp. acquires concession rights from the Deed Administrators.

Ownership

The mineral leases consist of three individual tenements, MLN 1070, MLN1071, and MLN1127 comprising some 5,365 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.

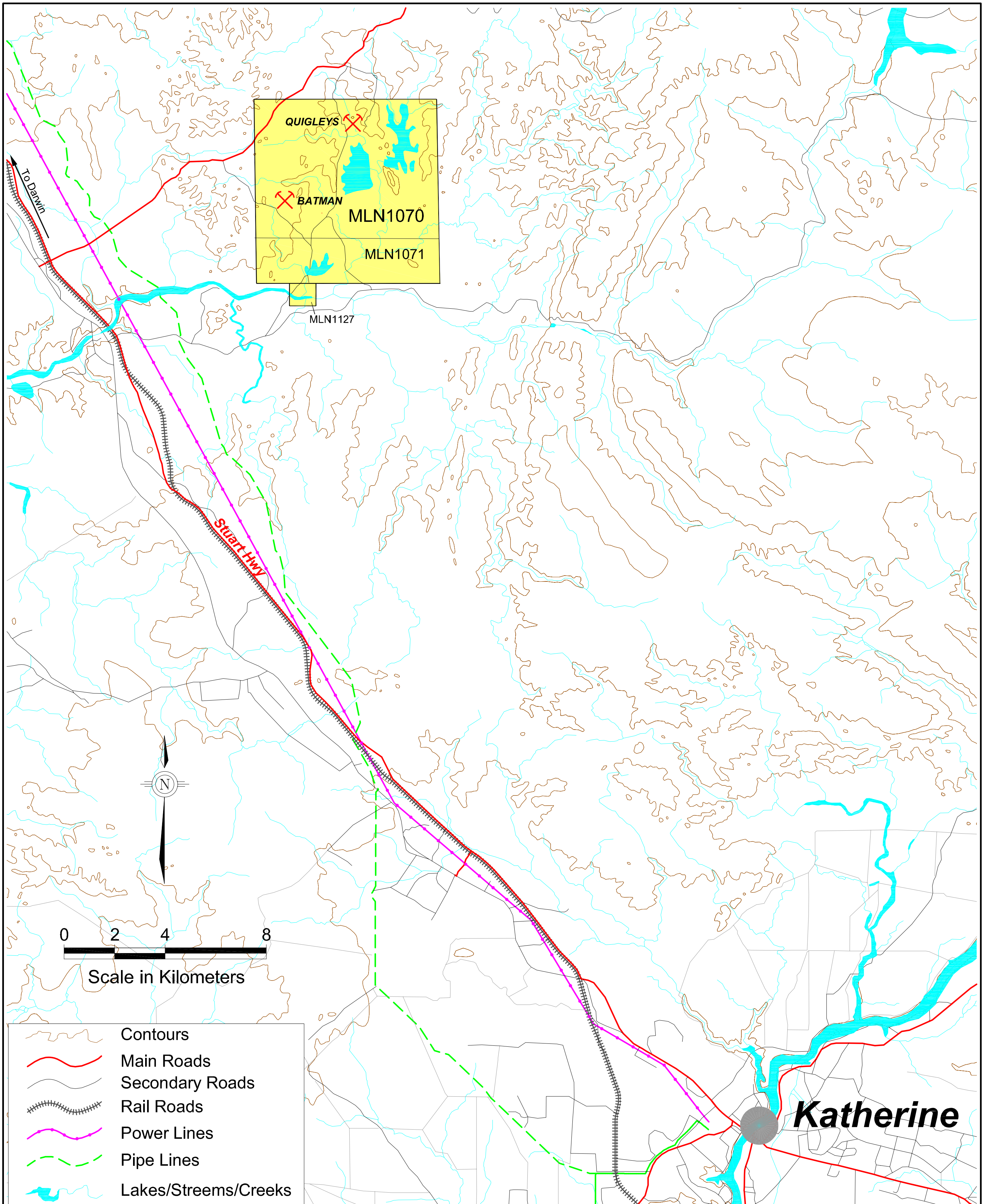
Geology

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 chalcophyrite, 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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GUSTAVSON ASSOCIATES GEOLOGISTS · ENGINEERS · APPRAISERS	DATE OF ISSUE. 08/03/2006	CLIENT NAME. **VISTA GOLD CORP.**	FIGURE I-2 CONCESSIONS AND INFRASTRUCTURE MAP
	PROJECT NO. VIS005	PROJECT NAME. MT. TODD GOLD PROJECT	
	DRAWING FILE NAME. Fig1-2_4-2.dwg		

Estimated Resources

At the present time, resources have only been estimated for the Batman Deposit. Gustavson created three-dimensional computerized geologic and grade models of the Batman Deposit. While the deposit model also contains the Quigleys Deposit, no geologic resource estimate has been made for this deposit at the present time.

The geologic model of the Batman Deposit was created by General Gold Corporation ("GGC") and audited by Gustavson. 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 GGC's own experience. It is Gustavson's opinion that the GGC's geologic model accurately portrays the geologic environment of the Batman Deposit.

Gustavson 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 Gustavson'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. 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
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Batman Resource Classification Criteria
December 2006

Category	Kriging Variance	No. of Sectors	No. of Points/Sector
Measured	JAS Model < 0.30	4	4 to 16
Indicated	JAS Model >=0.30<0.55	4	4 to 16
Inferred	Halo Model < 0.45	<4	2 to 8

TABLE 1-3 details the estimated in-place resources by classification and by cutoff grade for the Batman Deposit. All of the resources quoted are contained on Vista's mineral leases.

TABLE 1-3
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Batman Deposit Classified Resources
December 2006

Cutoff Grade g Au/t		INCREMENTAL			CUMULATIVE		
MEASURED RESOURCES							
		Tonnes (1000)	**Average Grade (g Au/t)**	**Contained Ounces**	**Tonnes (1000)**	**Average Grade (g Au/t)**	**Contained Ounces**
0.3	0.4	4,018	0.35	44,955	26,113	0.80	673,885
0.4	0.5	3,674	0.45	52,800	22,095	0.89	628,930
0.5	0.75	7,050	0.62	139,851	18,421	0.97	576,130
0.75	1.0	4,768	0.87	133,366	11,371	1.19	436,279
1.0	2.0	6,034	1.34	259,375	6,603	1.43	302,912
2.0	3.0	526	2.29	38,727	569	2.38	43,538
>3.0		43	3.48	4,811	43	3.48	4,811
INDICATED RESOURCES							
Cutoff Grade g Au/t		**INCREMENTAL**			**CUMULATIVE**		
		Tonnes (1000)	**Average Grade (g Au/t)**	**Contained Ounces**	**Tonnes (1000)**	**Average Grade (g Au/t)**	**Contained Ounces**
0.3	0.4	8,069	0.30	77,827	53,784	0.79	1,371,440
0.4	0.5	7,758	0.45	111,992	45,715	0.88	1,293,612
0.5	0.75	14,752	0.62	293,110	37,957	0.97	1,181,620
0.75	1.0	9,776	0.87	272,503	23,205	1.19	888,511
1.0	2.0	12,300	1.34	531,095	13,429	1.43	616,008
2.0	3.0	1,043	2.26	75,886	1,129	2.34	84,913
>3.0		86	3.27	9,028	86	3.27	9,028
MEASURED + INDICATED RESOURCES							
Cutoff Grade g Au/t		**INCREMENTAL**			**CUMULATIVE**		
		Tonnes (1000)	**Average Grade (g Au/t)**	**Contained Ounces**	**Tonnes (1000)**	**Average Grade (g Au/t)**	**Contained Ounces**
0.3	0.4	12,087	0.32	122,782	79,897	0.80	2,045,325
0.4	0.5	11,432	0.45	164,792	67,810	0.88	1,922,542
0.5	0.75	21,802	0.62	432,960	56,378	0.97	1,757,750
0.75	1.0	14,544	0.87	405,869	34,576	1.19	1,324,789
1.0	2.0	18,334	1.34	790,469	20,032	1.43	918,920
2.0	3.0	1,569	2.27	114,612	1,698	2.35	128,451
>3.0		129	3.34	13,839	129	3.34	13,839

INFERRED RESOURCES							
		INCREMENTAL			CUMULATIVE		
Cutoff Grade g Au/t		Tonnes (1000)	Average Grade (g Au/t)	Contained Ounces	Tonnes (1000)	Average Grade (g Au/t)	Contained Ounces
0.3	0.4	15,032	0.35	168,775	76,786	0.75	1,840,504
0.4	0.5	11,701	0.45	168,536	61,754	0.84	1,671,729
0.5	0.75	20,365	0.62	402,756	50,053	0.93	1,503,194
0.75	1.0	12,611	0.87	351,299	29,688	1.15	1100,438
1.0	2.0	16,234	1.32	686,837	17,077	1.36	749,139
2.0	3.0	819	2.27	59,7228	843	2.30	62,302
>3.0		24	3.34	2,580	24	3.34	2,580

Exploration Potential

The following discussion details by deposit some of the more important areas that have been identified by Gustavson 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 gold data and resulting creation of an independent gold grade model was the identification of areas within the existing defined deposit that are "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. This has resulted in a number of areas that contain no estimated resources, but in all likelihood, based on the geology and surrounding drill hole data, are mineralized and would contain resources if additional drilling were completed. In addition to theses areas, there are also extensions to the known mineralization that are likely to occur based on the limits of the current drilling.

Insufficient data exist within the Batman database to predict the copper-gold relationship. The Quigleys database has more copper data and has been used to "estimate" the relationship for Batman Deposit; however; the existing Batman core needs to be re-assayed for copper in order to gain an accurate understanding of the relationship and adequately predict the impact or lack thereof that copper will have on the project economics. It is for this reason that Gustavson has recommended that a phased assaying program be developed to prove the relationship and thereby allow modelling of this other metal.

Quigleys Deposit

The Quigleys Deposit is 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 an accurate resource estimate. Gustavson 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 modelling 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.

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.

Other Mineralized Occurrences

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

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

Mine Plan and Mineable Resources

Potentially mineable pit shapes were evaluated using a Lerchs-Grossman (LG) analyses performed with the Whittle 4 pit planning software and the Mt Todd GEMS® geologic model. The primary purpose of this was to determine ultimate pit limits and the best extraction sequence for open pit mine design. For this PA, measured, indicated, and inferred resources were considered potential ore. The parameters assumed for the LG analyses are summarized in Table 1-4 (all prices and costs are reported in fourth quarter 2006 US dollars).

Table 1-4 VISTA GOLD CORP. – MT TODD GOLD PROJECT Parameters for Lerchs-Grossman Analyses December 2006	
Average Pit Slopes	All 55 degrees
Gold Price	US$600 per oz Au
Gold Recovery	82 percent
Copper Price	US$2.00 per pound Cu
Copper Recovery	50 percent
Mining Cost	US$0.90 per tonne processed
Processing Cost	US$8.95 per tonne processed
General and Administrative Cost	US$0.25 per tonne processed
Environmental/Regulatory Cost	US$0.10 per tonne processed

The base case ultimate pit shell is defined by the economic factors listed in Table 1-4. A total of 15 LG runs, including the base case, were run to determine sensitivities to gold price and pit slopes. Gold price sensitivity was analyzed in $25.00 per ounce increments from $200 to $600 per ounce Au. The results of the $400, $500, and $600 per ounce Au cases were used to phase the pit in mine planning.

Using the base case, the ultimate pit was designed for medium-sized mining equipment, including 18-cubic meter hydraulic front shovels and 141-tonne haul trucks. The design includes smoothed pit walls, haulage ramps, benches, and pit access. After the ultimate pit was designed, two intermediate phases were created for production scheduling and enhancing the economics of the project. Without specific detailed geotechnical data, but utilizing reports from the previous operations it is believed that the geotechnical conditions are favorable for development of the proposed ultimate pit. Interramp slopes were assumed to average 55 degrees, with the bench heights and haul road widths designed to accommodate the midsize equipment fleet planned for Mt Todd.

Table 1-5 summarizes mineable resources resulting from the base case ultimate pit.

Table 1-5
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Classification of Mineable Resources ($600 per oz Au and $2.00 per lb Copper Designed Pit)
December 2006

Class	Ore Tonnes (x 1000)	Average Gold Grade (gm/t))	Contained Gold (oz)	Waste Tonnes (x 1000)	Total Tonnes (x 1000)	Stripping Ratio (W:O)
Measured	20,521.1	0.902	595,036	-NA-	-NA-	-NA-
Indicated	41,182.9	0.908	1,202,307	-NA-	-NA-	-NA-
Inferred	45,947.5	0.923	1,363,645	-NA-	-NA-	-NA-
TOTAL	**107,651.5**	**0.913**	**3,160,988**	**187,118**	**294,769**	**1.74**

The production schedule for this PA assumes a 30,000-tonne-per-day (10.65-million-tonne-per-year) ore production rate, resulting in a ten-year operating life, as shown in TABLE 1-6.

Table 1-6 VISTA GOLD CORP. – MT TODD GOLD PROJECT Production Schedule December 2006				
Year	**"Ore" Tonnes (x 1000)**	**Avg. Grade (g Au/tonne)**	**Waste Tonnes (x 1000)**	**Stripping Ratio (W:O)**
PP2	0		0	
PP1	0		0	
1	10,650	1.20	22,000	2.1
2	10,650	1.15	22,000	2.1
3	10,650	1.10	22,000	2.1
4	10,650	1.00	22,000	2.1
5	10,650	1.00	22,000	2.1
6	10,650	0.90	21,000	2.0
7	10,650	0.80	21,000	2.0
8	10,650	0.70	21,000	2.0
9	10,650	0.70	10,000	0.9
10	10,650	0.60	4,000	0.4
	106,500	**0.91**	**187,000**	**1.8**

Processing and Process Flowsheet

Run-of-mine (ROM) ore will be sent to the primary crusher (Gyratory Crusher) and conveyed to a stockpile. The ore will be reclaimed from the stockpile and subjected to secondary and tertiary crushing. The fine crushed ore will be ground to P_{80} of 104 microns in a ball milling circuit. The ground slurry would be sent to the rougher flotation circuit. The rougher concentrate would recover ± 7.5% of weight, ± 90% of gold and ± 75% of copper. The rougher tailings would have negligible amount of sulfides and would be environmentally friendly and sent to the existing or new tailings pond. The rougher concentrate would be reground to P_{80} of 37 microns and subjected to three stages of cleaner flotation to recover copper and gold. The final concentrate is projected to assay 24% Cu and contain ± 50% of the gold. The cleaner flotation tailings would be cyanide leached in the CIL circuit. The leach residue would be subjected to cyanide destruction and the residue will be thickened and filtered and trucked to a lined tailing area.

Existing Environmental Conditions

The Draft Environmental Impact Statement for the Mt Todd mine (Zapopan, 1992) gave the following as the specific environmental issues to be considered for the project: conservation of the Gouldian Finch in the Yinberrie Hills; control of acid drainage; heap leach solution containment; tailings containment; water management; rehabilitation planning; impacts of noise, dust and blasting; impacts on vegetation and fauna; impacts on Aboriginal sites of cultural significance; impacts on historical and Aboriginal archaeological sites; impacts on regional urban and social infrastructure; and general site management issues, such as weeds, mosquito-borne diseases, wildlife and workforce behavior.

The major environmental considerations for the Mt Todd site currently and going forward could be regarded as site water management and, potentially, the conservation of the Gouldian Finch.

The Gouldian Finch was classified as "Endangered" in 2001 by the NT Parks and Wildlife Commission (NT PWC, 2001). There are currently believed to be no specific conservation practices enforced at Mt Todd for the finch. The primary environmental challenge for Mt Todd is the area of water management. The site contains several sources of acidic water high in dissolved metals. These include Batman Pit, waste rock dump repository (RP1), tailings dam (RP7). HLP moat and low-grade ore dump pond (RP2). This water is managed through a combination of evaporation, pumping for containment, and controlled discharge to streams during major flow events.

A database has been constructed for the collation of Mt Todd hydro-chemical data, and potentially for other data types (e.g. groundwater levels and pumping rates) in the future. The "guidelines" referenced in the following discussion of Mt Todd waters chemistry are the ANZECC and ANZMARC (2000) guidelines for aquatic ecosystem protection (at the 95% species protection level) and for recreation.

In all the water retention ponds (excluding the raw water dam) and in Batman Pit, the median concentrations of all metals measured (except arsenic) exceed the guideline levels, usually by a considerable margin. Copper and zinc have the highest levels relative to guidelines, requiring dilution factors of approximately 9000 and 5000 respectively to meet the guidelines. This demonstrates why the compliance focus for Mt Todd is on copper concentrations. Metal levels in all the ponds and pits (except RWD) are generally within the same order of magnitude. The pH of the waters ranges from approximately 3 to 4.5. The waters are brackish, with EC ranging between 1700uS/cm and 5000uS/cm.

The ephemeral streams on site (Stow, Horseshoe and Batman) exhibit metal concentration records (particularly copper and zinc) indicative of periodic flushing of contaminants from site into the streams.

The impact of the Mt Todd site on the perennial Edith River is apparent in the monitoring results from sites along the river. Sulphate concentrations progress from very low upstream of site (less than 1 mg/L) to approximately 10 mg/L downstream of site during the wet season, with occasional excursions above 100mg/L. This seasonality is not observed upstream of site and likely represents flushing of mine waters to the river with wet season rainfall. There are similar indications for copper. A license criteria for the site is that the copper concentration at downstream site SW10 be no more than 10ug/L higher than at background site SW2. This criteria was breached several times in each of the previous three wet seasons. In the 2005/06 wet season it is understood that this was due in part to delays in installation of the water management infrastructure. The results also suggest significant intermittent contributions of zinc to the Edith River from the Mt Todd site, and lesser contributions of aluminum, cadmium and cobalt. The upstream water quality occasionally transgresses the aquatic guideline value for copper.

The hydro-chemical monitoring data displays no clear indication of seepage from the facilities. However, surface seeps are visible around the tailings dam, the heap leach pond and RP1. This suggests seepage to groundwater is either currently not detected or will occur in the future. Further work, including installation of new monitoring bores, is required to characterize the occurrence of seepage with more confidence.

Water Management

The major environmental challenge for Mt Todd lies in the area of water management. The site contains several ponds with acidic water high in dissolved metals which include Batman Pit (RP3), the waste rock dump repository (RP1), the tailings dam (RP7), the heap leach facility, and the low grade ore dump pond (RP2). This water is managed through a combination of evaporation, pumping for containment, and controlled discharge to streams during major flow events.

The license conditions for the site have been breached several times during each wet season while the site has been under care and maintenance. These breaches have taken the form of uncontrolled discharges of wastewater from several ponds, and occasional exceedances of the downstream copper concentration limit. They have occurred despite significant effort and resources applied to water management on the site by the NT government, demonstrating the water management challenges for Mt Todd.

The overflows were caused largely by lack of pumping capability from the heap leach facility and RP5, inadequate pumping capability from RP1 and RP2, and undersizing of the RP1 pond.

During 2006 the NT government installed pumps at the heap leach facility and RP5, which should greatly reduce the future overflows from these ponds. A new pumping system was installed at RP1 in 2006 as part of a strategy to pump excess water from RP1 to RP3, rather than to RP7 as previous. At the time of writing, this pumping system is reportedly operating at a rate of 450m^3/hr, which is lower than the 540m^3/hr design pumping rate.

A new water balance model for Mt Todd has been constructed using the GoldSim platform. The key findings of scenarios run with the model are:

- *The current water management strategy has a probable lifetime of two to four years (until RP3 fills). During this time the management strategy should decrease, but not eliminate, the occurrence of overflows and ARD releases from the site.*
- The water balance excess (defined as pumped water, excluding controlled discharges, plus overflowed water) for the site ranges from 1.5 to 2.1 million cubic meters per year;
- The breakdown in excess water contribution from the ponds is approximately: RP1 – 80%, RP2 – 11%, RP5 – 8%, heap leach facility – 1%;
- the controlled discharge to Edith River from RP1 is a relatively small proportion of the balance, being around 60,000 to 100,000 m^3/year, or 5% of the water balance excess;
- Catchment inflow to RP7 and RP3 is potentially significant. Diversion of catchment flow around RP7 could make the tailings dam a net sink for approximately 1 million m^3/year. However, uncertainty in the catchment flow parameters needs to be resolved;
- the Raw Water Dam overflows an average volume of approximately 8,700,000 m^3/year

of good quality water. This represents a potential dilutant source;

- a water treatment plant designed to treat the excess water from site (without mitigation measures) should have a peak design rate of 10,000 m^3/d and an average throughflow of 6,800 m^3/day; and
- it appears that the volumes of uncontrolled discharges from RP1 may have been significantly underestimated by water balance modelling in previous years. Therefore, there is a strong possibility that the reported overflow volumes for the 2006/07 wet season will increase from previous years despite similar management strategies.

The major uncertainties in the model relate to:

- water levels in RP3, RP2, RP5 and the heap leach pond, none of which are currently recorded; and
- catchment runoff contributions (particularly for RP3 and RP7).

The plan codifies the current water management practices by the NT government.

Beyond the 2006/2007 wet season, it is likely that some form of treat and release scheme will be required within two to four years. Broadly speaking, the site water management strategy should take the form of adopting mitigation measures to extend the current practice of storing water in RP3 with the intent of ultimately introducing a 'treat and release' scheme. Earlier introduction of a treatment scheme will have time and cost benefits for the removal of pit water in advance of mining.

Routing of excess water through evaporation cannons discharging into RP3 could be a valuable method to 'buy time' for the RP3 storage strategy and reduce the time needed to empty the pit in advance of mining. The evaporation cannons could potentially be moved to RP7 once the site moves into operation. Other mitigation measures which should be considered are the diversion of catchment flow from RP3 and RP7, installation of telemetered monitoring instrumentation at monitoring locations SW2 and SW4, and controlled release of water from the RWD to dilute treated water release.

Reclamation and Closure

Vista commissioned MWH to prepare the conceptual closure plan ("CCP") to support a preliminary feasibility study of the restart of mining operations. This CCP 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 separate reports prepared by MWH on the environmental status and water management at the site.

The major environmental challenge for Mt Todd currently lies in the area of water management. The site contains several sources of acidic water high in dissolved metals, including Batman Pit, the waste rock dump repository (RP1), the tailings dam (RP7), the heap leach facility and the low-grade ore stockpile pond (RP2). This water is managed through a combination of evaporation, pumping for containment, and controlled discharge to streams during major flow events. Similar conditions will need to be avoided for closure of a new mining operation at the site.

There are five primary facilities that currently exist that will be carried forward as part of the new mine plan, as well as ancillary facilities and disturbed ground. These are included in this CCP, as listed below:

- Batman Pit and pit lake (RP3);
- Waste Rock Dump (WRD), waste rock dump pond, and runoff containment pond (RP1);
- The existing Tailings Storage Facility (TSF), and tailings pond (RP7);
- Plant Area (not including stockpiles);
- Miscellaneous facilities (e.g., pipelines); and
- Disturbed ground (e.g., stockpile footprints).

In addition to the above-listed mine features, it is anticipated that a small lined tailings facility and a second unlined large tailings storage facility will be constructed during operation to contain sulfide-bearing and benign tailings, respectively. The mine pit and WRD will be significantly enlarged.

The closure costs were estimated based on the proposed design (areas and volumes) of each of the closure facilities and MWH's experience with similar projects. Using MWH's experience on similar projects, including current reclamation programs, unit rates were developed for each element of the closure strategy, which were then applied to the area or volume of each feature. The majority of the unit rates is per unit volume or area and have been applied to conditions where mine labor is used to conduct the reclamation. Based on this, the conceptual estimated costs for implementing this CCP are US**$30,500,000** including ten years of post-closure care and maintenance but before contingency, as summarized in TABLE 1-7.

Option 2, which includes a more robust cover on the STF increases the capital closure cost by approximately $4,100,000. The total cost difference including the engineering and construction management components is approximately $4,800,000. Post-closure care and maintenance between the two options is not considered to be significantly different.

TABLE 1-7 summarizes the MWH estimated closure costs for the Mt Todd site.

TABLE 1-7 VISTA GOLD CORP. – MT TODD GOLD PROJECT MWH Conceptual Closure Cost Estimate Summary December 2006	
Area	**Cost (US$)**
Batman Pit	$200,000
Waste Rock Dump	$9,200,000
Tailings Storage Facility - Existing	$4,200,000
Tailings Storage Facility - New	$3,500,000
Sulfide Tailings Facility Lined – New (option 1)	$1,300,000
Plant Area	$500,000
Disturbed Ground	$600,000
Water Management	$300,000
Subtotal	**$19,800,000**
Engineering & Construction Management	$3,200,000
Total Capital Cost for Closure	**$23,000,000**
Operations & Maintenance	$7,500,000
Total Cost	**$30,500,000**
Annual O&M costs until full closure accepted	$300,000

Notes: (1) Cost rounded to nearest $100,000 in current US$.
(2) Lower cost option 1 components included.
(3) Assumes that closure of the HLP estimated to cost $6,900,000 will be completed by the NT prior to project development.

It was necessary to make various assumptions in developing the CCP. Some of the key assumptions, which must be better understood as the closure process proceeds include the following:

- The heap leach pad will not be used in any way by the restart of mining operations and will be reclaimed by the NT at some date prior to commencement of mining operations;
- Sufficient water resources will be available to flood Batman Pit in a reasonable time period (e.g., 6 years or less);
- The Batman Pit lake limnology and watershed hydrology will allow for the establishment of a long-term stable closure condition without long-term water treatment;
- The inert waste rock that will be placed under the cover for the waste rock dump will be suitable to support the soil cover as plant growth media both chemically and in terms of water holding capacity (i.e. it will provide enough water storage to effectively eliminate infiltration);
- Sufficient inert waste rock will be available to allow for TSF embankment construction and for encapsulation of potentially acid generating waste rock;
- The heap leach pile will not have to be rinsed or otherwise treated prior to closure;
- In one scenario, the stabilized sulfide tailings will not interfere with the establishment of vegetation in a 1 m soil cover section, will be demonstrated to be chemically stable long-

term, and will be of sufficiently low permeability to act as a low-permeability layer.

- The "rougher tailings" that will be placed over the tailings disposal facility will be suitable be suitable as a plant growth media both in terms of water holding capacity and chemically;
- Burial by benign rougher tailings will be sufficient for limiting any future ARD production from the existing tailings;
- The proposed water treatment plant that will be part of the proposed mining facility will be available for closure and early post-closure water treatment; and
- Potential impacts to groundwater are assumed to be minimal and therefore no closure activities associated with groundwater are included in this CCP.

While the HLP will be closed by the NT prior to mine construction, another important assumption is that the HLP material will not have to be rinsed or otherwise treated prior to closure.

Several studies to gather information to confirm these assumptions and to provide the other necessary input parameters to model and finalize the design for the various mine facilities will be required prior to construction and closure.

Capital and Operating Costs

The estimated capital expenditures for the life of the mine are summarized on Table 1-8. Startup Capital is estimated to total about $264 million.

Operating costs for the cash flow are summarized below:

Mining	$1.21 per tonne material mined
Milling	$6.48 per tonne ore processed
G&A	$0.14 per tonne material mined

Cash Flow Estimates

A cash flow analysis and sensitivity studies were completed for the base case, which includes mining the measured, indicated and inferred resources and assumes a gold price of $600 per ounce. Gold recovery is assumed at 87 percent. TABLE 1-9 summarizes results from the pre-tax, 100 percent equity, constant 2006 US dollar, cash flow analysis. TABLE 1-10 summarizes the sensitivity of the Net Present Value of the cash flows in TABLE 1-9 at discount factors of 0, 5, 10, 15, 20, and 25 percent. Sensitivities for gold prices of $500, $600, and $700 per ounce and plus and minus 20 percent for operating and capital costs are presented. The base case has an approximate Discounted Cash Flow Rate of Return (DCFROR) of 17 percent. The breakeven gold price is a nominal $532 and $568 per ounce gold for DCFROR rates of 0 and 10 percent, respectively.

TABLE 1-8 VISTA GOLD CORP. - MT. TODD GOLD PROJECT Capital Cost Summary (US $000) December 2006										
Year	**Access & Site Prep**	**Plant & Facilities**	**Infrastructure**	**General Surface Mobile Equip.**	**Concentrator Tailings Disposal**	**Mine Development**	**Mine Equipment**	**Permitting/ Mine Closure**	**G&A, OH, Contingency**	**TOTAL**
PP2	300	950	1,150	1,300	79,800	5,550	713	500	10,126	100,389
PP1	150	1,000	500	486	106,400	2,400	24,332		14,627	149,895
1				1,510	1,445		9,439		1,239	13,633
2					1,658		108		177	1,943
3					7,904		216		812	8,932
4					6296		8,856	2,323	1,748	19,223
5					5,994		9,385	2,556	1,794	19,729
6					1,955		7,668		962	10,585
7					3,460		15,455		1,891	20,806
8					1,277		1,037		231	2,545
9							108		11	119
10							108		11	119
11								581	58	639
12								6,214	621	6,835
13								6,215	622	6,837
14								5,111	511	5,622
15								7,500	750	8,250
TOTAL	**450**	**1,950**	**1,650**	**3,296**	**216,189**	**7,950**	**77,425**	**31,000**	**36,191**	**376,101**

TABLE 1-9
VISTA GOLD CORP. - MT. TODD GOLD PROJECT
Base Case Before Tax Cash Flow Summary
December 2006

Case:		Copper Price ($/lb)	$2.00	Gold Price ($/oz)		$600															
30,000 TPD																					
M+I+I; 55 degree slopes				**PP 2**	**PP 1**	**Yr 1**	**Yr 2**	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yr 8**	**Yr 9**	**Yr 10**	**Yr 11**	**Yr 12**	**Yr 13**	**Yr 14**	**Yr 15**	**Totals**
MINE PRODUCTION SCHEDULE (000s tons)																					
	ORE																				
		TOTAL		0	0	10650	10650	10650	10650	10650	10650	10650	10650	10650	10650	0	0	0	0	0	106500
		% Cu		0.00	0.00	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03						
		oz Au/ton		0.000	0.000	0.039	0.037	0.035	0.032	0.032	0.029	0.026	0.023	0.023	0.019						
	WASTE			0	0	22000	22000	22000	22000	22000	21000	21000	21000	10000	4000	0	0	0	0	0	187000
		TOTAL		**0**	**0**	**32,650**	**32,650**	**32,650**	**32,650**	**32,650**	**31,650**	**31,650**	**31,650**	**20,650**	**14,650**	**0**	**0**	**0**	**0**	**0**	**187,000**
		Stripping Ratio				2.1	2.1	2.1	2.1	2.1	2.0	2.0	2.0	0.9	0.4						1.8
CONCENTRATOR SCHEDULE																					
	Mill Feed	tons (000s)		0	0	10650	10650	10650	10650	10650	10650	10650	10650	10650	10650	0	0	0	0	0	106500
	Recovery	Cu %	**70**	70	70	70	70	70	70	70	70	70	70	70	70						
		Au %	**87**	87	87	87	87	87	87	87	87	87	87	87	87						
	Cu Concentrate Grade	% Cu	**24**	24	24	24	24	24	24	24	24	24	24	24	24						
	Cu Concentrate Produced	tons (000s)		0.00	0.00	9.32	9.32	9.32	9.32	9.32	9.32	9.32	9.32	9.32	9.32						
		conc ratio		0	0	800	800	800	800	800	800	800	800	800	800						
PAYABLES SCHEDULE																					
	Copper S&R Rec = .96	lb (000s)		0	0	4295	4295	4295	4295	4295	4295	4295	4295	4295	4295						42,949
	Gold S&R Rec = .985	oz		0	0	348588	334063	319539	290490	290490	261441	232392	203343	203343	174294						2,657,983
REVENUE ($000)																					
	Copper	$/lb FOB Refinery	**2.00**	0	0	8,590	8,590	8,590	8,590	8,590	8,590	8,590	8,590	8,590	8,590						85,897
	Gold	$/oz FOB Refinery	**$600**	0	0	209,153	200,438	191,723	174,294	174,294	156,865	139,435	122,006	122,006	104,576						1,594,790
		TOTAL		**0**	**0**	**217,742**	**209,028**	**200,313**	**182,884**	**182,884**	**165,454**	**148,025**	**130,595**	**130,595**	**113,166**	**0**	**0**	**0**	**0**	**0**	**1,680,687**
MINE OPERATING COSTS ($000s)																					
	Mining																				
	Open Pit	$/ton mined	**1.21**	0	0	39507	39507	39507	39507	39507	38297	38297	38297	24987	17727						355135
	Milling	$/ ton milled	**6.48**	0	0	69012	69012	69012	69012	69012	69012	69012	69012	69012	69012						690120
	G&A	$/ton mined	**0.14**	0	0	4571	4571	4571	4571	4571	4431	4431	4431	2891	2051						41090
		TOTAL	**1.0**	**0**	**0**	**113,090**	**113,090**	**113,090**	**113,090**	**113,090**	**111,740**	**111,740**	**111,740**	**96,890**	**88,790**	**0**	**0**	**0**	**0**	**0**	**1,086,345**
FREIGHT, SMELTING & REFINING COST ($000s)																					
	Concentrate Freight	$/ton con shipped	**98.00**	0	0	913	913	913	913	913	913	913	913	913	913						9132
	Treatment Charge for Cu Conc	$/ton Cu con treated	**100.00**	0	0	932	932	932	932	932	932	932	932	932	932						9,319
	Refinery Charge	$/lb Cu recovered	**0.10**	0	0	429	429	429	429	429	429	429	429	429	429						4,295
		TOTAL		**0.0**	**0.0**	**2,274.6**	**2,274.6**	**2,274.6**	**2,274.6**	**2,274.6**	**2,274.6**	**2,274.6**	**2,274.6**	**2,274.6**	**2,274.6**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**22,746.0**
		$/lb Cu recovered				0.53	0.53	0.53	0.53	0.53	0.53	0.53	0.53	0.53	0.53						
ROYALTY		Deanhurst GSR%	**0.00**	0	0	0	0	0	0	0	0	0	0	0	0						**0**
		JAAC NSR%	**1.00**	0	0	2177	2090	2003	1829	1829	1655	1480	1306	1306	1132						**16,807**
		TOTAL																			
NET OPERATING REVENUE ($000s)				**0**	**0**	**100,201**	**91,573**	**82,946**	**65,691**	**65,691**	**49,786**	**32,531**	**15,275**	**30,125**	**20,970**	**0**	**0**	**0**	**0**	**0**	**554,789**
CAPITAL COST SUMMARY ($000s)																					
	Access and Site Prep			300	150	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**450**
	Surface Plant and Facilities			950	1000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**1,950**
	Site Infrastructure			1150	500	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**1,650**
	General Surface Mobile Equipment			1300	486	1510	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**3,296**
	Concentrator and Tailings Disposal			79800	106400	1445	1658	7904	6296	5994	1955	3460	1277	0	0	0	0	0	0	0	**216,189**
	Open Pit Mine Development			5550	2400	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**7,950**
	Open Pit Mine Equipment			712.8	24332	9439	108	216	8856	9385	7668	15455	1037	108	108	0	0	0	0	0	**77425**
	Permitting, Reclamation, and Closure			500	0	0	0	0	2323	2556	0	0	0	0	0	581	6214	6215	5111	7500	**31,000**
	G&A, OH, Contingency			10126	14627	1239	177	812	1748	1794	962	1891	231	11	11	58	621	622	511	750	**36,191**
		TOTAL	**1.0**	**100,389**	**149,895**	**13,634**	**1,943**	**8,932**	**19,223**	**19,729**	**10,585**	**20,806**	**2,545**	**119**	**119**	**639**	**6,835**	**6,837**	**5,622**	**8,250**	**376,101**
NET PRETAX CASH FLOW ($000s)				**-100,389**	**-149,895**	**86,567**	**89,631**	**74,014**	**46,468**	**45,962**	**39,200**	**11,724**	**12,730**	**30,007**	**20,852**	**-639**	**-6,835**	**-6,837**	**-5,622**	**-8,250**	**178,688**

Note: **Startup Capital (Yr PP2 +PP1+1)** **263,918**
No Working Capital, 100% Equity, Constant 2006 $US, Before Tax
Year 15 Closure Costs ($7.5 million) to pay for 10 years of Operations and Maintenance

TABLE 1-10
VISTA GOLD CORP. - MT. TODD GOLD PROJECT
Cash Flow Net Present Value Sensitivity Analysis
December 2006

Gold Price Sensitivity (NPV $000s)

Discount %	Base($600)	$500.00	$700.00
0	$178,688	($84,453)	$441,828
5	$100,088	($89,702)	$289,879
10	$45,739	($95,558)	$187,036
15	$8,059	($100,043)	$116,161
20	($18,186)	($102,851)	$66,479
25	($36,528)	($104,187)	$31,130

Operating Cost Sensitivity (NPV $000s)

Discount %	Base($600)	Op Cost-20%	Op Cost+20%
0	178,688	395,957	(38,581)
5	100,088	253,378	(53,201)
10	45,739	157,567	(66,090)
15	8,059	92,055	(75,938)
20	(18,186)	46,525	(82,897)
25	(36,528)	14,433	(87,490)

Capital Sensitivity (NPV $000s)

Discount %	Base($600)	CAPEX-20%	CAPEX+20%
0	178,688	253,908	103,467
5	100,088	163,256	36,921
10	45,739	100,646	(9,168)
15	8,059	56,901	(40,783)
20	(18,186)	25,966	(62,337)
25	(36,528)	3,852	(76,909)

2.0 INTRODUCTION

Gustavson Associates, LLC ("Gustavson") was commissioned by Vista Gold Corp. ("Vista") in May 2006 to prepare a Canadian National Instrument 43-101 (NI43-101) compliant Preliminary Economic Assessment Report on the Mt Todd Gold Project in the 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.

2.1 Terms of Reference

This report has been prepared in accordance with the guidelines provided in National Instrument 43-101 ("NI43-101"), Standards of Disclosure for Mineral Projects, dated December 23, 2005. The Qualified Person responsible for this report is Mr. John W. Rozelle P.G., Principal Geologist at Gustavson.

2.2 Scope of Work

The Mt Todd Mine was operated from 1993 to 1997 by Pegasus Gold Australia Pty Ltd. ("Pegasus") who reported investing more than A$200 million to develop the mine, and from 1999 to 2000 by a joint venture between Multiplex Resources Pty Ltd, General Gold Resources Ltd., and Pegasus. Low gold prices contributed to the mine closing both times. No resource estimates meeting NI43-101 standards have been completed until those that are the subject of this report.

The Mt Todd Mine property is made up of several gold deposits occurring in an area of some 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 Gustavson involved an in-depth review of the available documentation of the exploration, geology, mineral resource/reserve estimates and mining at Mt Todd, compiled by the Trustee and by previous operators. Based on these data, Gustavson estimated the mineral resources of the Batman Deposit.

2.3 Effective Date

The effective date of the mineral resource and mineral reserve statements in this report is June 26, 2006.

2.4 Units

All units unless other wise specified are metric. For the purpose of this report the exchange rates are CDN$1.00 = US$0.903 and A$1.00 = US$0.767.

2.5 Qualifications of Consultant

John W. Rozelle of Gustavson visited the Mt Todd property in June, 2005. During his visit Mr. Rozelle examined the Mt Todd mine site, core storage facility at the mine site and the data repository in Darwin. This report has been prepared based on a technical review and preparation of resource estimates by consultants sourced from Gustavson's Boulder, Colorado, office. These consultants are specialists in the fields of geology, mineral resource and mineral reserve estimation and classification, mining and mineral economics.

Neither Gustavson 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. Gustavson will be paid a fee for this work in accordance with normal professional consulting practice.

The individuals who have provided input to this technical report, who are listed below, have extensive experience in the mining industry and are members in good standing of appropriate professional institutions.

The key project personnel contributing to this report are listed in TABLE 2-1.

TABLE 2-1
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Key Project Personnel
December 2006

Company	Name	Title
Vista Gold Corp.	Michael Richings	President & CEO
	Fred Earnest	Senior Vice President
	Howard Harlan	Vice President
	Robert Perry	Vice President
Gustavson Associates, LLC	John Rozelle	Principal Geologist
	William Crowl	Vice President
	Stephen Krajewski	Senior Geologist
	Rex Bryan	Principal Geostatistician
	Leroy Aga	Sr. Mine Planning Technician
Resource Development Inc.	Deepak Malhotra	President, Metallurgist
	Ray Hyyppa	Consulting Metallurgist
MWH Consultants	Jed Youngs	Senior Hydrogeologist
	Cary L. Foulk, P.G.	Supervising Geologist/Geochemist
	Tatyana G. Alexivea, CPEng	Principal Engineer

2.6 Basis of Report

Gustavson 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.

Gustavson has not independently conducted any title or other searches, but has relied upon Vista for information on the status of the claims, property title, agreements, and other pertinent conditions. Gustavson has not independently conducted any mining, processing, or economic studies, or permitting and environmental studies.

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. Gustavson 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 the this report include:

Dr. Stephen A. Krajewski

> Graduated with Geography (B.S.-1964), Geology (M.S.-1971) and Earth Science (Ed.D.-1977) degrees from The Pennsylvania State University
>
> Is a Member of the American Institute of Professional Geologists, Member Number 4739, member of the Society for Mining, Metallurgy and Exploration, Inc. (SME); member of the American Association of Petroleum Geologists; and a member of the Rocky Mountain Association of Geologists.
>
> Has worked with computers to map and model mineral deposits since 1983. His geologic career has included forty-two years of domestic and international experience in the employ of Major and Junior Mining Industry Companies, Major and Minor Oil & Gas Companies, environmental consulting companies, a state geological survey, and universities.

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).
>
> Has worked as a geostatistical reserve analyst and mineral industry consultant for a total of twenty-six 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 in mine valuation, ore reserve estimation and environmental compliance.

William J. Crowl, P.G.

Graduated from University of Southern California in 1968 with a B.A. in Earth Science. Graduated from University of Arizona in 1979 with a M Sc in Economic geology.

Is a registered Professional Geologist in the State of Oregon (G573) and is a member of the Australasian Institute of Mining and Metallurgy (AUSIMM).

Has worked as a geologist, geostatistical reserve analyst and mineral industry consultant for a total of thirty-eight years since graduating from the University of California. During his professional career, he has worked for operating companies, engineering, procurement, and construction companies, and consultancies.

Dr. Deepak Malhotra

Graduated with a Mineral Economics Ph.D. from Colorado School of Mines, Golden, Colorado, in 1979. Graduated in 1974 from Colorado School of Mines in Golden, Colorado, with M.S. in Metallurgical Engineering. Graduated from Indian Institute of Technology, Kanpur, India, in 1970 with a B.S. in Metallurgical Engineering.

Is a member of the Society of Mining, Metallurgy and Exploration, Inc. (SME) and the Canadian Institute of Mining, Metallurgy and Petroleum (CIM).

Has worked as a metallurgist for seventeen years for a Major Mining Industry Company and since 1990 has worked as a consultant to Major and Junior Mining Industry Companies, International Finance Corporation (IFC), and United Nations. His area of expertise is in mineral processing, mineral economics, due diligences and plant audits.

Tatyana G. Alexieva, CPEng

Graduated with an M.S. in Geotechnical Engineering from the University of Architecture and Civil Engineering, Sofia, Bulgaria in 1988.

Is a registered Chartered Professional Engineer (CPEng) with the Institution of Engineers, Australia.

Has worked as a geotechnical engineer for eighteen years. For over fifteen years worked for international consulting companies providing services to the mining industry. Her area of expertise is analyses and design associated with mining facilities (tailing storage facilities, waste dumps, heap leach facilities, etc.). She has also been involved in numerous risk assessments, audits and due diligence mining projects.

Jed Youngs

Graduated from University of Western Australia in 1994 with a BEnv in Environmental

Engineering. Graduated from University of Technology Sydney in 2004 with an ME in Groundwater Management.

Is a member of the International Association of Hydrogeologists (IAH).

Has worked as a hydrogeologist and environmental engineer for ten years, primarily as consultant to the mining industry in northern Australia, Chile and South-east Asia. His main area of expertise is in groundwater studies (supply, dewatering and contamination investigation) and minesite water management.

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 50 kilometers northwest of Katherine, and approximately 250 kilometers 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

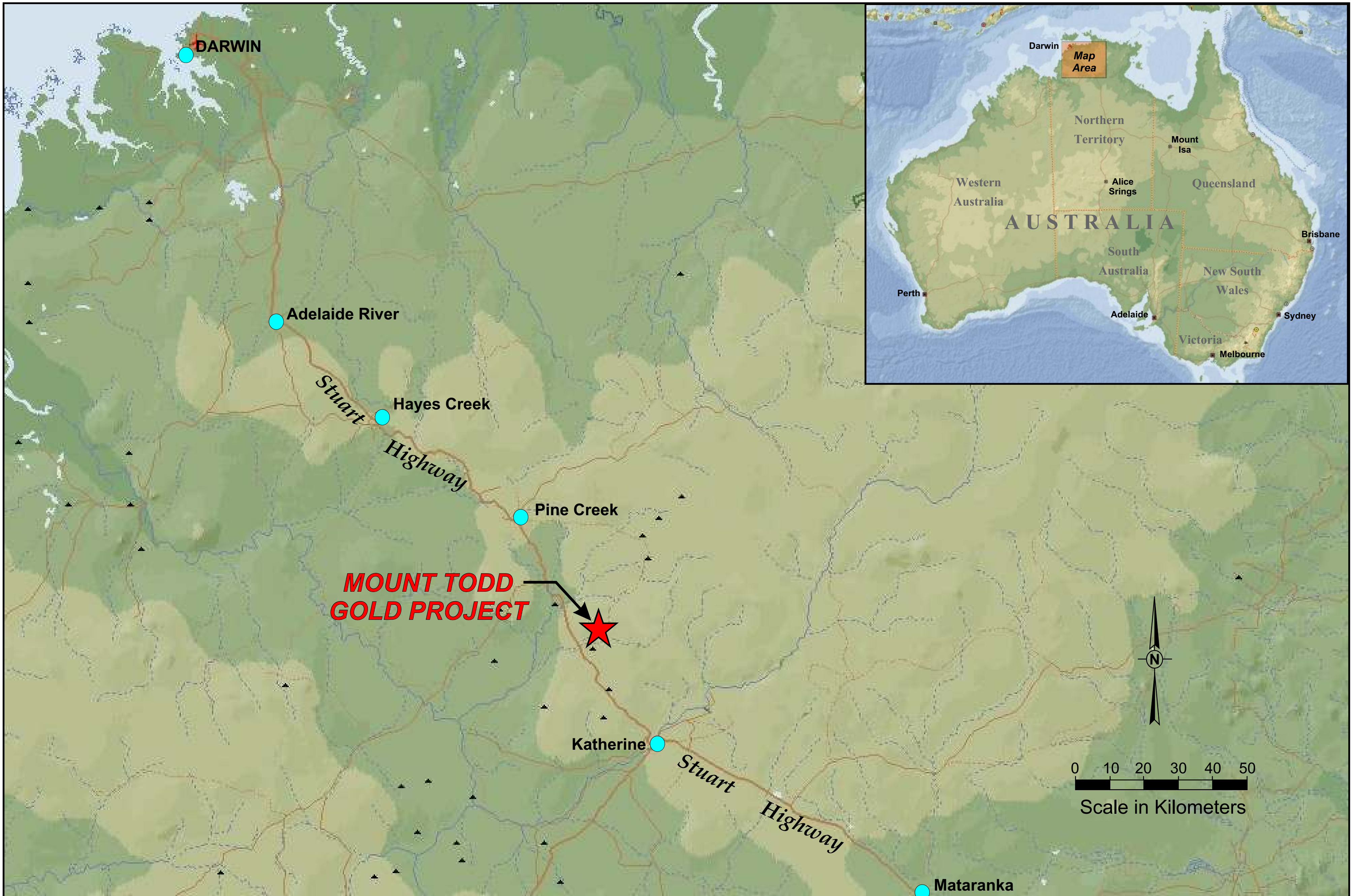
The concession consists of three individual tenements, MLN1070, MLN1071, and MLN1127 comprising some 5,365.27 hectares. FIGURE 4-2 illustrates the general location of the tenements and the relative position of the two primary mineral deposits: Batman and Quigleys.

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, 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 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.

Vista will pay 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 assume 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) calls 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



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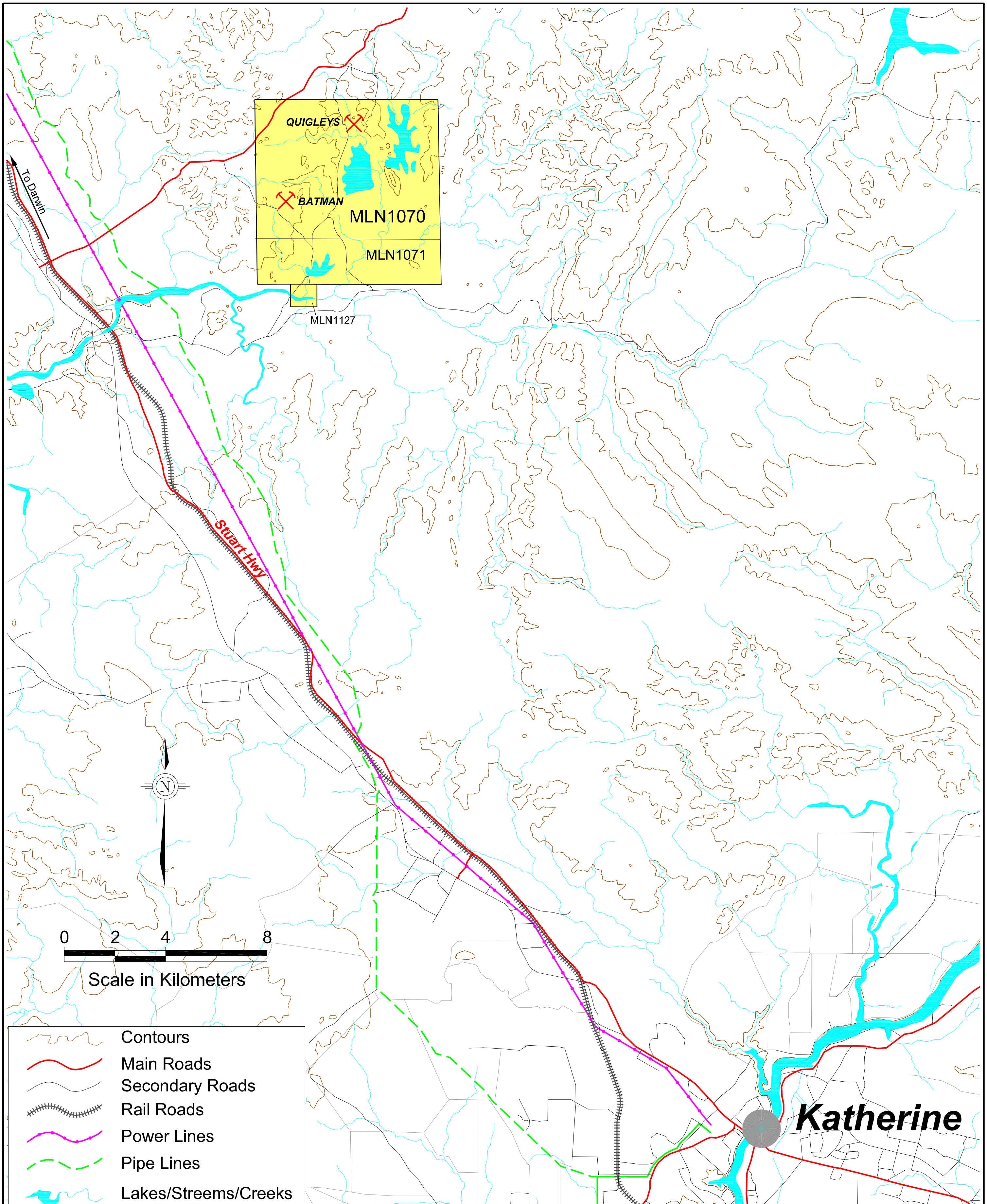
Date of Issue. 08/03/2006	Client Name. **VISTA GOLD CORP.**	FIGURE 4-1
Project No. VIS005	Project Name. Mt. Todd Gold Project	General Location Map
Drawing File Name. Fig4-1.cdr		Mount Todd Gold Project



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Date of Issue.	Client Name.	
08/03/2006	VISTA GOLD CORP.	FIGURE 4-2
Project No.	Project Name.	CONCESSIONS AND
VIS005	MT. TODD GOLD PROJECT	INFRASTRUCTURE MAP
Drawing File Name.		
Fig1-2_4-2.dwg		

during the period from the effective date until a decision is reached to begin production. Vista will also pay the JAAC A$5,000 per month in return for consulting with respect to Aboriginal, cultural and heritage issues. The JAAC will provide Vista with an office in Katherine (a regional center of population 11,000 approximately 50 kilometers from the mine site) and with secretarial services for a minimum of A$2,000 per month.

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 ACCESSIBILITTY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Mt Todd Project is located 50 kilometers northwest of Katherine, and approximately 250 kilometers 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. Some of these are:

> **The Katherine Gorge** is one of Katherine's main attractions. Katherine Gorge isn't just one spectacular gorge, but consists of 13 gorges separated by rapids. There are some fantastic walking trails and swimming areas around the Katherine Gorge, although some can be dangerous during the wet season. You can explore the Katherine Gorge and the surrounding areas by bushwalking, canoeing, scenic flights, boat cruises, and much more.
>
> **The Katherine Hot Springs** is located close to the Katherine River bed and pumps warm water from deep within the earth into various pools. The Hot Springs has a crystal clear appearance, making it the perfect place to swim in. There are also many shady spots and walking trails around Katherine Hot Springs.
>
> **The Katherine Low Level Nature Reserve** is an ideal swimming location. The Reserve

is also great for picnics and fishing. Visitors can follow the pathways around the Katherine Low Level Nature Reserve and look for birds and other kinds of wildlife.

Edith Falls is located 60 km north of Katherine. The falls consist of a series of cascading waterfalls and beautiful rock pools. There are also a number of good swimming areas at Edith Falls.

Cutta Cutta Caves Nature Park covers 1,499 hectares of limestone and caves close to Katherine. As the NT's only accessible tropical limestone cave, one can take a guided tour through the cavernous park. Cutta Cutta Caves Nature Park also has a large variety of wildlife including the deadly brown tree snake and rare orange horseshoe bats.

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

The following environmental section has been prepared by MWH Consultants (MWH) of Perth, Australia. MWH has had significant experience with mining projects both internationally and in Australia.

5.4.1 Existing Environmental Conditions

TABLE 5-1 below outlines the review of the most relevant environmental documents completed to date for Vista Gold. A comprehensive directory of reports exists for Mt Todd. The process of cataloguing and reviewing these documents is ongoing.

TABLE 5-1
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Summary of Current Site Environmental Conditions
December 2006

Document	Key Information
Terms of Reference for Development of Rehabilitation Plan for the Mt Todd (Legacy) Gold Mine, NT, *Unger, 2005, consultant report for DBIRD*	- Useful site overview - Basis for development of rehab plan, high-level recommended works program (water balance model, groundwater model, contaminant source/salt balance study) - Some rehabilitation and mitigation (eg automated monitoring) ideas - Digital terrain model

TABLE 5-1 VISTA GOLD CORP. – MT TODD GOLD PROJECT Summary of Current Site Environmental Conditions December 2006	
Waste Discharge License Reports 2003/2004, *DBIRD, 2004* Waste Discharge License Reports 2004/2005, *DBIRD, 2005*	- Reports on compliance with the site waste discharge license during the reporting period - Presents data for rainfall and evaporation, water levels in retention ponds, water quality in retention ponds, controlled releases, uncontrolled releases, wastewater and copper loadings to receiving waters, assessment of impact on the Edith River, macro invertebrate sampling, assessment of the management strategy, water balance (revised model to improve handling of RP1 sub-catchment) - Most of the water volumes (and calculated loadings) are estimated rather than measured - Vista has the spreadsheets used to develop the WDL reports - Issues identified: RP1 undersized for catchment, active water management is required to avoid uncontrolled discharges
Mount Todd Gold Mine Site Water Management Plan for 2001/2002 Wet Season, *Guan-hua Gao, 2001, Mines Division, Department of Mines and Energy*	- Adapted from General Gold 1999 / 2000 WMP - Uses spreadsheet-based water balance model - References groundwater investigations by Rockwater in 1988 and 1989 (reports on these investigations have been requested but not yet found), and by AGC Woodward-Clyde in 1992 (as part of EIS) - Description of mine water management system components, site hydrology - water balance calculations for 3 scenarios, essentially A) no transfer to storage B) transfer to tails dam for storage C) transfer to pit for storage - Scenario C was most effective in avoiding uncontrolled discharges but most costly, Scenario B was recommended as more economical - Water Management System: objectives and general strategies of water management, summary of water management strategies, operation of water management system, pre-wet season preparation, water monitoring program, contingency measures - Notes that RP1 and RP7 have insufficient storage for 1 in 10 years rainfall events without active management (pumping transfers)

TABLE 5-1 VISTA GOLD CORP. – MT TODD GOLD PROJECT Summary of Current Site Environmental Conditions December 2006	
Yimuyn Manjerr Mine Site: Review of Environmental Requirements Upon Suspension of Operations, *PPK, 2000, consultant report for General Gold Operations*	- Review of requirements for environmental management on suspension of operations, commissioned by General Gold through the receiver Ferrier Hodgson (July 2000) - Raises that water management is the major challenge, and also recommends focus on meeting statutory and reporting requirements, consultation and allocation of appropriate team - Very little monitoring of the acid-forming potential of waste rock and ore during operation, despite EIS (1992) flagging that 25% of mined rock was potentially acid-forming - General Gold attempted to model ARD potential of waste rock dump in 2000 but was severely restricted by lack of geochemical data "and was largely reliant on the sulphur content of material, even gold grades, for the early historical data" - Characterization of acid-forming potential of materials generally poor - 1999 / 2000 WMP (EWL Sciences, 1999) involved pumpage of poor quality water to tails dam. Uncontrolled discharge still resulted - tails water discharged through sprinklers adjacent to tails dam and on heap leach - other potential issues: Gouldian Finch (1992 EIS states at risk due to CN waters, risk now reduced due to low levels of WAD CN in the tailings dam), dust (less of an issue since use of quaternary and tertiary crushers was abandoned by General Gold), areas of exploration disturbance eg Quigleys (including uncapped drill holes), local areas of hydrocarbon contamination - possibility of pumping water to Batman Pit raised - tailings wall failure raised as worst possible scenario for site. Not considered as likely but geotechnical assessment recommended - "more generous release conditions" recommended for the waste discharge license
Environmental Management Plan September 2000, *Yimuyn Manjerr Investments) Pty Ltd, 2000*	- Identifies water management, waste rock management and Gouldian Finch management as the primary environmental issues of the site - Mine dewatering: pit water collects in sump and is pumped to RP2. No volumes noted, but small size of RP2 suggests very low inflow - Good figure of surface water and groundwater monitoring sites - Description of waste rock dump management - Conceptual plan for decommissioning and rehabilitation of the site - Environmental document list - Water body stage/volume data - Several procedural documents for water management (monitoring procedures, storm water mgmt, mgmt of retention pond water levels, RP1 mgmt procedures etc)
Mining and Environmental Management Plan, Annual Report August 2000, *Yimuyn Manjerr (Investments) Pty Ltd, 2000*	- EMP produced covering final year of mining - Includes a report on waste rock test work (May 2000). Found no correlation between sulphur and NAPP, suggesting the existing waste rock classifications (and stacking procedures?) may be flawed

TABLE 5-1 VISTA GOLD CORP. – MT TODD GOLD PROJECT Summary of Current Site Environmental Conditions December 2006	
Conceptual Rehabilitation Strategy for Yimuyn Manjerr, Draft, *EWL Sciences, 2000, consultant report for Yimuyn Manjerr Investments*	- summaries closure (and relevant operational) commitments made in the draft EIS (NSR, 1992) and later EMPs - 1996 EMP by Pegasus noted some key differences in operations at that time from the 1993 EMP, particularly construction of RP1, increase in size of waste rock dump and increase in size of the tailings dam - rehab strategy presented is very broad. Worth reviewing for possible input to conceptual strategy for Vista
Environmental Audit – Mt Todd Gold Mine, *PPK, 1999, consultant report for Pegasus Gold Australia Pty Ltd*	- audit conducted for Pegasus in March 1999 to assess significant environmental issues on assuming control of the site. Focused on identifying changes to environmental liabilities and mine closure costs during the period of Administration - list of major recommendations: five recommendations for water management plus a call for development of an EMS to demonstrate due diligence - "The Mt Todd facilities were generally found to be well managed. However, there are significant environmental issues faced by the site mainly associated with the management and control of acid mine drainage and surface waters"
Mt Todd Gold Mine Environmental Report 1993, *Zapopan N.L., 1993*	- baseline monitoring activities began in 1998 - high arsenic concentrations exceeding 50 ug/L were recorded in one bore (not specified) - Acid Mine Drainage investigations reported. - Heap leach: "Long term leaching experiments show that weathered heap leach ore has virtually no potential for generation of acid through sulfide oxidation due to the extremely low content of sulphur (<0.01%) within the ores". Primary heap leach ores were identified as "potentially acid forming". - Waste rock: "the kinetics of sulfide oxidation in waste rock were investigated by controlled leaching of a composite sample of waste considered to be representative of bulk waste rock from the Batman Deposit." Leaching trials were continued for 3 years and demonstrated "acceptable" pH, sulphate and calcium levels. The ANC of the waste rock was depleted by 2.5% "suggesting that unacceptable acid formation from weathered waste rock sources is unlikely". - Drill hole samples from Batman ridge area were screened for sulphur content in 1993. Total sulphur assays were considered to be low (0.1 – 0.8%), and NAG tests were considered to be unnecessary due to the low sulphur content. Cf. Mining and Environmental Management Plan (2000) above, which found there was no correlation between sulphur content and NAPP in the waste materials.

TABLE 5-1 VISTA GOLD CORP. – MT TODD GOLD PROJECT Summary of Current Site Environmental Conditions December 2006	
Mt Todd Gold Project Draft Environmental Impact Statement, ***Zapopan N.L., 1992***	- Specific environmental issues for the project: conservation of the Gouldian Finch in the Yinberrie Hills; control of acid drainage; heap leach solution containment; tailings containment; water management; rehabilitation planning; impacts of noise, dust and blasting; impacts on vegetation and fauna; impacts on Aboriginal sites of cultural significance; impacts on historical and Aboriginal archaeological sites; impacts on regional urban and social infrastructure; and general site management issues, such as weeds, mosquito-borne diseases, wildlife and workforce behavior. - Contains summary design report for water supply, heap leach and tailings disposal (AGC Woodward-Clyde, 1992)

The Draft Environmental Impact Statement for the mine released in 2002 gave the following as the specific environmental issues to be considered for the project: conservation of the Gouldian Finch in the Yinberrie Hills; control of acid drainage; heap leach solution containment; tailings containment; water management; rehabilitation planning; impacts of noise, dust and blasting; impacts on vegetation and fauna; impacts on Aboriginal sites of cultural significance; impacts on historical and Aboriginal archaeological sites; impacts on regional urban and social infrastructure; and general site management issues, such as weeds, mosquito-borne diseases, wildlife and workforce behavior.

The conservation of the Gouldian Finch was an important consideration at the start of mining operations in 1993, amidst concerns that the finch was confined to the Yinberrie Hills and may be affected by mining operations. It is now believed that the range of the finch is broader and less emphasis is placed on this issue for Mt Todd by the NT government.

Relationships between the local Aboriginal group (the Jawoyn people), Vista Gold and the NT government are good. The Jawoyn people have strong involvement in the planning for the future of Mt Todd, and at this time they have raised no concerns, which would hinder the re-opening of the mine.

The major environmental challenge for Mt Todd lies in the area of water management. The site contains several sources of acidic water high in dissolved metals (eg copper, cadmium, cobalt, aluminum), in the Batman Pit, the waste rock dump, the heap leach pad and the low-grade ore dump. This water is managed through a combination of evaporation, containment and controlled discharge to streams during major flow events.

The NT government recently expanded the existing pumping network in order to better manage the affected waters, and in particular to enable pumping of water from retention ponds to Batman Pit. Batman Pit has been used as a repository for ARD waters since 2005. Before that time the tailings dam was used; however, that practice was discontinued after it was found that the buffering capacity of the dam had been consumed.

There are other important aspects to the ARD situation of the site. The waste rock dump is a significant generator of ARD and the retention pond below it is undersized for the catchment it drains. The heap leach pad is unstable and vulnerable to erosion during heavy rainfall. There are seeps of poor-quality water below the waste rock dump, the heap-leach pad, and the tailings dam. The acidic waters stored in Batman Pit must be removed before mining can recommence. The reliance on Batman Pit as a repository for contaminated waters could not be continued under mining conditions.

Compliance for water management of the site is based on achieving no uncontrolled discharges from the site and maintaining dissolved copper levels below 10 ug/L (plus background) at a monitoring station approximately 5km downstream of the operation. The NT government has advised that under mining conditions the compliance criteria would be unlikely to change significantly.

The challenges posed by the ARD environment of the site are significant but are believed to be manageable. Vista Gold has engaged consultant MWH to conduct a preliminary assessment of the water management issues which will include preparation of a water balance model, investigation of low-cost mitigation measures and development of a conceptual closure plan.

The NT government has been an active manager of the site during the care and maintenance phase. It has demonstrated willingness to work with Vista to cooperatively address water management at Mt Todd.

5.4.2 Comments on Existing Known Liabilities

The major environmental challenge for Mt Todd lies in the area of water management. The site contains several ponds with acidic water high in dissolved metals which include Batman Pit (RP3), the waste rock dump repository (RP1), the tailings dam (RP7), the heap leach facility, and the low grade ore dump pond (RP2). This water is managed through a combination of evaporation, pumping for containment, and controlled discharge to streams during major flow events.

The license conditions for the site have been breached several times during each wet season while the site has been under care and maintenance. These breaches have taken the form of uncontrolled discharges of wastewater from several ponds, and occasional exceedances of the downstream copper concentration limit. They have occurred despite significant effort and resources applied to water management on the site by the NT government, demonstrating the water management challenges for Mt Todd.

The overflows were caused largely by lack of pumping capability from the heap leach facility and RP5, inadequate pumping capability from RP1 and RP2, and undersizing of the RP1 pond.

During 2006 the NT government installed pumps at the heap leach facility and RP5, which should greatly reduce the future overflows from these ponds. A new pumping system was installed at RP1 in 2006 as part of a strategy to pump excess water from RP1 to RP3, rather than to RP7 as previous. At the time of writing, this pumping system is reportedly operating at a rate of 450m^3/hr, which is lower than the 540m^3/hr design pumping rate.

A new water balance model for Mt Todd has been constructed using the GoldSim platform. The key findings of scenarios run with the model are:

- *The current water management strategy has a probable lifetime of two to four years (until RP3 fills). During this time the management strategy should decrease, but not eliminate, the occurrence of overflows and ARD releases from the site.*
- The water balance excess (defined as pumped water, excluding controlled discharges, plus overflowed water) for the site ranges from 1.5 to 2.1 million cubic meters per year;
- The breakdown in excess water contribution from the ponds is approximately: RP1 – 80%, RP2 – 11%, RP5 – 8%, heap leach facility – 1%;
- the controlled discharge to Edith River from RP1 is a relatively small proportion of the balance, being around 60,000 to 100,000 m^3/year, or 5% of the water balance excess;
- Catchment inflow to RP7 and RP3 is potentially significant. Diversion of catchment flow around RP7 could make the tailings dam a net sink for approximately 1 million m^3/year. However, uncertainty in the catchment flow parameters needs to be resolved;
- the Raw Water Dam overflows an average volume of approximately 8,700,000 m^3/year of good quality water. This represents a potential dilutant source;
- a water treatment plant designed to treat the excess water from site (without mitigation measures) should have a peak design rate of 10,000 m^3/d and an average throughflow of 6,800 m^3/day; and
- it appears that the volumes of uncontrolled discharges from RP1 may have been significantly underestimated by water balance modelling in previous years. Therefore, there is a strong possibility that the reported overflow volumes for the 2006/07 wet season will increase from previous years despite similar management strategies.

The major uncertainties in the model relate to:

- water levels in RP3, RP2, RP5 and the heap leach pond, none of which are currently recorded; and
- catchment runoff contributions (particularly for RP3 and RP7).

The plan codifies the current water management practices by the NT government.

Beyond the 2006/2007 wet season, it is likely that some form of treat and release scheme will be required within two to four years. Broadly speaking, the site water management strategy should take the form of adopting mitigation measures to extend the current practice of storing water in RP3 with the intent of ultimately introducing a 'treat and release' scheme. Earlier introduction of a treatment scheme will have time and cost benefits for the removal of pit water in advance of mining.

Routing of excess water through evaporation cannons discharging into RP3 could be a valuable method to 'buy time' for the RP3 storage strategy and reduce the time needed to empty the pit in advance of mining. The evaporation cannons could potentially be moved to RP7 once the site moves into operation. Other mitigation measures which should be considered are the diversion of catchment flow from RP3 and RP7, installation of telemetered monitoring instrumentation at monitoring locations SW2 and SW4, and controlled release of water from the RWD to dilute treated water release.

5.4.3 Permitting and other Regulatory Requirements

Permitting requirements have been discussed with the relevant authority in the NT government, being the Mining & Petroleum Authorizations & Evaluation Division, of the Department of Primary Industries, Fisheries and Mines ("DPIFM"). The process as described by the authority is outlined below.

Exploration

The following applications, forms and plans are mandatory as part of the exploration approvals process:

- Application for an authorization;
- Nomination for an Operator of a Mining Site;
- Security calculation form; and
- Small Mining/Exploration Operations Mining Management Plan.

The completion of applications and forms are likely to be straightforward. The Mining Management Plan is required to be submitted with the Application for an authorization of Mining Activities. Briefly, the plan will contain :

description of mining activities to be carried out;

safety, health and environmental issues relevant to the mining activities and the management system to be implemented at the mine site; and

a plan and costing of closure activities.

The NT government division advised that the key to approval at this phase is the existence of an effective safety and environment management plan. With such a plan in place to demonstrate good handling of safety and environmental issues, approvals can be expected to proceed.

Mining Development

The exact requirement for mining development approval for the site is currently unknown as three possible approvals paths may apply. The potential costs and timing of the three paths are addressed here and in Section 7 following.

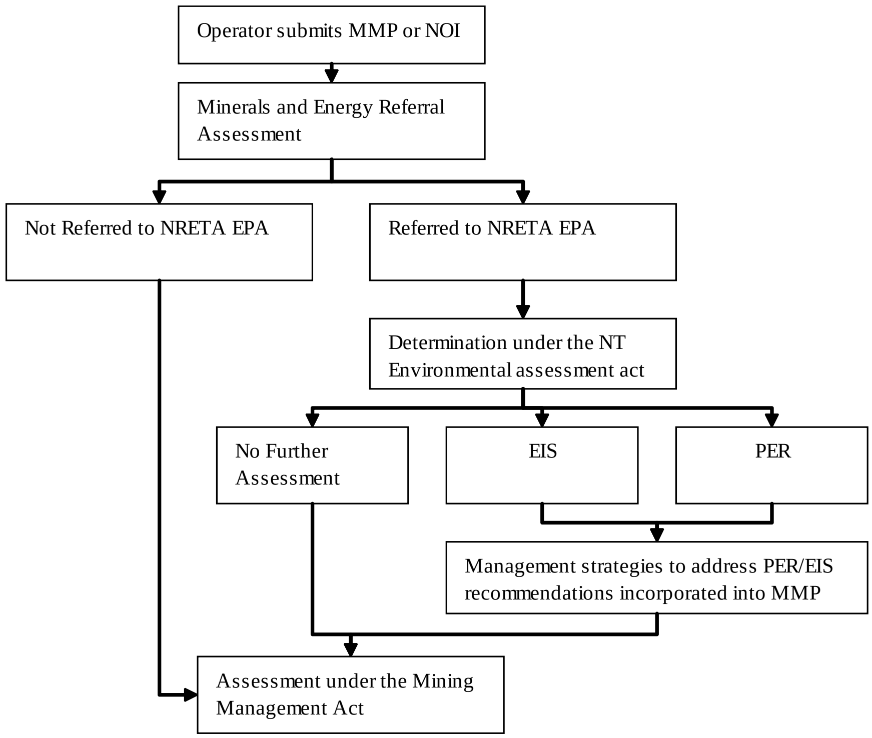
The first step in all cases is the submission of a Notice of Intent ("NOI") to the NT government. The NOI is intended to cover all the major issues relating to the mine development and provide sufficient information (background and technical) to allow a preliminary assessment by the DPIFM. The flow chart on the following page shows the different possible process paths, which may follow from the assessment by the DPIFM. The flow chart is taken from the DPIFM Advisory Note "Environmental Assessment of Mining Proposals".

Notice of Intent (NOI)

The Notice of Intent for the mining development is mandatory as part of the mining development application. The components of a NOI broadly include:

- A General Description of the Mine;
- Description of the Existing Environment;
- Description of the Proposed Works;
- Identification of Issues; and
- Environmental Management of Impacts.

This document should be as thorough as possible to minimize the amount of time taken to assess the document by government.



Public Environmental Report ("PER") and Environmental Impact Statement ("EIS)

If the DPIFM recommends referral to Department of Natural Resources, Environment and the Arts ("NRETA"), NRETA will advise on the requirement for either a PER or 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 project has support within the NT government, which would like to see the site returned to production. The site faces significant challenges in water management but the NT government recognizes that these issues can best be handled when the site is actively managed by an operator. It is believed the NT government would support the development of the site by an operator with demonstrated willingness to responsibly manage the environmental issues.

5.4.4 Estimated Permitting Costs and Spending Schedule for Developing the Project

Exploration

The costing and timing for the exploration approvals program are estimated in TABLE 5-2 below.

TABLE 5-2 VISTA GOLD CORP. – MT TODD GOLD PROJECT Estimated Exploration Permitting Costs December 2006		
Task	**Time[1]**	**Cost ($A)[2]**
Preparation of authorization and security forms and Small Mining Management Plan	2 weeks	$10,000 - $20,000
Total	*2 weeks*	*$10,000 - $20,000*

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

Mine Development

Three cases are presented in TABLE 5-3 are for the possible approval pathways the project could follow. The NT government is not able to advise at this time which pathway the Mt Todd Project would be likely to take.

Ranges are presented for preparation of the PER and NOI documents. It is anticipated that the costing and timing envelopes for the approvals would firm up significantly following submission of the NOI document and subsequent discussions with the NT government.

TABLE 5-3 VISTA GOLD CORP. – MT TODD GOLD PROJECT Estimated Mine Development Permitting Costs December 2006		
Task	**Time[1]**	**Cost ($A)[2]**
Case 1: Assessment under the Mining Management Act (not referred to NRETA)		
Notice of Intent	1 month	$25,000
Total	1 month	$25,000
Case 2: Referred to NRETA, Public Environmental Review Required		
Notice of Intent	1 month	$25,000
Public Environmental Report	3 – 4 months	$100, 000 - $180,000
Total	4 – 5 months	$125,000 - $205,000
Case 3: Referred to NRETA, Environmental Impact Statement Required		
Notice of Intent	1 month	$25,000
Environmental Impact Statement	3 – 6 months	$150,000 - $250,000
Total	4 – 7 months	$175,000 - $275,000

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

Dewatering Requirements and Costs

Hydrogeological information for the Mt Todd site is sparse. Hydrogeological investigations were undertaken by Rockwater in 1987 and 1988, but the reports on these works have not yet been uncovered. Limited further work was undertaken by AGC Woodward-Clyde in 1992, but this work focused on the mine processing areas (particularly the tailings dam and heap leach pad) and did not explore the Batman Pit mining area. The Quigleys Deposit is understood to lie entirely above the water table.

It appears to have been concluded early during the development process that the area of the Batman Pit was characterized by very low permeability, and little further work has been done to validate this assumption. The history of operations at Batman Pit does support the low-permeability finding. A single dewatering bore was installed early in the mining process but was mined through some time before 1996. The bore was not replaced, presumably because it was found to be unnecessary. Dewatering during the subsequent mining period was limited to pumpage from a single sump. Pumping rates from the sump are not known. Dewatering is not mentioned as a concern for the site in any of the environmental documents, which have been reviewed to date.

The Batman Deposit reportedly consists of a sequence of hornfelsed interbedded greywackes, and shales with minor thin beds of felsic tuff. The primary porosity of such a system will be very low, leading to low bulk permeability. However, locally higher secondary permeability zones may occur where related to fractures and fissures in the bedrock. High inflows could occur if such zones are encountered during mining. Fractured-rock secondary permeability zones are typically characterized by variable flow rates (potentially very high) but low storage, implying fractures can be drained readily.

The pre-mining water table in the Batman Pit was believed to be at approximately 60m below the pre-mining ground level, which is similar to the current level. The water level rebound on suspension of mining in 2000 was not recorded, and water levels in the pit have not been recorded during the mine's period of care and maintenance.

Future development of Mt Todd would require further hydrogeological investigations to improve the understanding of dewatering requirements. The investigations would form part of general hydrogeological investigation, which is required to characterize groundwater and establish a groundwater-monitoring program for the site. This work would as far as possible take advantage of drilling conducted for the exploration / infill drilling program.

It is noted that dewatering has been minor and very manageable during previous operations at Mt Todd. However, the hydrogeology of the mining area has not been investigated in sufficient detail to comment conclusively on the future dewatering requirements or provide any costings for dewatering at this time.

5.4.5 Reclamation and Closure

Vista commissioned MWH to prepare the CCP to support a preliminary feasibility study of the restart of mining operations. This CCP 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 separate reports prepared by MWH on the environmental status and water management at the site.

The major environmental challenge for Mt Todd currently lies in the area of water management. The site contains several sources of acidic water high in dissolved metals, including Batman Pit, the waste rock dump repository (“RP1”), the tailings dam (“RP7”), the heap leach facility and the low-grade ore stockpile pond (“RP2”). This water is managed through a combination of evaporation, pumping for containment, and controlled discharge to streams during major flow events. Similar conditions will need to be avoided for closure of a new mining operation at the site.

There are five primary facilities that currently exist that will be carried forward as part of the new mine plan, as well as ancillary facilities and disturbed ground. These are included in this CCP, as listed below:

- Batman Pit and pit lake (“RP3”);
- Waste Rock Dump (“WRD”), waste rock dump pond, and RP1;
- The existing Tailings Storage Facility (“TSF”), and RP7;
- Plant Area (not including stockpiles);
- Miscellaneous facilities (e.g., pipelines); and
- Disturbed ground (e.g., stockpile footprints).

In addition to the above-listed mine features, it is anticipated that a small lined tailings facility and a second unlined tailings storage facility will be constructed during operation to contain sulfide-bearing and benign tailings, respectively. The mine pit and WRD will be significantly enlarged.

The primary objectives of the closure strategies for each of these facilities are to provide physical, chemical and hydrologic stabilization of mine source components to help ensure that waters of the NT are not degraded and human health and the environment are protected. In addition, environmental risk factors are addressed and minimized to the extent practicable, and long-term maintenance is also reduced to the extent practicable. Other closure objectives include protecting public safety, providing a final, stable landform that is compatible with the natural surroundings, and promoting growth of native plant species. A final detailed closure plan will need to be developed and submitted for NT review and approval prior to the beginning of the closure period.

The Batman Pit is currently 114 m deep with a maximum aerial dimension of 950 m. The mine plan indicates that the area of the new pit will be approximately 66.6 ha with approximate dimensions of 700 m by 1,400 m, and will be approximately 470 m deep. Key elements of the pit closure are the depth of oxidation in the pit and the expected groundwater equilibrium level.

Based on the expected pit lake formation, the potential for unacceptable water quality, and the proposed configuration of the pit at the end of mining, the following closure strategies were developed:

- Flooding of the pit with clean water as rapidly as possible to help establish and maintain suitable water quality;
- Construction of safety berms and fencing to prevent unauthorized access; and
- Addition of soil covers and revegetation, if required, to minimize oxidation of select areas of potentially acid generating ("PAG") rock.

The assumption that a lake will develop and maintain the sulfides in a permanent subaqueous environment is key to the closure strategy. This assumption will have to be proven by developing a pit water balance and filling model. That model will require further assessment of the groundwater conditions and area hydrology. Availability of water sources for pit flooding will also have to be assessed.

The WRD, which currently covers an area of 69 ha, is also proposed to be expanded to 172 ha and will be approximately 120 m high. The side slopes will have grades of 33.3% (3H:1V), which is preferred for closure. Based on the physical and chemical data for the WRD and draindown water, the following closure strategies were developed to stabilize the WRD, limit infiltration and manage potential acid rock drainage ("ARD") generation:

- Side slope gradients of not more than 3H:1V;
- The use of non-ARD generating waste rock to construct the outside layer of the dump;
- Development of a dry cover system to limit infiltration of storm water; and
- Revegetation of the cover.

The HLP is 35 ha and 20 to 25 m thick. The side slopes of the HLP are steep (3H:1V to 1H:1.6V) and highly eroded with many rills. The HLP receives rainfall that permeates throughout the heap and fills the containment ponds and then overflows to the heap leach pond (a ring moat around the HLP). The heap leach pond periodically overflows into the environment. The HLP will no longer be used for processing at the site and can therefore be closed and reclaimed immediately. Based on the physical and chemical data for the HLP and draindown waters, the following closure strategies were developed to stabilize the pad, limit infiltration and manage draindown waters from the pad:

- Regrade the pad to a maximum of 3(H):1(V) slopes to limit erosion;
- Construct a cover system over the pad to promote evapotranspiration and reduce infiltration; and
- Revegetate the cover.

Because of the ARD generating character of the HLP, the most likely option for closure of the HLP includes the use of a robust 0.5 m thick clay cap covered by a store and release soil system.

Deposition of rougher tailings will be first to the existing TSF at the site. However, the existing facility does not have the capacity required for the total production of rougher tailings for the project. Therefore, a second TSF will have to be constructed. The area of the current TSF out to the toe of the tailings dam is approximately 201 ha, and will be 236 ha at complete build-out of the facility.

A conceptual design for the second TSF has been developed and will be presented in a separate report. The configuration of this new facility will include a surface area of 195 ha. The facility will be located to the southeast of the current facility. Based on available physical and chemical data for the TSF waters, the following closure strategies were developed:

- Cover the current TSF with non-ARD tailings during operation (primary rougher tailings deposition);
- Deposition of tailings during the final stages of operation to approximate the final surface and reduce grading;
- Regrade TSF surface to develop positive drainage and eliminate ponding, and construct a channel from the TSF to Horseshoe Creek;
- Revegetation of the TSF surface and embankment (may include fertilization or organic amendment); and
- Active draindown water management.

Both of the TSFs will be closed using the same strategies: directly revegetating the benign tailings plus the use of non-PAG waste rock to construct the embankments. The key assumption for the TSFs is that the benign tailings can be directly revegetated with only minor amendment of organic matter or fertilizer. This closure strategy will have to be supported by vegetation test plots to determine the best amendment and seeding plan for the tailings.

The new milling process will generate approximately 7.8 Mt of sulfide tailings. The sulfide content of these tailings will be on the order of 30 to 45 percent, and therefore, should be considered to have a very high potential for generating ARD. Due to this high ARD potential and the residual cyanide component, a lined impoundment will be constructed during construction of the mine to contain the sulfide tailings, hereafter referred to as the Sulfide Tailings Facility ("STF"). The facility will abut the current HLP on the south side and have an area of approximately 21 ha. The conceptual plan for this facility will be described in separate conceptual TSF design report. Two options for tailings closure have been considered depending upon whether the tailings are (1) filter-pressed and dry stacked, or (2) filter-pressed, dry stacked and chemically stabilized. Stabilization would possibly include the use of an alkaline cementaceous agent to produce a neutral low-permeability material. Based on the proposed configuration of the facility and the nature of the tailings (potentially acid-generating with cyanide) and the two disposal options, the following closure strategies were developed:

- Final grading of the surface to promote runoff and minimize erosion;
- Construction of a cover system over the facility; and
- Vegetation of the TSF surface.

For the chemically stabilized tailings, the cover system would simply consist of a soil cover to support vegetation. A more robust cap and cover system using a synthetic low permeability layer would be used if the tailings were disposed of without stabilization.

The current Plant Area has an area of 20 ha. It is assumed that the new plant will be at the same location, be the same approximate size, and contain similar buildings, tanks, piping and milling facilities as the previous plant. It is assumed that the low-grade ore and run-of-mine stockpiles will no longer be present at the beginning of the closure period, and those areas will only contain disturbed ground. Once mining ceases, it is assumed that the Plant will no longer be needed and that mining will not be restarted at the Mt Todd mine in the future. An exception to this is the lime treatment plant, which will be left in place during the early stages of closure. Based on this, the following closure strategies were developed:

- Demolition and removal of all plant buildings and structures;
- Removal and disposal of reagents and other chemicals;
- Removal of all residual ore and tailings, and any impacted soils;
- Closure of ponds and landfills;
- Regrading of the ground surface, as needed; and
- Revegetation of the entire area.

Outside the limits of the above-mentioned facilities will be areas of denuded and/or disturbed ground that will need to be addressed during closure. These areas are a result of roads, parking areas, general disturbance areas, etc. For the purposes of this CCP, it is assumed that the area of disturbed ground will be 83 ha, which accounts for approximately 50% of the areas outside of the facilities presented above and within the limits of the current mine footprint. The primary closure objectives for the disturbed ground not included in the areas discussed in the previous sections will be to reclaim the areas to blend in with the natural terrain and to ensure natural drainage over the long-term. Based on these criteria, the following closure strategies were developed:

- Regrading of the ground surface, as needed;
- The addition of a thin soil cover or organic amendment to allow revegetation, if necessary; and
- Revegetation of the impacted areas.

The last closure consideration is that of mine waters. The primary closure objectives for managing water at the site are to ensure that in the short-term and in the long-term, water impacted from mining activities does not pose health or safety risks to people or negative impact to the environment. Based on the current understanding of the status of the Site at closure, the strategies for managing mine water during closure are separated into two phases:

- Phase 1 – active water management and treatment during the first one to three years, until the seepage rates from the WRD have been significantly reduced, and the Batman Pit water level is at or near the anticipated static, long-term level.

- Phase 2 – passive water treatment once seepage rates have been reduced (after one to three years).

During phase 1 of closure water management, the following strategies will be used:

- Treat impacted water from RP1 and other ARD sources in the lime treatment plant, prior to discharging to Batman Pit.

- Discharge clean water from RP7, the new TSF, if necessary, and outflow from the lime treatment plant to Batman Pit to facilitate more rapid flooding.

Discharging the clean water to Batman Pit will be used as part of the strategy to flood the pit at the start of closure, as described above.

Phase 2 of closure water management will begin after the soil covers have been constructed and the rate of draindown water seepage has subsided. The time that this will require will need to be evaluated during closure, but it is assumed that it will occur within one to three years. Once seepage rates have subsided, then draindown water from the HLP, the WRD, and TSF seepage can be treated in a passive treatment system, if required. Alternatively, the residual draindown water from these facilities may be clean enough (compliant with relevant water quality standards) to be discharged directly to the natural drainages (i.e., Horseshoe Creek, Batman Creek, Stow Creek and/or Edith River). It is assumed that once the phase 2 period commences, RP7 and the reclaim pond in the second TSF will no longer exist. The actual required seepage rate and the nature of the treatment system will need to be evaluated in detail during closure. Examples of passive treatment system elements include anoxic limestone drains, wetlands and bioreactors.

The closure costs were estimated based on the proposed design (areas and volumes) of each of the closure facilities and MWH's experience with similar projects. Using MWH's experience on similar projects, including current reclamation programs, unit rates were developed for each element of the closure strategy, which were then applied to the area or volume of each feature. The majority of the unit rates are per unit volume or area and have been applied to conditions where mine labor is used to conduct the reclamation. Based on this, the conceptual estimated costs for implementing this CCP are US**$30,500,000** including ten years of post-closure care and maintenance but before contingency, as summarized in TABLE 5-4.

Option 2, which includes a more robust cover on the STF increases the capital closure cost by approximately $4,100,000. The total cost difference including the engineering and construction management components is approximately $4,800,000. Post-closure care and maintenance between the two options is not considered to be significantly different.

TABLE 5-4 details MWH's estimated closure costs for the Mt Todd Project.

TABLE 5-4 VISTA GOLD CORP. – MT TODD GOLD PROJECT MWH Conceptual Closure Cost Estimate Summary December 2006	
Area	**Cost (US$)**
Batman Pit	$200,000
Waste Rock Dump	$9,200,000
Tailings Storage Facility - Existing	$4,200,000
Tailings Storage Facility - New	$3,500,000
Sulfide Tailings Facility Lined – New (option 1)	$1,300,000
Plant Area	$500,000
Disturbed Ground	$600,000
Water Management	$300,000
Subtotal	**$19,800,000**
Engineering & Construction Management	$3,200,000
Total Capital Cost for Closure	**$23,000,000**
Operations & Maintenance	$7,500,000
Total Cost	**$30,900,000**
Annual O&M costs until full closure accepted	$300,000

Notes: (1) Cost rounded to nearest $100,000 in current US$.
(2) Lower cost option 1 components included.
(3) Assumes that closure of the HLP estimated to cost $6,900,000 will be completed by the NT prior to project development.

It was necessary to make various assumptions in developing the CCP. Some of the key assumptions, which must be better understood as the closure process proceeds included the following:

- The heap leach pad will not be used in any way by the restart of mining operations and will be reclaimed by the NT at some date prior to commencement of mining operations;
- Sufficient water resources will be available to flood Batman Pit in a reasonable time period (e.g., 6 years or less);
- The Batman Pit lake limnology and watershed hydrology will allow for the establishment of a long-term stable closure condition without long-term water treatment;
- The inert waste rock that will be placed under the cover for the waste rock dump will be suitable to support the soil cover as plant growth media both chemically and in terms of water holding capacity (i.e. it will provide enough water storage to effectively eliminate infiltration);
- Sufficient inert waste rock will be available to allow for TSF embankment construction and for encapsulation of potentially acid generating waste rock;
- The heap leach pile will not have to be rinsed or otherwise treated prior to closure;
- In one scenario, the stabilized sulfide tailings will not interfere with the establishment of

vegetation in a 1 m soil cover section, will demonstrate to be chemically stable long-term, and will be of sufficiently low permeability to act as a low-permeability layer.

- The "rougher tailings" that will be placed over the tailings disposal facility will be suitable as a plant growth media both in terms of water holding capacity and chemically;
- Burial by benign rougher tailings will be sufficient for limiting any future ARD production from the existing tailings;
- The proposed water treatment plant that will be part of the proposed mining facility will be available for closure and early post-closure water treatment; and
- Potential impacts to groundwater are assumed to be minimal and therefore no closure activities associated with groundwater are included in this CCP.

While the HLP will be closed by the NT Government prior to mine construction, another important assumption is that the HLP material will not have to be rinsed or otherwise treated prior to closure.

Several studies to gather information to confirm these assumptions and to provide the other necessary input parameters to model and finalize the design for the various mine facilities will be required prior to construction and closure.

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.

Historical heap leach production compared to the feasibility study is as presented in TABLE 6-1.

TABLE 6-1
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Heap Leach – Feasibility Estimates vs. Actual Production
December 2006

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 tonne per annum was achieved by mid 1997. However, problems with high soluble copper necessitated the closure of the flotation circuit which resulted in reduced recoveries. 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 as General Gold 2 percent, Multiplex Resources 93 percent, and Pegasus Gold Australia 5 percent. The joint venture appointed General Gold as mine operator, which contributed the operating plan in exchange for a 50 percent share of the net cash flow generated by the project, after allowing for acquisition costs and environmental sinking fund contributions. General Gold operated the mine from March 1999 to July 2000.

6.1 History of Previous Exploration

The Batman gold prospect, located about 3.5 km west of Mt Todd, is part of a goldfield that was worked from early in the 20th century. Gold and tin were discovered in the Mt Todd area in 1889. Most deposits were worked in the period from 1902 to 1914. A total of 7.80 tonnes of tin concentrate was obtained from cassiterite-bearing quartz-kaolin lodes at the Morris and Shamrock mines. The Jones Brothers reef was the most extensively mined gold-bearing quartz vein, with a recorded production of 28.45kg. 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, 130kg of molybdenite and a small quantity of bismuth.

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

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

Follow-up work concentrated on alluvial tin and, later, auriferous reefs. Backhoe trenching, costeaning, and ground follow-up were the favored mode of exploration. Two diamond drill holes were drilled at Quigleys Reef. Despite determining that the gold potential of the reefs in the area was promising, AOM ceased work around Mt Todd. The Arafura Mining Corporation, CRA Exploration and Marriaz Pty Ltd all explored the Mt Todd area at different times between 1975 and 1983. In late 1981, CRA Exploration conducted grid surveys, geological mapping and a 14-diamond drill hole program, with an aggregate meterage of 676.5m, to test the gold content of Quigleys Reef over a strike length of 800m. 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
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Property History
December 2006

<u>1986</u> October 1986 – January 1987:	Conceptual Studies, Australia Gold PTY LTD (Billiton); Regional Screening; (Higgins), Ground Acquisition by Zapopan N.L.
<u>1987</u> February:	Joint Venture finalized between Zapopan and Billiton. Geological Reconnaissance,
June-July:	Regional BCL, stream sediment sampling.
October:	Follow-up BCL stream sediment sampling, rock chip sampling and geological mapping (Geonorth)
<u>1988</u> Feb-March:	Data reassessment (Truelove)
March-April:	Gridding, BCL grid soil sampling, grid based rock chip sampling and geological mapping (Truelove)
May:	Percussion drilling Batman (Truelove) - (BP1-17, 1475m percussion)
May-June:	Follow-up BCL soil and rock chip sampling (Ruxton, Mackay)
July:	Percussion drilling Robin (Truelove, Mackay) - RP1-14, (1584m percussion)
July-Dec:	Batman diamond, percussion and RC drilling (Kenny, Wegmann, Fuccenecco) - BP18-70, (6263m percussion); BD1-71, (8562m Diamond); BP71-100, (3065m R.C.)

1989 Feb-June: June: July-Dec:	Batman diamond and RC drilling:BD72-85 (5060m diamond); BP101-208, (8072m RC). Penguin, Regatta, Golf, Tollis Reef Exploration Drilling : PP1-8, PD1, RGP1-32, GP1-8, BP108, TP1-7 (202m diamond, 3090m RC); TR1-159 (501m RAB). Mining lease application (MLA's 1070, 1071) lodged. Resource Estimates; mining-related studies; Batman EM-drilling: BD12, BD8690 (1375m diamond); RC pre-collars and H/W drilling, BP209-220 (1320m RC); Exploration EM and exploration drilling: Tollis, Quigleys, TP9, TD1, QP1-3, QD1-4 (1141 diamond, 278m RC); Negative Exploration Tailings Dam: E1-16 (318m RC); DR1-144 (701. RAB) (Kenny, Wegmann, Fuccenecco, Gibbs).
1990 Jan-March:	Pre-feasibility 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 to Present	Vista Gold Corp. acquires concession rights from the Deed Administrators.

6.2 Drilling

Neither Gustavson nor Vista completed any independent drilling at the property to date. The following discussion of drilling and the current drill hole database is taken from reports by prior interested parties.

The following discussion centers on the drill hole databases that were provided to Gustavson for use in this report. At the present time, Vista has not completed any new drilling. Based on the reports by companies, individuals and other consultants, it is Gustavson's opinion that the drill hole databases used as the bases of this report contain all of the available data. Gustavson is unaware of any drill hole data that have been excluded from this report.

Batman Deposit

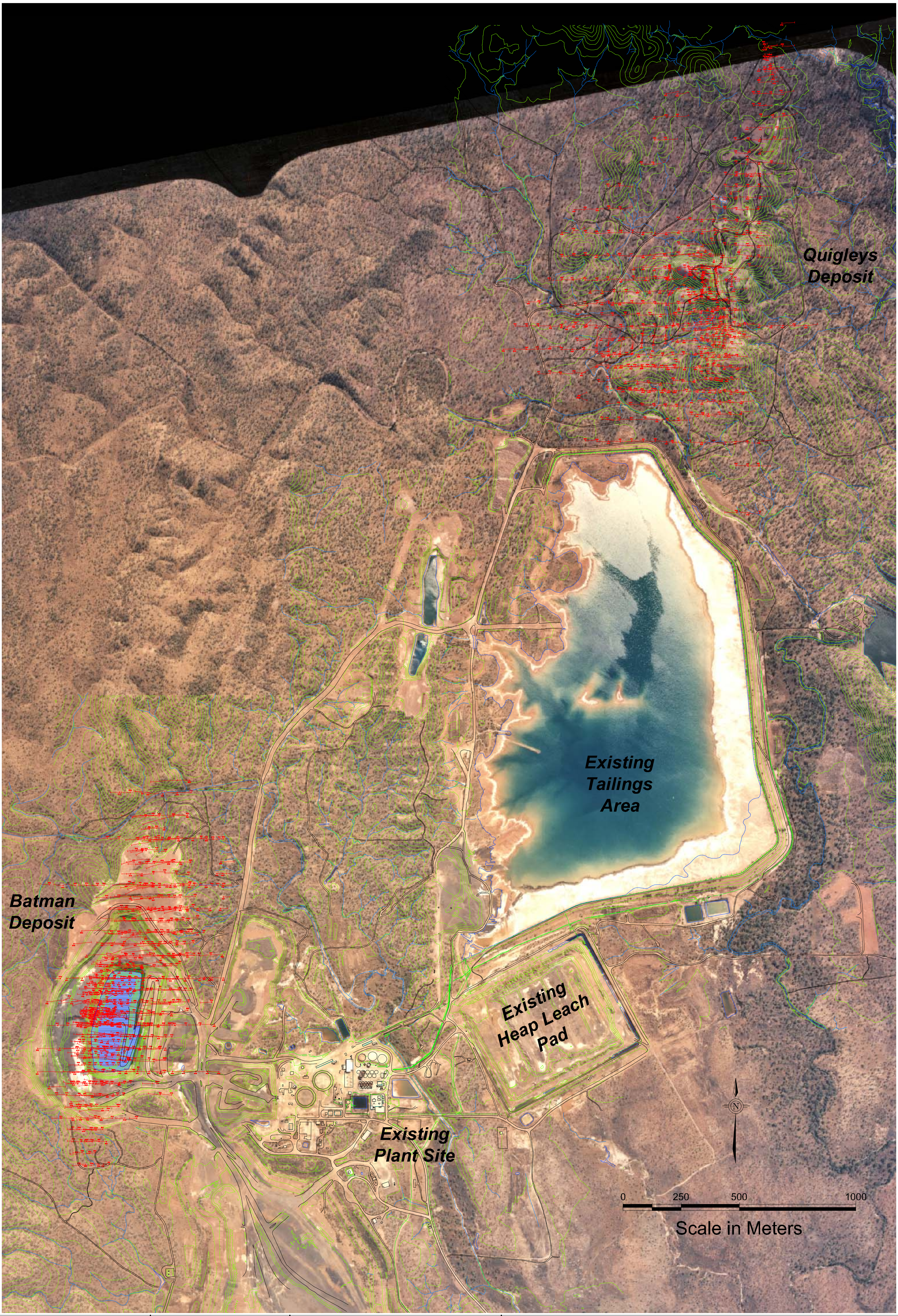
There are 730 drill holes in the Batman Deposit assay database. FIGURE 6-1 shows the drill hole locations for the Batman Deposit. These holes include 225-diamond drill core ("DDH"), 435 reverse circulation holes ("RVC"), and 70 open rotary holes ("OP"). Nearly all of the DDH and RVC holes were inclined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

A series of vertical RVC infill holes were drilled on a 25-meter-by-12.5-meter grid in the core of the deposit to depths between 50 and 85 meters below the surface. Zapopan elected to exclude these holes from modelling 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.

Drill Hole Density and Orientation

Pegasus was aware of the problem of drill hole density within the Batman Deposit. According to Pegasus management, the decision to not drill out the lower portion of the Batman Deposit was based on economic considerations. Section 7.0 of the 1995 BKK feasibility study detailed the decrease in drill hole density with depth. At the time of that study, there were 593 holes in the assay database of which 531 were used in the construction of the MRT block model. Reserve Services Group ("RSG") reported that the drilling density in the Central area oxide and transition zone ore was generally 25 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 drill hole database numbered 730 holes. It is not known if any holes were excluded from the Pegasus exploration models. Most of the new drilling that had been added since the 1994 MRT model was relatively shallow. TWC reviewed PGA's 50-meter drill sections through the Batman Deposit and saw that there was a marked decrease in drill hole spacing below 1000 RL (the model has had constant 1000 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 drill hole 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



GUSTAVSON ASSOCIATES GEOLOGISTS · ENGINEERS · APPRAISERS	DATE OF ISSUE. 12/22/2006	CLIENT NAME. **VISTA GOLD CORP.**	FIGURE 6-1 DRILL HOLE LOCATION MAP
	PROJECT NO. VIS005	PROJECT NAME. MT. TODD GOLD PROJECT	
	DRAWING FILE NAME. Fig6-1.dwg		

300 meters.

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

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

Quigleys

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

TABLE 6-3
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Summary of Quigleys Exploration Database
December 2006

Drill Holes	Gold Assays (approx 1m)	Copper Assays (approx 1m)	Lithologic Codes
632	49,178	41,673	51,205

Snowden completed a statistical study of the Quigleys drill hole database in order to bias test it. The following discussion is from their report and details their findings.

6.3 Comparison of Drilling Programs (After Snowden)

Preliminary variography indicated a high nugget effect of up to 0.6. This means that 60% of the variability in grade is attributable to random error (such as occurs in grade results when assaying two halves of the same drill core). A nugget of 0.4 is reasonably common in shear hosted gold deposits while a nugget of 0.6 is high. While much of the nugget effect may be due to inherent variability in the deposit, several other factors can contribute to a high nugget effect, including variations in the quality of drilling methods, sample collection, sample preparation and assaying techniques.

A comparison was made of the older drilling with the recent Pegasus drilling (defined by Pegasus personnel, with recent being QP087 to QP629 plus QD0ll to QD038). This comparison was made within the wide envelope prior to final trimming, referred to as Shear 2. Initially the comparison was made on all samples except those north of 13400 N where there was uneven representation of the older and new drilling. Summary statistics are tabulated in TABLE 6-4.

TABLE 6-4
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Summary Statistics old and new drilling within Shear 2 – Quigleys Deposit
December 2006

	Old	**New**
Number of samples	2592	4241
Minimum gold value (g/t)	0.0	0.0
Maximum gold value (g/t)	24.8	32.95
Mean gold value (g/t)	0.87	0.65
Variance	3.43	1.97
Geometric Mean gold value (g/t)	0.33	0.29
Log. Variance	1.96	1.58
Sichel's mean	0.87	0.64
Ref.	DDC2N	DDC2S

This global comparison suggested a bias in results, with the new drilling returning a lower average grade than the old drilling. A scatter plot of grades of the new drilling against grades of the old drilling at selected percentiles from the respective sample populations ("Q-Q plot") illustrates the bias.

The bias could be due to a difference in sampling or assay methods, clustering of the old drilling in the high grade area or could be related to drilling orientation. A significant difference between the old and new drilling is the drilling orientation. Recent drilling is almost universally directed at -60° to -90°, whereas earlier campaigns were mainly at this orientation but included some holes at varying orientations. This raises two possibilities; that a component of the mineralization could be oriented so that recent drilling at -60° to -90° would not regularly intersect it, or that the earlier drilling was aligned parallel to the mineralization.

To investigate these possibilities, two areas were compared from the central and southern portions of the shear zone. Firstly, old and new drilling were compared within a central area where the older drilling is variably oriented (approx 10900 to 11175 E, 13090 to 13340N). Secondly, a similar comparison was made within a southern area where the old and new drilling are both oriented at -60° to -90° (approx 11100 to 11230 E, 12850 to 13020 N). A summary of the findings is presented in TABLE 6-5.

TABLE 6-5 VISTA GOLD CORP. – MT TODD GOLD PROJECT Summary statistics old and new drilling in selected areas December 2006				
	Northern Area (variable orientation)		Southern Area (similar orientation)	
	Old	**New**	**Old**	**New**
No. of samples	1314	899	769	826
Minimum gold value (g/t)	0.0	0.0	0.0	0.0
Maximum gold value (g/t)	24.80	32.95	10.57	12.10
Mean gold value (g/t)	1.14	0.87	0.56	0.62
Variance	5.5	3.98	0.72	0.74
Geometric Mean of gold (g/t)	0.39	0.37	0.30	0.39
Log. Variance	2.09	1.56	1.45	0.93
Sichel's mean gold value (g/t)	1.11	0.81	0.61	0.61

The comparative statistics show that it is the bias in the area of variable orientations which gives rise to the bias seen in the overall data. For the area where both the old and new drilling is at a similar orientation, there is little evidence of any bias.

The comparison was based on a simple "cookie-cut" selection of holes and is not conclusive. Some bias between old and new drilling is evident for a selected area where the orientation of drill holes varies. However, the selected area is known to include some older drilling clustered in a high grade portion of the resource (along 13228N), which could also contribute to the differences seen.

Snowden’s conclusion was that the 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, although the source of the bias should also be resolved.

6.4 Sampling Method and Approach

Neither Gustavson nor Vista has completed any independent re-sampling or assaying to verify any of the historic data. The following discussion of sampling methodology and approach is taken from reports by prior interested parties.

Since the property is currently not operating, Gustavson did not witness any sampling personally. However, based on company, individual, and other independent consultants reports, the following description details the sampling methodology and approach used at the Batman and Quigleys Deposits.

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

Since the property is currently not operating, Gustavson 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 Gustavson'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 a assay database that fairly represents the tenor of the mineralization at Batman.

6.5 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 modelling purposes.

Drill hole 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 three-kg to four-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 prudent practice. Every 20th assay sample was subjected to assay by an independent lab. Standards were run periodically as well, using a non-coded sample number to prevent inadvertent bias in the labs.

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

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

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

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 five percent of all RVC rejects were sent as duplicates and duplicate pulps were analyzed for 2.5 percent of all DDH intervals. Duplicate halves of 130 core intervals were analyzed as well. Overall, Mt Todd's check assay work is systematic and acceptable. The feasibility study showed that the precision of field duplicates of RVC samples is poor and that high errors exist in the database.

The 1995 feasibility study stressed that because of the problems with the RVC assays, the RVC and OP assays should be kept in a separate database from the DDH assays. However, since that time, the majority of the identified assaying issues have been corrected by GGC based on recommendations of consultants. It is Gustavson'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

Gustavson 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.

6.6 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.

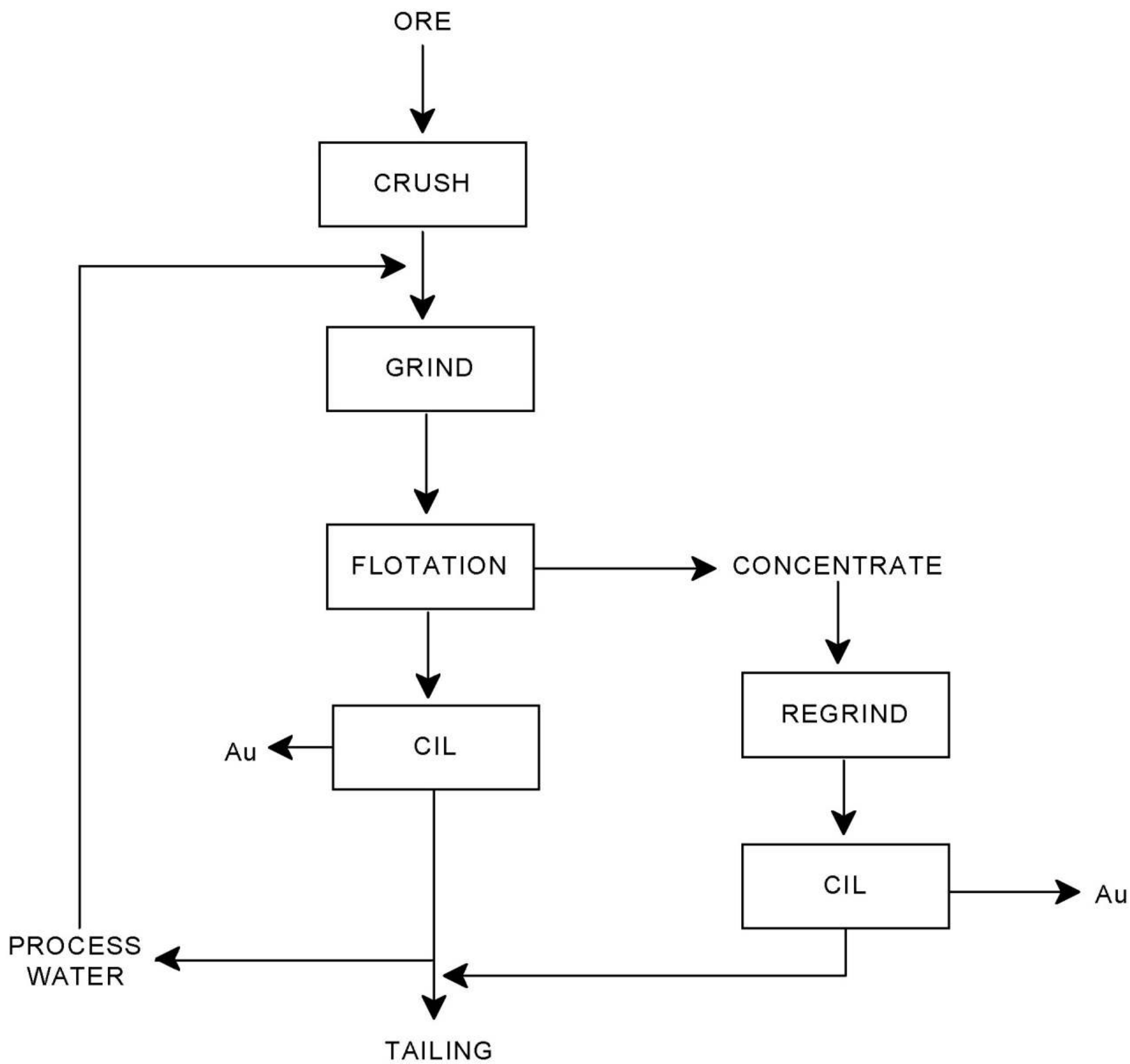
Pegasus Gold Australia Pty Ltd. and earlier owners did extensive metallurgical testing from 1988 to 1995 to develop a process flowsheet for recovering gold from low-grade extremely hard rock. The treatment route, based on the metallurgical studies, was engineered to provide for the recovery of a sulfide 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.



Note: from RDI

GUSTAVSON ASSOCIATES GEOLOGISTS · ENGINEERS · APPRAISERS	DATE OF ISSUE. 12/29/2006	CLIENT NAME. ***VISTA GOLD CORP.***	FIGURE 6-2 PLANT PROCESS FLOWSHEET FOR MT. TODD PROJECT (AS DESIGNED)
	PROJECT NO. VIS005	PROJECT NAME. MT. TODD GOLD PROJECT	
	DRAWING FILE NAME. Fig 6-2.cdr		

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 start up due to a collapse in gold price, under performance of the processing plant and higher than projected operating costs.

6.7 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.

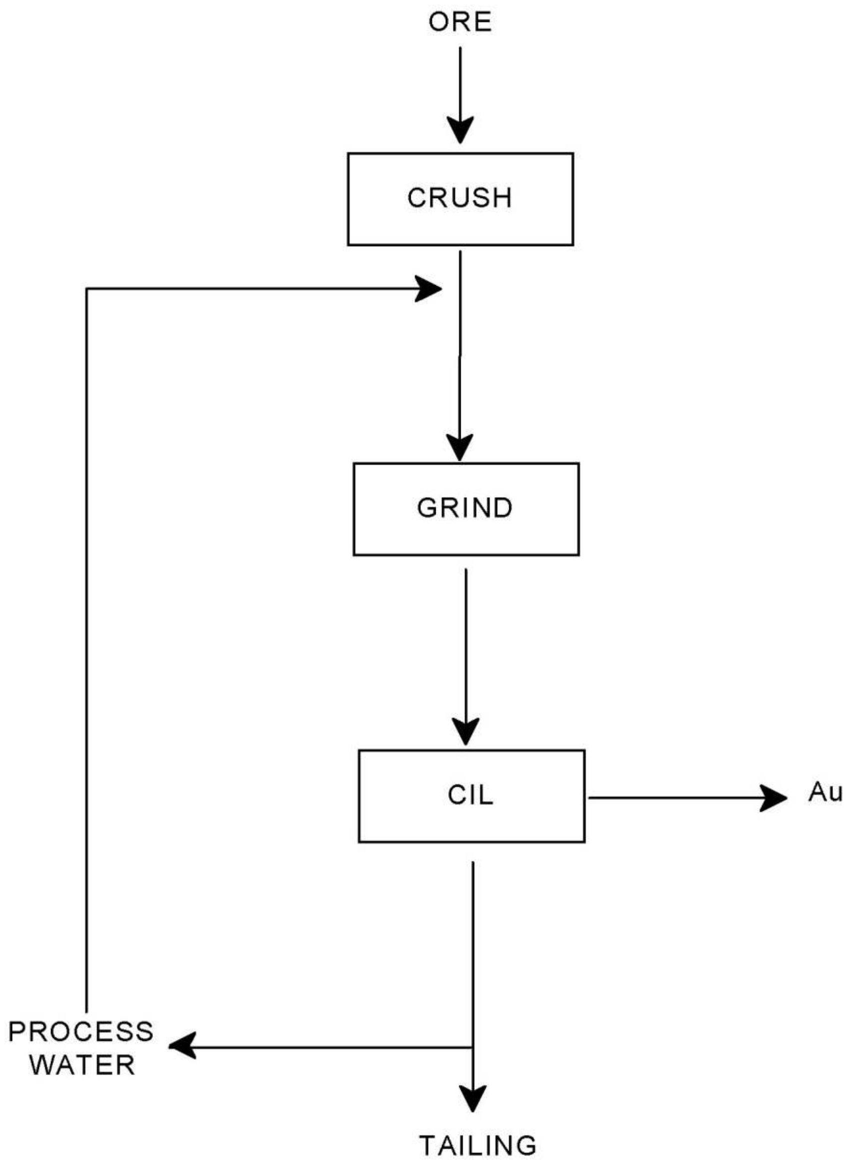
Crushing

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.

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.



Note: from RDI

GUSTAVSON ASSOCIATES GEOLOGISTS · ENGINEERS · APPRAISERS	DATE OF ISSUE. 12/29/2006	CLIENT NAME. ***VISTA GOLD CORP.***	FIGURE 6-3 MODIFIED PLANT PROCESS FLOWSHEET FOR MT. TODD PROJECT
	PROJECT NO. VIS005	PROJECT NAME. MT. TODD GOLD PROJECT	
	DRAWING FILE NAME. Fig 6-3.cdr		

- 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.

- 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.

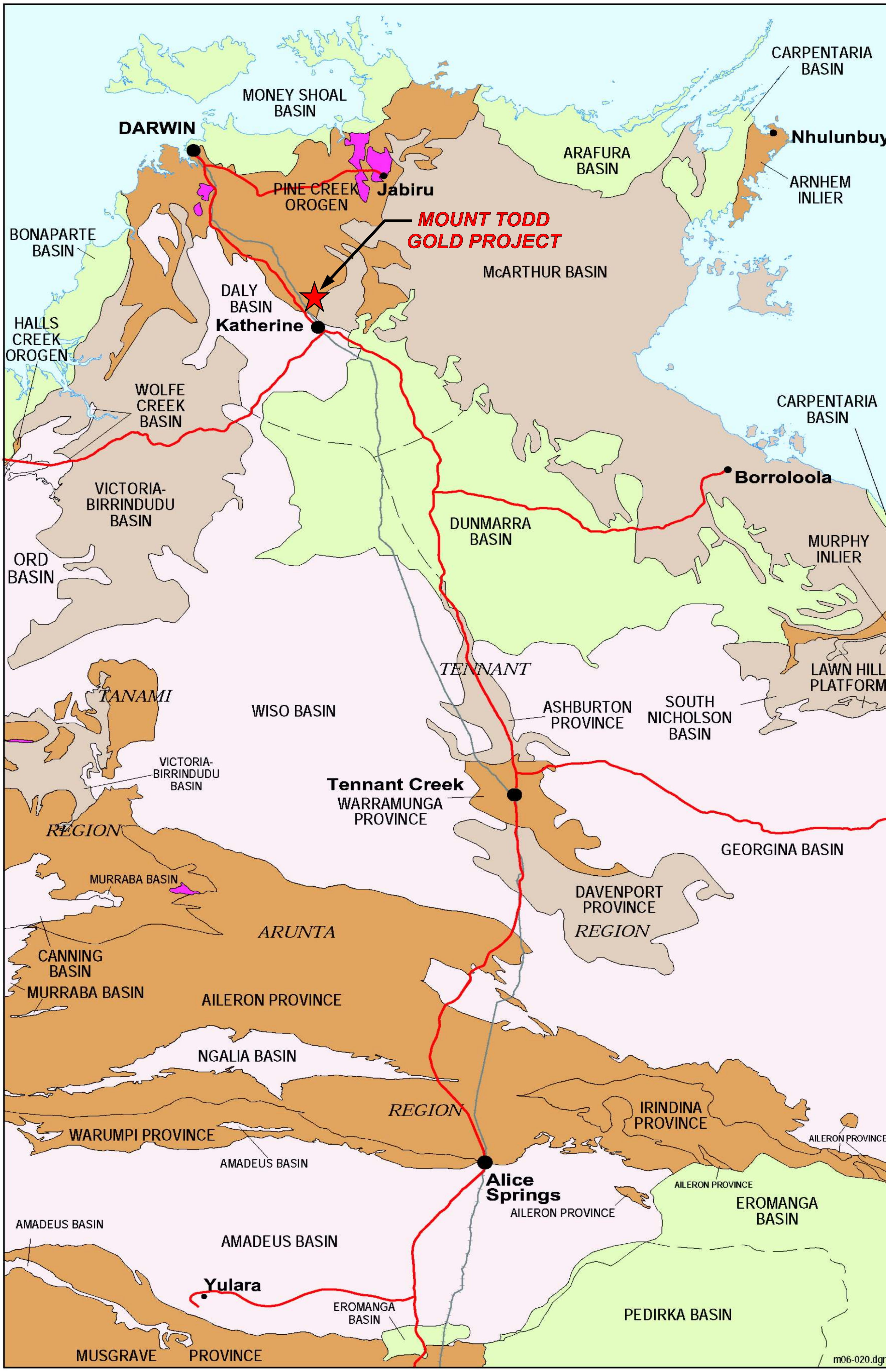
Nineteen lithological units have been identified within the deposit and are listed in TABLE 7-1 below from south to north (oldest to youngest).

TABLE 7-1 VISTA GOLD CORP. – MT TODD GOLD PROJECT Geologic Codes and Lithologic Units December 2006		
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

GUSTAVSON ASSOCIATES GEOLOGISTS · ENGINEERS · APPRAISERS	Date of Issue. 08/01/2006 Project No. VIS005 Drawing File Name. Fig7-1.cdr	Client Name. **VISTA GOLD CORP.** Project Name. Mt. Todd Gold Project	FIGURE 7-1 General Geologic Map

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 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 canalization 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:

(a) retrogressive contact metamorphism during cooling and crystallization of the Tennysons Leucogranite;
(b) fracturing of the country rock carapace;
(c) sinistral reactivation of a NE-trending basement strike–slip fault;
(d) brittle failure and fluid-assisted brecciation; and
(e) 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 Zone

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 Zone due to quartz veining not being as abundant as the Core Zone. 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 zones in places.

Footwall Zone

Like the Hanging Wall Zone, the mineralization is patchier than the Core Zone 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 Zone. Veining is both bedding parallel and north south trending. The mineralization appears to have migrated from the south along narrow north-south trending zones and “balloon out” parallel to bedding around the felsic tuffs.

9.2 Quigleys Deposit

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

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

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

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

10.0 EXPLORATION

Vista Gold Corp. acquired the Mt Todd properties on June 16, 2006. No new exploration activities have been conducted by Vista or Gustavson to date. Vista is in the process of evaluating pre-existing exploration programs and the accompanying data to ascertain the "best" development program to be applied to the property to take it to a development decision point.

This Report summarizes the work carried out to date. Prior exploration programs are discussed throughout this report and therefore this "Exploration" section is not used.

Any "new" exploration work discussed in this report will be documented in the Mineral Resource and Mineral Reserve Estimates section of this report, as that section provides a description and discussion of the resource estimation activities of Gustavson Associates as supervised by John W. Rozelle, on behalf of Vista.

11.0 DRILLING

Neither Gustavson nor Vista completed any independent drilling at the property to date. The following discussion of drilling and the current drill hole database is taken from reports by prior interested parties.

12.0 SAMPLING METHOD AND APPROACH

Neither Gustavson nor Vista has completed any independent re-sampling or assaying to verify any of the historic data.

13.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Neither Gustavson nor Vista has completed any independent re-sampling or assaying to verify any of the historic data. The following discussion of sampling methodology and approach is taken from reports by prior interested parties.

14.0 DATA VERIFICATION

14.1 Drill Core and Geologic Logs

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

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

14.2 Topography

The topographic map of the project area was delivered electronically in an AutoCAD® compatible format and is dated December, 1999. The surveyed drill hole collar coordinates agree well with the topographic map; it is Gustavson'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

Gustavson has not completed any independent analytical verification on the drill hole assays. While the property has had some analytical issues in the past, it is Gustavson's opinion that these issues have been rectified and that the current assay database fairly represents the tenor of the mineralization present. These issues are discussed in greater detail in SECTION 13.0 of this report. Gustavson's willingness to accept the current assay database as representative is in part based on work and reports by The Winters Company (TWC), Snowden Associates PTY Ltd (Snowden), Mineral Resource Development Incorporated (MRDI), Pincock, Allen & Holt, Inc. (PAH) and General Gold Operations PTY Ltd (GGC).

Since Vista will be re-assaying a portion of the Batman diamond drill core for copper, additional opportunities will exist to also independently verify the reported gold grades.

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

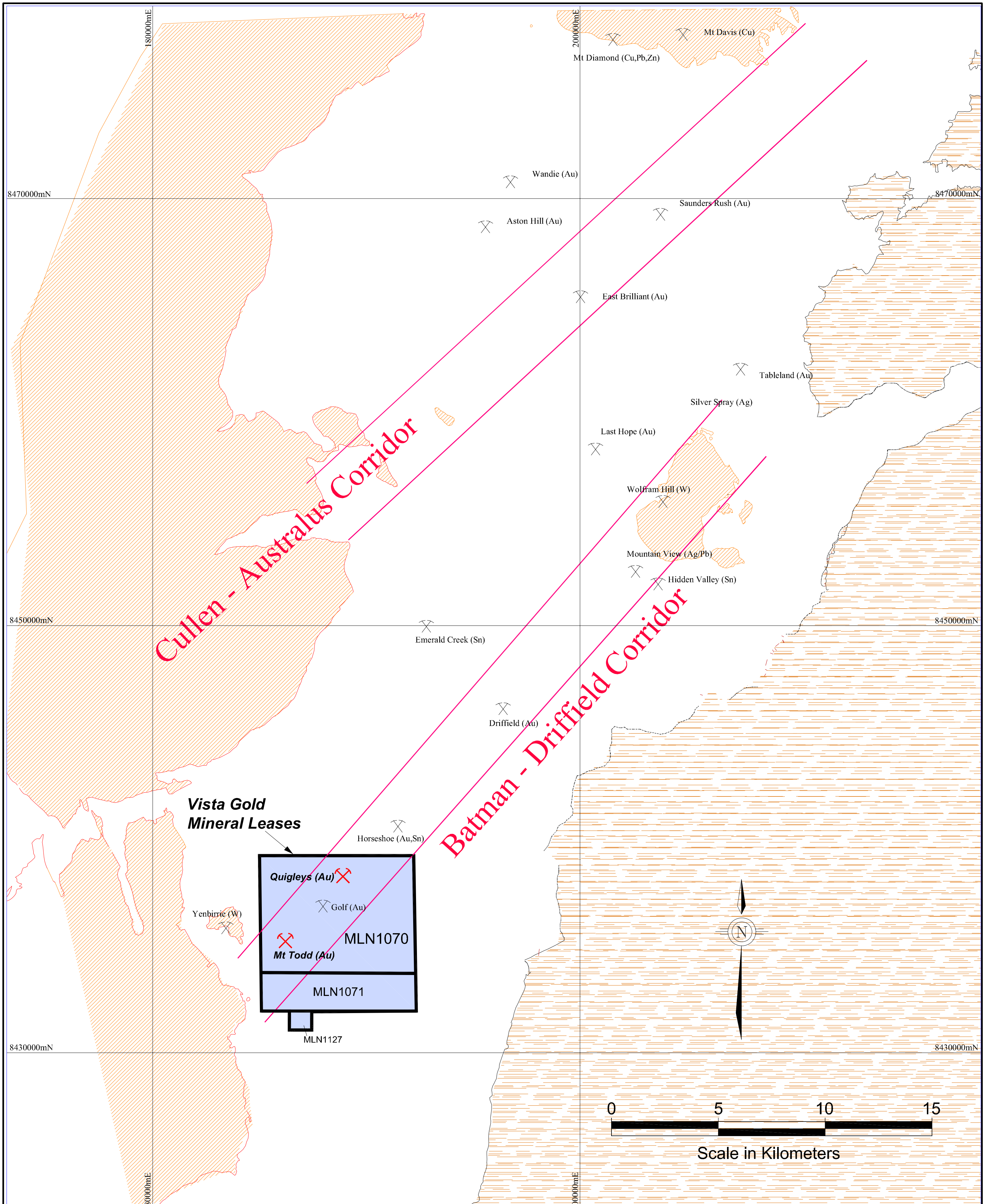
Previous work defined two gold prospects. At Anomaly One, RC drilling by Billiton returned peak gold intercepts of 5 m of 2.93 g Au/t and 33 m of 1.21 g Au/t (including 6 m @ 2.54 g Au/t). Pegasus drill tested Anomaly One with 16 RC holes, for 1599 m on four sections between 10200N to 10700N. Intersections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



GUSTAVSON ASSOCIATES
GEOLOGISTS · ENGINEERS · APPRAISERS

Date of Issue.	Client Name.	
07/18/2006	**VISTA GOLD CORP.**	
Project No. VIS005	Project Name.	
Drawing File Name. Fig15-1.dwg	MT. TODD GOLD PROJECT	

FIGURE 15-1
STRUCTURAL TRENDS WITH MINES & PROSPECTS

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

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

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

Resource Development Inc., (RDi) was contracted by Vista to undertake a study to develop order-of-magnitude capital and operating costs for the conceptual process flowsheet for the Mt Todd Gold Project located in NT, Australia. The conceptual process flowsheet was developed by RDi based on the review of the historical metallurgical testwork, the plant process flowsheet, and problems encountered during the operation of the plant[1].

This report discusses the conceptual process flowsheet, the assumptions made for sizing and costing of equipment and provides the order-of-magnitude capital and operating costs for processing 10.65 million metric (MM) tonnes per year of ore.

16.1 Process Flowsheet Description

The conceptual process flowsheet developed in an earlier study is given in Figure 16-1[1]. The run-of-mine (ROM) ore will be sent to the primary crusher (Gyratory Crusher) and conveyed to a stockpile. The ore will be reclaimed from the stockpile and subjected to secondary and tertiary crushing. The fine crushed ore will be ground to P_{80} of 104 microns in a ball milling circuit. The ground slurry would be sent to the rougher flotation circuit. The rougher concentrate would recover ± 7.5% of weight, ± 90% of gold and ± 75% of copper. The rougher tailings would have negligible amount of sulfides and would be environmentally friendly and sent to the existing or new tailings pond. The rougher concentrate would be reground to P_{80} of 37 microns and subjected to three stages of cleaner flotation to recover copper and gold. The final concentrate is projected to assay 24% Cu and contain ± 50% of the gold. The cleaner flotation tailings would be cyanide leached in the CIL circuit. The leach residue would be subjected to cyanide destruction and the residue will be thickened and filtered and trucked to a lined tailing area.

16.2 Process Design Criteria

Majority of the process design criteria was taken from the Pegasus feasibility studies for the processing of the sulfide ores. The conceptual process flowsheet recommended by RDi differs slightly from the flowsheet incorporated in the Mt Todd Project by Pegasus.

The process design criteria used to develop the capital and operating costs are given in TABLE 16-1.

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

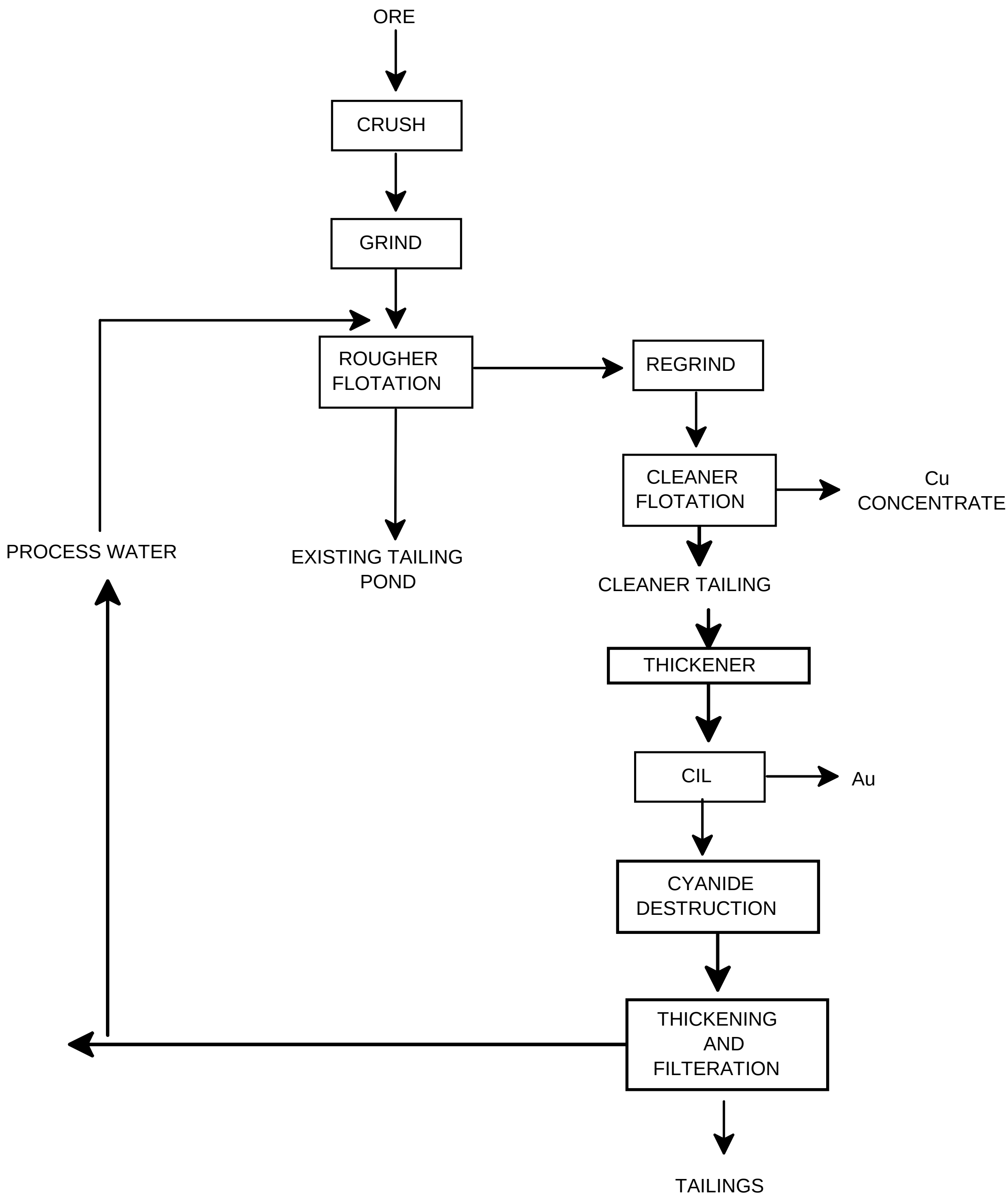


FIGURE 16-1
Conceptual Process Flowsheet for Mt Todd Ore

TABLE 16-1 VISTA GOLD CORP. – MT TODD GOLD PROJECT Process Design Criteria for Mt Todd Project December 2006	
GENERAL	
Feed Grade, % Cu	0.04
Feed Grade, g/t Au	1.10
Ore Specific Gravity, t/m^3	2.65
Moisture in ore, %	3
Abrasion Index	0.15
Crushing Work Index	20
PRIMARY CRUSHER	
Maximum ROM size, mm	1100
Crusher Product, P_{80} mm	150
Coarse Ore Live Stockpile, t	30,000
Coarse Ore Total Storage Capacity, t	120,000
3.0 GRINDING	
Bond's Ball Mill Work Index	26
SAG mill, F_{80}, mm	150
P_{80} Microns	1000
% to Pebble Crusher	25
Ball Mill, F_{80}, mm	6.5
P_{80} Microns	106
Circulating load, %	300
ROUGHER FLOTATION CONDITIONER	
Conditioner, minute	1
Flotation Pulp Density, %	32
Flotation Time, Rougher, min	15
Flotation Time, Scavenger, min	7.5
Rougher Concentrate Weight, %	7.5
Rougher Conc, Grade % Cu	0.40
g/t Au	13.17
REGRIND MILL	
F_{80} Microns	106
P_{80} Microns	38
Bond's Ball Mill W_i	19
FIRST CLEANER FLOTATION	
Flotation Pulp Density, %	15.7
Flotation Time, min	10
Concentrate Weight, %	0.75
Concentrate Grade, % Cu	4
g/t Au	125
SECOND CLEANER FLOTATION	
Flotation Pulp Density, %	28.1
Flotation Time, min	6
Concentrate Weight, %	0.25
Concentrate Grade, % Cu	12
g/t Au	335
THIRD CLEANER FLOTATION	
Flotation Pulp Density, %	20
Flotation Time, min	4

TABLE 16-1 VISTA GOLD CORP. – MT TODD GOLD PROJECT Process Design Criteria for Mt Todd Project December 2006	
Concentrate Weight, %	0.11
Concentrate Grade, % Cu	24
g/t Au	493
CIL CIRCUIT	
Leach Pulp Density, % solids	40
Leach Time, hrs	36
Specific Gravity, t/m^3	3.2
Feed Grade, % Cu	0.021
g/t Au	5.47
Extraction % Au	94.0
COPPER CONCENTRATE	
Thickening Rate, $ft^2/t/24$ hr	7
Filtrate Rate, $t/hr/ft^2$	0.09
LEACH RESIDUE	
Thickening Settling Rate, $ft^2/t/24$ hrs	1.05
Filtrate Rate, $t/hr/ft^2$	0.063

16.3 Material Balance

The material balance for the process flowsheet was simulated using METSIM. The recovery of the gold and copper is given in TABLE 16-2.

TABLE 16-2 VISTA GOLD CORP. – MT TODD GOLD PROJECT Overall Gold and Copper Recovery December 2006					
Product	**Recovery, %**			**Grade**	
	Wt.	**Cu**	**Au**	**% Cu**	**g/t Au**
Copper concentrate	0.11	70.0	52.3	24.0	493
Leach Circuit	–	–	34.6	–	–
Total	–	70.0	86.9	–	–

17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

The following sections detail the thought processes, procedures, and results of Gustavson's independent estimate of the contained gold resources of the Batman Deposit. Only the Batman Deposit currently has classified resource estimates. As detailed elsewhere in this report, the Quigleys Deposit, even though considerable data are available, will require additional work prior to estimation of a resource.

17.1 BATMAN DEPOSIT

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
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Summary of Batman SG Diamond Core Data by Oxidation State
December 2006

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	3.9	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
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Batman Pit Sample SG Data
December 2006

	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

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
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Quigleys Deposit SG Data
December 2006

Oxide within modelled shear (t/cm)	2.60
Oxide Waste (t/cm)	2.622
Transition within modelled shear (t/cm)	2.65
Transition Waste (t/cm)	2.577
Primary within modelled 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.

Batman Exploration Database

The exploration database consists of 730 drill holes, 226 diamond holes and 504 percussion holes. A total of 47,029 samples exist within the 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.
- Some samples with assays below detection have been incorrectly coded as not sampled.

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

MicroMine 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.).

Quigleys Exploration Database

TABLE 17-4 details the Quigleys exploration database.

TABLE 17-4
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Summary of Quigleys Exploration Database
December 2006

Drill Holes	Gold Assays (approx 1m)	Copper Assays (approx 1m)	Lithologic Codes
632	49,178	41,673	51,205

At the present time, no resource estimates have been made for the Quigleys Deposit.

Validation of the Batman Exploration Database

The exploration database has been validated in Gemcom for missing intervals, missing holes, invalid interval lengths and erroneous azimuth, dip and collar co-ordinates. The assay file was validated against the collar and survey file for interval length errors. No significant errors were encountered.

17.4 BATMAN SOLIDS

In previous resource models, the Batman Deposit resource was calculated either non-constrained, or with a grade shell for grade interpolation. Lithological units had not fully been taken into account. The GGC resource model and the Gustavson resource model both incorporate the lithological units interaction with mineralization within the deposit.

Solids have been created in Gemcom to flag the assay data and block model for oxidation state, lithological boundaries and mineralized zone.

Oxidation Solids

Pegasus oxidation solids were found for oxide, transitional, and primary. Close inspection of exploration data, pit inspections, and specific gravity test work showed these solids to predict the oxidation states with a high degree of certainty. These oxidation solids have been used to flag the block model and assay data. Coding of oxidation is the same as the block model rock type coding. TABLE 17-5 details the codes and SGs assigned to the oxidation solids.

TABLE 17-5
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Oxidation Model Codes & Associated SG – Batman Deposit
December 2006

Oxidation State	Code	SG (t/cm)
OXIDE	100	2.47
TRANSITIONAL	200	2.67
PRIMARY	300	2.77

Lithological Solids

Close inspection of the grade control and exploration data shows that the lithology interacts with the orientation and nature of mineralization. Pegasus mine geologists had created a solid interpretation of the lithology using Gemcom. These lithological solids predict the lithological boundaries well. The Pegasus lithological solids were then extracted into a GEMCOM polygon database and projected to fit the size of the entire block model. These polygons were extracted as three-dimensional rubber sheets (3drs) and solids were created that were then used to code both the assay data and the block model. Some units, notably lithologic unit SHGW23, could be broken up into sub units of shale and greywacke, but the grade distribution within the unit is relatively consistent.

Mineralization Zoning

Close inspection of the grade control and exploration data, shows that the mineralization can be zoned in to areas that have similar characteristics. Four major zones exist, the core, the hanging wall, the footwall and outside. All these zones show changes in mineralization characteristics across lithological boundaries.

These zones were created visually using a combination of assay data (both grade control and exploration assays), quartz percent, quartz veining per meter, vein orientation, fractures per meter and lithology and sulfides. Below are the details describing the zones.

Core Zone (GGC Code = 10000, Gustavson Code = 1000)

This zone is the main mineralized zone within the deposit. It is characterized by having a high quartz percentage with a high vein frequency. Veins are orientated at 0^0 to 20^0 to the north and dipping at around 80 to 60 degrees to the east. Mineralization within the Core zone is more consistent than in the other zones.

Footwall Zone (GGC Code = 20000, Gustavson Code = 2000)

The Footwall zone is adjacent to and to the west of the Core zone. Less quartz veining and patchier grade, distinguish the footwall zone from the core zone. The north south jointing is present, but not all joints are filled with quartz/sulfides. Lithology tends to control the intensity of mineralization. The western boundary of the footwall zone is where the north-south jointing intensity decreases dramatically.

Hanging wall Zone (GGC Code = 30000, Gustavson Code = 3000)

The Hanging wall zone is adjacent to and to the east of the core zone. As with the Footwall zone, less quartz veining and patchier mineralization distinguish the hanging wall zone from the Core zone. The north-south jointing is present but not all joints are filled with quartz/sulfides. Lithology tends to control the intensity and style of mineralization. The hanging wall Zone doesn’t occur north of T21 and south of T21 to GWSH23 the mineralization has a bedding trend. A large quartz-sulfide vein, up to two meter in thickness, consisting of quartz and pyrrhotite marks the boundary of the Hanging wall Zone and the Core Zones in places (it is possible source of large magnetic anomaly). This vein is only slightly mineralized. The eastern boundary of the hanging wall zone is marked by the last (eastern most) of the consistent quartz filled joints.

Outside Zone (GGC Code = 0, Gustavson Code = 0)

The Outside zone is all material outside the other zones. Narrow inconsistent north-south trending zones are present, as well as bedding parallel mineralization around bedding faults and areas of shale and felsic tuff, namely SH22 to SH20.

FIGURE 17-1 provides a detailed picture of these three main mineralized zones for the Batman Deposit. The Hanging Wall zone is blue, the Core is yellow and Foot Wall is green. The 3-D view shows partial drill holes within a given lower and upper elevation horizontal slice.

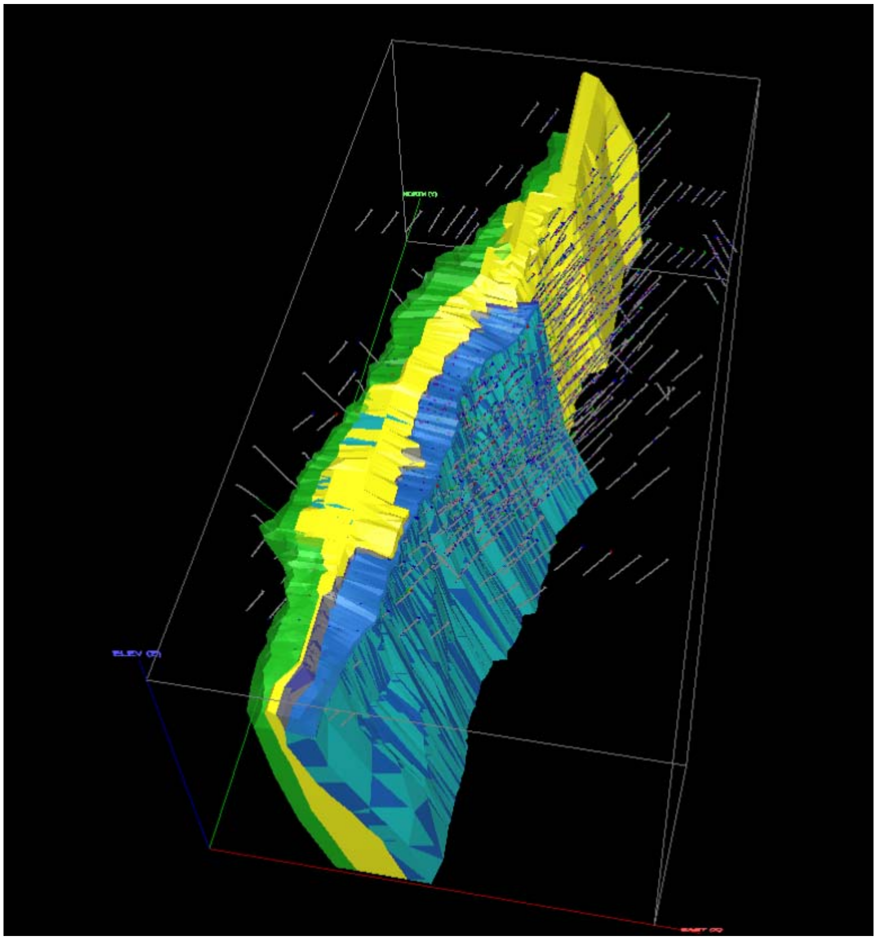


FIGURE 17-1: Simplified Lithological Units

17.5 BATMAN DRILLHOLE CODING

The drill hole databases were coded for oxidation, geological zone and mineralized zone, into separate tables in the database. These tables were named;

- Complith: geological zone coding.
- Compox: oxidation coding.
- Compzone: mineralization zones coding

Coding was checked visually in section, plan and 3D for errors.

Lithological Coding

GGC modelled eighteen lithological units identified within the deposit and are listed in TABLE 17-6 from south to north (oldest to youngest). These lithological codes were further consolidated into five codes within mineralization zones. FIGURE 17-2 shows the pattern of the 18 lithologies. The N-S trend of the "mineralized zones" crosscuts across a NW-SE pattern of lithologies. TABLE 17-6 also lists a five category "simplified lithology" that is shown in FIGURE 17-2.

TABLE 17-6
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Summary of Geologic Model Coding – Batman Deposit
December 2006

Lithology	Lithologic Unit Code	Description	Geological Zone Code (core, footwall, outside)	Geological Zone Code (hangingwall)
GW25	1	greywacke	1	1
SH24	2	shale	1	1
GW24A	3	greywacke	1	1
SHG24A	4	shale/greywacke	1	1
GW24	5	greywacke	1	1
SHGW23	6	shale/greywacke	2	2
GWSH23	7	greywacke/shale	3	3
GW23	8	greywacke	3	3
SH22	9	shale	4	3
T21	10	felsic tuff	4	
SH21	11	shale	4	
T20	12	felsic tuff	4	
SH20	13	shale	4	
GWSH20	14	greywacke/shale	5	
SH19	15	shale	5	
T18	16	felsic tuff	5	
SH18	17	shale	5	
GW18	18	greywacke	5	

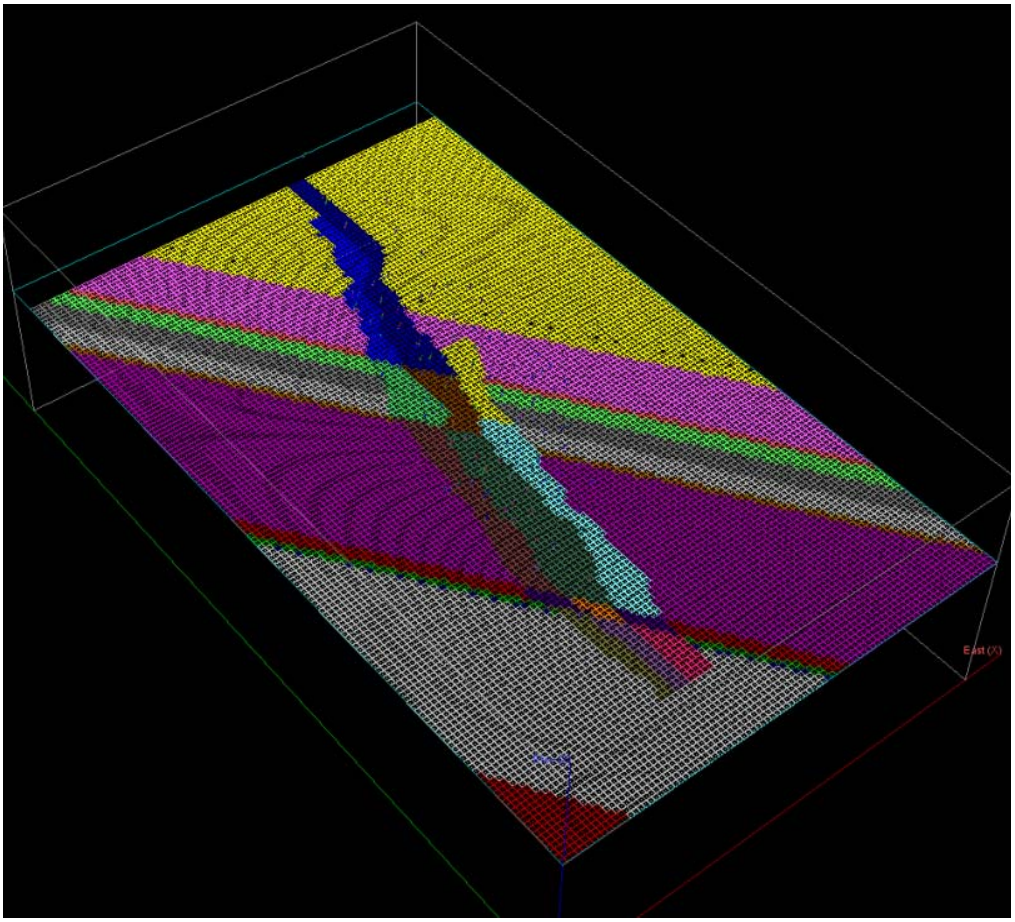


FIGURE 17-2: The 18 "detailed" Complith codes show a general pattern.

North-South Trending Corridor

The north-south trending mineralization occurs in all rock units. Inspection of grade control and exploration data, drill logs, diamond core and the pit has shown that the north-south trending mineralization can be divided into three major zones based on veining and jointing intensity. These three zones were given Compzone codes that have values in the ten thousands, i.e. 10000, 20000, and 30000 by GGC, which Gustavson has modified to be 1000, 2000, and 3000, respectively. Outside of the corridor, the code is 0. Note that in FIGURE 17-2, two sets of lithology produce a striped pattern. The first pattern is one of NW-SE, while the other is a pattern of N-S. The major gold mineralization falls within the latter pattern which allows the Complith codes to be simplified and remapped as codes 1 to 5 depending on whether they fall within the Hanging Wall, Core or Footwall Zones. The relationship between the detailed and simplified Complith codes is shown in TABLE 17-6.

Mineralization Zone Coding

The mineralized zones were coded in the drill hole database in field designated as COMPZONE (as in TABLE 17-7). Gustavson changed the GGC codes to 1000 series numbers in order to be compatible with the GEMCOM software.

TABLE 17-7 VISTA GOLD CORP. – MT TODD GOLD PROJECT Mineralized Zone Model Codes – Batman Deposit December 2006		
Zone	**GGC ID**	**GUSTAVSON ID**
CORE	10000	1000
COREHW	30000	3000
COREFW	2000	2000
OUTSIDECORE	0	0

Rock Type Model

A block model was created for oxidation, lithology and mineralized zones. The blocks were coded by intersecting with solids. A minimum of 51% intersection with the solids was the block coding criteria.

Density Model

The density model was coded from the oxidation model. Blocks were coded on the specific gravity given to the solids as presented in TABLE 17-5.

17.6 MINERAL RESOURCE ESTIMATE

A Mt Todd gold resource estimate was independently developed by Gustavson using the GEMCOM software package. The Gustavson study did not use any new drilling and assay data, as none exists, but carefully reviewed and re-interpreted existing data that were generated by previous studies. Many fundamental model parameters including: topography, drill assay and composite location, and the rock model were found to be acceptable. This included the interpretation by GGC of the lithologic rock designation, the oxidation level and the type and location of mineralized zones. The three-part designation of lithology, oxidation and zone (LOZ) was incorporated into a block model framework used by GEMCOM software. Crucial differences between the GGC and Gustavson resource classification are found within the interpretation of variogram and kriging parameters. Gold values follow a three-parameter lognormal distribution and were modelled with general relative (genRel) variograms. This interpretation resulted in a considerable shortening of variogram ranges used in the ordinary kriging (OK) of gold values. This observation that the variogram ranges are short (rarely longer than 50 meters) has been observed by other consultants (PAH, Snowden). The GGC model has no copper assays results. This is a crucial shortcoming in that the existence of copper appears to have had a negative impact on the earlier Pegasus Mt Todd operation. A copper to gold regression relationship was developed from a nearby mineralized deposit (Quigleys). The resultant block kriging gold estimation errors were in turn used to classify estimated blocks into measured, indicated and inferred categorizations. Until existing core from Mt Todd is re-assayed for copper, the regression relationship will be used for imputing inferred copper values. Finally, a sampling program was designed to efficiently upgrade the gold indicated and inferred blocks to a measured class.

In conclusion, Gustavson's interpretation of the data had impacts on the resource estimation. They are:

1. Gold values follow a three-parameter lognormal distribution, which was modelled with general relative (genRel) variograms.
2. Gustavson geostatistical interpretation of these variograms resulted in a considerable shortening of ranges (1/3 in most case) when compared to GGC.
3. Gustavson used a maximum of 12 samples in an octant search versus GGC's 30 samples.
4. Gustavson variogram models were also simplified, utilizing a nugget and single spherical model as compared to the multiple nested structures (up to 10) proposed by GGC.
5. Gustavson used kriging variance for determining whether a kriged block falls within a measured, indicated or inferred resource class. GGC used a resource classification solely on distance and number of drill holes used in the estimation.
6. The previous block model by GGC did not have copper estimates. This deficiency goes back to a failure to analyze for copper. An imputed inferred copper value was included in the Gustavson block model. The copper values were estimated by a regression relationship derived from the nearby Quigleys Deposit. These copper values will not be used in any resource tabulation.
7. Jackknife calculations were used to validate the inferred classification scheme.
8. Kriging variance was used to propose future sampling locations.

Model Dimensions

TABLE 17-8 provides the details associated with the Batman block model. It is important to note that the elevation (z) data have had a constant 1000 meters added to them in order to avoid having negative (below sea level) elevation values, consequently, these values appear with an "RL" for relative elevation.

TABLE 17-8
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Block Model Physical Parameters – Batman Deposit
December 2006

Direction	Minimum	Maximum	Block size	#Blocks
x-dir	7996mE	9112mE	12m	93
y-dir	9196mN	10984mN	12m	149
z-dir	1224mRL	612mRL	12m	51

Geostatistics of the Batman Deposit

The drilled resource forming the Batman Deposit is situated within a rectangular area approximately 1,500 meters N-S, and 1,125 meters E-W (FIGURE 17-1). The geology of the Batman Deposit consists of a sequence of hornfelsed interbedded greywackes, and shales with minor thin beds of felsic tuffs. Minor lamprophyre dykes trending north-south crosscut the bedding. The major "mineralized package" consists of a tabular deposit striking at 325^o with a dip of 40^o to 60^o to the southeast. The majority of drilling slants at a dip of approximately 65^o with an azimuth of 270^o. FIGURE 17-3 shows a 3-D image of the Batman Deposit, with topography shown in green, drill holes shown in red (gold assay >= 0.6 g Au/t) and blue (gold assays < 0.6 g Au/t). The Batman mine is now an abandoned open pit resulting in drill hole traces that are shown above the present topography.

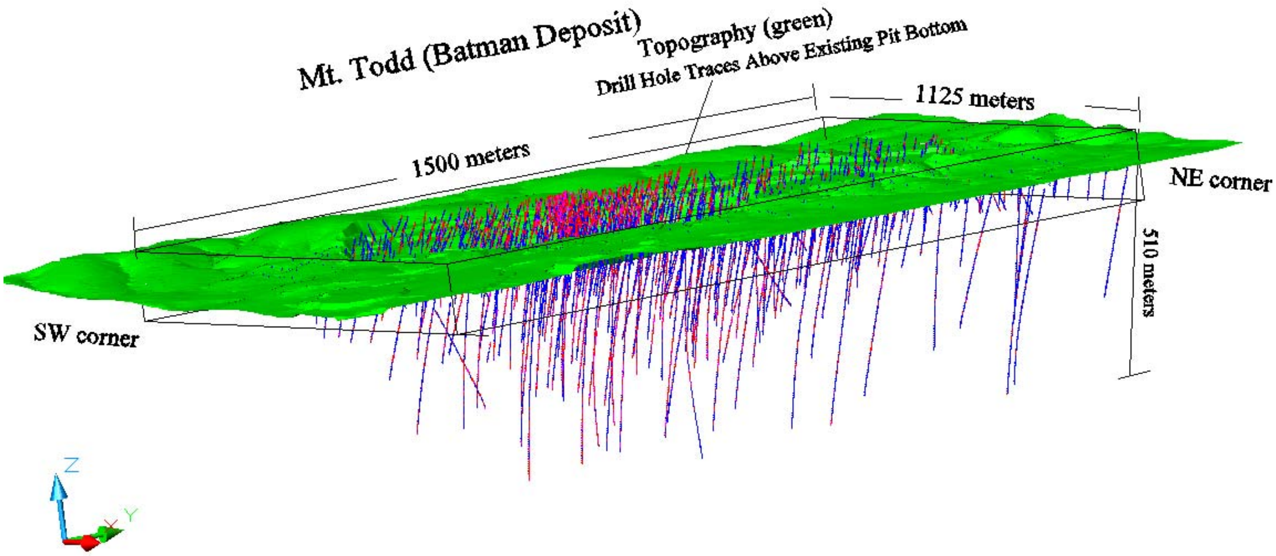


Figure 17-3: Key Map of the Batman Deposit

Bedding parallel shears are present in some of the shale horizons (especially in lithologic units SHGW23, GWSH23 and SH22). These bedding shears are identified by quartz/ calcite sulfidic breccias. Pyrite, pyrrhotite, chalcoprite, 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.

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 bedding and the east-west trending faults and joints.

There have been several previous resource studies, with the most recent being GGC in the year 2000, Snowden in 1997, PAH, in 1995 and Pegasus (MRT) also in 1995. The PAH study was a Due Diligence Review of Mt Todd. In their report, PAH recommend that the down-hole assay data be composited to four meters. PAH examined the variogram interpretations by Pegasus for their Mt Todd Feasibility Study. PAH concluded that Pegasus (MRT) interpreted variogram ranges to be too long and the variogram nugget effect was too low. The report commented:

" … the Feasibility Study…(had) … interpreted a second structure to the variogram data that extends the range significantly beyond the more obvious range, working on a portion of the structure that represents ten percent of the total sill (PAH 1995).

The present review by Gustavson finds that GGC's variogram modelling has the same problems. GGC's variograms were modelled with multiple nested structures, all within the last 10% of the sill. While GGC appears to have followed PAH recommendation in using log-variography, this may have contributed to GGC overestimating the variogram ranges, which in turn were used to specify overly optimistic search ellipsoids sizes, which were used in kriging. A final PAH recommendation that indicator kriging be used was explored by GGC. In the end, GGC used ordinary kriging (OK) with log-variograms on the exploration data for their kriged block model. Gustavson has also used ordinary kriging for the development of our independent resource estimate for the Batman Deposit.

GGC compared mineralized zones with each other and with the simplified lithological zones. The mineralized zones showed a straight-line graph (i.e., lognormal distribution), with hangingwall and footwall having lower grades than the core zone, suggesting that the sample populations are similar with a lower mean. For this reason, GGC decided to do the variography on the lithology zone (apart from Zones 3 and 4 in the footwall and core zones). GGC separated the sample codes and block model codes into ten zones for variography and interpolation (TABLE 17-9). The Gustavson study focused primarily on Zones 1, 2, 3, 3a and 5 for detailed review.

TABLE 17-9 VISTA GOLD CORP. – MT TODD GOLD PROJECT Variography And Interpolation Domains – Batman Deposit December 2006			
zone	**description**	**Block model codes**	**Sample codes**
Os1	Outside mineralized zones in lithological zone 1	1,2,3,4,5	1,
Os2	Outside mineralized zones in lithological zone 2	6,	2,
Os3	Outside mineralized zones in lithological zone 3	7,8	3,
Os4	Outside mineralized zones in lithological zone 4	9, 10, 11, 12, 13, 3010, 3011, 3012, 3013	4,
Os5	Outside mineralized zones in lithological zone 5	14, 15, 16, 17, 18, 3014, 3015, 3016 , 3017, 3018	5,
Zone1	Mineralized zones in lithological zone 1	1001, 1002, 1003, 1004, 1005, 2001, 2002, 2003, 2004, 2005, 3001, 3002, 3003, 3004, 3005	10001, 20001, 30001
Zone2	Mineralized zones in lithological zone 2	1006, 2006, 3006	10002, 20002, 30002
Zone3	Mineralized zones apart form hangingwall in lithological zones 3 and 4	1007, 1008, 1009, 1010, 1011, 1012, 1013, 2007, 2008, 2009, 2010, 2011, 2012, 2013	10003, 10004, 20003, 20004
Zone3a	Hangingwall zone in lithological zone 3	3007, 3008, 3009	30003,
Zone5	Mineralized zones apart form hangingwall in lithological zones 5	1014, 1015, 1016, 1017, 1018, 2014, 2015, 2016, 2017, 2018	10005, 20005

The statistics for the ten Interpolation Domains are shown in TABLE 17-10. These ten domains were further studied with variogram analysis. GGC used a computer program called Visor to analyze the variograms. Gustavson broke out the statistical analysis using different codes based on the previously discussed LOZ coding. It was determined that there are 96 possible combinations of Complith (6 = Codes 0, 1, 2, 3, 4, 5), Compox (4= Codes 0, 100, 200, 300) and Compzone (4= codes 0, 1000, 2000, 3000).

TABLE 17-10 VISTA GOLD CORP. – MT TODD GOLD PROJECT Summary Normal Statistics – Interpolation Domains – GGC Model December 2006										
Zone	**Os1**	**Os2**	**Os3**	**Os4**	**Os5**	**Zone1**	**Zone2**	**Zone3**	**Zone3a**	**Zone5**
Count	296	2064	837	1703	5346	1353	7450	650	294	2175
Maximum (g Au/t)	1.868	6.628	6.22	10.59	7.27	6.75	15.37	7.88	5.33	7.88
Minimum (g Au/t)	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005
Mean (g Au/t)	0.098	0.247	0.297	0.453	0.32	0.436	0.866	0.796	0.526	0.593
Median g Au/t)	0.05	0.155	0.175	0.265	0.195	0.294	0.673	0.592	0.323	0.429
Std dev	0.16	0.365	0.44	0.685	0.436	0.543	0.865	0.815	0.664	0.633
Coeff var	1.66	1.478	1.48	1.513	1.361	1.208	0.96	1.024	1.262	1.067
97.5 %tile	0.9	0.98	1.1	1.95	1.38	1.77	3.04	2.72	2.33	2.35
99 %tile	1.7	1.43	1.75	3.94	2.09	2.77	4.1	4.05	3.74	3.22

The spread of gold grades at each Complith (Lith designation) is shown in FIGURE 17-4. The small box encloses the mean, and the larger box all values within one standard error of the mean. The "whiskers" represent composites with a range of values that go from 1.96 times the standard error (SE). Note that the highest mean grade is within Complith Code 2, which has an average value above 0.70 g Au/t.

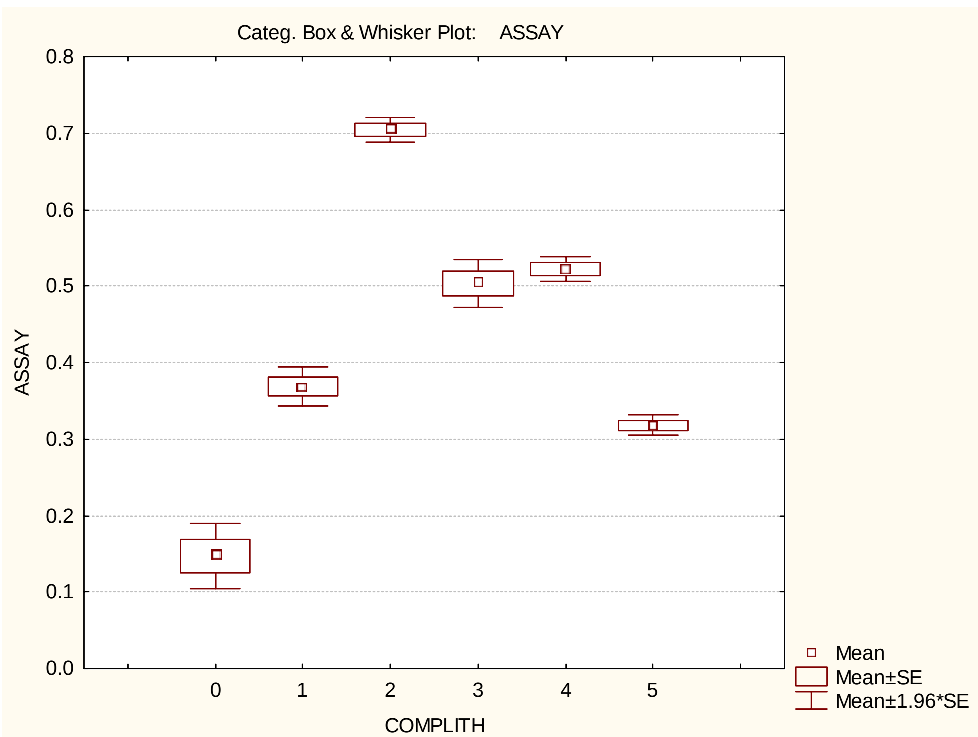


FIGURE 17-4: Box and Whisker Plot of Assay versus Complith (Gustavson Analysis)

A more detailed box-and-whisker plot breaks out gold grades by Complith Codes 1 through 5 and by Zone codes 0 and 10000 through 30000. Note that the highest grades are again in Complith 2, with zone 10000 having averages approaching 1.2 g Au/t.

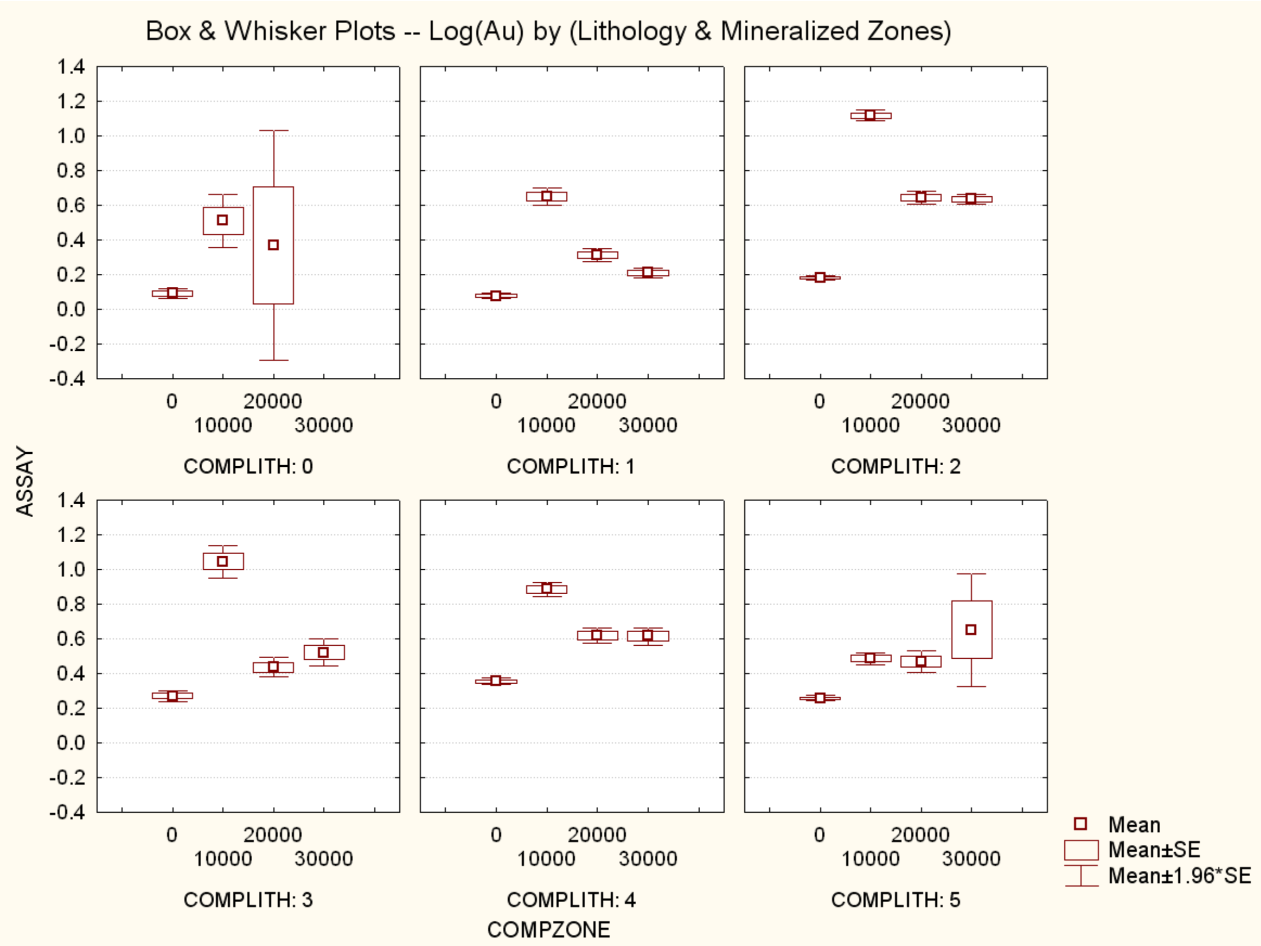


FIGURE 17-5: Box and Whisker Plot of Assay versus Complith, and Zone (Gustavson Analysis)

Looking north in FIGURE 17-6, shows the majority of the drill holes slanting approximately 65^{o} to the west. In the following variogram analysis, drill data that are in areas that have been mined are not discarded. The information of the missing data is still useful in producing spatial statistics for application to the remaining mineralization.

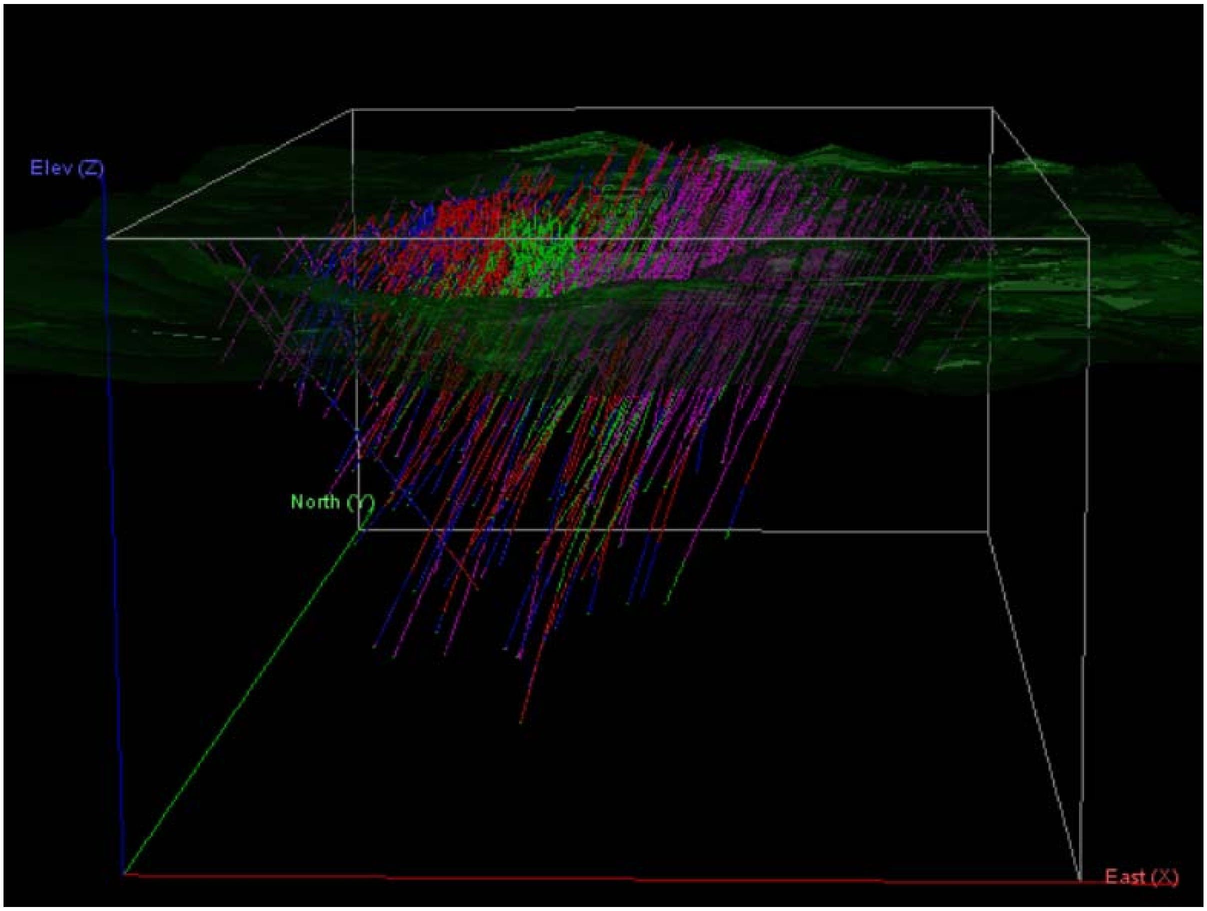


FIGURE 17-6: Drilling pattern, looking north shown with GEMCOM software.

TABLE 17-11 contains ten LOZ codes that contain more than 70% of gold content of the Batman Deposit. In fact, the LOZ code 10302 contains over 20 percent of the total gold content. Within this particular LOZ code, the highest gold value of 15.373 g Au/t was analyzed.

TABLE 17-11
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Top 10 Groups of LOZ with Greater than 70% of Gold Content – Batman Deposit
December 2006

Rank	Lith Code	Ox Code	Zone	Assay Mean g Au/t	No. Assay	proxy metal content g*t	% of Total gold	Assay Std. (g Au/t)	Assay Min. (g Au/t)	Assay Max. (g Au/t)	Assay Q25 (g Au/t)	Assay Median (g Au/t)	Assay Q75 (g Au/t	%tile	%tile
									.	.				95	99
1	2	300	1000	1.223	2169	2653.1	20.10%	1.022	0.005	15.373	0.578	0.973	1.573	3.01	4.9
2	4	300	1000	0.919	1266	1163.7	8.80%	0.823	0.045	6.818	0.42	0.665	1.1	2.598	3.888
3	2	100	1000	1.117	889	993.3	7.50%	0.723	0.015	7.105	0.635	0.95	1.48	2.38	3.665
4	4	300	0	0.369	2670	984.6	7.50%	0.568	0.005	10.595	0.085	0.215	0.438	1.195	2.833
5	2	300	3000	0.678	1235	837	6.30%	0.736	0.005	6.725	0.203	0.45	0.885	2.13	3.49
6	2	300	2000	0.608	1182	718.3	5.40%	0.782	0.005	12.115	0.16	0.399	0.768	1.905	3.85
7	4	300	2000	0.648	755	489.1	3.70%	0.692	0.015	7.78	0.238	0.448	0.808	1.905	3.503
8	2	200	1000	1.121	334	374.3	2.80%	0.72	0.009	4.225	0.62	0.985	1.475	2.605	3.517
9	3	300	1000	1.13	313	353.6	2.70%	0.958	0.115	7.88	0.578	0.875	1.35	2.648	4.548
10	2	100	2000	0.711	475	337.5	2.60%	0.606	0.01	4.363	0.303	0.57	0.92	1.953	2.958
			Top 10 Groups	0.789	11288	8904.6	71.70%			15.373					
		All Groups		0.581	22709	13183.1	100.00%	0.732	0.005	15.373	0.133	0.348	0.76	1.91	3.517
		notes:					top 10					>10 Au g Au/t			
							% metal								

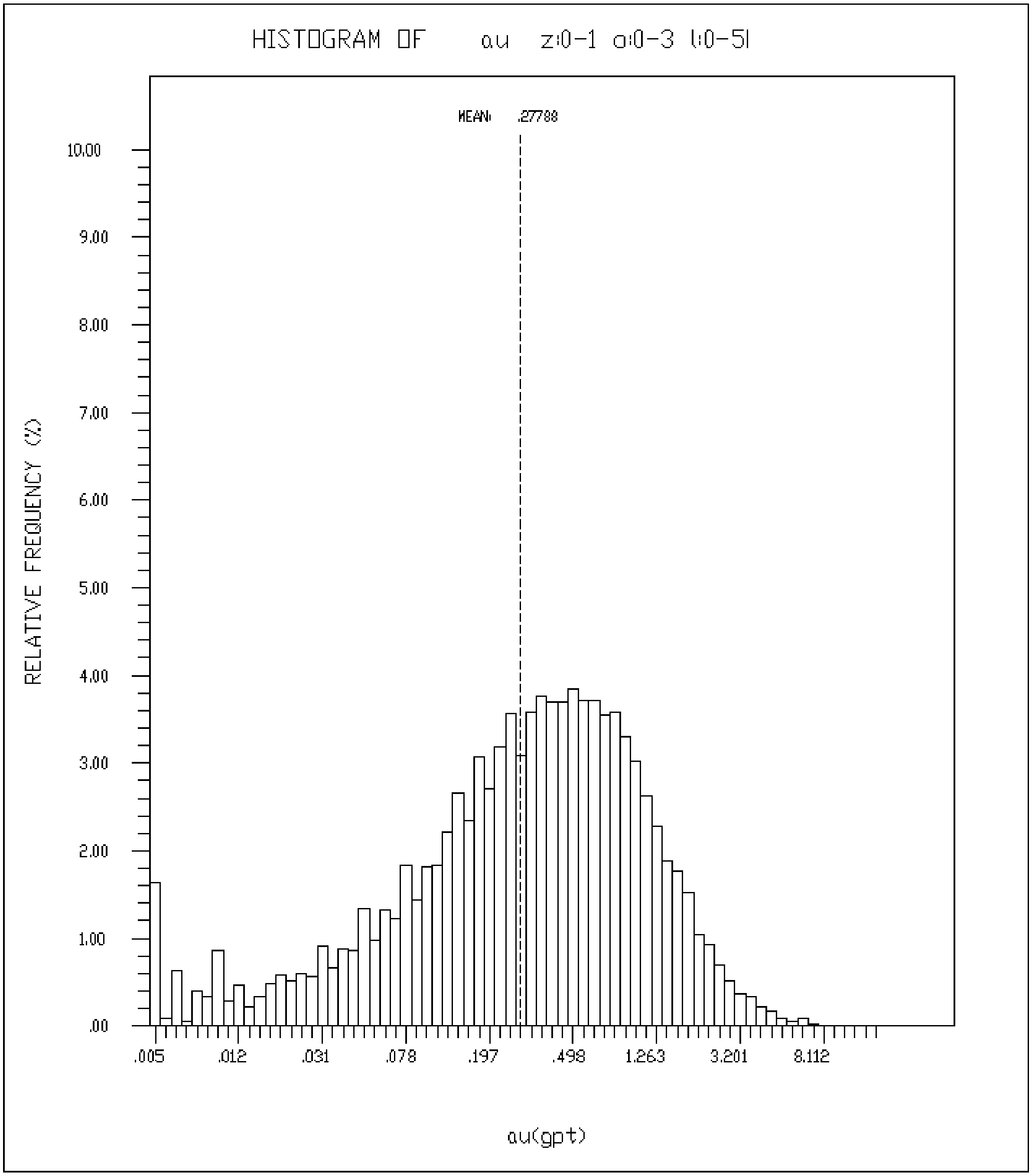


FIGURE 17-7: Histogram of Au g Au/t for all LOZ.

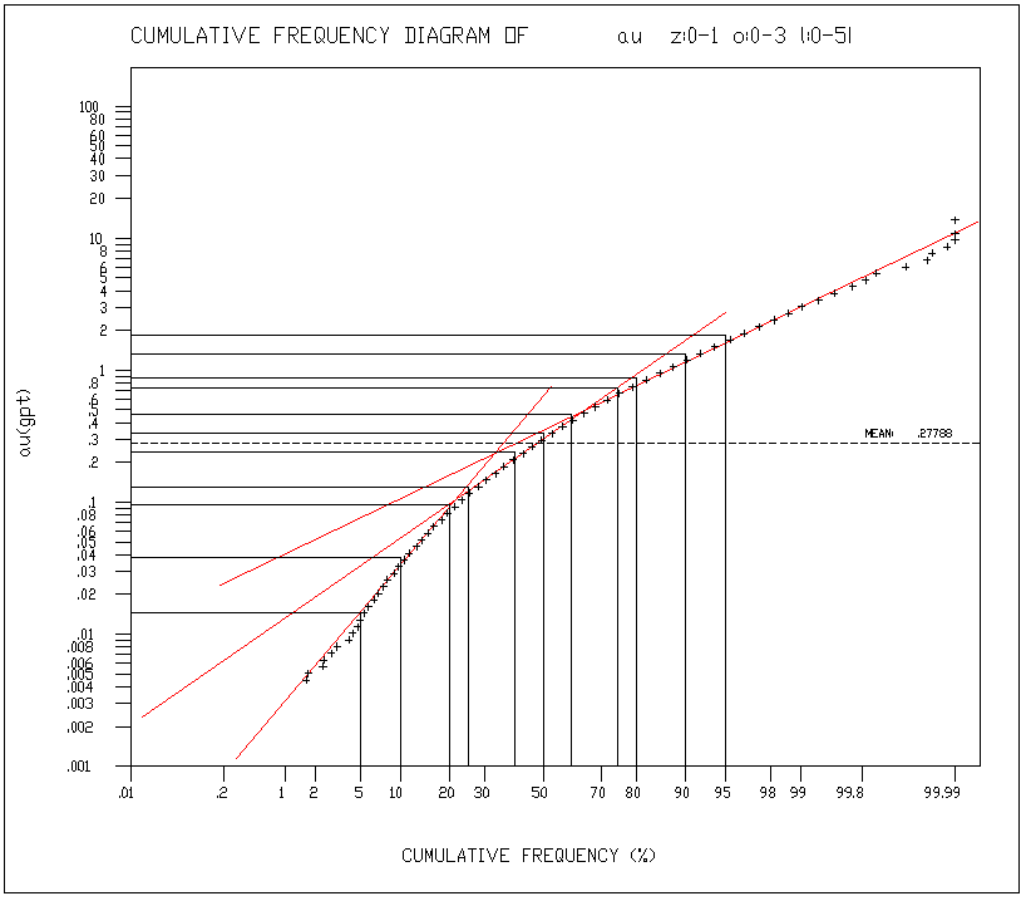


FIGURE 17-8: Cumulative probability plot of gold grades in grams per ton for all LOZ.

FIGURE 17-7 shows a histogram of g Au/t for all LOZ classes. The height of the vertical bars charts the relative frequency (y-axis) of composites falling within grade classes (x-axis). Note that the grade classes (bins) are log scaled. FIGURE 17-8 charts the same data on a log-probability plot. This is a specialized form of a cumulative frequency plot such that a lognormal distribution will plot as a straight line. A break from a normal curve occurs around 0.1 g Au/t. The gentle flexure of the curve exists above 0.1 Au g Au/t. A second break point has been modelled at 0.5 g Au/t. TABLE 17-12 lists the statistics of the curve, with 5% of the gold is below 0.15 g Au/t, 20% below 0.095 g Au/t and 95% below 1.83 g Au/t. FIGURE 17-9 shows the cumulative probability plot of a three-parameter lognormal model, with 0.1 g Au/t as the third parameter. Note that the curve is essentially a straight line, implying a single mode, lognormal distribution.

TABLE 17-12
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Statistics All LOZ – Batman Deposit
December 2006

```
mt-todd au z:0-1 o:0-3 l:0-5|
au  z:0-1 o:0-3 l:0-5|
Limits on the variable  : ** NONE **
Limits on the data axis : -.5403678E+01  .2732580E+01
Limits on the freq axis :  .0000000E+00  .3848694E+01

SAMPLES DISTRIBUTION INFORMATIONS
Number of samples      :        22709
Samples under minimum  :            0
Samples over maximum   :            0
Missing values         :            0
Out by restrictions    :            0
Out by logarithm       :            0
Minimum                :              .00450
Percentile   5%        :              .01463
            10%        :              .03848
            20%        :              .09573
            25%        :              .12877
            40%        :              .23978
            50%        :              .33697
            60%        :              .46070
            75%        :              .73707
            80%        :              .86924
            90%        :             1.31928
            95%        :             1.83121
Maximum                :            15.37250
```

```
STATISTICS INFORMATION                      LOGARITHMIC STATISTICS

Samples kept  :     22709                   Samples kept  :     22709

Median        :              .33697         Median        :            -1.08775
Average       :              .58052         Average       :            -1.28055
Mode          :              .49844         Mode          :             -.69627
Variance      :              .53522         Variance      :             2.01357
Std deviation :              .73159         Std deviation :             1.41900
Coefficient                                 Coefficient
of variation  :             1.26021         of variation  :            -1.10810
Skewness      :             3.69785         Skewness      :             -.77127
Kurtosis      :            30.47623         Kurtosis      :             3.42226
```

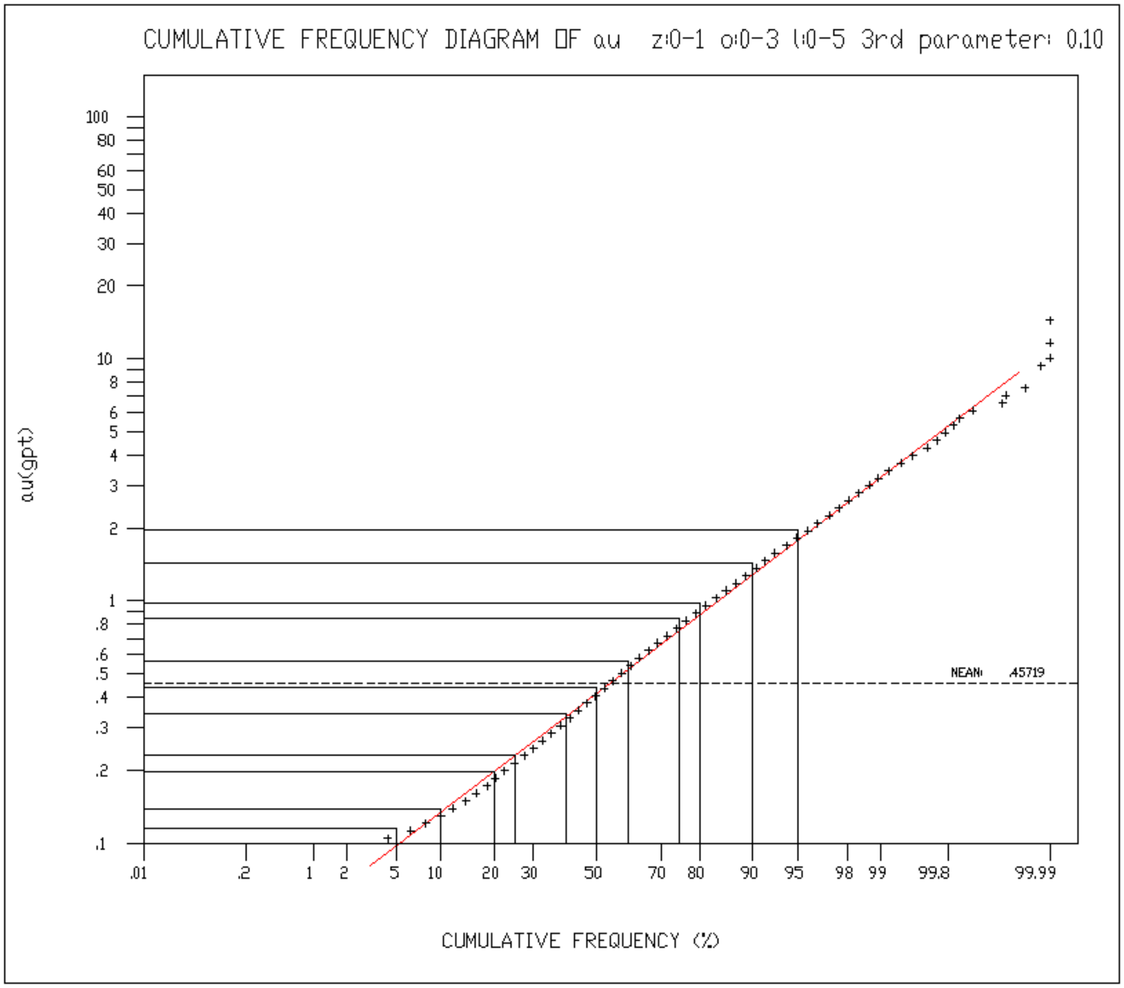


FIGURE 17-9: Cumulative Frequency Diagram of 3-parameter lognormal distribution for gold with all ZOL composites.

Estimation of Batman Copper Relationship from the Quigleys Database

The Quiqleys deposit is approximately 4000 meters northeast of the Batman Deposit. The GGC model contained very few (approximately 282) copper assays. There were insufficient data pairs at Batman to determine a definitive relationship. This is a crucial shortcoming in that the existence of copper appears to have had a negative impact on the earlier Pegasus Mt Todd operation. A copper to gold regression relationship was therefore developed from the nearby Quigleys mineralized deposit. Until existing core from Mt Todd is re-assayed for copper, this regression relationship will be used for imputing inferred copper values. These copper values will be used for in-house studies and will not be reported within the resource statement.

Since this is the only copper data available, Gustavson has assumed that the Quigleys gold to copper relationship may be similar to Batman. Within the Quigleys Deposit there are 631 drill holes, which contain 49,178 gold assays of which the majority is 1 meter long. Nearly every gold assay has a corresponding copper assay. In total, the Quigleys database contains 41,671 Au/Cu assay pairs.

FIGURE 17-10 is a scatter plot of copper versus gold assays from the Quigleys Deposit.

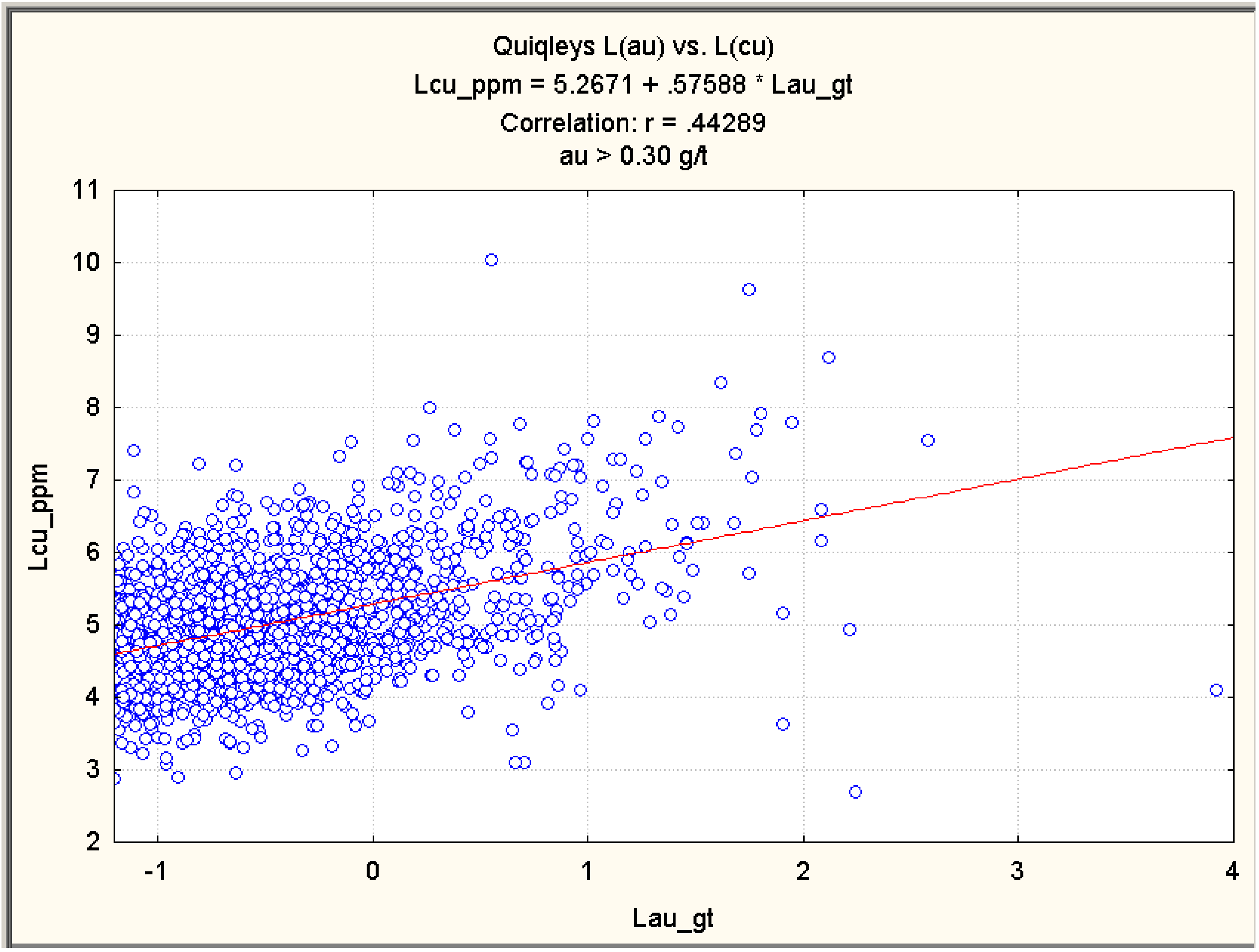


FIGURE 17-10: X-Y Scatter plot of gold and copper assays, with a regression line predicting copper from gold grades greater than 0.30 g Au/t.

The regression formula proposes a linear relationship when copper and gold are plotted on a log-log graph. There are two formats shown. The first relates gold in grams per tonne to copper in parts per million. The second form relates gold grams per tonne with percent copper.

$$\ln(cu\_ppm) = 5.2671 + 0.5759 \times \ln(au\_gt)$$

$$or:$$

$$\ln(cu\_\%) = -3.92 + 0.5759 \times \ln(au\_gt)$$

Using the second formula, if the gold grade is 1.0 g Au/t, then copper has an imputed value of 0.02 percent. The regression formula was used to create a copper resource model. The resources associated with this model are not reported in this report, as they do not meet any of the criteria even for inferred resources. The copper model has been produced merely to assist in the evaluation of potential copper that may follow the gold should Vista decide to complete scoping or other studies.

Variography

Geostat Systems® Vario3™ was used to calculate 3-D variograms. Vario3 uses the Marecx method of defining variogram bins. The program breaks the Z direction into "elevation" layers and then calculates variograms within chosen direction and angle bins. By swapping the Z coordinate, 3-D variograms can be explored. This method produces variograms sets called XYZ, XZY and YZX. Each of these three sets looks in five directions, 0^{o}, 45^{o}, 90^{o}, 135^{o} and omni-directional. FIGURE 17-11 shows the three sets for the LOZ 20302 coded unit for the General Relative variograms. Note that the modelled nugget is quite high relative to the sill, and that the ranges go from 20 to 45 meters.

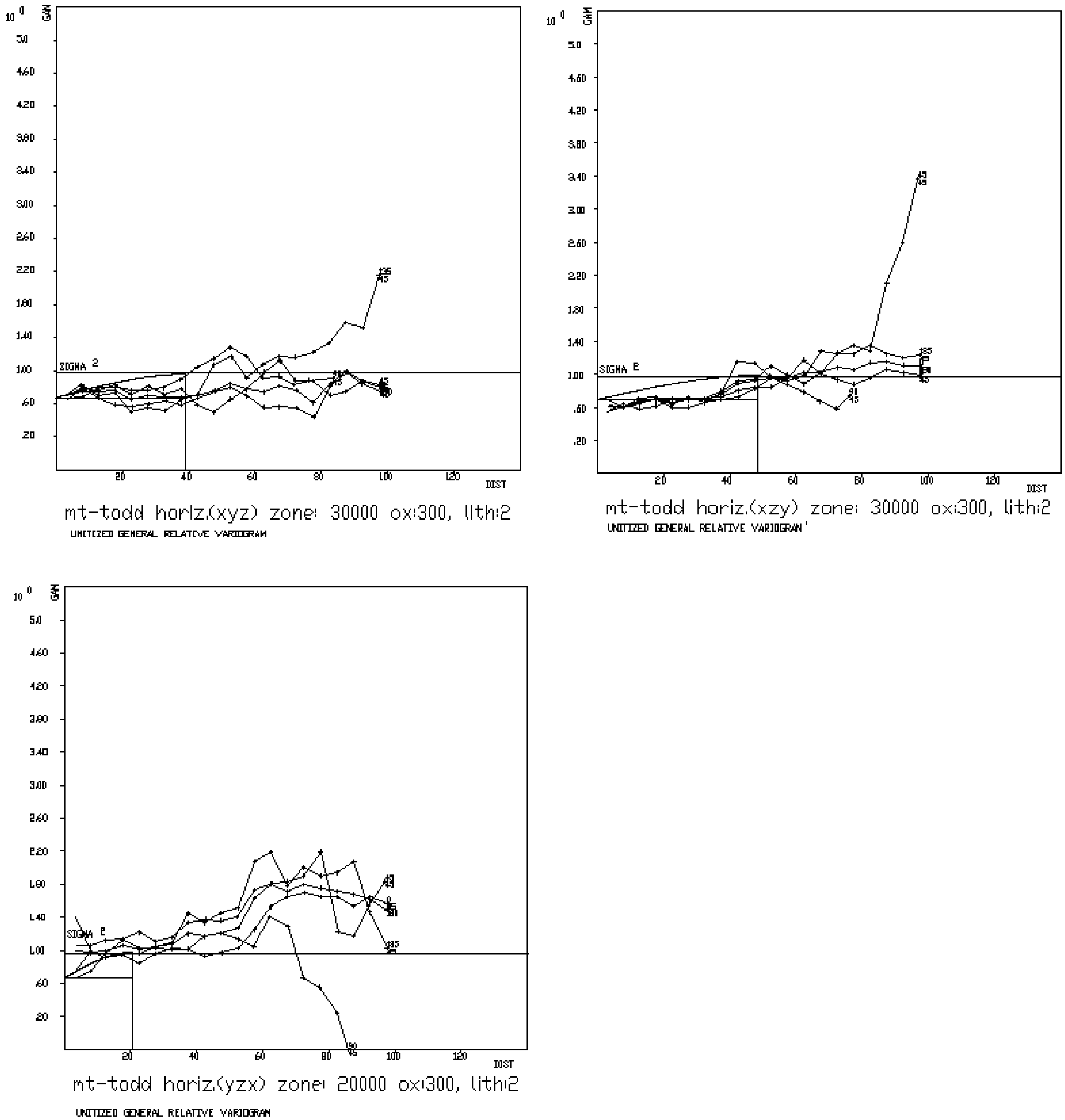


FIGURE 17-11: General Relative variograms looking in the 3-D

FIGURE 17-12 is an extract of the printer listing from Vario3. It shows the General Relative and Logarithmic variograms in the 0 degree directions, with a 45 degree angular window for YZX set. The nugget for the General Relative is 60% of the sill, and the range is 50 meters. The nugget for the log variogram has a nugget of 50% and a range exceeding 100 meters. It appears that GGC modelled log variograms. As seen in this example, log variograms tend to underestimate nuggets and overestimate ranges.

```
                          mt-todd horiz.(yzx) zone: 30000 ox:300, lith:2
                                 GENERAL RELATIVE VARIOGRAM
 VARIABLE (au  ) +++ ANGULAR DIRECTION (    .0) +++ ANGULAR WINDOW (  45.0)

 2.120     I
 2.032     I
 1.944     I
 1.855     I
 1.767     I
 1.679     I
 1.590     I
 1.502     I
 1.414     I
 1.325     I                                           *  *
 1.237     I                                       *          *      *  *
 1.149     I--------------------------------*---*----------------*----------*--*--
 1.060     I                  *   *  *   *
 .9718     I    *   *  *   *
 .8835     I *
 .7951     I
 .7068     I
 .6184     I
 .5301     I
 .4417     I
 .3534     I
 .2650     I
 .1767     I
8.8348E-02 I
 0.0       +---------+---------+---------+---------+---------+---------+---------+
           .0000     14.29     28.57     42.86     57.14     71.43     85.71     100.0
                                                           AVERAGE DISTANCE

                           mt-todd horiz.(yzx) zone: 30000 ox:300, lith:2
                                   LOGARITHMIC      VARIOGRAM
 VARIABLE (au  ) +++ ANGULAR DIRECTION (    .0) +++ ANGULAR WINDOW (  45.0)

 2.083     I
 1.983     I
 1.884     I
 1.785     I
 1.686     I
 1.587     I
 1.488     I
 1.388     I
 1.289     I----------------------------------------------------------------------
 1.190     I                                                  *         *
 1.091     I                                    *  *   *  *      *   *      *  *
 .9917     I        *  *   *  *   *  *   *  *
 .8926     I    *
 .7934     I *
 .6942     I
 .5950     I
 .4959     I
 .3967     I
 .2975     I
 .1983     I
```

FIGURE 17-12: Printer output of General Relative and Logarithmic variograms for LOZ 30302.

TABLE 17-13 contains the variogram parameters written in red, which were modelled by Gustavson. In general, the variograms ranges used by GGC are almost three times longer than those modelled by Gustavson. Also, the Gustavson modelled variogram structures as a simple nugget of 60% of the sill and single spherical curve, while GGC tended to have multiple nested variogram structures in the last 10% of the sill. GGC utilized Visor, an automatic variogram modeler. It is noted by GGC that GEMCOM has a limit of 8 variogram structures. At times Visor defined up to 10 structures. The GGC variogram range issue becomes doubly important in that it is used to specify the search ellipse. GGC's search parameters are typed in black and Gustavson in red.

Octant Search, Target Codes and Ellipsoids

GGC used a minimum of 4 sample points and a maximum of 30 sample points for kriging within an ellipsoid search. Gustavson used a minimum of 3 sample points and a maximum of 12 sample points. Gustavson, as well as other independent consultants, believes that the maximum of 30 points is "over-smoothing" the grade model and providing an inaccurate picture of the actual distribution and tenor of the mineralization.

TABLE 17-13 also contains the target code for blocks and the required LOZ code for composites for each interpolation zone. For example, a zone3ok has block codes 1007, 2007,1008, 2008, 1009, 2009, 1010, 2010,1011, 1011, 1012, 2012, 1013, 2013, 3007, 3008 or 3009. Only Composites with LOZ codes 10002, 20002, 10001, 20001, 30001, 10003, 20003, 30004, 10004 or 20004 that fall within the search ellipse and meet the octant search criteria can be to estimate the block. The codes in red were found missing from GGC's technical write-up of their kriging procedure. FIGURE 17-13 is a GEMCOM generated "photo" that illustrates the matching that takes place between the drill hole composites and the block model.

Note that GEMCOM uses ZYZ rotation method to specify 3-D orientation of both variogram anisotropy and search ellipsoids. FIGURE 17-14 shows a set of rectangular boxes that would contain search or anisotropy ellipsoids. Generalized size and orientation of search ellipsoids are shown from various 3-D views—the large rectangular box would enclose GGC and the larger ellipsoid used to estimate inferred blocks. The small rectangular box is 1/3 the larger one's size. It would enclose the search ellipsoid that is used to estimate measured and indicated blocks. The cube represents the scale and orientation of a 12x12x12 meter mining block. The line intersecting large block illustrates general drill-hole direction. (Left panel is a SW view; top right panel is a NE view and bottom right panel is a top view.)

TABLE 17-13
VISTA GOLD CORP. – MT TODD GOLD PROJECT
GGC versus Gustavson Search and Variogram Parameters – Batman Deposit
December 2006

Kriging Profile	Block Model target rock codes	Composite File sample rock codes	Zrot	Yrot	Zrot	r1 (m)	r2 (m)	r3 (m)	Co	C1
os1ok	1, 2, 3, 4, 5	1, 2, 30001, 20001	165	85	-5	**50** **168**	**20** **63**	**10** **18**	**0.60**	**0.40**
os2ok	6	1, 2, 3, 30002, 20002	170	105	-30	**50** ***228***	**20** **45**	**10** **29**	**0.60**	**0.40**
os3ok	7,8	2, 3, 4, 30003, 20003	-10	90	-20	44	22	18	**0.60**	**0.40**
os4ok	9, 10, 11, 12, 13, 3010, 3011, 3012, 3013	3, 4 ,5, 30004, 20004	-144	50	-70	46	35	14	**0.60**	**0.40**
os5ok	14,15.16,17,18, 3014, 3015, 3016, 3017, 3018	4, 5, 30005, 20005	175	109	-9	**50** **169**	**25** **64**	**18** **53**	**0.60**	**0.40**
zone1ok	1001,2001,3001,1002,2002, 3002, 1003,2003, 3003,1004,2004, 3004, 1005, 2005, 3005	10001, 20001, 30001,10002, 20002, 30002	**170**	**-80**	**-30**	**50** **121**	**35** **41**	**20** **18**	**0.60**	**0.40**
zone2ok	1006.2006, 3006	10002, 20002, 30002, 10001, 20001, 30001, 10003, 2000, 30003	**170** **165**	**-80** **100**	**-30**	**50** ***128***	**35** ***98***	**20** ***29***	**0.60**	**0.40**
zone3aok	3007, 3008, 3009	10002, 30002, 10004, 30004, 30003, 10003, 3, 4	**170** **50**	**-80** **125**	**-30** **-80**	**50** ***36***	**35** ***30***	**20** **18**	**0.60**	**0.40**
zone3ok	1007, 2007,1008, 2008, 1009, 2009, 1010, 2010,1011, 1011, 1012, 2012, 1013, 2013, 3007, 3008, 3009	10002, 20002, 10001, 20001, 30001, 10003, 20003, 30004, **10004, 20004**	**170** **165**	**-80** **95**	**-30**	**50** **137**	**35** **122**	**20** **38**	**0.60**	**0.40**
zone5ok	1014, 2014, 1015, 2015, 1016, 2016, 1017, 2017, 1018, 2018	10004, 20004, 10005, 20005	**170**	**-80** **100**	**-20**	**50** **156**	**35** **130**	**20** **120**	**0.60**	**0.40**

Note: ZYZ rotation.

Key:

Red: Gustavson search and variogram parameters that are different

Black: GGC search parameters unchanged

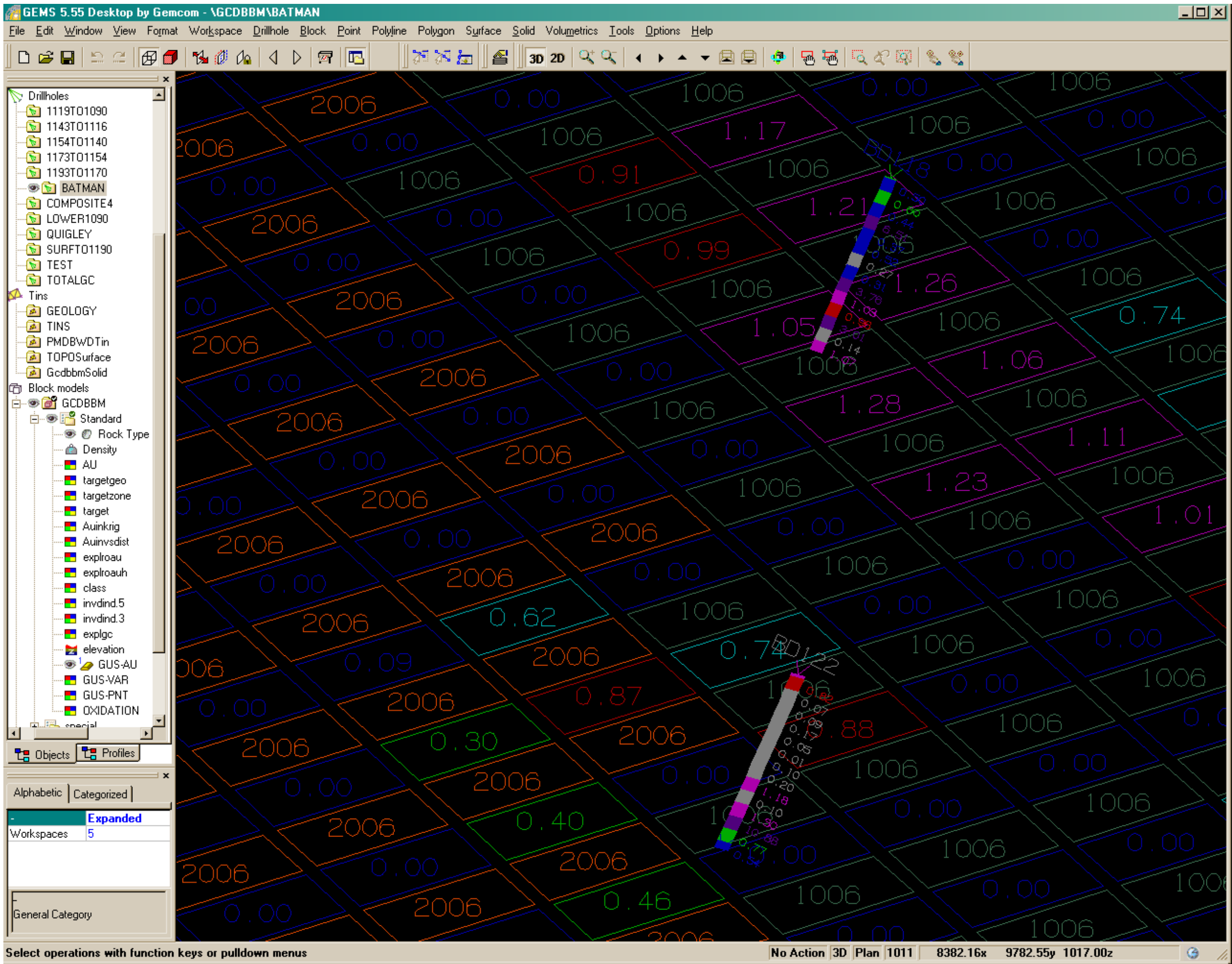


FIGURE 17-13: GEMCOM "photo" showing the process of matching composite codes to block model codes for kriging.

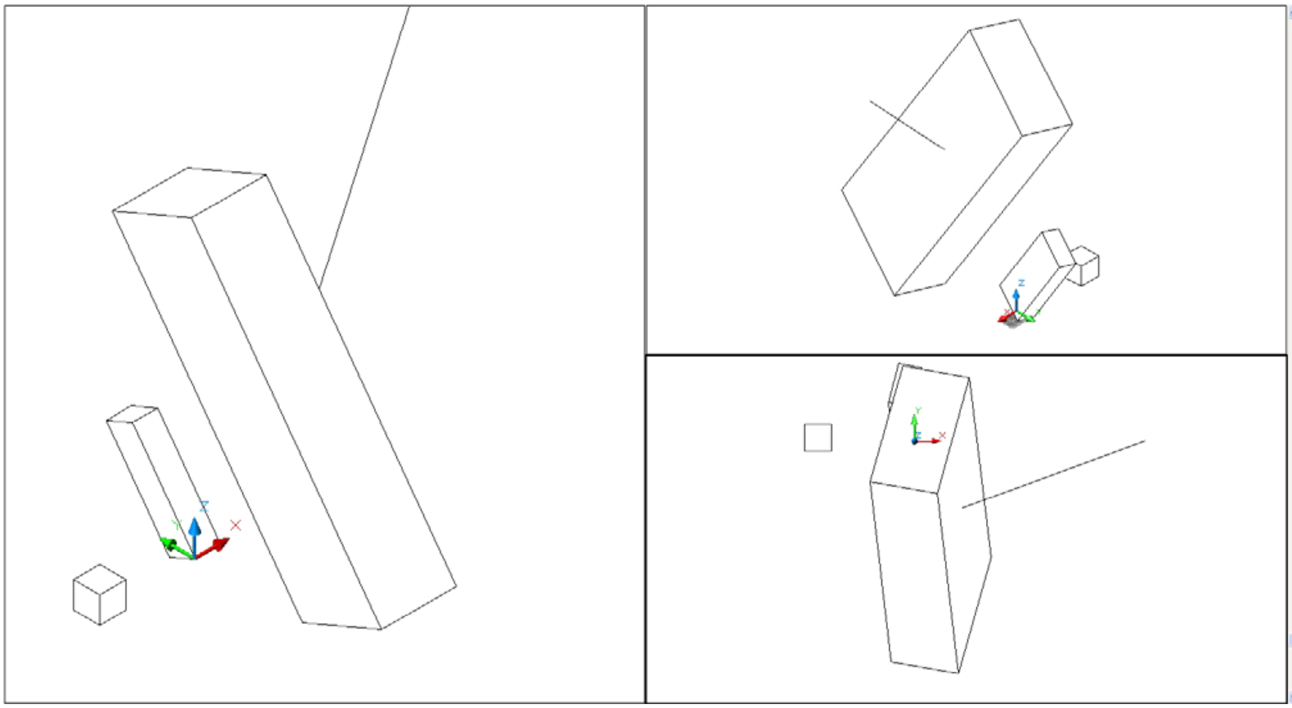


FIGURE 17-14: Generalized size and orientation of search ellipsoids shown from various 3-D views—large rectangular box encloses GGC and GUS-JAL ellipsoid, smaller one encloses GUS-JAS search ellipsoid. Cube is drawn to represent scale and orientation of a 12x12x12 m mining block. Line intersecting large block illustrates general drill-hole direction. (Left panel is a SW view; top right panel is a NE view and bottom right panel is a top view.)

TABLE 17-14 provides a comparison between the GGC and the Gustavson gold grade models and the base data used to create them.

TABLE 17-14 VISTA GOLD CORP. – MT TODD GOLD PROJECT Comparison of GGC and Gustavson Block Models – Batman Deposit December 2006	
GGC. Explroau	OK using exploration data only. -- Long Ranges; multiple structures
GUS-JAL	OK using exploration data only. -- Long Ranges; two structures
GUS-JAS	OK using exploration data only. -- Short Ranges; two structures
GUS-HALO	HALO=JAL-JAS

Additional differences are as follows:

- GGC's Explroau model uses four-meter composited exploration data. It has been used as the template for the two Gustavson models. The first is the GUS-JAS that uses search ellipses specified in TABLE 17-13. During kriging the minimum distance of a valid sample point used in the estimate and the kriging variance were written out to a file. Analysis of the kriging variance using cumulative frequency plots shows a reasonable break at 0.30 krige_var. This kriging variance was chosen as the break between Measured and Indicated resources. Only a small number of blocks are above 0.55 krige_var. Hence the break point of Inferred was found by producing the GUS-JAL model.

- The model GUS-JAL is similar to GGC's model in that the ranges are three times the values shown in TABLE 17-13. FIGURE 17-15 shows the relative difference in search ranges. Once again the minimum distance of a valid sample point used in the estimate and the kriging variance was written out to a file. The break between Measured and Indicated is when the closest sample is 10 meters.

- The final step was to produce a GUS-HALO model by doing a Boolean subtraction of GUS-JAS from GUS-JAL (FIGURE 17-16). This leaves a void where blocks are for the most part measured and indicated. Blocks that remain with a krige_var less than 0.45 krige_var were classified as inferred.

TABLE 17-15 details the differences in the determination of the resource classification between the GGC and the Gustavson grade models. It is important to note that the Gustavson classification uses significantly shorter searches, fewer points, and incorporates the block variances.

TABLE 17-15
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Comparison of GGC and Gustavson Classification Criteria– Batman Deposit
December 2006

Resource Class	GGC Model	Gustavson Model
Measured (Class 30)	Within 25 meters of data point. At least 16 samples used to estimate the block grade. At least 2 two drill holes used to provide data	**Core Model Kriging (JAS) Unitized Relative Variance: < .30**
Indicated (Class 20)	Between 25 and 50 meters of a data point. At least 10 samples used to estimate the block grade.	**Core Model Kriging (JAS) Unitized Relative Variance: >= .30 & < .55**
Inferred (Class 10)	Greater then 50 meters from a data point. At least 4 samples used to estimate the block grade	**Halo Model Kriging (JAL minus JAS) Unitized Relative Variance: <= 0.45**

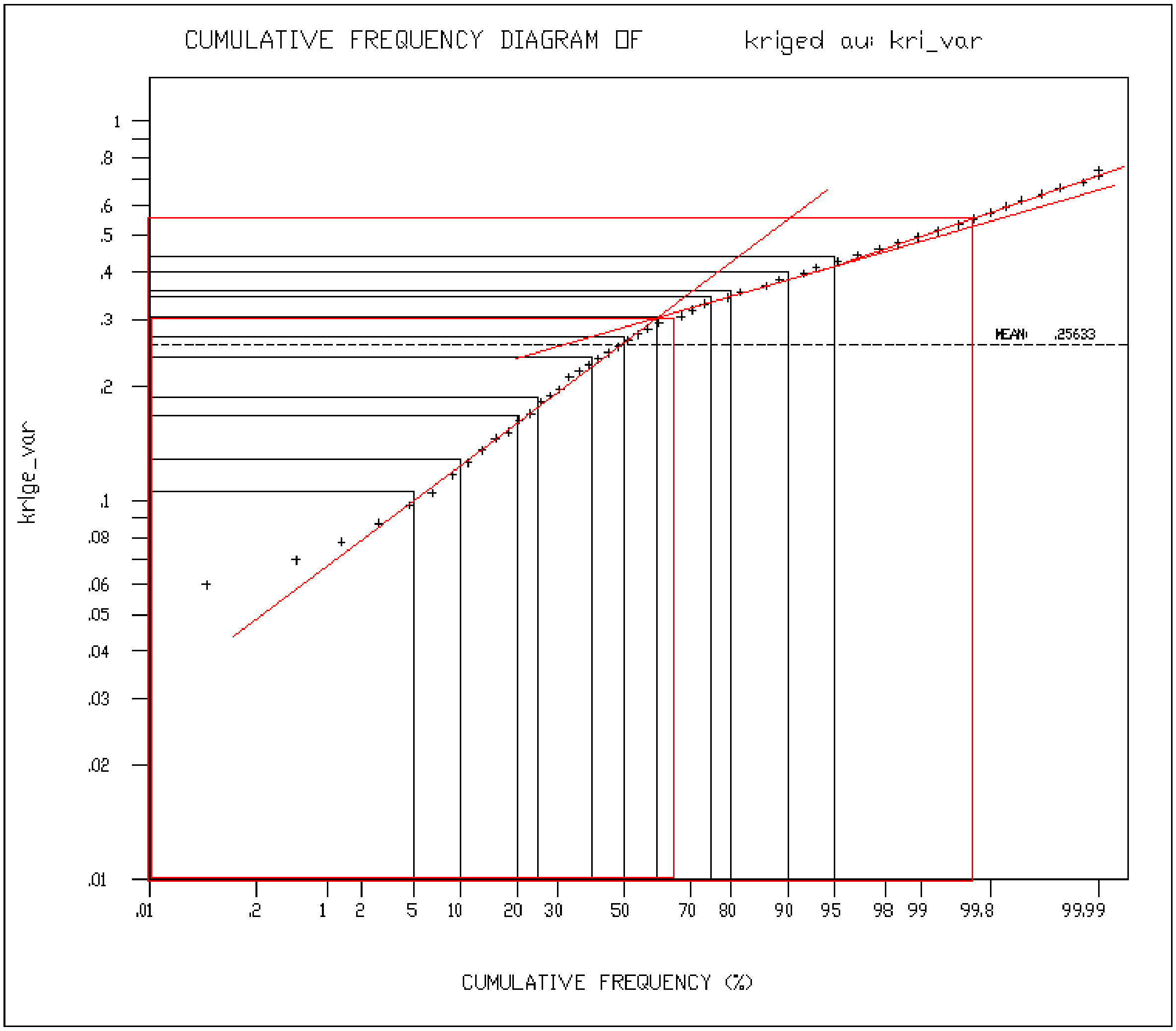


FIGURE 17-15: Cumulative frequency plot of GUS-JAS kriging variance. Measured to Indicated resource class at 0.30 krige_var. Indicated to Inferred resource class at 0.55 krige_var.

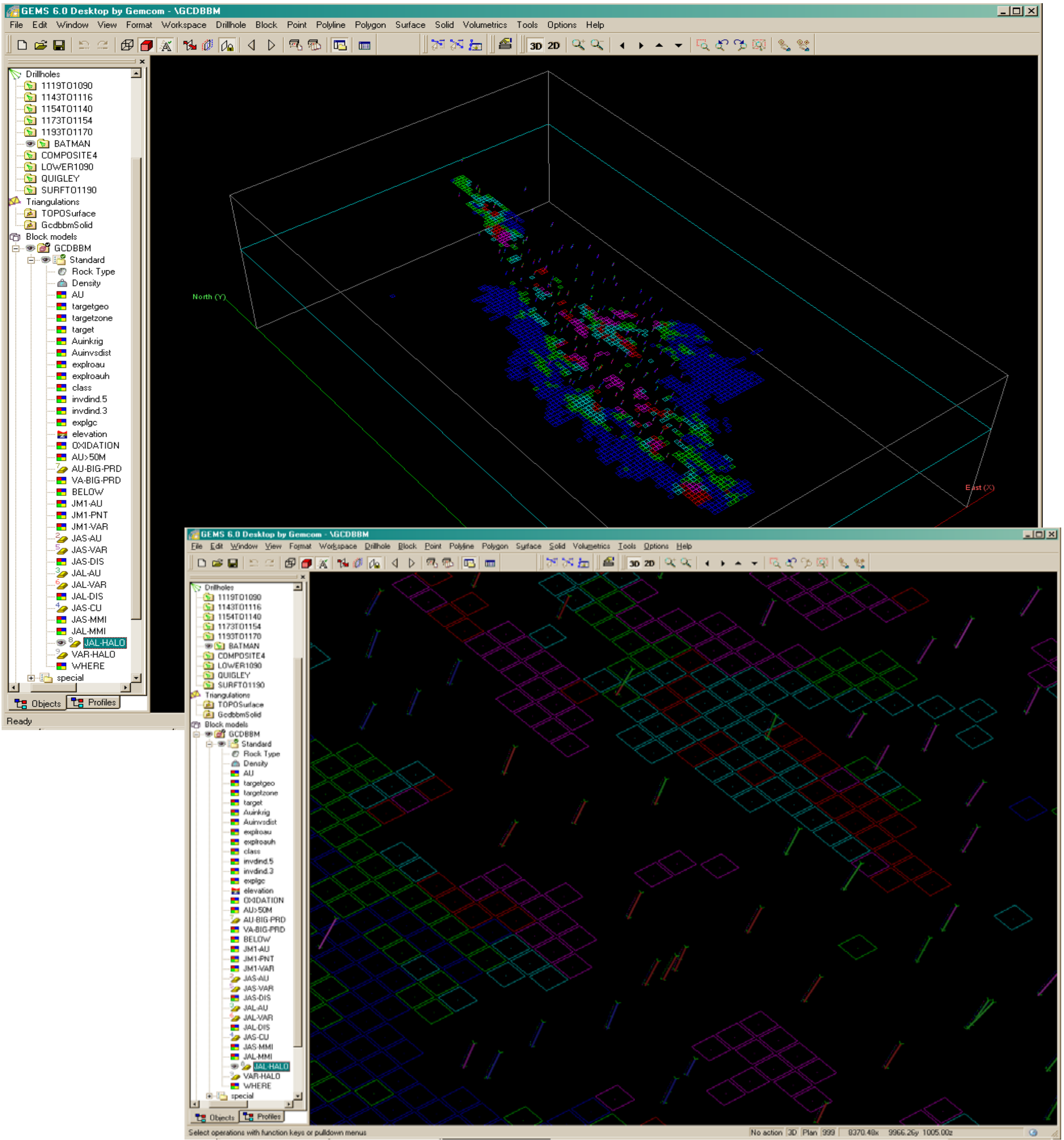


FIGURE 17-16: HALO, GUS-JAL minus GUS-JAS leaves a void where blocks are measured and indicated. Blocks that remain are inferred. Color of blocks going from blue to magenta indicates higher estimated grades. Strategy chosen for future sampling is to design a exploration program, which drills areas that have red and magenta blocks.

FIGURES 17-17 through 17-22 are east-west cross sections looking north that illustrate the drill hole traces, estimated gold blocks, and primary mineralized zones for the Batman Deposit. FIGURES 17-23 through 17-25 are north-south longitudinal sections looking east that show the drill hole traces, the estimated gold blocks, and the primary mineralized zones. It is important to note that the cross sections and longitudinal sections show estimated gold blocks above the current topographic surface. This is because all of the drill hole assay data were used to estimate the gold grades. These blocks have been removed prior to tabulating the in place geologic resources.

Finally, FIGURES 17-26 through 17-30 are plan view maps of the estimated gold grades with drill hole pierce points and the primary mineralized zones.

17.6.1 Resource Estimate

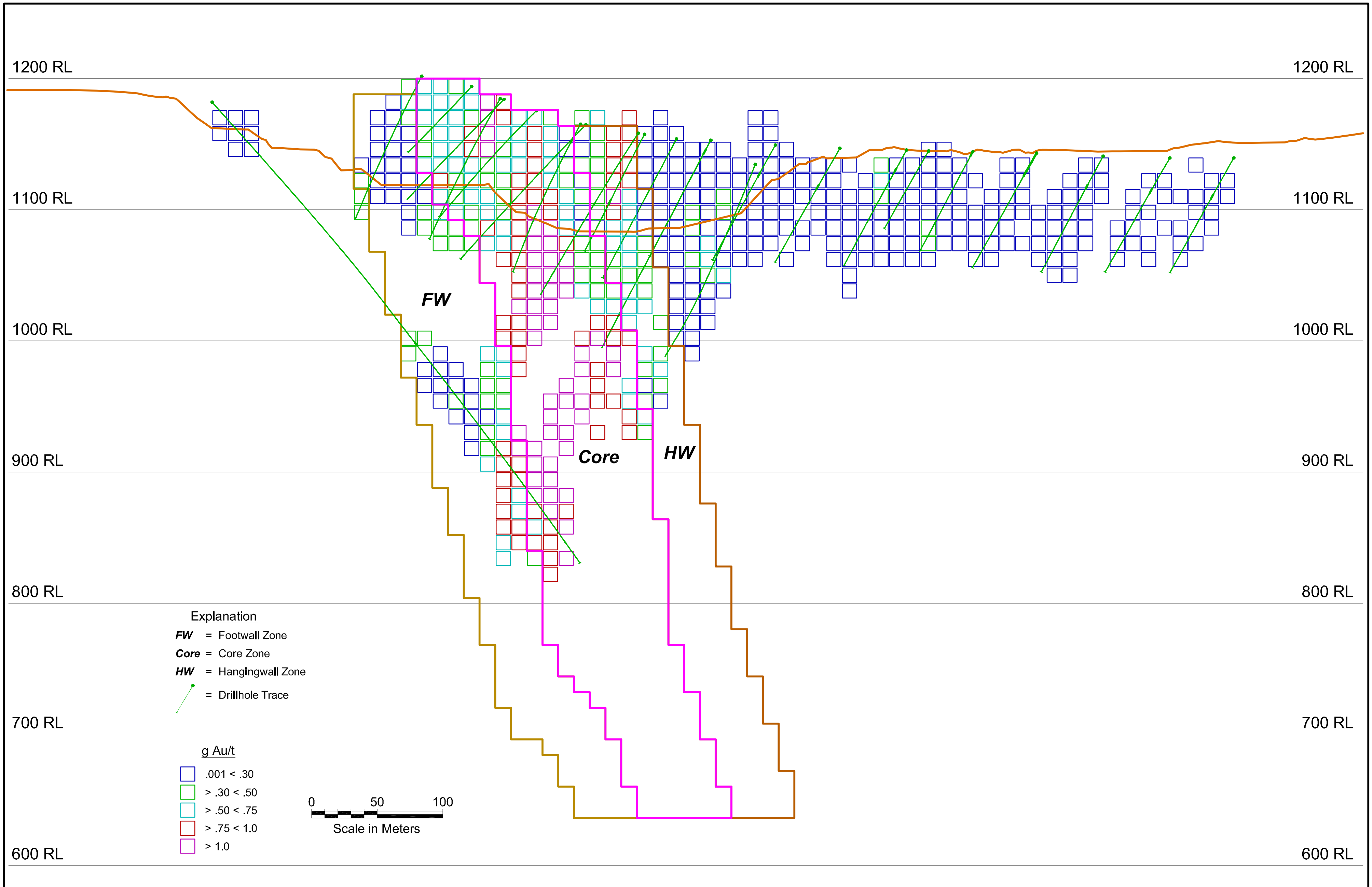
At the present time, resources have only been estimated for the Batman Deposit. Gustavson created three-dimensional computerized geologic and grade models of the Batman Deposit. While the deposit model also contains the Quigleys Deposit, no geologic resource estimate has been made for this deposit at the present time.

The geologic model of the Batman Deposit was created by GGC and audited by Gustavson. 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 Gustavson’s opinion that the GGC geologic model accurately portrays the geologic environment of the Batman Deposit.

Gustavson 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 Gustavson’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 according to the parameters detailed in TABLE 17-15. The classification was accomplished by a combination of kriging variance, number of points used in the estimate, and number of sectors used. TABLE 17-16 details the results of the classification.



Note: All drillhole data were used for resource estimation. Blocks above topography have been subtracted from reported resources.

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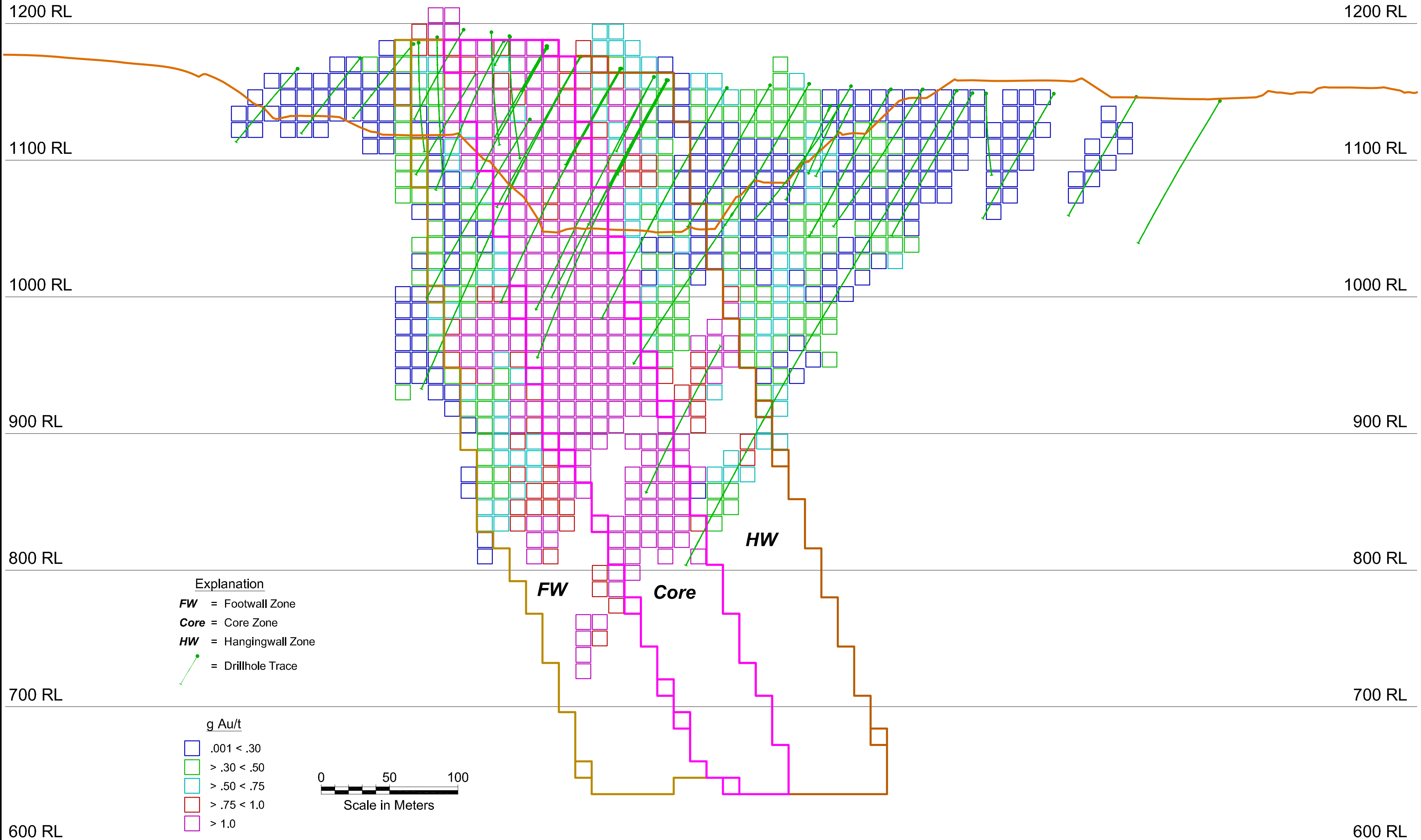
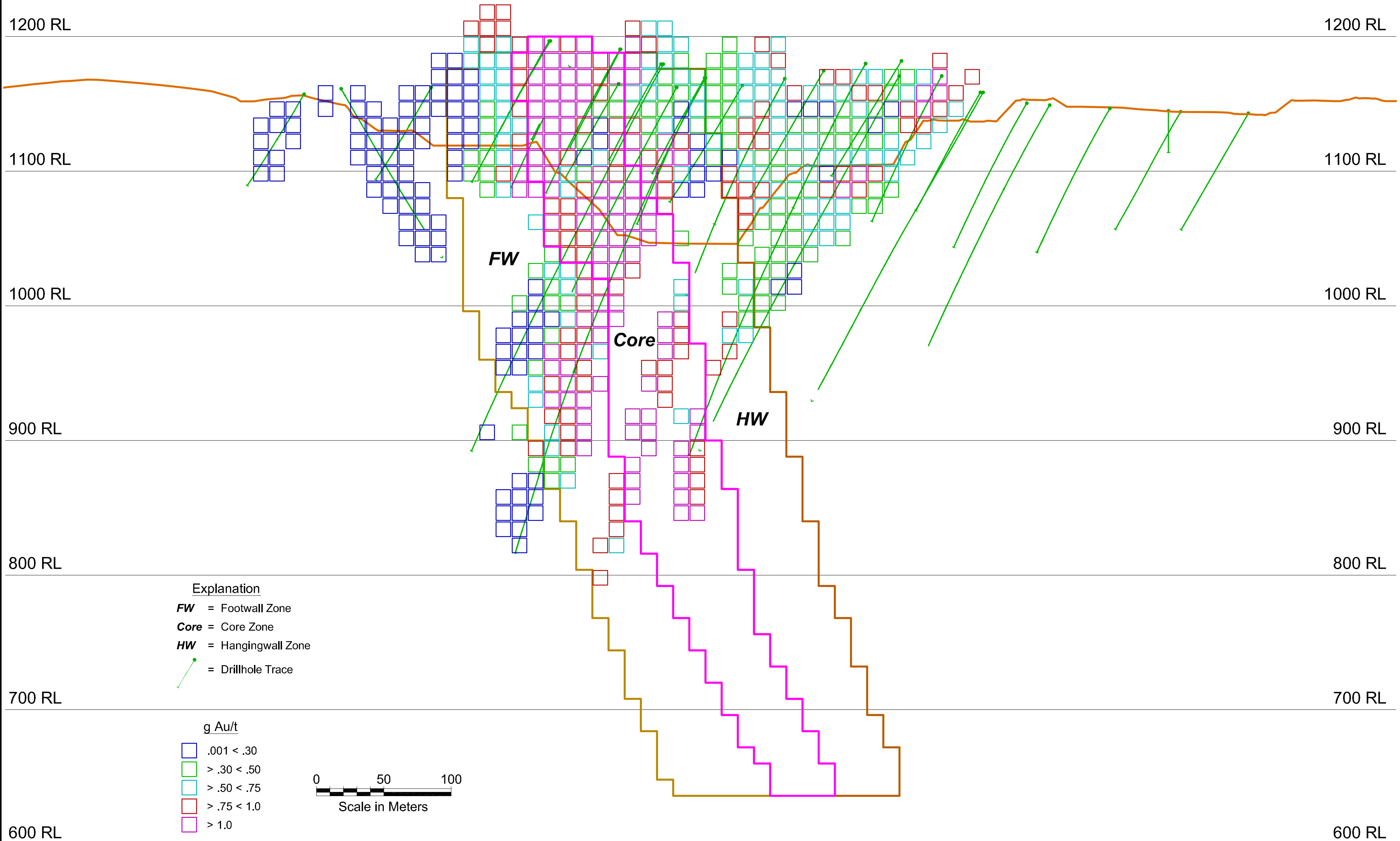
Date of Issue.	08/02/2006
Project No.	VIS005
Drawing File Name.	Fig17-17.dwg
Client Name.	**VISTA GOLD CORP.**
Project Name.	Mt. Todd Gold Project

FIGURE 17-17
SECTION WE-9800
(LOOKING NORTH)



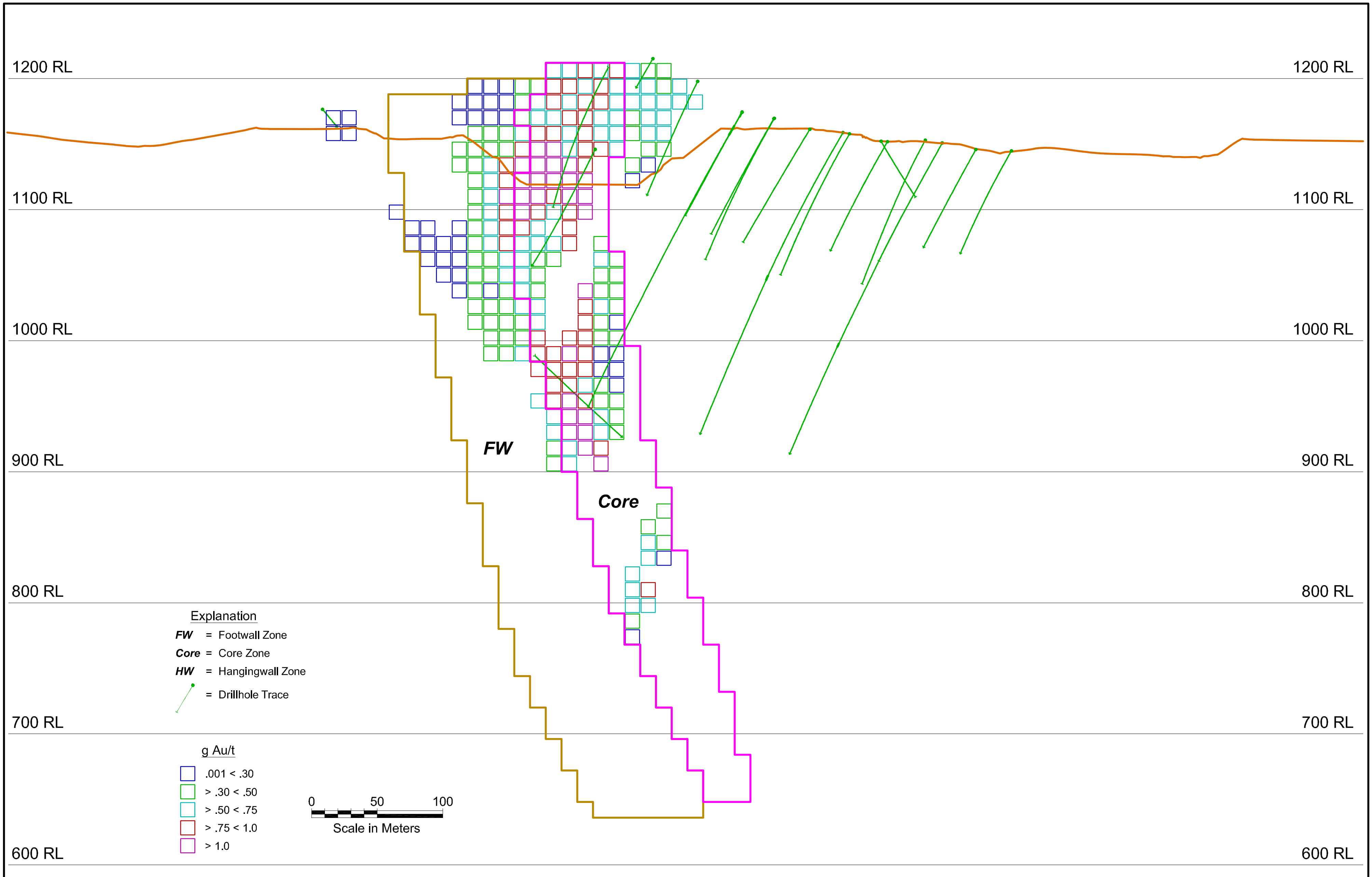
Note: All drillhole data were used for resource estimation. Blocks above topography have been subtracted from reported resources.

GUSTAVSON ASSOCIATES GEOLOGISTS · ENGINEERS · APPRAISERS	Date of Issue. 08/02/2006	Client Name. **VISTA GOLD CORP.**	FIGURE 17-18 SECTION WE-10000 (LOOKING NORTH)
	Project No. VIS005	Project Name. MT. TODD GOLD PROJECT	
	Drawing File Name. Fig17-18.dwg		



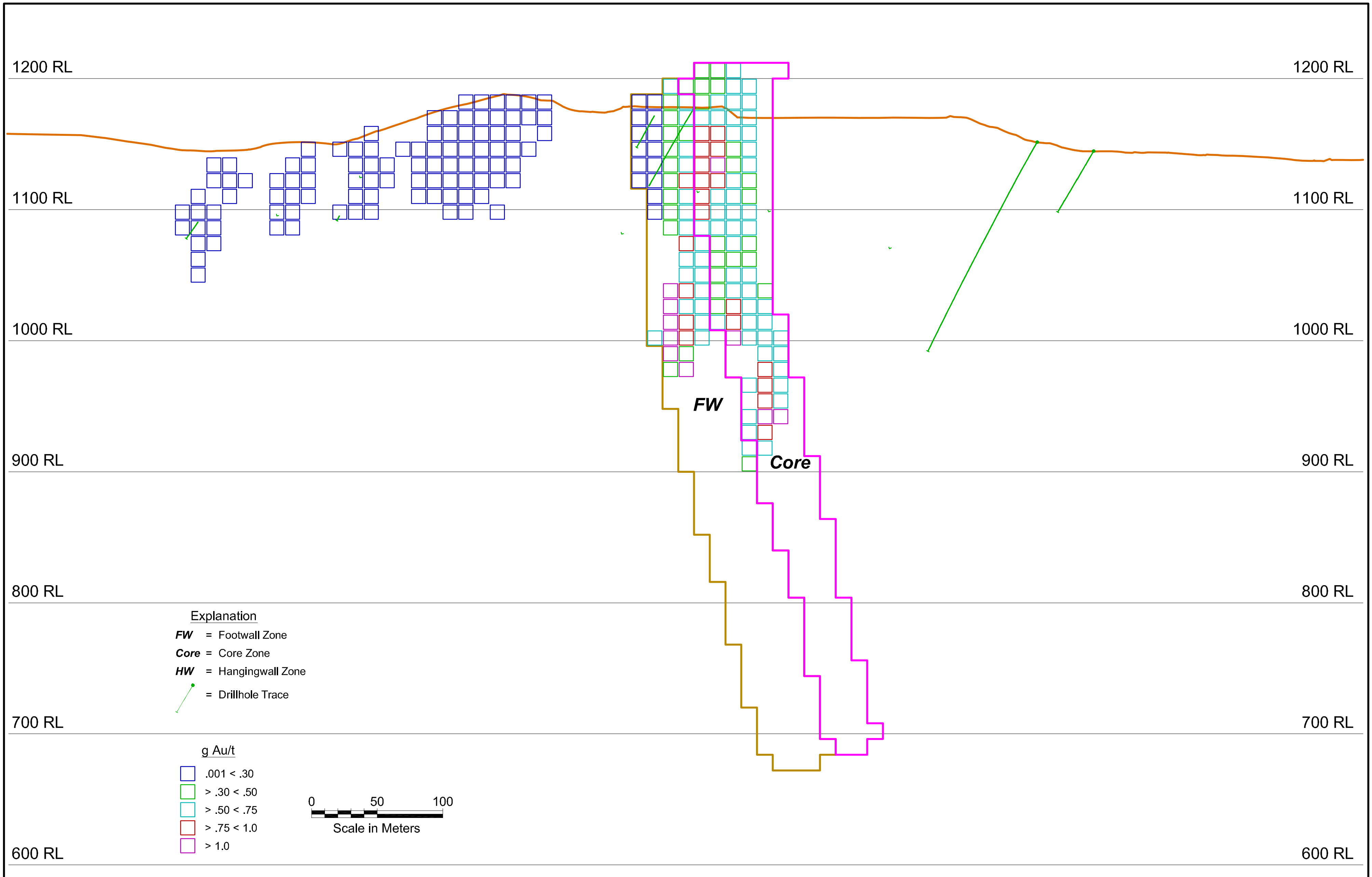
Note: All drillhole data were used for resource estimation. Blocks above topography have been subtracted from reported resources.

GUSTAVSON ASSOCIATES GEOLOGISTS · ENGINEERS · APPRAISERS	DATE OF ISSUE. 08/02/2006 PROJECT NO. VIS005 DRAWING FILE NAME. Fig17-19.dwg	CLIENT NAME. **VISTA GOLD CORP.** PROJECT NAME. MT. TODD GOLD PROJECT	FIGURE 17-19 SECTION WE-10200 (LOOKING NORTH)



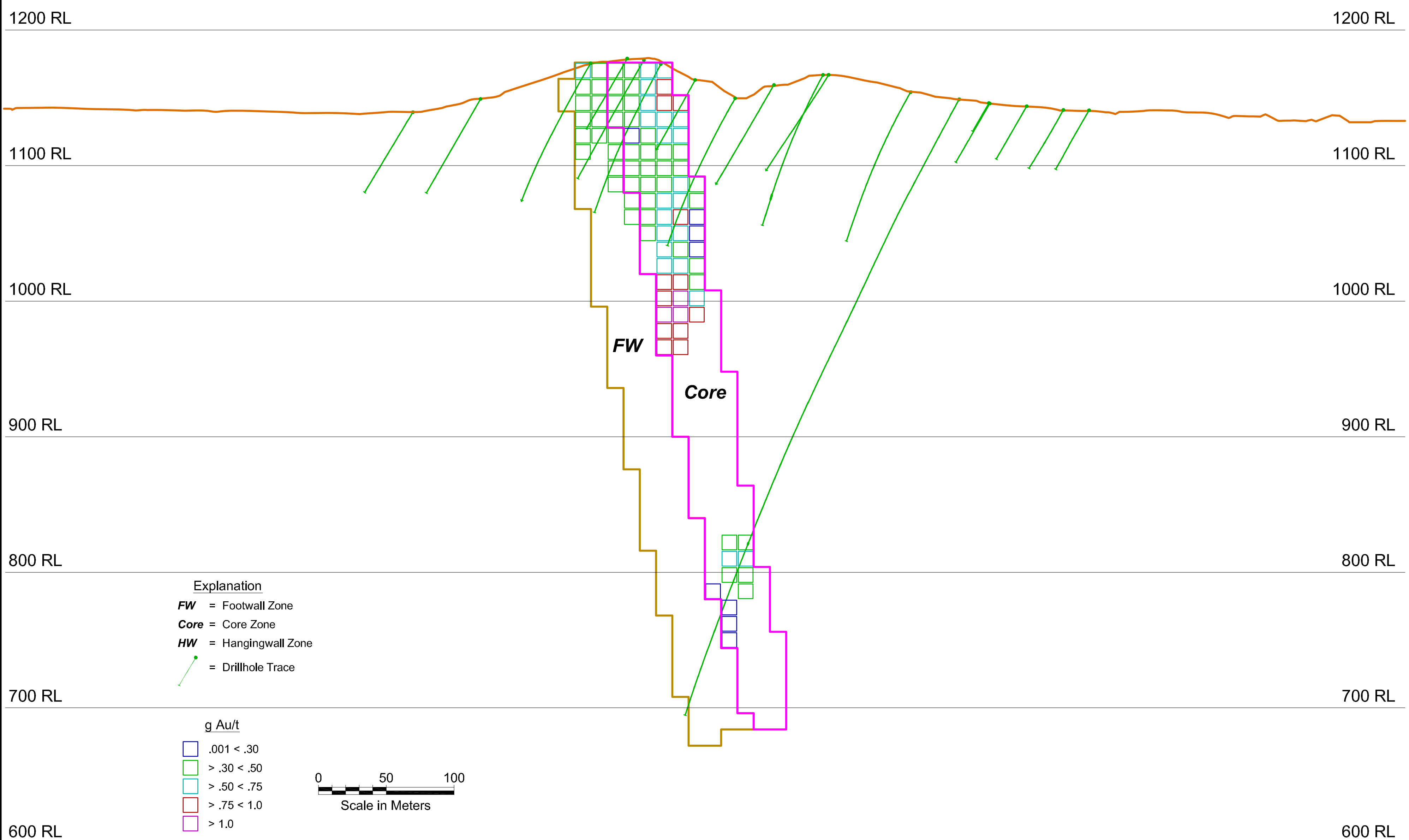
Note: All drillhole data were used for resource estimation. Blocks above topography have been subtracted from reported resources.

GUSTAVSON ASSOCIATES GEOLOGISTS · ENGINEERS · APPRAISERS	DATE OF ISSUE. 08/02/2006 PROJECT NO. VIS005 DRAWING FILE NAME. Fig17-20.dwg	CLIENT NAME. **VISTA GOLD CORP.** PROJECT NAME. MT. TODD GOLD PROJECT	FIGURE 17-20 SECTION WE-10400 (LOOKING NORTH)



Note: All drillhole data were used for resource estimation. Blocks above topography have been subtracted from reported resources.

GUSTAVSON ASSOCIATES GEOLOGISTS · ENGINEERS · APPRAISERS	Date of Issue. 08/02/2006 Project No. VIS005 Drawing File Name. Fig17-21.dwg	Client Name. **VISTA GOLD CORP.** Project Name. Mt. Todd Gold Project	FIGURE 17-21 Section WE-10610 (Looking North)



Note: All drillhole data were used for resource estimation. Blocks above topography have been subtracted from reported resources.

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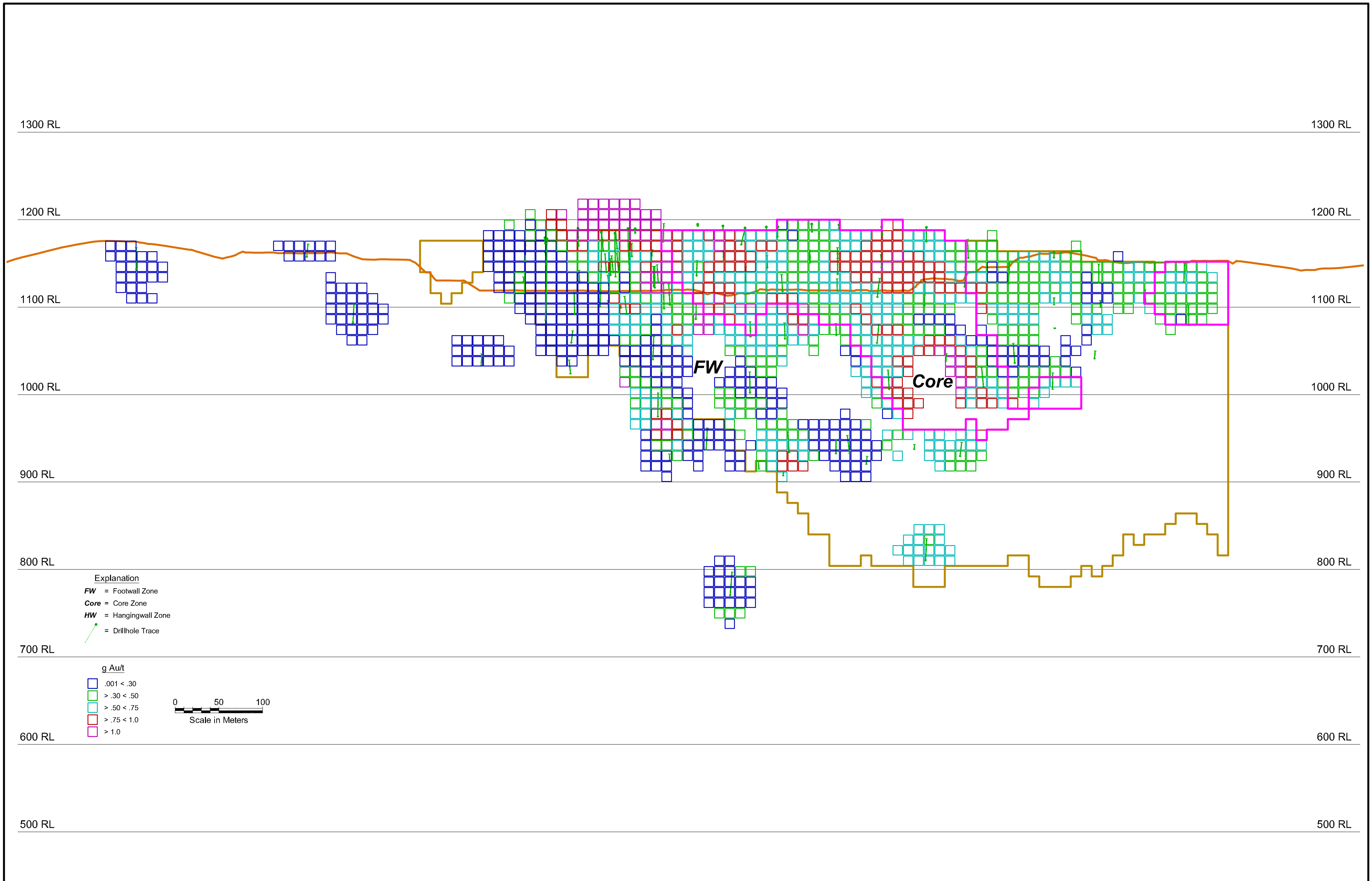
DATE OF ISSUE.	08/02/2006
PROJECT NO.	VIS005
DRAWING FILE NAME.	Fig17-22.dwg
CLIENT NAME.	***VISTA GOLD CORP.***
PROJECT NAME.	MT.TODD GOLD PROJECT

FIGURE 17-22
SECTION WE-10800
(LOOKING NORTH)



Note: All drillhole data were used for resource estimation. Blocks above topography have been subtracted from reported resources.

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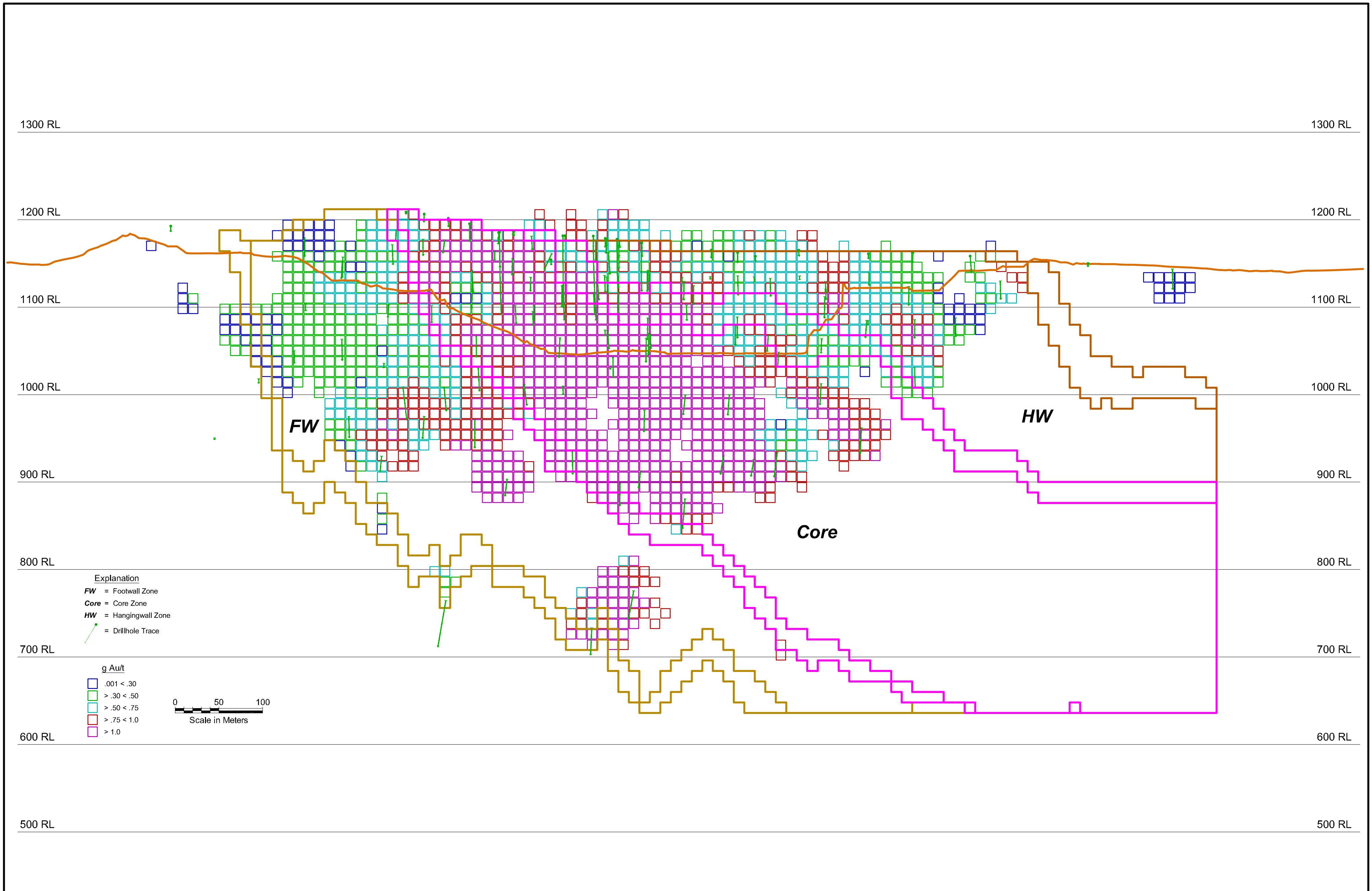
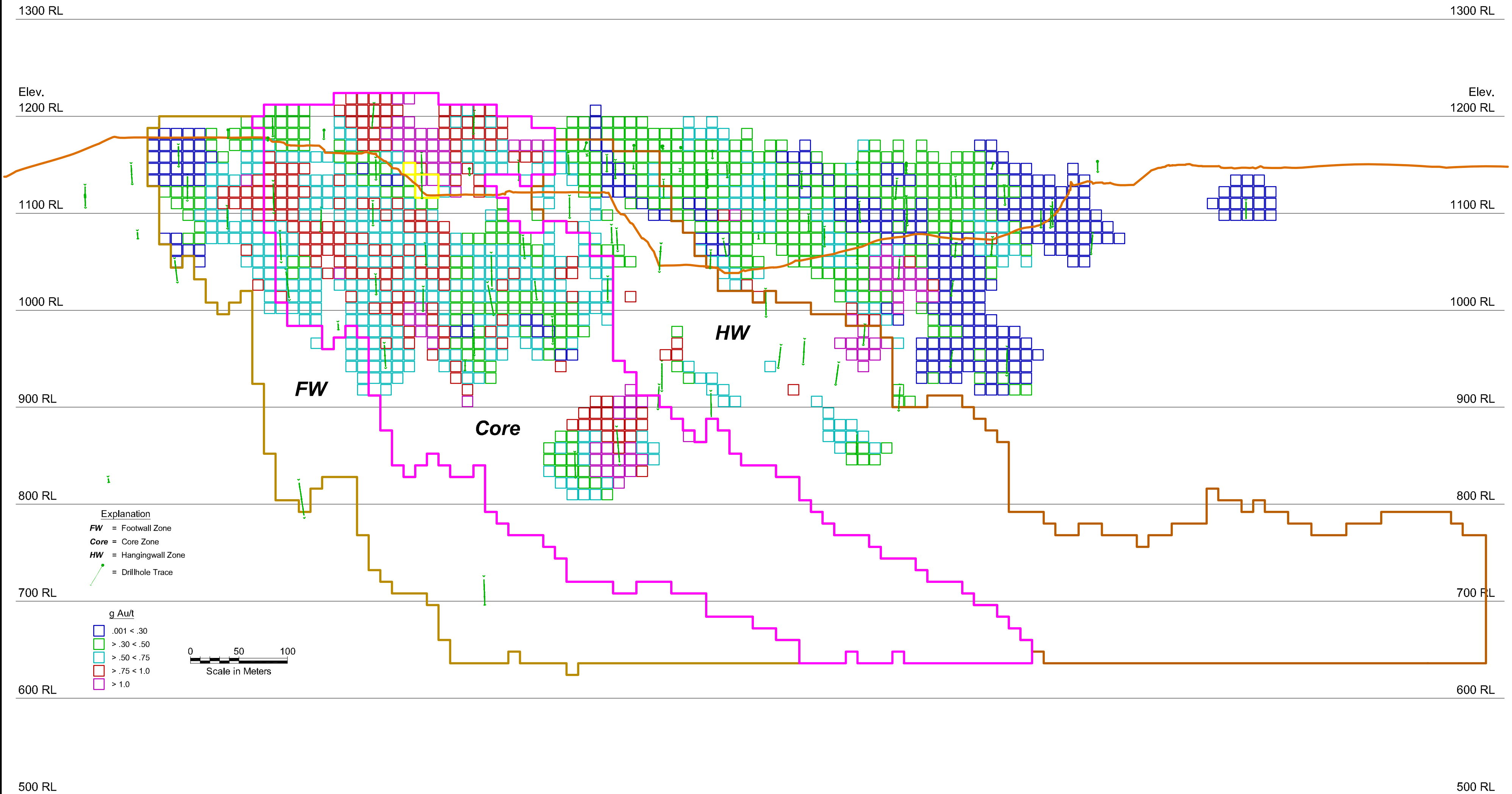
DATE OF ISSUE. 08/02/2006	CLIENT NAME. **VISTA GOLD CORP.**	FIGURE 17-23
PROJECT NO. VIS005	PROJECT NAME. MT. TODD GOLD PROJECT	SECTION NS-8400
DRAWING FILE NAME. Fig17-23.dwg		(LOOKING EAST)



Note: All drillhole data were used for resource estimation. Blocks above topography have been subtracted from reported resources.

GUSTAVSON ASSOCIATES GEOLOGISTS · ENGINEERS · APPRAISERS	DATE OF ISSUE. 08/02/2006	CLIENT NAME. VISTA GOLD CORP.	FIGURE 17-24 SECTION NS-8500 (LOOKING EAST)
	PROJECT NO. VIS005	PROJECT NAME. MT. TODD GOLD PROJECT	
	DRAWING FILE NAME. Fig17-24.dwg		



Note: All drillhole data were used for resource estimation. Blocks above topography have been subtracted from reported resources.

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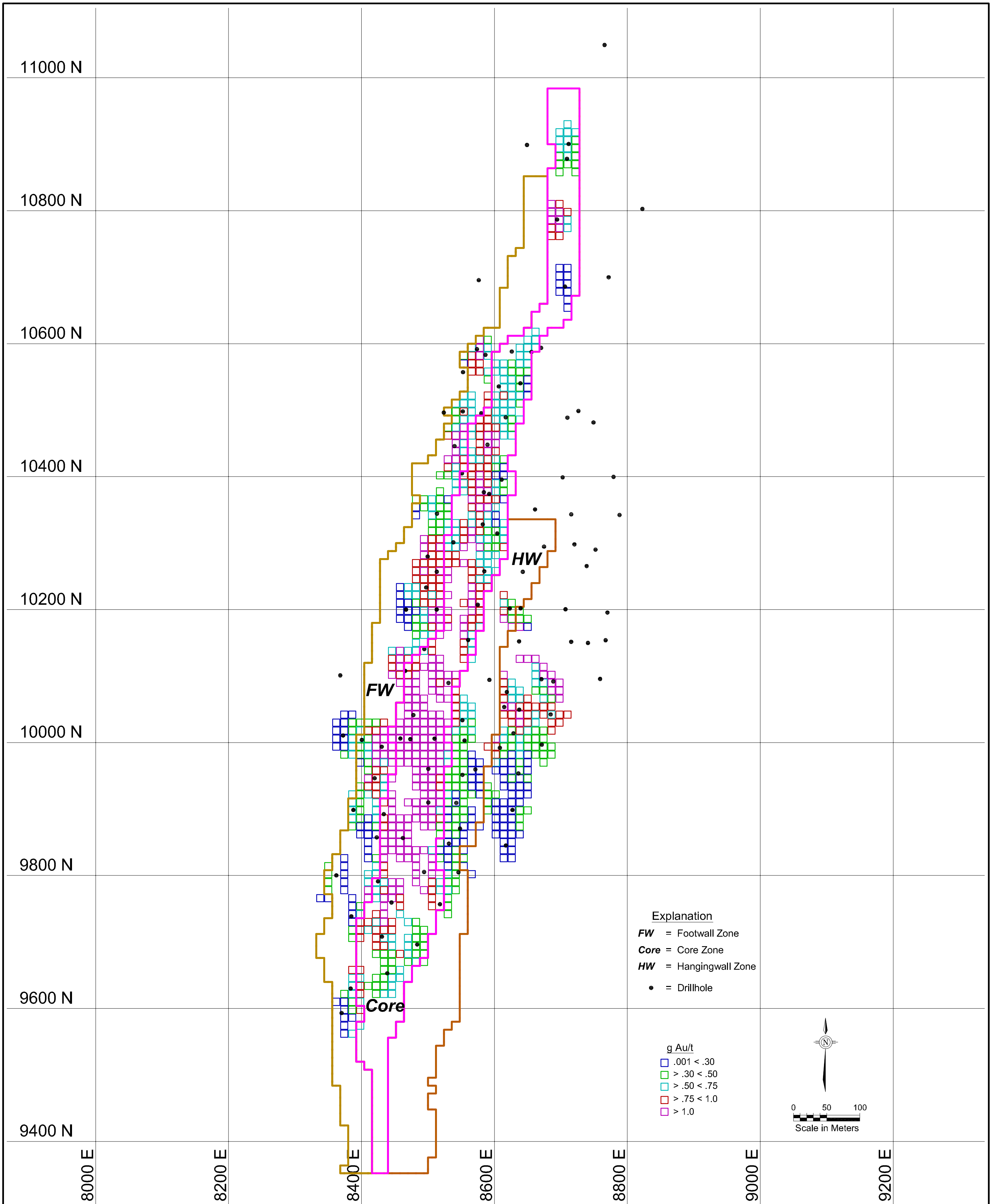
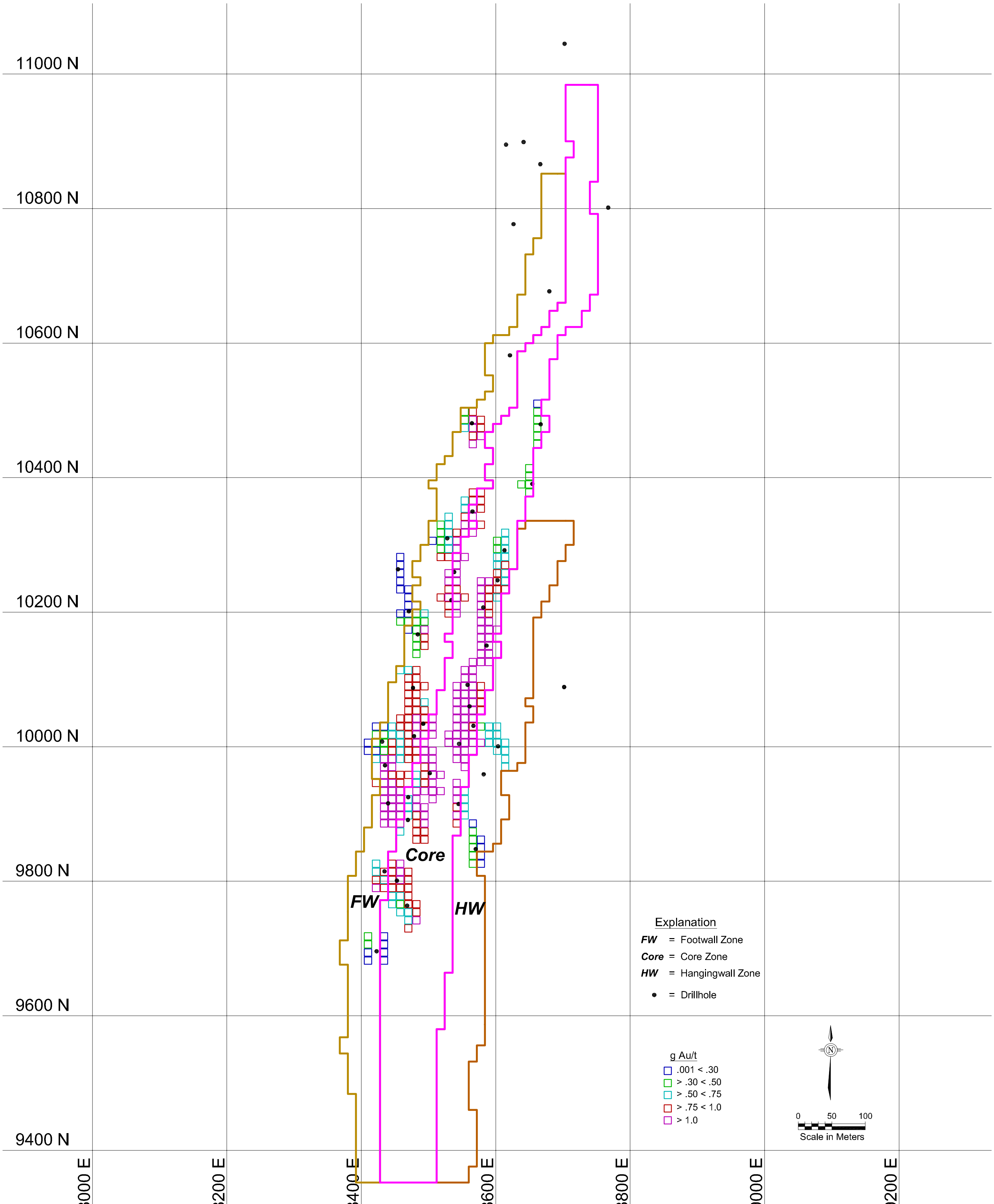
DATE OF ISSUE. 08/02/2006	CLIENT NAME. **VISTA GOLD CORP.**	FIGURE 17-25
PROJECT NO. VIS005	PROJECT NAME. MT. TODD GOLD PROJECT	SECTION NS-8600
DRAWING FILE NAME. Fig17-25.dwg		(LOOKING EAST)



Note: All drillhole data were used for resource estimation. Blocks above topography have been subtracted from reported resources.

GUSTAVSON ASSOCIATES GEOLOGISTS · ENGINEERS · APPRAISERS	DATE OF ISSUE. 08/02/2006 PROJECT NO. VIS005 DRAWING FILE NAME. Fig17-26.dwg	CLIENT NAME. **VISTA GOLD CORP.** PROJECT NAME. MT. TODD GOLD PROJECT	FIGURE 17-26 LEVEL 990 RL



Note: All drillhole data were used for resource estimation. Blocks above topography have been subtracted from reported resources.

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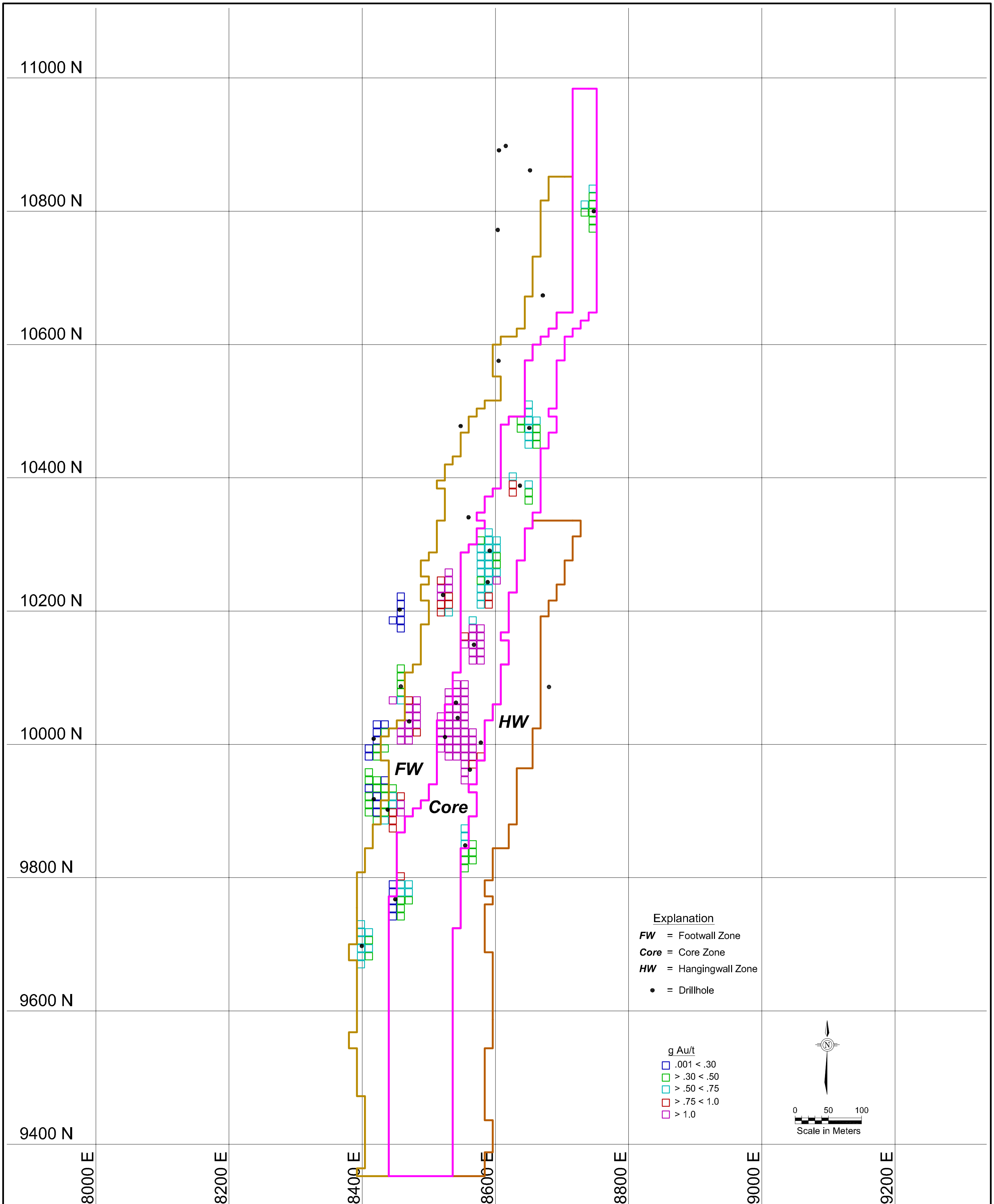
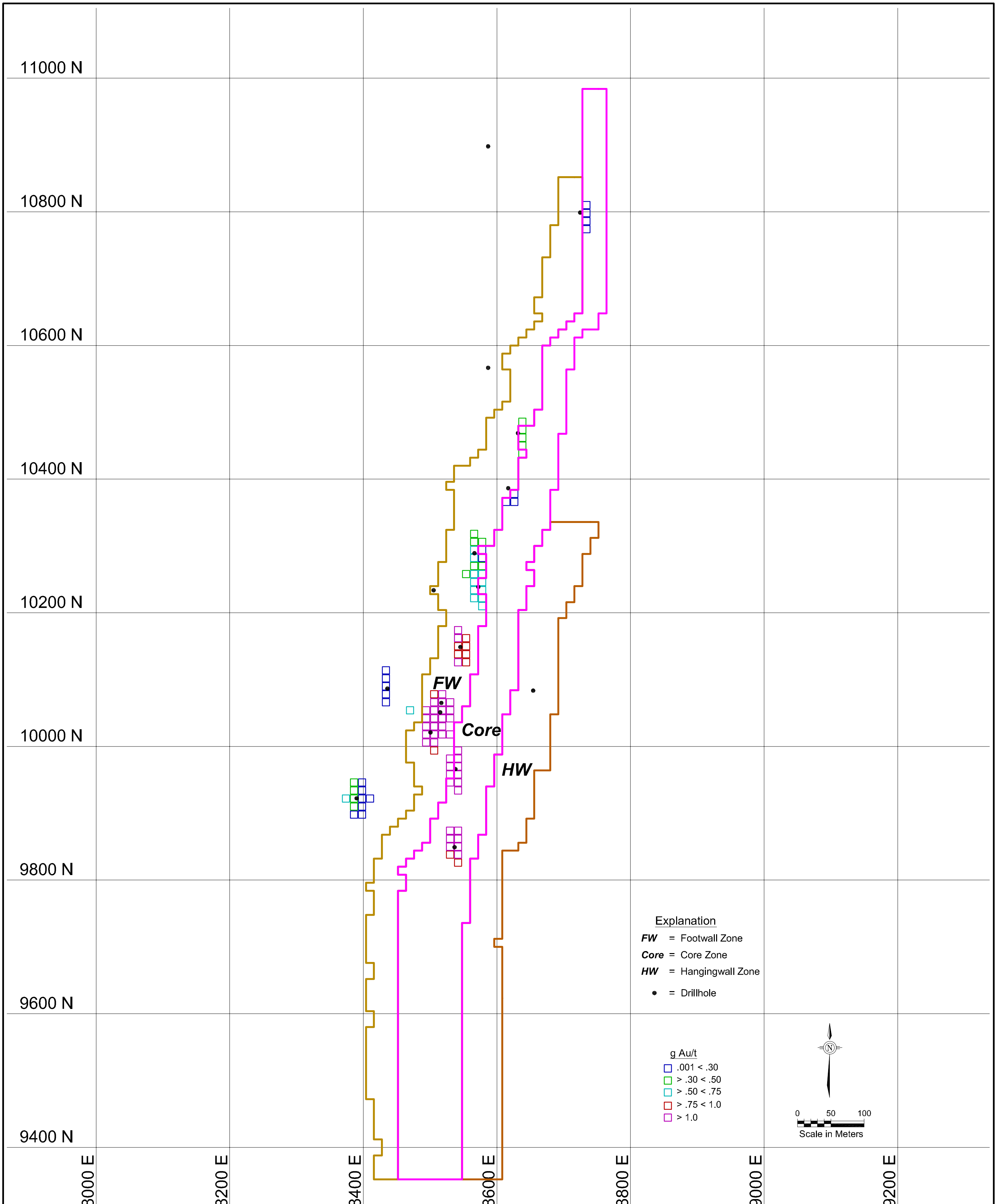
DATE OF ISSUE. 08/02/2006	CLIENT NAME.	**VISTA GOLD CORP.**	FIGURE 17-27 LEVEL 870 RL
PROJECT NO. VIS005	PROJECT NAME.	MT. TODD GOLD PROJECT	
DRAWING FILE NAME. Fig17-27.dwg			



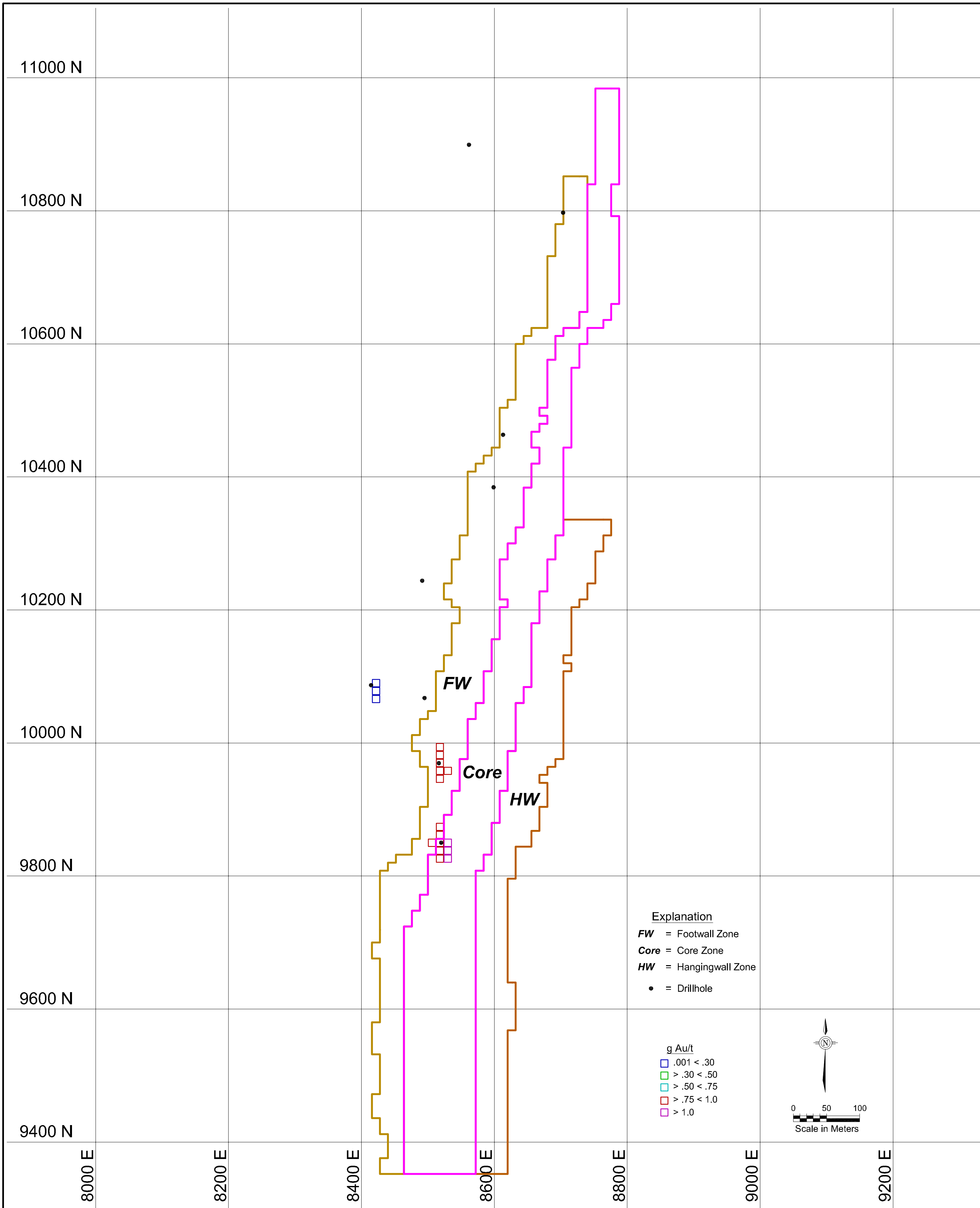
Note: All drillhole data were used for resource estimation. Blocks above topography have been subtracted from reported resources.

GUSTAVSON ASSOCIATES GEOLOGISTS · ENGINEERS · APPRAISERS	Date of Issue. 08/02/2006 Project No. VIS005 Drawing File Name. Fig17-28.dwg	Client Name. **VISTA GOLD CORP.** Project Name. Mt. Todd Gold Project	FIGURE 17-28 LEVEL 822 RL



Note: All drillhole data were used for resource estimation. Blocks above topography have been subtracted from reported resources.

GUSTAVSON ASSOCIATES GEOLOGISTS · ENGINEERS · APPRAISERS	DATE OF ISSUE. 08/02/2006	CLIENT NAME. **VISTA GOLD CORP.**	FIGURE 17-29 LEVEL 762 RL
	PROJECT NO. VIS005	PROJECT NAME. MT. TODD GOLD PROJECT	
	DRAWING FILE NAME. Fig17-29.dwg		



Note: All drillhole data were used for resource estimation. Blocks above topography have been subtracted from reported resources.

GUSTAVSON ASSOCIATES GEOLOGISTS · ENGINEERS · APPRAISERS	DATE OF ISSUE. 08/02/2006	CLIENT NAME. **VISTA GOLD CORP.**	FIGURE 17-30 LEVEL 702 RL
	PROJECT NO. VIS005	PROJECT NAME. MT. TODD GOLD PROJECT	
	DRAWING FILE NAME. Fig17-30.dwg		

TABLE 17-16 VISTA GOLD CORP. – MT TODD GOLD PROJECT Batman Resource Classification Criteria December 2006			
Category	**Kriging Variance**	**No. of Sectors**	**No. of Points/Sector**
Measured	JAS Model < 0.30	4	4 to 16
Indicated	JAS Model >=0.30<0.55	4	4 to 16
Inferred	Halo Model < 0.45	<4	2 to 8

TABLE 17-17 details the estimated in-place resources by classification and by cutoff grade for the Batman Deposit. All of the resources quoted are contained on Vista's mineral leases.

TABLE 17-17 VISTA GOLD CORP. – MT TODD GOLD PROJECT Batman Deposit Classified Resources December 2006							
MEASURED RESOURCES							
		INCREMENTAL			**CUMULATIVE**		
Cutoff Grade g Au/t		**Tonnes (1000)**	**Average Grade (g Au/t)**	**Contained Ounces**	**Tonnes (1000)**	**Average Grade (g Au/t)**	**Contained Ounces**
0.3	0.4	4,018	0.35	44,955	26,113	0.80	673,885
0.4	0.5	3,674	0.45	52,800	22,095	0.89	628,930
0.5	0.75	7,050	0.62	139,851	18,421	0.97	576,130
0.75	1.0	4,768	0.87	133,366	11,371	1.19	436,279
1.0	2.0	6,034	1.34	259,375	6,603	1.43	302,913
2.0	3.0	526	2.29	38,727	569	2.38	43,538
>3.0		43	3.48	4,811	43	3.48	4,811
INDICATED RESOURCES							
		INCREMENTAL			**CUMULATIVE**		
Cutoff Grade g Au/t		**Tonnes (1000)**	**Average Grade (g Au/t)**	**Contained Ounces**	**Tonnes (1000)**	**Average Grade (g Au/t)**	**Contained Ounces**
0.3	0.4	8,069	0.30	77,827	53,784	0.79	1,371,440
0.4	0.5	7,758	0.45	111,992	45,715	0.88	1,293,612
0.5	0.75	14,752	0.62	293,110	37,957	0.97	1,181,620
0.75	1.0	9,776	0.87	272,503	23,205	1.19	888,511
1.0	2.0	12,300	1.34	531,095	13,429	1.43	616,008
2.0	3.0	1,043	2.26	75,886	1,129	2.34	84,913
>3.0		86	3.27	9,028	86	3.27	9,028

MEASURED + INDICATED RESOURCES							
		INCREMENTAL			CUMULATIVE		
Cutoff Grade g Au/t		Tonnes (1000)	Average Grade (g Au/t)	Contained Ounces	Tonnes (1000)	Average Grade (g Au/t)	Contained Ounces
0.3	0.4	12,087	0.32	122,782	79,897	0.80	2,045,325
0.4	0.5	11,432	0.45	164,792	67,810	0.88	1,922,542
0.5	0.75	21,802	0.62	432,960	56,378	0.97	1,757,750
0.75	1.0	14,544	0.87	405,869	34,576	1.19	1,324,789
1.0	2.0	18,334	1.34	790,469	20,032	1.43	918,920
2.0	3.0	1,569	2.27	114,612	1,698	2.35	128,451
>3.0		129	3.34	13,839	129	3.34	13,839

INFERRED RESOURCES							
		INCREMENTAL			CUMULATIVE		
Cutoff Grade g Au/t		Tonnes (1000)	Average Grade (g Au/t)	Contained Ounces	Tonnes (1000)	Average Grade (g Au/t)	Contained Ounces
0.3	0.4	15,032	0.35	168,775	76,786	0.75	1,840,504
0.4	0.5	11,701	0.45	168,536	61,754	0.84	1,671,729
0.5	0.75	20,365	0.62	402,756	50,053	0.93	1,503,194
0.75	1.0	12,611	0.87	351,299	29,688	1.15	1100,438
1.0	2.0	16,234	1.32	686,837	17,077	1.36	749,139
2.0	3.0	819	2.27	59,7228	843	2.30	62,302
>3.0		24	3.34	2,580	24	3.34	2,580

18.0 MINERAL RESERVE ESTIMATE

At the present time, the Mt Todd gold project contains no CIM definable mineral reserves. Since this study is a Preliminary Assessment (PA), we have used all of the estimated resources, i.e. measured, indicated, and inferred, for determination of the potentially mineable mineral resources.

18.1 Geotechnical Data

Very little geotechnical data are available for the Mt Todd Project. Based on the mining that has occurred to date, rock conditions are assumed to be favorable for mining, with no unusual ground stability issues. It is believed that the addition of the silica as part of the mineralizing event has significantly improved the rock strength and ultimate stability of any pit that will be developed. However, additional work will need to be completed prior to re-initiation of the mining. This work should include, as a minimum the following items:

1) Re-logging of the existing core to characterize the geotechnical conditions present.
2) Examination of the core photographs for geotechnical purposes.
3) Addition of geotechnical logging to the planned exploration program to improve the stability database.
4) Inclusion of core drilling in the unmineralized hangingwall and footwall units to confirm their stability conditions.
5) Completion of a hydrologic study to determine the potential impact any water may have on pit stability as it deepens.

18.2 Pit Optimization

Potentially mineable pit shapes were evaluated using a Lerchs-Grossman (LG) analyses performed with the Whittle 4 pit planning software and the Mt Todd GEMS® geologic model. The primary purpose of this was to determine ultimate pit limits and the best extraction sequence for open pit mine design. For this PA, measured, indicated, and inferred resources were considered potential ore. The parameters assumed for the LG analyses are summarized in TABLE 18-1 (all prices and costs are reported in fourth quarter 2006 US dollars).

Table 18-1 VISTA GOLD CORP. – MT TODD GOLD PROJECT Parameters for Lerchs-Grossman Analyses December 2006	
Average Pit Slopes	All 55 degrees
Gold Price	US$600 per oz Au
Gold Recovery	82 percent
Copper Price	US$2.00 per pound Cu
Copper Recovery	50 percent
Mining Cost	US$0.90 per tonne processed
Processing Cost	US$8.95 per tonne processed
General and Administrative Cost	US$0.25 per tonne processed
Environmental/Regulatory Cost	US$0.10 per tonne processed

The base case ultimate pit shell is defined by the economic factors listed in TABLE 18-1 and is illustrated by mid-bench contours on Figure 18-1. A total of 15 LG runs, including the base case, were run to determine sensitivities to gold price and pit slopes. Gold price sensitivity was analyzed in $25.00 per ounce increments from $200 to $600 per ounce Au. The results of the $400, $500, and $600 per ounce Au cases were used for preliminary phasing of the pit for mine planning. TABLE 18-2 summarizes the results from the $400, $450, $500, $550, and $600 per ounce Au runs:

TABLE 18-2
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Lerchs-Grossman Price Sensitivity
December 2006

Pit Slope (degrees)	Gold Price (US$/oz)	Copper price (US$/lb)	Mineable Tonnes (millions)	Avg. Au Grade (g)	Avg Cu Grade %(1)	Contained Oz Au (x 1000)	Waste Tonnes (Millions)	Stripping Ratio (W:O)
50	400	2.00	35.2	1.263	0.03	1,429.3	64.7	1.8
50	450	2.00	55.4	1.169	0.03	2,082.2	124.5	2.3
50	500	2.00	73.3	1.092	0.03	2,573.5	165.5	2.3
50	550	2.00	88.3	1.036	0.03	2,940.3	194.5	2.2
50	600	2.00	104.0	0.986	0.03	3,297.5	255.9	2.2
55	400	2.00	41.5	1.257	0.03	1,677.6	69.5	1.7
55	450	2.00	59.4	1.169	0.03	2,232.1	106.5	1.8
55	500	2.00	79.4	1.088	0.03	2,776.9	148.5	1.9
55	550	2.00	95.4	1.028	0.03	3,153.7	172.2	1.8
55	600	2.00	108.1	0.985	0.03	3,423.0	189.3	1.8

(1) Copper grades set to 75 percent of estimated copper grade of 0.04 percent based on the mathematical relationship from the Quigleys deposit and additional metallurgical testwork undertaken by RDi in 2006.

18.3 Pit Design

Using the base case, the ultimate pit was designed for medium-sized mining equipment, including 18-cubic meter hydraulic front shovels and 141-tonne haul trucks. The design includes smoothed pit walls, haulage ramps, benches, and pit access. After the ultimate pit was designed, two intermediate phases were created for production scheduling and enhancing the economics of the project. Phase 1 is based on the $400 per oz Au pit, and Phase 2 is based on the $500 per oz Au pit. These phases were also designed with smoothed pit walls, haul roads, and access. Phases 1 and 2 and the ultimate pit are illustrated by Figures 18-2, 18-3, and 18-4, respectively.

Pit Design Parameters

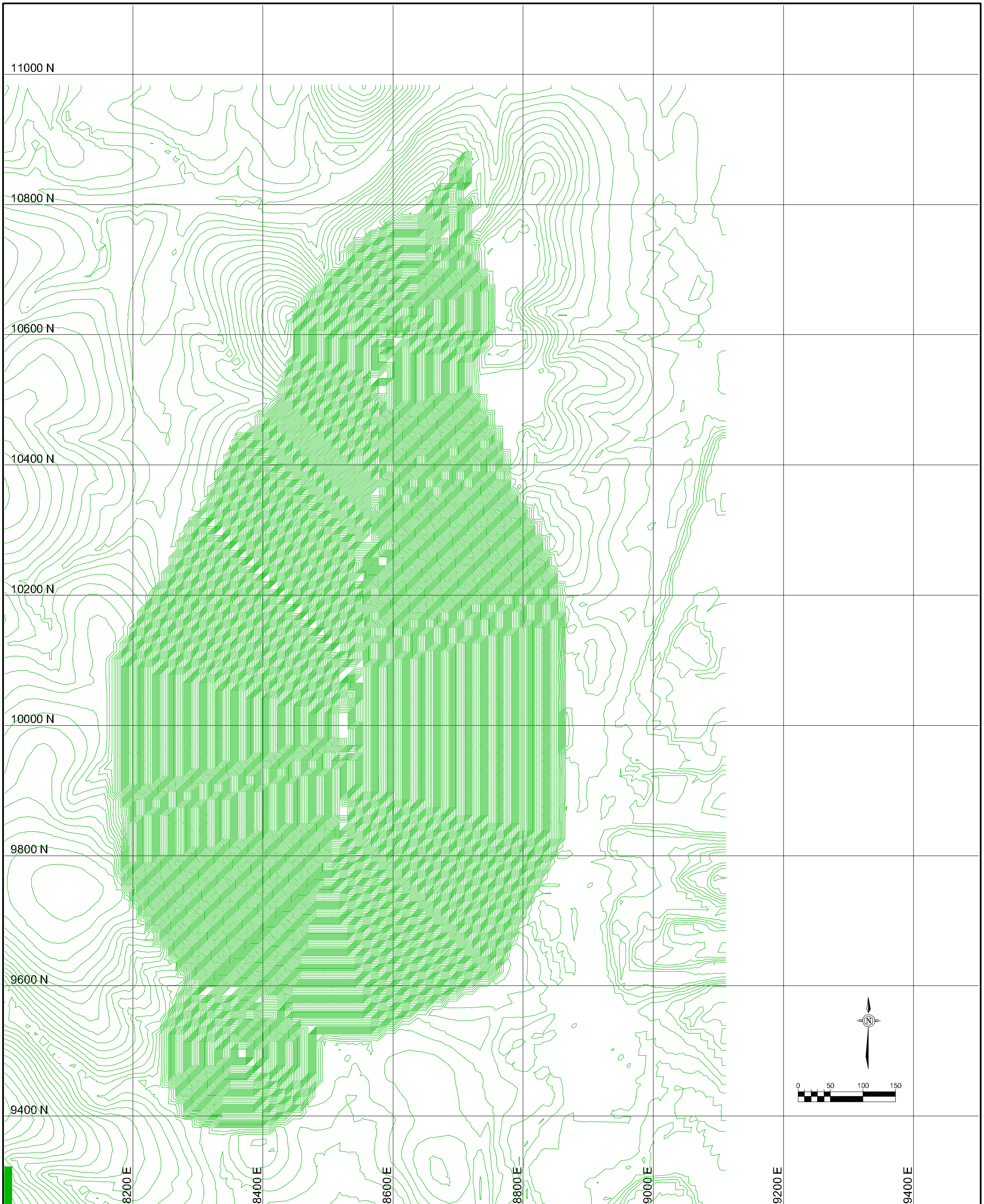
Without specific detailed geotechnical data, but utilizing reports from the previous operations it is believed that the geotechnical conditions are favorable for development of the proposed ultimate pit. Interramp slopes were assumed to average 55 degrees, with the bench heights and haul road widths designed to accommodate the midsize equipment fleet planned for Mt Todd.

The pit was developed with 12-meter benches and 24-meter intervals between catch benches (double bench). Bench face angles were designed at 78 degrees, and minimum catch bench width is set to 5.8 meters. Haulage ramps were designed using a 28 meter width and at a maximum grade of 10 percent. Near the bottom of the pit (specifically, the bottom three benches), ramp width was reduced to 20 meters (one-way traffic) and grades were increased to 15 percent in order to recover as much of the resource as possible.

Mining Phases

The preliminary production schedule presented in this PA is based on two intermediate phases and the ultimate pit (Phase 3), as discussed above. Phase 1 is based on the $400 per ounce Au LG pit, and Phase 2 is based on the $500 per ounce Au LG pit. In general, the plan mines Phase 1 for the first 4 years, followed by both Phases 2 and 3 during Years 4 to 8, and Phase 3 in Years 6 to10. The anticipated mining rate of about 10.65 million tonnes per year (30,0000 tonnes per day) results in a mine life of years.

The primary crusher will be located to the east of the pit, and all the waste will be hauled to a single waste stockpile to the south of the pit. Each phase of the mine plan includes access to these facilities.



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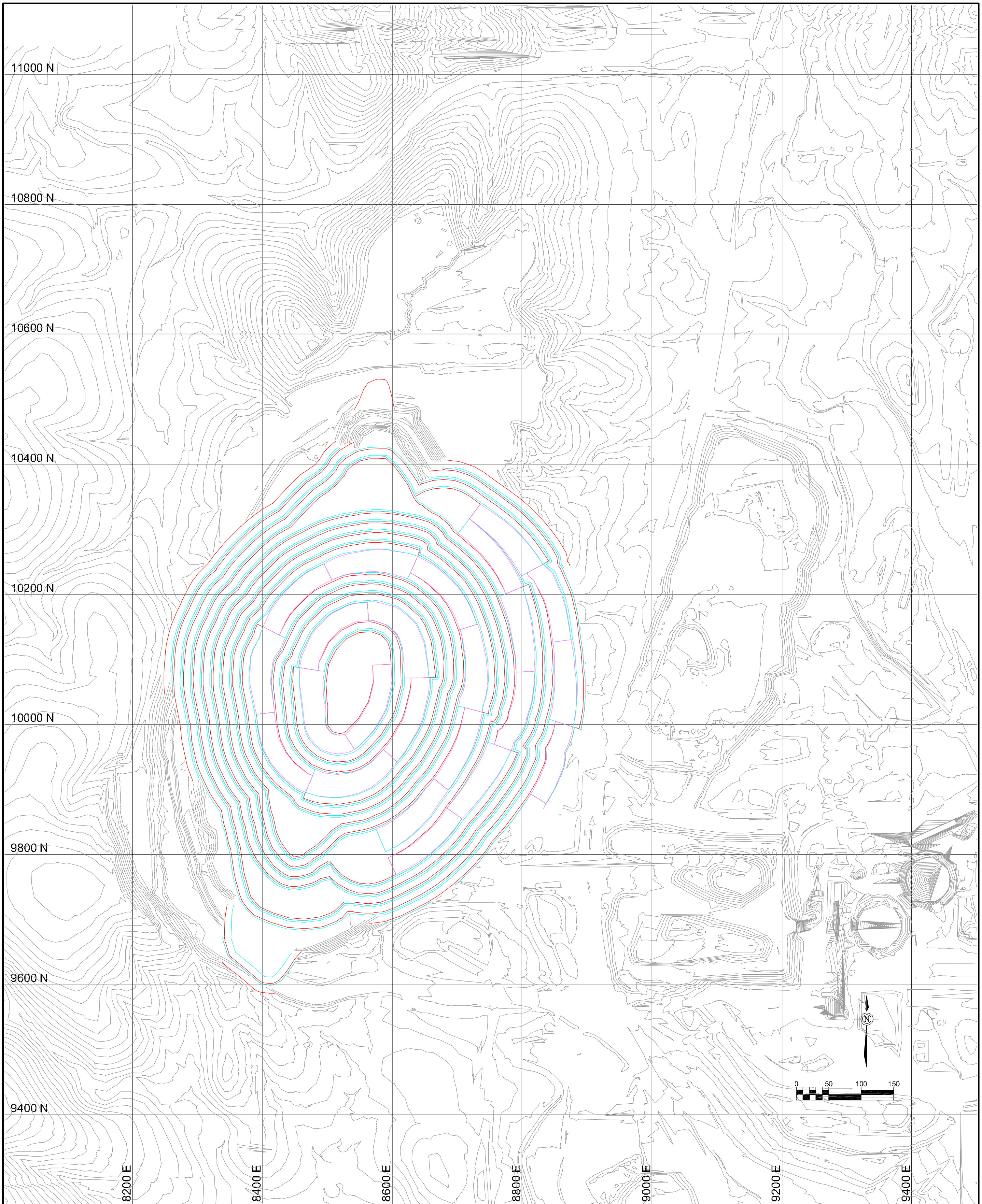
DATE OF ISSUE. 12/15/2006	CLIENT NAME. **VISTA GOLD CORP.**	FIGURE 18-1 $600AU WITH 55° SLOPE CONE PIT
PROJECT NO. VIS005	PROJECT NAME. MT. TODD GOLD PROJECT	
DRAWING FILE NAME. Fig18-1.dwg		



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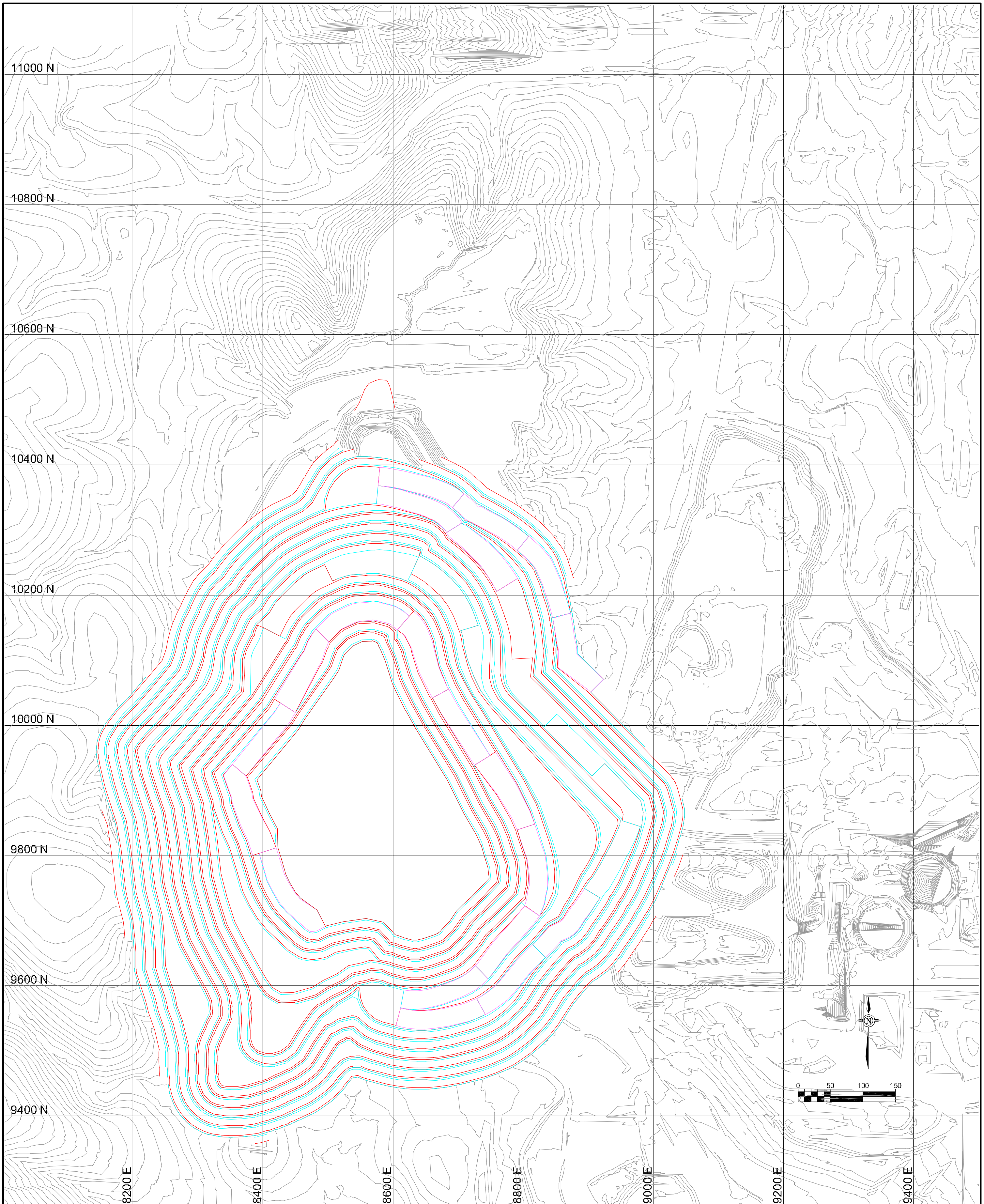
DATE OF ISSUE. 12/15/2006	CLIENT NAME. **VISTA GOLD CORP.**	FIGURE 18-2 $600 PHASE I PIT CONFIGURATION
PROJECT NO. VIS005	PROJECT NAME. MT. TODD GOLD PROJECT	
DRAWING FILE NAME. Fig18-2.dwg		



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Date of Issue.	Client Name.	
12/15/2006	**VISTA GOLD CORP.**	FIGURE 18-3
Project No. VIS005	Project Name.	$600 Phase 2 Pit
Drawing File Name. Fig18-3.dwg	Mt. Todd Gold Project	Configuration

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DATE OF ISSUE.	12/15/2006
PROJECT NO.	VIS005
DRAWING FILE NAME.	FNL_Pit-Dump_Dec14.dwg
CLIENT NAME.	VISTA GOLD CORP.
PROJECT NAME.	MT. TODD GOLD PROJECT

FIGURE 18-4
$600 FINAL PIT
WITH 55° SLOPES

18.4 Potentially Mineable Mineral Resource

This PA includes an inventory of "potentially mineable mineral resource," and no mineral reserve estimate is offered. Measured, Indicated, and Inferred (M+I+I) resources are included in the potentially mineable resource estimate, rather than the normal prefeasibility and feasibility requirement restricting "reserves" to only mineable measured and indicated (M+I) resources.

In-situ Bulk Density

Individual bulk densities of the material that will be mined are contained in TABLE 17-1 of SECTION 17 of this report.

Cutoff Grade

The LG analysis calculates the internal cutoff grade as it determines if a block should be mined as either ore or waste. Using the economic assumptions listed in TABLE 18-1, the internal cutoff grade for the Mt Todd deposit is calculated to be approximately 0.4g Au per tonne.

Dilution and Recovery

This PA does not apply a dilution factor to the material grades generated by the geologic model. The model incorporates some dilution in the compositing process, as does smoothing from grade interpolations. In addition, the ore zones are very continuous and ore control should be effective. Similarly, this analysis assumes 100 percent resource recovery. It is Gustavson’s opinion that these are not unrealistic expectations since the potentially mineralized material is distinct from the surrounding waste rock units due to the extensive silica flooding that occurred as part of the mineralizing event. In addition, once in the potentially mineralized zone, the majority of the material is above the economic cutoff.

Potentially Mineable Resource Summary

For this PA, no mineable reserves have been estimated. Gustavson based the ultimate pit design on the contained measured (M), indicated (IND), and inferred (INF) resource blocks. Gustavson has therefore defined the resultant pit tonnages as the “potentially mineable mineral resource” (mineable resources) at Mt Todd. These mineable resources are summarized in TABLE 18-3 by phase for the base case ($600 per oz Au and $2.00 per lb Cu). The reported mineable resources summarized in TABLE 18-3 are calculated from the base case ultimate pit shell using the slightly lower cutoff grade resulting from lower unit processing costs and high gold recovery estimated by RDi.

The base case mineable resource is classified as summarized in TABLE 18-4.

TABLE 18-3
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Summary of Mineable Resources by Phase ($600 per oz Au and $2.00 per lb Copper Designed Pit)
December 2006

Mining Phase	"Ore" Tonnes (x 1000)	Average Gold Grade (gm/t))	Contained Gold (oz)	Waste Tonnes (x 1000)	Total Tonnes (x 1000)	Stripping Ratio (W:O)
1	41,500	1.257	1,677,600	69,500	111,000	1.7
2	37,900	0.902	1,099,300	79,000	116,900	2.1
3	28,300	0.422	383,000	38,600	66,900	1.4
TOTAL	**107,651.5**	**0.913**	**3,160,988**	**187,118**	**294,769**	**1.74**

TABLE 18-4
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Classification of Mineable Resources ($600 per oz Au and $2.00 per lb Copper Designed Pit)
December 2006

Class	"Ore" Tonnes (x 1000)	Average Gold Grade (gm/t))	Contained Gold (oz)	Waste Tonnes (x 1000)	Total Tonnes (x 1000)	Stripping Ratio (W:O)
Measured	20,521.1	0.902	595,036	-NA-	-NA-	-NA-
Indicated	41,182.9	0.908	1,202,307	-NA-	-NA-	-NA-
Inferred	45,947.5	0.923	1,363,645	-NA-	-NA-	-NA-
TOTAL	**107,651.5**	**0.913**	**3,160,988**	**187,118**	**294,769**	**1.74**

19.0 OTHER RELEVANT DATA AND INFORMATION

Gustavson is unaware of any other data and/or information that would be relevant to this report and is not contained in one of the SECTIONS of this report.

20.0 INTERPRETATION AND CONCLUSIONS

20.1 Interpretation

While much of the data used in the development of this report predate Canadian National Instrument NI43-101 standards for reporting, they are the result of well thought out and executed exploration programs. Additionally, it is Gustavson's opinion that most of the past work would meet current standards and those areas that do not meet current standards have been identified within the body of this report. The work was 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. Finally, the database and supporting data have been meticulously preserved and maintained and are readily available for confirmation of the results of the various studies.

20.2 Conclusions

It is Gustavson's opinion that the data used in support of and for the estimation of the geologic resources quoted in this Technical Report are compliant with CIM definitions and that the geologic resources presented meet the requirements of measured, indicated, and inferred resources under current CIM definitions.

21.0 RECOMMENDATIONS

Based on Gustavson's review of the database, previous studies and work products, and as an outgrowth of the geologic modelling and grade estimation work, Gustavson has developed the following list of recommendations for Vista's consideration.

Batman Deposit

Other issues that will still need to be addressed include:

1) Pegasus was aware of the problem of drill hole density within the Batman Deposit. According to former Pegasus management and based on personal communications, court documents, file data, etc., the decision to not drill out the lower portion of the Batman Deposit was based on economic considerations. Section 7.0 of the 1995 feasibility study detailed the decrease in drill hole density with depth. At the time of that study, there were 593 holes in the assay database of which 531 were used in the construction of the MRT block model. Reserve Services Group (RSG) reported that the drilling density in the Central area oxide and transition zone ore was generally 25 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 TWC's site visit, the drill hole database numbered 730 holes. It is not known if any holes were excluded from the Pegasus exploration models. Most of the new drilling that had been added since the 1994 MRT model was relatively shallow. TWC reviewed PGA's 50-meter drill sections through the Batman Deposit and saw that there was a marked decrease in drill hole spacing below 1000 RL and another sharp break below 900 RL. The drill hole spacing in the south of 1000 N on the 954 RL bench plan approached 80 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.

As presented later in this section, Vista has identified and designed a drilling program that will begin to address and correct theses identified deficiencies.

2) Another problem related to drilling is the preferred orientation of the drill holes. Most of the holes in the assay database are inclined to the west to capture the vein set which strikes N 10° 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 thrown out of the drill hole database for the 1994 MRT model because their assay results appeared to be too low compared to other hole orientations. If vertical hole orientations were actually underestimating the gold content during exploration drilling, the vertical and often wet blastholes, which are used for ore control pose a similar problem.

The Vista program presented later in this section has been designed to also begin to address and correct theses identified deficiencies.

Quigleys Deposit

The Quigleys Deposit is 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 an accurate resource estimate. Gustavson proposes that the following items be considered when preparing the work plan:

1) Surface mapping and subsequent re-interpretation of the footwall contact to the shear zone mineralization are recommended. Any additional structural complexity that results should, where appropriate, be used to refine the mineralized envelope upon which modelling 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 suitable envelope constraints.

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 resource calculations.

Other Mineralized Occurrences

Several other known mineral occurrences occur 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:

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

Water Management Recommendations

MWH has prepared the following recommendations for dealing with the water management issues at the Mt Todd Project site.

TABLE 21-1
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Proposed Water Management Program
December 2006

No.	Mitigation Methods	Cost Estimate (Aus$)
	Care and Maintenance Phase	
1	Catchment inflow diversion at RP3 (to extend the lifetime of the RP3 storage strategy and RP7 (to assist in drying and consolidation of tailings before carrying out a dam lift)	ND
2	Installation of monitoring instrumentation at Edith River gauging sites SW2 and SW4 to increase discharge from RP1 and improve the hydrological dataset	$20,000
3	Evaporation sprays at RP3 to extend the lifetime of the RP3 ARD storage strategy	$500,000 - $2,000,000
4	Water treatment (lime dosing) in a pond to allow year-round release of ARD excess and reduce the pit water removal requirements in advance of mining	Variable, dependant on location
5	Construction of a water treatment plant to allow year-round release of ARD excess and reduce the pit water removal requirements in advance of mining	$9,200,000
6	Release of water from the Raw Water Dam conjunctively with discharge of treated water to further dilute the discharge to environment	$0
	Operational / Closure Phases	
1	Continued application of care and maintenance mitigation methods as appropriate	As above
2	Wetland polishing of *moderately contaminated* waters prior to discharge	ND
3	Land application of *treated* wastewater to reduce sulphate levels before discharge	ND
4	Closure of the Heap Leach facility to remove one source of ARD generation	See conceptual closure plan (in prep.)
5	Incorporation of ARD generation considerations during further development of the waste rock dump.	ND

ND = Not Determined

Closure Recommendations

There are opportunities during the Mt Todd Project to conduct closure of a number of the facilities prior to or during operation, including the current HLP and TSF. Once the final raises of both TSFs are completed, then revegetation of the embankments can be initiated. In addition, it may be possible to close portions of the WRD, but this opportunity may be limited by the need for a selective waste rock placement program to help mitigate potential ARD.

As the closure plan develops, the following considerations should be made, some of which are discussed above:

- Immediate closure of the HLP;
- Early closure of existing TSF (once deposition complete);
- Locating and evaluating sources of borrow materials;
- A waste rock management strategy to reduce ARD concerns;
- Stockpiling benign waste materials for use in closure (e.g., for rock cover);
- Final placement of tailings in the TSF to minimize the need for regrading during closure;
- Vegetation test plots on TSF to determine its suitability as a growth medium, and/or amendment requirements; and
- Consideration of waste rock placement to facilitate a geomorphic slope (i.e., convex at the top and concave on the lower slopes); such designs are more erosionally stable and have a more "natural" appearance.

Major assumptions had to be made regarding the properties of the waste materials and soils that could be used for cover materials. Characterization of the waste and borrow materials which should include the physical and chemical properties should be initiated before the closure process can proceed beyond this conceptual level. The results from the characterization testing would then be used with climate and plant data to finalize the cover designs. Additional assumptions regarding the physical and erosional stability and the short and long-term water treatment requirements should also be checked using site-specific information.

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. This program is detailed in TABLE 21-2. 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 Gustavson's opinion that the proposed program is designed to address many of the 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 21-2 VISTA GOLD CORP. – MT TODD GOLD PROJECT Mt Todd Gold Project – Proposed Work Plan and Budget December 2006				
Description	**#/Units**	**Units**	**US$/Unit**	**Cost (US$)**
Infill & Metallurgical Drilling	7,000	m	US $150/m	1,050,000
Metallurgical Testing				25,000
Assess Previous Exploration on the mineral Leases				25,000
Geochemical Sampling				15,000
Geophysical Studies				15,000
Geologic Mapping and Prospecting				50,000
Technical Scoping Study on Batman				50,000
TOTAL				**1,230,000**

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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 Gustavson Associates, LLC at:

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   Suite D
   Boulder, Colorado 80301

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 ECONOMIC ASSESSMENT OF THE MOUNT TODD GOLD PROJECT, NORTHERN TERRITORY, AUSTRALIA.*" and dated 29 December 2006 (the "Technical Report"). I visited the subject property on June 20, 2005.
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 not had prior involvement with Vista Gold Corp. 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 two- (2) kilometer 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 Technical 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 Technical Report.

Dated this 29th Day of December, 2006.

_______________.
Signature of Qualified Person

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

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29 Dec 2006

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

24.1 Mining

The preliminary assessment of the economic potential of reopening the Mt Todd Project was based on the geologic model presented earlier in this report. The base case ultimate pit was designed from the $600 per Au oz , 55-degree average pit slope, optimized pit for measured, indicated, and inferred resources. The mine plan assumes that all measured, indicated, and inferred resources are mineable.

Production Schedule

Intermediate phases were developed during the LG pit optimization runs using a gold price of $400 and $500 per Au oz for Phases 1 and 2, respectively. Phase 3 is the ultimate $600 per Au oz pit. Each of the phases was designed with smoothed pit walls and haul roads, as shown in FIGURES 18-2, 18-3, and 18-4**.** Mining Phases 1 and 2 during the early years enhance the ore grades and improves economic results. The production rate for the cash flow analyses is 10,650,000 tonnes per year (30,000 tonnes per day), and a conceptual production schedule was developed to provide a ten-year mine life with two pre-production years for site construction activities, as shown in TABLE 24-1.

TABLE 24-1
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Production Schedule
December 2006

Year	Ore Tonnes (x 1000)	Avg. Grade (g Au/tonne)	Waste Tonnes (x 1000)	Stripping Ratio (W:O)
-2	0		0	
-1	0		0	
1	10,650	1.20	22,000	2.1
2	10,650	1.15	22,000	2.1
3	10,650	1.10	22,000	2.1
4	10,650	1.00	22,000	2.1
5	10,650	1.00	22,000	2.1
6	10,650	0.90	21,000	2.0
7	10,650	0.80	21,000	2.0
8	10,650	0.70	21,000	2.0
9	10,650	0.70	10,000	0.9
10	10,650	0.60	4,000	0.4
	106,500	**0.91**	**187,000**	**1.8**

Production scheduling parameters are summarized in TABLE 24-2.

TABLE 24-2 VISTA GOLD CORP. – MT TODD GOLD PROJECT Mine Production Scheduling Parameters December 2006	
Daily Mill Feed Rate (tonnes)	30,000
Annual Ore Production Rate (tonnes)	10,650,000
Mine Operating Hours per Shift	12
Mine Operating Shifts per Day	2
Mine Operating Days per Week	7
Scheduled Mine Operating Days per Year	355
Number of Mine Crews	4

Haul Roads

The plant facilities are located to the east of the open pit, and the mine waste stockpile is located to the southeast; therefore, the primary haul road and pit access is from the east side of the pit. Haul roads are designed at 27-meters wide to accommodate the proposed equipment fleet, ditches, and berms. The maximum designed road gradient is 10 percent except for the bottom three benches, where road widths are narrowed for one-way traffic and steepened to a 15 percent gradient.

Waste Stockpile

As shown in FIGURE 24-1, the mine waste stockpile has been designed to accommodate 25 percent swell of the waste rock over the expected ten-year mine life. Average side slopes are designed at 3H:1V to facilitate reclamation and mine closure. The haul roads are designed at a 10 percent gradient and are also 27-meters wide. Total capacity of the designed stockpile is over 196 million tonnes. No segregation of mine waste has been anticipated in this study.

Mine Equipment Selection and Requirements

The 30,000 tonne-per-day (tpd) mill feed requirement provided the basis for equipment selection and fleet requirements. Mining will take place on 12-meter benches, and a mid-sized fleet of 203-mm diameter blasthole drills, 18-cubic meter hydraulic front shovels, and 141-tonne haul trucks was selected. The mine will work two 12-hour shifts per day, 355 days per year. Equipment requirements are estimated based on 10 operating hours per shift, which allows time for lunch, fueling, safety training and personal breaks during the shift. Project capital cost estimates assume all new equipment. TABLE 24-3 summarizes the expected fleet size and annual operating shifts for the major mine equipment.

<u>Drilling</u>

Crawler-mounted, 25,000-kg pulldown, down-the-hole hammer drills were selected for primary blasthole drilling. Drill productivity is based on a blasthole diameter of 203 mm and 14-meter depth, assuming 2 meters of subgrade drilling. The drill pattern will be about 6 meters by 6 meters with about six meters of stemming, based on an assumed powder factor of 0.19 kg

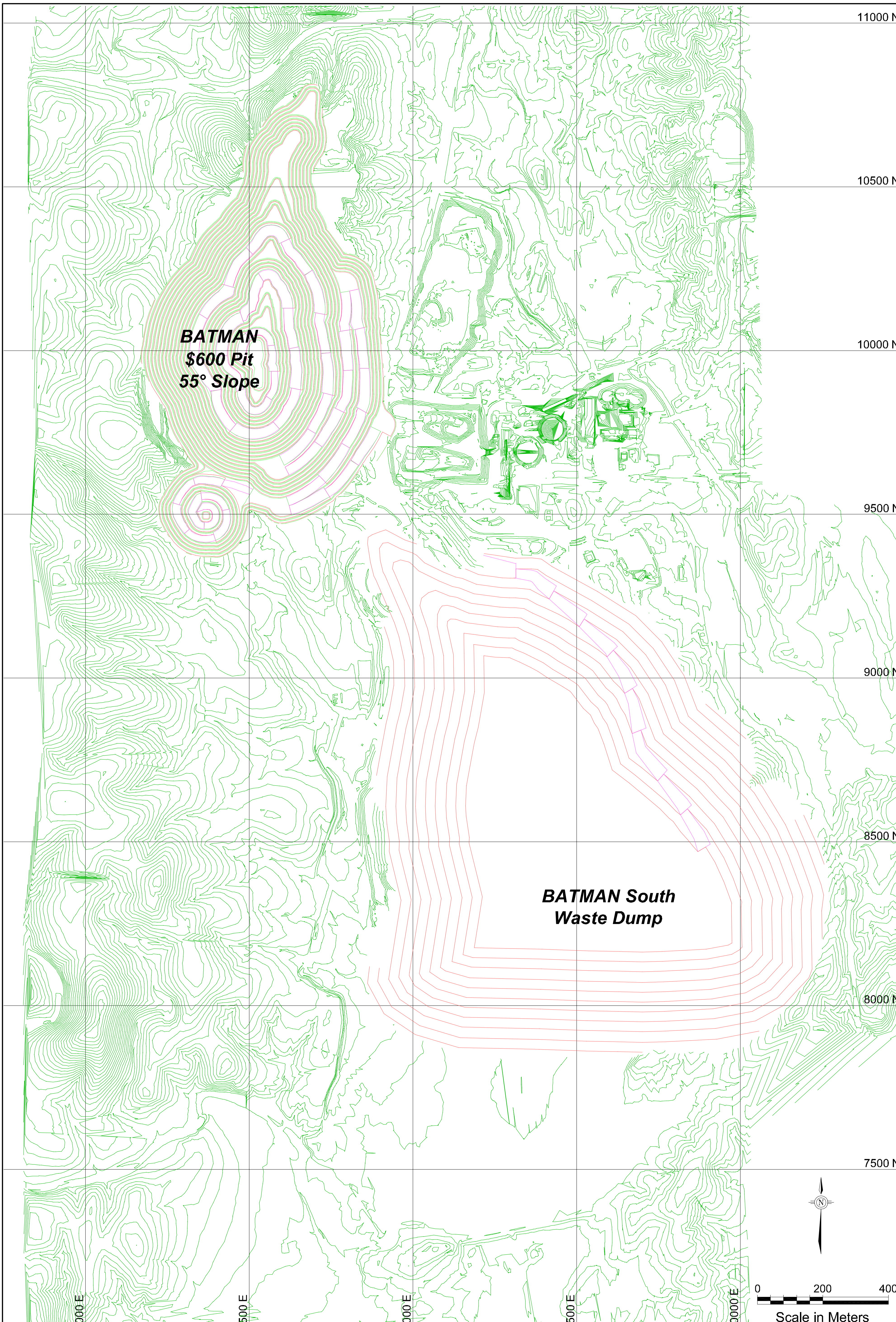

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FIGURE 24-1
SITE LAYOUT -$600AU PIT WITH 55° SLOPE
& MINE WASTE ROCK STOCKPILE

TABLE 24-3
VISTA GOLD CORP. - MT. TODD GOLD PROJECT
Equipment Operating Shifts and Fleet Requirements
December 2006

	Preprod	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Material Handled (ktonnes)												
Ore	0	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	10,650	106,500
Waste	0	22,000	22,000	22,000	22,000	22,000	21,000	21,000	21,000	10,000	4,000	187,000
TOTAL	0	32,650	32,650	32,650	32,650	32,650	31,650	31,650	31,650	20,650	14,650	293,500
Scheduled Shifts	0	720	720	720	720	720	720	720	720	720	720	7,200
Major Equipment Operating Shifts Required												
Rotary Drills, 203 mm	0	2,674	2,674	2,674	2,674	2,674	2,674	2,674	2,674	2,674	2,674	26,740
Hydraulic Shovels, 18 cu m	0	1,296	1,296	1,296	1,296	1,296	1,296	1,296	1,296	1,296	1,296	12,960
Loaders , 12 cu m	0	360	360	360	360	360	360	360	360	360	360	3,600
Trucks, 144 t	0	6,120	6,120	6,120	6,120	6,120	6,120	6,120	6,120	6,120	6,120	61,200
Track Drill, 89 mm	0	180	180	180	180	180	180	180	180	180	180	1,800
Track Dozers, D9R	0	1,620	1,620	1,620	1,620	1,620	1,620	1,620	1,620	1,620	1,620	16,200
RT Dozers, 834 B	0	540	540	540	540	540	540	540	540	540	540	5,400
Graders, 16H	0	720	720	720	720	720	720	720	720	720	720	7,200
Water Truck, 50,000 ltr	0	720	720	720	720	720	720	720	720	720	720	7,200
Fleet Size												
Rotary Drills, 203 mm	0	4	4	4	4	4	4	4	4	4	4	
Hydraulic Shovels, 18 cu m	0	2	2	2	2	2	2	2	2	2	2	
Loaders , 12 cu m	0	1	1	1	1	1	1	1	1	1	1	
Trucks, 144 t	0	9	9	9	9	9	9	9	9	9	9	
Track Drill, 89 mm	0	1	1	1	1	1	1	1	1	1	1	
Track Dozers, D9R	0	3	3	3	3	3	3	3	3	3	3	
RT Dozers, 834 B	0	1	1	1	1	1	1	1	1	1	1	
Graders, 16H	0	1	1	1	1	1	1	1	1	1	1	
Water Truck, 50,000 ltr	0	1	1	1	1	1	1	1	1	1	1	

blasting agent per tonne of rock. Most of the blasting will be done using ANFO. Assuming an average penetration rate of 23 meters per hour, drill productivities are expected to be 157 meters per shift. A crawler-mounted air track drill will be required for pioneering and road construction, as well as for secondary breakage within the pit. This rotary percussion drill will drill holes of about 89 mm diameter. Four 203-mm rotary blasthole drills and one air track drill will be required when the mine is in full production.

Loading

Two 18-cubic meter hydraulic front shovels were selected for the primary loading units for ore and waste. They will be backed up by a 12-cubic meter front-end loader. The shovel should be able to load the trucks in four passes, and productivity is projected at about 33,000 tonnes per shift.

Hauling

Ore and waste haulage will be handled by nine (eight plus one spare) 141-tonne off-highway haul trucks at the maximum mining rate of about 31.6 million tonnes per year. Each haul truck is expected to haul over 7,000 tonnes per shift (averaged over the mine life). Although truck productivities will decline over time as the pit deepens and the waste stockpile is higher and farther away, the projections in this PA are based on the average for the short mine life.

Support Equipment

Auxiliary equipment are required for miscellaneous activities such as waste stockpile, haul road, and bench maintenance, dust control, storm water management, etc. Some of this equipment will serve the entire mine and mill complex. TABLE 24-4 lists the major support equipment assigned to mining operations.

TABLE 24-4
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Mine Support Equipment
December 2006

Equipment Type	No.
Air Track Drill (89 mm)	1
Rubber-Tired Dozer	1
Track Dozers	3
Motor Grader	1
Water Truck (50,000 liter)	1

Mine Operations

Mine manpower requirements were estimated on the basis of working two 12-hour shifts per day, 7 days per week, 52 weeks per year. A standard rotating four-crew work schedule has been assumed for the PA. Each person would average 42 hours per week. Operations and maintenance levels are increased by 7 percent to account for vacation, sickness, and other absenteeism (VSA). TABLE 24-5 lists projected hourly mine operations personnel. At peak levels, a total of 107 hourly mine operations personnel will be required, mostly distributed among the four work crews.

Mine Maintenance

Most of the mine equipment maintenance will take place on site, including repairs and preventative maintenance. Maintenance personnel will be assigned to the four rotating crews. At peak operations, the PA assumes a total mine maintenance requirement of about 50 personnel, as shown on TABLE 24-5.

Mine Supervision and Technical Services

TABLE 24-6 lists the mine supervision and technical services personnel for the Mt Todd Project. It is presumed that all supervisory and technical staff will be Australian nationals. Approximately 26 personnel will be required in operations and maintenance supervision, and 17 personnel will be required for mine technical services.

24.2 General and Administrative

TABLE 24-7 lists the general and administrative personnel required for the project. These positions are responsible for overall site operations such as site management, accounting, purchasing, human resources, safety, security, and environmental. Approximately 31 personnel are required for these positions.

24.3 Infrastructure

The Mt Todd property contains a number of known occurrences of gold, which have been explored and/or exploited to various degrees over the years. The largest and best-known deposits are the Batman and Quigleys. Both of these have had historic mining, with Batman having the most production and exploration completed. Some of the infrastructure, such as access roads, buildings, radio tower, storm water ponds, tailings disposal facility, and local line power are available from past operations.

Access

The Mt Todd Project is located 50 kilometers northwest of Katherine, and approximately 250 kilometers southeast of Darwin in the NT of Australia. Access to the mine is via high quality, two-lane paved roads from the Stuart Highway, the main artery within the territory.

TABLE 24-5
VISTA GOLD CORP. - MT. TODD GOLD PROJECT
Mine Hourly Personnel
December 2006

Position	Preprod	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Operations											
Driller, blasthole	0	16	16	16	16	16	16	16	16	16	16
Driller Helper, blasthole	0	4	4	4	4	4	4	4	4	4	4
Driller, track drill	0	1	1	1	1	1	1	1	1	1	1
Blaster	0	1	1	1	1	1	1	1	1	1	1
Blaster Helper	0	3	3	3	3	3	3	3	3	3	3
Shovel/Loader Operator	0	8	8	8	8	8	8	8	8	8	8
Truck Driver	0	34	34	34	34	34	34	34	34	34	34
Track Dozer Operator	0	12	12	12	12	12	12	12	12	12	12
RT Dozer Operator	0	4	4	4	4	4	4	4	4	4	4
Grader Operator	0	4	4	4	4	4	4	4	4	4	4
Water Truck Driver	0	4	4	4	4	4	4	4	4	4	4
Pumpman/Utilityman	0	1	1	1	1	1	1	1	1	1	1
Laborer/Trainee	0	8	8	8	8	8	8	8	8	8	8
VSA Operator*	0	5	5	5	5	5	5	5	5	5	5
VSA Laborer/Trainee**	0	2	2	2	2	2	2	2	2	2	2
Subtotal	0	107	107	107	107	107	107	107	107	107	107
Maintenance											
Heavy Equip. Mechanic	0	12	13	14	14	16	16	16	16	16	16
Welder/Mechanic	0	6	6	6	6	7	7	7	7	7	7
Electrician/Instrumentman	0	2	2	2	2	2	2	2	2	2	2
Lubeman/PM Mechanic	0	8	8	8	8	8	8	8	8	8	8
Tireman	0	3	3	3	3	3	3	3	3	3	3
Machinest	0	1	1	1	1	1	1	1	1	1	1
Utilityman	0	4	4	4	4	4	4	4	4	4	4
Laborer/Trainee	0	6	6	6	6	6	6	6	6	6	6
VSA Mechanic*	0	2	2	2	2	2	2	2	2	2	2
VSA Laborer**	0	1	1	1	1	1	1	1	1	1	1
Subtotal	0	45	46	47	47	50	50	50	50	50	50
TOTALS	0	152	153	154	154	157	157	157	157	157	157

* 5% of total for Vacations, Sickness, and Absenteeism (VSA)

** 2% of total for Vacations, Sickness, and Absenteeism (VSA)

TABLE 24-6
VISTA GOLD CORP. - MT. TODD GOLD PROJECT
Mine Salaried Personnel
December 2006

Position	Preprod	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Supervision											
Mine Superintendent	0	1	1	1	1	1	1	1	1	1	1
General Mine Foreman	0	1	1	1	1	1	1	1	1	1	1
Drill & Blast Foreman	0	2	2	2	2	2	2	2	2	2	2
Mine Shift Foremen	0	8	8	8	8	8	8	8	8	8	8
Maintenance Superintendent	0	1	1	1	1	1	1	1	1	1	1
Maintenance Planner	0	1	1	1	1	1	1	1	1	1	1
Maintenance Shift Foremen	0	8	8	8	8	8	8	8	8	8	8
Secretary/Clerk	0	4	4	4	4	4	4	4	4	4	4
Subtotal	0	**26**	**26**	**26**	**26**	**26**	**26**	**26**	**26**	**26**	**26**
Technical Services											
Chief Mine Engineer	0	1	1	1	1	1	1	1	1	1	1
Sr. Mine Planning Eng.	0	1	1	1	1	1	1	1	1	1	1
Ore Control Engineer	0	1	1	1	1	1	1	1	1	1	1
Mine Engineer	0	1	1	1	1	1	1	1	1	1	1
Sr. Mine Geologist	0	1	1	1	1	1	1	1	1	1	1
Mine Geologist	0	2	2	2	2	2	2	2	2	2	2
Surveyor	0	1	1	1	1	1	1	1	1	1	1
Rodman	0	1	1	1	1	1	1	1	1	1	1
Eng Tech/Ore Control	0	8	8	8	8	8	8	8	8	8	8
Subtotal	0	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**17**	**17**
TOTAL	**0**	**43**	**43**	**43**	**43**	**43**	**43**	**43**	**43**	**43**	**43**

TABLE 24-7
VISTA GOLD CORP. - MT. TODD GOLD PROJECT
General and Administrative Costs
December 2006

STAFFING				
	No. Emp	**Salary ($/Yr)**	**Burden at 48% ($/Yr)**	**Total Cost ($/Yr)**
Management				
General Site Manager	1	152,000	72,960	224,960
Administrative Assistant	1	49,430	23,726	73,156
Subtotal				298,116
Accounting				
Administrative Manager	1	79,848	38,327	118,175
Senior Accountant	2	64,639	31,027	191,331
Payroll	1	45,627	21,901	67,528
Accounts Receivable	1	45,627	21,901	67,528
Subtotal				444,562
Purchasing				
Purchase Agent	1	68,441	32,852	101,293
Warehouseman	3	34,221	16,426	151,941
Inventory Control	2	34,221	16,426	101,294
Buyer	1	53,232	25,551	78,783
Subtotal				433,311
Human Resources				
HR Manager	1	72,243	34,677	106,920
Clerk	2	45,627	21,901	135,056
Subtotal				241,976
Safety, Security & Environmental				
Safety Manager	1	64,639	31,027	95,666
Safety Trainer/Inspector	2	64,639	31,027	191,331
Security	8	38,023	18,251	450,192
Environmental Manager	1	68,441	32,852	101,293
Environmental Technicians	2	49,430	23,726	146,313
Subtotal				984,795
TOTAL G&A PAYROLL				**2,402,761**
EXPENSES ($/Yr)				**2,156,000**
TOTAL G&A Cost	**($)**			**4,558,761**
	($/tonne mined)			**0.14**

sufficiently close to the city of Katherine to allow for an easy commute for workers. Because there has been both historic and relatively 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.

Power and Water

The property has an existing high-pressure gas line and an electric line that was used by previous operators. In addition, wells for potable water and a dam for process water are also located on the site.

Communications

The existing radio tower will allow sufficient communications for the project.

Site Buildings

The site has some existing buildings in place, including a process building, a small shop/office, and two electrical buildings. At full operation, the project will require construction of an administration building, a truck shop, a warehouse building and other storage, new processing facilities, and a fuel and lube facility.

24.4 Capital Costs

Mine Capital Costs

TABLE 24-8 summarizes the mine equipment capital expenditure schedule over the life of the project. The annual fleet requirements were listed in TABLE 24-3. Pre-production and Year 1 expenditures total approximately US $34.5 million. Major and support operations equipment, maintenance equipment, and other miscellaneous support equipment such as pumps, light plants, shop equipment, and pickups are included. Spare parts were estimated at eight percent of the total capital. A contingency factor of five percent is included to cover small items not listed in the table. Because the mine will operate within a pre-existing open pit, no pre-stripping is required. No ANFO trucks and blasthole loading equipment, including storage silos and explosive magazine, are included because these items will be provided by contract services.

Processing Capital Costs

TABLE 24-9 summarizes RDi's capital cost estimates for construction of the 30,000 tonne-per-day crushing and processing facilities. The total start-up cost is estimated to total US $179 million in Pre-production Years 1 and 2.

The following methodology was used for developing the capital cost for the project at 10.65 MM tonnes per year capacity:

TABLE 24-8
VISTA GOLD CORP. - MT. TODD GOLD PROJECT
Mine Equipment Capital Expenditure Schedule
December 2006

	Unit Cost ($000)	Preprod	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Major Equipment													
Rotary Drills, 203 mm, 50K	860	1720	1720	0	0	1720	1720	0	1720	860	0	0	9460
Hydraulic Shovels, 18 cu m	3090	6180	0	0	0	6180	0	0	6180	0	0	0	18540
Loaders , 12 cu m	1390	0	1390	0	0	0	0	1390	0	0	0	0	2780
Trucks, 141 t	1650	9900	4950	0	0	0	4950	4950	4950	0	0	0	29700
Track Drill, 89 mm	150	150	0	0	0	0	0	0	0	0	0	0	150
Track Dozers, D9R	580	1160	580	0	0	0	0	580	1160	0	0	0	3480
RT Dozers, 834 B	550	550	0	0	0	0	550	0	0	0	0	0	1100
Graders, 16H	610	610	0	0	0	0	610	0	0	0	0	0	1220
Water Truck, 50,000 ltr	760	760	0	0	0	0	760	0	0	0	0	0	1520
Support Equipment													
Backhoe/Loader, 2 cy	240	240	0	0	0	0	0	0	0	0	0	0	240
Crane, 40 ton	270	270	0	0	0	0	0	0	0	0	0	0	270
Forklift	30	30	0	0	0	0	0	0	0	0	0	0	30
Fuel/Lube Truck	120	120	0	0	0	0	0	0	0	0	0	0	120
Service Truck	120	240	0	0	0	0	0	0	0	0	0	0	240
Tire Truck	120	120	0	0	0	0	0	0	0	0	0	0	120
Light Plant	20	80	0	0	0	0	0	80	0	0	0	0	160
Crew Van	30	60	0	0	0	0	0	60	0	0	0	0	120
Pickups	20	300	0	0	100	200	0	0	200	0	0	0	800
Welder/Generator	40	40	0	0	0	0	0	0	0	0	0	0	40
*													
Miscellaneous													
Shop Equipment	200	200	0	0	0	0	0	0	0	0	0	0	200
Pumps, 125 HP	30	60	0	0	0	0	0	0	0	0	0	0	60
Engineering & Office Equipment	150	150	0	0	0	0	0	0	0	0	0	0	150
Water Storage	50	50	0	0	0	0	0	0	0	0	0	0	50
Base Radio & GPS System	100	100	0	0	0	0	0	0	0	0	0	0	100
Other	100	100	100	100	100	100	100	40	100	100	100	100	1040
Equipment Subtotal		23,190	8,740	100	200	8,200	8,690	7,100	14,310	960	100	100	71690
Spares Inventory	0.08	1,855	699	8	16	656	695	568	1,145	77	8	8	5735
TOTAL		**25,045**	**9,439**	**108**	**216**	**8,856**	**9,385**	**7,668**	**15,455**	**1,037**	**108**	**108**	**77,425**

***Note:** **ANFO Slurry Truck, Stemming/Dump Truck, Powder/Flatbed Truck, AN Storage Bin, Powder Magazine, Cap Magazine are all the responsibility of the BLASTING CONTRACTOR**

- Majority of the major equipment was sized for processing 10.65 MM ton per year.

- Percentage factors were used for freight (20%), GST (10%), installation of equipment (20%), concrete (10%), structural steel (15%), piping (30%), electrical distribution ($400/kw) and indirect (25%) of purchased equipment cost in the capital cost estimation.

- Indirect costs were estimated at (25%) of purchased equipment cost.

- Instrumentation was estimated at 20% of electrical distribution.

- Percentage factors were also used for EPCM (15%) and spare parts (5%).

- Contingency of 25% was applied to the capital cost.

TABLE 24-9
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Capital Cost Estimate for Processing 10.65 MM Tonnes Per Year
December 2006

Area	Description	Cost, US$
100	Crushing Circuit	19,444,500
200	Coarse Ore Stockpile & Reclaim	2,097,000
300	Grinding	32,619,000
400	Flotation	5,467,200
500	Carbon-in-Leach	865,000
600	Copper/Gold Stripping Circuit	268,800
700	Gold Electrowinning Circuit	274,400
800	Copper Electrowinning	274,400
900	Acid Wash Circuit	110,600
1000	Carbon Regeneration Circuit	305,200
1100	Refinery	333,200
1200	Carbon Conditioning Circuit	134,400
1200	Cyanide Destruction	328,000
1400	Copper Concentrate Thickening & Filtration	204,400
1500	Tailings Pipeline & Reclaim Water Barge & Piping	4,965,300
1600	Sulfide Tailings Thickening & Filtration	2,100,000
	Subtotal Direct Cost	**69,791,400**
	Indirect Cost @ 25% *Purchased Equipment	10,653,200
	Concrete @ 10% *Purchased Equipment	4,261,280
	Structure Steel @ 15% *Purchased Equipment	6,391,920
	Piping @ 30% *Installed Equipment	12,783,840
	Electrical Distribution @ $400 per kW	18,759,800
	Instrumentation @ 20% *Electrical Distribution	3,751,960
	Subtotal Direct + Indirect Cost	**126,393,400**
	EPCM @ 15% of Above Costs	18,959,010
	Contingency @ 25% of Above Costs	31,598,350
	Spare Parts @ 5% of Purchased Equipment	2,130,640
	Mobile Equipment Allowance	0
	TOTAL ESTIMATED CAPITAL COST	**179,081,000**

Other Capital Costs

Startup capital for other project components are summarized in TABLE 24-10 and discussed elsewhere in this report.

TABLE 24-10
VISTA GOLD CORP. – MT TODD GOLD PROJECT
Other Capital Costs
December 2006

Item	Start-up Cost (US$ * 1000)
Access and Site Preparation	450
Surface Facilities	1,950
Site Infrastructure	1,650
General Surface Mobile Equipment	3,296
Pit Dewatering	5,300
Mine Dewatering	1,750
Tailings Disposal Facilities	10,741
Open Pit Mine Development	900
Permitting	500
General Project Management	2,000
TOTAL	**28,537**
Total Estimated Closure	**30,500**

MWH estimates the capital cost of expanding the existing tailings disposal facility to be $10.74 million during Years PP1, 1, and 2.

Capital Cost Summary

The estimated capital expenditures for the life of the mine are summarized on TABLE 24-11.

TABLE 24-11
VISTA GOLD CORP. - MT. TODD GOLD PROJECT
Capital Cost Summary (US $000)
December 2006

Year	Access & Site Prep	Plant & Facilities	Infrastructure	General Surface Mobile Equip.	Concentrator Tailings Disposal	Mine Development	Mine Equipment	Permitting/ Mine Closure	G&A, OH, Contingency Overhauls, Eq. Repl	TOTAL
PP2	300	950	1,150	1,300	79,800	5,550	713	500	10,126	100,389
PP1	150	1,000	500	486	106,400	2,400	24,332		14,627	149,895
1				1,510	1,445		9,439		1,239	13,633
2					1,658		108		177	1,943
3					7,904		216		812	8,932
4					6296		8,856	2,323	1,748	19,223
5					5,994		9,385	2,556	1,794	19,729
6					1,955		7,668		962	10,585
7					3,460		15,455		1,891	20,806
8					1,277		1,037		231	2,545
9							108		11	119
10							108		11	119
11								581	58	639
12								6,214	621	6,835
13								6,215	622	6,837
14								5,111	511	5,622
15								7,500	750	8,250
TOTAL	**450**	**1,950**	**1,650**	**3,296**	**216,189**	**7,950**	**77,425**	**31,000**	**36,191**	**376,101**

24.5 Operating Cost Estimate

Mine Operating Costs

Mine operating costs were estimated based on the equipment operating and manpower requirements discussed above. Mining costs include pit and waste stockpile operations, road maintenance, in-pit storm water management, mine supervision, and technical services from the mine to the primary crusher facilities located on the east side of the pit. Key operating cost parameters are as follows:

Diesel Fuel	$0.54 per liter
Electric Power	$0.113 per kWh
ANFO	$0.74 per kg

Hourly labor rates range from $11.98 to $25.05 per hour plus a burden rate of 48 percent, as shown in TABLE 24-12. The typical hourly labor cost is about $0.47 per tonne of material mined (ore or waste). As shown in TABLE 24-13, salaried labor costs are typically about $0.14 per tonne of material mined. Hourly and salaried labor rates are based on local prevailing rates in north-central Australia.

Unit operating costs (Cost per Operating Hour) for equipment were developed from current cost estimating manuals. Operating hours were estimated based on expected equipment productivity and two-shift-per-day work schedules discussed previously. Hourly operating costs include costs for fuel and lubes, tires, wear and maintenance parts, power, and supplies. These costs do not include maintenance and operating labor, which are included in the labor cost estimate. TABLE 24-14 summarizes equipment operating costs, typically $0.45 per tonne material mined.

In addition to the above costs for labor and equipment, this PA assumes that blasting operations will be handled by a contractor for $0.15 per tonne. Typical total mining costs are summarized below:

Hourly Labor	$0.47 per tonne material mined
Salaried Labor	$0.14 per tonne material mined
Equipment Operating	$0.45 per tonne material mined
Blasting Contractor	$0.15 per tonne material mined
Total	$1.21 per tonne material mined

Mine operating costs in the cash flow analysis are based on an overall average of $1.21 per tonne material mined over the life of the project.

Plant Operating Costs

Plant operating costs as provided by RDi are summarized in TABLE 24-15. TABLE 24-16 summarizes estimated labor requirements for the plant.

TABLE 24-12
VISTA GOLD CORP. - MT. TODD GOLD PROJECT
Mine Hourly Labor Cost (US $000)
December 2006

	Wage Rate ($US/Hr)													
Position	**Base**	**Fr @ 48%**	**Total**	**Preprod**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**
Operations														
Driller, blasthole	45.55	21.86	67.41	0	2,071	2,071	2,071	2,071	2,071	2,071	2,071	2,071	2,071	2,071
Driller Helper, blasthole	27.73	13.31	41.04	0	315	315	315	315	315	315	315	315	315	315
Driller, track drill	33.67	16.16	49.83	0	96	96	96	96	96	96	96	96	96	96
Blaster	45.55	21.86	67.41	0	129	129	129	129	129	129	129	129	129	129
Blaster Helper	27.73	13.31	41.04	0	236	236	236	236	236	236	236	236	236	236
Shovel/Loader Operator	45.55	21.86	67.41	0	1,035	1,035	1,035	1,035	1,035	1,035	1,035	1,035	1,035	1,035
Truck Driver	35.65	17.11	52.76	0	3,444	3,444	3,444	3,444	3,444	3,444	3,444	3,444	3,444	3,444
Track Dozer Operator	35.65	17.11	52.76	0	1,216	1,216	1,216	1,216	1,216	1,216	1,216	1,216	1,216	1,216
RT Dozer Operator	35.65	17.11	52.76	0	405	405	405	405	405	405	405	405	405	405
Grader Operator	35.65	17.11	52.76	0	405	405	405	405	405	405	405	405	405	405
Water Truck Driver	35.65	17.11	52.76	0	405	405	405	405	405	405	405	405	405	405
Pumpman/Utilityman	33.67	16.16	49.83	0	96	96	96	96	96	96	96	96	96	96
Laborer/Trainee	21.78	10.45	32.23	0	495	495	495	495	495	495	495	495	495	495
VSA Operator*	35.65	17.11	52.76	0	507	507	507	507	507	507	507	507	507	507
VSA Laborer/Trainee**	21.78	10.45	32.23	0	124	124	124	124	124	124	124	124	124	124
Subtotal				0	**10,980**	**10,980**	**10,980**	**10,980**	**10,980**	**10,980**	**10,980**	**10,980**	**10,980**	**10,980**
Maintenance														
Heavy Equip. Mechanic	35.65	17.11	52.76	0	1,216	1,317	1,418	1,418	1,621	1,621	1,621	1,621	1,621	1,621
Welder/Mechanic	35.65	17.11	52.76	0	608	608	608	608	709	709	709	709	709	709
Electrician/Instrumentman	35.65	17.11	52.76	0	203	203	203	203	203	203	203	203	203	203
Lubeman/PM Mechanic	35.65	17.11	52.76	0	810	810	810	810	810	810	810	810	810	810
Tireman	35.65	17.11	52.76	0	304	304	304	304	304	304	304	304	304	304
Machinest	35.65	17.11	52.76	0	101	101	101	101	101	101	101	101	101	101
Utilityman	35.65	17.11	52.76	0	405	405	405	405	405	405	405	405	405	405
Laborer/Trainee	23.76	11.40	35.16	0	405	405	405	405	405	405	405	405	405	405
VSA Mechanic*	35.65	17.11	52.76	0	213	218	223	223	238	238	238	238	238	238
VSA Laborer**	23.76	11.40	35.16	0	57	58	59	59	63	63	63	63	63	63
Subtotal				0	**4,321**	**4,429**	**4,537**	**4,537**	**4,860**	**4,860**	**4,860**	**4,860**	**4,860**	**4,860**
TOTALS				0	**15,301**	**15,409**	**15,517**	**15,517**	**15,840**	**15,840**	**15,840**	**15,840**	**15,840**	**15,840**
		$/Tonne Mat.			0.47	0.47	0.48	0.48	0.49	0.50	0.50	0.50	0.77	1.08

* 5% of total for Vacations, Sickness, and Absenteeism (VSA)
** 2% of total for Vacations, Sickness, and Absenteeism (VSA)

TABLE 24-13
VISTA GOLD CORP. - MT. TODD GOLD PROJECT
Mine Salaried Labor Cost (US $000)
December 2006

Position	Salary (US$/Yr) Base	Fr @ 48%	Total	Preprod	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Supervision														
Mine Superintendent	106000	50880	156880	0	156.88	156.88	156.88	156.88	156.88	156.88	156.88	156.88	156.88	156.88
General Mine Foreman	87000	41760	128760	0	128.76	128.76	128.76	128.76	128.76	128.76	128.76	128.76	128.76	128.76
Drill & Blast Foreman	87000	41760	128760	0	258	258	258	258	258	258	258	258	258	258
Mine Shift Foremen	87000	41760	128760	0	1,030	1,030	1,030	1,030	1,030	1,030	1,030	1,030	1,030	1,030
Maintenance Superintendent	87000	41760	128760	0	129	129	129	129	129	129	129	129	129	129
Maintenance Planner	80000	38400	118400	0	118	118	118	118	118	118	118	118	118	118
Maintenance Shift Foremen	76000	36480	112480	0	900	900	900	900	900	900	900	900	900	900
Secretary/Clerk	49000	23520	72520	0	290	290	290	290	290	290	290	290	290	290
Subtotal				0	3,010	3,010	3,010	3,010	3,010	3,010	3,010	3,010	3,010	3,010
Technical Services														
Chief Mine Engineer	106000	50880	156880	0	156.88	156.88	156.88	156.88	156.88	156.88	156.88	156.88	156.88	156.88
Sr. Mine Planning Eng.	95000	45600	140600	0	140.6	140.6	140.6	140.6	140.6	140.6	140.6	140.6	140.6	140.6
Ore Control Engineer	90000	43200	133200	0	133	133	133	133	133	133	133	133	133	133
Mine Engineer	90000	43200	133200	0	133	133	133	133	133	133	133	133	133	133
Sr. Mine Geologist	95000	45600	140600	0	141	141	141	141	141	141	141	141	141	141
Mine Geologist	90000	43200	133200	0	266	266	266	266	266	266	266	266	266	266
Surveyor	64000	30720	94720	0	95	95	95	95	95	95	95	95	95	95
Rodman	49000	23520	72520	0	73	73	73	73	73	73	73	73	73	73
Eng Tech/Ore Control	49000	23520	72520	0	580	580	580	580	580	580	580	580	580	580
Subtotal				0	1,718	1,718	1,718	1,718	1,718	1,718	1,718	1,718	1,718	1,718
TOTAL				0	**4,729**	**4,729**	**4,729**	**4,729**	**4,729**	**4,729**	**4,729**	**4,729**	**4,729**	**4,729**
		$/Tonne Material			0.14	0.14	0.14	0.14	0.14	0.15	0.15	0.15	0.23	0.32

TABLE 24-14
VISTA GOLD CORP. - MT. TODD GOLD PROJECT
Major Mine Equipment Operating Costs ($000)
December 2006

Equipment Item	$/Op Hour	Preprod	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Major Equipment													
Rotary Drills, 203 mm, 50K	77	0	2,059	2,059	2,059	2,059	2,059	2,059	2,059	2,059	2,059	2,059	20,590
Hydraulic Shovels, 18 cu m	200	0	2,592	2,592	2,592	2,592	2,592	2,592	2,592	2,592	2,592	2,592	25,920
Loaders , 12 cu m	94	0	338	338	338	338	338	338	338	338	338	338	3,384
Trucks, 144 t	117	0	7,160	7,160	7,160	7,160	7,160	7,160	7,160	7,160	7,160	7,160	71,604
Track Drill, 89 mm	39	0	70	70	70	70	70	70	70	70	70	70	702
Track Dozers, D9R	44	0	713	713	713	713	713	713	713	713	713	713	7,128
RT Dozers, 834 B	48	0	259	259	259	259	259	259	259	259	259	259	2,592
Graders, 16H	41	0	295	295	295	295	295	295	295	295	295	295	2,952
Water Truck, 50,000 ltr	86	0	619	619	619	619	619	619	619	619	619	619	6,192
Support Equipment													
Backhoe/Loader, 2 cy	19	0	139	139	139	139	139	139	139	139	139	139	1,436
Crane, 40 ton	25	0	45	45	45	45	45	45	45	45	45	45	463
Forklift	10	0	36	36	36	36	36	36	36	36	36	36	371
Fuel/Lube Truck	10	0	70	70	70	70	70	70	70	70	70	70	724
Service Truck	9	0	136	136	136	136	136	136	136	136	136	136	1,399
Tire Truck	9	0	68	68	68	68	68	68	68	68	68	68	699
Light Plant	1	0	17	17	17	17	17	17	17	17	17	17	178
Crew Van	5	0	18	18	18	18	18	18	18	18	18	18	185
Pickups	5	0	135	135	135	135	135	135	135	135	135	135	1,391
Welder/Generator	5	0	10	10	10	10	10	10	10	10	10	10	101
TOTAL		0	14,781	14,781	14,781	14,781	14,781	14,781	14,781	14,781	14,781	14,781	148,010
	$/Tonne Mat.		0.45	0.45	0.45	0.45	0.45	0.47	0.47	0.47	0.72	1.01	0.50

TABLE 24-15 VISTA GOLD CORP. – MT TODD GOLD PROJECT Operating Cost Estimate for Processing 10.65 MM Tonnes per Year December 2006			
Cost Category	**Cost Annual $US**	**US$ per Tonne Milled**	**A$ per Tonne Milled**
LABOR			
Salaried Personnel	1,766,483	0.17	0.22
Hourly Operations Personnel	6,441,810	0.60	0.80
Hourly Maintenance Personnel	3,148,463	0.30	0.39
Total Labor	11,356,755	1.07	1.42
CONSUMABLES			
Power	34,494,019	3.24	4.31
Reagents	13,211,099	1.24	1.65
Grinding Steel	5,488,169	0.52	0.69
Maintenance Supplies	3,227,502	0.30	0.40
Misc. Operating Supplies	677,775	0.06	0.08
Trucking Sulfide Tailings	585,750	0.06	0.07
Estimated Operating Cost	**69,041,069**	**6.48**	**8.62**

TABLE 24-16
VISTA GOLD CORP. - MT. TODD GOLD PROJECT
Process Labor Summary
December 2006

Salaried Labor

Position	Total Number	US$/Year
Mill Manager	1	$93,750
Operations Superintendent	1	$86,250
Mechanical Maintenance Superintendent	1	$86,250
Electrical Maintenance Superintendent	1	$86,250
Operations Foreman	4	$86,250
Mechanical Maintenance Foreman	1	$75,000
Electrical Maintenance Foreman	1	$75,000
Chief Metallurgist	1	$90,000
Plant Metallurgist	1	$86,250
Instrument Foreman	1	$75,000
Clerk	2	$45,000
	15	

Operating Labor

Position	Total Number	US$/Year
Crusher Operator	12	$67,500
Control Room Operator	4	$52,500
Grinding Mill Operator	4	$52,500
Flotation Operator	4	$67,500
CIL Operator	4	$67,500
Carbon Stripping Operator	4	$67,500
Tailings Operator	4	$67,500
Assayers	6	$67,500
Samplers	6	$97,500
Laborers	16	$48,750
Equipment Operator	4	$63,750
	68	

Maintenance Labor

Position	Total Number	US$/Year
Mechanical Maintenance Man I	15	$67,500
Mechanical Maintenance Man II	15	$41,250
Electrical Maintenance Man I	4	$67,500
Electrical Maintenance Man II	2	$41,250
Instrument Man I	2	$67,500
Instrument Man II		
	38	

General and Administrative Costs

TABLE 24-7 summarizes staffing, salary costs, and estimated expenses for general and administrative services for the Mt Todd Gold Project. The general and administrative payroll is about $2.5 million per year, and the operating expenses are about $2.2 million per year, for an average operating cost of about $0.14 per tonne of material mined.

24.6 Cash Flow Estimates

A cash flow analysis and sensitivity studies were completed for the base case, which includes mining the measured, indicated and inferred resources and assumes a gold price of $600 per ounce, average pit slopes of 55 degrees, start-up capital of $264 million, and operating costs of $1.21 per tonne mined for mining, $6.48 per tonne milled for processing, and $0.14 per tonne mined for G&A. TABLE 24-1**7** summarizes results from the before tax, 100 percent equity, constant 2006 US dollar, cash flow analysis. TABLE 24-18 and FIGURE 24-2 summarize the sensitivity of the Net Present Value of the projected cash flows at discount factors of 0, 5, 10, 15, 20, and 25 percent. Sensitivities for gold prices of $500, $600, and $700 per ounce and plus and minus 20 percent for operating and capital costs are presented. The base case has a Discounted Cash Flow Rate of Return of approximately 17 percent. The breakeven gold price is a nominal $532 and $568 per ounce gold for DCFROR rates of 0 and 10 percent, respectively.

TABLE 24-17
VISTA GOLD CORP. - MT. TODD GOLD PROJECT
Base Case Before Tax Cash Flow Summary
December 2006

Case:		Copper Price ($/lb)	$2.00	Gold Price ($/oz)	$600																
30,000 TPD																					
M+I+I; 55 degree slopes				**PP 2**	**PP 1**	**Yr 1**	**Yr 2**	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yr 8**	**Yr 9**	**Yr 10**	**Yr 11**	**Yr 12**	**Yr 13**	**Yr 14**	**Yr 15**	**Totals**
MINE PRODUCTION SCHEDULE (000s tons)																					
	ORE																				
		TOTAL		0	0	10650	10650	10650	10650	10650	10650	10650	10650	10650	10650	0	0	0	0	0	106500
		% Cu		0.00	0.00	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03						
		oz Au/ton		0.000	0.000	0.039	0.037	0.035	0.032	0.032	0.029	0.026	0.023	0.023	0.019						
	WASTE			0	0	22000	22000	22000	22000	22000	21000	21000	21000	10000	4000	0	0	0	0	0	187000
		TOTAL		**0**	**0**	**32,650**	**32,650**	**32,650**	**32,650**	**32,650**	**31,650**	**31,650**	**31,650**	**20,650**	**14,650**	**0**	**0**	**0**	**0**	**0**	**187,000**
		Stripping Ratio				2.1	2.1	2.1	2.1	2.1	2.0	2.0	2.0	0.9	0.4						1.8
CONCENTRATOR SCHEDULE																					
	Mill Feed	tons (000s)		0	0	10650	10650	10650	10650	10650	10650	10650	10650	10650	10650	0	0	0	0	0	106500
	Recovery	Cu %	**70**	70	70	70	70	70	70	70	70	70	70	70	70						
		Au %	**87**	87	87	87	87	87	87	87	87	87	87	87	87						
	Cu Concentrate Grade	% Cu	**24**	24	24	24	24	24	24	24	24	24	24	24	24						
	Cu Concentrate Produced	tons (000s)		0.00	0.00	9.32	9.32	9.32	9.32	9.32	9.32	9.32	9.32	9.32	9.32						
		conc ratio		0	0	800	800	800	800	800	800	800	800	800	800						
PAYABLES SCHEDULE																					
	Copper S&R Rec = .96	lb (000s)		0	0	4295	4295	4295	4295	4295	4295	4295	4295	4295	4295						42,949
	Gold S&R Rec = .985	oz		0	0	348588	334063	319539	290490	290490	261441	232392	203343	203343	174294						2,657,983
REVENUE ($000)																					
	Copper	$/lb FOB Refinery	**2.00**	0	0	8,590	8,590	8,590	8,590	8,590	8,590	8,590	8,590	8,590	8,590						85,897
	Gold	$/oz FOB Refinery	**$600**	0	0	209,153	200,438	191,723	174,294	174,294	156,865	139,435	122,006	122,006	104,576						1,594,790
		TOTAL		**0**	**0**	**217,742**	**209,028**	**200,313**	**182,884**	**182,884**	**165,454**	**148,025**	**130,595**	**130,595**	**113,166**	**0**	**0**	**0**	**0**	**0**	**1,680,687**
MINE OPERATING COSTS ($000s)																					
	Mining																				
	Open Pit	$/ton mined	**1.21**	0	0	39507	39507	39507	39507	39507	38297	38297	38297	24987	17727						355135
	Milling	$/ ton milled	**6.48**	0	0	69012	69012	69012	69012	69012	69012	69012	69012	69012	69012						690120
	G&A	$/ton mined	**0.14**	0	0	4571	4571	4571	4571	4571	4431	4431	4431	2891	2051						41090
		TOTAL	**1.0**	**0**	**0**	**113,090**	**113,090**	**113,090**	**113,090**	**113,090**	**111,740**	**111,740**	**111,740**	**96,890**	**88,790**	**0**	**0**	**0**	**0**	**0**	**1,086,345**
FREIGHT, SMELTING & REFINING COST ($000s)																					
	Concentrate Freight	$/ton con shipped	**98.00**	0	0	913	913	913	913	913	913	913	913	913	913						9132
	Treatment Charge for Cu Conc	$/ton Cu con treated	**100.00**	0	0	932	932	932	932	932	932	932	932	932	932						9,319
	Refinery Charge	$/lb Cu recovered	**0.10**	0	0	429	429	429	429	429	429	429	429	429	429						4,295
		TOTAL		**0**	**0**	**2,275**	**2,275**	**2,275**	**2,275**	**2,275**	**2,275**	**2,275**	**2,275**	**2,275**	**2,275**	**0**	**0**	**0**	**0**	**0**	**22,746**
		$/lb Cu recovered				0.53	0.53	0.53	0.53	0.53	0.53	0.53	0.53	0.53	0.53						
ROYALTY		Deanhurst GSR%	**0.00**	0	0	0	0	0	0	0	0	0	0	0	0						**0**
		JAAC NSR%	**1.00**	0	0	2177	2090	2003	1829	1829	1655	1480	1306	1306	1132						**16,807**
		TOTAL																			
NET OPERATING REVENUE ($000s)				**0**	**0**	**100,201**	**91,573**	**82,946**	**65,691**	**65,691**	**49,786**	**32,531**	**15,275**	**30,125**	**20,970**	**0**	**0**	**0**	**0**	**0**	**554,789**
CAPITAL COST SUMMARY ($000s)																					
	Access and Site Prep			300	150	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**450**
	Surface Plant and Facilities			950	1000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**1,950**
	Site Infrastructure			1150	500	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**1,650**
	General Surface Mobile Equipment			1300	486	1510	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**3,296**
	Concentrator and Tailings Disposal			79800	106400	1445	1658	7904	6296	5994	1955	3460	1277	0	0	0	0	0	0	0	**216,189**
	Open Pit Mine Development			5550	2400	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	**7,950**
	Open Pit Mine Equipment			712.8	24332	9439	108	216	8856	9385	7668	15455	1037	108	108	0	0	0	0	0	**77425**
	Permitting, Reclamation, and Closure			500	0	0	0	0	2323	2556	0	0	0	0	0	581	6214	6215	5111	7500	**31,000**
	G&A, OH, Contingency			10126	14627	1239	177	812	1748	1794	962	1891	231	11	11	58	621	622	511	750	**36,191**
		TOTAL	**1.0**	**100,389**	**149,895**	**13,634**	**1,943**	**8,932**	**19,223**	**19,729**	**10,585**	**20,806**	**2,545**	**119**	**119**	**639**	**6,835**	**6,837**	**5,622**	**8,250**	**376,101**
NET PRETAX CASH FLOW ($000s)				**-100,389**	**-149,895**	**86,567**	**89,631**	**74,014**	**46,468**	**45,962**	**39,200**	**11,724**	**12,730**	**30,007**	**20,852**	**-639**	**-6,835**	**-6,837**	**-5,622**	**-8,250**	**178,688**

Note: **Startup Capital (Yr PP2 +PP1+1) 263,918**
No Working Capital, 100% Equity, Constant 2006 $US, Before Tax
Year 15 Closure Costs ($7.5 million) to pay for 10 years of Operations and Maintenance

TABLE 24-18
VISTA GOLD CORP. - MT. TODD GOLD PROJECT
Cash Flow Net Present Value Sensitivity Analysis
December 2006

Gold Price Sensitivity (NPV $000s)

Discount %	Base($600)	$500.00	$700.00
0	$178,688	($84,453)	$441,828
5	$100,088	($89,702)	$289,879
10	$45,739	($95,558)	$187,036
15	$8,059	($100,043)	$116,161
20	($18,186)	($102,851)	$66,479
25	($36,528)	($104,187)	$31,130

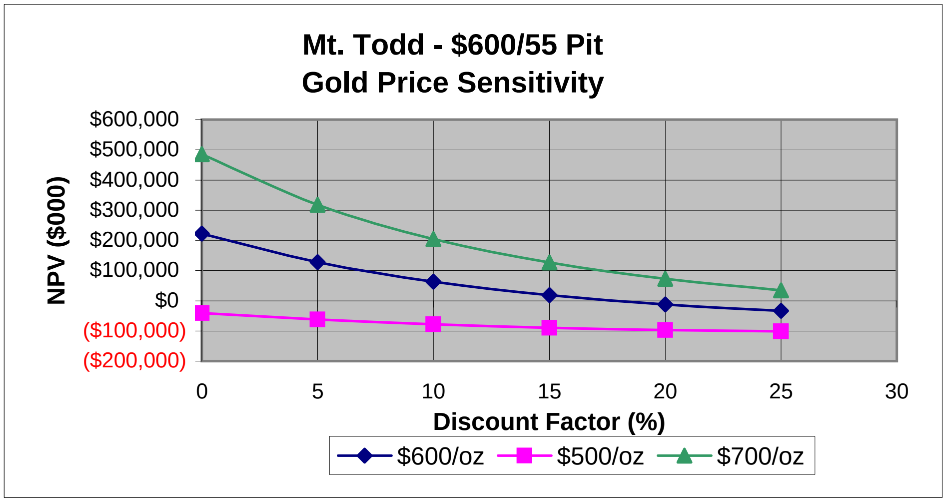
Operating Cost Sensitivity (NPV $000s)

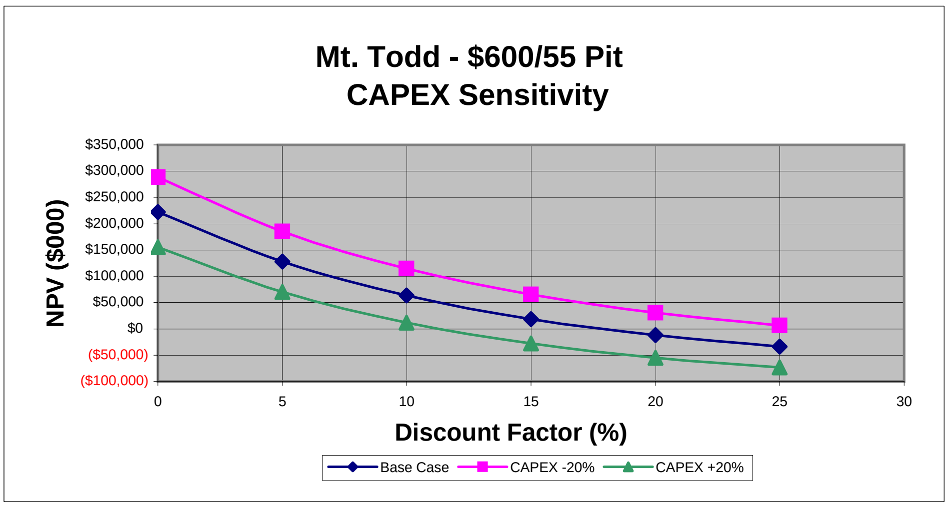
Discount %	Base($600)	Op Cost-20%	Op Cost+20%
0	178,688	395,957	(38,581)
5	100,088	253,378	(53,201)
10	45,739	157,567	(66,090)
15	8,059	92,055	(75,938)
20	(18,186)	46,525	(82,897)
25	(36,528)	14,433	(87,490)

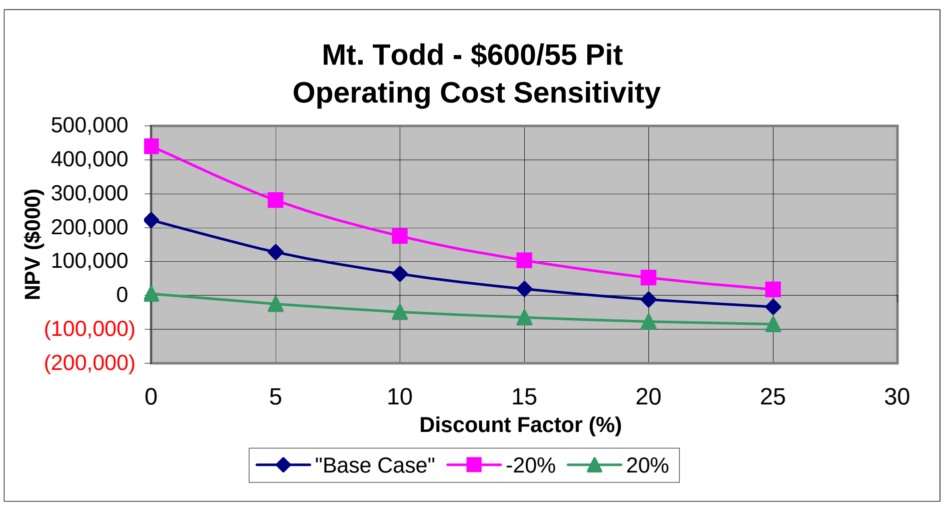
Capital Sensitivity (NPV $000s)

Discount %	Base($600)	CAPEX-20%	CAPEX+20%
0	178,688	253,908	103,467
5	100,088	163,256	36,921
10	45,739	100,646	(9,168)
15	8,059	56,901	(40,783)
20	(18,186)	25,966	(62,337)
25	(36,528)	3,852	(76,909)

FIGURE 24-2
VISTA GOLD CORP. - MT. TODD GOLD PROJECT
Cash Flow Sensitivity Analysis
December 2006







25.0 ILLUSTRATIONS

All of the illustrations used in the preparation of this report appear in each of their respective sections.