RICHMONT

RICHMONT MINES INC.

TECHNICAL REPORT 43-101
Estimation of mineral resources and reserves
EAST AMPHI PROJECT
Malartic, Quebec

PREPARED BY
JULES RIOPEL, M.Sc, P.Geo, MBA

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1.0 EXECUTIVE SUMMARY

This Technical Report (the "Report") on the East Amphi project provides a description of the overall work and parameters used in the development of mineral reserve and resource estimates and a technical review of the project as of December 31, 2005. This report was prepared by Qualified Persons at Richmont Mines. The parameters used for the geological and structural interpretation of the project, as well as for the mineral inventory estimate and the economic study are mainly based on the best estimation of Richmont Mines. This Technical Report was prepared in accordance with National Instrument 43-101 and companion policies regarding technical reporting, and also complies with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) guidelines on the estimation and classification of mineral resources and reserves.

1.1 Location and Property Description

The East Amphi property consists of 34 contiguous claims for a total of 1077 hectares including a mining lease (BM 848), wholly owned by Richmont Mines (100%). This property is located some 30 kilometres west of the city of Val-d'Or and was acquired by Richmont Mines in December 2003 from McWatters Mining Inc. The city of Malartic is located less than 3 kilometres from the centre of the mining property. The property is crossed by Highway 117 which runs from Val-d'Or to Rouyn-Noranda. A 2% NSR (Net Smelter Return) royalty is payable to McWatters Mining Inc. after 300,000 ounces of gold have been produced from the property.

1.2 History

In 1937, McIntyre Porcupine Mines Ltd explored the Cadillac fault and intersected mineralized diorite and feldspar porphyry. East Amphi Gold Mines Ltd became the owner in 1940 and started underground development in early 1946. Following the sinking of a three-compartment shaft to a depth of 155 metres, 1,490 metres of drifting and 415 metres of cross-cuts, the company announced its decision to suspend underground activities. From 1948 to 1987, sporadic drilling was completed.

Breakwater Resources carried out work on the property from 1987 to 1990 and drilled 85 surface diamond drill-holes, 92 underground drill-holes and geological mapping of underground developments. In 1998, McWatters bought the property from Placer Dome, drilled an additional 44 drill-holes on the East Amphi property and excavated a total of 120,427 tonnes of ore from an open pit at an average diluted grade of 5.66 g/t Au. This ore was transported to the Sigma mill for processing.

1.3 Geology and Mineralization

The East Amphi property is located in the southern part of Abitibi greenstone belt, which is an Archean volcano-sedimentary complex in the Superior Structural Province of the Canadian Shield. The property is located in the Malartic mining camp. The Cadillac–Larder Lake deformation zone runs across the central part of the East Amphi property and lies along the boundary between metasedimentary rocks of the Pontiac Subprovince and basaltic to ultramafic volcanic rocks of the Piché Group in the Abitibi Subprovince. This deformation zone produced a talc-chlorite schist unit into which the gold-bearing diorite dykes were injected.

These mineralized zones were defined stratigraphically as zones A, B, and P, in which distinct lenses have been identified and named as zones A1-A2-A2' and A3, and zones B2-Bn and B1-B1'. The B2-Bn zones occur at the interface between the feldspar porphyry and the talc-chlorite schist. These zones are marked by the injection of several generations of intrusions, generally subparallel to the foliation and showing a boudinaged structure. A1-A2-A2', A3 zones, and B1-B1 zones are situated within the talc-chlorite schists. The P zones are developed within the porphyritic body to the south. The zones have been identified over a strike length of more than 1 km, from the surface to a depth of 600 metres, and they can reach a thickness of more than 20 metres.

The gold mineralization occurs in two distinct settings. In general, gold is associated with widespread coarse-grained (up to 5 mm) automorphic pyrite. This type of mineralization represents the majority of the resources and is typical of the gold-bearing mineralization found on the property. The second form is where gold is associated with injected quartz veins and veinlets hosted in the feldspar porphyry intrusion to the south and in a shear zone that transects the first type of mineralization (Bn).

1.4 2004-2005 Exploration Program

Following the acquisition at the end of 2003, Richmont Mines launched an underground exploration program involving drilling and drifting in order to improve the quality of the resources, transfer resources to reserve categories and increase the resources. Table 1.1 presents the total exploration program. A total of $23,719,554 was therefore invested in exploration in 2004-2005.

Table 1.1 Summary of 2004-2005 Exploration

Work	Metreage 2004	Metreage 2005	Total 2004-2005	# Holes	Comments
Ramp	1,037	294	1,331		(4.7 m wide x 4.5 m high) (4980 to 4792 El)
Exploration drift	274	194	468		100 level (4.5 m x 4.5 m) (4900 El)
	310	39	349		125 level (4.5 m x 4.5 m) (4875 El)
	110	201	311		150 level (4.5 m x 4.5 m) (4850 El)
	0	201	201		175 level (4.5 m x 4.5 m) (4825 El)
	0	186	186		200 level (4.5 m x 4.5 m) (4800 El)
Miscellaneous excavation	392	1,402	1,794		Cross-cut, mucking bay, sump, lunch room, electrical station, drilling bay
Ore excavation	0	604	604		Sill out in B2 zone
Underground delineation drilling	1,323	19,913	21 236	311	19 ddh in 2004 – 311 ddh in 2005
Surface drilling	12,713	2,373	15,087	56	47 ddh in 2004 – 9 ddh in 2005
Test mining/milling			40 581		24,917 T development ore 15,664 T test mining (stope)

1.5 Drilling

A total of 969 holes totalling 109,554 metres of surface and underground diamond drilling were completed within the bounds of the study area from the early 1930's to September 2005. Approximately 430 holes totalling 80,147 metres were drilled from the surface, and 539 holes totalling 29,407 metres from underground, including Richmont Mines drilling. During the 2004-2005 exploration program, 367 holes totalling 36,323 metres were drilled. Of these, 56 NQ-size diamond drill-holes totalling 15,087 metres and 311 BQ-size diamond drill-holes totalling 21,236 metres were completed from underground infrastructures from December 2004 until the end of September 2005. The majority of the drill-holes used for resource and reserve estimates were BQ size.

A rock quality designation (RQD) analysis was completed for all drill-holes drilled in the 2004–2005 program. Table 1.2 gives an overview of the RQD study. The RQD for zones B2 and Bn is relatively good.

Table 1.2 RQD Study

Zone	RQD (%)
B2	87.3
Bn	70.8
B2-Bn Footwall	88.3
B2-Bn Hanging wall	44.6
A3	53.9
A3 Hanging wall	53.8
A3 Footwall	54.7

1.6 Sampling - QA/QC Protocol

In 2004–2005, a total of 25,643 samples were taken and Table 1.3 presents the distribution per type of sample. All 2004–2005 samples were sent to ALS Chemex Chimitec of Val-d'Or, certified ISO 9001:2000. For drill-holes, a total of 22,783 metres of core were sampled for 62.7% of total drilling: 16,015 metres for underground samples (75.4% of total UG drilling) and 6,768 metres of surface drilling (44.9% of total surface drilling).

Table 1.3 Sample Distribution by Type – 2004-2005 Exploration Program

UG DDH	SURFACE DDH	CHIPS	MUCK	TEST HOLE	TOTAL
16,286	6,249	1,076	1,981	51	25,643

During the course of the geological confirmation program, an evaluation of "Quality Assurance/Quality Control" (QA/QC) data was done to address the three (3) main concerns of analytical determination protocols, namely: (i) contamination, (ii) accuracy, and (iii) precision, as measured by the results obtained from field and analytical blank standards, certified reference standards and an assortment of specific duplicate samples collected and/or prepared, in addition to the regular samples submitted to the laboratory. The results of field and analytical blanks used to monitor for potential contamination during sample processing and assaying indicate that no significant contamination is likely to have occurred during the sampling/assaying programs completed.

Good accuracy was demonstrated by the laboratory during the sampling/assaying programs, as monitored internally and externally by the assaying of certified reference standards. Precision results from evaluation of AAS, gravimetric finish sample duplicates as well as assorted pulp duplicate results for the various original sample types demonstrate overall poor precision is being realized for the 50 g pulp samples. It is recommended that the split size be increased from the 250 g to 1000 g size. Future sampling/assaying programs should include collection and assaying of coarse crush sample duplicate splits as well as field duplicate samples for evaluation of total and incremental precision levels and better identification as to the primary source of error.

A comparison of the AAS versus gravimetric results demonstrates that precision for the AAS finish results is better up to grades of 3.5 g/t Au when gravimetric grades are more precise. Re-assaying of initial AAS determinations should be completed using gravimetric methods for results reporting greater than 3.5 g/t Au.

1.7 Mill Test

The 2004-2005 exploration program included extraction of a bulk sample. This exercise served as a final step in the grade validation of East Amphi reserves, and also served to test the behaviour of the stopes particularly with respect to the dilution rate and ground conditions. The East Amphi ore was hauled by truck to Richmont Mines' Camflo mill located at an approximate distance of 13 km from the minesite. The Camflo mill is a traditional gold recovery mill using a conventional Merrill-Crow type process, with circuits for crushing, grinding, gold cyanidation and precipitation using zinc powder.

During 2005, 3 batches were processed for a total of 40,581 tonnes; the first two batches were development ore, and the last one included four stopes and development ore. Table 1.4 presents the description of ore milling in 2005. For the third batch, 10,256 dry tonnes of development ore and 15,664 tonnes of ore from four stopes were transported and milled.

Table 1.4 Description of Ore Milling in 2005

BATCH	DATE		TONNAGE (METRIC TONNES)	DESCRIPTION
	FROM	TO		
1	July 24, 2005	July 30, 2005	6,349	Development ore
2	September 26, 2005	October 3, 2005	8,312	Development ore
3	December 7, 2005	January 01, 2006	10,256 15,664	Development ore 4 stopes
Total			**40,581**	

The head grade was calculated based on the 2005 East Amphi milling program. For the bulk sample from batch 3, the head grade averaged 4.0 g/t Au, whereas the head grade for the entire 2005 mill test, including development ore from batches 1 and 2, was 3.7 g/t. Gold recovery of the zinc precipitation circuit at the Camflo mill was established at 97.5%.

The main conclusions of the reconciliation between planning, mining and milling are:

- The head grade is 3.7 g/t Au for the entire 2005 mill test, and 4.0 g/t Au for the third batch only.
- The muck grade was lower than the actual head grade by 10.7% for the entire 2005 mill test, and by 10.5% for batch 3 only.
- The mill feed head grade was overestimated by 2.2% for the entire 2005 mill test and overestimated by 3.1% for the third batch only.
- The estimated grade of CMS-surveyed stopes (block model from drill core and chip samples) was lower than the actual head grade by 6.3% for the entire 2005 mill test, and by 4.7% for the third batch only.
- Average grade milled from the four stopes returned a content of 4.14 g/t compared to an undiluted grade planned of 3,85 g/t and an average grade of 3,80 g/t according to outlined tonnage effectively withdraw in the block model evaluation. This represents an upgrade of over 9% based on CMS measurement evaluation.

1.8 Resource and Reserve Estimates

The mineral reserve and resource estimate was calculated using reasonable parameters and results were reported in accordance with CIM guidelines for the estimation, classification and reporting of reserves and resources and with regulations under National Instrument 43-101.

Richmont Mines has estimated the Global Resources using all information from its 2004-2005 exploration program including information from underground openings and historical diamond drilling. The East Amphi Global Mineral Resource estimate served as a basis for the East Amphi mineral reserve and resource estimate. The conversion of mineral resources into mineral reserves is based on economic studies comparable to a feasibility study in terms of accuracy and detail, carried out by the mining engineers of Richmont Mines. The source data and the parameters used for the calculation of mineral resources and mineral reserves correspond to acquired knowledge, best estimation and the situation as at December 31, 2005.

At East Amphi, Global Mineral Resources are estimated using the polygonal method (on longitudinal section) for the A1-A2-A2', A3, and B1-B1' zones, and using inverse distance to power (IDP) block modelling methods with Gemcom software for the B2-Bn and P zones. The main parameters used to estimate the global mineral resources are as follows:

- A cut-off grade of 3 g/t Au was used;
- Grade capping at 30 g/t Au for all zones;
- A minimum true thickness of 3.0 metres based on the proposed mining method (long hole);
- An average rock density of 2.8 t/m^3 is defined.

The main parameters for the block model used to estimate the global mineral resources are as follows:

- A wireframe of 3 g/t;
- Compositing of gold assay results was done on 1-metre equal lengths;
- A 30-metre ellipsoid search was fixed by the distance;
- The block size for the B2 zone is 1.25 metres by 1.0 metres by 1.5 metres;
- Interpolation method: the IDP interpolation method to the power 1 was used to minimize the nugget effect of gold assays;
- A maximum of 7 composite samples per diamond drill-hole within a maximum of 15 composites was fixed.

1.8.1 Resource Category

Measured mineral resources were confirmed by a high density of drill-holes, some cross-cuts through the mineralized zones and a bulk sampling program. In the polygonal method, blocks are considered measured resources if drill spacing is less than 10 metres and several drill-holes form a cluster of similar results, especially if there is some underground development in the mineralization to confirm its location and grade. In the block model, areas within 10 metres of a drill-hole are considered as measured resources.

Mineral resources are considered in the indicated category if drill spacing is between 10 and 20 metres and several drill-holes form a cluster of similar results. In the polygonal method, the maximum extension of each block is 20 metres laterally and 40 metres vertically. For the block model, indicated resources lie within a 20-metre radius of a drill-hole.

Mineral resources are considered in the inferred category if drilling is sparser but again drill intercepts must be grouped in a cluster of similar (geological) results within a reasonable distance of no more than 30 metres at most depending on geology. For the block model, inferred mineral resources are defined within a maximum distance of 30 metres from a drill-hole. In the polygonal method, the area of influence is a maximum of 30 metres laterally and 60 metres vertically.

As can be seen in Table 1.5 below, resources in the measured and indicated categories as of December 31, 2005 amount to a total of 1,436,052 tonnes at a grade of 5.33 g/t Au for a total of 246,295 ounces of gold, and inferred resources stand at 332,711 tonnes at 6.09 g/t Au for 65,134 ounces of gold. The global mineral resources include the tonnage coming from surface pillars. Dilution and mining recovery factors are not included in the above resource estimate.

Table 1.5 Summary of Global Mineral Resource Estimate as of December 31, 2005 (Cut-off 3.0 g/t Au and ≥ 3.0 metres true width)

DESCRIPTION	TONNES	GRADE G/T	OUNCES
Measured			
B1-B1' Zone	94,732	6.80	20,719
B2 Zone Mine	432,289	4.25	59,068
Bn Zone Mine	134,003	4.14	17,836
A1-A2-A2' Zone	36,965	7.26	8,628
Total Measured*	**697,989**	**4.73**	**106,252**
Indicated			
B1-B1' Zone	6,604	8.68	1,843
B2 Zone Mine	15,739	3.94	1,994
Bn Zone Mine	8,481	4.0	1,091
B2 Zone (4,800 – 4,725 elevation)	66,716	4.5	9,652
Bn Zone (4,800 – 4,725 elevation)	130,664	5.86	24,618
B2 Zone (Explo)	132,856	4.53	19,350
Bn Zone (Explo)	155,620	5.56	27,818
A1-A2-A2' Zone	173,117	7.16	39,851
A3 Zone	48,266	8.91	13,826
Total Indicated	**738,063**	**5.9**	**140,043**
Total Indicated and Measured	**1,436,052**	**5.33**	**246,295**
Inferred			
B1-B1' Zone	25,897	8.30	6,908
B2 Zone Mine	4,243	3.86	527
Bn Zone Mine	811	4.23	110
B2 Zone (4,800 – 4,725 elevation)	2,384	4.43	340
Bn Zone (4,800 – 4,725 elevation)	1,664	4.06	217
B2 Zone (Explo)	40,386	6.02	7,817
Bn Zone (Explo)	23,398	5.54	4,168
A1-A2-A2' Zone	114,494	6.01	22,109
A3 Zone	55,718	8.47	15,173
P Zone	63,716	3.79	7,766
Total Inferred	**332,711**	**6.09**	**65,134**

* Include 27,109 tonnes extracted during the exploration work program

1.8.2 Estimate of Mineral Reserves

The economic study has determined proven and probable reserves amounting to a total of 641,000 tonnes at an average grade of 4.88 g/t Au for a total of 100,500 ounces of gold as of December 31, 2005 (Table 1.6).

Table 1.6 Summary of Reserve Estimate as of December 31, 2005 (Cut-off 3.0 g/t Au and ≥ 3.0 metres width)

DESCRIPTION	TONNES* (METRIC)	DILUTED GRADE* (G/T AU)	RECOVERABLE OUNCES**
Proven Reserves			
B2 – Bn Zone (above 4,800 elevation)	288,891	4.04	37,500
Total Proven Reserves	**288,891**	**4.04**	**37,500**
Probable Reserves			
B2 – Bn Zone (4,725 – 4,800 elevation)	136,294	5.19	22,740
A1-A2-A2' and B1-B1' Zones	176,160	5.21	29,490
A3 Zone	39,480	8.49	10,780
Total Probable Reserves	**351,934**	**5.57**	**63,010**
Total Proven and Probable Reserves	**640,825**	**4.88**	**100,510**

* Including dilution and 100% mining recovery (excluding the pillars).
** Before milling recovery (97.5%).

The parameters and the basis of calculation used in the economic study are:

- The extraction methods will be transversal and longitudinal long hole and cut-and-fill with a minimum dip of the zone at 60° and a maximum panel length of 11 metres by 25 metres high;
- Ore recovery: 100% for stopes designed with 4-metre pillars between stopes clearly identified during the process of the mineral reserve estimate. The pillars are excluded from the reserves;
- Internal dilution: minimum mining width is 3 metres for the B2-Bn zones and 4 metres for the A1-A2-A2', A3, and B1-B1' zones;
- External dilution: a rate of 10% at a grade of 2.7 g/t Au for the B2-Bn Mine zone, 1.9 g/t Au for the B2-Bn zones below 4800 elevation, and 20% at a grade of 0.5 g/t Au for the A1-A2-A2', A3, and B1-B1' zones;
- Operating costs of $62.00/tonne;
- Cut-off grade of 3 g/t Au fixed with a gold price at US$450/oz (or CAN$540/oz with an exchange rate of CAN$1.2 for US$1). However, the final cut-off grade will be based on the current gold price at the time of extraction of the reserves;
- The mill recovery is 97.5%.

1.8.3 Estimate of Mineral Resources

The mineral resources at East Amphi are calculated after transferring the reserves from the Global Mineral Resource as of December 31, 2005. Indicated and measured resources amount to a total of 820,200 tonnes at a grade of 5.15 g/t Au for a total of 135,800 ounces of gold, and inferred resources stand at 310,000 tonnes at 5.95 g/t Au for 59,000 ounces of gold (Table 1.7).

Table 1.7 Summary of Resource Estimate as of December 31, 2005 (Cut-off 3.0 g/t Au and ≥ 3.0 metres width)

DESCRIPTION	TONNES (METRIC)	GRADE (G/T AU)	OUNCES
Measured			
B2 Zone Mine	290,132	4.02	37,519
Bn Zone Mine			
B1-B1' Zone	33,502	8.73	9,406
A1-A2-A2' Zone	8,115	7.92	2,066
Total Measured	**331,749**	**4.59**	**48,991**
Indicated			
B2 Zone Mine	15,739	3.94	1,994
Bn Zone Mine	8,481	4.00	1,091
B2 Zone (4,800 – 4,725 elevation)	14,161	4.01	1,827
Bn Zone (4,800 – 4,725 elevation)	38,999	5.94	7,444
B2 Zone (Explo)	132,856	4.53	19,350
Bn Zone (Explo)	155,620	5.56	27,818
B1-B1' Zone	6,891	9.06	2,007
A1-A2-A2' Zone	95,703	6.88	21,165
A3 Zone	19,950	6.44	4,131
Total Indicated	**488,400**	**5.53**	**86,826**
Total Indicated and Measured	**820,149**	**5.15**	**135,817**
Inferred			
B1-B1' Zone	24,980	8.47	6,801
B2 Zone Mine	4,243	3.86	527
Bn Zone Mine	811	4.23	110
B2 Zone (4,800 – 4,725 elevation)	2,384	4.43	340
Bn Zone (4,800 – 4,725 elevation)	1,664	4.06	217
B2 Zone (Explo)	40,386	6.02	7,817
Bn Zone (Explo)	23,398	5.54	4,168
A1-A2-A2' Zone	101,167	6.01	19,540
A3 Zone	45,702	7.98	11,725
P Zone	63,716	3.79	7,766
Total Inferred	**308,451**	**5.95**	**59,010**

1.9 Pre-feasibility Study

Richmont Mines has reviewed the pre-feasibility study on the project to mine the East Amphi deposit. The East Amphi property contains approximately 100,000 ounces of gold from 641,000 tonnes of proven and probable ore grading 4,88 g/t Au.

1.9.1 Mining

All the mining facilities are already built near the deposit. The East Amphi underground mine will be accessed via a ramp which extends from surface via the existing pit from 4,980 elevation to 4,792 elevation. Based on ground quality, transverse and longitudinal long hole was chosen as the standard and effective mining method to lower capital investments and to ensure the lowest possible operating costs, while maintaining an excellent control on ground conditions and a low dilution rate. Stope size is planned at 11 metres along strike, 25 metres in the vertical axis with an average width of 7 metres, corresponding to the economic orebody thickness. This layout is valid for the reserves above the 200 level, in the B2 orebody.

A bulk sample, taken in December 2005, consisting partly of the first four stopes, has shown good to excellent wall control and a very low amount of dilution in three stopes that averaged only 3.9%. Production drilling and blasting is a critical portion of the cycle to maintain wall stability during the extraction phase, to minimize dilution.

The actual production is based on an operating schedule of 8-hour shifts, two shifts per day and five days per week. The daily production rate is set at 800 tonnes of ore plus the additional tonnage from waste development and backfill requirements. The annual production for 2006 is planned at 200,000 tonnes. The ore is brought back to surface with 26-ton trucks along the access ramp, which starts in the south wall of the existing open pit. The ore is then dumped on a surface pad.

Table 1.8 shows previous and forecasted development and production schedule for the B2 zone above 4800 elevation from early 2004 through to 2007.

Table 1.8 Development and Production Schedule (B2 zone – above the level 200)

Description	2004 Tonnes (grade)	2005 Tonnes (grade)	2006 Tonnes (grade)	2007 Tonnes (grade)
Waste				
Contractor's development	130,300	114,900		
Richmont Mines' development	0	43,800	13,500	
Total	**130,300**	**158,700**	**13,500**	
Ore				
Development	0	24,917 (3.35 g/t)	50,300 (3.39 g/t)	
Stope (diluted)	0	15,664 (4.25 g/t)	149,700 (4.18 g/t)	88,900 (4.18 g/t)
Total	**0**	**40,581 (3.98 g/t)**	**200,000 (3.98 g/t)**	**88,900 (4.18 g/t)**

1.9.2 Processing

East Amphi ore is hauled by truck to the Camflo mill located at an approximate distance of 13 km from the minesite. Transport Nord-Ouest ensures the ore transportation on the basis of a three (3)-year contract. No major problems of operation at this plant, other than the usual maintenance and repairs, were encountered nor are anticipated in the near future.

The mill's gold recovery rate is estimated at 97.5% for the East Amphi ore. The ore contains a large fraction of carbonates (CO_3) and is definitely not acid-generating, as its neutralisation potential (NP) is much higher than its acid potential (AP).

1.9.3 Environmental Considerations

The restoration program will mainly consist of:

- Dismantling of all buildings, structures and foundations;
- Soil characterization to determine contamination and required remedial measures;
- Site revegetation;
- Levelling of settling ponds; and
- Protecting open pit access.

The above work is the subject of the warranty of completion estimated at $82,600 and the funding is currently warranted.

1.9.4 Future Mining Plan

Mining of the satellite zones is forecast at $70/tonne and additional development costs of $10.8M are forecasted to extract 65,000 ounces. Small long-hole stopes, 15 metres in height and 6 to 7 metres along strike is the mining method retained for most of the tonnage. The cut-and-fill approach is also considered for a small portion (74,000 tonnes) of the ore reserves.

Table 1.9 below indicates the estimated annual tonnage to be extracted by zone based on current ore reserves from January 2006 to mid-2010.

Table 1.9 Production Forecast 2006-2010

ZONE	2006	2007	2008	2009	2010
B2-Bn (> 4800 El)	200,000	89,000			
B2-Bn (< 4800 El)		22,500	48,000	48,000	22,000
A2 and B1 (> 4848 El)		16,100	33,600	33,600	13,000
A2 and A2' (< 4848 El)		3,900	31,000	29,800	10,000
A3 E-O (> 4820 El)		7,500	14,400	13,600	5,000
Total:	200,000	139,000	127,000	125,000	50,000
Grade:	3.98 g/t	4.68 g/t	5.58 g/t	5.56 g/t	5.61 g/t
*Ounces recovered:	24,952	20,392	22,214	21,786	8,793

* Before milling recovery (97.5%)

1.9.5 Capital and Operating Costs

At year-end 2005, a total of $23,719,554 in capital costs were spent to provide access to the B2 ore reserves down to the 200 level. Operating costs to mine and mill the B2 ore reserves are forecasted at $62.00/tonne in 2006, including an investment of $1.15M for 900 metres of drifting ($5.71/tonne).

Table 1.10 shows the summary of the estimated average operating costs for the third year of production, which is considered to be a representative year for the East Amphi operation. Reserves in the B2 zone above the 200 level amount to 289,000 tonnes on January 1, 2006. Production from that zone will be at 200,000 tonnes/year for a period of 17 months. Production costs for the 2006 fiscal year are as follows:

Table 1.10 Forecasted Production Cost

	B2 - BN $/T BUDGET 2006	SATELLITE ZONES $/T FORECAST
U/G Operation	21.36	26.03
Maintenance and services	15.57	17.13
Technical services and Administration	5.58	7.25
Transportation and milling	19.64	19.64
Total	62.15	70.05

1.9.6 Financial Analysis, Payback and Taxes

A financial analysis for the East Amphi mining project was conducted using the parameters described in the report. The capital cost of the mine development and the surface infrastructures are taken into account. The operating costs are those shown in the above table.

A fixed gold price of US$450 an ounce and an exchange rate of US$1.00 = CAN$1.20 were used in the financial analysis. No debt financing was taken into account and the total East Amphi capital investment is therefore considered to be equity. No annual inflation was assumed for investments and for operating costs.

Total gold production (actual and future mining plan) is forecasted at just over 100,000 ounces over a period of 4.5 years. At CAN$540/oz, total revenues of $53M will be generated, with total spending for the same period of $50M. Cash flow generated will be at $3M, based on that gold price.

The project is sensitive mainly to the gold price, operation cost and capital expenditures, based on specific dilution parameters (10% in the B2 zone and 20% in satellite zones). The breakeven point is at approximately CAN$520/oz. Using the parameters listed above and no inflation on expenses, the analysis shows a project internal rate of return of 4.82% before income taxes and 4.19% after income taxes.

1.9.7 Mine Life

Mine life is anticipated to be around 4.5 years, based on current ore reserves and mine plans. On the positive side, the East Amphi property has a 4-km strike along the mineralized Malartic shear, and has the potential to find additional reserves. The actual resource base could also be partly transferred into mineable reserves.

But, on a more conservative side, the current probable reserves are economically viable at a CAN$540/oz scenario. Development costs for the satellites zones were evaluated at $1,230/metre for a total amount of $10.8M. These costs will have to be closely tracked and appropriate ground control measures taken, as part of the development will be driven in schist formation.

A complete review of the proposed mining methods, associated costs as well as mining productivity and risk based on ground conditions will be conducted prior to going ahead with the second phase of development. Actual and future short-term mining experience in the B2 zone above the 200 level will serve as a basis for that review. Actual ground conditions look more stable than predicted, mainly because of low depth of mining and the absence of lateral constraints.

2.0 INTRODUCTION

The present Technical Report (the "Report") on the East Amphi project was prepared by the technical staff of Richmont Mines under the supervision of Jules Riopel, M.Sc., P.Geo, MBA in order to provide a technical review of the project including a summary of scientific and technical information concerning mineral exploration and development activity. This document is intended to be submitted by Richmont Mines to regulatory authorities and monitoring organizations such as the Toronto Stock Exchange and the *Autorité des marchés financiers* as recommended by applicable laws and regulations.

This Report provides a description of the overall work and parameters used in the development of mineral resource estimates for the East Amphi project along with a study of the project economics in order to determine the mineral reserves as of December 31, 2005.

The information and data used to prepare this Report are based on a number of documents, including geological reports, other earlier reports and procedural guidelines developed by Richmont Mines personnel along with past and recent drilling, mapping, and bulk sampling. The parameters used for the geological and structural interpretation of the project, as well as for the mineral inventory estimate and the economic study are mainly based on the best estimation of the Company.

This Technical Report was prepared in accordance with National Instrument 43-101 and companion policies regarding technical reporting, and also complies with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) guidelines on the estimation and classification of mineral resources and reserves.

The authors of the present Technical Report are Richmont Mines personnel who are Qualified Persons as per National Instrument 43-101 or technicians working under their direct supervision. Qualified Persons with their respective areas of expertise for the preparation of the different sections of this Technical Report are listed below along with their responsibility and title. The detail of authors participation of each section is presented in Appendix 1.

Geology – Exploration and Mineral Resource and Reserve estimations	• Jacques Daigneault, P.Geo • Jules Riopel, M.Sc., P.Geo, MBA • Christian Bézy, P.Geo	• Senior Exploration Geologist • Geology and Exploration Manager • Chief Geologist
Mining - Mineral reserve estimation and Economical study	• Alain Mercier, Eng. • Christian Pichette, Eng., M.Sc., A	• Manager, Technical Services • Vice-President, Operations
Finance - Sales, royalties and taxes	• J.Y. Laliberté, CA • Nicole Veilleux CA	• Vice-President, Finance • Corporate controller
Environment	• Alain Mercier, Eng.	• Manager, Technical Services
Metallurgy	• Richard Nolet,	• Mill Superintendent,Camflo

3.0 TERMS OF REFERENCE

Unless otherwise noted, all units of measurement used in this Report are expressed according to the metric system. The following conversion factors and their respective abbreviations are used in this Report:

- 1 ounce troy (oz) = 31.1035 grams (g)
- 1 tonne (t) = 1.1 short tons (or 2,000 pounds)
- 1 metre (m) = 3.28 feet (ft)

Other frequently used abbreviations in this Report are as follows:

- Au: gold
- cfm: cubic feet per minute
- ddh: drill hole
- g: gram
- g/t: grams per metric tonne
- ha: hectare
- kg: kilogram
- km: kilometre
- kV: kilovolt
- m: metre
- mm: millimetre
- NPI: net profit interest
- NSR: net smelter return
- t/d: tonnes per day
- T: metric tonne

Unless otherwise indicated, all financial data on revenues and costs are expressed in Canadian dollars. Royalty amounts are in Canadian or American dollars, depending on the contractual terms agreed to between parties. Gold price is expressed in US$, which is the most internationally recognised currency for precious metal prices.

Qualified Persons have directly supervised the collection of data used in the preparation of this Report. As mentioned earlier, a large number of mine plans, sections and documents were generated and used in the determination of the mineral resource and reserve estimates as well as in the preparation of the economic study. Consequently, only a few schematized mine plans and sections in smaller format are included to understand and to illustrate the present Technical Report. It should be noted that all documents, calculation backup sheets, plans and sections, external technical reports, are kept at East

Amphi and are available on demand at any time for review and audit purposes, please contact Julie Normandeau, Investor Relations at Richmont Mines Montréal office.

4.0 DISCLAIMER

This Technical Report 43-101 contains forward-looking statements that include risks and uncertainties. The factors that could cause actual results to differ materially from those indicated in such forward-looking statements include changes in the prevailing price of gold, the Canadian-United States exchange rate, grade of ore mined and unforeseen difficulties in mining operations that could affect revenues and production costs. Other factors such as uncertainties regarding government regulations could also affect the results. Other risks may be detailed in Richmont Mines' Annual Information Form, Annual Report and periodic reports.

Cautionary Note to U.S. Investors Concerning Resource Estimates

The resource estimates in this Technical Report 43-101 were prepared in accordance with National Instrument 43-101 adopted by the Canadian Securities Administrators. The requirements of NI 43-101 differ significantly from the requirements of the United States Securities and Exchange Commission (the "SEC"). In this Technical Report 43-101, we use the terms "measured", "indicated" and "inferred" resources. Although these terms are recognized and required in Canada, the SEC does not recognize them. The SEC permits U.S. mining companies, in their filings with the SEC, to disclose only those mineral deposits that constitute "reserves". Under United States standards, mineralization may not be classified as a reserve unless the determination has been made that the mineralization could be economically and legally extracted at the time the determination is made. United States investors should not assume that all or any portion of a measured or indicated resource will ever be converted into "reserves". Further, "inferred resources" have a great amount of uncertainty as to their existence and whether they can be mined economically or legally, and United States investors should not assume that "inferred resources" exist or can be legally or economically mined, or that they will ever be upgraded to a higher category.

U.S. Investors are urged to consider the disclosure in our annual report on Form 20-F, File No. 0-28816, which may be obtained from us or from the SEC's web site: http://sec.gov/edgar.shtml.

5.0 PROPERTY DESCRIPTION AND LOCATION

5.1 Location

The East Amphi property is located within the limits of the town of Malartic, in Malartic Township (32D01), some 30 km west of the center of Val-d'Or, northwestern Québec. The project is located approximately 575 km northwest of Montréal, Québec. Figure 5.1 shows the project location with respect to the general area.

5.2 Description of Mining Titles

The East Amphi property consists of 34 contiguous claims for a total of 1077.1 hectares. It belongs to Richmont Mines, who also holds contiguous mining claims in the Fourax, Malartic Extension, Radium-Nord, Reservoir, and West Amphi properties (Figure 5.2 and Table 5.1). These lands are contiguous to the town of Malartic, which is part of the Vallée de l'Or MRC (regional county municipality). The East Amphi property covers part of Range I (batches 8 to 35 incl.) and part of Range II (batches 16 to 20 incl.) in Malartic Township. In 1998, McWatters Mining acquired a mining lease (ML #848) covering part of these claims to perform an open pit operation. This lease covers an area of 119.1 hectares. The limits of the mining lease are surveyed. The surface rights are held by the Crown.

A detailed list of the mining titles is provided in Appendix 2. All the claims are registered under the name of Richmont Mines and are in good standing.

Mining titles on the East Amphi property as well as on adjacent properties give Richmont Mines the following rights:

MINING TITLES	ASSOCIATED RIGHTS
Claims	• Exploration for mineral substances • Rights to subsurface only • Work required for renewal of right
Mining Lease	• 20-year period • No obligation or work requirement • Payment of annual fee • Surface rights limited to mining activities • Possibility to renew for an additional period of 10 years

Table 5.1 Description of Properties

PROPERTY	RICHMONT MINES INTEREST	# CLAIMS	SIZE (HECTARES)	ROYALTIES
East Amphi	100%	34	1077.1	2% NSR payable to McWatters Mining after 300,000 ounces of gold have been produced from the property. The NSR can be purchased for a cash payment of $1.5M.
Fourax	100%	12	270.3	3% NSR in favour of Royal Oak Mines Inc.
Malartic Extension	100%	11	443.2	No royalty
Radium-Nord	85% 15% to Currie	7	297.2	2-3% NSR in favour of Barrick Gold 15% NPI to Barrick Gold
Reservoir	100%	5	240.0	2-3% NSR in favour of Barrick Gold
West Amphi	100%	19	859.3	No royalty

5.3 Ownership of Mineral Rights

All the mining titles of the properties (Figure 5.2, Table 5.1) are held 100% by Richmont Mines, except Radium-Nord, where Richmont Mines owns 85% and the remaining 15% is held by the Currie Mill's estate.

5.4 Mineral Royalties

Four properties are subject to the payment of royalties and financial contractual obligations as follows:

5.4.1 East Amphi

A 2% NSR (Net Smelter Return) royalty is payable to McWatters Mining Inc. after 300,000 ounces of gold have been produced from the property. The determination of the amount of royalties to be paid is based on the gold price and the amount of gold produced on a quarterly basis. Richmont Mines has a right to purchase this royalty for $1.5M at any time.

5.4.2 Fourax

For every ounce of gold produced from the Fourax property, a 3% NSR (royalty calculation) is to be paid quarterly to Royal Oak Mines Inc. based on the prevailing price of gold.

5.4.3 Radium-Nord

For every ounce of gold produced from the Radium-Nord property, a royalty based on the gold price and is payable quarterly to Barrick Gold as follow:

Gold price (US$/oz)	Royalties (CAN$/oz Au produced)
< 350	2% NSR
> 350	3% NSR

A 15% NPI (Net Profit Interest) amount is payable on a monthly basis to Barrick Gold.

5.4.4 Reservoir

For every ounce of gold produced from the Reservoir property, a royalty based on the gold price is payable quarterly to Barrick Gold as follow:

Gold price (US$/oz)	Royalties (CAN$/oz Au produced)
< 350	2% NSR
> 350	3% NSR

5.5 Surface Infrastructure

5.5.1 General

The area is well served by existing infrastructure. The town of Malartic with 4,000 inhabitants is located less than 3 km east from the center of the mining property. The city of Val-d'Or, 30 km to the east, counts a population of approximately 32,500 inhabitants. It is accessible by a network of roads and is served by an airport with daily commercial flights. The property is crossed by the TransCanada highway 117 between Val-d'Or and Rouyn-Noranda (Figure 5.2).

The Camflo mill is located some 13 km east of the property. A CN railroad line crosses the southwestern part of the property. Two 120 kV hydro lines linking the Cadillac and Malartic stations pass less than 2 km north of the property and Hydro-Québec supplies the minesite with a 25 kV power line.

The city of Malartic gets its drinking water from an artesian well located 1.5 km to the northwest of the East Amphi shaft. Therefore, the mining project would have no effect on the drinking water source for the town of Malartic.

Besides the mine water collecting pond and a swamp-fed stream, there are no other apparent sources of surface water close to the mine.

5.5.2 Surface Facilities on Site

The following facilities are installed at the site and are shown on the Figure 5.3.

- The administration, geology and engineering offices (285 m^2);
- The lockers, mine dry, showers and shift boss dispatch (385 m^2 – capacity: 80 people). The sewage facilities can accommodate a shift of 60 people;
- The core-shack, garage, warehouse, cold warehouse and compressor room (4,000 cfm capacity);
- The backfill plant, consisting of a cement silo of 155 m^3 (160 tonnes), a screw conveyor of 10 inches in diameter, a 20-m^3 mixing tank, a mixer, an 8-m^3 water tank and an 8-m^3 retardant tank;
- A gate and gatehouse for security personnel;
- A parking lot for 35 vehicles;
- The site is currently supplied by Hydro-Québec with a 25 kV power line. Two electrical substations are in place. They are equipped with transformers and switchgears to bring the incoming 25 kV down to 4,160 volts and 600 volts, respectively for underground and surface distribution;
- The communication system for the East Amphi site consists of 25 optic fibre lines buried along the access road. Five lines are currently in service to meet telephone, fax and computer Internet requirements;
- The mine ventilation system consists of two fans, 150 hp each, pushing 160,000 cfm of fresh air underground. Noise is cut down with two appropriate silencers. Two propane heaters of 8 MBTU each heat the fresh air during winter season. The mine air is routed down through the old shaft, the two old track drifts, a distribution raise and finally, exhausts to the surface by the main ramp;
- The fuel storage tank of 2,000 gallons;
- The propane storage tank of 18,000 gallons;
- Explosive and detonator magazines. Currently, both materials are stored underground;
- The mine water settling and polishing ponds;
- The surface ramp to the underground ramp portal;
- Primary crushing is done at the Camflo mill. On site, a movable hydraulic hammer is used to break down larger blocks prior to road haulage.

5.6 Environmental Obligations

No environmental obligations.

5.7 Rehabilitation Plan - Permits

The rehabilitation plan for the East Amphi project was approved by the Ministère des Ressources naturelles et de la Faune du Québec and is valid until September 2008. A copy of the rehabilitation plan is available for consultation at the East Amphi site.

The schedule of annual guarantee payments for the rehabilitation plan is as follows:

YEAR	PAYMENT ($)
2005	$82,600 (100%)

All necessary permits and authorizations have been requested and issued. No other permits are needed. The following certificates of authorization have been issued:

- Certificate of authorization from the Ministère du Développement durable, de l'Environnement et des Parcs (May 14, 2004): "*Pour extraire 1 300 000 tonnes de minerai par voie souterraine, entreposer 361 000 tonnes de stérile, entreposer du minerai sur une halde de 3 650 m^2 et opérer une usine à remblai d'une capacité nominale de 100 tonnes à l'heure*" (to extract 1,300,000 tonnes of ore from an underground operation, stockpile 361,000 tonnes of waste, stockpile ore in a heap of 3,650 m^2, and operate a backfill plant at a nominal capacity of 100 tonnes per hour).
- Certificate of authorization from the Ministère du Développement durable, de l'Environnement et des Parcs (May 14, 2004): "*Pour construire et opérer un bassin d'eau de mine*" (to build and operate a mine water pond).
- Certificate of authorization from the Ministère du Développement durable, de l'Environnement et des Parcs (May 14, 2004): "*Pour aménager une halde à stérile de 30 000 m^2 d'une hauteur maximale de 6 mètres, aménager une halde de mort terrain de 12 000 m^2 et augmenter la capacité de la halde à minerai existante de 3 650 à 9 000 m^2*" (to set up a waste stockpile of 30,000 m^2 at a maximum height of 6 metres, to set up an overburden stockpile of 12,000 m^2, and to increase the capacity of the existing ore pile from 3,650 to 9,000 m^2).
- Certificate of authorization from the Ministère du Développement durable, de l'Environnement et des Parcs (August 26, 2004): "*Pour déplacer la halde à stérile #2 d'une superficie de 30 000 m^2 entre la route d'accès et la voie ferrée*" (to move waste stockpile #2, covering a surface area of 30,000 m^2, between the access road and the railway line).

6.0 ACCESS, CLIMATE, LOCAL RESOURCES AND PHYSIOGRAPHY

6.1 Access

The East Amphi project may be accessed from main highway 117 going west from Val-d'Or to Rouyn-Noranda just to the western limit of the town of Malartic. A 2-kilometre gravel road, going west from highway 117 and bordering the Malartic-Fournière township line in an east-west direction, provides easy access to the site and leads to the ramp, the old East Amphi Gold Mines shaft and the open pit, in the south-central part of the property. The access roads to the site are maintained by Richmont Mines.

6.2 Climate

The climatological information is based on data from the Val-d'Or meteorological station, except for evaporation data which comes from a station in Amos. The average annual precipitation is some 954 mm, peaking in September (some 102 mm). However, it was in July that the largest 24-hour amount of rainfall was recorded (68 mm). Snow falls between October and May with the most snowfall occurring between November and March. The average for that period is about 54 mm (expressed in mm of water).

Evaporation and potential evapotranspiration are highest in the summer months and almost nil during the winter months. The average annual evaporation and evapotranspiration are 541 mm and 489 mm respectively. In general, the net annual precipitation on a free water surface is 413 mm and the net annual precipitation on a vegetated surface is 465 mm.

The average daily temperature in Val-d'Or is slightly above freezing namely 1.2°C. The average temperature for July reaches 17.1°C while in January the temperature falls to –17.0°C. The lowest recorded temperature was -43.9°C and the highest recorded temperature was 36.1°C. It freezes an average of 209 days per year.

Anemometric data is based on the period from 1977 to 1989. Between August and January, southerly winds are dominant while between February and July northwesterly winds are more frequent. The average velocity varies between 9 and 17 km/h.

6.3 Local Resources

The local manpower is well trained. As the mining towns of Val-d'Or and Malartic are very active, it is relatively easy to recruit and to keep a mining workforce. Local professionals, engineers, geologists and technicians are also well available. Local regulations require that all personnel working in an underground mine must follow the modular course for miners, specifically modules 1, 2, 3, 5 and 7. Training sessions are held in Val-d'Or. The latter also hosts an excellent base of suppliers and manufacturers for the mining industry.

However, it is possible that in the short term, increased mining activity in the area may create a temporary shortage of certain categories of skilled workers.

At this time all technical staff and underground workers are hired and are working on the site. For the operation at the mine, a total of 47 people will be necessary. Table 6.1 shows the breakdown by department.

Table 6.1 Breakdown by Department

DEPARTMENT	# OF PEOPLE
Administration	1
Engineering/Geology	8
Service/Mechanical/Electrical	6
Mine - Underground Workers	23
Contractor Mirado Machine Rogers	 5 4

6.4 Physiography

The property is located in the lowlands of the Abitibi, which are part of the James Bay physiographic area. The East Amphi deposit is located in a very flat sector, near a major swamp for which the drainage is poorly defined. McWatters in built a series of channels to improve swamp drainage. The swamp either drains towards the north in the direction of Lake Malartic through an unnamed brook, or via the Malartic River into Lake Malartic, which is actually a widening of the Harricana River. A few rocky hills, roughly oriented east-west, mark the area's southern and western territory.

The overburden on the site of the project consists of two major groups. The first is related to the last glaciation and includes a till covered by lake deposits. The second group corresponds to recent formations and includes organic matter (bog and marsh) as well as alluvial deposits associated with flood plains.

Most of the area affected by the project is either sparsely wooded or clear of trees. A small wooded area close to the site is dominated by black spruce, ranging between 10 to 15 metres in height, and which has a density of 40 to 60%.

7.0 HISTORY

Gold was initially discovered on the property by an independent prospector in 1923. The discovery consisted of a weakly mineralized quartz vein, within a granitic formation (Cartier dome) on batches 16 and 17, Range I in Malartic Township. This showing became the focus of a number of drilling programs undertaken by several different companies.

In 1937, McIntyre Porcupine Mines Ltd. explored the Cadillac fault and drilled 18 surface holes (for a total of 3,823 m), intersecting mineralized diorite and feldspar porphyry containing minor mineralization.

In 1940, Canadian Malartic drilled 8 surface drill-holes on the property (for a total of 1,668 m). The same year Howey Gold Mines drilled another 20 holes (for a total of 3,193 m).

East Amphi Gold Mines Ltd. became the owner in 1940 and carried out surface drilling between 1940 and 1945. A total of 31 surface holes were drilled (for a total of 4,054 m). At the beginning of 1946, the company considered the surface drilling results sufficiently positive to justify underground development. Following the sinking of a three-compartment shaft to a depth of 155 metres, 1,490 metres of drifting and 415 metres of cross-cuts were excavated, on the -100 metres and -145 metres levels. It is reported that some 4,925 metres of underground drilling was carried out but data from only 101 of these underground drill-holes (for a total of 4,383 m) are included in the current resource calculation database. Geological work performed in 1946 allowed the identification of six ore zones on the -100 level and one zone on the –145 metres level. In each case the gold was associated with porphyry dykes. A resource of 623,695 tonnes at 8.84 g/t Au was estimated in 1946.

Early in 1948, the mining industry went through a difficult period and East Amphi Gold Mines Ltd. announced its decision to suspend underground activities.

In 1958/59 East Amphi Gold Mines drilled another 8 surface holes (for a total of 2,008 m).

In 1979/80 Darius Gold Mines carried out a 5-hole drilling program (for a total of 1,259 m).

Between 1981 and 1984, Sulpetro Minerals Ltd. (Novamin Resources) began exploration for the Darius Joint Venture on the East Amphi property and nearby lands (Malartic 7M and Malartic 8M properties) by completing magnetometer, VLF, IP and geological surveys. In 1986 the company drilled 8 surface drill-holes on the property (for a total of 2,604 m). These holes were located in the immediate vicinity of the mine.

During the winter of 1987-88 Breakwater Resources Ltd. carried out a surface drilling program in the mine area consisting of 56 diamond drill-holes (for a total of 12,335 metres, including two wedged holes). In 1988-89, the positive results of this program prompted the company to undertake an underground exploration program. The former East Amphi Gold Mines shaft and the two levels (-325' and -475' which are 100 metres and 145 metres below surface) were dewatered and the underground openings were mapped and sampled. A total of 92 underground drill-holes were drilled (for a total of 3,246 m). Additional surface drilling was carried out as well: an additional 9 holes in 1989 (for a total of 3,264 m), another 9 holes in 1990 (for a total of 3,587 m), and an additional 11 holes in 1994 (for a total of 5,262 m). A resource calculation was prepared in 1990 (758,015 tonnes of indicated resources at an average grade of 11.02 g/t).

In 1995 Placer Dome optioned the property and started an exploration program. A magnetic survey and two IP surveys were carried out, as well as a surface diamond drilling program. In 1995, 20 holes were drilled (for a total of 4,858 m) and in 1996 the company drilled another 23 surface drill-holes (for a total of 8,450 m). Another three holes were drilled south of the property but the results are not included in the database. A resource calculation was carried out, based on their geological model (850,000 tonnes of indicated resources at an average grade of 8.11 g/t).

In 1998, McWatters acquired the property and carried out a surface drilling program. During this program 27 drill-holes were drilled on the East Amphi property (for a total of 2,516 m) and 20 holes south of the East Amphi zones (for an additional 1,907 m). The results of the latter program are not included in the database. Using this information and all other data, McWatters reviewed the geological interpretation, in particular the link between the work of Breakwater (concentrated in the eastern part of the property) and the work of Placer Dome (concentrated in the west of the property). Based on this reassessment, the zones were redefined and renamed for consistency. A new resource calculation was carried out resulting in an estimated 2.29 million tonnes of measured and indicated resources (cut-off grade of 3.0 g/t) at an average grade of 5.98 g/t.

In the winter of 1999, an additional 17 surface drill-holes were drilled (for a total of 3,034 m) to delineate and validate the portion of indicated resources in the Zone B-West block, which McWatters planned to mine as an open pit. A limited portion of A-2 Zone was drilled as well.

Analytical results from McWatters' Kiena laboratory were reportedly verified in the Abilab laboratory using current methods and regulations for inter-laboratory verification. Some differences were identified and these were subjected to closer verification. Two mineralogical studies were conducted on the East Amphi ore. One study took place in the fall of 1998 at the *Centre de Recherche minérale du Québec* where the distribution, association and recovery rate of gold were examined, using particle size and gravity separation methods. A second study was carried out in 1999 involving macro- and microscopic examinations.

McWatters also completed sampling and analytical studies to confirm the results of earlier studies by Breakwater (1988) and Placer Dome (1995) in which it was shown that the East Amphi deposit and the surrounding host rocks are not acid generators.

Ore was excavated from an open pit from December 1998 to August 1999 and a total of 120,427 tonnes of ore was shipped to the Sigma mill for processing. The average diluted grade of the ore was 5.66 g/t, which correlated well with the reserve estimate.

In July 1999, McWatters prepared a study for the East Amphi deposit, prepared an underground mining plan and calculated the resources and reserves. This plan and the resource/reserve estimate were reviewed by an external consultant (John V. Tully & Ass.) and were reported at 1,124,600 tonnes of probable reserves at an average grade of 5.07 g/t Au. The reserves were calculated using the polygon method.

SNC-Lavalin was mandated in August 2002 by McWatters to carry out a feasibility study for an underground ramp mining project. This study shows an internal rate of return (IRR) of 43.6% before taxes and 33.6% after income taxes for an initial capital investment of $13.5M based on a fixed gold price of US$315 an ounce and an exchange rate of US$1.00 = CAN$1.59. The operating costs were estimated at an average of $44.0 per tonne of ore milled. Proven reserves were estimated at 170,000 T at 4.7 g/t Au and probable reserves at 1.2 MT at 4.1 g/t Au for a total of 1.4 million metric tonnes at 4.2 g/t Au.

In 2003, Richmont Mines bough the property for $7M and invested $23M to conduct an advances exploration program in 2004 and 2005.

7.1 Historical East Amphi Resource and Reserve Estimates

A number of resource and reserve estimates have been prepared over the years for the East Amphi Project. They are summarized below and in Table 7.1. Richmont Mines has not reviewed the historical estimates in sufficient detail to comment on their reliability and because more drilling has been carried out since that time. These historical resource and reserve estimates should not be relied upon, as they likely do not conform to NI 43-101 standards and definitions except for the SNC-Lavalin 2002 estimate. They are included in this section for indicative purposes only and should not be disclosed out of context.

Table 7.1 Historical Resource and Reserve Estimates

COMPANY	YEAR	CATEGORY	TONNAGE (METRIC)	GRADE (G/T AU)	ZONE
East Amphi Gold Mines Ltd	1946	Unknown	623,695	8.84	
Breakwater Resources	1990	Indicated resource	758,015	11.02	A2
Placer Dome	1995	Indicated resource	850,000	8.11	B2
McWatters	1998	Indicated-measured resource	2,300,000	6.00	All zones
McWatters(John V. Tully & Ass.)	1999	Probable reserve	1,124,600	5.07	All zones
McWatters (SNC-Lavalin)	2002	Proven reserve Probable reserve	170,000 1,200,000	4.70 4.20	All zones

The 2002 SNC-Lavalin estimate (Tables 7.2 and 7.3) covered a large area that included the A, B and P zones with 3 different methodologies used for the estimation (block model). The SNC-Lavalin estimate is the current estimate reported on SEDAR in a NI 43-101 technical report. The SNC estimate can no longer be considered current because of the significant amount of drilling and underground work that was carried out in 2004 and 2005. The resource estimate reported in this report therefore replaces it. The reader is invited to consult the report available on SEDAR for more information. A detail for each zone is presented in Appendix 3.

Table 7.2 2002 SNC-Lavalin – East Amphi Resource Estimate

	Grade Contour Model	GT Contour Model	Kriged GT Model
Classification	**Tonnes/Grade**	**Tonnes/Grade**	**Tonnes/Grade**
Measured	486,035 at 5.67	380,027 at 5.69	453,109 at 4.71
Indicated	1,212,641 at 4.22	844,329 at 5.14	1,221,373 at 4.65
Total	1,698,676 at 4.63	1,222,356 at 5.31	1,674,482 at 4.67
Inferred	1,239,518 at 3.64	798,661 at 5.88	723,574 at 4.35

All resource-modeling methods are based on a regular grid, as opposed to the classic polygonal method (half-distance between drill-holes). For resource modeling, the middle point of the drill-hole intercepts for each of the A Zone and the B Zone are projected on longitudinal sections (2D model). The composite grade of each drill-hole intercept (as mentioned above) and the horizontally projected thickness (perpendicular to the section matching the areas measured on the section and not the true geological thickness) are projected on a regular 10 by 10-metre grid using the Laplace interpolation method. In the base model prepared by SNC-Lavalin, the grade contour model, no preferential direction of continuity, limits in range of projection or minimum number of samples were specified as typically required with kriging or inverse distance models. Hence, the model is unidirectional, allowing any local rake to manifest itself. SNC-Lavalin is of the opinion that the grade contour model is representative of the available resources.

A cut-off grade of 2.725 g/t Au was used and high assays were cut to 30 g/t Au. The minimum width used was 2.0 metres. The resource classification is based on the following:

- Blocks are considered measured resources if drill spacing is less than 25 metres, especially if there is some underground development in the mineralization to confirm its location and grade.
- Blocks are considered indicated resources if drill spacing is between 25 and 50 metres and several drill-holes form a cluster of similar results.
- Blocks are considered inferred resources if drilling is sparser but again drill intercepts must be grouped in a cluster of similar (geological) results within a reasonable distance of no more than 100 to 200 metres at most depending on geology.

An important consideration in the SNC-Lavalin evaluation is the comparison with the results of the open pit ore (120,427 tonnes at 5.66 g/t Au) extracted in 1998. All 3 models were checked against the pit model and excavation results. They all indicate similar grades in that area and show little differences in the measured resources.

Table 7.3 2002 SNC-Lavalin - East Amphi Reserve Estimate

Grade Contour Model (Cut-off 2.725 g/t Au and 2.0 metres minimum width)		
Classification	**Tonnes (metric)**	**Grade (g/t)**
Proven	168,793	4.67
Probable	1,199,954	4.09
Total	1,368,747	4.16

The reserves were estimated using the grade contour model resources and a mining plan in which a crown pillar of 30 metres was left below the overburden (B2 and A zones) and 20 metres below the existing pit bottom (B1 Zone). The dilution was assumed at 15% with an average of 0.5 g/t Au. The reserves are therefore included in the measured and indicated resources above the 4775 metres elevation.

The mining reserves are calculated in order to estimate the volume and grade of ore that can be mined and processed at a potential profit from the East Amphi project. Only the measured and indicated resources can be used for the preparation of a mining plan to determine the proven and probable reserves. For the calculation of the measured and indicated resources, a cut-off grade of 2.725 g/t Au was determined as follows:

- Preliminary total operating costs (earlier McWatters study): $43.60/tonne
- Gold price: US$315/ounce
- Exchange rate: CAN$1.59 = US$1.00
- 1 ounce = 31.1035 grams

A minimum mining width of 2.0 metres was chosen to allow for the mining equipment to operate in the East Amphi mine.

8.0 GEOLOGICAL SETTING

8.1 Regional Geology

The Malartic area is located in the southeastern portion of the Abitibi Subprovince, a typical granite-greenstone terrane. The Abitibi Subprovince is located in the southeastern part of the Superior Province of the Canadian Shield. The Abitibi belt is the largest greenstone belt in the world (85,000 km^2; Card, 1990) and also one of the richest mining areas (Hodgson and Hamilton, 1989; Poulsen et al., 1992). The Abitibi Subprovince extends approximately 700 km from the Kapuskasing Structural Zone in northeastern Ontario eastward to the Grenville Front (Province) in northwestern Québec. The Abitibi Subprovince is divided into a "Northern Volcanic Zone" and a younger "Southern Volcanic Zone" (Ludden et al., 1986; Chown et al., 1992; Mueller et al., 1996). The Porcupine-Destor Fault Zone (PDF) is considered to be the limit separating the Northern Volcanic Zone from the Southern Volcanic Zone. The Northern Volcanic Zone is interpreted as an older diffuse volcanic arc, 2730-2710 MA and the Southern Volcanic Zone is interpreted as a segment of a younger arc, 2705-2698 MA (Mueller et al., 1996). The Malartic area and the East Amphi property are located within the Southern Volcanic Zone (Figure 8.1).

The Abitibi Subprovince is bounded to the north by gneiss and the plutonic terrane of the Opatica Subprovince, while to the south it is bounded by metasedimentary rocks and plutons of the Pontiac Subprovince. The contact between the group of Pontiac and the greenstone belt rocks is characterized by a tectonic episode, the tectonic zone of Cadillac. The age of installation of the volcanic rocks varies between 2,747 and 2,698 million years (My) (Mortensen, 1993).

The regional stratigraphy is divided into geological groups of sedimentary and volcanic rocks. The main geological groups in the Malartic area include the Piché Group, the Cadillac Group, the Malartic Group (or "Malartic Composite Block," described by Desrochers et al., 1996; Desrochers and Hubert, 1996). The Piché Group forms tectonic slices along the Larder Lake – Cadillac tectonic zone. The Piché group is defined by talc-chlorite and locally by carbonate schist, for which the protolith corresponds to magnesian basaltic to komatiitic flows, with local olivine cumulate or spinifex and highly altered with tremolite and carbonate. The Cadillac Group is a sedimentary unit corresponding to quartz wackes, quartzo-feldspathic wackes, pelites and polymictic conglomerates. Locally, the conglomerates contain iron formation fragments. The units of the Cadillac Group show a schistosity of variable intensity which is defined by micas (biotite and/or chlorite and/or muscovite). The Malartic Composite Block (Desrochers et al, 1996) is mainly composed of tholeiitic mafic to ultramafic volcanic rocks (komatiites) and can be subdivided into seven litho-tectonic domains: North, Central, Vassan, Baie Carpentier, South, Val-d'Or and de Montigny.

The East Amphi property hosts rocks from the Cadillac, Piché and Malartic groups and from the Pontiac Subprovince (Figure 8.2). The property is located in the Piché Group volcanics, which are sandwiched between sedimentary rocks of the Cadillac Group to the north and of the Pontiac Group to the south. The metavolcanics of this dominant stratigrafic unit form (Piché Group) a thick band (450 to 1,000 metres) in a NW–SE direction. The Larder Lake – Cadillac tectonic zone straddles the East Amphi property in its central part. This tectonic zone is a deformation zone located at the limit between the metasedimentary rocks of the Pontiac Subprovince (2683 MA) and the basaltic to ultramafic volcanic rocks of the Piché Group. This group, which hosts the Cadillac tectonic zone, represents the most promising zone on the East Amphi property for hosting gold. The Cadillac tectonic zone, comprised of a series of shears, is about 50 to 350 metres wide and is situated in the lower sequence of the Piché Group. The rocks are intensely deformed and sheared, with abundant talc-chlorite schists and minor diorite and feldspar porphyries. The degree of metamorphism varies between the lower greenschist and lower epidote-amphibolite facies. To the south of the Cadillac fault, rocks of the Pontiac Group occur. These are predominantly sediments, mainly massive greywackes but with some intercalations of sandstone or siltstone, with occurrences of minor felsic intrusive rocks (granite) in the southern part of the property.

The property is located on the southern flank of the Malartic Syncline, which strikes from east to west and traces a sub-vertical fold axis on the surface.

8.2 East Amphi Property Geology

8.2.1 General Overview

The description of the geology is based on work on the property, particularly on the recent drilling and drifting undertaken by Richmont Mines in 2004 and 2005. The East Amphi property is cut by the tectonic zone of Larder Lake–Cadillac. On the property, this deformation zone is found in the ultramafic rocks of komatiitic type in the Piché Group, which have been transformed into schists of chlorite and talc. This group is bordered on the north by the sediments of Cadillac and on the south by the Pontiac Group. An intrusive mass of feldspathic porphyry is found in the southern part of the Piché Group, south of the band of talc-chlorite schists from the Cadillac fault proper (Figure 8.2, Figure 8.3 and map in the Appendix 4).

The so-called East Amphi property includes major concentrations of gold-bearing ore. The zones have been identified over more than 1 km, from the surface to a depth of 600 metres, and they can reach a strength of more than 20 metres. These mineralized zones were defined stratigraphically as A, B and P zones, in which distinct concentrations have been identified and named as A1-A2 and A3 zones and B2-Bn and B1 zones. B2-Bn zones is found at the junction (interface) of the feldspathic porphyry and the talc-chlorite schist-a junction said to be transitional. This transition zone is characterized by a zone marked by the injection of several generations of intrusions, generally sub parallel to the foliation. A1-A2, A3 and B1 zones are situated within the talc-chlorite schists. P Zones are developed within the porphyric mass to the south.

The mineralization of the gold ore assumes 2 forms. In general, gold is associated with the spreading of large automorphic pyrites up to 5 mm in the lightly silicified zones associated with compacted and broken intrusions, either of a variably biotized dioritic nature or hematized and silicified felsic ones, or in the ultramafic deformed host rocks of the intrusions. This type of mineralization represents the majority of the resources and is typical of the gold-bearing mineralization found on the property. The second form is that the gold is associated with injected quartz veins and veinlets encased in the porphyric feldspathic intrusion to the south and in a shearing zone that divides the first type of mineralization. The gold is generally found within pyrite, and a few specks of free gold have been observed.

Given the association with the Lader-Lake Cadillac fault, we can see a complexity in the history of deformation and hydrothermalism in the gold deposits which is marked by at least 2 episodes. The first phase is marked by the development of penetrative foliation oriented towards 310, with a dip to the north varying from 50° in the upper part (100 level) to vertical towards the bottom (200 level), and seems to slope sharply to the south in depth (400 level). This episode of deformation is associated with the creation of the different generations of dykes that make up B2 Zone. Next, a second phase of deformation affect and dragged the first one, as is well documented by our on-site observations. Its orientation is parallel to that of the first, but its dip is sharply sloped to the north, varying between 70° and 90°. This phase of deformation dragged the foliation and the mineralized dykes of the first phase. The hydrothermalisme in the gold deposits associated with this second episode of deformation is characterized by the development of quartz veins injected parallel to the foliation. The presence of quarts veins marks the location of Bn Zone and is the sole marker of geological correlation.

In addition, the excavation work at levels 100 - 125 has demonstrated that the foliation to the south of mineralized B2-Bn zones seems to be sloped to the south at 40°. To date, no gold-bearing value has been associated with this system. These changes in attitude are poorly understood at present and could represent either the presence of an additional phase of deformation, a conjugated system associated with one of the early phases, or folded behind the deformation and at the site of the mineralization.

8.2.2 Description of Local Lithology

Five principal lithologies are observed on the property from north to south (Piché Group to Pontiac Group):

Sediment (Cadillac Group)

The Cadillac Group is made up primarily of greywackes and lens-shaped layers of conglomerates. No significant mineralization has been discovered inside this group in the Malartic region.

Mafic Volcanics - Gabbro (Piché Group)

Comprised principally of massive to occasionally pillowed basalts with inclusions of dykes of diorite and gabbro, which form the northern margin of the Cadillac fault.

Ultramafic Rocks and Chlorite - Talc Schist (Larder Lake - Cadillac Deformation Zone - Piché Group)

Progressively, the ultramafic rocks become deformed as a result of an intense deformation/alteration of an ultramafic protolith by the Larder Lake-Cadillac deformation zone. The dominant lithology within this deformation zone is the talc-chlorite schist, which is the altered resultant of a komatiitic sequence. This rock is typically bluish-grey and variably holds numerous veinlets of talc carbonate. The talc-chlorite rock unit ranges in composition from a talc-chlorite schist to a chlorite-rich, relatively unaltered ultramafic that lacks significant concentrations of talc. The former rock type is typically pervasively foliated, while the relatively unaltered ultramafic can be fairly massive. The overall rock unit is generally dark green in colour, fresh to slightly weathered and aphanic to fine-grained.

There are abundant units of talc-carbonate (dolomite) and chlorite layers and quartz veins. In general, the chlorite schists, which are intensely foliated, are composed of chlorite, with some amphiboles, carbonate and fine quartz feldspar clusters.

Within the East Amphi project area, the schist, comprising both the chlorite and talc-chlorite members, has an average thickness of some 60 metres and may attain up to 120 metres. The talc-chlorite schist is located north of the Bn Zone and host rock in the A1-A2, A3 and B1 zones. The chlorite schist is associated inside the transition zone (B2 Zone).

Mineralized diorite dykes of variable thickness (1-20 metres) are often intersected in the talc-chlorite schists unit. The presence of an intrusion indicates the transition zone and mineralized zone of A1-A2 and A3. These diorites are altered with biotites and carbonates and may be locally silicified. Small quartz veins are also found. The gold values are often associated with the presence of coarse pyrite. The rocks have a strong magnetic signature.

Between sections 150E and 1125E, a massive mafic volcanic flow is present, which becomes increasingly large, attaining a thickness of between 20 to 50 metres at depth, as does the diorite between the schist and porphyry units. This lava is particularly massive and was not affected by the Cadillac fault.

Diorite-Gabbro - Pyroxenite - Felsique Dykes (Transitional zone - B2 Zone - Larder Lake Cadillac Deformation Zone)

Different generations and compositional dykes are found in the transitional zone located between the talc-chlorite schist and the feldpathic porphyry stock. These dykes are injected during the deformation (phase 1) inside an ultramafic rock called komatiite, and few dykes are post phase 1. Foliation is not really present, but some of these dykes are broken up show a boudinage structure. This komatiite is variably deformed, less than the chlorite and talc-chlorite schists. The correlation of these dykes is not evident and is difficult to correlate, but it is possible to draw an envelope of the komatiite and dykes.

There are three categories of dykes, described below in order of increasing size:

1) Mafic dykes (Diorite): These are intrusive rocks, generally medium to dark grey in colour, fine- to medium-grained, composed mainly of albite and sodium-rich hornblende. Their general composition approaches that of gabbro. Chilled margins and cooling fractures are common, and alteration (biotite, amphibole and sulphides) is generally concentrated along these margins. They typically have a salt-and-pepper texture from disseminated carbonate (dolomite) clusters of up to 4 mm. It seems that in the transition zone of the B2 area, visually, the diorite dykes have an accrued affinity with gold mineralization (pyrite), but gold mineralization is also present in equal if not greater amounts in the felsic dykes and in ultramafic rocks.

2) Felsic dykes: These are generally grey to pinkish-beige, massive, fine-grained, variably siliceous, and may have seldom disseminated feldspathic phenocrysts. The silica content appears variable, ranging from dacitic, rhyodacitic and rhyolitic equivalents. Also, these dykes may be thin apophyses of the feldspathic porphyry stock or some intermediate phase between those two. The strength of these felsic dykes ranges from a few centimetres to more than 10 metres; in the latter case, they can easily be protruded by dioritic dykes.

3) Pyroxinetic dykes: These are grey to typically greenish-grey, medium- to coarse-grained, and typically massive, showing very minor deformation or fracturation with a low content of quartz veinlets, and are less inclined to gold mineralization. They are composed of amphiboles resulting of alteration procesus.

The diorite dykes often possess moderate to high magnetism, and their thickness varies between 5 and 30 metres. Locally the thickness of the diorite/gabbro dykes may attain 50–90 metres. Gold values are generally associated with these diorite dykes, but gold is also present inside the felsic dykes and ultramafic host rock.

Feldspathic Porphyry Stock

A mass of feldspar porphyry is found within the southern border of the Larder Lake-Cadillac deformation zone. The porphyry is a coarse-grained monzonite, which often demonstrates differentiation phases that could be named Quartz Porphyry with albite, Albitite Quartzite or Albitite. This unit generally occurs as dykes and discrete bodies along the southern contact of the Cadillac shear and within the talc-chlorite schist. It is characterized by a massive, undeformed rock (post-tectonic deformation), with a predominantly porphyritic texture (1-12 mm). The colour varies from grey (unaltered) to salmon pink (potassic and silicic alteration). The unit is often cut by mafic dykes and small quartz veins. Fine pyrite is disseminated in the matrix (trace to 5%). The unit is massive and laterally inter-fingered with the talc-chlorite schist and the sediments.

This lithologic unit, which forms the footwall of the B2 Zone, is often the site of anomalous gold values to the north contact and has an average thickness of 100 metres in the centre of the property. The unit splits laterally into several parts towards the east and west. The P zones are developed inside this unit. A hematization is associated.

Sediments (Pontiac Group)

This lithological unit consists of massive to rippled greywacke, with silty to microsilty beds in places. This contact is irregular, and volcanic and diorite bands and felsic dykes are often present.

9.0 TYPE OF MINERAL DEPOSIT

The East Amphi deposit is located in the Malartic Mining Camp, which produces 7.7 million ounces of gold. The mineralization is generally associated with the fault system of the Cadillac Lader Lake deformation zone. This mining camp hosts old gold products, like Canadian Malartic, Barnat, Malartic Gold Fields and East Malartic, with 1 to 3 million ounces of gold in its deposits (Trudel et Sauvé, 1992). The average grade has been 4.9 g/t, and few mines extend over 1,000 metres at depth. Table 9.1 shows the production of the main mines at the Malartic Mining Camp.

Table 9.1 Historical Production at the Malartic Mining Camp

Mines	**Production (year period)**	**Tonnage (Millions of metric tonnes)**	**Grade g/t Au**	**Geology**
Canadian Malartic	1935–1965	9.93	3.4	Pontiac Group
Sladen Malartic and Barnat	1938–1970 1976–1979	8.45	4.5	Piché and Pontiac Group
East Malartic	1938–1979	17.9	4.9	Piché Group
Malartic Gold Fields	1941–1965	9.0	5.9	Piché Group
Camflo	1965–1992	8.9	5.8	
Total		**54.1**	**4.9**	**7.7 million ounces**

The mineralization is similar between the mines and is characterized primarily by structural control and secondarily by lithological control. Regarding the Piché Group, which hosts the Cadillac fault, it is well established that most of the ore produced came from porphyry and dioritic intrusive led inside sheared volcanic rock. The gold deposits are found in zones of high deformation, where competent intrusive rocks are set within more ductile rocks. In these zones, deformation-resistant rocks have been fractured and mineralized into gold, while incompetent rocks were deformed in a ductile manner, creating fewer openings favourable to the depositing of gold.

The lithological context and the morphology of the ore from the Barnat - Sladen Mine are similar, if not identical, to those of the East Amphi deposit. The ore at Barnat comes primarily from the zone of diorites, or “buckshot,” which includes a swarm of dykes, chimneys, or segments of cigar-shaped intrusions running between 20° and 40° to the west, as at the East Amphi property all encased in talc-chlorite schists. In addition, at the East Malartic Mine, as indicated by Cormie (1948), the gold ore contains only a small number of quartz veins. Instead, the gold is spread out through the pyritized rock, which can be fractured or foliated, as at East Amphi.

At all the mines in the Malartic sector, the gold grade is generally proportional to the amount of pyrite present in the host rock. The richest ore is found in diorite, and it is noteworthy that the grade is directly proportional to the particle size of the pyrite; the same goes for East Amphi. This first type of ore is generally associated biotite, chlorite and carbonate altération.

The second type of ore is characterized by a classic alteration from an enrichment in silica and sodium. This type of alteration is found in felsic porphyries associated with veins and veinlets of quartz, carbonates and, locally, tourmaline; it appears in the form vein stockwork cutting almost exclusively through felsic intrusive. This type of mineralization was widely observed in the three largest mines of the Malartic camp, as well as in the Camflo mine. This type of mineralization has been identified at East Amphi but has not been adequately evaluated, although it does offer strong potential for exploration.

10.0 EAST AMPHI MINERALIZATION AND ALTERATION

The Larder Lake-Cadillac Deformation Zone, which crosses the ultramafic rocks of the Piché Group on the property, produced a gold-bearing talc-chlorite schist unit into which the gold-rich dykes were injected in contact with feldspathic porphyry to the south. The two principal gold-bearing zones, A and B zones, were developed within this structural corridor, in which distinct concentrations were identified and named as A1-A2 and A3 zones and B2-Bn and B1 zones. The P zones are associated inside the porphyric feldspathic stock and are relatively unknown (Figure 10.1 and map in the Appendix 4).

B2-Bn zones are situated at the juncture between the talc-chlorite schists and the intrusive porphyric mass in a zone called the transition zone. These zones have been traced over 1 km from east to west, 150 metres from north to south, and to a vertical depth of 600 metres. This transition zone is characterized by the injection of several generations of intrusion, generally sloped parallel to the foliation inside the ultramafic rocks. These intrusions seem to be lenticular and/or compressed and show little extension. Their correlation between the boreholes and crosscuts is difficult. We can, however, note a dominant regional rake steeply plunging between 20° and 40° to the west in the intrusions, such as that documented at the Barmat Mine.

Gold appears, first of all, in dykes generally containing less than 5% of automorphic pyrite disseminated in fine or coarse grains. The dykes are generally massive and of a dioritic composition, or even felsic poles with phenocrystals of feldspath. The gold values range from the komatiitic host schists, associated with the presence of coarse pyrite, to jagged quartz veins and a potassic (biotite) alteration. Grades are hard to estimate visually. For instance, dykes of diorite containing a significant amount of pyrite can yield appreciable gold values. The opposite is also true. Nonetheless, the local presence of biotite and pyrite in the schist does suggest that jagged stringers of diorite could be responsible for the deposits. The development of automorphic gold-bearing pyrite suggests a remobilization of gold during metamorphism and also suggests a complex history of hydrothermalism.

A statistical study of the gold-bearing content by type of unit has shown that the gold-bearing mineralization in B2 Zone is primarily associated at the level of 33% with felsic dykes, at 23% with mafic dykes (diorite), at 27% with schists and at 17% with komatiitic lava.

A1-A2, A3 and B1 zones are located within the talc-chlorite schists. The gold-bearing mineralization is generally associated with recrystallized coarse-grained pyrite and with biotized dioritic intrusions similar to B2 Zone. The extent of these dykes is variable, and the strength of the dykes varies between several centimetres and 6 metres. These dykes also seem to show a boudinage structure.

The second form is that the gold is associated with inject quartz veins and veinlets encased in the porphyric feldspathic intrusion to the south and in a shearing zone that divides the first type of mineralization and that is called Bn Zone. Gold is generally found in pyrites, and a few specks of free gold have been observed.

P zones are developed within the porphyric mass to the south. They are associated with quartz veins and veinlets associated with dykes of diorite and a shearing zone at a dip opposite to that of the other zones. A hematization of the walls of porphyry is also associated therewith. The geometry has not been well defined.

10.1 Characteristics of A1-A2, A3 and B1 Zones (talc-chlorite schist corridor)

- The mineralization in these zones is localized within the diorite dykes, which are often biotized and pyritized (up to 5% coarse-grained pyrite). These intrusives are lens-shaped and show a boudinage structure, with thicknesses varying generally between a few centimetres and 6 metres. As the dykes increase in thickness, they are less fractured and therefore less likely to be gold-bearing. The mineralization is seldom associated with quartz veining disposed parallel to lithology contacts and “oblique” to the same.
- These zones appear to be restricted between minor parallel faults and developed within the Cadillac Tectonic Zone. These minor faults can be correlated from section to section.
- Mineralization in these zones often proves to be less than three metres, as in the example in A3 Zone, in which the average mineralization thickness is near two metres without surpassing 3 metres. But elsewhere as it is in A2 Zone, mineralizations often surpass 3 metres and run as high as 6 metres in thickness. Also, it happens locally that there are two parallel dykes that could bring the coalescent zone to up to 9 metres. Finally, in A2 and B1 zones, these separate into two zones that are called, respectively, A2’ and B1’.

10.2 Characteristics of B2-Bn Zones (Transition Zone)

- The mineralization in B2 Zone is localized within the different composition and phases of the dykes, which are often biotized and pyritized (up to 5% coarse-grained pyrite) in the transition zone. These intrusives are lens-shaped and show a boudinage structure, with thicknesses varying generally between 1 and 20 metres. As the dykes increase in thickness, they are less fractured and therefore less likely to be gold-bearing. Sometimes the dykes affect the shape of the foliation, as in the phenomenon of diapirism.
- The mineralization in the B2 Zone also extends outwards into the schist and is associated with coarse pyrite, biotite and silica alteration that is generally found around the edges of the intrusive. Anomalous gold envelopes define a mineralized envelope traceable over 200 metres strike and 25 metres wide.
- Several feldspar porphyry dykes were also injected into the sheared komatiite. These dykes show potassic and silicic alterations and may contain up to 5% disseminated pyrite. A good gold value is associated with them.
- Bn Zone is different because of the presence of quartz veins associated with a phase 2 deformation. The veins are typical a sheared quartz vein, parallel to the shear zone. The veins contain 2% of pyrite, and minor amounts of visible gold are observed. The host rock contains dissemination of automorphic pyrite, which is anomalous in gold.

The best gold value is inside the quartz vein. This zone is a good geological marker, but at the 100 elevation, the zone shows few complications. Laterally, Bn Zone is discontinuous.

10.3 Characteristics of the P Zones (Feldspathic porphyry stock)

- Within the P zones, gold is generally associated with ≤ 5% pyrite disseminated within the porphyry and/or localized within millimetric microfractures, with quartz veins (1–2%), occasionally pyrite and molybdenite; it can also be associated with mafic dykes and shear zone. The mineralization forms a series of narrow parallel bands (echelons), separated by several metres, which extend both laterally and vertically. Potassic alteration and silicification, which are very common in the Malartic Camp, generally accompany gold mineralization.
- These zones dip to the south, and the direction and dip also vary. The relationship with the B2-Bn system is unknown. It is possible that this system could be conjugate with the Bn system, which is of the same type (quartz vein).
- These zones are not well understood and show good potential for future exploration work, especially by the development of staking of zones inside the porphyry stock. The majority of the drilling has been sub-parallel to the orientation of these systems.

11.0 EXPLORATION

11.1 Past Exploration

For a review of past exploration, the reader is referred to item 7 entitled "History" for a complete description of previous work including various resource and reserve estimates.

11.2 2004-2005 Richmont Mines Exploration

Following the acquisition at the end of 2003, Richmont Mines launched an underground exploration program involving drilling and drifting in order to improve the quality of the resources, convert resources to reserve categories and increase the resources. Table 11.1 presents the total exploration program. The total cost of exploration for 2004 was \$10,504,244. For 2005, the exploration program amounted to \$13,215,310. A total of \$23,719,554 was therefore invested in exploration in 2004-2005. A total of 1,331 metres of ramp, 1,515 metres of drift, 1,794 metres of miscellaneous excavation, and 604 metres of ore were excavated, 21,236 metres of delineation drilling (311 ddh) and 15,087 metres of surface drilling (56 holes) were completed, and 40,581 tonnes of development and stope ore was milled.

Table 11.1 Summary of 2004-2005 Exploration

WORK	METREAGE 2004	METREAGE 2005	TOTAL 2004-2005	# HOLES	COMMENTS
Ramp	1,037	294	1,331		(4.7 m wide x 4.5 m high) (4980 to 4792 El)
Exploration drift	274	194	468		100 level (4.5 m x 4.5 m) (4900 El)
	310	39	349		125 level (4.5 m x 4.5 m) (4875 El)
	110	201	311		150 level (4.5 m x 4.5 m) (4850 El)
	0	201	201		175 level (4.5 m x 4.5 m) (4825 El)
	0	186	186		200 level (4.5 m x 4.5 m) (4800 El)
Miscellaneous excavation	392	1,402	1,794		Cross-cut, mucking bay, sump, lunch room, electrical station, drilling bay
Ore excavation	0	604	604		Sill out in B2 Zone
Underground delineation drilling	1,323	19,913	21 236	311	19 ddh in 2004 – 311 ddh in 2005
Surface drilling	12,713	2,373	15,087	56	47 ddh in 2004 – 9 ddh in 2005
Test mining/milling			40 581		24,917 T development ore 15,664 T test mining (stope)

In 2004, Richmont Mines completed 47 surface drill-holes totalling 12,713 metres to confirm and increase geological resources identified by SNC-Lavalin. This work was performed by Forages Benoit and Forage Mercier. This program started in February 2004 and was completed in June of the same year. Subsequently, an underground exploration program was initiated. The portal was built in April 2004 and excavation of the ramp started on April 14, 2004. Dumas Contracting, from Timmins, was chosen to develop the access ramp and drifting until the end of June 2005. In 2004, 1,037 metres of ramp, 694 metres of drift and 392 metres of miscellaneous excavation was completed.

The 2004 exploration program consisted of:

- Dewatering the open pit;
- Infrastructures, namely core shack, garage, cold warehouse, surface electric substation, main ventilation and propane heating system;
- Preparation of waste pad #2 (30,000 m^2);
- 12,713 metres of surface drilling (47 ddh);
- 1,037 metres of ramp (below the 150 level);
- 694 metres of drift (100, 125, 150 levels);
- 392 metres of miscellaneous excavation (cross-cut, mucking bay, sump, lunch room, electrical station, drilling bay);
- 1,323 metres of delineation drilling on the 100 level (19 ddh);
- Resource estimation.

In December 2004, a delineation program was started at the 100 level which was available by then. Forage Orbit was contracted for the job. A new resource evaluation was completed. This resource evaluation indicated a resource of 926,646 tonnes grading 6.05 g/t Au in the measured and indicated categories and 943,228 tonnes grading 5.72 g/t Au in the inferred category. Table 11.2 summarizes this evaluation.

Table 11.2 Richmont Mines - Resource Evaluation, December 31, 2004

	MEASURED RESOURCE		INDICATED RESOURCE		INFERRED RESOURCE	
	Metric tonnes	**Grade (g/t)**	**Metric tonnes**	**Grade (g/t)**	**Metric tonnes**	**Grade (g/t)**
B1 Zone	122,839	6.20	28,777	6.08	6,362	6.36
B2 Zone	142,578	4.92	440,502	5.13	569,072	5.06
A1-A2 zones	51,124	8.77	105,781	9.32	140,028	8.53
A3 Zone			35,044	7.90	74,270	7.51
C Zone					115,024	4.44
An Zone					38,472	5.46
Total	**316,541**	**6.04**	**610,105**	**6.06**	**943,228**	**5.72**

The polygon method was used to calculate all ore zones on longitudinal sections. The half-distance rule was applied. Calculations are based on a minimum cut-off grade of 3 g/t

Au and a minimum mining width of 3 metres. The maximum cut-off grade was set at 18 g/t Au in the B2 Zone, and at 30 g/t Au in the A1-A2, A3, and B1 zones. Resource categories are based on the distance between drill-holes and their spatial relationship. Where the drilling density is 10 metres or less, resources are considered in the measured category. Where the drilling density exceeded 10 metres, the half-distance rule was applied and an area of influence of 10 metres laterally and 20 metres vertically from the drill-hole was used to determine indicated resources. Inferred resources constitute either isolated zones or indicated resources beyond the maximum 40-metre distance from a drill-hole.

In 2005, the underground exploration program continued. Dumas Contracting, from Timmins, was mandated to complete the access ramp and drifts until the end of June 2005. Subsequently, underground work was performed by Richmont Mines personnel. A total of 294 metres of ramp, 821 metres of drift and 1,402 metres of miscellaneous excavation was performed. The latter included a cross-cut to reach the zone for geological verifications and bulk sample preparation. Underground delineation and exploration drilling continued until the end of September 2005. A total of 311 drill-holes (19,913 metres) were drilled from the ramp and drifts on the 100, 125, 150, 175, and 200 levels. These holes intersected the B2-Bn zones and a few holes targeted the A3 Zone. From August to September 2005, a surface drilling program of 9 holes totalling 2,373 metres was performed to confirm and increase geological resources defined by SNC-Lavalin for the A1-A2 zones. This work was performed by Forage Mercier.

Underground mapping was carried out for the new ramp development down to the 200 level, at all levels and drop points on each level. Mapping information was plotted on plans located at the minesite and this information was digitized in AutoCAD in 2004 and transferred to Promine. In addition, back mapping and chip sampling was done after all development work was completed.

Richmont Mines collected muck (1981 samples) and chip samples (1076 samples) and drilled test holes (51 test holes) on all levels.

From November 4 to December 21, 2005, four stopes were prepared and excavated in order to complete a bulk-sampling program. A total of 15,664 tonnes of ore from four stopes and 10,256 tonnes of development ore was shipped and milled in December 2005 at the Camflo mill (Table 11.3). Prior to this, 2 batches of development ore were milled in July (6,349 tonnes) and September (8,312 tonnes).

The 2005 exploration program consisted of:

- Infrastructures, namely backfill plant, underground electric substation, warehouse;
- 294 metres of ramp below the 200 level (4792 El);
- 821 metres of drift (100, 125, 150, 175 and 200 levels);
- 1,402 metres of miscellaneous excavation (mainly cross-cut, ore sill out);
- 2,373 metres of surface drilling (9 ddh);

- 19,913 metres of delineation drill-holes on the 100, 125, 150, 175, 200 levels and ramp for a total of 311 holes;
- Preparation of an ore pad on surface;
- Milling of 24,917 tonnes of development ore and of a test mining batch of 15,664 tonnes;
- Resource and reserve estimation.

Table 11.3 Summary of 2005 milling

BATCH	DATE		TONNAGE (METRIC TONNES)	DESCRIPTION
	FROM	TO		
1	July 24, 2005	July 30, 2005	6,349	Development ore
2	September 26, 2005	October 3, 2005	8,312	Development ore
3	December 7, 2005	January 01, 2006	10,256 15,664	Development ore 4 stopes
Total			**40,581**	

12.0 DRILLING

12.1 Historical

A total of 969 holes totalling 109,554 metres of surface and underground diamond drilling has been completed within the bounds of the study area from the early 1930's to July 2005 (Table 12-1). Approximately 430 holes totalling 80,147 metres have been drilled from the surface, and 539 holes totalling 29,407 metres from the underground, including Richmont Mines' drilling. In 1988-1989, the breakwaters completed 92 underground drill holes for 3,246 metres to evaluate the B1 Zone. In 1944-45, East Amphi Mines completed 136 underground ddh for a total of 4,925 metres, intercepting the A1-A2-A2' zones.

The various drilling programs produced at least five (5) different core diameters during the history of the East Amphi project, namely AQ, BQ, LTK, ATW and NQ. The majority of the drill holes used for the estimates of resources and reserves were of BQ size. Thereafter, for surface drilling Richmont Mines chose the NQ diameter and used the BQ size for underground drilling.

Table 12.1 Historical Drilling Summary

EAST AMPHI PROJECT				
PROGRAM	PERIOD	SIZE	HOLES	METRES
McIntyre Porcupine Mine	1937	Unknow	18	3,823
Canadian Malartic	1940	Unknow	8	1,668
Howey Gold Mine	1940	Unknow	20	3,193
East Amphi Gold Mines	1940–45	Unknow	31	4,054
	1948	Unknow	5	1,259
Sulpetro Mineral	1986	Unknow	8	2,604
Breakwaters	1987–88	BQ	56	12,335
	1988–89	BQ	92	3,246
	1989	BQ	9	3,264
	1990	BQ	9	3,587
	1994	BQ	11	5,262
Placer Dome	1995	NQ	20	4,858
	1996	NQ	23	8,450
McWatters	1998	BQ	47	4,423
	1999	BQ	17	3,034
Richmont Mines	2004–2005	NQ	56	15,087
Total Surface Drilling			**430**	**80,147**
East Amphi Gold Mines	1940–45	AQ	136	4,925
Breakwaters	1988–89	AQ	92	3,246
Richmont Mines	2004–2005	BQ	311	21,236
Total Underground Drilling			**539**	**29,407**
Grand Total Drilling			**969**	**109,554**

12.2 2004-2005 Richmont Mines Drilling

As mentioned in the section 9, 2004-2005 exploration program consisted of 367 holes totalling 36 323 metres. A detailed summary of the 2004 and 2005 drilling exploration programs is presented in Appendix 5. The 2004-2005 surface drilling program was 56 holes for 15 087 metres. The 2004 surface program was initiated in mid-February and completed in June 2004, comprising 47 holes totalling 12,713.3 metres. In 2005, an additional 9 holes were drilled between August and September, for a total of 2,373 metres. The surface program tested the extension of the B2, A3 and A1-A2-A2' zones. For the B2 Zone, the objective was to specify the 100 level and test the perimeter extension of the known resources. For the A1-A2-A2' and A3 zones, the program had the objective of testing the extension of these zones and increasing the confidence level of the A3 Zone and the inferred resources of the A1-A2-A2' zones. In 2004, 3 drills were used with NQ diameter (1'' 7/8) and one for 2005 of the same size. This drilling was performed by Forage Benoit and Forage Mercier in 2004 and Forage Mercier in 2005.

During the course of the 2004-2005 underground drilling program, a total of 21,236 metres of diamond drilling from 311 holes was carried out from the underground infrastructures from December 2004 until the end of September 2005. One type of drill was used underground: an electric drill with a BQ diameter (1" 7/16). All the underground drilling operations were carried out by Forage Orbit, and between one and three drill rigs were used. For B2-Bn, over the 200 level, the average length of the holes was 51 metres. Below the 200 level, the length varied from 91 to 281 metres, with an average of 154 metres. For the A3 Zone, the average length was 154 metres.

12.3 Methodology and Planning

The majority of the drill holes are planned on cross-sections in order to intersect the zone perpendicularly. In the 2005 program, few drill holes are out-section whenever no underground opening is available or to explore the extensions of the known zones. All drill holes used to investigate the A3 Zone were azimuth holes.

Drill-hole spacing varied from 10 metres to 20 metres apart, depending on density of previous surface and underground drilling. Most of the holes intersected the projected structural trends around 15 metres above or below known intersections. This drilling was performed from levels 100, 125, 150, 175, and 200 and from the ramp. The objective was to increase the confidence level, extend the known resources and identify the location of the footwall drifts.

Below the 200 level, the section spacing was 20 to 40 metres and the vertical spacing varied from 20 to 40 metres. This drilling was performed from the ramp. The objective was to increase the confidence level until 75 metres below the 200 level and extend the known resources.

Each drill hole drilled by Richmont Mines at East Amphi has a unique identification that is related on the section name where fund the location of the collar.

12.4 Geology and Analysis

A detailed description of the drill cores is carried out by experienced and qualified personnel (graduate geologist) who are members of the OIQ (Quebec Order of Engineers) or the OGQ (Quebec Order of Geologists), according to a pre-established model at the East Amphi project. A computerized log is prepared for each drill hole with the following basic information:

- location
- main and secondary units
- texture and structure
- mineralization and alteration: mineralogy, thickness, type
- sample location

The length and location of the sample is controlled by the geology: *i.e.*, geological unit, alteration package or mineralized zone. The sampled intervals of the drill holes are sawn or split in order to preserve a sample of core-witness at the mine site. Once the sample results are returned from the laboratories, the results are plotted on sections and plans at the appropriate scale. The complete guide for core description, legends and the various symbols used is available for consultation at the East Amphi site.

At East Amphi, the relationship between the core length and the true thickness of the mineralization is as follows: On the longitudinal and maps, the horizontal thickness of the mineralization zone intersected is always reported. The horizontal thickness and true thickness were computed with the Gemcom software or measured directly from an interpreted drill-hole plan or section for the B2, BN and P zones; for all other zones, trigonometric calculation was employed.

12.5 Core Storage

Drill cores from all exploration holes are stored in their entirety in the East Amphi core library for the 2004–2005 drilling program. Each stored core box is identified with an aluminium tag that has the appropriate drill-hole information embossed on it (including the hole number, the box number and the core interval stored in the box). Boxes belonging to individual drill holes are stored consecutively in a core rack located outside on the East Amphi site. An inventory is kept for each core rack and is copied into an electronic databank by the geology department. For other old holes, all these holes are stored at the Kiena mine site and O'Brien mine site.

12.6 Recovery

The core recovery in mineralized zones is 100%. In the case of loss of the mineralized core, the routine procedure is to install a wedge or to redrill the missing mineralized interval. The rock quality designation (RQD) was completed for all drill holes drilled in the 2004-2005 drilling program. The RQD of the East Amphi project was performed in the following manner:

- Irrespective of the size of the core (BQ or NQ), the reference spacing on which the RQD is measured is basically 3 metres, corresponding to the drilling blocks in the core box. But for the schist contacts, shear fault or anomalous interval will prevail over those blocks one. For the BQ type, the length of the core pieces being discounted should be less than twice the core diameter; however, we use 8 cm for BQ and 10 cm for NQ. Also, in the RQD process the number of natural fractures in the ore is reported.

Table 12.2 gives an overview of the study of the RQD. The RQD for B2 and Bn zones is relatively good, and so is the footwall of B2 Zone, with RQD ranging from 70% to 88%; however, A3 Zone and the hanging wall of Bn Zone are not so good, with RQD between 45% and 55%.

Table 12.2 The Study of the RQD

ZONE	RQD (%)
B2	87.3
Bn	70.8
B2-Bn Footwall	88.3
B2-Bn Hanging wall	44.6
A3	53.9
A3 Hanging wall	53.8
A3 Footwall	54.7

12.7 Cementing of Drill Holes

In accordance with the Quebec mining regulations, after the drill holes are completed and surveyed, they are cemented either at the collar (over a 6-metre length). Sometimes, when those holes are suspected to cut through future developments works, they will be completely filled using a grout cement mixture. A contractor completes the plugging of the borehole. The list of cemented holes is also kept in a handwritten registry stored in the geology vault. The hole cementation is also registered in the database of the DH logger system and identified on the front page of every drill-hole log.

12.8 Collar Surveying

Key exploration data contained within the study area include diamond drilling, drill-hole survey data, and underground development. The spatial location of most of these data is usually defined with reference to the East Amphi grid systems, in which the heading used as north makes 38°E with the Geographic North Pole. This grid system was established in 1987-1988, during the Breakwaters surface and underground exploration program, and was the main reference grid for all underground data as well as most surface data collected by Richmont Mines, McWatters, Placer Dome and Breakwaters.

Procedures for surveying diamond drill-hole collars from the surface have varied considerably across programs. The information from most programs is relatively complete and is shown on the front page of the drill logs. The collar locations for the holes drilled from 1937 to 1948 were originally determined from measurements with a chain on a cut grid. The collars that are known to have been surveyed are from the Sulpetro-Breakwaters works, in 1986 and up to the Richmont Mines surface and underground drilling programs of 2004-2005.

Underground drilling, the establishment of the back and foresights is marked using surveying instruments. The contractor sets the diamond drill onto the collar and aligns the drill along a string tied taut between the front and back sight pads. The plunge of the drill is fixed using a spirit level.

Once the hole is completed, the surveying department that who retums to the collar location of the hole and directly measures the final coordinates, azimuth and plunge. These data are entered into both a handwritten drill-hole registry and an electronic databank.

12.9 Down-Hole Surveying

Procedures for down-hole surveying have varied with time. Down-hole surveying was conducted mainly with a Tropari, with some acid tests as done by McWatters in 1998-1999. Placer Dome used the light log survey in 1995-1996. Before that, Breakwater and Noramco used acid tests, occasionally with Sperry Sun and only once by Gyroscope. Down-hole surveying for the Breakwaters underground drilling was conducted mainly with acid tests.

During the 2004-2005 Richmont Mines surface drilling program, down-hole Reflex tests were recorded for every 30 metres down-hole, with azimuth readings referenced to magnetic north. A-52° azimuth correction was applied to all readings to adjust for the 14° magnetic declination and the 38° rotation between True North and Mine Grid North. The drillers used a susceptibility meter to position Reflex tests, avoiding magnetite-rich intervals. All the surface diamond-drill holes, drilled by Forage Benoit and Mercier, used 3-metre-long NQ diameter core barrels with two 18-inch stabilizing shells.

The underground drill holes for 2004-2005, drilled by Orbit Diamond Drilling, used thin-walled BQ tubing and one 3-metre hexagonal core barrel with a 6-inch-long stabilizing shell, except for A3 Zone drilling, where azimuth holes were produced, using an 18-inch stabilizing shell and two 3-metre hexagonal core barrels. For underground holes exceeding 150 metres, mainly produced below the 200 level, we used one nine-inch stabilizing shell and two 3-metre hexagonal core barrels.

When a drill hole is completed, the collar is surveyed and a measurement of dip using the acid test sometimes reflex at 30-metre spacing for short holes, tropari or reflex at generally 30-metre spacing for holes exceeding 150 metres. The history of surface drill holes intercepted in the drifts and ramp shows that the deviation is generally negligible at the East Amphi project.

13.0 SAMPLING METHOD

Sampling of gold mineralization within the study area has included surface and underground diamond drilling, chips, muck and test holes. Diamond drilling and chip sampling are the initial methods of collecting a continuous series of samples through zones of mineralization on a regular pattern. The 2005 Mineral Resource and Reserve estimate is supported by surface and underground diamond drill hole samples and chip sampling.

For the muck, this type of sample is the piece of rock sampled after the blast. It is used to evaluate the grade estimate during mining and development for a large volume of broken ore. Although muck sample results were not used in Mineral Resource estimates for East Amphi, they were used to monitor the grade of mineralized material being sent to the processing plant (because mill reconciliation is used to calibrate resource estimates, it is appropriate to briefly describe muck sampling procedures).

In 2004-2005, a total of 25,643 samples was taken. Table 13.1 presents the distribution of the type of sample. All samples in 2004-2005 were sent to ALS Chemex Chimitec of Val-d'Or, certified ISO 9001:2000. For the drill hole, the sample represent 22,535 metres of core was sampling for 62.3% of total drilling: 16,015 metres for the underground sample (76.4% of total UG drilling) and 6,768 metres of surface drilling (44.2% of total surface drilling).

Table 13.1 Sample Distribution by Type – 2004-2005 Exploration Program

UG DDH	SURFACE DDH	CHIPS	MUCK	TEST HOLE	TOTAL
16,286	6,249	1,076	1,981	51	25,643

13.1 Core Sample Collection

The method for sampling drill core (from 1987 to 1999) shows very little variation over time. Sampling was carried out with samples that typically varied between 0.25 and 1.5 metres and that did not necessarily coincide with geological boundaries. Concerning drilling programs before 1948, lengths of sampling characteristically show extreme variations from 0.1 to more than 2.5 metres, and sampling was very spotty.

The revised sampling approach by Richmont Mines was planned to coincide with lithological contacts. Each analysis is connected to a geological description in the log book. All core sampling in 2004-2005 was marked up and tagged by a geologist using three-part sample tags supplied by the commercial laboratory. One part was retained by the geologist and used for data entry, one was placed in the core box to delimit the beginning of the samples taken and one was sent along with the sample to the laboratory. The width of most 2004 and 2005 samples was 0.96 metre. The samples are taken over a length of 1.5 metres maximum and 0.50 metre minimum. A few samples have been taken with a length below .5 metre for different special reasons, mainly for ore distribution understanding. Samples of ore must always be properly bordered by samples of waste. Should an anomalous value be returned from an isolated sample, the geologist is required to return to the core interval and take additional bordering samples. Generally samples 1.0 meter-long are purposefully taken on the borders of obvious ore zones in order to minimize the effect of sample contamination of wall rock by high-grade ore.

Core Size

All of the surface diamond drilling completed in 2004-2005 at East Amphi are NQ size (60.3 mm diameter) core using industry-standard wire-line methods. All the diamond drill holes from under ground were BQ size (132 mm diameter).

Core Storage

The 2004-2005 drilling program was carefully sawed in half (longitudinally, in order to obtain a representative sample), and the witness portions of the drill core were stored and catalogued for future reference purposes in the core library located at East Amphi.

Core Sampling

Once the drill holes samples have been extracted, the method for taking core samples is as follows:

1. The core is washed with fresh water using a hose.
2. Once the geology and location of the samples have been described, the geologist carefully marks the start and end of the sample directly on the core with a coloured wax crayon while the core is still intact in the core box.
3. A sample tag, specially made of waterproof paper and indelible ink, is placed at the beginning of the sample interval. Each sample number is unique.
4. The core is generally sampled over regular intervals that vary between 30 cm and 1.5 metres.
5. Samples are measured to the nearest tenth of a metre, but sample intervals have to coincide with major lithological boundaries.
6. The whole core is split into two using a saw with a diamond.
7. The diamond saw is properly cleaned with a brush prior to cutting every sample.
8. As the core sample is cut in half lengthwise, the samples chosen for assay are collected in individual plastic sample bags. The other identical half core witness sample is replaced carefully in the box according to its original orientation (the correct end of the core up hole, for example). One of the two sample tags is placed in the plastic bag, which is then securely stapled shut.
9. The other identical sample tag is stapled to the core box at the end of the marked sample interval.

A sample request form is to be completed prior to dispatch of the samples. The request specifies the name of the laboratory, the person making the request, the date, the sample series, the elements to be assayed (gold, almost exclusively), the units in which the results should be reported (grams per tonne), the analytical method and any special instructions.

Core Sample Quality and Representativeness

At East Amphi, samples recovered through diamond drilling are of high quality (the mineralization in the core is intact, with no possibility of loss due to washout). Rarely, the core can be ground over short lengths of less than 0.5 metre and a sample not recovered. Overall drill core samples recovered for East Amphi (including the historical samples) can be considered to be representative. Each core sample weights an average of 1.15 kg for AQ, 1.37 kg for BQ and 2.52 kg for NQ.

13.2 Panel Sample

Sampling Method

The panel (or chip) sampling method generally consisted of taking horizontal representative samples, from 0.5 metre to 1.5 metres long, of the geology (units or alteration) that was exposed either in the face or in the adjacent walls. At East Amphi, we systematically chip sample primary draw point cross cut on both walls, starting at two rounds before ore grade was expected from drilling data and up through the mineralization zone to the end of the crosscut. For secondary draw points cross cut, the same method was applied but sampling was done on one wall only. The average length was 1 metre. The geologist/technician gave a 5 kg sample was chipped with a hammer for a zone of 2 metres vertically by 0.5 to 1.5 metres horizontally. Each geological unit was sampled proportionally to its importance in order to obtain a representative sample. Samples could vary in weight from around 1.5 kg to 6 kg (or more), with an average of 2.93 kg.

The geologist noted the location and the geology of each panel in a sample notebook. Sample numbers and results were plotted on hand-drawn maps, which were validated regularly.

13.3 Muck Sample

Sampling Method

The mucking was done by taking one sample of broken rock pulling either four 6-tonne bucket. With this method, a 5 kg sample was taken by the operator from every 24 tonnes of material extracted. This method gave a representative sample for the broken ore. A sample tag was included in every 5 kg sample. Up to now, we average 5.05 kg samples. Each round represented 200 tonnes of ore; therefore 8 to 10 samples of muck were taken in each round. The result for each round is given by the mean of the 10 assays. Results from muck samples were not used in the East Amphi Mineral Resource and Reserve Estimate.

14.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

Historical procedures for sample preparation varied. Most drill core samples collected before 2004 were split with manual and hydraulic core splitters. Routine assaying was often conducted on final sub-samples that weighed only one-half assay-ton and very little metallic and gravimetric assaying was carried out.

For the 2004-2005 exploration program, Richmont Mines developed a quality control program for sampling and shipping, and monitored QA/QC measures from commercial analytical laboratories. In 2004, a core logging facility and a core storage area were established on the East Amphi project. Samples were collected and prepared for shipping to the laboratory in a sample room adjacent to the core logging area by a sample technician. When the drill core was sawn, one half was placed into a plastic sample bag along with a sample tag and sealed with a plastic tie wrap. Individual sample bags were sealed with tape. The samples were placed in large rice fibre bags that were sealed with tape and wire and placed on pallets. Samples were shipped by Parco Transport or were picked up at the project site by the commercial laboratory. Final sample preparation and assaying was conducted at ALS Chemex Laboratories in Val-d'Or. A few samples were also assayed at Bourlamaque Laboratory.

14.1 Chemical Assay Laboratory

Several chemical assay laboratories were used in the past. ALS Chemex Chimitec Laboratories, in Val-d'Or, was selected to conduct sample analyses on the East Amphi project for the 2004-2005 exploration program. The successive stages of analysis for all types of samples (drill core, chips and muck) are briefly described as follows:

14.1.1 Sample Preparation

All samples received by Chemex Laboratories are tagged with an Internal Sample Control Number and entered into the Laboratory Information Management System (LIMS). The benefit of this system is the reduction of human error by controlling the labelling, sample throughput, and data entry of results from the instrumentation to the LIMS program. The system has the ability to generate all reports both on certificate and electronic formats.

The samples are dried. The routine procedure used for all samples involves crushing the total sample to 70% passing -10 mesh (1.7 mm-1700 micron) and then splitting out a sub-sample weighing approximately 250 g for pulverization. Sub-samples are then pulverized to 85% passing -200 mesh (75 micron) before a 30-g sub-sample is taken.

14.1.2 Sample Analysis

Samples are then analyzed by fire assay with atomic absorption finish. Samples exceeding seven grams per tonne gold (7 g/t Au) on the first analysis are re-analyzed with a gravimetric finish. In the latter case, 30 g of representative material are then analyzed by fire assay and gravimetric finish. Rejects and pulps are preserved by the laboratory or stored at the East Amphi site.

14.1.3 Laboratory Certification

All ALS Chemex laboratories in North America are certified ISO 9001-2000 for the "Supply of assays and geochemical analysis services" by BSI Quality Registrars.

ALS Chemex also takes part assiduously in the "Proficiency Testing Program – Minerals Analysis Laboratories" (PTP-MAL) and has done so for many years. A certificate demonstrates the success of ALS Chemex in this program for the following elements: Au, Ag, Cu, Zn, Pb, Ni, Co.

This success is compulsory to obtain the ISO 17025 certification.

14.1.4 Quality Control (QC) by Laboratories

As mentioned previously, ALS Chemex Chimitec Laboratories use the "LIMS" data management system. The main functions of this system are to generate the list of samples and the worksheets, print the tags for the sample pulp bags, and manage the insertion of blanks, control standards and duplicates. Weighing results, calculations and result validations are also dealt with automatically by the system.

The quality control procedures used by the laboratory at different stages of the process are as follows:

- Crushing, pulverizing, weighing: daily check;
- Fire assays: each batch of 84 fusion crucibles includes one blank, two standards and three duplicate samples along with 78 client samples.

14.2 Quality Control and Quality Assurance Analysis

This section presents an analysis of the quality assurance and quality control (QA/QC) data collected during the 2004-2005 exploration program for the East Amphi project. Evaluation of QA/QC data addresses the three principal concerns of analytical determination protocols, namely: Contamination, Accuracy, and Precision, as measured by the results obtained from Field and Analytical Blank Standards, Certified Reference Standards and an assortment of specific duplicate samples collected and/or prepared, in addition to, the regular samples submitted to the laboratory. QA/QC results internal to the laboratory (ALS Chemex – Chimitec) were evaluated independently from QA/QC results that were initiated externally by Richmont Mines.

14.2.1 Past Drilling

Assay results of drill core samples from old logs are taken into account, even if their validity is not entirely guaranteed and sampling techniques used are not clearly established. Historical procedures for quality control have included the use of quality control standards as well as re-sampling of core, rejects and pulps. Duplicate data has been collected for most past programs. In most cases duplicates were done on a non-blind basis and were not checked by a third party laboratory.

Data for the Breakwater and McWatters drill programs consisted of blanks and two types of in-house reference material. Standards were inserted into the sample stream at the rate of one per twenty samples and blanks at the rate of one per forty samples. The reference materials used were MV-1 and RB-1 and covered a grade range of 1.4 g/t Au to 1.9 g/t Au. No major problems with blanks or standards were noted.

McWatters checked assay results given by McWatters' own laboratory at the Kiena mine. Abilab Laboratory, in Val-d'Or, proceeded with sample assays using current methods and standards. Certain discrepancies were noted and were subjected to close verification. Samples for which fire assay results were above 3 g/t Au upon first analysis (A.A. finish) were re-analyzed, starting from a total pulverization of the reject (if sufficient). This served to confirm the initial results overall, except in a few cases attributable to the presence of coarse gold. In the latter case, the value that was eventually selected was the one resulting from total pulverization, considered as the most reliable.

14.2.2 2004-2005 Richmont Mines Exploration Program

QA/QC measures for the 2004-2005 exploration program consisted in the insertion of blanks and standards for each drill-hole, re-assaying pulps for samples that yielded assay results over 7 g/t Au and monitoring the results of QA/QC measures from the laboratory procedure. No quartered core duplicates were done and no systematic re-assaying by another laboratory was done. Only a few samples (30 samples) were sent to Techni-Lab in Sainte-Germaine-Boulé for verification. All data for the Richmont Mines 2004 and 2005 program are shown on certified assay certificates from Chemex Laboratories.

Seven type of certified reference materials (CRM) were acquired from Rocklabs. Of these, 33 samples at 1.315 g/t Au, 102 at 2.643 g/t Au, 29 at 4.048 g/t Au, 39 at 4.823 g/t Au, 100 at 5.911 g/t Au, 37 at 7.378 g/t Au, and 32 at 8.367 g/t Au were sent for laboratory checks. The blanks were collected by Richmont Mines geology staff, from barren BQ and NQ diamond drill-holes. For the 2004-2005 exploration program, a total of 372 standards and 381 blanks were sent to the laboratory. One CRM sample and one blank sample were introduced in each drill-hole by Richmont Mines staff.

As mentioned, for 2004-2005 drill-holes, check assays were carried out for all samples which yielded assay results exceeding 7 g/t Au, using another method which gives a better precision; the method used was fire assay with gravimetric finish. The results of this method were used for the reserve and resource estimation in conjunction with the first method but all AA data over 10 g/t Au were rejected.

According to QA/QC reports from ALS Chemex, one blank and two standards were inserted into the sample stream for every 78 client samples processed. Moreover, the laboratory reassayed three samples in each batch. ALS Chemex used 11 different standards during the 2004-2005 exploration program. The standards used were OXE-20, OXE-21, OXF-28, OXH-19, OXK-18, OXK35, OXL-17, OXM-16, OXP-32, SI-15, and Sp-17, and covered a grade range from 0.55 g/t Au to 18.13 g/t Au.

14.3 QC/QA Conclusions

The assays supporting the East Amphi Mineral Resource Estimate are based on sample preparation and analytical protocols that meet standard industry practice and are reasonable and acceptable. Horvath and Pelletier completed an evaluation for the quality control sample analytical results from East Amphi. The following discussion provide from their report.

Contamination

Contamination was monitored internally by ALS-Chemex Chimitec laboratories by means of an analytical blank. The result indicates no contamination has occurred during the analytical process.

The field blank standard, submitted by Richmont Mines also monitored for potential contamination during sample preparation. Results from the field blank standards are not 100% conclusive regarding potential contamination during sample processing. At nine (9) samples report significant gold concentrations. It is difficult to quantify whether these “failures” are erratic high grade values characteristic of the standard or a result of carry-forward contamination during sample processing.

Accuracy

Accuracy of the laboratory determinations is monitored by the regular submission of blind certified reference standard (CRM) samples within each batch of regular samples. In addition, the laboratory monitors accuracy by including samples of its own internal certified reference standards within each batch of samples processed.

The standard results internal to the laboratory demonstrate that for the most part the accuracy of the laboratory has been acceptable over the course of the sampling/assaying program. Good accuracy was demonstrated by the laboratory during the sampling/assaying programs. The results from Richmont Mines external CRM samples is has been generally very good during the course of the sampling and assaying programs.

Both the internal and external verifications demonstrated excellent levels of accuracy during the 2004-2005 program.

Precision

Precision is the measure of reproducibility of a sample value. Precision is monitored and measured by comparing results of assorted duplicate samples. A detailed Thompson-Howarth incremental precision analysis of chip, muck, surface and under ground diamond drill core sample on the pulp duplicate results using a variety of determination methods (i.e. FA-AAS and FA-gravimetric) has clearly demonstrated that the East Amphi ores contain a nuggety gold (Figure 14.1).

Several conclusions can be drawn from the Thompson-Howarth Plot of the various original sample type pulps:

- All sample types are demonstrating similar rather poor levels of precision ranging from 29-42% error for the 50 g pulp fusion weights used. For pulp sample duplicates, this precision level should be less than 10% and preferably in the 3-5% error range.
- The smaller sized drill core samples (1-1.5 kg for underground core, 1.0-2 kg for surface core) provide slightly better precision levels ranging from 29-34% error for duplicate assays of the same pulp in comparison to the larger chip/channel and muck samples (1.5-2.5 kg for chip/channel samples and 3-4 kg for muck samples) displaying precision levels from 40-42% error.
- For all sample duplicate types, no significant additional loss of precision is realized until grades below 1 g/t Au.

The results demonstrate that relatively poor precision is being realized in the analytical results from re-assaying of the same sample pulps regardless of the original sample type. In addition, the smaller sized original samples demonstrate slightly better precision than the larger sized samples indicating that representation or homogeneity of the larger samples is not being achieved in the pulp samples.

The precision results would suggest that the size of the sub-sample split taken after coarse sample crushing is too small and the resulting pulp not representative of the larger original samples. Hence the 50 g individual pulp assays produce more precise results for the more homogenous smaller original samples. It is important to note that the results do NOT suggest that field samples should be smaller but rather that larger coarse crush sample splits are pulverized.

The results demonstrate further test work is justified to provide duplicate samples of the coarse crushed product (i.e. 2nd split for pulverization) as well as the original field sample (i.e. collect second sample or use other half of split core). Duplicate samples at each of the pulp, coarse crush and field duplicate stages, would provide data to evaluate total sample precision as well as incremental loss at each stage and better optimize the sampling/ preparation and assay process.

Over 800 duplicate AAS versus gravimetric assays of the same sample pulp were analysed. The Thompson-Howarth Precision Plot for the precision (error %) versus grade concentration (g/t Au) for each of the AAS and gravimetric sample duplicates is shown in Figure 14.2. The plot demonstrates that below grades of approximate 3.5 g/t Au the AAS results are more precise than gravimetric results. Re-assaying of samples by gravimetric methods is not required below grades of 3.5 g/t Au.

Recommendations

Horvath and Pelletier (2006) recommended:

- Future sampling/assaying programs should characterize the field blank standard in a small round robin survey. In addition, the “potential” failures identified should be investigated to determine whether any carry-forward contamination may have occurred.

- That the split size be increased from the 250 g to 1000 kg size. Future sampling/assaying programs should include collection and assaying of coarse crush sample duplicate splits as well as field duplicate samples for evaluation of total and incremental precision levels and better identification as to the primary source of error.
- Re-assaying of initial AAS determinations should be completed using gravimetric methods for results reporting greater than 3.5 g/t Au.

15.0 DATABASE INTEGRITY AND VERIFICATION

For the purpose of modelling the resources, Richmont Mines started with the database received from McWatters (reference PXDBEA database). This database was evaluated and used by SNC Lavalin in 2002 to complete the reserve and resource evaluation. This database included all the drill holes and the underground sampling at the project site. SNC Lavalin was verified only for data-entry errors and to measure the influence of anomalies such as high-grade samples and the nugget effect, typical of gold ores. These studies are available in the report from SNC Lavalin on SEDAR.

Richmont Mines verified the database, some checks against the original data were completed (5% of assays) and a few modifications were completed. The modifications are:

- Standardization of geological code
- Skip 565 doubloon (doublon) in the assays table
- 3569 assay modifications, especially right codification for assays below 0.005 g/t Au with correction for ‘’ < ‘’
- Importation of 3569 sample numbers
- Re-importation assays from the original DBase (dbf file) of the assays

No significant omissions were founded during this evaluation. It was concluded that the database’s integrity appears to be very good. The new database (reference GD_PXDBEA) was used for entering the new data from the 2004-2005 exploration program.

At the East Amphi project, all the geoscientific data are gathered in a single Access database whose structure and programming were completed by the personnel of East Amphi. This database is linked with the Gemcom software used on the site for geological and resource and reserve evaluations.

Richmont Mines used a number of queries in MS Access, the Gemcom data validation routine, and 3-D visual inspection to validate the drill-hole database. Internal procedures were developed in order to validate the information in the database. All work that is carried out by the department of geology of the East Amphi, from initial data acquisition to drawing preparation, follows a pre-established and rigorous procedure, with specific checks to ensure validity. Access to the database is strictly controlled and limited to duly authorized personnel in order to ensure its absolute integrity. The database was backed up over a network on a twice-a-day basis during the 2004-2005 exploration work.

All of the assays from 2004-2005 were sent electronically to Richmont Mines’ East Amphi project. The sample intervals, sample numbers, and other information on the sample tags were manually entered into the computer by Richmont Mines’ personnel. The assays were digitally transferred in the database at times by a program developed by Richmont Mines’ personnel. The Gemcom database is valid and acceptable for supporting resource estimation work.

16.0 ADJACENT PROPERTIES

Properties adjacent to the East Amphi project are in a geological environment similar to that of the East Amphi deposit. The Fourax property, held by Richmont Mines, is adjacent and contiguous to the east of the East Amphi property. It contains interesting gold occurrences. Gold mineralization on the Fourax property is known to occur in two distinct settings: within a series of porphyry dykes and sills south of the Cadillac Break comprising the Western Porphyry Zone, and within the Cadillac Break itself. Here the mineralization is hosted within a series of tuff horizons in the Townsite Diorite Zone and the Fourax Shear Zone. These units lie along the southern contact of the Cadillac Break north of the Western Porphyry Zone (Figure 8.2). These historical resource and reserve estimates presented in this section should not be relied upon as they likely do not conform to NI 43-101 standards and definitions and not compliant with CIM standards. They are included in this section for indicative purposes only and should not be disclosed out of context.

Townsite Diorite Zone

The Townsite Diorite Zone has been partially delineated in the centre of the property. Gold mineralization occurs in a series of stratigraphically continuous tuff horizons with sulphide-rich zones (5-10% disseminated pyrite and pyrrhotite) and with quartz stringers and veinlets that cross-cut the cleavage. Mineralization generally occurs in two closely-spaced parallel horizons trending N295° and dipping 80° north. The dip steeps to vertical in the northwestern corner of the property. These tuffs have an average width of 6.5 feet.

Ingham calculated total inferred resources of 388,279 tonnes at 3.43 g/t Au (43,650 ounces) in 1980 for the two mineralized lenses. The mineralized horizons are open along strike and at depth. This resource evaluation corresponds to drill-indicated inferred resources.

Western Porphyry Zone

The Western Porphyry Zone lies approximately 915 metres west of the Townsite Diorite Zone. Gold mineralization in this part of the property is contained within a series of unconformable felsic intrusive dykes and sills, which range in composition from granite and feldspar porphyry to diorite. These intrusive bodies lie south of the Cadillac Break and appear to form apophyses or lens-shaped bodies emanating from a major granitic porphyry intrusion. These dykes are approximately parallel to the Cadillac Break trending N295°.

The gold-bearing zones are characterized by moderate to high levels of pervasive silicification and pyritization associated with high background gold values, and by the coincidental alignment of the zone which parallels the intrusive contacts of the granodiorite. High gold values are generally attained at the periphery of the felsic intrusive bodies around conformable and subvertical crushed and bleached zones 3 to 12 metres wide. The gold, which is closely associated with pyrite, is disseminated and contained in numerous tensional quartz veinlets. The presence of mafic and ultramafic volcanic material appears important in that the most consistent and highest gold values generally appear in their close proximity. Also present in these alteration zones are pink feldspar phenocrysts, abundant carbonate, sericite and chlorite, with traces of chalcopyrite, magnetite, molybdenite, galena, and native gold.

W.M. Kilbourne, an independent geologist, carried out an evaluation of tonnage and grade in 1987. His interpretation resulted in the correlation of 8 continuous gold-bearing zones. W.M. Kilbourne estimated resources at 354,443 tonnes of mineralized material grading 6.86 g/t Au (for 78,000 ounces) utilizing a 15 metres area of influence. Alternately, the resources increased to 997,800 tonnes at a grade of 6.5 g/t Au (for 144,450 ounces) utilizing a 30-metre area of influence.

The following criteria were used by Kilbourne (1987), in calculating ore resources:

1. a lower cut-off grade of 3.43 g/t;
2. a minimum mining width of 1.5 metres;
3. high assays were cut to 34.29 g/t;
4. an area of influence of 15 metres and alternately 30 metres;
5. a tonnage factor of 2.8 t/m^3 per tonne for rock density;
6. the plane of the zone lies in the plane of the longitudinal;
7. no dilution factor was taken in account.

The potential of the Western Porphyry Zone is excellent; the mineralized zones remain open both along strike and at depth.

Fourax Shear Zone

The Fourax Shear Zone lies along the southern contact of the Cadillac Break north of the Western Porphyry Zone. A quartz breccia unit consisting of 50% milky white quartz and 50% felsic intrusive clasts hosts the gold mineralization. The horizon was intersected over a core length of approximately 140 feet (true width is approximately 90 feet) in hole FX-8858.

Units of intensely sheared mafic to ultramafic tholeiitic volcanic rocks and intercalated finely laminated sediments and magnetite iron formation are also found within the Fourax Shear Zone. They are probably representative of the regionally designated Piché Group.

Between sections 350E and 1800E, Geologica Consulting (2002) calculated inferred resources, using all available historical and recent data, at 255,078 tonnes grading 2.74 g/t Au (for 22,800 ounces), using a 1 g/t lower cut-off grade. This horizon requires further drilling along strike and at depth to ascertain its potential as a continuous mineralized zone.

The Parbec property, held by Globex, is adjacent and contiguous to the west of the East Amphi property. This property presents interesting gold mineralization in the Camp zone, the Number 2 Zone and the Discovery zone. A geological resource of 412,773 tonnes grading 4.63 g/t over an average width of 1.9 metres was defined.

Past producers in the Malartic Camp, namely East Malartic (2,836,950 ounces of gold), Barnat-Sladen (1,213,434 ounces of gold), Canadian Malartic (1,076,053 ounces of gold) and Malartic Goldfields (1,700,000 ounces of gold) are located along the same stratigraphic horizon and associated with the Cadillac-Larder Lake Break (Trudel et Sauvé, 1992).

17.0 MINERAL PROCESSING AND METALLURGICAL TESTING

The 2004-2005 exploration program included extraction of a bulk sample. This exercise served as a final step in the grade validation of East Amphi reserves. It was extremely important to complete a bulk sampling program to verify the grade and test the behaviour of the stopes particularly with respect to the dilution rate and ground conditions.

The ore from East Amphi was hauled by truck to the Camflo mill, also property of Richmont Mines and located at an approximate distance of 13 km from the mine site. Transportation of the ore was ensured by Transport Nord-Ouest Inc. The Camflo mill is a traditional gold recovery mill using a conventional Merrill-Crow type process, with circuits for crushing, grinding, gold cyanidation and precipitation using zinc powder. The mill has a rated capacity of 1,180 tonnes per day.

During 2005, three batches were processed for a total of 40,581 dry tonnes; the first two batches were development ore, and the last one included four stopes and development ore. The first batch of 6,349 tonnes was processed from July 24 to July 30, 2005, the second batch of 8,312 dry tonnes was processed from September 26 to October 3, 2005 and the third batch of 25,920 dry tonnes was processed between December 7, 2005 and January 1, 2006. For the third batch, 10,256 tonnes of development ore and 15,664 tonnes of ore from four stopes were transported and milled.

For the third batch, specific procedures were agreed upon to insure better gold accounting measures. Although processed sequentially, focus was set on overall bulk sample head grade validation for further data correlation requirements.

The following outlines the major steps taken for the third batch:

- At East Amphi, the ore was stockpiled in five different places, one for development ore and one for each stope. The four stopes were excavated in distinct areas within the outlined reserves providing a diversified global sample.
- Sample batches (development ore and each stope) were trucked separately to the Camflo mill.
- At the Camflo mill, a dedicated pad was assigned to the East Amphi ore in five different places, one for development and one for each stope. The pad was inspected prior to the start of the ore transfer from the East Amphi site. The ground was found levelled and frozen. This ground has consolidated over the years as it serves regularly as an ore stockpiling area. Stockpiled ore is handled via a front-end loader and dumped into the jaw crusher feed hopper. An underlying layer of magnetite base material acted as a reference. Any such material scooped by the loader will be recovered via a belt magnet indicating stockpile bottom.
- Care was also taken at the East Amphi site while retrieving each of the five sample stockpiles. Registered lot tonnages at Camflo mill were crosschecked with mining results confirming adequate overall ore handling procedures.
- The development ore was processed first followed by each of the four stopes. The objective was to clean the mill with the development ore to improve the quality of

testing for the four stopes. All changes in the source of mill feed were noted. There was no cleaning of the grinding circuit prior to or after the milling program was completed.

17.1 Tonnage

East Amphi ore was milled at a rate of 45 tonnes per hour at a targeted fineness of grind of near 80% -200 mesh. While milling throughput is controlled via a weighing scale unit at the rod mill feed conveyor (#8), the official tonnage is measured from the fines ore bin feed conveyor (#5) weighing scale unit. Measurement Canada, an agency of Industry Canada, assures the unit certification. Unit electronics are sealed and only zero adjustments are allowed. Unit accuracy is said to be within 0.5%. Tonnage correction reported by Camflo mill operation was established on this basis and is presented in Table 17.1 for the three lots. If the difference between the rod mill feed conveyor (#8) weighing scale unit and the fines ore bin feed conveyor (#5) weighing scale unit is less than 0.5%, the tonnage from the rod mill feed conveyor (#8) weighing scale unit is used as the official tonnage.

Table 17.1 Tonnage Calculation

EAST AMPHI - BATCH 1	
Registered wet tonnage at conveyor #8	6,474 tonnes
Mean correction factor	1.00
Corrected wet tonnage at conveyor #8	6,474 tonnes
Moisture content %	1,93 (98,069%)
Processed final dry tonnage	**6,349 tonnes**

EAST AMPHI - BATCH 2	
Registered wet tonnage at conveyor #8	8,519 tonnes
Mean correction factor	1.00
Corrected wet tonnage at conveyor #8	8,519 tonnes
Moisture content %	2,43 (97.570%)
Processed final dry tonnage	**8,312 tonnes**

EAST AMPHI - BATCH 3	
Registered wet tonnage at conveyor #5	26,970 tonnes
Mean correction factor	0,379 (99.621%)
Corrected wet tonnage at conveyor #5	26,868 tonnes
Moisture content %	3,53 (96.472%)
Processed final dry tonnage	**25,920 tonnes**

2005 Processed final dry tonnage	**40, 581 tonnes**

17.2 Total Gold Content

17.2.1 Precipitation Circuit Products

Soluble gold was recovered via the Camflo mill zinc precipitation circuit. Precipitate produced over the East Amphi program was smelted onsite on six occasions, once for lot 1, once for lot 2 and four times for the lot 3. Six gold bars were produced and sent to the Royal Canadian Mint for further refining. Approximately 96.8% of the total gold produced from the zinc precipitation circuit is confirmed from recognized refineries. Various smelting by-products were also produced. These include slags, buttons and sample filings which were assayed. Approximately 72.5% of the total gold produced from the by-products is recoverable. Table 17.2 summarizes gold recovered from this circuit.

17.2.2 Mill Tails

An automated sampler unit is installed on the mill tails recovering shift composites that in turn provide daily composite samples for gold analysis on solids and solution fractions. Approximately 2.3% of the total gold was lost to the tailings. Table 17.2 summarizes gold distribution in mill tails.

17.2.3 Head Grade

For the 2005 East Amphi milling program, head grade was calculated on the basis of the information contained within the previous sections. The fine gold distribution is summarized in Table 17.2 and Table 17.3. below. For the bulk sample from lot 3, the head grade averaged 3.98 g/t Au, whereas the head grade for the entire 2005 mill test, including development ore from lots 1 and 2, was 3.70 g/t.

Table 17.2 Fine Gold Content Distribution Summary

EAST AMPHI – BATCH 1	**CORRECTED ESTIMATES**		
		Fine Gold (ounces)	**Distribution (%)**
Mill Tails	Solids	17.081	2.4
	Solution	2.363	0.3
Circuit Products	Refining Products	677.930	95.7
	Refining By-Products	11.172	1.6
Total		**708.546**	**100.0**

TONNAGE (DRY METRIC TONNES)	**OUNCES**	**HEAD GRADE (G/T)**
6,349	**708.546**	**3.47**

EAST AMPHI – BATCH 2	**CORRECTED ESTIMATES**		
		Fine Gold (ounces)	**Distribution (%)**
Mill Tails	Solids	18.324	2.3
	Solution	2.889	0.4
Circuit Products	Refining Products	762.982	95.5
	Refining By-Products	14.580	1.8
	Refining By-Products Lot 1	- 2.04	
Total		**796.735**	**100.0**

TONNAGE (DRY METRIC TONNES)	**OUNCES**	**HEAD GRADE (G/T)**
8,312	**796.735**	**2.98**

EAST AMPHI – BATCH 3	**CORRECTED ESTIMATES**		
		Fine Gold (ounces)	**Distribution (%)**
Mill Tails	Solids	57.144	1.7
	Solution	12.397	0.4
Circuit Products	Refining Products	3,231.638	97
	Refining By-Products	29.25	0.9
	Refining By-Products Lot 2	-9.61	
Total		**3,320.819**	**100.0**

TONNAGE (DRY METRIC TONNES)	**OUNCES**	**HEAD GRADE (G/T)**
25,920	**3,320.819**	**3.98**

Table 17.3 Total 2005 Mill Test

TONNAGE (DRY METRIC TONNES)	**OUNCES**	**HEAD GRADE (G/T)**
40,581	**4,826.10**	**3.70**

17.3 Recovery

Gold recovery of the zinc precipitation circuit at the Camflo mill was established at 97.5%.

17.4 Reconciliation Mining - Milling

The reconciliation table between planning, mining and milling is shown in Table 17.4. The main conclusions are:

- The head grade is 3.7 g/t Au for the entire 2005 mill test, and 4.0 g/t Au for the third lot only.
- The muck grade was lower than the actual head grade by 10.7% for the entire 2005 mill test, and by 10.5% for lot 3 only.
- The mill feed head grade was overestimated by 2.2% for the entire 2005 mill test and overestimated by 3.1% for the third lot only.
- The estimated grade of CMS-surveyed stopes (block model from drill core and chips samples) was lower than the actual head grade by 6.3% for the entire 2005 mill test, and by 4.7% for the third lot only.
- Average grade milled from the four stopes returned a content of 4.14 g/tonne compared to an undiluted grade planned of 3,85 g/tonne and an average grade of 3.80 g/tonne according to outlined tonnage effectively withdraw in model evaluation. This represents an upgrade of over 9% based on CMS measurement evaluation.

Table 17.4 RECONCILIATION

Stope/ Development	Planned tonnage	Planned tonnage	Grade	Grade	Grade	Grade	Grade	Tonnes CMS	Grade CMS	Estimated Contained Tonnes not recovered	Tonnes total	Tonnes transported by TNO	Dry tonnes processed
	(without dilution)	(dilution 10%)	(without dilution)	(diluted)	(muck)	(mill fead)	(head grade)						
	(tm)	(tm)	(g)	(g)	(g)	(g)	(g)	(tm)	(g)	(tm)	(tm)	(tm)	(tm)
Development July 05 batch #1	6,150	6,150	3.34	3.34	3.34	4.64	3.47	6,150	3.34		6,150	6,476	6,349
Development Sept. 05 batch #2	8,666	8,666	2.58	2.58	2.58	2.94	2.98	8,666	2.58		8,666	8,519	8,312
Development Dec. 05 batch #3	10,600	10,600	3.69	3.69	2.97	3.43	3.58	10,600	3.69	300	10,900	9,913	10,256
Total	**25,416**	**25,416**	**3.23**	**3.23**	**2.93**	**3.57**	**3.35**	**25,416**	**3.23**	**300**	**25,716**	**24,908**	**24,917**
CH200-13 batch #3	3,132	3,445	5.58	5.32	2.71	3.08	3.19	2,750	5.52	50	2,800	2,790	2,754
CH200-11 batch #3	4,472	4,919	3.91	3.80	3.81	3.82	3.93	4,709	3.94	125	4,834	3,793	4,355
CH175-13 batch #3	4,931	5,424	3.21	3.16	4.95	4.50	4.61	6,203	3.17	300	6,503	4,948	4,975
CH200-15 batch #3	4,181	4,599	3.66	3.57	3.98	4.85	4.96	3,736	3.72	150	3,886	3,770	3,580
Total Stope	**16,716**	**18,387**	**3.95**	**3.84**	**4.02**	**4.14**	**4.25**	**17,398**	**3.87**	**625**	**18,023**	**15,301**	**15,664**
Grand total batch #3	**27,316**	**28,987**	**3.85**	**3.78**	**3.60**	**3.86**	**3.98**	**27,998**	**3.80**	**925**	**28,923**	**25,214**	**25,920**
Grand total 2005	**42,132**	**43,803**	**3.52**	**3.47**	**3.35**	**3.79**	**3.70**	**42,814**	**3.49**	**925**	**43,739**	**40,209**	**40,581**

18.0 MINERAL RESOURCE AND RESERVE ESTIMATES

The 2005 Mineral Resource and Mineral Reserve Estimates combine grade and tonnage information. This estimation were carried out by the technical staff of Richmont Mines under the supervision of Qualified Persons as per National Instrument 43-101.

The estimate of mineral resources and reserves was made in compliance with the recommendations and regulations of National Instrument 43-101. The mineral resources and reserves were grouped according to the classification established by the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") and adopted by the CIM Council.

The CIM Standards describe completion of a Preliminary Feasibility Study as the minimum prerequisite for the conversion of Mineral Resources to Mineral Reserves. A Preliminary Feasibility Study is a comprehensive study of the viability of a mineral project that is advanced to a stage where the mining method has been established, and where an effective method of mineral processing has been determined. This Study includes a financial analysis based on reasonable assumptions of technical, engineering, operating and economic factors and evaluation of other relevant factors that are sufficient to determine if all or part of the Mineral Resources may be classified as Mineral Reserves.

The source data and the parameters used for the calculation of mineral resources and mineral reserves correspond to acquired knowledge, best estimation and the situation as at December 31, 2005. The technical staff of Richmont Mines calculated mineral resources and reserves by zone and by level, according to local conventions used at East Amphi as follows:

- Zones: B2-Bn, B1-B1', A1-A2-A2', A3, and P;
- The B2-Bn zones are divided in three areas:
  - Mine Area: above 4,800 elevation (200 level) and between sections 1000 E and 1225 E;
  - Below 4,800 elevation (200 level) to 4,725 elevation;
  - Explo Area: below 4,725 elevation and west to section 1000 E;
- Levels: 100 (4,900 elevation), 125 (4,875 elevation), 150 (4,850 elevation), 175 (4,825 elevation), and 200 (4,800 elevation).

Before defining reserves and resources, a global resource evaluation was performed to identify the mineralized zones that meet technical parameters fixed by the head office. This global resource evaluation was completed for all the zones and the same technical parameters were used except for the method of evaluation. Subsequently, the global resource was reviewed by the engineering department assisted by the geological staff to define the reserves which could be economically extracted and to propose a mining plan. At the end of this exercise, a remaining resource was recalculated, which for the moment needs additional work to complete the evaluation before being transferred to reserves.

A series longitudinal sections are presented in Appendix 6 to illustrate the mineral reserve and resource blocks.

18.1 Estimate of Global Mineral Resources

A Global Mineral Resource estimation was performed for all zones. The global inventory of mineral resources at East Amphi is comprised between sections 730E and 1690E according to the local coordinate system. At East Amphi, global mineral resources are estimated using the polygonal method (on longitudinal section) for the A1-A2-A2', A3, and B1-B1' zones, and using inverse distance to power (IDP) block modelling methods with Gemcom software for the B2-Bn and P zones. However, the technical parameters used were the same for all zones. This section describes the processes common with each method.

The diamond drilling unit of the Gemcom software is used to generate and trace view plans, cross-sections and longitudinal sections. Data input for drill-hole descriptions is carried out using an Access program developed by Denis Sarrazin specifically for East Amphi geologists. The long section used for global resource evaluation was transferred to AutoCAD to complete the estimation for the A1-A2-A2', A3, and B1-B1' zones. Drilling data are plotted on a set of cross-sections equally spaced and oriented N-S (mine grid), perpendicular to the mineralized zone. For the B2-Bn zones, cross-sections were spaced every 5 metres, and 10 metres for the A1-A2-A2', A3, and B1-B1' zones. The drill-hole spacing increases at depth (below 4,800 elevation) to 20 metres with some areas of scattered drilling (below 4,725 elevation). Plan views were done on 12-metre spacing for the B2-Bn zones. All sections were drawn using Borehole Manager, Gemcom and AutoCAD software.

A geological interpretation is first made on both cross-sections and plan views before a proper global resource estimate can be made in order to ensure a good understanding of the structures and mineralized zones. A shell of 3 g/t Au is then drawn and considered as a wireframe. In the geological interpretation at East Amphi, there is a dominant regional rake steeply plunging west, but it appears that a minor local rake axis is going in another direction plunging east. This interpretation was not integrated in the model.

Results of the interpretation for all zones were digitized and only the B2-Bn and P zones were linked as 3D solids using Gemcom software. For the resource model done in Gemcom, a composite table was prepared using the current geological interpretation to group assays by zones of economic interest.

18.1.1 Database Review

Richmont Mines has estimated the Global Mineral Resource using all information from its 2004-2005 exploration program including information from underground openings and historical diamond drilling. These estimates (information inside the wireframe of the 3 g/t Au cut-off envelope) are supported by 440 underground drill-holes and 106 surface drill-holes for a total of 546 drill-holes. These holes came from following the drilling program of Breakwaters, Placer Dome, McWatters and Richmont Mines. The distribution of holes and assay results used for the estimation of each zone is presented in Table 18-1. Moreover, for the B2-Bn and P zones, 110 chip samples stripes (channel sample) were used from the 2004-2005 underground exploration and development program. The quality of the database has been discussed in detail in a previous section 15.

In the old underground development (B1 and A1-A2 zones), a large number of channel samples (close to 3,000) were taken. These are biased toward a higher average grade value (~5 g/t Au) since the drift is often located directly in the middle of the high-grade ore. However, this data could not be used by Richmont Mines (SNC-Lavalin also excluded this information) because it does not provide information on the full width of the potential ore. Also, 82 historical drill-holes were excluded due to inaccurate location (Appendix 7).

18.1.2 Basic Statistics

The composite control for the global resource evaluation is presented in Table 18.1 and Table 18.2. This table summarizes the distribution of composites and assays by zone. A total of 656 mineralized intersections was used for the Global Mineral Resource including 568 mineralized intersections for the B2, Bn, and P zones and 3 411 -1 meter composites.

Table 18.1 Drill-hole and Composite Intersections by Zone Global Resource Evaluation

Zone	# DDH Used	# Chips Used	# Assays	Statistic Mean Grade Au (uncut)	Statistic Mean Grade Au (cut 30 g/t)	Average Length (m)	# Composites (1 m) Used
A1-A2	48	-	*	*	6.76	*	*
A3	15	-	*	*	8.67	*	*
B1	25	-	*	*	7.20	*	*
B2	224	74	2306	4.69	4.54	0.98	2444
Bn	189	28	916	5.94	5.16	0.92	967
P	45	8	272	6.20	4.49	0.92	-
Total	**546**	**110**	**3494**				**3411**

* Not available, because not estimated with Gemcom.

Table 18.2 Drill Hole Distribution by Zone for 2005 Reserve and Resource Estimate

ZONE	UNDERGROUND DRILLING	SURFACE DRILLING	TOTAL
B2	186	38	224
Bn	154	35	189
B1 –B1'	24	1	25
A1-A2-A2'	35	13	48
A3	6	9	15
P	35	10	45
Total	**440**	**106**	**546**

Most sample values are in the range of 0.5 to 4.5 g/t Au with a typical high-grade tail from 25 up to about 50 g/t Au, suggesting average grades for the geological resources ranging between 4 and 6 g/t Au statistically, depending on the cut-off grade.

Following are some additional details about the data statistics:

- To obtain the 568 mineralized intercepts used in the resource model for the B2, Bn, and P zones, selected samples had to be grouped from a total of 3,494 drill core and chip samples, of which only 1,214 samples had significant amounts of gold (>4.5 g/t Au), *i.e.* 35% of the total population.
- Of those 3,494 samples, 90% had grades between 0.1 and 10 g/t Au. Among the higher-grade samples, 3 were over 200 g/t, 3 in the 100 to 200 g/t Au range, 2 in the 75 to 100 g/t Au range, 6 in the 50 to 75 g/t Au range, and 41 between 30 and 50 g/t Au. In other words, only a handful out of thousands of samples, could show a significant nugget effect to worry about.
- There are only 55 samples with more than 30 g/t Au, or 1.6% of samples higher than 30 g/t Au.

18.1.3 Estimation Parameters

The main parameters used to estimate the Global Mineral Resources are as follows:

A) Cut-off grade

A cut-off grade of 3 g/t Au was used. Cut-off grades are determined by the mining department based on criteria such as the mining method and the operating costs. This topic is covered in more details in section 24 and section 18.2.1.

B) Grade capping

Based on the results of statistical analyses, the grade capping value for all samples (drill core and chips) was set at 30 g/t Au for all zones. Individual assays were capped and not the composite. Statistical studies, supported by a Geostat study (2005) were carried out to determine the capping grades for assays.

C) Minimum width

Each potentially economic intercept was calculated using a minimum true thickness of 3.0 metres. All diamond drill-hole intercepts were calculated at that minimum, using the grade of the adjacent material when assayed or a value of zero when not assayed. This value is a realistic estimate for the proposed mining method (long hole). This topic is covered in more detail in section 24.1.

D) Density

Density data comes from tests conducted for a study on rock mechanics (Golder and Associates, 1997). For the B Zone, the density was evaluated as follows:

	Percentage	Density
Mafic Dyke:	50%	2.80
Quartz Vein:	30%	2.75
Altered Granodiorite:	20%	2.75
	Average	2.78

An average rock density of 2.8 t/m^3 is defined. This value is a realistic estimate of the average composition of the ore from a theoretical point of view.

In 2004-2005, Richmont Mines evaluated the density of three different zones (Table 18.3). The results indicate that a fixed density of 2.8 t/m^3 per tonne should be used. The results did not show any major differences between the zones and were very similar to the test done by Golder in 1997. A density of 2.8 t/m^3 was therefore used to calculate the tonnage estimate.

Table 18.3 Density Estimation by Richmont Mines

ZONE	NUMBER OF SAMPLES	DENSITY MEASUREMENTS (T/M^3 PER METRIC TONNE)	AVERAGE (T/M^3)
B2	6	2.79 - 2.71 - 2.88 - 2.91 - 2.88 - 2.69	2.81
A	4	2.83 - 2.94 - 2.78 - 3.01	2.89
P	7	2.70 - 2.74 - 2.80 - 2.85 - 2.81 - 2.73 - 2.72	2.76
TOTAL	**17**		**2.81**

18.1.4 Longitudinal Polygonal Method

The polygonal method on longitudinal section was used at East Amphi for the first resource estimation published by Richmont Mines on December 31, 2004 and for the global resource estimate for zones A1-A2-A2’, A3, and B1-B1’ on December 31, 2005. It is a simple and quick method that is adequate for narrow and planar zones where gold values are relatively uniform and well distributed. The disadvantage of the polygonal method is related to its tendency to bias results (because the method underestimates the grade in low-grade areas and, inversely, overestimates the grade in high-grade areas; see Glacken, 1999).

In a polygonal estimation, the influence of each composite drill-hole value is fixed at mid-distance to surrounding intercepts, and samples are given equal weights within a volume. The polygonal method on longitudinal section consists firstly of measuring the horizontal north-south thickness of each drill intercept for a zone traced directly from an interpreted level plan or a north-south cross-section. The mid-point coordinate of each zone intersection is then projected onto an east-west vertical longitudinal section that is unique for each zone.

A polygonal shaped perimeter surrounding each intercept point which links the mid-distance marks to surrounding intercepts is then drawn on the longitudinal section. The surface area of each polygon is measured off from the longitudinal section. The tonnage is the product of the polygon surface area multiplied by the intercept horizontal thickness and the specific gravity determined for each zone.

The polygons are then reproduced on long sections for better control. The area of each polygon is calculated using AutoCAD. The surface, horizontal width, tonnage and average grade are also noted. The data for all polygons are finally transferred in Excel spreadsheets in order to carry out mineral resource calculations per block, zone, level and sublevel, category and stope.

18.1.5 Block Model

A block model was used at East Amphi for the 2002 SNC-Lavalin estimate. In the Global Mineral Resource evaluation, Richmont Mines also used a block model to estimate the B2–Bn and P zones. This method is appropriate where the zones are larger, irregularly shaped and thicker and the gold is distributed irregularly in three dimensions. The method is also quick, relatively simple, statistically based and, to a certain extent, accounts for grade variability according to distance and direction (based on variograms).

The method consists firstly of building a three-dimensional envelope (wireframe model) of a particular orebody using confident geological information and technical parameters. The orebody envelope is then filled with several unit-sized blocks to which a grade is then interpolated. The grade of each individual block within the orebody envelope is assigned using the inverse distance to power (IDP) interpolation method. In practice, surrounding samples (those comprising various drill-hole intercepts for example) are weighed inversely to their linear distance^{power} from a particular estimation point (the unit-sized block).

For example:

Inverse distance squared grade estimate = weigh each sample's grade inversely to distance^2 and divide by sum of inverse distance^2.

In an IDP estimate, samples are weighed inversely to their distance from the estimation point. This method is based on the premise that the grade of a particular unit block is more like samples that are closer to it; closer samples should therefore be given a higher weight. The use of different power levels according to search distance is a function of the information density and of the zone.

At East Amphi, the block model method involves using Gemcom ore reserve modelling software. An envelope of 3 g/t Au was drawn based on diamond drilling, chip sampling and underground mapping results to build the wireframe. Gemcom captures all the sampling data within the envelope (wireframe). Intersections are then composited so that the grade values for each drill-hole and chip sample can be represented over standard unit lengths of 1 metre. The interpolation method retained for the estimation was the inverse distance to power 1 method (IDP 1). The method was chosen based on the recommendations of a Geostat study (2005), taking into account variogram results and ore continuity.

A dilution envelope was calculated outside of the 3 g/t Au shell. This envelope was used to define the dilution grade.

The discussion below of the various parameters used for the block model estimation is based on an analysis by the technical staff of Richmont Mines and discussions with Robert de l'Étoile from Geostat (2005).

A) Sample selection

At East Amphi, the selection of drill core samples by zone within the wireframe of 3 g/t Au for use in the estimation is based on the Access database table of composites (Gemcom), compiled by the geology department.

B) Length of composite samples

The block modelling method requires that all samples have an equal length and that the gold grade is uniformly distributed throughout the sample interval. A statistical analysis of samples from the B2-Bn zones shows that 38% of samples had a length of 1.0 metre and that the average length was 0.96 metre. The final compositing of gold assay results was done on 1-metre equal lengths and constrained within the interpreted wireframe for each zone.

C) Variography to establish search parameters

Directional variograms and correlograms were used to establish the relationship between composite samples. The same relationships are to be applied to the blocks in the estimation.

In order to look for directions of variability, the variography was measured and compared along different directions until the best results for range were found. Orthogonal intermediate axes were then established. The whole idea behind the concept is that two nearby samples have values more similar than two samples that are far apart, and the distance in which samples could be considered similar is variant with direction. It is called directional variation (anisotropy).

Samples in the B2-Bn zones show a correlation within 12 to 15 metres horizontally and vertically, and within 3 metres perpendicular in a N-S direction (from Geostat, 2005). To complete adequately the evaluation, an ellipsoid search of 30 metres was applied but we imposed restrictions on the minimum and maximum number of composites and an obligation to look for a minimum of 2 drill-holes. With these restriction specifications and as a function of the spacing of drill-hole sections (10 metres by 15 metres vertical), the size of the ellipsoid search is minimized. A 30-metre ellipsoid search was fixed by the distance for inferred resources.

D) Block dimension

The size of the blocks (or cells) that will fill the wireframe model is a function of the sample grid size and the orebody geometry. The rule of thumb is that the blocks should never be smaller than 1/3 of the sample grid spacing. Because the delineation drill intercept spacing at East Amphi is roughly 10 metres by 15 metres, the block size for the B2 zone is 1.25 metres by 1.0 metre by 1.5 metres. Also, the size of the blocks should fit within the stope size.

E) Interpolation method

The choice of power used in the IDP interpolation method (either $distance^1$, $distance^2$ or $distance^3$) was set at power 1, as recommended by Geostat study (2005), to minimize the nugget effect of gold assays.

F) Number of composites

When using this linear estimation method, clustering of samples in certain areas can cause possible grade biases during the grade interpolation process. In order to reduce the effect of sample clustering, an octant search method was applied. This method limits the number of samples that will be considered within a given octant and thereby forces the estimation calculation to use a more logically spread group of nearby samples.

At East Amphi, a maximum of 7 composite samples per diamond drill-hole within a maximum of 15 composites was fixed.

18.1.6 <u>Classification of Resources</u>

The mineral resource estimate was carried out according to the rules and procedures of the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") and adopted by the CIM Council.

The following definitions have been applied for the classification of the mineral resource at East Amphi. Mineral resources are defined as mineral-bearing material whose grade and quantity present reasonable prospects for economic extraction.

A) Measured mineral resources

Part of mineral resources for which grade, geometry and continuity are established with a high degree of confidence from drilling or mining works. Geological continuity and grades are confirmed. At East Amphi, measured mineral resources were confirmed by a high density of drill-holes, some cross-cuts through the mineralized zones and a bulk sampling program. In the polygonal method, blocks are considered measured resources if drill spacing is less than 10 metres and several drill-holes form a cluster of similar results, especially if there is some underground development in the mineralization to confirm its location and grade. In the block model, areas within 10 metres of a drill-hole are considered as measured resources.

B) Indicated mineral resources

Part of mineral resources for which geological continuity, geometry and grade are sufficiently known to allow for a reliable estimate of tonnage and grade. At East Amphi, blocks are considered indicated resources if drill spacing is between 10 to 20 metres and several drill-holes form a cluster of similar results. In the polygonal method, the maximum extension of each block is 20 metres laterally and 40 metres vertically. For the block model, indicated resources lie within a 20-metre radius of a drill-hole.

C) Inferred mineral resources

Part of mineral resources based on limited information but for which geological evidence and comprehension can reasonably presume the existence of an orebody, without being able to check the geological continuity and the grades. At East Amphi, blocks are considered inferred resources if drilling is sparser but again drill intercepts must be grouped in a cluster of similar (geological) results within a reasonable distance of no more than 30 metres at most depending on geology. For the block model, inferred mineral resources are defined within a maximum distance of 30 metres from a drill-hole. In the polygonal method, the area of influence is a maximum of 30 metres laterally and 60 metres vertically.

18.1.7 <u>Results of the Global Mineral Resource Estimate</u>

The Global Mineral Resources estimate at East Amphi were calculated as of December 31, 2005 and are presented in Table 18.4. Resources are presented by zone, and for the B2-Bn zones, by area.

Table 18.4 Summary of Global Mineral Resource Estimate, December 31, 2005 (Cut-off 3.0 g/t Au and ≥ 3.0 metres true width)

DESCRIPTION	TONNES	GRADE G/T	OUNCES
Measured			
B1-B1' Zone	94,732	6.80	20,719
B2 Zone Mine	432,289	4.25	59,068
Bn Zone Mine	134,003	4.14	17,836
A1-A2-A2' Zone	36,965	7.26	8,628
Total Measured*	**697,989**	**4.73**	**106,252**
Indicated			
B1-B1' Zone	6,604	8.68	1,843
B2 Zone Mine	15,739	3.94	1,994
Bn Zone Mine	8,481	4.0	1,091
B2 Zone (4,800 – 4,725 elevation)	66,716	4.5	9,652
Bn Zone (4,800 – 4,725 elevation)	130,664	5.86	24,618
B2 Zone (Explo)	132,856	4.53	19,350
Bn Zone (Explo)	155,620	5.56	27,818
A1-A2-A2' Zone	173,117	7.16	39,851
A3 Zone	48,266	8.91	13,826
Total Indicated	**738,063**	**5.9**	**140,043**
Total Indicated and Measured	**1,436,052**	**5.33**	**246,295**
Inferred			
B1-B1' Zone	25,897	8.30	6,908
B2 Zone Mine	4,243	3.86	527
Bn Zone Mine	811	4.23	110
B2 Zone (4,800 – 4,725 elevation)	2,384	4.43	340
Bn Zone (4,800 – 4,725 elevation)	1,664	4.06	217
B2 Zone (Explo)	40,386	6.02	7,817
Bn Zone (Explo)	23,398	5.54	4,168
A1-A2-A2' Zone	114,494	6.01	22,109
A3 Zone	55,718	8.47	15,173
P Zone	63,716	3.79	7,766
Total Inferred	**332,711**	**6.09**	**65,134**

* Include 27,109 tonnes extracted during the exploration work program

The Global Mineral Resources include the tonnage coming from surface pillars. Dilution and mining recovery factors are not included in the above resource estimate. As can be seen in the table above, resources in the measured and indicated categories as of December 31, 2005 amount to a total of 1,436,052 tonnes at a grade of 5.33 g/t Au for a total of 246,295 ounces of gold, and inferred resources stand at 332,711 tonnes at 6.09 g/t Au for 65,134 ounces of gold.

18.2 Estimate of Mineral Reserves

The calculation of mineral reserves is made to estimate the volume and grade of ore which can be mined and processed at a potential profit. The conversion of mineral resources into mineral reserves is based on economic studies comparable to a feasibility study in terms of accuracy and detail, carried out by the mining engineers of Richmont Mines (section 24). As stipulated in National Instrument 43-101, only mineral resources classified in the measured and indicated categories by Richmont Mines geologists were used in the economic calculations to estimate reserves as of December 31, 2005.

The calculation of ore reserves occasionally includes lower-grade ore blocks or parts of ore blocks within an integrated mining plan. A mining plan was prepared, the details of which are explained in the section 24 and shown on long section in Appendix 6.Ore dilution and recovery rates for each mining method and budgetary costs used in the present study are calculated according to Richmont Mines best estimates as of December 31, 2005. These factors and parameters will be revised each year in order to take into consideration the realities encountered in the mining operation and will be updated according to gained experience and changes in the economic situation.

18.2.1 Estimation Parameters

The parameters and the basis of calculation used in the economic study are described in the following sections.

A) Mining methods

The extraction methods will be transversal and longitudinal long hole and drift-and-fill. Table 8.5 presents the mining method for each zone. Their main design criteria and operational parameters are summarized as follows:

- Geometry: maximum panel length of 11 metres by 25 metres high; this size of stope is used to limit dilution, based on preliminary rock mechanics calculations by Golder & Associates (2005) and the recommendations of John Henning (2005);
- Minimum dip of the zone: 65°;
- Ore recovery: 100% for stopes designed with 4-metre pillars between stopes clearly identified during the process of the mineral reserve estimate. The pillars are excluded from the reserves;
- Internal dilution: minimum mining width is 3 metres for the B2-Bn zones and 4 metres for the A1-A2-A2’, A3, and B1-B1’ zones.

Table 18.5 Mining Method by Zone

MINING METHOD	ZONE
Transversal long hole	B2- Bn
Longitudinal long hole	A1-A2, B1, B2-Bn
Drif-and-fill	A1-A2, A3

B) Operating costs

According to budget estimates of operating costs for 2006, the mine will produce 25,000 ounces of gold from 200,000 tonnes at a direct operation cost of CAN$62.15/tonne. Operating costs for B2-Bn zones and satellites zones are shown here:

	B2 - BN $/T BUDGET 2006	SATELLITE ZONES $/T FORECAST
U/G Operation	21,36	26,03
Maintenance and services	15,57	17,13
Technical services and Administration	5,58	7,25
Transportation and milling	19,64	19,64
Total	62,15	70,05

C) Cut-off grade

A 3 g/t Au cut-off grade was calculated based on the long-hole mining method (transversal and longitudinal) and estimation of operating costs.

The main criteria used in the economic evaluation are as follows:

- No profit margin is included in the calculation;
- The deferred or capitalized costs are not taken into account;
- Only the gold price is taken into account in the economic evaluation and the gold price used is US$450/oz (or CAN$540/oz with an exchange of CAN$1.2 for US$1). However, the final cut-off grade will be based on the current gold price at the time of extraction of the reserves;
- Costs are based on estimation of production costs and include the following major elements:
  - Development cost;
  - Variable costs including contract transportation and milling;
  - Cost of mining;
- $62,15/tonne (operation cost) - $11,00/tonne (fixed cost) =$51,15/tonne or 3 g/t.

D) Dilution rate

The grade of dilution was calculated in the immediate envelope in the hanging wall of the reserve block for the B2-Bn zones. Following the test mining of four stopes, a dilution rate of 10% was deemed appropriate. The dilution grade is based on a statistical analysis of all material included in the 10% in the hanging wall. The grade was 2.7 g/t Au for the B2-Bn zones above the 200 level (4,800 elevation) and 1.9 g/t Au for the B2-Bn zones below the 200 level. For the A1-A2-A2’, A3, and B1-B1’ zones, based on geological information, a grade of 0.5 g/t Au and a dilution of 20% was applied.

E) Mill Recovery

The mill recovery estimated was 97.5%.

18.2.2 Classification of Reserves

The classification of mineral reserves for East Amphi was carried out according to definitions adopted by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM).

Proven mineral reserves

The geologists and mining engineers at East Amphi estimated the tonnage and grade of the mineral reserves in the proven category based on the conclusions of an economic study to determine the part of the measured mineral resources that are economically mineable.

Probable mineral reserves

The estimate of the mineral reserves in the probable category was made based on an economic study of measured and indicated mineral resources in order to determine the part which can be mined economically as probable reserves.

18.2.3 Results of the Mineral Reserve Estimate

The economic study has determined proven and probable reserves amounting to a total of 641,000 tonnes at an average grade of 4.88 g/t Au for 10,500 ounces of gold as of December 31, 2005 (Table 18.6). The estimate of the reserves - tonnage and grade - is inclusive of dilution as well as mining recovery factors.

Table 18.6 Summary of Reserve Estimate, December 31, 2005
(Cut-off 3.0 g/t Au and ≥ 3.0 metres width)

DESCRIPTION	TONNES* (METRIC)	DILUTED GRADE* (G/T AU)	RECOVERABLE OUNCES**
Proven Reserves			
B2 – Bn Zone (above 4,800 elevation)	288,891	4.04	37,500
Total Proven Reserves	**288,891**	**4.04**	**37,500**
Probable Reserves			
B2 – Bn Zone (4,725 – 4,800 elevation)	136,294	5.19	22,740
A1-A2-A2' and B1-B1' Zones	176,160	5.21	29,490
A3 Zone	39,480	8.49	10,780
Total Probable Reserves	**351,934**	**5.57**	**63,010**
Total Proven and Probable Reserves	**640,825**	**4.88**	**100,510**

* Including dilution and 100% mining recovery except the pillars.
** Before milling recovery (97.5%).

18.3 Estimate of Mineral Resources

The mineral resources at East Amphi, calculated after transferring the reserves from the Global Mineral Resource as of December 31, 2005, are presented in Table 18.7.

Table 18.7 Summary of Resource Estimate, December 31, 2005
(Cut-off 3.0 g/t Au and ≥ 3.0 metres width)

DESCRIPTION	TONNES (METRIC)	GRADE (G/T AU)	OUNCES
Measured			
B2 Zone Mine	290,132	4.02	37,519
Bn Zone Mine			
B1-B1' Zone	33,502	8.73	9,406
A1-A2-A2' Zone	8,115	7.92	2,066
Total Measured	**331,749**	**4.59**	**48,991**
Indicated			
B2 Zone Mine	15,739	3.94	1,994
Bn Zone Mine	8,481	4.00	1,091
B2 Zone (4,800 – 4,725 elevation)	14,161	4.01	1,827
Bn Zone (4,800 – 4,725 elevation)	38,999	5.94	7,444
B2 Zone (Explo)	132,856	4.53	19,350
Bn Zone (Explo)	155,620	5.56	27,818
B1-B1' Zone	6,891	9.06	2,007
A1-A2-A2' Zone	95,703	6.88	21,165
A3 Zone	19,950	6.44	4,131
Total Indicated	**488,400**	**5.53**	**86,826**
Total Indicated and Measured	**820,149**	**5.15**	**135,817**
Inferred			
B1-B1' Zone	24,980	8.47	6,801
B2 Zone Mine	4,243	3.86	527
Bn Zone Mine	811	4.23	110
B2 Zone (4,800 – 4,725 elevation)	2,384	4.43	340
Bn Zone (4,800 – 4,725 elevation)	1,664	4.06	217
B2 Zone (Explo)	40,386	6.02	7,817
Bn Zone (Explo)	23,398	5.54	4,168
A1-A2-A2' Zone	101,167	6.01	19,540
A3 Zone	45,702	7.98	11,725
P Zone	63,716	3.79	7,766
Total Inferred	**308,451**	**5.95**	**59,010**

These mineral resources are defined as mineral-bearing material whose grade and quantity present reasonable prospects for economic extraction. At the moment, these resources do not have enough information not the confidence to be transferred in the reserve category.

Additional drilling and development is required to increase the confidence of these resource estimates. These resources are established with the same parameters described in section 18.1.

A crown pillar of 30 metres was excluded for all zones. For the B1-B1' and A1-A2 zones, the resources include the pillar below the open pit; a total of 37,367 tonnes at 7.30 g/t Au. The resources in the B2-Bn zones exclude all 4-metre pillars and sill which represent 135,600 tonnes at 4.36 g/t Au.

19.0 OTHER RELEVANT DATA AND INFORMATION

19.1 Historical Production - Open Pit Reconciliation

Following the evaluation of resources and the pre-feasibility study performed in 1998, McWatters Mines launched a mining development program on the East Amphi deposit. The surface portion of the B-1 zone of the East Amphi deposit has been mined by the open-pit method during the course of 1999. Ore was excavated from an open pit from December 1998 to August 1999, and a total of 120,427 tonnes of ore at 5.66 g/t Au was extracted, yielding 21,257 ounces of gold. The ore was transported to the Sigma mill for treatment. Operations were disrupted by halt in extraction lasting approximately one month, following the subsidence of a volume of 35,000 m^3 of overburden, the northwest walls of the pit. The surface pit now extends from the surface at an elevation of 5,000 metres down to the elevation of 4,937 metres (9 level).

Surface ground stability problems were encountered during the course of the work, and a ground slide occurred on the west wall of the pit. The pit walls on the western, northern and eastern sides of the pit were stabilized by means of remodelling the slopes and the placement of rip-rap on the overburden slopes. An array of piezometers and monitoring devices was installed to monitor and measure ground movement. Golder and Associates provided their services in the assessment, monitoring and control of the overburden movement during the pit operation after the ground slide until the completion of the open pit operation.

This operation required an investment totalling $8M. The realized cash cost of operations at the site was US$208/ounce, while the cash cost of this operation were US$235/ounce, or US$36/tonne.

A summary of the operations at this site is presented in Tables 19.1 and 19.2.

Table 19.1 Summary of Operations

	PLANNED	EXTRACTED
Depth of the pit	- 65 m/9 levels	-63 m/9 levels
Excavation of overburden (m^3)	716,000	688,000
Tailings (tonnes)	615,000	597,000
Tailings/ore ratio	5.7: 1	5.2: 1
Rate of production (t/day)	600	1,025

Table 19.2 Summary of Extraction

	BUDGET			PLANNED (GEOSTATS) (B HEART) DILUTED			PRODUCTION		
	Tonnage (tm)	**Grade g/t Au**	**Ounces***	**Tonnage (tm)**	**Grade g/t Au**	**Ounces***	**Tonnage (tm)**	**Grade G/t Au**	**Ounces***
Zone B-1	124,000	5.73	22,158	99,900	6.7	20,874	120,427	5.66	21,257
Zone A-2	7,000	10	2,184						
Total	**131,000**	**6.0**	**24,342**	**99,900**	**6.7**	**20,874**	**120,427**	**5.66**	**21,257**

* 97% recovery (after milling)

20.0 INTERPRETATION AND CONCLUSION

The Qualified Persons at Richmont Mines have conducted an economic study that has resulted in the estimation of East Amphi mineral resources and reserves as of December 31, 2005. This study rests primarily on the geological and structural interpretation of drill-hole results and some cross-cuts. The parameters used in the economic study were essentially based on the best estimate of Richmont Mines based on experience acquired during the project operation in 2005.

All work was carried out according to best practices in the industry and in compliance with the rules and recommendations of National Instrument 43-101 and its companion policies, including those adopted by the CIM. Moreover, the qualifications and the experience of the mine employees who took part in the estimation of the resources and reserves as well as in the determination of parameters give a high level of confidence in the quality of the interpretation and the accuracy of the economic calculations for the East Amphi project.

The economic study determined proven and probable reserves of 640,825 T at an average grade of 4.88 g/t Au for a total of 101,500 ounces of gold after application of dilution and mining recovery factors. An undiluted total of 820,150 T at a grade of 5.15 g/t Au remains in the form of measured and indicated resources, and 308,500 T at a grade of 5.95 g/t Au of inferred resources. These cannot be mined at present on an economic basis or the degree of geological confidence must be raised by the development of drifts or additional drilling. The definition diamond drilling completed in 2005 in the B2-Bn zones has modified the December 31, 2004 Mineral Resource Estimate calculated by Richmont Mines and the 2002 Mineral Resource Estimate by SNC Lavalin. The 2004-2005 exploration program set up by Richmont Mines made it possible to increase the mineral reserve estimate. A good potential exists at depth in the B2-Bn zones, where inferred resources are already identified but cannot be taken into account in the reserve evaluation.

Based on an annual rate of production estimated at 200,000 T for 2006 and an average of 125,000 T thereafter, the reserves available at December 31, 2005 should allow the continuity of mining production for a period of 4.5 years.

Low dilution level from production and ground stability of future developments to access satellite zones are key elements in the success and profitability of the overall project. On the other hand, if ground conditions or dilution level do not adversely affect the economics, the quality of the resources should make it possible to reclassify a good part of the latter into mineable reserves following work to be performed in the next few years, ensuring an even longer operation.

After a non-cash write-down of $26,040,953, the financial forecasts estimate an operating profit of CAN$3M before taxes on revenue of CAN$53M. The projected total production amounts to 100,000 ounces of gold at a direct cash cost of operation of CAN$520/oz.

The project shows an internal rate of return after taxes of 4.82% for a total investment of $50M, including a working capital of $11.9M and an average operating cost of $66 per tonne of ore milled. The project is, however, very sensitive to variations in capital investments, operating cost and the gold price.

Project profitability was affected by a number of negative elements. The 2004-2005 drilling program failed to extend previously defined ore zones and to consolidate reserves established by SNC-Lavalin in 2002. Furthermore, definition drilling also demonstrated significant grade variability related to an important nugget effect.

Cross-cut development revealed the geological complexity of the B2-Bn zones and the discontinuous nature of the latter. The zone is characterized by the presence of several generations of dykes of distinct compositions and variable content in coarse-grained gold-bearing pyrite, such that the mineralization is discontinuous and boudinaged. To consider the entire architecture of mineralized zones, an ore shell of 3 g/t Au was defined, which resulted in a lower ore grade.

Following the conclusion of geotechnical studies in late 2005, modifications were brought to the mining plan, with the result that the width of stope openings was reduced, permanent pillars were added, and the use of cement back-fill was abandoned in most stopes. These studies established that the hanging wall in the B2-Bn zones and the country rock in satellite zones were of poor quality. In addition to these important modifications to the original design of the mining plan, the tonnage was also significantly reduced as a result.

During the mining test, 40,581 tonnes of development ore including 15,664 T from four stopes, were processed. The ore grade was 4.25 g/t Au. The head grade for the entire 2005 mill test, including development ore from lots 1 and 2, was 3.70 g/t Au, whereas for the bulk sample from lot 3, the head grade averaged 3.98 g/t Au. Gold recovery of the zinc precipitation circuit at the Camflo mill was established at 97.5%. The conclusions of the reconciliation between planning, mining and milling are:

- The muck grade was lower than the actual head grade by 10.7% for the entire 2005 mill test, and by 10.5% for lot 3 only.
- The mill feed head grade was overestimated by 2.2% for the entire 2005 mill test and overestimated by 3.1% for the third lot only.
- The estimated grade of CMS-surveyed stopes (block model from drill core and chips samples) was lower than the actual head grade by 6.3% for the entire 2005 mill test, and by 4.7% for the third lot only.
- From an operational viewpoint, overall dilution was only 10%, even taking into account one stope where the dilution reached 25%, due to the excessive deviation of a drill-hole.
- Average grade milled from the four stopes returned a content of 4.14 g/t compared to an undiluted grade planned of 3,85 g/t and an average grade of 3,80 g/t according to outlined tonnage effectively withdraw in the block model evaluation. This represents an upgrade of over 9% based on CMS measurement evaluation.

- Block modelling using inverse distance to power 1 and listed parameters appears to be a suitable method for mineral reserve and resource estimations at East Amphi, as the mill test results confirm, showing a variation of less than 9% relative to results obtained for the ore during the mining test.

As a result of the modifications to the mining plan and the general cost increase, production costs and capital costs were substantially higher than initial forecasts. All of these elements affected the project profitability.

21.0 RECOMMENDATIONS

Given the marginal profitability of the project, a write-down of $26,040,953 was recorded at the end of 2005. The project is particularly sensitive to the dilution rate and consequently, to the higher production and development costs the latter could generate. In light of the available information contained in this Technical Report on exploration work and operation of the East Amphi project, particular conditions (gold price, grade, dilution, etc.) could adversely affect the continuation or the profitability of the East Amphi operation in the years to come. An action plan has been proposed and the following measures are being put forward to improve (or maintain) profitability:

- Modify stope size according to confidence level to recover additional resources relative to dilution;
- Reduce costs and maintain an effective cost control on the operation;
- Maintain a good dilution control program;
- Adjust the cut-off grade according to gold price and operating costs on a quarterly basis;
- Complete the geological interpretation at depth on the B2-Bn and P zones and adequately evaluate their potential.

The following general recommendations can be made:

- Complete a resource and reserve estimate of the satellite zones by block modeling using Gemcom;
- Footwall and hanging wall dilution of reserve blocks from satellite zones should be modelled by wireframe and evaluated by block modelling techniques. This would complete all the lenses in the proven and probable resource and indicated resource categories;
- Continue to review the current block model parameters periodically using grade and tonnage reconciliation data;
- Pursue the deep exploration program using standard sectional drilling from the ramp;
- Modify the assay technique to improve grade determination accuracy by pulverizing 1,000 g of material and introduce standard re-assaying by gravimetric finish for all assays over 3.5 g/t Au initially determined by atomic absorption.

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23.0 CERTIFICATE OF AUTHOR

I, Jules Riopel, from Rouyn-Noranda (Québec), Canada, do hereby certify that:

1) I am a Professional Geologist.
2) I received a bachelor's degree in geology from the Université de Montréal (Montréal, Québec) in 1987, a master's degree from the same university in 1992 and an MBA from the Université du Québec à Montréal in 2004.
3) I am a registered member of the Ordre des Géologues du Québec (OGQ, licence number 490).
4) I have over 17 years' experience as a geologist in the mining industry. My experience has been acquired with Cambior and Noranda. I have been working at Richmont Mines since June 2003 as the Principal Geologist - Exploration and since 2006 as Geology and Exploration Manager.
5) I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
6) I am responsible for the preparation of all sections of this technical report entitled "Technical Report on the Mineral Resource and Reserves Estimation of the East Amphi" and dated March 8, 2006, relating to the East Amphi property. As a geologist, I conducted numerous underground visits to the East Amphi project between April 2004 and December 31, 2005. The 2005 Mineral and Reserve Estimate is based on my direct supervision. The feasibility study, including cut of grade and operation cost, was prepared by the engineering staff.
7) I have never had any prior involvement with the property that is the subject of the Technical Report.
8) 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.
9) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
10) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.
11) I hold the option to purchase a certain number of shares of Richmont Mines.

Dated this 8th of March, 2006.

(S) Jules Riopel, M.Sc., P.Geo., MBA

I, Chrisitan Pichette, from Montreal (Québec), Canada, do hereby certify that:

1) I am a Professional Mining Engineer.

2) I received a bachelor's degree in mining engineering from the Université Polytechnique (Montréal, Québec) in 1977 and a master's degree in rock mechanics from the same university in 1978.

3) I am a registered member of the Ordre des Ingénieurs du Québec (OIQ, licence number 31632).

4) I have over 26 years experience in the mining industry. My mining expertise has been acquired with Noranda, Lab Chrysotile, Placer Dome, Barrick and Cambior. I have been working for Richmont since September 2005 as Vice-President, Operations.

5) I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

6) As vice-president, Operations, I approved the 2006 East Amphi budget prepared by the staff of Richmont. I conducted some underground visits to the East Amphi project between September 2005 and December 31, 2005. During those visits, I mainly spent time on evaluating rock quality, ground conditions and the overall aspects of the operation. The East Amphi mine is under my overall responsibility. The feasibility study, including cut of grade and operation costs, was prepared by the engineering staff and reviewed by myself.

7) 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.

8) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

9) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

10) I hold the option to purchase a certain number of shares of Richmont Mines.

Dated March, 8th, 2006.

(S) Christian Pichette Eng.

I, Alain Mercier, from Rouyn-Noranda (Québec), Canada, do hereby certify that:

1) I am a Professional Mining Engineer.
2) I received a bachelor's degree in mining engineering from Université Laval (Québec City, Québec) in 1974.
3) I am a registered member of the "Ordre des Ingénieurs du Québec" (OIQ, licence number 26583).
4) I have over 28 years' experience in the mining industry. My mining expertise has been acquired with Noranda, Northgate-Patino and Barrick. I have been working for Richmont since April 2003 as Director of Technical Services and Environment.
5) I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
6) As Director of Technical Services, I designed the actual underground mining project "surface to 200 meters depth." I conducted numerous underground visits to the East Amphi project between April 2004 and December 31, 2005. During those visits, I mainly spent time on the design aspect, ground support, rock quality evaluation, the criteria for the B2 and Bnorth ore zones and the overall aspects of the operation. With the contributions of several Richmont employees, I prepared the 2006 East Amphi production budget in the B2 and Bnorth ore zones. In this report, I directly took part in establishing the general mining design of those satellite zones, the scheduling of development, the choice of mining method and the production table. I also wrote the environmental section.
7) I have never had prior involvement with the property that is the subject of the Technical Report.
8) 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.
9) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
10) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 28th of February 2006

Alain Mercier Eng.

I, Christian Bézy, from Dubuisson (Québec), Canada, do hereby certify that:

1) I am a Professional Geologist.

2) I received a bachelor's degree in geology from the Université du Québec (Montréal, Québec) in 1978.

3) I am a registered member of the Ordre des Géologues du Québec (OGQ, licence number 117).

4) I have over 27 years' experience as a geologist in the mining industry. My mining expertise has been acquired with Quebec Cartier Mining from 1980 to 1984, Placer Dome (1988–1998), Mc watters (1998–2003). I have been working at Richmont since June 2005 as a Mining Geologist.

5) I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

6) I am involved in the preparation of the resource and reserve section of this technical report, entitled "Technical Report on the Mineral Resource and Reserves Estimation of the East Amphi" and dated March 8, 2006, relating to the East Amphi property. As geologist, I have been at East Amphi as senior production geologist since May 2004. I have been involved particularly in the underground drilling campaigns, underground mapping and the resources–reserves calculation for the B2-Bn zones at East Amphi.

7) I have never had any prior involvement with the property that is the subject of the Technical Report.

8) 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.

9) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

10) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

11) I hold the option to purchase a certain number of shares of Richmont Mines.

Dated this 8th of March, 2006.

(S) Christian Bézy, P.Geo.

I, Jacques Daigneault, from Val-d'Or (Québec), Canada, do hereby certify that:

1) I am a Professional Geologist.

2) I received a bachelor's degree in geology from the Université du Québec (Montréal, Québec) in 1973 and a D.E.C. (geology) degree from the Ecole Polytechnique de Montréal (Montréal, Québec) in 1976.

3) I am a registered member of the Ordre des Géologues du Québec (OGQ, licence number 641).

4) I have over 30 years' experience as a geologist in the mining industry. My exploration expertise has been acquired, since my graduation, first with Iron Ore Co. Ltd. My experience in gold mining has been acquired since 1980 with Camflo Mines Ltd., Barrick Gold, where I became chief geologist from 1989 to 1993 and subsequently chief geologist with Richmont Mines Ltd (1993–2003). Now I serve Richmont Mines as senior exploration project geologist.

5) I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6) I am involved in the preparation of most of sections of this technical report entitled "Technical Report on the Mineral Resource and Reserves Estimation of the East Amphi" and dated March 8, 2006, relating to the East Amphi property. As a geologist, I have been at East Amphi as senior exploration geologist since the beginning of the work done by Richmont Mines. I have been involved particularly in the surface and underground drilling campaigns, underground mapping and the resources–reserves calculation for 5 of the 8 reported zones at East Amphi.

7) I have never had any prior involvement with the property that is the subject of the Technical Report.

8) 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 have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

12) I hold the option to purchase a certain number of shares of Richmont Mines.

Dated this 8th of March, 2006.

(S) Jacques Daigneault, P.Geo.

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

The information contained in this section has been prepared by East Amphi engineering staff and Richmont Mines' director of technical services.

24.1 Mining

24.1.1 Infrastructure

Richmont Mines started its advanced underground exploration program in January 2004. All the mining facilities are already built near the deposit. A decline has been excavated at a grade of minus 16% grade from the surface to a vertical depth of 200 metres. Footwall drifts have been developed at levels 100, 125, 150, 175 and 200 levels giving as many accesses for definition drilling of the upper part and further depth exploration drilling up to a vertical depth of 400 metres. The Table 11.1 in Section 11 summarizes the underground infrastructures at the end of the year 2005.

The East Amphi deposit is accessible via a ramp from surface at 5,002 metres elevation, the surface ramp uses an old open pit segment to reach the portal at an elevation of 4,980 metres. The portal was built on the open pit south wall, which was the most appropriated and economical approach:

- The portal and the underground ramp start are located in porphyry rock which provides safe ground conditions on a long-term basis;
- The overburden is at its minimum thickness of about 10 metres;
- From 5,002 to 4,980 metres' elevation, the old open pit ramp segment reduced the underground excavation by as much as 130 metres;
- Considered as stable, the south pit wall (rock and overburden) monitoring does not required.

The portal was stabilized with reinforced concrete anchors and wire mesh on the immediate rock face dipping at 70°. From the rock contact, the overburden has been profiled at a slope of 3 to 1. The portal and the first 15 metres of underground ramp (4.7 m x 4.5 m) were shotcreted. At gradient of -16%, the underground ramp was extended down to 4,785 elevation. Intermediate accesses were created at level 100 (4,900 El), level 125 (4,875 El), level 150 (4,850 El), level 175 (4,825 El) and level 200 (4,800 El). In all, 1,330 metres of ramp were excavated. Along the ramp, safety bays were cut every 30 metres and muck bays were developed every 160 metres. The ramp is used as haulage road, ventilation exhaust and main access for personnel and services. Its size allows the use of a 6.0 v3 scooptram and 30 tonnes low-profile haulage trucks.

All level footwall drifts were excavated at 4.5 metres wide x 4.5 metres in heigh. A total of 1,515 metres were advanced in 2004 and 2005. As seen in figure 24.1, ramp access is on the south side of the orebody zone. Development is shown as done at the end of 2005. Some 900 metres of development to complete preparation of stopes preparation is still to be excavated in 2006.

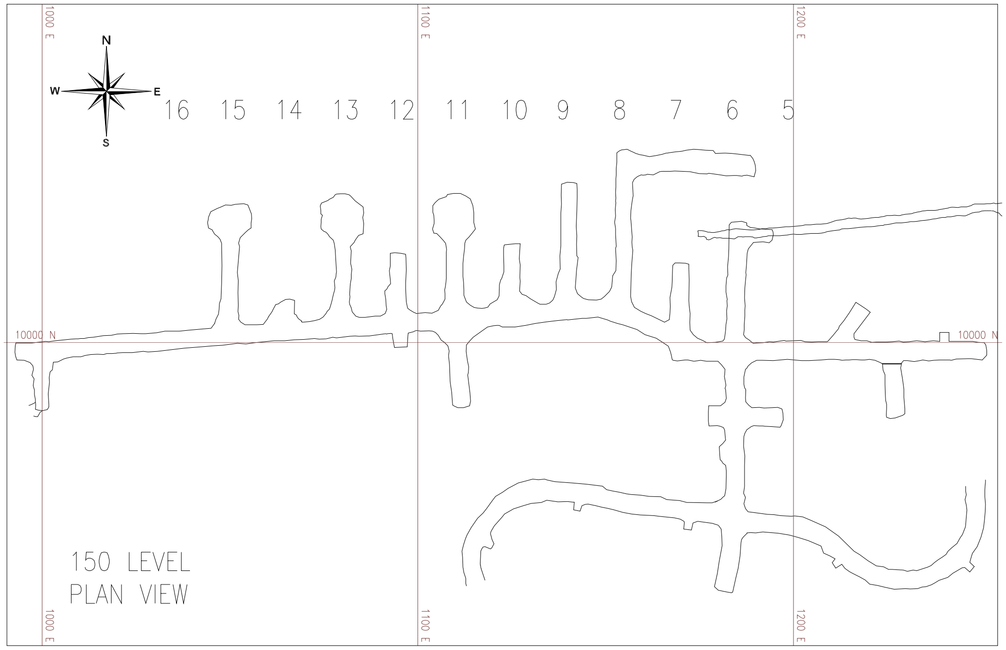


Figure 24.1 – Footwall drift at level 150

Between levels 100 (4,900 El) and 200 (4,800 El) levels, a ventilation raise 3 metres in diameter was developed at a 57° angle. Access to the ventilation raise is shown at the west end of level 150 in figure 24.1. The second mine exit to the surface (an old exploration shaft, level 100 and main vent raise) has been completed by installing a manway in the vent raise.

24.1.2 Underground Mining Approach

The underground mining approach described here makes reference only to the B2 Zone located between 4,800 and 4,915 metres elevation. A crown pillar of a minimum of 30 metres in thickness will be kept in place.

The extraction of the B2 Zone has been designed with a long-hole mining method including transversal (90%) and longitudinal (10%) options. The long-hole stoping method suits well the geometry and the slope (65° to 85°) of the ore body. It is a low cost mining method but the transversal approach requires a relatively large development effort. The thickness of the stopes varies from three metres to a maximum of 17 metres.

According to ground quality, a standard and effective mining method has been chosen to lower the capital investment and ensure an operating cost as low as possible, while ensuring an excellent control on ground conditions and a low amount of dilution. On the basis on the 25-metres span between levels, the presence of rock schists at the hanging wall and studies of rock mechanics (Henning 2005; Golder et Associate 2005), the maximum stope hanging wall exposure is limited at 11 metres along strike. To minimize dilution, each stope is separated by a lateral pillar of 4 metres along strike. The risk of pillar failure is increases importantly with stope thicknesses over seven metres. The ore reserve left in the pillars averages 25%. This layout is valid for the reserves over the 200 meter elevation, in the B2 orebody.

At the end of the fourth quarter 2005, a bulk sample of the first four stopes has been mined and milled. Dilution results are shown below in Table 24.1.

Table 24.1 Grade Comparison and Dilution Results From Mining of Four Stopes

Planning and Dilution Results From the first four Stopes										
Sequence	**Planned tonnage (without dilution) (mt)**	**Grade (without dilution) (g)**	**Planned tonnage (diluted 10%)**	**Grade (diluted) (g)**	**Milled tonnes**	**Milled grade**	**Tonnes CMS (mt)**	**Grade CMS**	**Tonnage left in place (mt)**	**Total tonnes**
Stopes 2005										
CH200-13	3,132	5.58	3,445	5.32	2,754	3.08	2,750	5.52	50	**2,800**
CH200-11	4,472	3.91	4,919	3.80	4,355	3.82	4,709	3.94	125	**4,834**
CH175-13	4,931	3.21	5,424	3.16	4,975	4.50	6,203	3.17	300	**6,503**
CH200-15	4,181	3.66	4,599	3.57	3,578	4.85	3,736	3.72	150	**3,886**
	16,716	**3.95**	**19,223**	**3.79**	**15,662**	**4.14**	**17,398**	**3.87**	**625**	**18,023**

Dilution								
	Hangning-wall H/W			**Footwall F/W**			**H/W + F/W**	
	Tonnage	**Grade**	**%**	**Tonnage**	**Grade**	**%**	**Tonnage**	**%**
CH200-13	52	2.6	1.7	31	3.3	1.0	83.0	2.7
CH200-11	270	2.9	6.0	76	4.76	1.7	346.0	7.7
CH175-13	1,015	2.5	20.6	232.4	3.06	4.7	1,247.4	25.3
CH200-15	32.6	3.55	0.8	3.3	3.9	0.1	35.9	0.9
	1,369.6	**2.61**	**8.2**	**342.7**	**3.47**	**2.1**	**1,712.3**	**10.2**

Dilution from the first four stopes averaged 10.2%. Stope 200-13 returned a 25.3% dilution mainly because the hanging wall was damaged following the deviation from a production hole that did not break through on the level 200. Since then, measures have been taken to minimize hole deviation and ensure that holes break through. Besides that result, dilution from the others three stopes that were part of the bulk sample averaged only 3.9%. From those results, a 10% dilution has been included in the calculation of ore reserves for the B2-Bn zones.

Since December 31, 2005, two others stopes have been put into production. To date mining has shown a good to excellent walls control and a low dilution levels. Average dilution from the first six stopes (including 175-13) was 7.4%. Dilution excluding stope 175-13 is at 4.1%. Typical results from cavity monitoring measurements done in stope 175-11 are shown in Figures 24.2, 24.3 and 24.4.

Figure 24.2 – Longitudinal view- B2 Zone (above level 200)

Figure 24.3 – Cross-section view - outline stope 175-11

Figure 24.4 – Longitudinal view - outline stope 175-11

Production drilling is carried out with two ITH drills, which are also used to drill the slot raises with a V30 hammer. Figures 24.5 and 24.6 show the typical pattern of drilling layout and slot- raise locations. Production drilling and blasting is a critical portion of the cycle to maintain walls stability during the extraction phase, to minimize dilution.

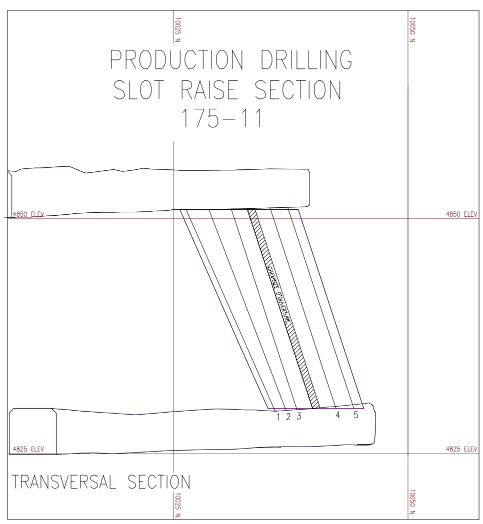


Figure 24.5 – Transversal section - production drilling

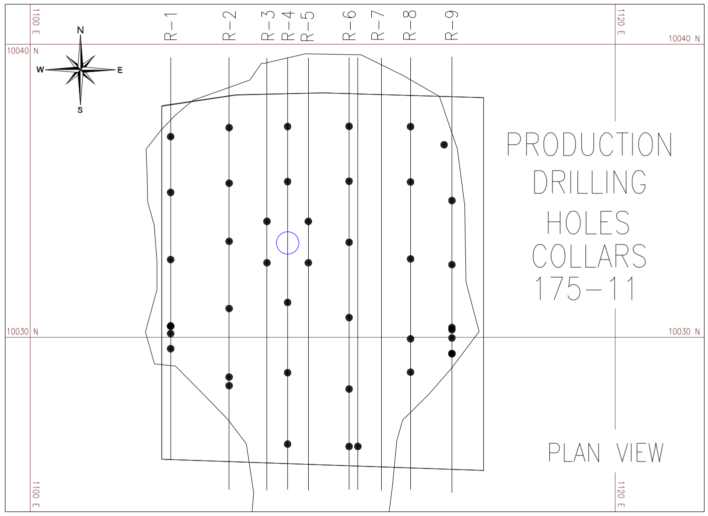


Figure 24.6 – Plan view - production pattern

Three 6 cy scooptrams are used for stope extraction and haulage is done with two low profile trucks of 26 tonnes and one of 30 tonnes. On the surface, oversized ore blocks are reduced to a maximum of 16’’ in size with a mobile rock breaker. All of the ore produced will haule to the Camflo mill over a distance of 13 km by 40 tonnes trucks.

During the exploration project, the development waste was hauled up to the surface and disposed on a waste pad. With stope extraction, development waste is mainly kept underground as stope backfill and the complementary backfill required is taken from the surface waste pad.

It is planned to backfill some stopes with cemented rock fill (CRF) where pillar failure is anticipated. In those cases, 20% sand will be added to the rock development waste. A 4.5% Portland cement slurry will be prepared in a surface backfill plant and added to the waste mixture during the underground stope backfilling process.

The old exploration shaft was refurbished in 2004, and it is presently used as a second man exit to surface, as the main downcast ventilation raise and for shaft water pumping.

The 3D view (figure 24.7) shows an overview of the overall underground infrastructures developed at the end of 2005, including the ramp, the levels and the main vent raise excavations.

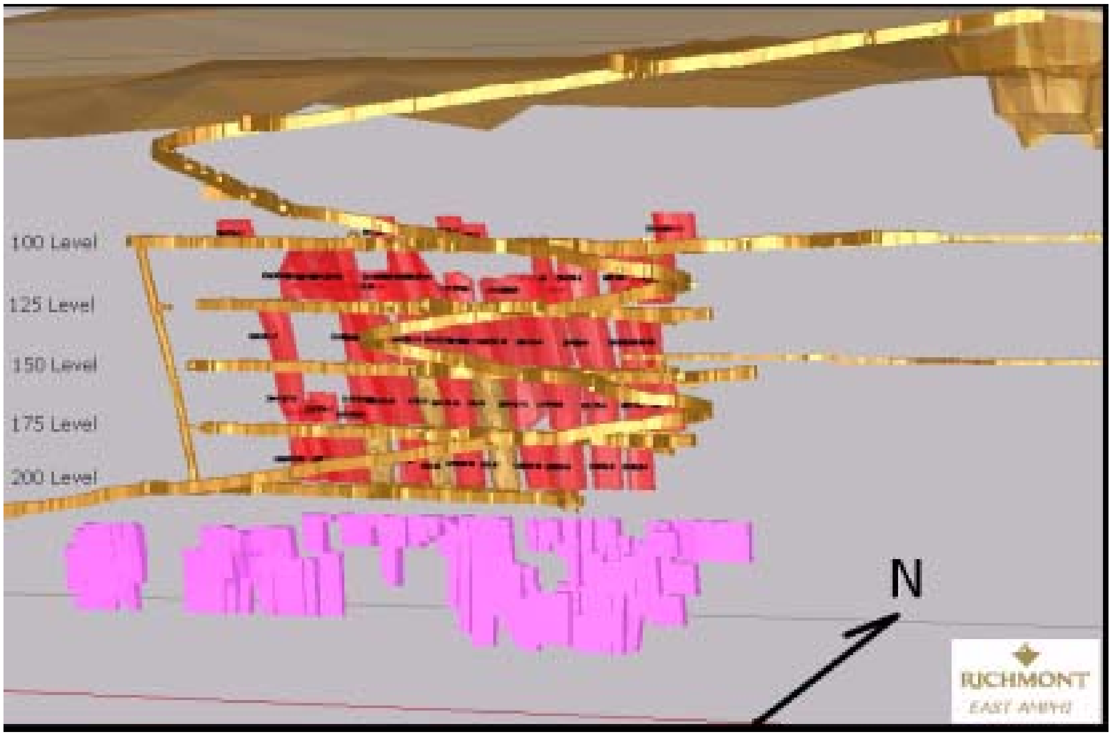


Figure 24.7 – 3D view of general actual u/g infrastructure - B2 Zone

24.1.3 Operating Schedule

The actual production is based on an operating schedule of 8-hours shifts, two shifts per day and five days per week. The daily production rate is set at 800 tonnes of ore plus the additional tonnage from waste development and backfill requirements. Annual production for 2006 is planned at 200,000 tonnes.

Manpower requirements to maintain the East Amphi operation are described below in Table 24.2.

Table 24.2 On Site Manpower Requirements

MANPOWER	SHIFTS/DAY	#/SHIFT	TOTAL	OFF-SHIFT	TOTAL
Mine and maintenance					
Mining					
Mine captain	1	1	1	0	1
Shift bosses	2	1	2	0	2
Development miners	2	2	4	0	4
Rehab, longhole blasting, cablebolt installation + construction	2	2	4	0	4
Jumbo operators	2	1	2	0	2
ITH + cablebolts drillers (contractors)	2	2	4	0	4
Service miners	2	1	2	0	2
Scooptram + truck operators	2	4	8	0	8
Maintenance and Surface					
Maintenance foreman	1	1	1	0	1
Mechanics	2	2	3*	0	3
Electricians	1	1	1	0	1
Surf. loader operator	1	1	1	0	1
Total			33	0	33
Security guards **	3	1	3	2	5
Administration and Technical Services					
Mine superintendent	1	1	1	0	1
Purchasing agent	1	1	1	0	1
Chief engineer					
Mining engineer	1	1	1	0	1
Mining technician	1	1	1	0	1
Geologist	1	1	1	0	1
Geologist technician	1	1	1	0	1
Surveyor	1	2	2	0	2
Wharehouse clerk	1	1	1	0	1
Total			9	0	9
Grand Total East Amphi					47

* only one on night shift
** two12 -hour shifts/day

Table 24.3, below, shows the development and the production previously done and forecast for the B2-Bn zones above elevation 4,800 from the beginning of 2004 through 2007.

Table 24.3 Development and Production Schedule (B2-Bn zones - Above Level 200)

Description	2004 Tonnes (grade)	2005 Tonnes (grade)	2006 Tonnes (grade)	2007 Tonnes (grade)
Waste				
Contractor’s development	130,300	114,900		-
Richmont Mines’ development	0	43,800	13,500	-
Total	**130,300**	**158,700**	**13,500**	**-**
Ore				
Development	0	24,917 (3.35 g/t)	50,300 (3.39 g/t)	-
Stope (diluted)	0	15,664 (4.25 g/t)	149,700 (4.18 g/t)	88,900 (4.18 g/t)
Total	**0**	**40,581 (3.98 g/t)**	**200,000 (3.98 g/t)**	**88,900 (4.18 g/t)**

As shown in Table 24.3, the pre-production schedule covered a 25-month period. The project was put into commercial production at the beginning of February 2006. Additional mine development will continue for the next 30 months to give access to the A1-A2-A2’, A3, B1-B1’ and B2-Bn zones (below level 200), as shown in the future mining plan (Section 24 E).

24.1.4 <u>Mining Equipment</u>

The development and its ground support is performed using one two-booms Jumbo, two 322 scissors lifts, one Bolter and several jacklegs and stopers. Production drilling is done using two ITH drills with V30 hammer for slot opening. Ore, waste and backfill handling are performed using three scooptrams (6 v3) and three low-profile trucks (26 and 30 tonnes). All underground services duties and transport of personnel are performed using one boom truck, three tractors and two Land Cruisers. Ore, waste and backfill handling is performed using a Loader 966G. Table 24.4 describes the mining equipment fleet actually in operation at East Amphi.

Table 24.4 Mining Equipment Fleet

EQUIPMENT	FLEET	REMARKS
Hydraulic Jumbo (322)	1	Two booms for development excavation
ITH Drill	2	Both used in production drilling 37/8'' (98 mm) diameter hole and one drill set for slot raising with V30 hammer. Done with Machine Roger Contractor.
Bolter (McClean)	1	Development ground support and rehabilitation
Scooptrams (EJC 210 - 6 cy)	3	Two with remote control for stope extraction and one for stope backfilling and development.
Haulage truck (26 mt) Haulage truck (30 mt)	2 1	Low profile trucks for ore, waste and backfill haulage.
Scissor lift	2	Development ground support and rehabilitation
Boom truck	1	Services duties
Tractors	3	Services duties
Toyota Land Cruiser	2	Men carriers
Loader (Cat – 966G)	1	Ore and waste surface handling and material receiving.

24.1.5 Stope Ground Support

The secondary ground support added to support the roof of the stopes consists of 6- metre cemented anchor cables (pattern 1.5m x 1.8m) at the top sill out excavation. Where the hanging wall of the stope is exposed by the development excavation, rows of five cemented anchor cables of six metres in length are installed under the hanging wall belly and directly on the drift wall. Those cables are tied up in a birdcage style within a pattern of 1.2 m x 1.2 m.

Each long-hole stope is surveyed by Cavity Monitoring Survey (CMS). The results are treated to calculate the dilution average and to quantify the pillar damage.

24.1.6 Ventilation and Services

A global fresh-air flow of 160,000 cfm is provided for mine ventilation purposes. The two main fans (150 HP each) and the two propane air heaters (8 MBTU each) are located at surface on top of the old exploration shaft. Table 24.5 shows the basis for the ventilation requirements of the East Amphi project.

Fresh air for the mine is routed to the new underground project via the old shaft and the two track drifts (6' x 6') located on levels 100 and 150. Ventilation doors are installed on those two levels forcing the fresh air to reach the main vent raise located on the west side of the ore body. Where needed, regulators are installed in that vent raise for proper distribution of fresh air to the different levels. Finally, the air is exhausted to the surface through the main ramp. Secondary fans are installed as mining dictates.

Electricity is the main form of energy used in the underground mine to operate the mining equipments is electrical. A 750 MCM cable supplies five electrical sub-stations (1,000 kva each) based on each production level. Two 2,000 cfm screw compressors are installed on the surface. They provide the mine with the compressed air required for operation.

A backfill plant is erected south of the open pit. The main components of the backfill plant are a cement silo (160 tonnes), a screw conveyor (10’’ dia), a mixing tank (20 m3), a mixer, a water tank (8 m3) and a retardant tank (8 m3). From the backfill plant, the slurry line passes by the portal and into the main ramp down to the underground slurry transfer plant on level 100. The main components of the underground slurry transfer plant are a mixing tank (14 m3), two small mixers, two SRL slurry pumps and an “Airlift” system. Primed with the airlift system, the slurry is distributed by gravity to each level using holes in the rock and various piping.

Two refuges are installed on levels 100 and 150. Powder and detonator magazines are both located on level 125.

The open pit is used as a large decantation basin. From the underground pumping circuit, the used mine water is discharged into the open pit. A hole has been drilled from level 100 (4,900 El) and breakthrough in the lower part of the open pit (4,955 El). A 4.5’’ ID pipe (HDPE 5’’ DR-11) has been cemented in the hole. This pipe feeds by gravity the industrial water to the underground mine and the water. In this second application, the flow of the exhaust water is measured and the water is pumped to the surface. Then, the water is treated for suspended solids and toxicity before being discharged to the environment. The environmental pumping circuit and treatment do not operate during the frozen months.

There is currently no indication that the ground water level is affected by the mining operations.

Table 24.5 Underground Ventilation Requirements

EQUIPMENT	FLEET	UNIT REQ. (CFM)	UTILIZATION FACTOR	FLOW REQUIRED (CFM)
Hydraulic Jumbo (322)	1	4,500	0.6	2,700
Bolter (McClean)	1	4,500	0.6	2,700
Scooptrams (EJC 210 - 6 v^3)	3	17,600	2.5	44,000
Haulage truck (26 & 30 mt)	3	19,300	2.5	48,250
Scissor lift	2	(1x) 4,500 (1x) 11,900	1.2	9,800
Boom truck	1	4,500	0.6	2,700
Tractors	3	(2x) 10,000 (1x) 8,700	2.5	23,900
Toyota Land Cruiser	2	7,300	1.5	10,950
			Total	**145,000**

24.2 Processing and Metallurgy

This section presents a description of the following subjects:

- A description of the different steps involved into the processing of the East Amphi gold ore by the Camflo mill,
- A summary of the Lakefield Research mineralogical and metallurgical laboratory test work and results,
- An evaluation of the Camflo mill for mill the East Amphi ore.

24.2.1 Milling facilities - Camflo Mill

East Amphi's ore is hauled by truck towards the Camflo mill, located at an approximate distance of 13 km from the mining site. Transport Nord-Ouest ensures ore transportation on the basis of a three-years contract. Richmont Mines owns 100% of the Camflo mill.

The Camflo mill is a traditional gold recovery mill with a rated capacity of 1,180 tonnes per day. This mill is a conventional Merrill-Crow type, with circuits for crushing, grinding, gold cyanidation, precipitation using zinc powder and tailings area circuits. Ore is crushed in a 92 cm x 102 cm standard surface jaw crusher and stored in a coarse ore bin. Coarse ore bin material is screened, with minus 1.9 cm material sent to a dedicated fines ore bin and oversize material being re-crushed via an open-circuit standard cone crusher. Crushed material is screened, with undersize (-1.9 cm) being sent to the fines ore bin and oversize material feeding a short head cone crusher in a closed circuit with the latter screen. Fines ore bin material is metered into the grinding circuit at the desired throughput. Ore passes through an open-circuit rod mill. Lime and lead nitrate reagents are added to the rod mill for chemical adjustments. The rod mill's discharge pump box feeds a cyclone cluster. Cyclone overflow is sent to thickening prior to leaching. Underflow is split to feed two parallel ball mills. Cyanide is added to the cyclone underflow. The ball mills discharge flow to the rod mill discharge pump box, thus closing the loop.

Cyclone overflow, having already come into contact with cyanide, is thickened prior to the leaching circuit. Underflow is sent to the leach circuit. The circuit comprises a series of six leach tanks in combination with three sets of drum filters. The circuit flow is set as a three-stage solution recovery circuit. After the initial three leach tanks, the first stage of filtering removes solution, and the washed cake is then re-pulped prior to feeding the fourth and fifth leach tanks.

Slurry from the latter tank feeds in turn the second solution-recovery stage. The third and final stage filtered cake is washed, re-pulped and sent to the tailings impoundment area. Tailings water is recycled as barren wash and third-stage repulp solution. Solution from the first two filtering stages is combined with the grinding product prior to feeding the thickener. Overflow from the thickener is stored as pregnant solution, which will feed the gold-recovery circuit. The gold-recovery procedure is standard zinc cementation (Merrill Crowe). The pregnant solution is bag-clarified to remove any particulate matter. Clarifier solution discharge is processed through a de-aeration tower. Once de-aerated, the solution is exposed to zinc dust. The gold-bearing zinc precipitate is collected in standard press filters. The filter cake is smelted and gold bars are poured. The filter solution is recycled as barren solution and serves primarily as make-up water at the grinding circuit as well as wash solution for stage 1 and two drum filters.

24.2.2 Metallurgical Test Work from Lakefield

- The mill's gold-recovery rate is fixed at 97.5% for the East Amphi ore. The ore has been evaluated using the mineralogical and metallurgical laboratory test results generated by Lakefield Research (2002). The three bulk tests done in 2005 returned a gold recovery of 97.7% on 40,581 tonnes treated at the Camflo mill and confirmed Lakefield's results.

The Lakefield Research mineralogical and metallurgical laboratory test work and results are summarized as follows.

Mineralogy

Three ore samples were prepared namely:

- A and B zones consisting of diorites containing talc and chlorite;
- B Zone consisting of fractured diorite without talc and chlorite;
- P zones consisting of porphyry that are currently not included in the reserves;

The three samples generally contained the same accessory minerals including non-opaque minerals, pyrite, magnetite, chalcopyrite and pyrrhotite.

Ore sample no 1 also contained minor amounts of molybdenite and galena. Magnetite was present as free grains and as inclusions within pyrite. Gold occurred as free grains, or was associated with molybdenite and pyrite.

Ore sample no 2 contained a greater amount of ilmenite than ore samples no 1 and no 3. It contained finer-sized gold grains that were mostly locked or attached to pyrite. Minor proportions of gold occurred as free grains.

Ore sample no 3 was composed of pyrite, chalcopyrite, pyrrhotite, galena and rutile. Galena was more prominent than in ore samples no 1 and no 2. Ore sample no 3 contained the smallest number of grains, although relatively large portions were liberated. Gold associated with pyrite also occurred, mainly as locked inclusions with attachments or intergrowths with galena.

Head Grade Assays

Table 24.6 shows the head grades that were assayed or calculated during the test work program. The presence of coarse gold in the ore samples makes the gold grade determinations more difficult (nugget effect) as evidenced in the table.

Table 24.6 Head Grade

	AU GRADE (g/t)				AG (g/t)	S (%)
Sample	**Metallic**	**CN test**	**Grav +CN test**	**Ave**	**Direct**	**Direct**
Ore #1	7.0	8.1	8.15	7.75	<0.5	0.30
Ore #2	8.4	7.2	7.06	7.55	<0.5	1.08
Ore #3	3.6	3.0	-	3.30	0.6	1.46

Grinding

Bond rod mill grindability and ball mill grindability tests were performed on the three samples. The results are summarized in Table 24.7.

Table 24.7 Grinding Test Results Summary

	ROD MILL WORK INDEX	BALL MILL WORK INDEX
Sample	**kWh/t**	**kWh/t**
Ore #1	10.6	9.0
Ore #2	11.7	9.7
Ore #3	14.5	11.6

Overall, the ore samples tested are all relatively soft, with their work indexes lower than the ore currently milled at the Camflo mill.

Gravity Concentration

Gravity recovery tests using a Knelson concentrator and a Mozley shaking table were performed on the ore samples no 1 and 2. As shown in Table 24.8, the results were similar for the two ores, as 41% of the gold was recovered by gravity separation. Lakefield reported that a fair amount of coarse gold flakes was observed in the table concentrates, especially for ore sample no 1.

Table 24.8 Summary of Gravity Test Results

		AU GRADE (g/t)	AU DISTRIBUTION (%)
Ore #1	Concentrate	15,455	41
	Tailings	4.85	59
	Head	8.15	100
Ore #2	Concentrate	4,070	41
	Tailings	4.15	59
	Head	7.06	100

In the mill, an overall lower gravity recovery, in the 25-35% range, can be expected, as the gravity concentrate has to be upgraded to about 50-60% gold (500,000 – 600,000 g Au/t) for smelting.

Cyanidation

A summary of the cyanidation test results is presented in Table 24.9. Initially, the three samples were submitted to a series of four cyanidation tests at various grind size. The tests were carried out over 32 hours under Kiena mill cyanidation conditions (density of 53% solids, pH of 11 and NaCN concentration of 450 mg/l). The results of this first series of tests (CN-01 to CN-12) showed relatively slow kinetics. It was suspected that coarse free gold was partly responsible for the slow kinetics.

A second series of tests was run to investigate the effect of removing coarse gold by gravity separation (CN-13 to CN-18) and the effect of using higher NaCN concentrations (CN-19 and CN-20).

Table 24.9 Summary of Cyanidation Test Results

TEST	SAMPLE	GRAVITY	GRINDING		% AU RECOVERY		CALC. HEAD
No.		(Y/N)	K_{80}(μ)	%<200 mesh	24 h	32 h	g /t
CN-01	Ore no1	N	64	86	94	97.0	6.58
CN-02	Ore no1	N	78	79	82	85.2	8.45
CN-03	Ore no1	N	91	73	94	97.3	8.39
CN-04	Ore no1	N	117	64	92	96.2	9.30
CN-05	Ore no2	N	58	89	90	95.8	6.32
CN-06	Ore no2	N	67	85	92	96.5	8.35
CN-07	Ore no2	N	85	75	90	94.7	7.21
CN-08	Ore no2	N	109	66	92	93.3	6.72
CN-09	Ore no3	N	53	91	94	95.9	2.72
CN-10	Ore no3	N	67	84	89	95.7	3.29
CN-11	Ore no3	N	80	78	94	95.0	2.93
CN-12	Ore no3	N	101	68	95	93.3	2.99
CN-13	Ore no1	Y	161	56	90	93.4	8.11
CN-14	Ore no1	Y	104	68	95	96.7	8.01
CN-15	Ore no1	Y	74	81	97	97.8	8.09
CN-16	Ore no2	Y	168	52	87	89.4	7.05
CN-17	Ore no2	Y	109	66	93	94.0	7.17
CN-18	Ore no2	Y	75	80	94	95.5	6.98
CN-19	Ore no1	N	88	75	93	97.3	7.9
CN-20	Ore no2	N	95	72	92	95.1	7.39

The effect of grind size was fairly strong for the three ores. The finer the grind size, the higher the recovery (CN-01, 05 and 09). Gravity separation has a limited effect on the final gold recovery (after 32 hours) but has the benefit of increasing the kinetics of leaching. Recoveries in the 95-97% range were obtained after 24 hours for ore no 1 (CN-14 and 15) and 93-94% for ore no 2 (CN-17 and 18). The use of a higher NaCN concentration as a way to improve the leaching kinetics produced a limited effect (CN-03 vs CN-19, and CN-07 vs CN-20). Conditions of tests CN-14 for ore no 1 and CN-17 for ore no 2 are proposed to develop the design criteria.

Settling Tests

Settling rate tests were performed to assess the effect of flocculants (type and dosage) on the thickener area requirement for different thickener underflow densities. The flocculants Magnafloc 351 and E-10 were found to be superior to the others tested. The differences between the two were small, and the flocculant Magnafloc 351 was selected, as it is already used at the Camflo mill. The testwork results show that ore sample no.1 is the most problematic.

Acid Generation Potential

Ore samples no 1 and no 2 contained a large fraction of carbonates (CO_3) and are definitely not acid generating, as their neutralization potential (NP) is much higher than their acid potential (AP). Ore sample no. 3 must be considered uncertain with respect to acid-generation, due to its low ratio (NP/AP) of 1.4. Samples with ratios between 3.0 and 1.0 require more extensive work (humidity cells) to confirm whether they will be acid generating or not.

24.2.3 Camflo Mill Evaluation

In 2005, Soutex was given the mandate of preparing a pilot project for the East Amphi ore at the Camflo mill. This section presents their conclusions as well as their observations and comments.

The main conclusions drawn from their report are:

- In view of the hardness of the ore and the historical data on the operation of the Camflo mill, capacity is not expected to pose a problem. Nonetheless, given the hardness of the material, it is recommended to begin with a tonnage of approximately 50 tonnes per hour;
- The particle size that can be achieved in the grinding circuit will allow for the recovery of 95%, provided that the material be similar to the Type 2 material of the previous assays;
- The cyanidation time will be sufficient;
- Since cyanide is added during grinding, it will not be necessary to have a gravimetric recovery circuit;
- The determination of the feed grades according to the current method poses a risk of possible accumulation of a large amount of gold in the grinding circuit;
- Sampling of the waste matter from the rod mill is recommended for a better assessment of the feed grades;
- Improving the sampling of the residues is recommended. The current method has the potential for bias that could hide possible losses;
- Starting with a pilot project is recommended in order to confirm the behaviour of the ore at the mill;
- The pilot project should last at least four or five days in order to yield conclusive results. No potential for acid production is expected.

Grinding

The material comes from a geologic unit different from the one that was exploited in 2002. This unit will differ in the hardness of the rocks and in the physical characteristics of the gold. The ore contains primarily fine gold and can be expected to resemble Type #2 of the assay that has been done.

The history of the operation of the concentrator shows that different ores with grindability indices varying by 30% have been processed at the Camflo mill. The ore from the East Amphi mine can be expected to lie within the limits already observed. The ore may come from three mineralogical units, namely porphyry, diorite and Unit V4.

With regard to grinding, it is possible that the grindability index of the rod mill will be higher. The result would be coarser waste matter from the rod mill, which could cause problems for pumping towards the hydrocyclone. This situation has been seen in the past with harder ores.

Cyanidation

The ore contains approximately 1% pyrite. According to the geological staff of Richmont Mines, it contains no other sulphites, such as pyrrhotite, arsenopyrite, chalcopyrite or other sulphites containing copper. The gold is not visible but is associated with the presence of pyrite, which probably do contain gold.

Usually the presence of sulphides entails the consumption of oxygen in the cyanidation tanks. The reactivity of the sulphides can be reduced by adding lead nitrate. The process must, however, be carefully monitored to see whether it is necessary to adjust the addition of these reagents.

The tests of cyanidation have shown that a minimum time of 32 hours is required for cyanidation. The current cyanidation circuit allows for a retention time of 36 to 40 hours. That is sufficient to ensure that the recoverable gold will be completely dissolved.

The consumption of cyanide in the laboratory is an indication of the consumption that will be observed in the mill. By extrapolating from the results of the laboratory tests, we can conclude that the ore will consume a similar amount of cyanide, taking into account the concentration that is maintained at the mill.

Gravimetry

Since cyanide is added to the grinding circuit, it is not necessary to install equipment for gravimetric recovery. There is no possible benefit in recovery that could justify this extra expense. In addition, the ore that will be extracted from the East Amphi mine will probably contain little free gold; thus the use of gravimetry is even less attractive.

Acidity

During the preliminary analysis of the material, it was emphasized that ore of Type 3 has characteristics that pose a risk of acid production. Since the material that will be processed is a mixture of porphyry and diorite, the amount of carbonate will be sufficient to ensure an adequate acid-neutralizing capacity. There is no cause for worry that the material may produce acid.

24.3 Market and Contracts

For the year 2006, Richmont Mines has signed three contracts:

- The "***Production drilling & Slot raising***" contract has been signed with Machine Roger International. It covers 100% of the activities for long-hole drilling at 37/8'' diameter and the slot-raise opening at 30'' diameter;
- The "***Ore transportation***" contract has been signed with Transport Nord Ouest Inc. It covers the ore trucking between the East Amphi site and the Camflo mill located at 13 km apart. Truck loading is under Richmont Mines' responsibility;
- The "***Ore custom milling***" agreement has been signed with Camflo Mill Inc. which is a subdivision of Richmont Mines. It covers ore processing and gold-recovery terms.

The gold lingots are refined by the Royal Canadian Mint. The gold is then sold based on the higher spot prices offered by two different buyers. Richmont Mines has no hedging contracts in place.

24.4 Environmental Considerations

Environmental considerations as outlined in the McWatters report for the project's environmental permit request and various reports from Golder and Associates and Laboratoire d'expertise de Québec (LEQ) (1999) have also been taken into account during the conceptual design phase of the project and facilities, as well as their cost estimate.

24.4.1 Restoration

In January 1999, the East Amphi closure plan was evaluated by Roche Groupe Conseil for McWatters's open-pit operation phase. In October 2004, the revision for Richmont Mines underground operation phase was done by the same consultant. The restoration program will mainly consist of:

- Dismantling all buildings, structures and foundations;
- Soils characterization for contaminations, and any remedial measures required;
- Site revegetation;
- Levelling of the settling ponds;
- Securing the open-pit access.

The above work is the subject of the warranty of completion estimated at $82,600 and the funding is currently warranted.

All development waste material of the exploration phase of the project has been placed on a surface waste dump. The waste rock is not acid generating and does not require any particular environmental control measures.

24.4.2 Water Management

The phase of dewatering the open pit proceeded at a rate of 4 m^3/min from August to October 2003 and from January to June 2004. Given the excessively high concentration of suspended matter in the water found in the open pit, flocculent treatment was installed in April 2004. The surface settling basin was thus put into service to trap the flocculent matter.

Water management for the underground exploration/operation phase can be summarized as follows:

- The open pit acts as a primary settling basin and also serves as a reservoir for the accumulation of water during the winter;
- The surface basin has the effect of buffing;
- The industrial water and drainage water for the mine are maintained in a closed circuit in the open pit;
- Any excess water must be processed with a flocculent and an acid in order to reduce the concentration of suspended matter and the pH, respectively;
- The surplus water from the pit is currently treated at a rate below 1 m^3/min, sent to the buffing basin and returned to the environment through a drainage ditch.

24.5 Future Mine Plan

While production has started in the B2-Bn Zone (over level 200), and when development has been completed, development of the satellite zones (A1-A2-A2’, A3, B1-B1’ and B2-Bn, below level 200) is anticipated to be undertaken over a 30-month period to allow continuous production up to the depletion of the reserves. The reserves, estimated at 641,000 tonnes (including B2 Zone over level 200), would therefore be mined out in a period of some 54 months from the beginning of 2006.

Table 24.10, shown below indicates the estimated annual tonnage by zone planned to be extracted according to actual ore reserves from January 2006 to mid-2010.

Table 24.10 Production Forecast 2006-2010

ZONE	2006	2007	2008	2009	2010
B2-Bn (> 4,800 El)	200,000	89,000			
B2-Bn (< 4,800 El)		22,500	48,000	48,000	22,000
A2-B1 (> 4,848 El)		16,100	33,600	33,600	13,000
A2-A2’ (< 4,848 El)		3,900	31,000	29,800	10,000
A3 (> 4,820 El)		7,500	14,400	13,600	5,000
Total:	200,000	139,000	127,000	125,000	50,000
Grade (g/t):	3.98	4.68	5.58	5.56	5.61
*Ounces recovered:	24,952	20,392	22,214	21,786	8,793

* After mill recuperation at 97.5%

A total of 8,775 metres of excavation (drifting and ramp) will be required to give access to the satellite ore zones over the next 30 months, starting in mid-2006.

The B2 and Bn zones under 200 metres in elevation will be accessed by driving the actual ramp down to elevation 4,750. Four sub-levels (footwalls drifts) in the eastern portion will excavated, while three sub-levels will be driven on the western side. A total of 2,600 metres of excavation are planned to extract the ore using the long-hole method with cemented backfill. Figure 24.8, shown below sketches the longitudinal view of the proposed mining plan.

Figure 24.8 – Longitudinal view - B2-Bn zones (under 200 level)

The A1-A2-A2’ zones, these zones are located between elevations 4,770 and 4,906 (Figure 24.9). The A2 Zone between elevations 4,848 and 4,906 is parallel to the B1 Zone, discussed below. The upper limit is related more to grade than to the surface pillar under the former open pit, which is offset from the zone. The long-hole method is mainly proposed except for the first 10 metres in elevation, between elevations 4,848 and 4,858, where drift-and-fill method is anticipated.

Figure 24.9 – Longitudinal view - Mining approach - A1-A2-A2' zones

Accesses to that zone and to the parallel B1 Zone will come from the eastern side of the footwall drifts of the B2 Zone, on levels 100, 125 and 150. The total development required comes to 2,635 metres for the A2 Zone, as sketched in Figure 24.9, and the B1 Zone, described below and in Figure 24.10.

Under elevation 4,848, the A2 Zone shows continuity at depth. A seven metres pillar will be left in place. Accessing to the zone is also achieved from B2-Bn zones, but on levels 175 and 200. The total development required is 2,250 metres. The main mining method forecast is drift-and-fill with successive 2.8 metres cuts from two main 23-metre horizons. The long-hole method is used to recover the portion between elevations 4,793 and 4,808 in successive panels 15 metres height by 6 metres. Also, 10-metre uppers are forecast to mine below the 7 metres pillar, between elevations 4,831 and 4,841.

Figure 24.10 – B1 zone longitudinal layout

In the B1 Zone, a 30-metre surface pillar has to be left in place, according to Golder and Associates (1999). As for the parallel A Zone, the first 10 metres in elevation are taken with drift-and-fill mining, while the main part will be taken with the long-hole method.

For the A3 Zone, access to this zone is done by west-end footwall drifts from the B2 Zone on levels 150 and 175. Some 1,290 metres of development is required to mine the zone from elevations 4,812 and 4,836 by drift-and-fill and the upper part of the zone by the long-hole method (Figure 24.11).

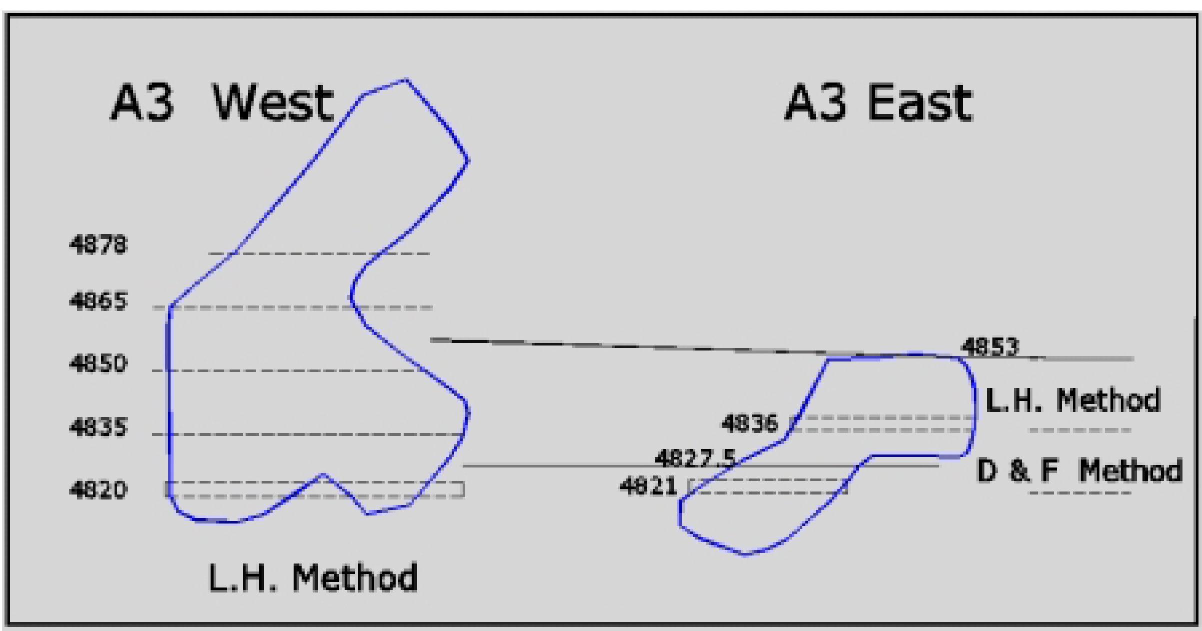


Figure 24.11 – A3 Zone longitudinal layout

24.6 Capital and Operating Costs

Reserves in the B2 Zone above level 200 accounted for 289,000 tonnes as of January 1, 2006. The production level from that zone will be 200,000 tonnes/year for a period of 17 months. The production costs for the 2006 budget is presented in the Table 24.11. Operating costs to mine and mill the B2 ore reserves are forecasted at $62/tonne in 2006, including a spending of $1,15M in 900 metres of drifting in ore.

Table 24.11 Forecasted Production Cost

	B2 - BN $/T BUDGET 2006	SATELLITE ZONES $/T FORECAST
U/G Operation	21,36	26,03
Maintenance and services	15,57	17,13
Technical services and Administration	5,58	7,25
Transportation and milling	19,64	19,64
Total	62,15	70,05

Reserves in the satellites zones total 352,000 tonnes (65,000 ounces). Over 8,775 metres of development are included totalling $10.8M in ore and waste development. The forecast of operating costs is estimated at $70/tonne reflecting the use of cemented backfill and the lower annual mining rate. Overall, the forecast production cost is $66.22/tonne or CAN$520/ounce based on US$450 and 1.2% exchange rate.

Long-hole mining is the approach for A2, B1 (4,898 to 4,906 el), A3 (4,820 to 4,878 el) and B2-Bn zones (4,740 to 4,800 el) with cemented backfill to increase ore recovery, with mining openings limited to 15 metres in height and 6 metres along strike. Both walls (hanging and footwall) are in talc schists.

A 7-metre pillar will be left in the B2-Bn zones below level 200, where mining is currently being performed. In the B1 Zone, a 30-metre pillar is left under the open pit, as recommended by Golder and Associates.

The drift-and-fill method is used in the A2-A2’ zones between elevations 4,770 and 4,841 metres (74,000 tonnes).

Development for the satellite zones is expected to start by mid-2006 at a rate of 140 metres for three months, and 200 metres per month for the last three months of 2006. The forecast for 2007 is 360 metres per month. In 2008 and 2009, an average of 155 metres per month will be developed.

24.7 Economic Analysis, Payback and Taxes

24.7.1 Economic Analysis

Note that this section was completed after considering a non-cash write down of mining assets of $26,040,953 in fiscal 2005. This write down was accounted according to the method of discounted future cash-flow based on current proven and probable reserves and a gold price of CAN$575. More details on this asset write down can be found in the company's 2005 annual report.

A capital cost of $23,719,554 was spent to provide access to the B2 ore reserves down to level 200. The table 24.12 of cash-flow summarizes the required investment, the operational costs and the revenues anticipated over the 4.5-year period. A total cash flow of $2.9M (before taxes) and $2.6M (after taxes) is forecast from CAN$53.0M revenues at CAN$540/oz. The internal rate of return (IRR) for East Amphi at CAN$540 would be 4.82% before tax and 4.19% after tax. The breakeven point is set at around 54,800 ounces of gold produced or after two years and five months.

The sensitivities are shown in the following graphic related to the gold price (CAN$500 – CAN$600), the operation cost (-10% to +10%) and development cost (-10% to +10%) The project is sensitive mainly to gold price and production cost based on specific dilution parameters (10% in B2 Zone and 20% in satellite zones). The breakeven point is close to CAN$520/ounce.

24.7.2 Payback

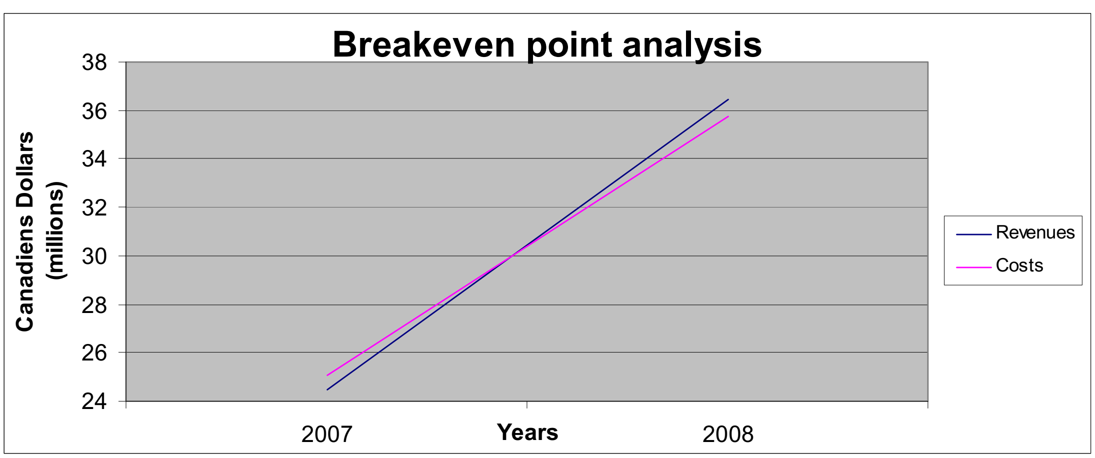
The payback period is around 2 years and 5 months based on a CAN$540/ounce scenario.

Table 24.12 Cash-flow

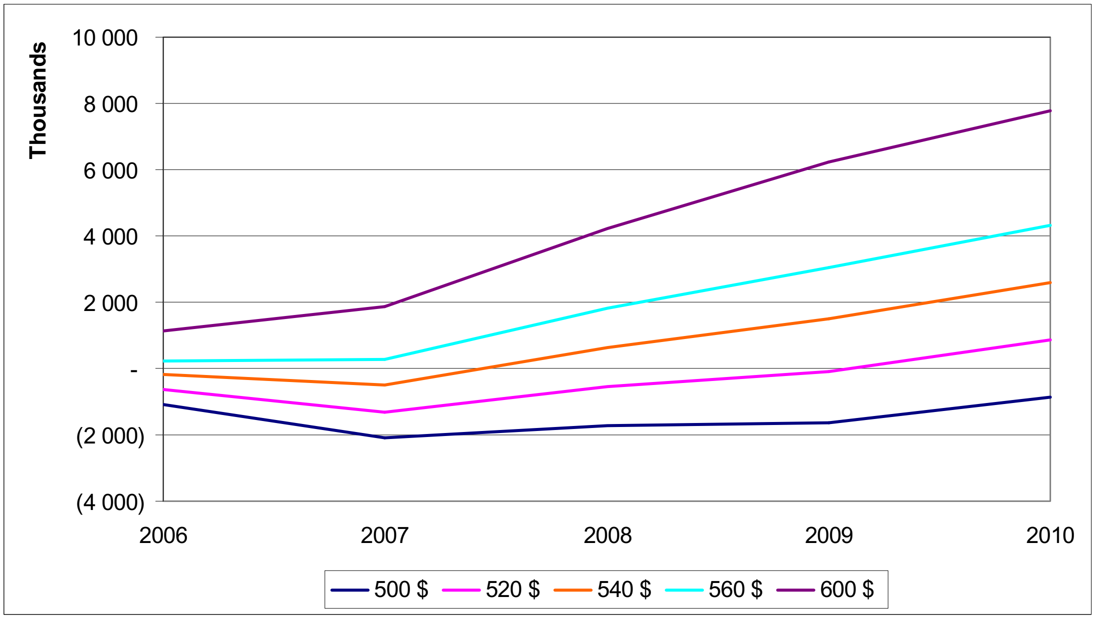
	2006	2007	2008	2009	2010	TOTAL
Metric tonnes	200,000	139,000	127,000	125,000	50,000	**641,000**
Grade	3.98	4.68	5.58	5.56	5.61	
Gold price CAN$	540.00	540.00	540.00	540.00	540.00	
Cost per metric tonne	62.00	64.24	69.88	69.92	70.00	**66.22**
Development cost	1,300,000	2,440,000	1,810,000	2,060,000	0	**7,610,000**
Ounces (recovered 97.5%)	24,952	20,392	22,214	21,786	8,793	**98,137**
Revenues	13,474	11,011,591	11,995,749	11,764,521	4,748,127	**52,994,173**
Operating cost	12,400,000	8,930,000	8,875,000	8,740,000	3,500,000	**42,445,000**
Development cost*	1,300,000	2,440,000	1,810,000	2,060,000	0	**7,610,000**
Cash flow	(225,815)	(358,409)	1,310,749	964,521	1,248,127	**2,939,173**
Cash flow after taxes, 12%	(198,717)	(315,400)	1,153,459	848,778	1,098,352	**2,586,472**
		NPV before taxes	5%	2,363,579		
			10%	1,917,062		
			15%	1,566,477		
		NPV after taxes	5%	2,079,949		
			10%	1,687,015		
			15%	1,378,500		
		IRR before taxes		4.82%		
		IRR after taxes		4.19%		

The internal rate of return (IRR) for East Amphi at CAN$540 would be 4.82% before tax 4.19% after tax.

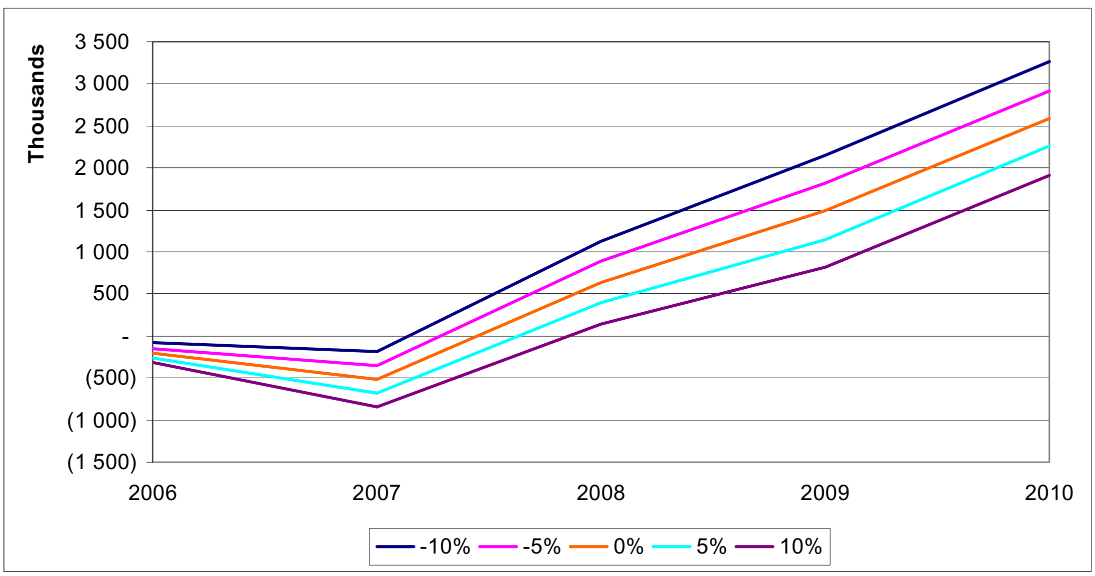
* Includes only capitalized development (waste). Ore development is included in operating costs for an additional amount of $2,045,000 in satellite zones and $1,145,000 in B2 Zone (above 200 level).



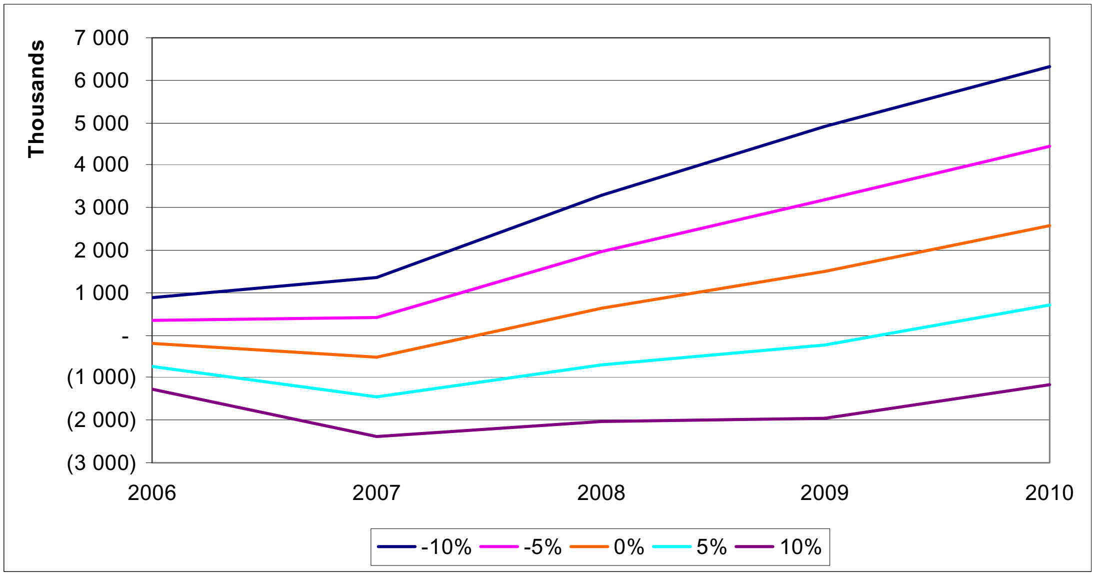
The breakeven point is reach after 2 years and 5 months.



East Amphi after-tax cumulative cash flow sensitivity to CAN$/ounce gold price variance



East Amphi after-tax cumulative cash flow sensitivity to development cost variance



East Amphi after-tax cumulative cash flow sensitivity to operating cost variance

24.7.3 Taxes

The future East Amphi operation will be subject to the following taxes:

- Quebec mining duties
- Federal and provincial income taxes

Mining duties are payable to the Quebec Ministry of Natural Resources, Wildlife and Parks, at a rate of 12% of income from mining operations.

The company will assume a charge of 12% for mining taxes since it does not have a tax pool allowing it to reduce its taxable income.

For 2005, federal income taxes are payable at a rate of 26%. This rate is effectively reduced by a resource allowance available to mining companies. The resource allowance is calculated at 25% of a company's resource profits. By 2007, the federal income tax rate will be reduced to 21% and the federal resource allowance will be eliminated. The new legislation will also allow for the deductibility of provincial mining duties paid. The Company expects the new legislation to have a neutral impact on income taxes. Provincial income taxes are assessed on a similar basis as the current federal system. The Company currently is assessed at a blended Quebec provincial rate of approximately 8% (after provincial resource allowances). Although many provinces have indicated their intention to mirror the federal changes in the resource taxation structure neither Ontario nor Quebec have not introduced such changes yet.

Richmont Mines has existing Canadian and Quebec income taxes pools of over CAN$40M and non-refundable provincial tax credits of $4.5M and federal tax credits of $2M may be used within the next ten years to reduce income taxes otherwise payable. East Amphi would pay virtually no tax.

24.8 Mine Life

Mine life is anticipated to be around 4.5 years, according to ore reserves and mine plans for a total of 100,000 ounces of gold.

On the positive side, the East Amphi property has a 4 km strike along the mineralized Malartic shear, and has potential to find additional reserves. The actual resources base could also be partly transferred in minable reserves.

But, on a more conservative side, the actual probable reserves are economically viable at a CAN$540/oz scenario. Ore and waste development costs for the satellites zones have been evaluated at $1,230/metre for a total amount of $10.8M. Those costs will have to be closely tracked and appropriate ground control measures taken as some of them are located in a schist formation.

A complete review of the proposed mining methods, associated costs as well as mining productivity and risk according to ground conditions will be done for that second phase of development with the actual and future short term mining experience acquired in the B2 Zone above the 200 level. Currently ground conditions look more stable than predicted, mainly because of the shallow depth of mining and the absence of lateral constraints.

FIGURE 5.1

GEOGRAPHIC LOCALISATION

Geographic Localisation

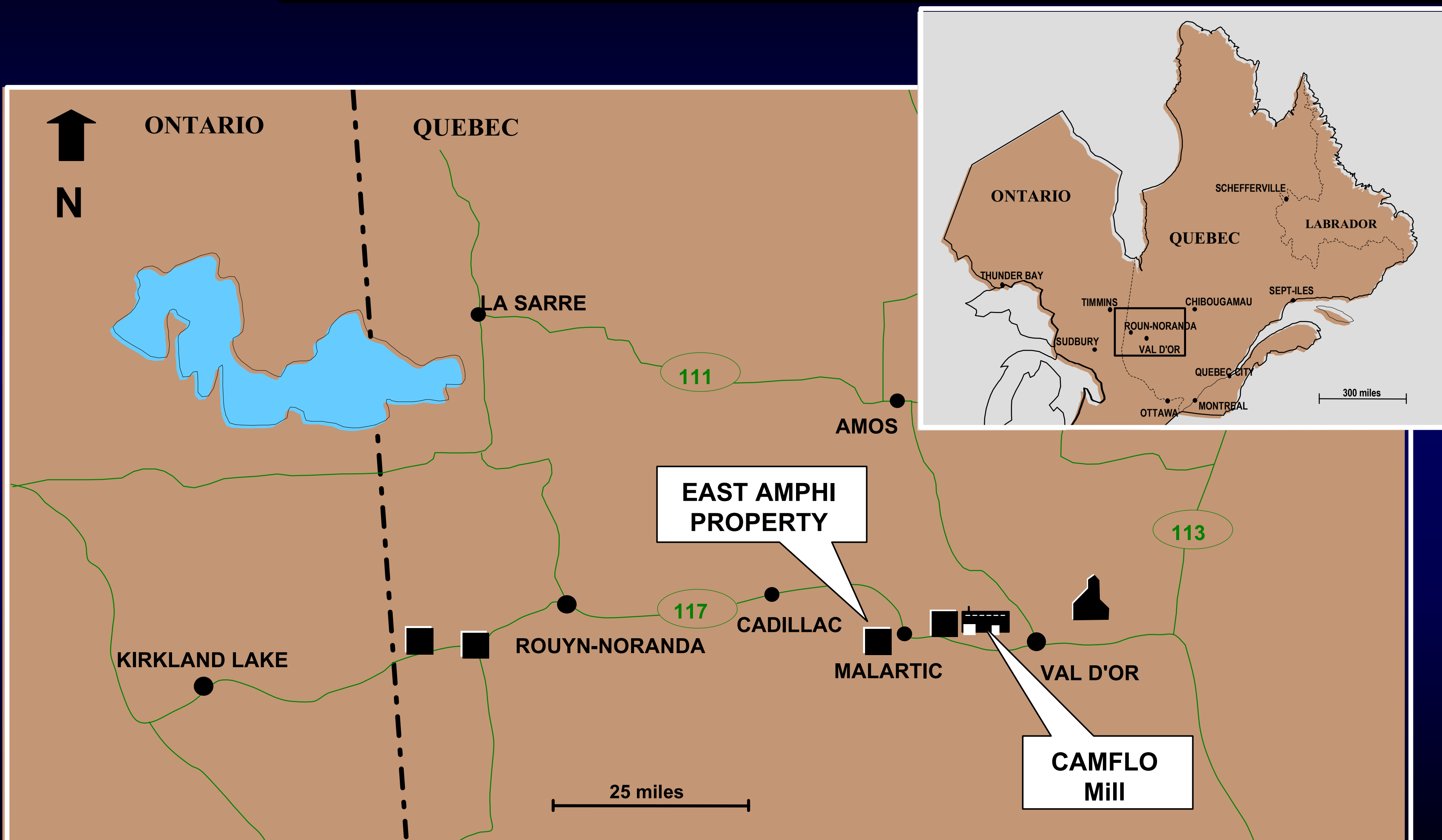
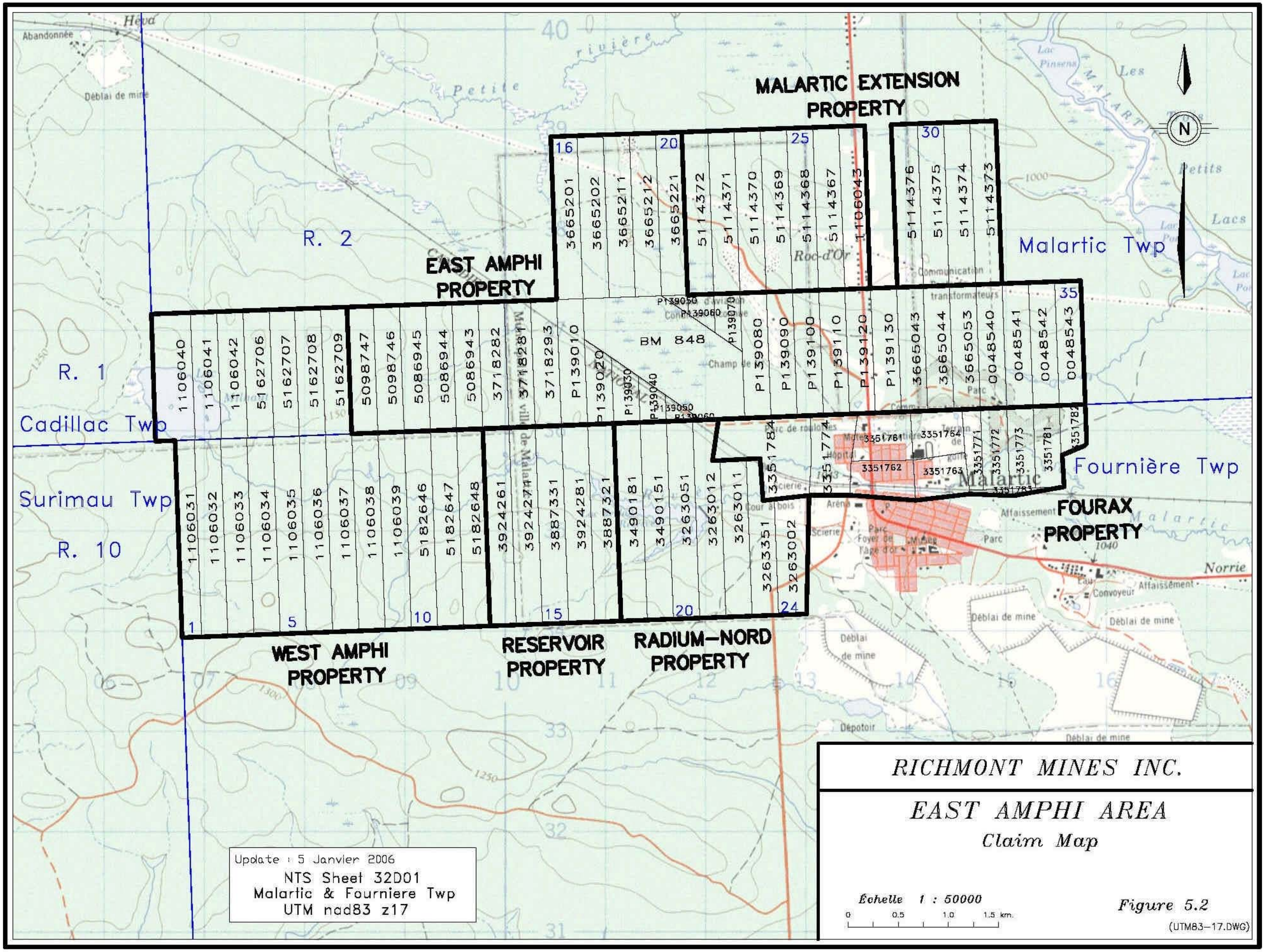


FIGURE 5.2

CLAIM MAP



MALARTIC EXTENSION PROPERTY
EAST AMPHI PROPERTY
WEST AMPHI PROPERTY
RESERVOIR PROPERTY
RADIUM-NORD PROPERTY
FOURAX PROPERTY
Malartic Twp
Fournière Twp
Cadillac Twp
Surimau Twp
R. 1
R. 2
R. 10
BM 848
Update : 5 Janvier 2006
NTS Sheet 32D01
Malartic & Fourniere Twp
UTM nad83 z17
RICHMONT MINES INC.
EAST AMPHI AREA
Claim Map
Échelle 1 : 50000
0 0.5 1.0 1.5 km.
Figure 5.2
(UTM83-17.DWG)

FIGURE 5.3

GENERAL SURFACE PLAN

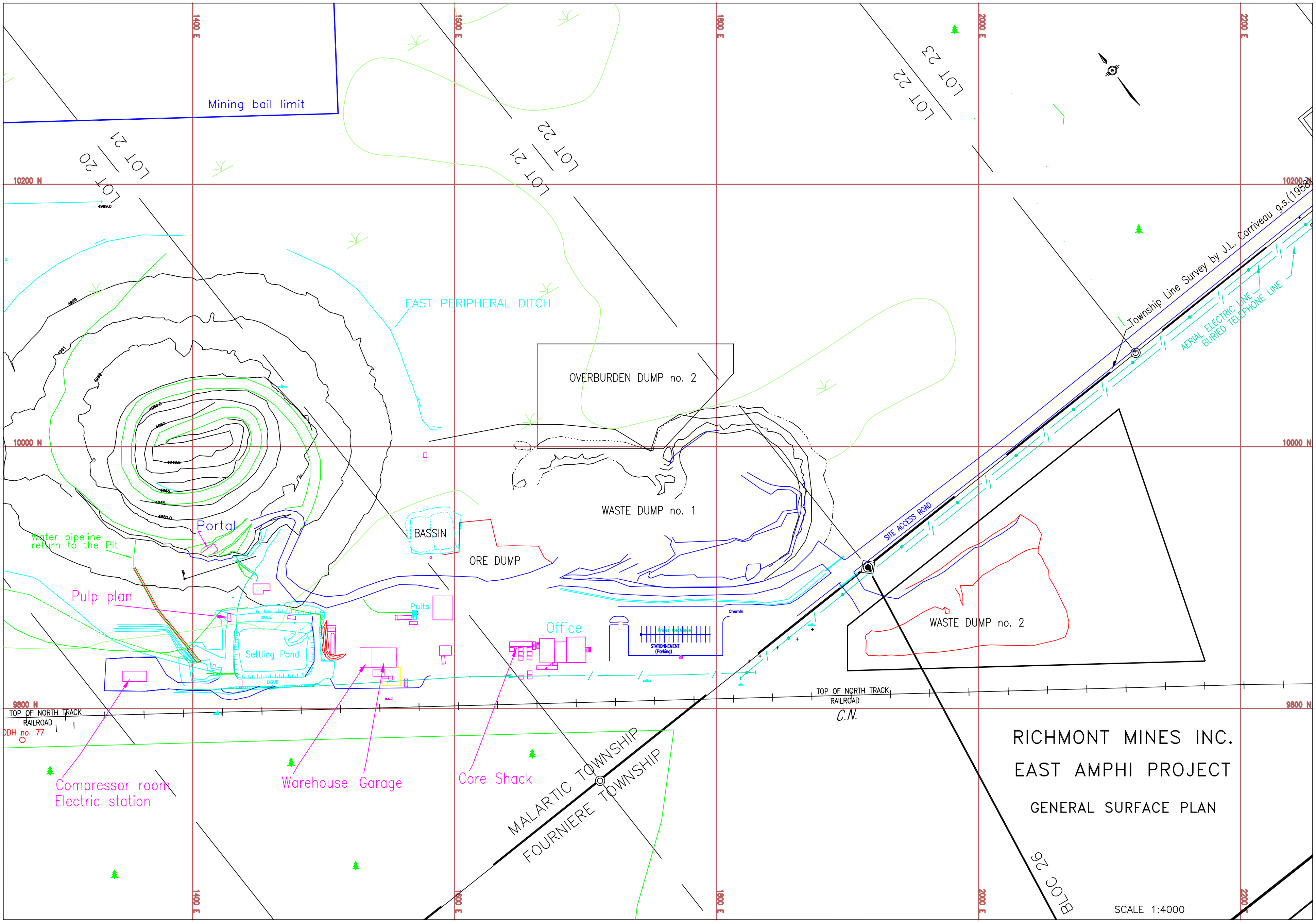

RICHMONT MINES INC.
EAST AMPHI PROJECT
GENERAL SURFACE PLAN
SCALE 1:4000
Mining bail limit
LOT 20
LOT 21
LOT 22
LOT 23
EAST PERIPHERAL DITCH
OVERBURDEN DUMP no. 2
WASTE DUMP no. 1
WASTE DUMP no. 2
ORE DUMP
BASSIN
Portal
Water pipeline return to the Pit
Pulp plan
Settling Pond
Puits
Office
STATIONNEMENT (Parking)
Chemin
Compressor room
Electric station
Warehouse
Garage
Core Shack
MALARTIC TOWNSHIP
FOURNIERE TOWNSHIP
BLOC 26
TOP OF NORTH TRACK RAILROAD
C.N.
DDH no. 77
SITE ACCESS ROAD
Township Line Survey by J.L. Corriveau g.s.(1988)
AERIAL ELECTRIC LINE
BURIED TELEPHONE LINE
10200 N
10000 N
9800 N
1400 E
1600 E
1800 E
2000 E
2200 E

FIGURE 8.1

REGIONAL GEOLOGY

Regional Geology

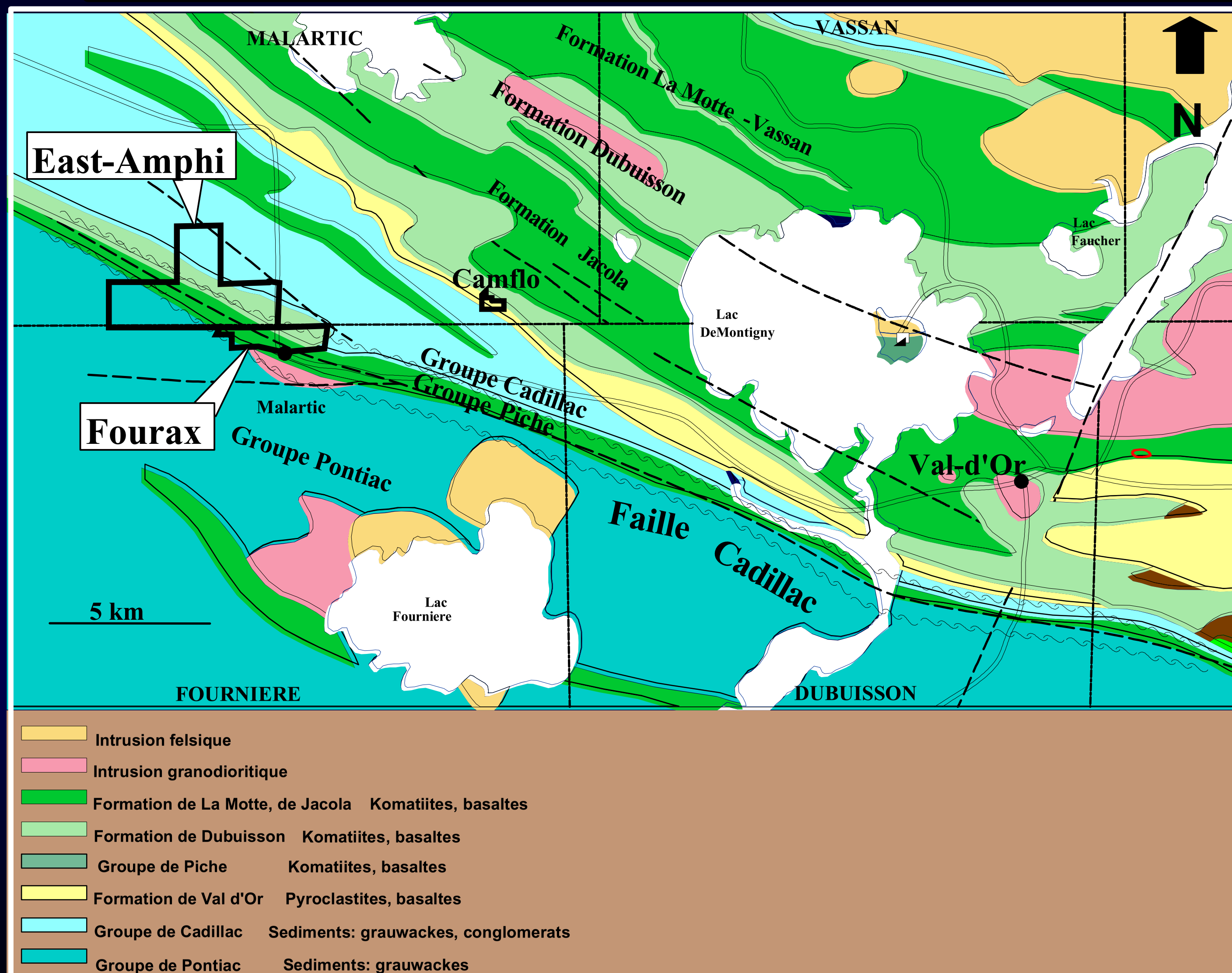


FIGURE 8.2

GEOLOGY PROPERTY

Geology Property

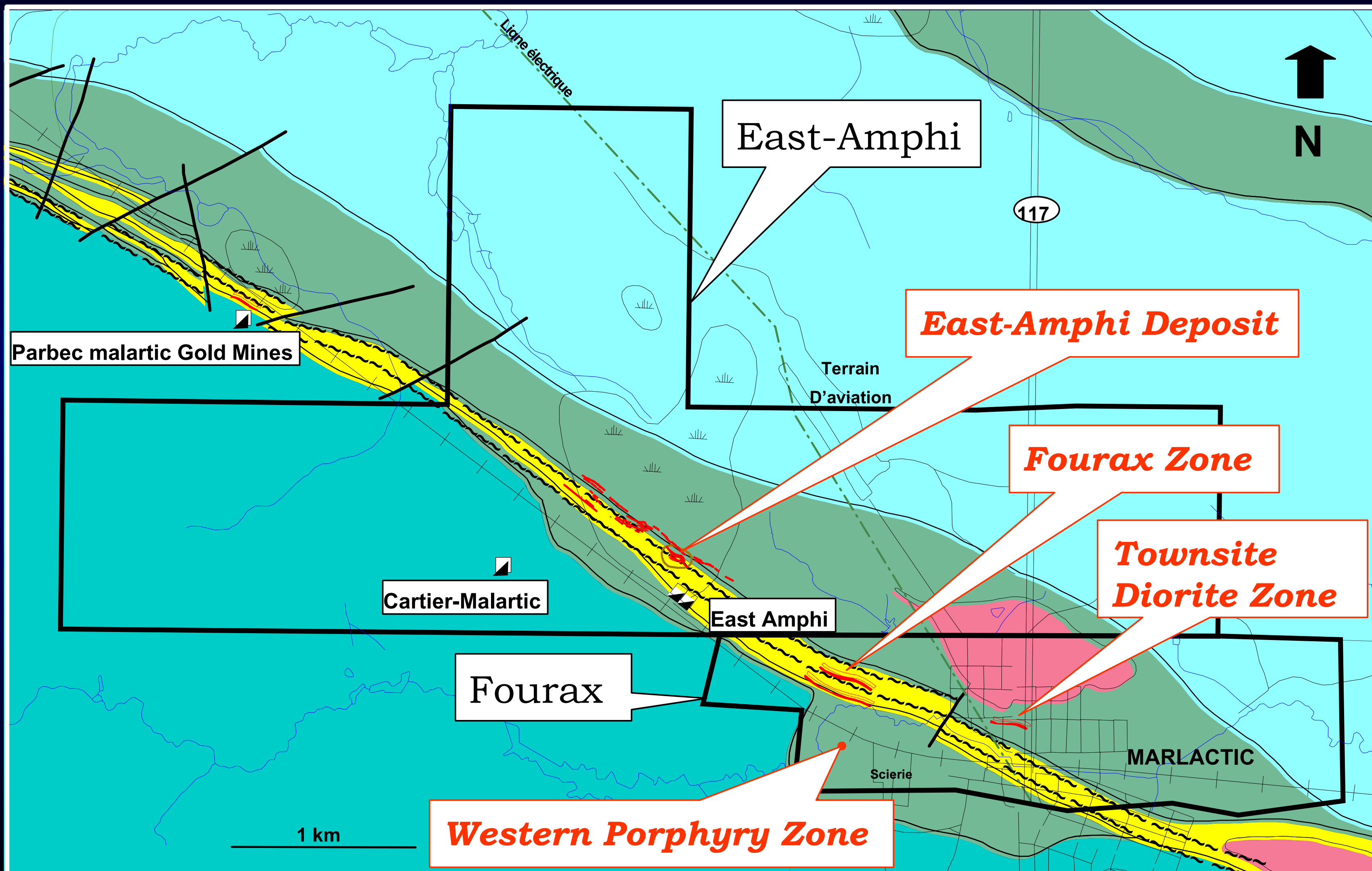


FIGURE 10.1

DETAIL GEOLOGY AND ZONE

Detail Geology and Zone

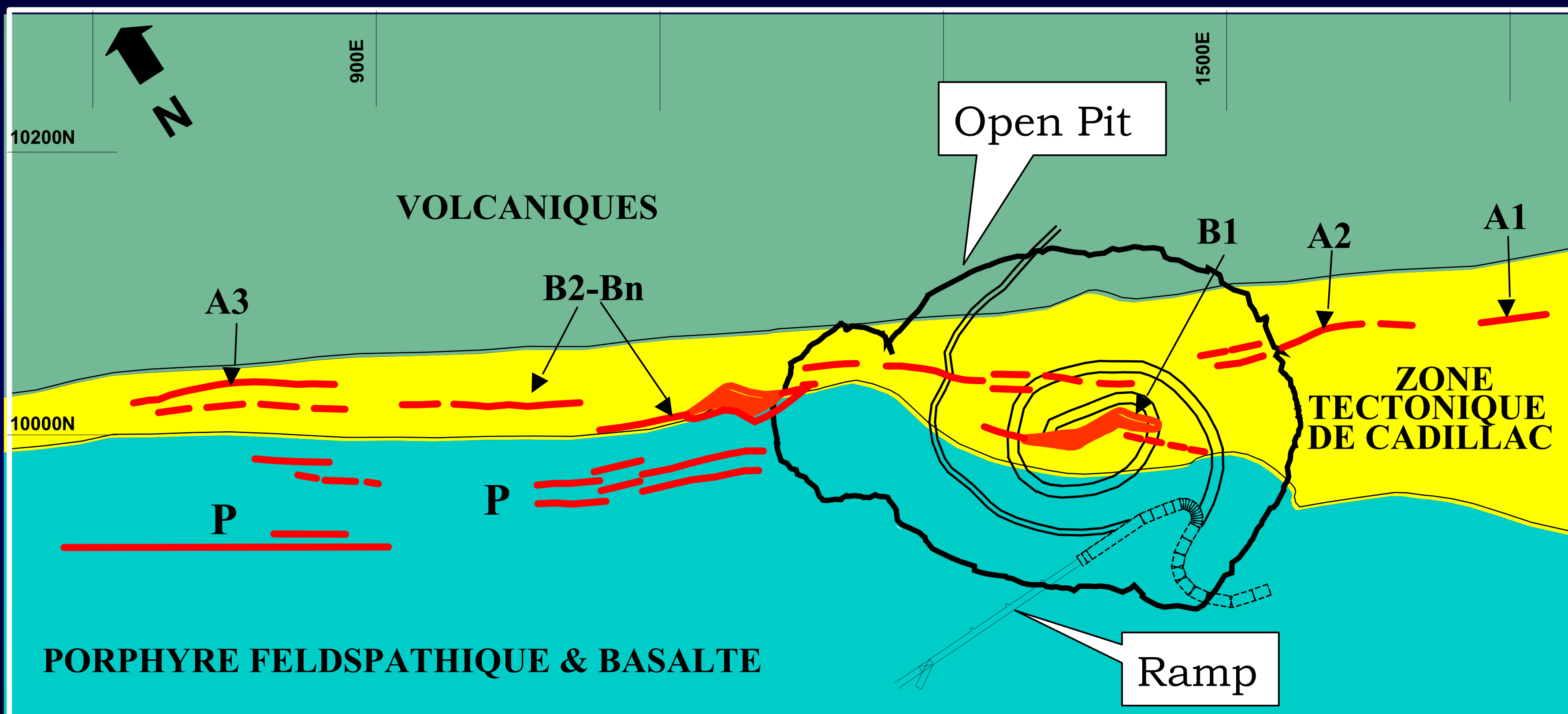
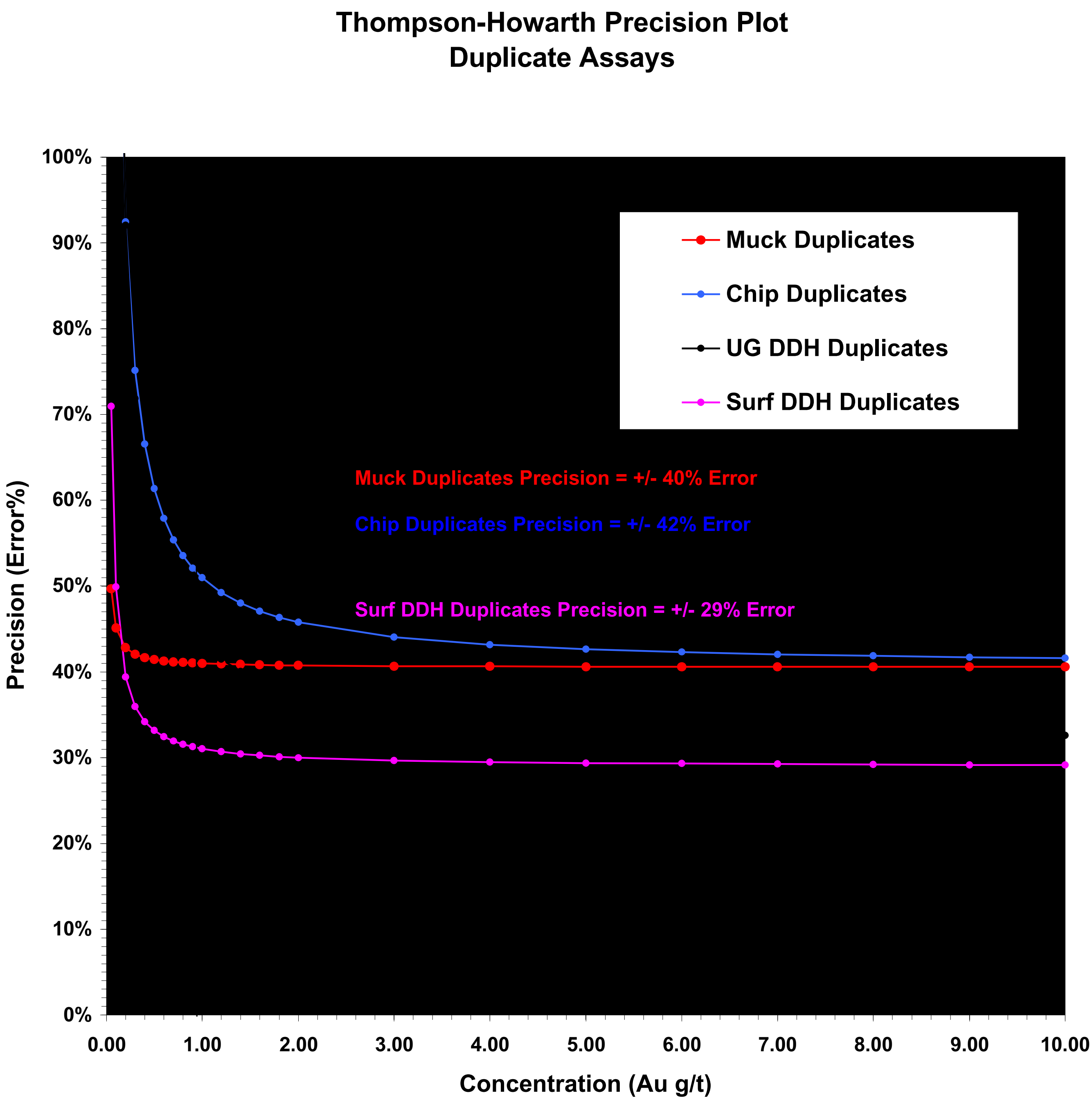


FIGURE 14.1

THOMPSON-HOWARTH PRECISION PLOT
DUPLICATE ASSAYS

FIGURE 14.1

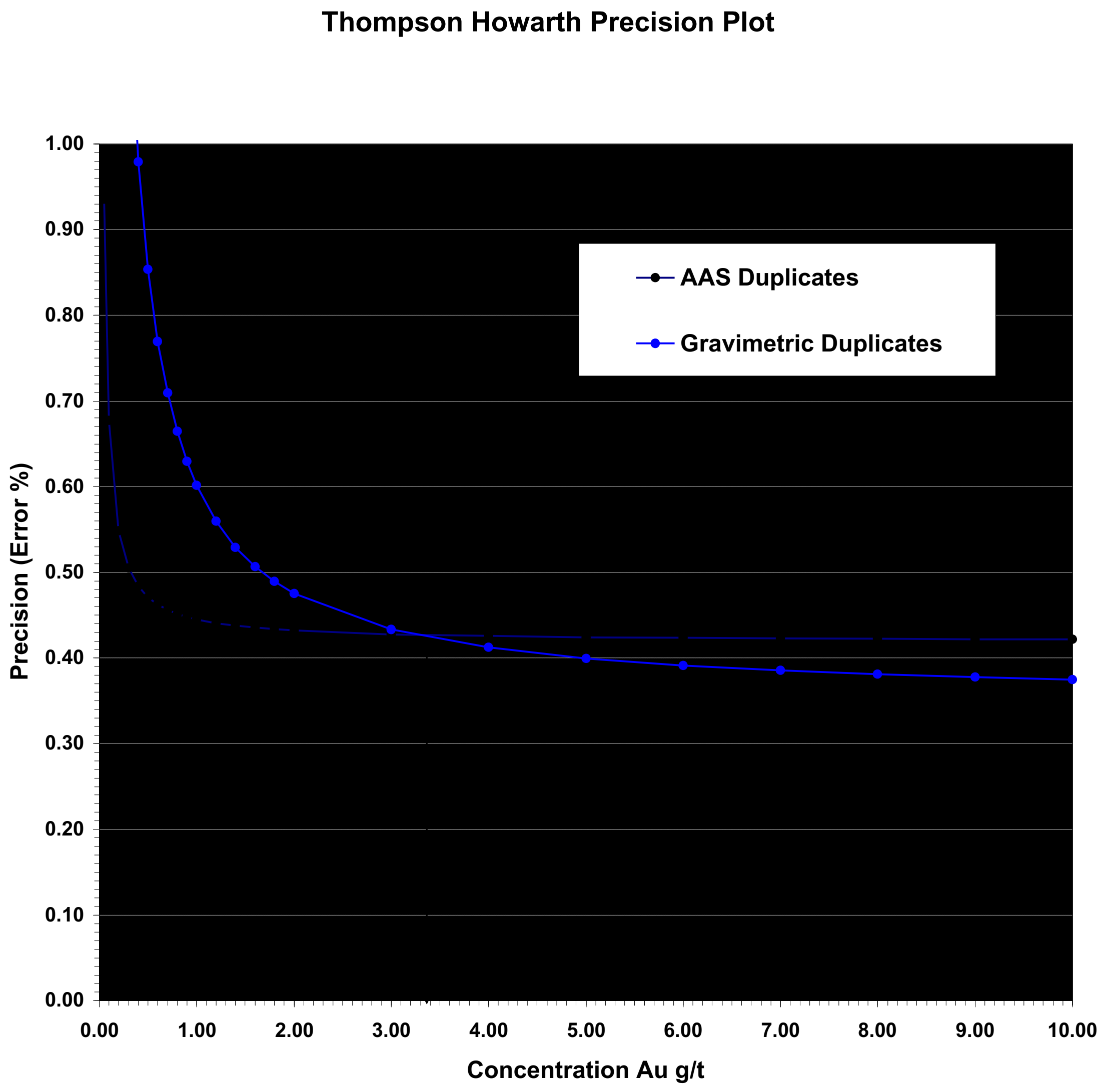


Thompson-Howarth Precision Plot
Duplicate Assays
Muck Duplicates
Chip Duplicates
UG DDH Duplicates
Surf DDH Duplicates
Muck Duplicates Precision = +/- 40% Error
Chip Duplicates Precision = +/- 42% Error
Surf DDH Duplicates Precision = +/- 29% Error
Precision (Error%)
100%
90%
80%
70%
60%
50%
40%
30%
20%
10%
0%
0.00
1.00
2.00
3.00
4.00
5.00
6.00
7.00
8.00
9.00
10.00
Concentration (Au g/t)

FIGURE 14.2

THOMPSON-HOWARTH PRECISION PLOT
AAS & GRAVIMETRIC DUPLICATES

FIGURE 14.2
THOMPSON-HOWARTH PRECISION PLOT AAS & GRAVIMETRIC DUPLICATES



APPENDIX 1

LIST OF AUTHOR PARTICIPATION BY SECTION

List of author participation by section

REPORT SECTION	WRITER	TITLE
Executive Summary	Jules Riopel, M.Sc., P.Geo, MBA	Geology and Exploration Manager
Introduction	Jules Riopel, M.Sc., P.Geo, MBA	Geology and Exploration Manager
Terms of Reference	Jules Riopel, M.Sc., P.Geo, MBA	Geology and Exploration Manager
Disclaimer	Jules Riopel, M.Sc., P.Geo, MBA	Geology and Exploration Manager
Property Description and location	Jules Riopel, M.Sc., P.Geo, MBA Jean-Yves Laliberté, CA Alain Mercier, Eng	Geology and Exploration Manager Vice-President, Finance Manager, Technical Services
Access, climate, local resource and physiography	Jules Riopel, M.Sc., P.Geo, MBA	Geology and Exploration Manager
History	Jules Riopel, M.Sc., P.Geo, MBA Jacques Daigneault, P.Geo	Geology and Exploration Manager Senior Exploration Geologist
Geological Setting -Mineralization	Jules Riopel, M.Sc., P.Geo, MBA Jacques Daigneault, P.Geo	Geology and Exploration Manager Senior Exploration Geologist
Exploration	Jules Riopel, M.Sc., P.Geo, MBA Jacques Daigneault, P.Geo Christian Bézy, Geo	Geology and Exploration Manager Senior Exploration Geologist Chief Geologist, East Amphi
Drilling	Jules Riopel, M.Sc., P.Geo, MBA Jacques Daigneault, P.Geo	Geology and Exploration Manager Senior Exploration Geologist
Sampling Method	Jules Riopel, M.Sc., P.Geo, MBA Jacques Daigneault, P.Geo	Geology and Exploration Manager Senior Exploration Geologist
Sample Preparation, Analyses and Security	Jules Riopel, M.Sc., P.Geo, MBA Jacques Daigneault, P.Geo	Geology and Exploration Manager Senior Exploration Geologist
Database Integrity	Jules Riopel, M.Sc., P.Geo, MBA Jacques Daigneault, P.Geo	Geology and Exploration Manager Senior Exploration Geologist
Adjacent Properties	Jules Riopel, M.Sc., P.Geo, MBA Jacques Daigneault, P.Geo	Geology and Exploration Manager Senior Exploration Geologist
Mineral Resource and Reserve Estimates	Jules Riopel, M.Sc., P.Geo, MBA Jacques Daigneault, P.Geo Christian Bézy, P.Geo Alain Mercier, Eng.	Geology and Exploration Manager Senior Exploration Geologist Chief Geologist, East Amphi Manager, Technical Services
Mining	Alain Mercier, Eng. Richard Lavallée, Eng. Jules Riopel, M.Sc., P.Geo, MBA	Manager, Technical Services Project Manager Geology and Exploration Manager
Mill Test	Jules Riopel, M.Sc., P.Geo, MBA Richard Nolet	Geology and Exploration Manager
Environmental Considerations	Alain Mercier, Eng. Jules Riopel, M.Sc., P.Geo, MBA	Manager, Technical Services Principal Geologist, Exploration
Capital Costs	Alain Mercier, Eng. Richard Lavallée, Eng.	Manager, Technical Services Project Manager
Operating Costs	Alain Mercier, Eng. Richard Lavallée, Eng.	Manager, Technical Services Project Manager
Financial Analysis	Jean-Yves Laliberté, C.A. Nicole Veilleux, CA	Vice-President, Finance Corporate Controler
Interpretation/Conclusion	Jules Riopel, M.Sc., P.Geo, MBA Christian Pichette, Eng	Geology and Exploration Manager Vice-President, Operation
Recommendations	Jules Riopel, M.Sc., P.Geo, MBA Christian Pichette, Eng	Geology and Exploration Manager Vice-President, Operation

APPENDIX 2

LIST OF CLAIMS

List of claims

PROJECT	NTS	TWP	RANGE	LOT	TITLE	CLAIM	DATE REG	DATE EXP	DATE REN	AREA	EXC CRED	FEES	WORK
EAST AMPHI	32D01	Malartic	0001	0032	CDC	0048540	2004-12-14	2006-12-13	2006-10-13	32,56	$0,00	$48,00	$1 200,00
EAST AMPHI	32D01	Malartic	0001	0033	CDC	0048541	2004-12-14	2006-12-13	2006-10-13	32,55	$0,00	$48,00	$1 200,00
EAST AMPHI	32D01	Malartic	0001	0034	CDC	0048542	2004-12-14	2006-12-13	2006-10-13	32,37	$0,00	$48,00	$1 200,00
EAST AMPHI	32D01	Malartic	0001	0035	CDC	0048543	2004-12-14	2006-12-13	2006-10-13	32,22	$0,00	$48,00	$1 200,00
EAST AMPHI	32D01	Malartic	0001	0029	CL	3665043	1977-01-13	2006-12-21	2006-10-21	30,2	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0001	0030	CL	3665044	1977-01-13	2006-12-21	2006-10-21	30,2	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0001	0031	CL	3665053	1977-01-13	2006-12-21	2006-10-21	30,2	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0002	0016	CL	3665201	1977-01-13	2006-12-20	2006-10-20	40	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0002	0017	CL	3665202	1977-01-13	2006-12-20	2006-10-20	40	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0002	0018	CL	3665211	1977-01-13	2006-12-20	2006-10-20	40	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0002	0019	CL	3665212	1977-01-13	2006-12-20	2006-10-20	40	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0002	0020	CL	3665221	1977-01-13	2006-12-20	2006-10-20	40	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0001	0014	CL	3718281	1978-05-25	2007-05-04	2007-03-04	31,2	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0001	0013	CL	3718282	1978-05-25	2007-05-04	2007-03-04	31,2	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0001	0015	CL	3718293	1978-05-25	2007-05-04	2007-03-04	30,8	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0001	0012	CL	5086943	1993-07-29	2007-07-28	2007-05-28	32	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0001	0011	CL	5086944	1993-07-29	2007-07-28	2007-05-28	32	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0001	0010	CL	5086945	1993-07-29	2007-07-28	2007-05-28	32	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0001	0009	CL	5098746	1993-07-29	2007-07-28	2007-05-28	32	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0001	0008	CL	5098747	1993-07-29	2007-07-28	2007-05-28	32	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0001	0017	BM	BM 848	1999-03-24	2019-03-23	2006-03-23	119,08	$0,00	$4 644,12	$0,00
EAST AMPHI	32D01	Malartic	0001	0016	CLD	P139010	1978-06-07	2007-06-06	2007-04-06	30,8	$56 979,78	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0001	0017	CLD	P139020	1978-06-07	2007-12-03	2007-10-03	18,93	$167 094,58	$24,00	$1 000,00
EAST AMPHI	32D01	Malartic	0001	0018	CLD	P139030	1978-06-07	2007-12-03	2007-10-03	13,78	$284 746,46	$24,00	$1 000,00
EAST AMPHI	32D01	Malartic	0001	0019	CLD	P139040	1978-06-07	2007-12-03	2007-10-03	8,61	$1 370 310,73	$24,00	$1 000,00
EAST AMPHI	32D01	Malartic	0001	0020	CLD	P139050	1978-06-07	2007-12-03	2007-10-03	6,6	$1 180 196,28	$24,00	$1 000,00
EAST AMPHI	32D01	Malartic	0001	0021	CLD	P139060	1978-06-07	2007-12-03	2007-10-03	9,37	$963 288,13	$24,00	$1 000,00
EAST AMPHI	32D01	Malartic	0001	0022	CLD	P139070	1978-06-07	2007-12-03	2007-10-03	12,42	$0,00	$24,00	$1 000,00
EAST AMPHI	32D01	Malartic	0001	0023	CLD	P139080	1978-06-07	2007-06-06	2007-04-06	30,8	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0001	0024	CLD	P139090	1978-06-07	2007-06-06	2007-04-06	30,8	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0001	0025	CLD	P139100	1978-06-07	2007-06-06	2007-04-06	30,8	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0001	0026	CLD	P139110	1978-06-07	2007-06-06	2007-04-06	30,8	$52 008,01	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0001	0027	CLD	P139120	1978-06-07	2007-06-06	2007-04-06	30,4	$0,00	$48,00	$2 500,00
EAST AMPHI	32D01	Malartic	0001	0028	CLD	P139130	1978-06-07	2007-06-06	2007-04-06	30,4	$0,00	$48,00	$2 500,00
Total					**34**					**1077,09**	**$4 074 623,97**	**$6 084,12**	**$68 300,00**

List of claims

PROJECT	NTS	TWP	RANGE	LOT	TITLE	CLAIM	DATE REG	DATE EXP	DATE REN	AREA	EXC CRED	FEES	WORK
FOURAX	32D01	Fournière	Bloc 15		CL	3351761	1973-08-03	2007-07-17	2007-05-17	24,8	$0,00	$24,00	$1 000,00
FOURAX	32D01	Fournière	Bloc 16		CL	3351762	1973-08-03	2007-07-17	2007-05-17	21,6	$57 796,66	$24,00	$1 000,00
FOURAX	32D01	Fournière	Bloc 19		CL	3351763	1973-08-03	2007-07-17	2007-05-17	20,4	$0,00	$24,00	$1 000,00
FOURAX	32D01	Fournière	Bloc 20		CL	3351764	1973-08-03	2007-07-17	2007-05-17	22	$0,00	$24,00	$1 000,00
FOURAX	32D01	Fournière	010N	0031	CL	3351771	1973-08-03	2007-07-17	2007-05-17	12,9	$0,00	$24,00	$1 000,00
FOURAX	32D01	Fournière	010N	0032	CL	3351772	1973-08-03	2007-07-17	2007-05-17	20,8	$0,00	$24,00	$1 000,00
FOURAX	32D01	Fournière	010N	0033	CL	3351773	1973-08-03	2007-07-17	2007-05-17	20,9	$0,00	$24,00	$1 000,00
FOURAX	32D01	Fournière	Bloc 27		CL	3351774	1973-08-03	2007-07-17	2007-05-17	33,3	$118 005,79	$48,00	$2 500,00
FOURAX	32D01	Fournière	010N	0034	CL	3351781	1973-08-03	2007-07-17	2007-05-17	20	$138 346,10	$24,00	$1 000,00
FOURAX	32D01	Fournière	010N	0035	CL	3351782	1973-08-03	2007-07-17	2007-05-17	12,5	$0,00	$24,00	$1 000,00
FOURAX	32D01	Fournière	010S	0031	CL	3351783	1973-08-03	2007-07-17	2007-05-17	4,3	$29 372,96	$24,00	$1 000,00
FOURAX	32D01	Fournière	Bloc 26		CL	3351784	1973-08-03	2007-07-17	2007-05-17	56,8	$1 573 609,90	$48,00	$2 500,00
MALARTIC EXTENSION	32D01	Malartic	0002	0027	CDC	1106043	2002-12-03	2006-12-02	2006-10-02	41,67	$0,00	$48,00	$1 200,00
MALARTIC EXTENSION	32D01	Malartic	0002	0026	CL	5114367	1993-09-11	2007-09-10	2007-07-11	40	$0,00	$48,00	$2 500,00
MALARTIC EXTENSION	32D01	Malartic	0002	0025	CL	5114368	1993-09-11	2007-09-10	2007-07-11	40	$0,00	$48,00	$2 500,00
MALARTIC EXTENSION	32D01	Malartic	0002	0024	CL	5114369	1993-09-11	2007-09-10	2007-07-11	40	$0,00	$48,00	$2 500,00
MALARTIC EXTENSION	32D01	Malartic	0002	0023	CL	5114370	1993-09-11	2007-09-10	2007-07-11	40	$0,00	$48,00	$2 500,00
MALARTIC EXTENSION	32D01	Malartic	0002	0022	CL	5114371	1993-09-11	2007-09-10	2007-07-11	40	$0,00	$48,00	$2 500,00
MALARTIC EXTENSION	32D01	Malartic	0002	0021	CL	5114372	1993-09-11	2007-09-10	2007-07-11	40	$0,00	$48,00	$2 500,00
MALARTIC EXTENSION	32D01	Malartic	0002	0032	CL	5114373	1993-09-11	2007-09-10	2007-07-11	40,5	$0,00	$48,00	$2 500,00
MALARTIC EXTENSION	32D01	Malartic	0002	0031	CL	5114374	1993-09-11	2007-09-10	2007-07-11	40,5	$0,00	$48,00	$2 500,00
MALARTIC EXTENSION	32D01	Malartic	0002	0030	CL	5114375	1993-09-11	2007-09-10	2007-07-11	40,5	$0,00	$48,00	$2 500,00
MALARTIC EXTENSION	32D01	Malartic	0002	0029	CL	5114376	1993-09-11	2007-09-10	2007-07-11	40	$0,00	$48,00	$2 500,00
RADIUM-NORD	32D01	Fournière	0010	0024	CL	3263002	1972-08-21	2007-08-01	2007-06-01	33,2	$0,00	$48,00	$2 500,00
RADIUM-NORD	32D01	Fournière	0010	0022	CL	3263011	1972-08-21	2007-08-01	2007-06-01	38	$45 883,27	$48,00	$2 500,00
RADIUM-NORD	32D01	Fournière	0010	0021	CL	3263012	1972-08-21	2007-08-01	2007-06-01	46,8	$49 248,59	$48,00	$2 500,00
RADIUM-NORD	32D01	Fournière	0010	0020	CL	3263051	1972-08-21	2007-08-02	2007-06-02	48,4	$5 058,74	$48,00	$2 500,00
RADIUM-NORD	32D01	Fournière	0010	0023	CL	3263351	1972-08-28	2007-08-09	2007-06-09	34	$762,28	$48,00	$2 500,00
RADIUM-NORD	32D01	Fournière	0010	0019	CL	3490151	1974-09-25	2007-09-08	2007-07-09	48,4	$9 696,33	$48,00	$2 500,00
RADIUM-NORD	32D01	Fournière	0010	0018	CL	3490181	1974-09-25	2007-09-01	2007-07-02	48,4	$12 537,90	$48,00	$2 500,00
RESERVOIR	32D01	Fournière	0010	0017	CL	3887321	1980-07-22	2007-06-19	2007-04-19	48	$0,00	$48,00	$2 500,00
RESERVOIR	32D01	Fournière	0010	0015	CL	3887331	1980-07-22	2007-06-19	2007-04-19	48	$0,00	$48,00	$2 500,00
RESERVOIR	32D01	Fournière	0010	0013	CL	3924261	1980-07-22	2007-07-04	2007-05-04	48	$0,00	$48,00	$2 500,00
RESERVOIR	32D01	Fournière	0010	0014	CL	3924271	1980-07-22	2007-07-04	2007-05-04	48	$0,00	$48,00	$2 500,00
RESERVOIR	32D01	Fournière	0010	0016	CL	3924281	1980-07-22	2007-07-04	2007-05-04	48	$0,00	$48,00	$2 500,00

List of claims

PROJECT	NTS	TWP	RANGE	LOT	TITLE	CLAIM	DATE REG	DATE EXP	DATE REN	AREA	EXC CRED	FEES	WORK
WEST AMPHI	32D01	Fournière	0010	0001	CDC	1106031	2002-12-03	2006-12-02	2006-10-02	37,9	$0,00	$48,00	$1 200,00
WEST AMPHI	32D01	Fournière	0010	0002	CDC	1106032	2002-12-03	2006-12-02	2006-10-02	51,4	$0,00	$48,00	$1 200,00
WEST AMPHI	32D01	Fournière	0010	0003	CDC	1106033	2002-12-03	2006-12-02	2006-10-02	51,39	$0,00	$48,00	$1 200,00
WEST AMPHI	32D01	Fournière	0010	0004	CDC	1106034	2002-12-03	2006-12-02	2006-10-02	51,37	$0,00	$48,00	$1 200,00
WEST AMPHI	32D01	Fournière	0010	0005	CDC	1106035	2002-12-03	2006-12-02	2006-10-02	51,36	$0,00	$48,00	$1 200,00
WEST AMPHI	32D01	Fournière	0010	0006	CDC	1106036	2002-12-03	2006-12-02	2006-10-02	51,39	$0,00	$48,00	$1 200,00
WEST AMPHI	32D01	Fournière	0010	0007	CDC	1106037	2002-12-03	2006-12-02	2006-10-02	51,4	$0,00	$48,00	$1 200,00
WEST AMPHI	32D01	Fournière	0010	0008	CDC	1106038	2002-12-03	2006-12-02	2006-10-02	51,42	$0,00	$48,00	$1 200,00
WEST AMPHI	32D01	Fournière	0010	0009	CDC	1106039	2002-12-03	2006-12-02	2006-10-02	51,41	$0,00	$48,00	$1 200,00
WEST AMPHI	32D01	Malartic	0001	0001	CDC	1106040	2002-12-03	2006-12-02	2006-10-02	49,72	$0,00	$48,00	$1 200,00
WEST AMPHI	32D01	Malartic	0001	0002	CDC	1106041	2002-12-03	2006-12-02	2006-10-02	33,26	$0,00	$48,00	$1 200,00
WEST AMPHI	32D01	Malartic	0001	0003	CDC	1106042	2002-12-03	2006-12-02	2006-10-02	33,25	$0,00	$48,00	$1 200,00
WEST AMPHI	32D01	Malartic	0001	0004	CL	5162706	1996-09-14	2006-09-13	2006-07-14	33	$0,00	$48,00	$1 800,00
WEST AMPHI	32D01	Malartic	0001	0005	CL	5162707	1996-09-14	2006-09-13	2006-07-14	33	$0,00	$48,00	$1 800,00
WEST AMPHI	32D01	Malartic	0001	0006	CL	5162708	1996-09-14	2006-09-13	2006-07-14	33	$0,00	$48,00	$1 800,00
WEST AMPHI	32D01	Malartic	0001	0007	CL	5162709	1996-09-14	2006-09-13	2006-07-14	33	$0,00	$48,00	$1 800,00
WEST AMPHI	32D01	Fournière	0010	0010	CL	5182646	1996-11-02	2006-11-01	2006-09-01	54	$0,00	$48,00	$1 800,00
WEST AMPHI	32D01	Fournière	0010	0011	CL	5182647	1996-11-02	2006-11-01	2006-09-01	54	$0,00	$48,00	$1 800,00
WEST AMPHI	32D01	Fournière	0010	0012	CL	5182648	1996-11-02	2006-11-01	2006-09-01	54	$0,00	$48,00	$1 800,00

APPENDIX 3

SUMMARY OF RESOURCE ESTIMATE

Summary of Resource Estimate
Grade Contour Model
(Cut off 2.725 g/t and ≥ 2.0 m width)

DESCRIPTION	TONNES	GRADE G/T	THICKNESS (M)
Measured			
Zone A	262,506	5.58	4.0
Zone B	223,530	5.77	5.6
Total Measured	**486,035**	**5.67**	**4.6**
Indicated			
Zone A	279,899	3.71	5.4
Zone B	932,742	4.37	7.1
Total Indicated	**1,212,641**	**4.22**	**6.6**
Total Indicated and Measured	**1,698,676**	**4.63**	**5.9**
Inferred			
Zone A	755,947	3.42	5.7
Zone B	483,571	3.99	3.5
Total Inferred	**1,239,518**	**3.64**	**4.6**

Summary of Resource Estimate
Grade Thickness Contour Model
(Cut off 2.725 g/t and ≥ 2.0 m width)

DESCRIPTION	TONNES	GRADE G/T	THICKNESS (M)
Measured			
Zone A	139,095	5.82	3.3
Zone B	210,932	5.57	4.9
Total Measured	**380,027**	**5.69**	**4.0**
Indicated			
Zone A	90,362	5.41	2.6
Zone B	753,967	5.10	5.0
Total Indicated	**844,329**	**5.14**	**4.6**
Total Indicated and Measured	**1,224,356**	**5.31**	**4.4**
Inferred			
Zone A	371,322	4.85	2.9
Zone B	427,339	6.77	2.4
Total Inferred	**798,661**	**5.88**	**2.6**

Summary of Resource Estimate
Kriged Grade Thickness Contour Model
(Cut off 2.725 g/t and ≥ 2.0 m width)

DESCRIPTION	TONNES	GRADE G/T	THICKNESS M
Measured			
Zone A	197,419	4.74	3.3
Zone B	255,691	4.69	5.2
Total Measured	**453,109**	**4.71**	**4.2**
Indicated			
Zone A	121,964	5.33	4.3
Zone B	1,099,409	4.57	6.5
Total Indicated	**1,221,373**	**4.65**	**6.2**
Total Indicated and Measured	**1,674,482**	**4.67**	**5.5**
Inferred			
Zone A	419,701	4.35	4.5
Zone B	303,873	4.35	3.9
Total Inferred	**723,574**	**4.35**	**4.2**

Comparison of Resource Estimates

	MEASURED		INDICATED		INFERRED	
Description	**Tonnes**	**Grade g/t**	**Tonnes**	**Grade g/t**	**Tonnes**	**Grade g/t**
Grade Contour Model	486,035	5.67	1,212,641	4.22	1,239,518	3.64
GT Contour Model	380,027	5.69	844,329	5.14	798,661	5.88
Kriged GT Model	453,109	4.71	1,221,373	4.65	723,574	4.35

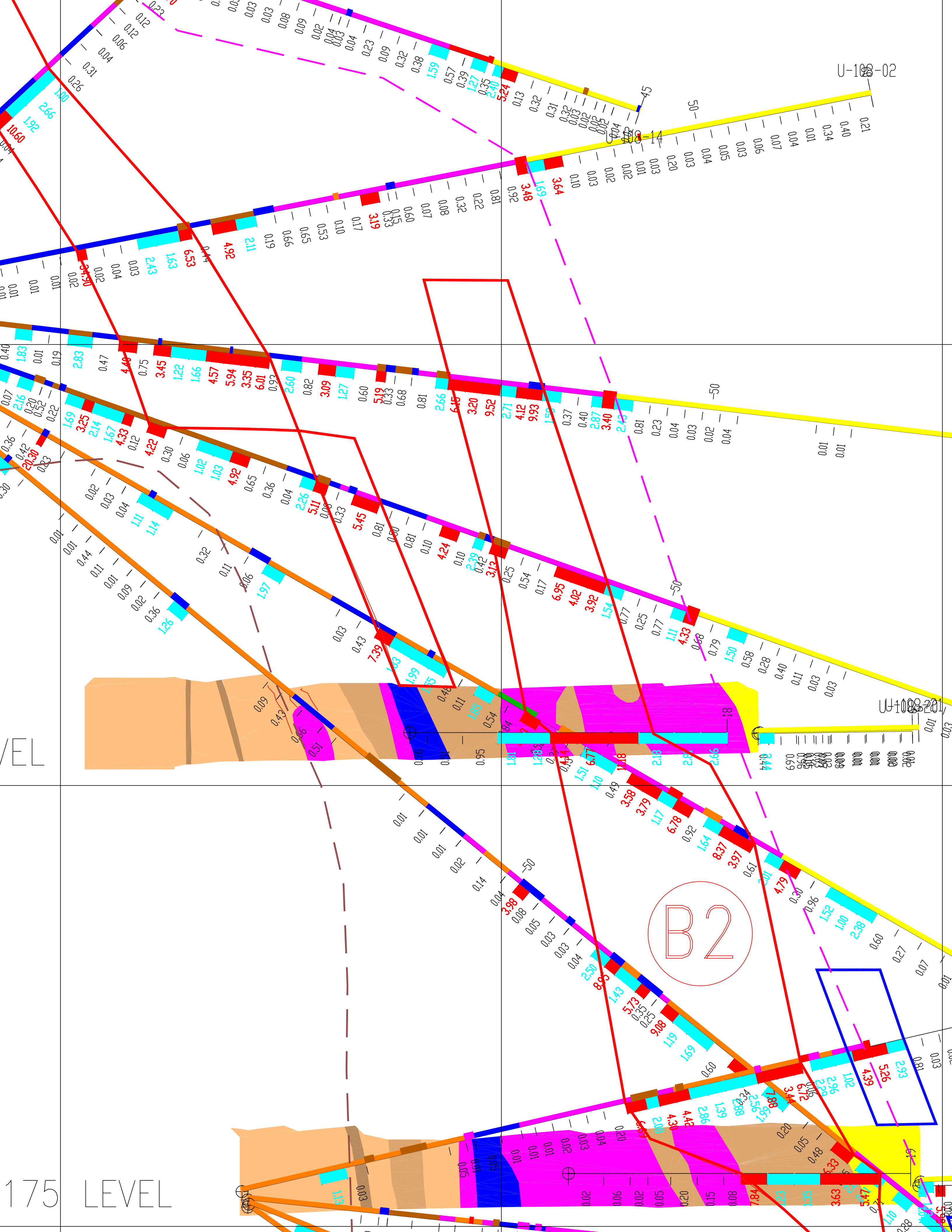
Summary of Reserve Estimate
(Cut off 2.725 g/t and ≥ 2.0 m width)
(estimated by SNC-Lavalin)

DESCRIPTION	RECOVERABLE TONNES	DILUTED GRADE G/T	RECOVERABLE OUNCES*
Proven Reserves			
Zone A	129,289	4.57	18,035
Zone B1	39,505	5.00	6,032
Zone B2	0	0	0
Total Proven Reserves	**168,793**	**4.67**	**24,067**
Probable Reserves			
Zone A	259,691	4.34	34,419
Zone B1	47,097	4.20	6,042
Zone B2	893,166	4.01	109,339
Total Probable Reserves	**1,199,954**	**4.09**	**149,800**
Total Reserves	**1,368,747**	**4.16**	**173,867**

* Gold recovery based on 95% mill recovery

APPENDIX 4

SECTION AND PLAN VIEW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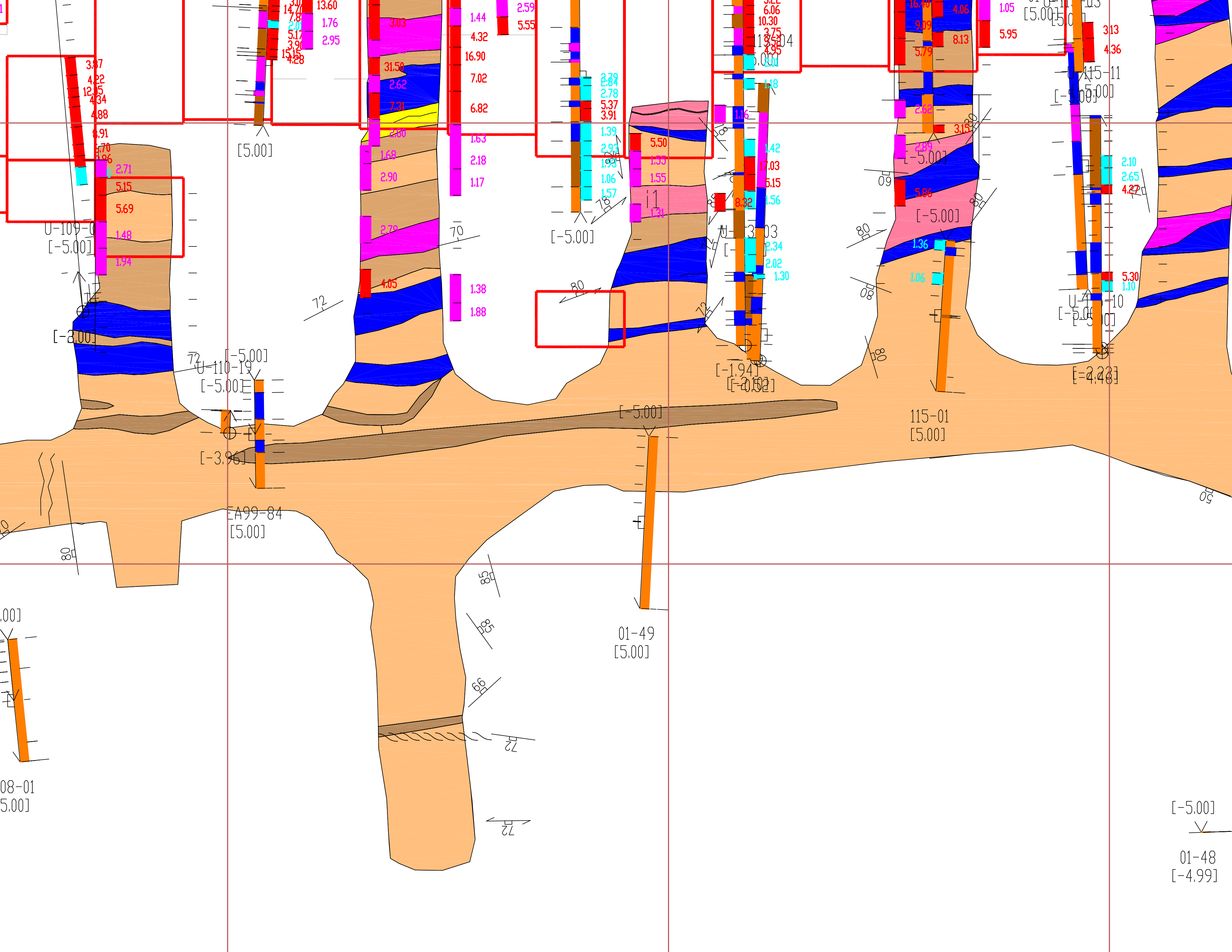
U-108-14
U-108-201
B2
175 LEVEL



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01-49
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01-48
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08-01
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APPENDIX 5

DRILLING SUMMARY

Drilling Summary

2004 - 2005 DRILLING SUMMARY							
SURFACE							
Core Size	**Hole**	**Length**	**MineGrid East**	**MineGrid North**	**MineGrid Elev**	**Target**	**Sample number**
NQ	100-01	135,00	999,89	10026,56	4998,71	A3,Bn,B2	48
NQ	100-02	222,00	999,9	10140,03	4998,82	A3,Bn,B2	55
NQ	101-01	342,00	1015,15	10235,45	4999,564	A3,Bn,B2	102
NQ	101-02	430,00	1015	10236	5000	A3,Bn,B2	134
NQ	102-01	139,00	1024,47	10010,78	4998,8	Bn,B2	59
NQ	104-01	226,00	1040	10112	4998,6	Bn,B2,P	108
NQ	105-01	150,00	1050,81	10066,28	4998,41	B2	59
NQ	107-01	300,00	1074,41	10185,21	4999,02	A3,Bn,B2	82
NQ	107-02	270,00	1070,06	10145,67	4998,81	B2,P	161
NQ	108-01	231,00	1079,87	10093,18	4998,69	Bn,B2	102
NQ	110-01	380,00	1105,5	10239,2	4999,7	Bn,B2	122
NQ	110-02	468,00	1105	10260	5000	A3,Bn,B2	191
NQ	111-01	373,50	1109,4	10160,22	4998,7	Bn,B2	201
NQ	114-01	414,00	1140,26	10229,22	5000,05	Bn,B2	182
NQ	115-01	237,00	1148,86	10139,13	4998,98	Bn,B2	87
NQ	117-01	285,00	1169,86	10126,83	4998,85	Bn,B2	137
NQ	121-01	179,70	1209,39	10134,26	4998,68	Bn	133
NQ	123-01	267,00	1230,25	10155,36	4998,88	Bn	98
NQ	125-01	165,00	1250,07	10120,88	4998,69	Bn	46
NQ	127-01	201,00	1269,49	10130,13	4998,48	Bn	50
NQ	127-02	243,00	1269,52	10130,21	4998,52	Bn	75
NQ	148-01	268,50	1480	10165	5000	A2	62
NQ	152-01	250,00	1521,645	10183,265	4999,006	A2,A2'	87
NQ	152-02	294,00	1521,645	10183,265	4999,006	A2,A2'	144
NQ	159-01	310,00	1595,236	10179,534	4998,711	A2	95
NQ	159-02	270,00	1595,236	10179,534	4998,711	A2	132
NQ	162-01	226,00	1624,917	10160,862	4998,654	A2	99
NQ	162-02	235,00	1624,917	10160,862	4998,654	A1	107
NQ	162-03	301,00	1622,366	10190,33	4998,809	A2	115
NQ	166-01	218,90	1664,753	10161,105	4998,554	A2	88
NQ	168-01	192,00	1679,43	10153,31	4998,45	A1	49
NQ	173-01	152,50	1729,69	10171,29	4998,17	A1	64
NQ	173-02	312,00	1730	10116	4998,2	A1	202
NQ	66-01	235,00	659,71	10061,83	4997,6	Bn,B2,P	91
NQ	68-01	325,00	685,03	10113,73	4997,65	A3,Bn,B2,P	162
NQ	74-01	321,00	740	10099,03	4998,5	A3,Bn,B2	206
NQ	77-01	215,00	769,83	10142,73	4998,47	A3,Bn,B2	74
NQ	78-02	300,00	780	10062	5000	A3,Bn,B2	153
NQ	80-01	340,00	800,11	10086,71	4998,23	A3,Bn,B2	164
NQ	81-01	270,00	809,69	10154,44	4998,7	A3,Bn,B2	56
NQ	83-01	326,60	830,16	10142,84	4998,67	A3,Bn,B2	133
NQ	84-01	212,00	839,93	10052,63	4998,5	A3,Bn,B2	116
NQ	84-02	175,00	839,79	10006,25	4998,2	Bn,B2	67
NQ	85-01	303,00	849,86	10177,11	4998,83	A3,Bn,B2	64
NQ	88-01	261,00	885,08	10161,98	4998,79	A3,Bn,B2	93
NQ	88-02	392,00	886,12	10130,84	4998,79	A3,Bn,B2	305
NQ	90-01	471,00	899,9	10238,17	4999	A3,Bn,B2	149
NQ	90-01A	198,00	900	10240	4999	A3,Bn,B2	89
NQ	92-01	159,00	919,79	10047,88	4998,61	Bn,B2	57
NQ	92-02	273,00	919,56	10139,95	4998,71	A3,Bn,B2	124
NQ	92-03	330,00	920	10167,65	4998,83	A3,Bn,B2	185
NQ	94-01	315,00	939,52	10219,62	4999,55	A3,Bn,B2	101
NQ	96-01	372,00	960	10219	5000	A3,Bn,B2	118
NQ	97-01	249,00	971,69	10128,23	4998,74	A3,Bn,B2	96
NQ	97-02	156,00	974,58	10095,52	4998,72	Bn,B2	70
NQ	97-03	200,00	975,07	10102,67	4998,71	A3,Bn,B2,P	100
SUBTOTAL	**56**	**15086,70**					**6249**

Drilling Summary

UNDERGROUND							
Core Size	Hole	Length	MineGrid East	MineGrid North	MineGrid Elev	Target	Sample number
BQ	U-100-01	120,00	1000,805	9954,997	4804,394	A3,Bn,P	74
BQ	U-100-02	45,00	1000,174	10016,217	4803,283	Bn	44
BQ	U-101-01	49,00	1010,858	10016,148	4804,538	Bn	61
BQ	U-101-02	36,00	1010,733	10001,009	4854,655	Bn	43
BQ	U-101-03	43,50	1010,77	10005,478	4829,238	P	54
BQ	U-102-01	40,00	1020,457	9974,839	4903,64	B2	41
BQ	U-102-02	55,15	1020,476	9974,825	4903,386	B2	32
BQ	U-102-03	65,10	1020,445	9974,754	4903,247	B2	54
BQ	U-102-04	28,50	1020,449	10029,678	4805,118	Bn,B2	32
BQ	U-102-05	55,00	1020,514	10029,755	4802,877	Bn,B2	64
BQ	U-102-06	45,20	1020,435	9987,334	4878,828	B2	59
BQ	U-102-07	36,00	1020,933	10002,3	4855,079	B2	48
BQ	U-102-08	47,00	1020,511	10006,347	4828,034	Bn,B2	62
BQ	U-102-09	45,00	1020,569	10006,26	4830,305	B2	56
BQ	U-103-01	100,00	1030,008	9963,174	4806,508	Bn,B2,P	54
BQ	U-103-02	110,30	1030,029	9963,041	4806,328	Bn,P	91
BQ	U-103-03	201,30	1030,173	9962,926	4805,714	Bn,P	136
BQ	U-103-04	43,50	1030,504	9987,331	4878,778	Bn,B2	61
BQ	U-103-05	60,00	1030,509	9987,536	4878,034	Bn	48
BQ	U-103-06	45,00	1030,722	10006,981	4828,383	B2	62
BQ	U-103-07	133,00	1029,979	9963,084	4806,105	Bn,B2,P	101
BQ	U-103-08	155,80	1029,978	9963,011	4805,91	Bn,B2,P	75
BQ	U-103-09	234,40	1030,027	9962,897	4805,659	Bn,B2,P	183
BQ	U-103-10	51,10	1030,249	9975,03	4903,008	Bn	31
BQ	U-103-11	55,50	1030,897	10007,112	4827,06	Bn,B2,P	52
BQ	U-103-12	49,50	1030,899	10007,112	4827,244	Bn,B2	44
BQ	U-103-13	50,00	1030,936	10006,966	4828,375	B2	31
BQ	U-103-14	55,50	1030,947	10007,005	4829,86	Bn	42
BQ	U-104-01	40,00	1040,1	9986,896	4880,887	B2	40
BQ	U-104-02	36,50	1040,113	9987,139	4879,866	B2	42
BQ	U-104-03	45,60	1040,037	9987,137	4878,662	Bn,B2	45
BQ	U-104-04	45,00	1039,978	9987,267	4878,086	B2	39
BQ	U-104-05	56,50	1040,415	10007,361	4829,884	Bn,B2	46
BQ	U-104-06	45,00	1040,452	10007,492	4827,467	Bn,B2	36
BQ	U-104-07	50,50	1040,453	10007,569	4826,962	Bn,B2,P	40
BQ	U-104-08	35,60	1040,316	9974,964	4903,241	Bn,B2	25
BQ	U-104-09	42,00	1040,152	10016,073	4802,74	B2	44
BQ	U-104-10	49,00	1040,16	10016,073	4802,471	Bn,B2	33
BQ	U-104-11	65,50	1040	10016	4802	Bn	41
BQ	U-104-12	42,00	1039,456	10003,201	4854,384	Bn	52
BQ	U-104-13	45,00	1039,549	10007,529	4827,523	Bn,B2,P	50
BQ	U-104-14	45,00	1039,576	10007,532	4829,076	B2	63
BQ	U-105-01	35,00	1049,841	9975,238	4903,569	B2	37
BQ	U-105-02	55,00	1049,829	9975,238	4902,841	Bn,B2	45
BQ	U-105-03	46,00	1049,84	10008,44	4828,744	Bn,B2	34
BQ	U-105-04	29,00	1050,005	10033,228	4804,153	B2	27
BQ	U-105-05	35,50	1049,98	10033,013	4802,566	B2	31
BQ	U-105-06	48,40	1049,747	10008,466	4826,895	Bn,B2	37
BQ	U-105-07	44,10	1049,781	10008,401	4827,568	Bn,B2	34
BQ	U-105-08	54,70	1050,672	9987,094	4877,472	Bn,B2	47
BQ	U-105-09	43,70	1050,744	9986,954	4878,048	Bn,B2	43
BQ	U-105-10	44,00	1050,558	9975,011	4903,22	Bn	47
ATW	U-105-11	8,80	1048,728	10052,518	4803,522	Bn,B2	10
ATW	U-105-12	9,20	1050,889	10052,136	4803,721	Bn,B2	12
BQ	U-105-13	42,00	1054,159	10004,642	4854,985	Bn,B2	50
ATW	U-105-14	8,80	1048,559	10033,727	4853,548	Bn	10
ATW	U-105-15	8,50	1050,983	10033,584	4853,432	Bn	11
BQ	U-106-01	42,50	1060,031	9986,819	4879,614	B2	37
BQ	U-106-02	40,00	1059,979	9987,055	4878,245	Bn,B2	40
BQ	U-106-03	50,00	1059,996	9987,14	4877,492	Bn,B2	46
BQ	U-106-04	65,20	1059,965	9987,22	4877,228	B2	62
BQ	U-106-05	105,00	1059,778	9964,986	4810,91	Bn,B2	74
BQ	U-106-06	137,30	1059,802	9964,929	4810,725	B2,P	65

Drilling Summary

Core Size	Hole	Length	MineGrid East	MineGrid North	MineGrid Elev	Target	Sample number
BQ	U-106-07	51,50	1060,118	10009,113	4828,4974	B2	39
BQ	U-106-08	50,00	1060,105	10009,361	4827,521	Bn,B2	39
BQ	U-106-09	55,00	1060,023	10009,258	4826,745	Bn,B2	47
BQ	U-106-10	60,10	1060,03	10009,302	4826,548	A3,Bn,B2	50
BQ	U-106-11	40,50	1059,387	10004,943	4855,022	B2	40
BQ	U-106-12	42,00	1059,347	10005,18	4853,603	B2	52
BQ	U-106-13	240,00	1059,909	9964,548	4810,077	Bn,B2,P	155
BQ	U-106-14	281,30	1060,168	9964,471	4810,133	Bn,P	195
BQ	U-106-15	39,10	1060,116	9975,476	4902,796	Bn	39
BQ	U-106-16	138,70	1060,133	9964,684	4810,301	Bn,P	89
BQ	U-106-17	45,00	1059,319	10005,33	4852,776	Bn,B2	60
BQ	U-106-18	81,00	1064,863	10032,087	4802,176	Bn	54
BQ	U-106-19	40,00	1065,614	10032,44	4802,551	B2	43
BQ	U-106-20	55,00	1065,562	10032,297	4802,319	Bn,B2	55
BQ	U-107-01	44,50	1070,231	9987,501	4878,898	Bn,B2	44
BQ	U-107-02	45,40	1070,223	9987,347	4877,72	Bn,B2	47
BQ	U-107-03	60,00	1070,219	9987,058	4877,048	Bn,B2	58
BQ	U-107-04	75,00	1070,221	9987,091	4876,716	B2	69
BQ	U-107-05	50,50	1069,994	10009,731	4826,766	A3,B2	39
BQ	U-107-06	50,00	1069,943	10009,706	4826,454	Bn,B2	34
BQ	U-107-07	75,50	1069,942	10009,726	4826,234	Bn,B2	55
BQ	U-107-08	39,45	1070,36	10009,668	4827,607	Bn,B2	28
BQ	U-107-09	39,00	1070,38	10005,428	4855,099	Bn,B2	48
BQ	U-107-10	39,00	1070,339	10005,4	4853,566	Bn,B2	51
BQ	U-108-01	50,50	1079,893	9986,813	4878,978	B2	48
BQ	U-108-02	60,00	1079,871	9987,069	4877,633	B2	61
BQ	U-108-03	70,00	1079,889	9987,279	4877,138	B2	57
BQ	U-108-04	91,00	1080,165	9987,315	4876,57	B2	70
BQ	U-108-05	115,00	1080,161	9987,257	4876,443	Bn,B2	82
BQ	U-108-06	81,00	1079,932	9987,387	4876,884	B2	68
BQ	U-108-07	50,00	1080,179	10010,62	4826,505	Bn,B2	43
BQ	U-108-08	61,40	1080,181	10010,687	4826,265	B2	56
BQ	U-108-09	69,50	1080,19	10010,623	4826,1	Bn,B2	61
BQ	U-108-10	31,50	1079,974	9974,794	4902,34	B2	31
BQ	U-108-11	43,30	1080,511	10010,296	4826,976	Bn,B2	35
BQ	U-108-12	60,50	1083,88	10016,332	4801,917	Bn	63
BQ	U-108-13	75,00	1083,884	10016,292	4801,713	Bn	55
BQ	U-108-14	45,25	1080,539	9989,872	4902,436	B2	56
BQ	U-108-15	30,00	1080	9990	4903	B2	34
ATW	U-108-16	8,70	1081,211	10048,667	4827,303	Bn	12
ATW	U-108-17	8,00	1078,784	10048,775	4827,521	Bn	8
ATW	U-108-18	8,70	1078,833	10051,203	4803,372	Bn	10
ATW	U-108-19	9,40	1080,811	10051,225	4803,418	Bn	11
ATW	U-108-20	8,80	1078,832	10039,552	4852,942	Bn	9
ATW	U-108-21	9,10	1080,509	10039,639	4852,989	Bn	10
BQ	U-109-01	36,00	1092,497	10026,562	4802,801	B2	46
BQ	U-109-02	36,00	1092,501	10026,488	4804,104	B2	46
BQ	U-109-03	30,80	1093,751	10023,396	4826,888	B2	39
BQ	U-109-04	33,20	1093,658	10023,282	4828,333	Bn,B2	43
BQ	U-109-05	30,00	1092,052	10015,013	4852,504	B2	33
BQ	U-109-06	34,00	1092,232	10014,791	4854,091	B2	44
BQ	U-109-07	45,80	1090,561	9988,365	4877,76	B2	54
BQ	U-110-01	54,00	1099,936	9987,589	4878,771	B2	49
BQ	U-110-02	50,50	1099,875	9987,644	4877,941	B2	45
BQ	U-110-03	55,00	1099,81	9987,778	4877,256	Bn,B2	46
BQ	U-110-04	56,00	1099,979	9987,744	4876,576	B2	46
BQ	U-110-05	31,00	1100,147	9982,789	4876,609	P	21
BQ	U-110-06	16,00	1096,357	10025,525	4877,545	B2	12
BQ	U-110-07	95,40	1100,18	9967,971	4816,891	Bn,B2	80
BQ	U-110-08	110,80	1100,158	9967,751	4816,728	Bn	68
BQ	U-110-09	159,50	1100,156	9967,866	4816,328	Bn,B2	125
BQ	U-110-10	150,00	1100,235	9967,9	4816,265	Bn	107
BQ	U-110-11	40,00	1098,3453	10014,383	4827,991	B2	56
BQ	U-110-12	45,10	1101,452	10012,153	4828,067	Bn,B2	34

Drilling Summary

Core Size	Hole	Length	MineGrid East	MineGrid North	MineGrid Elev	Target	Sample number
BQ	U-110-13	39,50	1101,364	10012,174	4826,59	Bn,B2	39
BQ	U-110-14	45,20	1101,348	10012,188	4826,039	B2	43
BQ	U-110-15	50,00	1101,335	10012,27	4825,772	Bn,B2	53
BQ	U-110-16	129,80	1100,102	9967,744	4816,479	Bn,B2,P	89
BQ	U-110-17	30,40	1095,3	10034,862	4804,05	Bn,B2	33
BQ	U-110-18	40,00	1095,329	10034,656	4801,966	Bn	46
BQ	U-110-19	42,00	1100,126	10007,418	4853,957	B2	53
BQ	U-111-01	66,00	1110,359	9987,383	4876,351	B2	59
BQ	U-111-02	50,00	1110,358	9987,433	4877,315	B2	44
BQ	U-111-03	41,30	1110,247	10012,716	4827,107	Bn,B2	35
BQ	U-111-04	45,30	1110,262	10012,702	4825,976	Bn,B2	37
BQ	U-111-05	49,30	1110,37	10012,749	4825,476	Bn,B2	36
BQ	U-111-06	40,00	1110,49	10001,379	4876,775	Bn,B2	38
BQ	U-111-07	35,00	1110,572	10030,551	4801,315	B2	38
BQ	U-111-08	50,30	1110,576	10030,324	4801,044	Bn	44
BQ	U-111-09	70,00	1114,117	10016,188	4800,812	Bn,P	83
ATW	U-111-10	9,00	1109,103	10046,455	4827,11	Bn	11
ATW	U-111-11	9,00	1110,459	10046,521	4827,242	Bn	9
ATW	U-111-12	9,50	1108,842	10049,836	4802,311	Bn	12
ATW	U-111-13	8,50	1110,465	10049,832	4802,249	Bn,B2	11
ATW	U-111-14	8,60	1109,855	10039,67	4852,342	Bn	11
BQ	U-112-01	46,00	1120,155	9987,156	4877,59	B2	46
BQ	U-112-02	55,20	1120,178	9987,057	4876,913	Bn,B2	58
BQ	U-112-03	74,50	1120,158	9986,998	4876,488	Bn,B2	75
BQ	U-112-04	61,70	1120,166	9987,105	4876,265	Bn,B2	63
BQ	U-112-05	70,50	1120,16	9987,159	4876,45	B2	72
BQ	U-112-06	61,60	1120,015	10013,255	4826,25	B2	58
BQ	U-112-07	65,00	1120,011	10013,253	4825,599	Bn,B2	43
BQ	U-112-08	54,70	1120,004	10013,268	4825,285	B2	38
BQ	U-112-09	41,60	1120,249	9974,745	4901,527	B2	41
BQ	U-112-10	76,50	1125,273	9990,857	4902,41	Bn,B2	85
BQ	U-112-11	66,00	1119,913	9976,382	4901,429	Bn,B2	67
BQ	U-112-12	70,20	1119,878	9975,933	4900,968	Bn,B2	86
BQ	U-112-13	41,00	1122,133	10020,69	4827,915	Bn,B2	52
BQ	U-113-01	40,00	1130,109	10013,992	4825,672	Bn,B2	24
BQ	U-113-02	45,85	1130,103	10014,015	4825,153	B2	43
BQ	U-113-03	35,50	1129,982	10011,633	4852,1	Bn,B2	35
ATW	U-113-04	41,00	1130,198	10011,518	4850,516	Bn,B2	40
BQ	U-113-05	45,00	1126,627	10029,018	4800,992	B2	51
BQ	U-113-06	29,00	1126,707	10029,268	4801,492	Bn,B2	37
BQ	U-113-07	57,00	1130,607	9987,146	4876,222	Bn,B2	71
BQ	U-113-08	56,00	1130	9986,535	4876,867	Bn,B2	69
BQ	U-113-09	42,00	1129,358	10012,394	4851,941	B2	53
BQ	U-114-01	50,20	1140,086	9974,718	4900,94	B2	48
BQ	U-114-02	45,70	1140,245	9989,409	4876,741	B2	45
BQ	U-114-03	56,00	1140,401	9989,312	4876,111	Bn,B2	57
BQ	U-114-04	66,70	1140,453	9989,37	4875,713	B2	64
BQ	U-114-05	69,50	1140,444	9989,329	4875,571	Bn,B2	49
BQ	U-114-06	91,00	1140,255	9972,318	4822,641	B2	87
BQ	U-114-07	113,00	1140,255	9972,339	4822,394	Bn	95
BQ	U-114-08	120,00	1140,245	9972,302	4822,22	Bn	72
BQ	U-114-09	200,00	1140,261	9972,196	4821,921	Bn	99
BQ	U-114-10	139,50	1139,456	9972,047	4822,254	Bn,B2	53
BQ	U-114-11	128,00	1139,689	9972,051	4822,228	Bn,B2,P	65
BQ	U-114-12	40,00	1140,089	10014,737	4826,245	Bn,B2	38
BQ	U-114-13	37,50	1140,073	10015,051	4825,649	Bn,B2	35
BQ	U-114-14	40,00	1140,06	10015,037	4825,264	Bn,B2	28
BQ	U-114-15	117,00	1139,726	9972,092	4822,322	Bn,P	88
BQ	U-114-16	96,00	1140,274	9972,365	4822,472	Bn	60
BQ	U-114-17	109,55	1140,255	9972,284	4822,177	Bn,B2	75
BQ	U-114-18	125,50	1140,239	9972,323	4822,033	Bn,B2	66
ATW	U-114-19	9,00	1138,894	10045,934	4826,295	Bn	11
ATW	U-114-20	10,20	1141,113	10045,878	4826,39	Bn	13
ATW	U-114-21	9,60	1139	10042,437	4851,923	Bn	12

Drilling Summary

Core Size	Hole	Length	MineGrid East	MineGrid North	MineGrid Elev	Target	Sample number
ATW	U-114-22	9,50	1141,252	10042,58	4851,932	Bn	14
BQ	U-115-01	65,00	1149,913	9975,128	4900,999	B2	49
BQ	U-115-02	80,00	1150,058	9987,388	4875,413	Bn,B2	58
BQ	U-115-03	94,00	1150,063	9987,375	4875,272	Bn,B2	52
BQ	U-115-04	70,00	1150,06	9987,411	4875,612	B2	48
BQ	U-115-05	66,30	1149,795	9987,335	4875,935	B2	62
BQ	U-115-06	61,00	1149,915	9987,311	4876,301	Bn,B2	60
BQ	U-115-07	61,20	1149,343	9987,136	4876,876	B2	56
BQ	U-115-08	40,00	1150,387	10015,429	4825,255	B2	39
BQ	U-115-09	47,00	1150,4	10015,394	4824,983	Bn,B2	29
BQ	U-115-10	47,00	1149,591	10011,974	4854,462	Bn,B2	59
BQ	U-115-11	43,00	1149,572	10012,231	4852,234	Bn,B2	55
BQ	U-116-01	70,50	1159,977	9974,621	4900,763	B2	52
BQ	U-116-02	80,00	1159,968	9974,531	4900,447	B2	68
BQ	U-116-03	65,50	1160,37	9987,546	4876,603	B2	69
BQ	U-116-04	65,50	1160,331	9987,386	4875,672	B2	64
BQ	U-116-05	78,00	1160,354	9987,385	4875,496	B2	54
BQ	U-116-06	47,20	1160,008	10016,107	4826,449	B2	42
BQ	U-116-07	42,00	1160,003	10015,943	4825,555	B2	36
BQ	U-116-08	50,00	1160,048	10015,86	4825,092	B2	44
BQ	U-116-09	56,00	1160,048	10015,839	4824,927	B2	42
ATW	U-116-10	22,20	1159,971	10044,821	4852,379	B2	24
ATW	U-116-11	20,60	1159,998	10044,623	4851,225	B2	24
ATW	U-116-12	15,50	1159,945	10045,041	4850,124	B2	18
ATW	U-116-13	22,50	1164,88	10045,264	4852,446	Bn,B2	23
ATW	U-116-14	21,60	1164,872	10045,504	4851,192	Bn,B2	23
ATW	U-116-15	16,50	1164,898	10045,625	4850,167	Bn,B2	17
BQ	U-116-16	50,00	1160,031	10007,912	4852,992	B2	54
BQ	U-117-01	70,00	1169,601	9974,976	4900,567	Bn,B2	63
BQ	U-117-02	86,50	1169,43	9974,991	4900,218	Bn,B2	81
BQ	U-117-03	78,50	1169,519	9975,155	4900,354	Bn,B2	73
BQ	U-117-04	35,00	1170,563	10030,349	4827,228	Bn,B2	33
BQ	U-117-05	50,00	1170,537	10030,09	4825,21	Bn,B2	46
BQ	U-117-06	27,15	1170,266	10030,383	4825,698	Bn,B2	26
BQ	U-117-07	51,00	1169,845	10001,639	4877,444	Bn,B2	59
ATW	U-117-08	9,20	1168,554	10053,14	4825,999	Bn	12
ATW	U-117-09	9,10	1170,765	10053,219	4826,082	Bn	12
ATW	U-117-14	22,00	1169,565	10046,239	4852,422	B2	22
ATW	U-117-15	21,00	1169,652	10046,067	4851,306	B2	20
ATW	U-117-16	17,00	1169,659	10046,54	4850,437	B2	17
ATW	U-117-17	39,75	1170,024	10020,938	4852,48	Bn,B2	41
BQ	U-118-01	65,50	1179,975	9975,55	4900,796	Bn	55
BQ	U-118-02	67,50	1179,975	9975,245	4900,364	B2	50
BQ	U-118-03	66,50	1179,948	9987,05	4875,553	Bn,B2	49
BQ	U-118-04	65,50	1179,936	9987,083	4875,302	Bn,B2	42
BQ	U-118-05	79,50	1179,93	9987,109	4875,098	Bn,B2	31
BQ	U-118-06	41,50	1180,466	10016,016	4825,284	Bn	36
BQ	U-118-07	45,00	1180,469	10016,042	4824,821	Bn	31
BQ	U-118-08	60,50	1180,485	10016,099	4824,519	Bn	58
BQ	U-118-09	175,60	1174,555	9941,052	4826,004	Bn,B2	97
BQ	U-118-10	161,60	1174,568	9941,152	4826,197	Bn,B2	160
BQ	U-118-11	145,50	1174,609	9941,173	4826,488	Bn,B2	100
BQ	U-118-12	59,00	1180,164	10000,237	4850,909	Bn,B2	61
BQ	U-119-01	67,00	1190,216	9976,026	4900,984	Bn,B2	40
BQ	U-119-02	75,00	1190,139	9976,196	4900,588	Bn	57
BQ	U-119-03	80,00	1190,167	9976,248	4900,424	Bn,B2	71
BQ	U-119-04	45,00	1189,85	9986,939	4874,856	Bn	39
BQ	U-119-05	94,50	1189,842	9986,863	4874,712	Bn	60
BQ	U-119-06	70,50	1189,88	9987,029	4875,047	B2	68
BQ	U-119-07	66,30	1189,877	9987,19	4875,334	Bn,B2	70
BQ	U-119-08	65,00	1190,254	10000,099	4851,225	Bn,B2	66
BQ	U-119-09	50,50	1186,071	10016,143	4826,528	Bn	45
BQ	U-119-10	46,00	1186,074	10016,322	4825,335	Bn	60
BQ	U-120-01	61,00	1199,682	9988,46	4900,645	Bn,B2	50

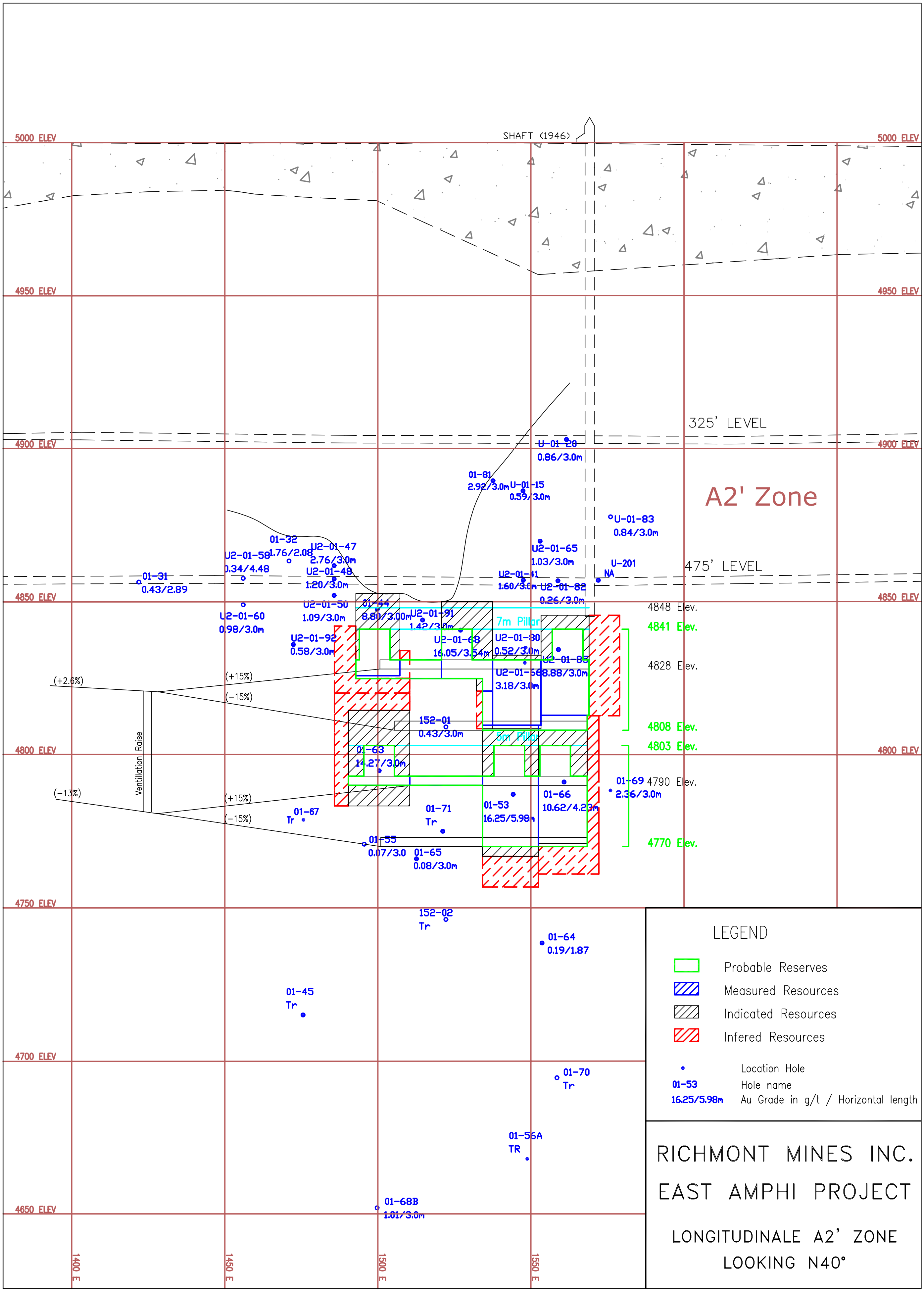
Drilling Summary

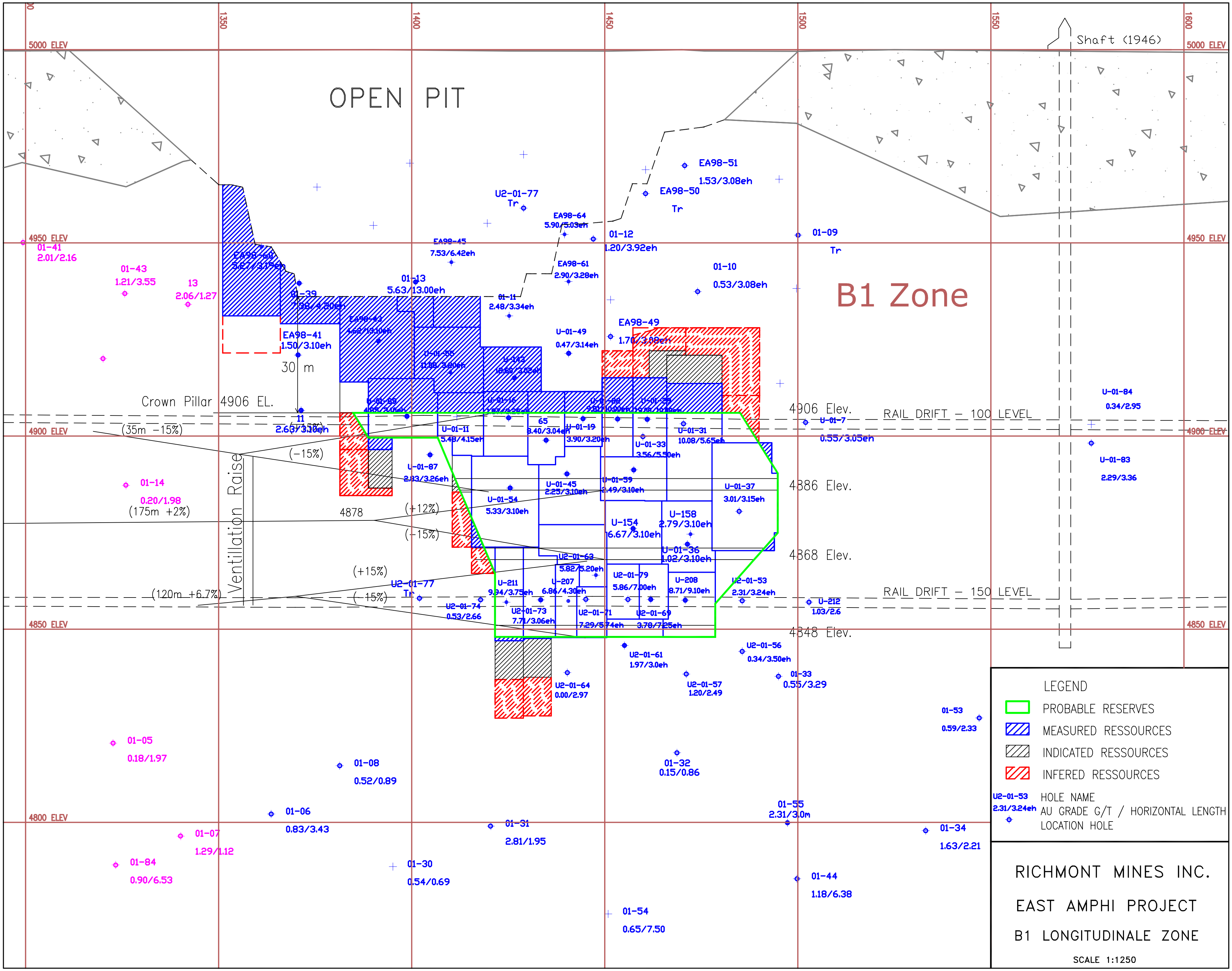
Core Size	Hole	Length	MineGrid East	MineGrid North	MineGrid Elev	Target	Sample number
BQ	U-120-02	68,00	1199,712	9988,479	4900,124	Bn,B2	61
BQ	U-120-03	60,00	1200,326	9998,325	4874,988	B2	54
BQ	U-120-04	60,50	1200,306	9998,163	4874,546	B2	37
BQ	U-120-05	60,00	1200,324	9998,03	4874,337	Bn,B2	38
BQ	U-120-06	70,00	1200,336	9997,943	4874,146	Bn,B2	71
BQ	U-120-07	60,00	1199,957	10015,672	4823,995	Bn,B2	45
BQ	U-121-01	61,70	1208,6	9975,84	4899,41	B2	37
BQ	U-121-02	70,00	1208,56	9987,612	4873,96	Bn	29
BQ	U-121-03	75,80	1208,553	9987,614	4873,812	Bn	53
BQ	U-121-04	75,00	1208,559	9987,842	4873,583	Bn	23
BQ	U-121-05	73,80	1208,727	9987,632	4874,908	Bn	79
BQ	U-122-01	80,50	1220,05	9975,05	4899,6	Bn	54
BQ	U-122-02	75,80	1220,03	9974,87	4900,04	Bn	57
BQ	U-122-03	71,63	1219,899	9987,442	4874,336	Bn	47
BQ	U-122-04	75,50	1219,927	9987,496	4874,093	Bn	20
BQ	U-122-05	88,00	1219,935	9987,517	4873,989	Bn	47
BQ	U-123-01	59,00	1230,887	9990,915	4900,672	Bn	43
BQ	U-123-02	64,00	1230,85	9990,785	4900,098	Bn	47
BQ	U-93-01	122,25	929,601	9950,606	4794,455	A3,Bn,B2,P	89
BQ	U-93-02	139,50	929,487	9950,8	4793,719	A3,Bn,B2,P	76
BQ	U-93-03	169,00	929,473	9950,677	4793,601	A3,Bn,B2,P	70
BQ	U-93-04	199,50	929,481	9950,653	4793,429	A3,Bn,B2,P	80
BQ	U-93-05	155,40	929,243	9950,658	4793,657	A3,Bn,B2,P	79
BQ	U-93-06	160,50	928,805	9950,663	4795,5	A3,Bn,B2,P	109
BQ	U-93-07	169,50	928,498	9950,696	4795,147	A3,Bn,B2,P	104
BQ	U-93-08	225,00	929,424	9950,613	4794,977	A3,Bn,B2,P	131
BQ	U-93-09	141,00	929,976	9950,447	4795,686	A3,Bn,B2,P	72
BQ	U-93-10	120,00	930,905	9950,589	4796,389	A3,Bn,B2,P	58
BQ	U-93-11	135,00	930,561	9949,852	4797,751	B2,Bn	60
BQ	U-93-12	140,00	930,304	9950,212	4797,044	A3,Bn,B2,P	81
BQ	U-93-13	201,00	929,738	9950,4	4795,906	A3,Bn,B2,P	63
BQ	U-93-14	195,00	929,495	9950,604	4795,286	A3,Bn,B2,P	65
BQ	U-93-15	140,40	930,74	9950,547	4794,336	A3,Bn,B2,P	73
BQ	U-93-16	144,00	929,938	9950,675	4794,825	A3,Bn,B2,P	68
BQ	U-93-17	120,00	930,644	9950,569	4794,959	A3,Bn,B2,P	64
BQ	U-93-18	146,00	930,659	9950,555	4794,316	A3,Bn,B2,P	66
BQ	U-93-19	180,00	929,719	9950,784	4794,287	A3,Bn,B2,P	76
BQ	U-93-20	150,00	929,885	9950,555	4794,603	A3,Bn,B2,P	74
BQ	U-93-21	145,00	929,508	9950,862	4793,974	A3,Bn,B2,P	56
BQ	U-94-01	37,90	945,932	9951,848	4796,745	P	25
BQ	U-95-01	129,00	949,798	9951,857	4798,295	A3,Bn,B2,P	106
BQ	U-95-02	167,00	949,77	9952,114	4796,075	A3,Bn,B2,P	96
BQ	U-95-03	126,00	949,617	9951,574	4798,75	A3,Bn,B2	55
BQ	U-96-01	120,00	960,354	9953,136	4798,236	A3,Bn,B2,P	70
BQ	U-96-02	130,00	960,297	9952,957	4797,628	A3,Bn,B2,P	126
BQ	U-98-01	165,40	982,082	9954,447	4801,108	A3,Bn,B2,P	89
BQ	U-98-02	176,00	982,067	9954,442	4800,956	A3,Bn,B2,P	84
BQ	U-98-03	202,00	982,055	9954,466	4800,783	A3,Bn,B2,P	104
BQ	U-98-04	120,50	980,599	9954,237	4801,375	A3,Bn,B2,P	54
BQ	U-98-05	131,00	980,07	9954,247	4800,898	A3,Bn,B2,P	70
BQ	U-98-06	36,00	982,697	10012,041	4806,596	A3,Bn,B2,P	25
BQ	U-98-07	54,85	982,89	10015,946	4805,429	A3,Bn,B2,P	43
BQ	U-98-08	36,00	980,931	10011,031	4803,593	P	23
BQ	U-98-09	45,20	980,25	10011,626	4805,178	P	32
BQ	U-98-10	75,30	980,308	10013,196	4806,073	P	37
BQ	U-98-11	56,00	981,201	10013,31	4806,62	B2,P	30
SUBTOTAL	**311**	**21236,28**					**16286**

APPENDIX 6

LONG SECTION WITH RESOURCES AND RESERVES FOR ALL ZONES

150−17
3104@3.42
150−15
2839@3.70
6921@ 3.89
150−14
6126@4.13
150−13
5102@3.90
150−12
4391@3.88
150−11
6084@3.90
150−10
12494@ 4.56
150−9
7933@ 4.83
3113@ 4.12
1129@ 4.98
175−17S
3138@ 4.27
175−17N
975@ 7.13
175−16
1511@ 3.55
175−15N
tonnes 7314
3.88 Au /gt
175−15S
858@ 4.05
175−14
7470@ 4.50
175−13
tonnes 6197
3.17 Au /gt
175−12
5281@ 3.98
175−11
tonnes 7,598
4.48 Au /gt
175−10
5725@4.10
175−9
6271@4.09
200−17N
2299@ 3.56
200−16
9450@ 3.70
200−15
tonnes 3746
3.72 Au /gt
200−14
7330@ 4.93
200−13
tonnes 2755
5.52 Au /gt
200−12
4299@ 5.60
200−11
tonnes 4709
3.94 Au /gt
200−10
3453@ 3.30
200−9
5334@ 3.69
PILIER
4.0
3.0
5.5
5.0
PILIER 7 m.


SHAFT (1946)
5000 ELEV
4950 ELEV
4900 ELEV
4850 ELEV
4800 ELEV
4750 ELEV
4700 ELEV
4650 ELEV
1400 E
1450 E
1500 E
1550 E
325' LEVEL
475' LEVEL
A2' Zone
U-01-20
0.86/3.0m
01-81
2.92/3.0m
U-01-15
0.59/3.0m
U-01-83
0.84/3.0m
U2-01-65
1.03/3.0m
U-201
NA
01-32
1.76/2.08
U2-01-47
2.76/3.0m
U2-01-58
0.34/4.48
U2-01-48
1.20/3.0m
01-31
0.43/2.89
U2-01-41
1.60/3.0m
U2-01-82
0.26/3.0m
U2-01-50
1.09/3.0m
01-44
8.80/3.00m
U2-01-91
1.42/3.0m
U2-01-60
0.98/3.0m
U2-01-92
0.58/3.0m
U2-01-68
16.05/3.54m
U2-01-80
0.52/3.0m
U2-01-85
U2-01-56
3.18/3.0m
8.88/3.0m
7m Pillar
5m Pillar
4848 Elev.
4841 Elev.
4828 Elev.
4808 Elev.
4803 Elev.
4790 Elev.
4770 Elev.
(+2.6%)
(-13%)
(+15%)
(-15%)
Ventillation Raise
152-01
0.43/3.0m
01-63
14.27/3.0m
01-69
2.36/3.0m
01-66
10.62/4.23m
01-53
16.25/5.98m
01-71
Tr
01-67
Tr
01-55
0.07/3.0
01-65
0.08/3.0m
152-02
Tr
01-64
0.19/1.87
01-45
Tr
01-70
Tr
01-56A
TR
01-68B
1.01/3.0m
LEGEND
Probable Reserves
Measured Resources
Indicated Resources
Infered Resources
Location Hole
01-53
Hole name
16.25/5.98m
Au Grade in g/t / Horizontal length
RICHMONT MINES INC.
EAST AMPHI PROJECT
LONGITUDINALE A2' ZONE
LOOKING N40°



OPEN PIT
B1 Zone
Shaft (1946)
5000 ELEV
4950 ELEV
4900 ELEV
4850 ELEV
4800 ELEV
1350
1400
1450
1500
1550
1600
Crown Pillar 4906 EL.
(35m -15%)
30 m
Ventillation Raise
(175m +2%)
(120m +6.7%)
4878
4906 Elev.
4886 Elev.
4868 Elev.
4848 Elev.
RAIL DRIFT - 100 LEVEL
RAIL DRIFT - 150 LEVEL
EA98-51
1.53/3.08eh
EA98-50
Tr
01-09
Tr
01-10
0.53/3.08eh
01-12
1.20/3.92eh
EA98-64
5.90/5.03eh
EA98-61
2.90/3.28eh
EA98-45
7.53/6.42eh
U2-01-77
Tr
01-13
5.63/13.00eh
01-11
2.48/3.34eh
EA98-49
1.70/3.08eh
U-01-49
0.47/3.14eh
EA98-41
1.50/3.10eh
01-41
2.01/2.16
01-43
1.21/3.55
13
2.06/1.27
01-14
0.20/1.98
01-05
0.18/1.97
01-07
1.29/1.12
01-84
0.90/6.53
01-08
0.52/0.89
01-06
0.83/3.43
01-30
0.54/0.69
01-31
2.81/1.95
01-54
0.65/7.50
01-32
0.15/0.86
01-55
2.31/3.0m
01-44
1.18/6.38
01-34
1.63/2.21
01-53
0.59/2.33
01-33
0.55/3.29
U2-01-56
0.34/3.50eh
U2-01-57
1.20/2.49
U2-01-61
1.97/3.0eh
U2-01-64
0.00/2.97
U-01-84
0.34/2.95
U-01-83
2.29/3.36
U-01-7
0.55/3.05eh
U-212
1.03/2.6
U-01-31
10.08/5.65eh
U-01-33
3.56/5.50eh
U-01-37
3.01/3.15eh
U-158
2.79/3.10eh
U-154
6.67/3.10eh
U-01-36
1.02/3.10eh
U-01-59
2.49/3.10eh
U-01-45
2.25/3.10eh
U-01-54
5.33/3.10eh
U-01-87
2.83/3.26eh
U-01-11
5.48/4.15eh
65
8.40/3.04eh
U-01-19
3.90/3.20eh
U2-01-63
5.82/5.20eh
U2-01-79
5.86/7.00eh
U-208
8.71/9.10eh
U2-01-53
2.31/3.24eh
U-211
9.34/3.75eh
U-207
6.86/4.30eh
U2-01-73
7.71/3.06eh
U2-01-71
7.29/5.74eh
U2-01-69
3.78/7.25eh
U2-01-74
0.53/2.66
U2-01-77
Tr
11
2.63/3.10eh
(-15%)
(+12%)
(+15%)
LEGEND
PROBABLE RESERVES
MEASURED RESSOURCES
INDICATED RESSOURCES
INFERED RESSOURCES
U2-01-53
2.31/3.24eh
HOLE NAME
AU GRADE G/T / HORIZONTAL LENGTH
LOCATION HOLE
RICHMONT MINES INC.
EAST AMPHI PROJECT
B1 LONGITUDINALE ZONE
SCALE 1:1250

APPENDIX 7

LIST OF DRILL HOLE UNUSED FOR RESERVE AND RESOURCE ESTIMATION

List of Drill Hole Unused for Reserve and Resource Estimation

BLOCK MODEL USED	HOLE-ID
No	71
No	72
No	S-035
No	S-036
No	S-037
No	S-038
No	S-039
No	S-040
No	S-041
No	S-042
No	S-050
No	S-065
No	S-066
No	S-067
No	S-068
No	S-069
No	S-070
No	S-073
No	S-074
No	S-078
No	S-082
No	S-083
No	S-084
No	S-085
No	S-086
No	S-087
No	S-088
No	S-089
No	S-090
No	S-091
No	S-092
No	S-093
No	S-094
No	S-095
No	S-096
No	S-097
No	S-098
No	S-099
No	S-100
No	S-103
No	S-104

BLOCK MODEL USED	HOLE-ID
No	S-106
No	S-108
No	S-118
No	S-120
No	S-121
No	S-122
No	S-123
No	S-124
No	S-125
No	S-126
No	S-164
No	S-165
No	S-166
No	S-167
No	S-168
No	S-169
No	S-170
No	S-171
No	S-172
No	S-173
No	S-174
No	S-175
No	S-176
No	S-177
No	S-178
No	S-179
No	S-180
No	U-162
No	U-163
No	U-167
No	U-168
No	U-169
No	U-170
No	U-171
No	U-172
No	U-231
No	U-232
No	U-233
No	U-234
No	U-235
No	U-237