Technical Report of the Preliminary Assessment on Barry-1 and Bachelor Properties of Metanor Resources Inc

Respectfully submitted to:
Metanor Resources Inc.
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By:
Systèmes Géostat International Inc.
10, boul de la Seigneurie, suite 203
Blainville, QC, J7C 3V5
Phone : (450) 433-1050
Fax : (4500 433-1048
E-mail : info@Geostat.com

FOREWORD

The objective of this study is to produce a preliminary economical assessment of the mineral resources of two gold Properties of Metanor Resources Inc. These two Properties are: Barry-1 and Bachelor Lake Mine where resources estimates were recently completed.

This preliminary assessment study is done with the objective of evaluating the economical results of exploiting these two Properties by using the existing concentrator at the Bachelor Lake Mine site that is actually under refurbishing. The proposed production program is to first exploit the Barry-1 Property by open-pit while the Bachelor Lake Mine underground property will be developed.

This study is done in accordance to a mandate given to Geostat Systems International by Metanor Resources Inc. of Val d’Or.

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Summary (Item 3)

Geostat Systems International Inc. (Geostat) was selected by Metanor Resources Inc. (Metanor) of Val d'Or, QC to prepare a Preliminary Economic Assessment report for the Barry-1 (Barry) and Bachelor Lake Mine (Bachelor) properties.

Geostat reminds that this Preliminary Assessment is preliminary in nature, that it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. This report is in accordance to National Instrument 43-101

Note: This report is showing both imperial and metric units. Production costs are all reported in CDN$ per short ton.

Properties Description

Barry

The Barry property is located in the Barry Township QC and is composed of 7 claims for a total of 111.6 Ha. This property is around 100 km east of the town of Lebel-sur-Quévillon and 65 km south of the Bachelor Lake concentrator. The access to the property either from Lebel-sur-Quévillon or Bachelor Lake is easy as good gravel roads are available all year long. Metanor Resources Inc. is holding 100% interest the Barry-1 property following the purchase agreement done during 2006.

Bachelor

The Bachelor property is located in the Le Sueur Township, Northwestern Quebec, only 3.5 km east of Desmaraisville along the provincial road No 113 going from Val d'Or to Chibougamau. This property is on the site of the former underground Bachelor Lake Mine that was in production from 1982 to 1989.

This property was formerly known as the Bachelor Lake Joint Venture (BLJV) in which Metanor Resources Inc and Halo Resources Ltd each had a 50% interest. Following a purchase agreement done in 2006, Metanor will be the sole owner of the property when all the purchase requirements will be completed on or before November 2007.

The extension of the known resources are covering an area located within 9 claims (CL) and one mining concession (CM) representing a total of 184.73 Ha. This is the site of the existing surface infrastructures. A large block of 137 claims and one mining concession totalizing 4,081.62 Ha are contiguous to the first 9 claims. All these claims and mining concessions belong to Metanor Resources Inc.

Geology and Mineralization

Barry

The Barry property is located along the Murgor Shear Zone. All the rocks of the Barry project property are metamorphosed to the greenschist facies. Three distinct rock units are present on the property:

1. Unaltered mafic metavolcanic rocks (massive and pillowed).
2. Altered mafic metavolcanic rocks (Fe-carbonate with quartz and albite veining with 2-5% pyrite).
3. Quartz-feldspar porphyry dykes and plugs.

Gold-bearing mineralization lies in pyritized and moderately altered volcanic flows near contacts with quartz-feldspar porphyry dykes and plugs. Alteration mineralogy includes albite, carbonate, biotite, ankerite, epidote, chlorite, sericite and garnet. Those minerals are distributed in a board halo of chlorite-sericite-calcite-magnetite crosscut by a magnetite destructive proximal halo of biotite-ankerite-albite and pyrite. Mineralization is characterized by a system of east-northeast and north-northeast sheeted quartz-carbonate veins dipping at 40-60 degrees to the south-southeast and the east-southeast respectively. The known mineralization is up to 500 metres long, up to 150 metres wide and tested, for the majority of the drill holes, to a vertical depth of 50 metres. Only 20 holes reached a vertical depth greater than 100 metres in the Main Zone Area (Main Zone, zones 43 and 45).

The Barry I gold mineralized envelopes are located south of a major shear and fold zone. The mineralized envelopes represent sub-horizontal elongated dome shapes even if the mineralized veins are dipping moderately. The mineralized envelopes show variable thicknesses, which vary from a few centimetres to more than 30 metres. The actual mineralized envelopes of the Main Zone Area occur mainly in the first 30 metres of rock below surface. The gold grades vary within the envelopes and visible gold is frequently observed in the core.

Exploration work, including extensive drilling, was performed during several phases and over several years, from 1983 to 2007. Some 78 drill holes were completed on the property in the second half of 2004 and the beginning of 2005 by Osisko. A total of 61 of these drill holes were drilled on the Barry I Main Zone Area and the remaining 17 tested additional exploration targets on the Barry property. Six drill holes were drilled at the end of 2005 and 32 in the beginning of 2006 by Murgor. Three major trenches and a broad stripped zone are present on the Main Zone. 58 new holes have been drilled in 2006-2007.

Ghislain Deschênes, professional geologist and qualified person assigned to the study, visited the site of the Barry property in December 2005, during the drilling campaign realized by Murgor in 2005. Geostat declares that the assay verification program confirms the gold values present in the database and that no statistical bias was observed.

The 2006-2007 drilling campaign permitted to link the mineralization from the zones 43 and 45 to the Main and found a new zone at the southeast of the Main Zone.

The staff of Murgor and Geostat carried out the geological interpretation of the gold-bearing envelopes. The gold-bearing intersections were defined in accordance with this interpretation. A set of composites of regular length of 1.5-metres was created in order to assess the continuity of the gold mineralization, define interpolation parameters and carry out the interpolation of the grades for the resource calculation.

Bachelor

Geological Setting and Mineralization

The property is located within the Northern Volcanic Zone of the Archean Abitibi Greenstone Belt, Superior Province of the Canadian Shield and lies along the major northeast-trending Wedding-Lamarck fault. The property hosts a wide variety of deposit types from volcanogenic polymetallic mineralization (zinc showings no.1 and no.2; Coniagas horizon) to syn- to late-orogenic gold deposit (Bachelor Lake gold deposit). The Bachelor Lake gold mineralization has been interpreted to be associated with a late-tectonic granodioritic intrusion (the O'Brien pluton located east of the deposit and associated dykes documented at the mine). The mineralized zones, six (6) gold-bearing zones ("Main", "A", "B", "C", "A West" and "B West"), usually consist of disseminated sulphides (pyrite) and variably developed stockworks in intensely altered wallrocks (red-colour silica-hematite alteration). The "Main", "A", and "B" zones were originally defined at the Bachelor mine and extend to the West on the Hewfran claims (Hewfran East zone area). The "A West" and "B West" zones are located in the Hewfran West zone area and can be interpreted as extensions of the "A" and "B" zones documented at the Bachelor mine. The "A" and "A West" zones are associated with later shearing and interpreted as gold remobilization from earlier formed gold-bearing zones.

Mineral Resources Estimate

Barry

The resources of the Barry I Main Zone Area (Main Zone, Zones 43 and 45) gold deposit were estimated by inverse distance composites of 1.5 metres length. A measured specific gravity of 2.8 g/cm^3 is used in this study for all rock types.

Even if the majority of the drill holes collars of the Main Zone Area still visible on the site were surveyed, the absence of a detailed survey of the topographic surface in the considered area, the incertitude regarding the position of the hole collars not surveyed and the difficulty to get a good anisotropic variogram cause by the short distance of influence of the samples do not allow us to declare measured resources at this stage. The topographic surface is acting as a mineralization contact boundary. The parameters used to define the indicated and inferred categories are the distance and the number of composites and are the following:

Category	Search ellipse (oriented 65N)	Minimum number of composites	Maximum number of composites per hole
Indicated	25 m, 12.5 m, 6m	4	2
Inferred	Inside the mineralized envelope, not indicated		

Different cut-offs grades of 1, 2, 3, 4 and 5 g/t Au were used for the resource calculation in various scenarios. Barry I Main Area resources (Main, 43 and 45), calculated by inverse distance and rounded:

Total resources inverse distance (No cut-off) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**Density**	**Au (g/t)**	**Oz Au**
Indicated	**415,000**	**148,000**	**2.8**	**4.05**	**54,000**
Total	**415,000**	**148,000**	**2.8**	**4.05**	**54,000**
Inferred	1,102,000	394,000	2.8	3.78	133,800
Total resources inverse distance (Cut-off of 1 g/t) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**2.80**	**4.00**	**36,100**
Indicated	**415,000**	**148,000**	**2.8**	**4.05**	**54,000**
Total	**415,000**	**148,000**	**2.8**	**4.05**	**54,000**
Inferred	1,102,000	394,000	2.8	3.78	133,800
Total resources inverse distance (Cut-off of 2 g/t) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**Density**	**Au (g/t)**	**Oz Au**
Indicated	**385,000**	**138,000**	**2.8**	**4.23**	**52,300**
Total	**385,000**	**138,000**	**2.8**	**4.23**	**52,300**
Inferred	966,000	345,000	2.8	4.07	126,600
Total resources inverse distance (Cut-off of 3 g/t) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**Density**	**Au (g/t)**	**Oz Au**
Indicated	**277,000**	**99,000**	**2.8**	**4.89**	**43,600**
Total	**277,000**	**99,000**	**2.8**	**4.89**	**43,600**
Inferred	690,000	246,000	2.8	4.70	104,300
Total resources inverse distance (Cut-off of 4 g/t) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**Density**	**Au (g/t)**	**Oz Au**
Indicated	**174,000**	**62,000**	**2.8**	**5.74**	**32,100**
Total	**174,000**	**62,000**	**2.8**	**5.74**	**32,100**
Inferred	404,000	144,000	2.8	5.59	72,600
Total resources inverse distance (Cut-off of 5 g/t) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**Density**	**Au (g/t)**	**Oz Au**
Indicated	**109,000**	**39,000**	**2.8**	**6.49**	**22,800**
Total	**109,000**	**39,000**	**2.8**	**6.49**	**22,800**
Inferred	225,000	80,000	2.8	6.46	46,700

Bachelor Resources by Geostat

In order to be able to calculate stopes in the indicated and inferred resource, all the zones listed in the NI43-101 December 2005 report from InnovExplo were recalculated for this report on East-West longitudinals. The following procedure was used:

To get an equivalent to the 2.5 ft composite cutting by InnovExplo, we did some cutting directly on the assays. For each vein:

- Geological intervals from InnovExplo were used.
- Two or three intervals in one hole were summed to make one unique intersection.
- Thicknesses of geological intercepts were calculated for the longitudinal projection. (Horizontal thicknesses in the North-South direction were calculated).
- A two dimensional longitudinal block model was then interpolated from these intercepts.

All details are shown under Title 16 of the actual study. The table below is showing the all categories resources result in Imperial Units.

UNDILUTED STOPE RESOURCES - FROM CONTACT TO CONTACT

Ref	Stope Name	Thickness (ft)	Au opt	Volume ft3	T/m3	Tonage (short tons)	Au (oz)	Category
1	M06-03W	8.2	0.24	131,212	2.75	11,239	2,699	Ind. & Inf.
2	M08-03W	13.3	0.20	753,507	2.75	64,544	12,973	Ind. & Inf.
3	M09-03W	11.7	0.22	240,832	2.75	20,629	4,464	Ind. & Inf.
4	M10-03W	7.2	0.21	139,120	2.75	11,917	2,529	Ind. & Inf.
5	M12-01W	10.2	0.25	314,468	2.75	26,937	6,635	Ind. & Inf.
6	M12-04E	9.2	0.15	396,070	2.75	33,927	4,930	Ind. & Inf.
7	M13-01W	13.5	0.20	798,713	2.75	68,417	13,458	Ind. & Inf.
8	M13-02E	17.1	0.13	614,622	2.75	52,648	6,713	Ind. & Inf.
9	M14-01W	16.3	0.25	634,521	2.75	54,352	13,441	Ind. & Inf.
10	M14-02E	13.2	0.15	456,390	2.75	39,094	5,758	Ind. & Inf.
11	M15-01W	13.2	0.24	460,281	2.75	39,427	9,541	Ind. & Inf.
12	8-02-E	8.5	0.36	18,223	2.75	1,561	568	Measured
13	8-03-E	8.5	0.25	90,567	2.75	7,758	1,932	Measured
14	1002W	12.5	0.32	253,910	2.75	21,750	7,000	Measured
15	11-02-E-1	8.3	0.38	85,115	2.75	7,291	2,741	Measured
16	11-02-E-2	7.8	0.22	151,110	2.75	12,944	2,783	Measured
17	1102W	9.7	0.34	146,112	2.75	12,516	4,270	Measured
18	1202W	9.9	0.32	241,150	2.75	20,657	6,630	Measured
	TOTAL Main	**12.8**	**0.202**			**507,607**	**102,435**	
19	B12-01W	8.7	0.25	108,465	2.75	9,291	2,298	Ind. & Inf.
20	B12-03W	4.8	0.26	54,208	2.75	4,643	1,193	Ind. & Inf.
21	B13-01W	12.8	0.23	345,237	2.75	29,572	6,781	Ind. & Inf.
22	B13-02E	27.1	0.22	596,232	2.75	51,072	11,113	Ind. & Inf.
23	B14-01W	9.1	0.36	250,547	2.75	21,461	7,713	Ind. & Inf.
24	B14-02E	26.2	0.20	1,737,160	2.75	148,802	29,091	Ind. & Inf.
25	B15-01W	4.2	0.20	129,232	2.75	11,070	2,255	Ind. & Inf.
	TOTAL B	**21.8**	**0.219**			**275,913**	**60,444**	
26	AW06-03-W	6.1	0.190	81,718	2.75	7,000	1,400	Ind. & Inf.
27	AW08-05-W	16.3	0.220	175,110	2.75	15,000	3,200	Ind. & Inf.
28	AW08-07-W	14.7	0.200	1,330,836	2.75	114,000	22,400	Ind. & Inf.
	Total AW	**14.70**	**0.199**			**136,000**	**27,000**	
	TOTAL: B+Main+AW		**0.212**			**919,520**	**189,878**	

Note: This resource estimate was prepared by M. Yann Camus, an engineer working for Geostat.

The diluted tonnage estimated to be send to the mill is illustrated in the following table and was prepared by leaving 10% of the resources in place as pillars and losses and adding 10% material at a grade of 0.03 opt

Estimation of the Bachelor Property "retained for mining" Resources of all Categories

Stope Name	Thickness (ft)*	Grade opt	Density	Total tons (st)	Total ounces in place (oz)	10% losses and pillars (st)	10% losses and pillars (oz)	Tons before dilution (st)	Ounces before dilution (oz)	Tons of dilution at 10%	Oz of dilution at 0.03 opt	Ounces (oz)	Mill feed diluted tons (st)	Mill feed grade opt	Class
M06-03-W	8.2	0.24	2.75	11,239	2,699	1,124	270	10,115	2,429	1,012	30	2,459	11,127	0.221	II
M08-03-W	13.3	0.20	2.75	64,544	12,973	6,454	1,297	58,090	11,676	5,809	174	11,850	63,899	0.185	II
M09-03-W	11.7	0.22	2.75	20,629	4,464	2,063	446	18,566	4,018	1,857	56	4,073	20,423	0.199	II
M10-03-W	7.2	0.21	2.75	11,917	2,529	1,192	253	10,725	2,276	1,073	32	2,308	11,798	0.196	II
M12-01-W	10.2	0.25	2.75	26,937	6,635	2,694	663	24,243	5,971	2,424	73	6,044	26,667	0.227	II
M12-02-E	9.2	0.15	2.75	33,927	4,930	3,393	493	30,534	4,437	3,053	92	4,528	33,587	0.135	II
M13-01-W	13.5	0.20	2.75	68,417	13,458	6,842	1,346	61,575	12,112	6,157	185	12,297	67,732	0.182	II
M13-02-E	17.1	0.13	2.75	52,648	6,713	5,265	671	47,383	6,041	4,738	142	6,183	52,121	0.119	II
M14-01-W	16.3	0.25	2.75	54,352	13,441	5,435	1,344	48,917	12,097	4,892	147	12,244	53,809	0.228	II
M14-02-E	13.2	0.15	2.75	39,094	5,758	3,909	576	35,184	5,183	3,518	106	5,288	38,703	0.137	II
M15-01-W	13.2	0.24	2.75	39,427	9,541	3,943	954	35,484	8,587	3,548	106	8,694	39,033	0.223	II
8-02-E	8.5	0.36	2.75	1,561	568	156	57	1,405	511	140	4	516	1,545	0.334	Measured
8-03-E	8.5	0.25	2.75	7,758	1,932	776	193	6,982	1,739	698	21	1,760	7,680	0.229	Measured
1002W	12.5	0.320	2.75	21,750	6,960	2,175	696	19,575	6,264	1,958	59	6,323	21,533	0.294	Measured
11-02-E-1	8.3	0.376	2.75	7,291	2,741	729	274	6,562	2,467	656	20	2,487	7,218	0.345	Measured
11-02-E-2	7.8	0.215	2.75	12,944	2,783	1,294	278	11,650	2,505	1,165	35	2,540	12,815	0.198	Measured
1102W	9.7	0.340	2.75	12,516	4,255	1,252	426	11,264	3,830	1,126	34	3,864	12,391	0.312	Measured
1202W	9.9	0.320	2.75	20,657	6,610	2,066	661	18,591	5,949	1,859	56	6,005	20,450	0.294	Measured
TOTAL Main	**13.0**	0.215	2.75	507,607	108,990	50,761	10,899	456,846	98,091	45,685	1,371	99,462	502,531	0.198	
B12-01-W	8.7	0.247	2.75	9,291	2,298	929	230	8,362	2,068	836	25	2,093	9,198	0.228	II
B12-03-W	4.8	0.257	2.75	4,643	1,193	464	119	4,179	1,074	418	13	1,086	4,597	0.236	II
B13-01-W	12.8	0.229	2.75	29,572	6,781	2,957	678	26,615	6,103	2,662	80	6,183	29,277	0.211	II
B13-02-E	27.1	0.218	2.75	51,072	11,113	5,107	1,111	45,965	10,002	4,597	138	10,140	50,562	0.201	II
B14-01-W	9.1	0.359	2.75	21,461	7,713	2,146	771	19,315	6,942	1,932	58	7,000	21,247	0.329	II
B14-02-E	26.2	0.196	2.75	148,802	29,091	14,880	2,909	133,922	26,182	13,392	402	26,584	147,314	0.180	II
B15-01-W	4.2	0.204	2.75	11,070	2,255	1,107	225	9,963	2,029	996	30	2,059	10,959	0.188	II
TOTAL B	**20.8**	0.219	2.75	275,913	60,444	27,591	6,044	248,322	54,399	24,832	745	55,144	273,154	0.202	
TOTAL AW		0.199	2.75	135,611	26,954	13,561	2,695	122,050	24,259	12,205	366	24,625	134,255	0.183	
GRAND TOTAL		**0.214**	**2.75**	**919,131**	**196,388**	**91,913**	**19,639**	**827,218**	**176,749**	**82,722**	**2,482**	**179,231**	**909,940**	**0.197**	

Bachelor (by InnovExplo)

Resources were published in December 2005 by InnovExplo of Val d'Or and the following results are reproduced from the Ni 43-101 Technical Report on 2005 Drilling Program and Mineral Resources Estimate for the Bachelor Property.

BACHELOR LAKE MINERAL RESOURCES (METRIC UNITS)

		BACHELOR	**HEWFRAN**	**TOTAL**
Measured	**Metric Tons (t)**	**177 898**	**14 696**	**192 594**
	Grade (g/t)	**8.83**	**8.50**	**8.80**
	Oz of Gold	*50 487*	*4 018*	***54 504***
	kg of Gold	*1 570*	*125*	***1 695***
Indicated	**Metric Tons (t)**	**465 928**	**183 069**	**648 997**
	Grade (g/t)	**7.63**	**7.14**	**7.49**
	Oz of Gold	114 329	42 024	**156 352**
	kg of Gold	*3 556*	*1 307*	***4 861***
Measured + Indicated	**Metric Tons (t)**	**643 826**	**197 765**	**841 591**
	Grade (g/t)	**7.96**	**7.24**	**7.79**
	Oz of Gold	*164 815*	*46 042*	***210 857***
	kg of Gold	*5 126*	*1 432*	***6 556***
Inferred	**Metric Tons (t)**	**207 517**	**218 630**	**426 148**
	Grade (g/t)	**6.76**	**6.30**	**6.52**
	Oz of Gold	*45 083*	*44 283*	***89 366***
	kg of Gold	*1 402*	*1 377*	***2 778***

1.) The Qualified People for the Mineral Resource estimates as defined by National Instrument 43-101 were Alain Carrier, M.Sc., P.Geo. and Julien Davy, M.Sc., P.Geo. (Innovexplo Inc.), and the effective date of the estimate is October 5, 2005.
2.) Mineral Resources are not Mineral Reserves having demonstrated economic viability.
3.) Results are presented undiluted and in situ, and some resource blocks may be locked in pillars. The estimate included six (6) gold-bearing zones ("Main", "A", "B", "C", "A West" and "B West") and covers the Bachelor Lake, Hewfran East and West areas.
4.) The resources were compiled using a cut-off grade of 3.43 g/t Au. This cut-off must be re-evaluated in the light of the present market conditions: gold price, exchange rate and mining cost. A fixed density of 2.755 g/cm^3 was used. A minimum of 1.5 m horizontal width was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed. High grade capping were fixed at 51.4 g/t Au for the "Main" zone, and to 34.3 g/t Au for the "A", "B", "C", "A" West and "B" West zones and were done on 0.75 m drill hole composite interval.
5.) Measured Resources were evaluated from a polygonal method using underground geological mapping and face sampling assay results.
6.) Indicated and Inferred Resources were evaluated from drill hole results using a block model approach (inverse distance squared interpolation) constrained within six (6) individual 3D wire frames ("Main", "A", "B", "C", "A West" and "B West" zones).
7.) Calculations used Imperial units (feet, short tons and oz/short ton Au) and results were rounded to reflect their "estimate" nature. These results were later converted into Metric using a factor of 0.90178

for the conversion of short tons into tonnes and a factor of 34.2865 for the conversion of oz/t Au into g/t Au.

8.) The companies are not aware of any known environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues that could materially affect the Mineral Resource estimates.

Expressed in Imperial Units the same resources are as shown below.

BACHELOR LAKE RESOURCES SUMMARY (IMPERIAL UNITS)

		BACHELOR	HEWFRAN	TOTAL
Measured	Short Tons (t)	196 100	16 200	**212 300**
	Grade (oz/t)	0.257	0.248	**0.257**
	Oz Gold	*50 487*	*4 018*	***54 504***
Indicated	Short Tons (t)	513 600	201 800	**715 400**
	Grade (oz/t)	0.223	0.208	**0.219**
	Oz Gold	*114 329*	*42 024*	***156 352***
Measured + Indicated	Short Tons (t)	**709 700**	**218 000**	**927 700**
	Grade (oz/t)	**0.232**	**0.211**	**0.223**
	Oz Gold	***164 815***	***46 042***	***210 857***
Inferred	Short Tons (t)	228 750	241 000	**469 750**
	Grade (oz/t)	0.197	0.184	**0.190**
	Oz Gold	*45 083*	*44 283*	***89 366***

Mining Plan

Barry

The total mineralized material at the Barry-1 project appeared to be economically mineable through the open pit mining method. From the beginning, the combined East and West pits represented some economical weaknesses on account of the 1,872,000 st (1,698,000 mt) of waste that would have to be mined out to access the 456,500 st (414,171 mt) of resources at an average grade of 0.149 opt (5.10 g/t Au), before dilution. After a tonnage dilution of 20% at a grade of 0.15 opt (0.5 g/t Au), the average grade to the mill drops to 0.126 opt (4.33 g/t Au).

To total waste material to be moved to access the ore material is increased by 491,000 st (445,591 mt) when taking the overburden into consideration. The Barry pit(s) being at near 100 km away by road from milling facilities automatically increases the cost of placing a tonne at the mill site substantially.

To decrease the waste and the overburden tonnage to a more acceptable level while improving the economical viability of the project a combination of open pitting and one underground mining option was scrutinized. An eastern open-pit with the following resources is proposed plus a mechanized open stope underground exploitation for a high grade portion of the west zone

East Pit Resources (metric tonnes)

Bench	Floor Elevation	Ore tonnage	Waste tonnage	OVB tonnage	Ore Au
1	2,014	0	5,756	3,129	
2	2,011	260	13,536	34,014	3.02
3	2,008	5,548	50,455	48,100	3.54
4	2,005	21,600	117,617	52,707	3.83
5	2,002	49,512	155,354	57,033	3.89
6	1,999	67,728	139,500	39,031	4.03
7	1,996	66,225	138,562	18,031	4.13
8	1,993	64,559	105,781	2,936	3.97
9	1,990	52,334	67,501	14	4.10
10	1,987	28,431	65,725	0	4.17
11	1,984	17,719	31,577	0	4.96
12	1,981	14,475	12,868	0	5.37
13	1,978	8,520	5,807	0	5.56
14	1,975	2,517	2,350	0	6.46
Total		399,427	912,390	254,995	4.16
		W/O ratio:	2.92		

West Zone diluted resources in imperial tons

Bench	Tons	Grade	Ounces
12	7,453	0.147	1,094
13	9,889	0.157	1,555
14	11,184	0.172	1,928
15	12,728	0.200	2,540
16	9,161	0.214	1,959
11	10,120	0.222	2,243
10	5,333	0.260	1,388
Sub-Total	65,868	0.193	12,706
Dilution 15% at 0.015 opt	9,880	0.015	144
Total	**75,748**	**0.170**	**12,850**

The proposed mining/milling tonnage forecasts as discussed with the owners is to run the mill at 500 short tons per day for 4 months, and increased it to 750 tons per day. The Barry exploitation would then last for twenty four (24) months.

Bachelor

The Bachelor ore is steeply dipping and competent as are both walls; leading to open stope mining methods. Three underground mining methods are proposed at Bachelor with the following proportions.

Bachelor Proposed Mining Methods

Description	%
Long-hole	52
Alimak vein mining	38
Shrinkage	10
Total	**100%**

Capital Expenditure (CAPEX)

Barry

The CAPEX that is applied against the revenues of Barry is the refurbishing of the mill at 500 tons/day, the increase to 750 tons per day, the tailing pond studies and rehabilitation and a provision for the exploitation closure.

Barry-1 CAPEX

Description	$ (total)
Miil refurbishing at 500 tons per day	$ 2,618,000
Mill increase to 750 tons per day	$ 1,100,000
Tailing pond rehabilitation	$ 2,190,000
Exploitation closure provision	$ 300,000
Total	$ 6,208,000

Bachelor

The overall summary of Bachelor CAPEX is shown below

Bachelor Mine CAPEX

Description	**CDN$**
Hoist installation	**$ 1,020,750**
Service building and Warehouse	**$ 600,000**
Compressors and generators repairs	**$ 385,000**
Shaft sinking, ore & waste passes	**$ 9,196,131**
Camp	**$ 600,000**
Explosive & detonators magazines	**$ 85,600**
Level developments (12-13-14-15-16)	**$ 3,031,248**
Equipment acquisition	**$ 2,343,000**
Ventilation study	**$ 16,050**
Mine closure provision	**$ 1,500,000**
Total	**$ 18,777,779**

The above amount will be needed over a period of 18 to 24 months depending of the owners' development schedule. For the Cash Flow estimate the costs are distributed over a period of 22 months.

Metallurgical Recovery

Barry Ore

A bench-scale testwork performed in 2006 on two composites samples of gold bearing ore at Queen's University in Ontario has returned results demonstrating that conventional cyanidation provided the highest extraction of gold yielding results of 94.2% to 97.5%.

A copy of the report was transmitted by Metanor. A two (2) pages document describing the content of composite sample is produced in Title 15 (Item 18) of the actual report, and the complete report from Queen's is shown in the appendices of the report.

In the economic study of Barry 1, a recovery of 95% is used for the first four (4) months and 96% thereafter.

Bachelor Ore

In his concentrator's study M. Gilbert Rousseau is mentioning that a mill run made at the Lake Short concentrator before the shut down in 1989 reported recovery of +95 %.

The mill recovery for the Bachelor ore is assumed to be 96%.

Milling Costs

The estimation of the milling costs was done by Gilbert Rousseau eng, a consultant who was hired by Geostat and visited the concentrator. The cost estimation for 500 tons per day and 750 tons are shown below.

BACHELOR LAKE MINE CONCENTRATOR OPERATING COSTS

Description	500 st/day	750 st/day
	350 days/year	350 days/year
	90% availability	90% availability
	157,500 st/year	236,250 mt/year
	13,125 st/month	19,687 st/month
Rounded to	**13,125**	**20,000 st/month**
Labour (incl 34% F.B.)	$ 14.03	$ 9.65
Consumables	$ 8.67	$ 8.67
Sub-total	$ 22.70	$ 18.32
Overhead at 10%	$ 2.27	$ 1.83
Total	**$ 24.97**	**$ 20.15**

According to Gilbert Rousseau estimation the mill could be operated by 27 people including the staff.

Mining Operating Costs

Barry

East Pit Costs

The excavation costs are from a contractor proposal and the milling costs are from the estimation done by Mr Gilbert Rousseau.

The total east open-pit costs are shown below.

East Pit Production Costs - imperial tonnes

Description	$/t	Q (tons)	$ total	$/t of ore
Overburden: $/ton of overburden	$ 2.72	281,000	$ 764,320	$ 1.74
Waste: $/ton of waste	$ 3.95	1,012,500	$ 3,999,375	$ 9.09
Ore: $/ton of ore	$ 5.75	440,000	$ 2,530,000	$ 5.75
Crushing: $/ton of ore	$ 1.28	440,000	$ 563,200	$ 1.28
Transport: $/ton of ore	$ 16.53	440,000	$ 7,273,200	$ 16.53
Ore selectivity	$ 1.25	440,000	$ 550,000	$ 1.25
Gen + Administration	$ 4.55	440,000	$ 2,002,000	$ 4.55
Sub-total			*$ 17,682,095*	*$ 40.19*
(milling at 500 tpd) Total	$ 24.97	52,500	$ 1,310,925	$ 65.16
(milling at 750 tpd) Total	$ 20.15	387,500	$ 7,808,125	$ 60.34
Average total cost for East Pit		**440,000**	**$ 26,801,145**	**$ 60.91**

West zone costs

West Zone Production Costs - in imperial

Description	$/t	Tons	$ total	$/t of ore
Development			$ 1,592,500	$ 20.95
Ore Mining: $/ton of ore	$ 28.13	76,000	$ 2,137,880	$ 31.10
Crushing: $/ton of ore	$ 1.28	76,000	$ 97,280	$ 1.42
Transport: $/ton of ore	$ 16.53	76,000	$ 1,256,280	$ 18.28
Gen + Administration	$ 5.00	76,000	$ 380,000	$ 5.53
Sub-total		76,000	*$ 5,463,940*	$ 79.49
Milling at 750 st per day	$ 20.15	76,000	$ 1,531,400	$ 22.28
Average total cost for West UG Zone		76,000	**$ 6,995,340**	**$ 92.04**

Bachelor Costs

The underground production mining costs are estimated as shown below.

Average Mining Costs for all methods			
Description	%	$/t	Total
Long-hole	52%	$ 61.44	$ 32.41
Alimak vein mining	38%	$ 66.65	$ 25.69
Shrinkage	10%	$ 83.10	$ 8.43
Total	100%	Average	**$ 66.54**
	Less administration		$ 29.03
Stope development, mining and services costs			**$ 37.51**

Estimated Preliminary Cash Flow of Barry-1 and Bachelor properties

The Base Case Cash Flow was prepared with the following parameters.

East Pit average production cost at 500 stpd	**$65.16/t**
East Pit average production cost at 750 tpd	**$60.34/t**
West Zone production cost (including development)	**$92.04/t**
Bachelor Lake production costs	**$66.54/t**
Mill recovery	**95-96%**
Gold price	**$C660/oz**
Barry-1 CAPEX	**$6,208,000**
Bachelor CAPEX	**$18,777,779**

The next page table is the final summary of the Cash Flow

SUMMARY of BARRY-1 & BACHELOR LAKE MINE CASH FLOW

BARRY: 4 months at 500 stpd, after 750 stpd										
Description	Preproduction	Unit costs	Year-1	Year-2	Year-3	Year-4	Year-5	Year-6	Year-7	**TOTAL**
CAPEX		$/ st (ore)								
Total CAPEX	**$ 6,208,000**									
East Pit Production Costs										
Total Expenses		$ 85.31	$ 11,349,895	**$ 12,769,110**						**$ 24,119,005**
REVENUES	440,000		212,500	231,500						444,000
Ounces produced			24,847							
Gross revenue @ $CDN / oz	$ 660		16,398,783	**$ 17,627,810**						**$ 34,026,593**
Gross profit before royalties			2,168,708	**$ 4,858,700**						$ 9,907,588
Ore NSR royalties - 10%			1,639,878	$ 1,762,781						$ 3,402,659
Milling NSR royalty - 1%			163,988	$ 176,278						$ 340,266
(East Pit) EBITDA			364,841	**$ 2,919,641**						**$ 3,284,483**
BARRY-1 West Zone - UG Mining		$/st(ore)	Year-1	Year-2	Year-3					**TOTAL**
Total Expenses		**$ 91.59**		**$ 732,752**	$ 6,228,388					**$ 6,961,140**
REVENUES										
Ounces produced				1,306	11,098					11,098
Gross revenue @ $CDN / oz	$ 660	$ 660		**$ 861,696**	**$ 7,324,416**					**$ 8,186,112**
Gross profit (loss) before royalties				**$ 128,944**	**$ 1,096,028**					**$ 1,224,972**
Ore NSR royalties - 10%				$ 86,170	$ 732,442					$ 818,611
Milling NSR royalty - 1%				$ 8,617	$ 73,244					$ 81,861
(Barry-1 UG) EBITDA				**$ 34,158**	**$ 290,342**					**324,500**
BARRY-1 : Annual Cash Flow			**$ 364,841**	**$ 2,953,799**	**$ 290,342**					**$ 3,608,982**
BARRY-1 : Cumulative Cash Flow	**-$ 6,208,000**		**-$ 5,843,159**	**-$ 2,889,360**	**-$ 2,599,018**					
BACHELOR LAKE MINE 750 st/day			Year - 1	Year - 2	Year - 3	Year-4	Year-5	Year-6	Year-7	Total
					172,000	240,000	240,000	240,000	18,000	910,000
CAPEX (CDN$)	CAPEX	$/ st of ore								
Surface Sub-total	***$ 2,691,350***	8 months	**$ 2,691,350**							**$ 2,691,350**
Mine Development Sub-total	***$ 14,586,429***	16 months	$ 911,650	$ 10,939,800	$ 2,734,950					**$ 14,586,400**
Mine closure provision	*$ 1,500,000*	5 months						$ 1,500,000		$ 1,500,000
Total on going CAPEX	***$ 18,777,779***		**$ 3,603,000**	**$ 10,939,800**	**$ 2,734,950**			**$ 1,500,000**		**$ 18,777,750**
Production Costs										
Zones: Main + B + AW (st)		910,000			172,000	240,000	240,000	240,000	18,000	910,000
Sub-total		$ 89.19			$ 89.19	$ 89.19	$ 89.19	$ 89.19	$ 89.19	$ 89.19
Total Production Costs					**$ 15,340,680**	**$ 21,405,600**	**$ 21,405,600**	**$ 21,405,600**	**$ 1,605,420**	**$ 81,162,900**
Total CAPEX & Production Costs			**$ 3,603,000**	**$ 10,939,800**	**$ 18,075,630**	**$ 21,405,600**	**$ 21,405,600**	**$ 22,905,600**	**$ 1,605,420**	**$ 99,940,650**
Revenues										
Milling rate - st/month	910,000	20,000			172,000	240,000	240,000	240,000	18,000	910,000
Diluted average grade of all Zones - opt		0.197			0.197	0.197	0.197	0.197	0.197	0.197
Milled recovery - %		0.96			0.96	0.96	0.96	0.96	0.96	0.96
Ounces produced					32,529	45,389	45,389	45,389	3,404	172,099
Gross revenue at $CDN/oz		$ 660.00		0	**$ 21,468,902**	**$ 29,956,608**	**$ 29,956,608**	**$ 29,956,608**	**$ 2,246,746**	**$ 113,585,472**
Gross profit (loss) before royalties			-$ 3,603,000	-$ 10,939,800	$ 3,393,272	$ 8,551,008	$ 8,551,008	$ 7,051,008	$ 641,326	$ 13,644,822
Ore NSR royalty - 3 %					$ 644,067	$ 898,698	$ 898,698	$ 898,698	$ 67,402	$ 3,407,564
Milling NSR royalty - 1%					$ 214,689	$ 299,566	$ 299,566	$ 299,566	$ 22,467	$ 1,135,855
(BACHELOR LAKE MINE) EBITDA*			**-$ 3,603,000**	**-$ 10,939,800**	**$ 2,534,516**	**$ 7,352,744**	**$ 7,352,744**	**$ 5,852,744**	**$ 551,456**	**$ 9,101,403**
Bachelor - Cumulative Cash Flow			-$ 3,603,000	-$ 14,542,800	-$ 12,008,284	-$ 4,655,540	$ 2,697,204	$ 8,549,947	$ 9,101,403	
METANOR ANNUAL CASH FLOW			**-$ 3,238,159**	**-$ 7,986,001**	**$ 2,824,858**	**$ 7,352,744**	**$ 7,352,744**	**$ 5,852,744**	**$ 551,456**	**$ 12,710,385**
METANOR CUMULATIVE CASH FLOW	-$ 6,208,000		-$ 9,446,159	-$ 17,432,160	-$ 14,607,301	-$ 7,254,558	$ 98,186	$ 5,950,930	**$ 6,502,385**	
Discounted Metanor Cash Flow at 10%										$ 137,442
Discounted Metanor Cash Flow at 7.5%										$ 1,353,258
Discounted Metanor Cash Flow at 5.0%										$ 2,789,439

Economic Analysis Results

As shown in the following table the exploitation of the two properties is generating a **Net Cash Flow of $6,502,385 for the expected 73 months of operation**. This Cash Flow is shown as EBITDA, (Estimated Benefit Before Tax Depreciation & Amortization), in other words this is a Pre-Tax Undiscounted Cash Flow.

The situation at the end of this period will leave Metanor Resources Inc with two properties that most likely will not be exhausted, plus a running concentrator.

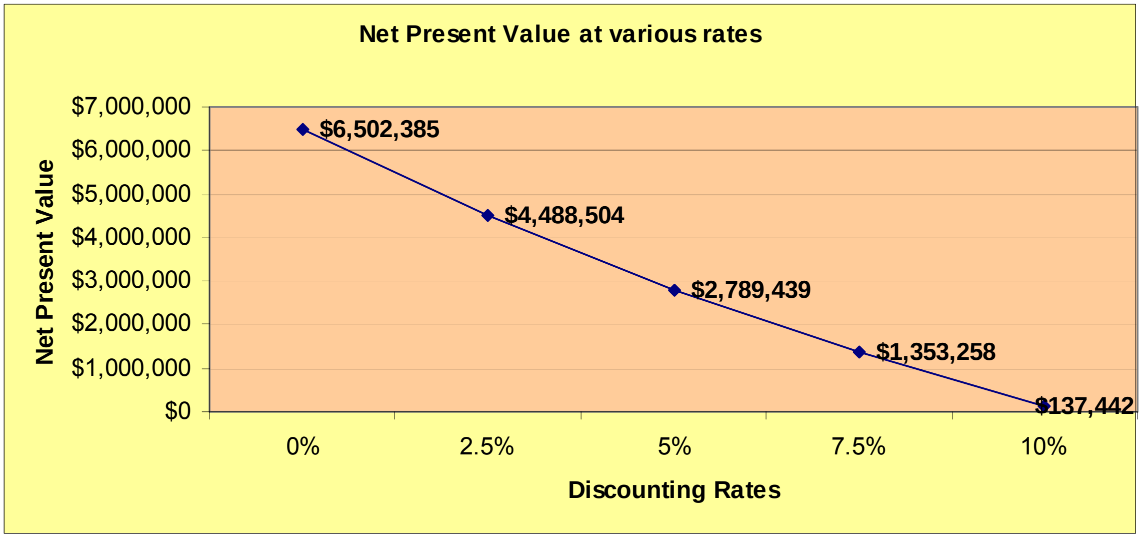
It is also important to note that no salvage values have been given to the assets in the Cash Flow estimate.

Discounted Cash Flow

The effect of discounting the Base Case result is illustrated in the following table.

Undiscounted Cash Flow	$	6,502,385
Discounted at 2.5 %	$	4,488,504
Discounted at 5.0 %	$	2,789,439
Discounted at 7.5%	$	1,353,258
Discounted at 10%	$	137,442

The same variation is shown in graphic mode



Benefit of the Actual Tax Regime

The preliminary Cash Flow has been prepared without any tax credit or fiscal taxation advantages.

In accordance with the Tax credit for resources documentation of the province of Quebec, both properties could benefit from the provisions of the actual mining taxation regime.

In the case of Barry-1 property, the advantage is related to the tax credit of the cost of the treatment of a bulk sample needed to better define the metallurgical characteristics of the ore before going into production.

At the Bachelor Lake Mine property the estimated resources have to be evaluated economically and technically. A major part of these resources is located below the existing shat bottom that has to be deepened by 675 ft to provide accesses to them. These development expenses qualify for tax credit.

Conclusions and Recommendations

- **Following the positive results of the estimated Cash Flow, Geostat is recommending to the owners to advance the properties in the direction of a commercial production**.
- Geostat also recommends that Metanor should proceeds with a pre-feasibility study in order to confirm our recommendation.
- Before proceeding to the next pre-feasibility phase, Geostat recommends that Metanor should prepare the followings:

At the Barry property

1. Better evaluate the full economic benefit of the treatment of a bulk sample

2. Define the cost saving resulting from the ore crushing at Barry-1 before sending it to Bachelor; in our Cash Flow no cost reduction has been applied. to that operation

3. Reassess the economic impact of the royalties, specially those of the Barry-1 property

4. Perform additional fill-in drilling to better define the known mineralized zones

5. Explore the surrondings of the proposed open-pit to avoid stockpiling waste or overburden over possible mineralized areas

6. Complete the survey of the topography and all the drill holes that have not already been surveyed in the area of the Barry-1 property

7. Realize a detailed new description of some of the old drill core to better understand the correlation between the mineralized envelopes and the geology of Barry.

8. Continue the exploration around the proposed East Pit and West Zone where the presence of mineralized zones could add resources to the exiting ones.

The costs of the recommended works at Barry-1 before the pre feasibility study are summarised below

Barry Property

Description			Cost
Ore definition at the Bulk Sample area - lump sum			$35,000
Exploration under the stockpiling areas	500 m	$120/m	$60,000
Resources in-fill drilling	2,000 m	$120/m	$240,000
General expansing drilling	2 000 m	$120/m	$240,000
Pre-feasibility study - lump sum			$150,000
Bulk sample exploitation			$2,900,000
Total			**$3,625,000**

At the Bachelor property

1. Proceed to replace the existing hoist

2. Initiate the shaft deepening to give acccess to the ore portion that is below the twelve level.

3. Proceed to the development of the proposed ore undercuts to have a full understanding of the geology and to assay the mineralized zones

4. Realize an infill drilling program estimated to 20,000 ft

The total of all these recommended workings before the pre feasibility study is shown below.

Bachelor Property

Description			Cost
Hoist & headframe repair			$ 1,020,750
Shaft sinking with services - 675 ft			$ 9,196,150
Excavation of undercuts (50% of all level developments)	4,000 ft	$356/ft	$ 1,424,000
In-fill drilling, c/w assays	20,000 ft	$35/ft	$ 700,000
Pre-feasibility study - lump sum			$ 150,000
Total			**$ 12,490,900**

1. Introduction (Item 4)

Geostat reminds that this Preliminary Assessment is preliminary in nature, that it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. This report is in accordance to National Instrument 43-101

This report is prepared for Metanor Resources Inc. of Val D'or QC. Geostat Systems International Inc. has received a mandate from Metanor to prepare a preliminary economic assessment report of the possible exploitation of the Barry-1 and the Bachelor Lake Mine properties.

The Barry-1 resources are suitable to be exploited by open pit, or a combination of underground and open pit, as the ore is exposed at surface. The Bachelor Lake Mine resources are located underground and are in the extensions of the former exploitation that took place from 1982 to 1989 and produced 131,029 ounces of gold from 958,368 short tons at 0.147 opt

The main sources of information for the Barry-1 property are the NI 43-101 Technical Report of the Resources Evaluation, prepared by Geostat in April 2007 and other relevant data supplied by Metanor Resouces during the preparation.

For the Bachelor Lake Mine Property the main source of information is the NI 43-101 Technical Report on the 2005 drilling program and Mineral Resources estimate for the Bachelor Lake property prepared by InnovExplo in December 2205 and general informations transmitted by Metanor Resources.

The Barry-1 and Bachelor Lake properties were visited on May 14-15, 2007 by Gaston Gagnon and Yann Camus two Geosat's engineers. The visit included the Barry-1 exposed mineralization site and the surroundings and all of Bachelor Lake Mine installations, mainly the hoisting facilities, two underground levels, the concentrator, and some of the drilled core.

Gilbert Rousseau, a consultant engineer hired by Geostat visited the concentrator, the crusher house and the tailings pond on May 14, 2007.

The following terms are used in this study:

Metanor: Metanor Resources Inc. of Val d'or, Qc
Geostat: Systèms Géostat International Inc, of Blainville Qc
Barry: Barry-1 Main Zone-1
Bachelor: Bachelor Lake Mine

1.1 List of abbreviations

In this report, monetary units are in Canadian dollars (CA$) unless when specified in United States dollars (US$). The metric and imperial system of measurements and units are used throughout the report except for the gold quantities, which are reported in Troy ounces.

A table showing abbreviations used in this report is provided below.

tonnes or mt	Metric tonnes
tpd	Tonnes per day
tons	Short tons (0.907185 tonnes)
kg	Kilograms
oz	Troy ounce (31.1035 grams)
g/t	Grams/tonne or ppm
ppm, ppb	Parts per million, parts per billion
ha	Hectares
m	Metres
km	Kilometres
m^3	Cubic metres
opt	ounces per ton

Table 1: List of abbreviations.

2. Reliance on Other Experts (Item 5)

The status of the Barry and Bachelor Property mining titles, i.e. claims and mining concessions have been obtained from GESTMIN, the official site of the Ministère des Ressources naturelles, de la Faune et des Parcs.

The references of the InnovExplo Technical Report of December 2005 for the Bachelor Lake Property and those of the April 2007 Technical Report on Resources Evaluation of the Barry-1 Property are used as mentioned in each title.

3. Property description and location (Item 6)

This following information is part of the April 2007 Technical Report on Resources Evaluation of the Barry-1 Property by Geostat.

3.1 Barry-1 property

This following information is part of the April 2007 Technical Report on Resources Evaluation of the Barry-1 Property by Geostat.

The Barry property is located 100 km east of Lebel-sur-Quevillon and 180 km southwest of Chibougamau, Abitibi, in the Barry Township. The Barry property is centred on UTM coordinates 443,690E and 5,426,450N (UTM-18, NAD 83) on the topographic map (NTS 32 B/13). The property is located south of the Municipalité de la Baie James border. It is not under the jurisdiction of the different agreements associated with this municipality.

Figure 3.1: Location of the Barry property.

The Barry I claims block (Next Figure), located in the centre part of the Barry property, consists of 7 claims.

The SDBJ owns 6 claims. Due to a bureaucratic constraint, Murgor and the SDBJ have an agreement where Murgor gets an exclusive and irrevocable interest of one hundred percent in the mineral substances that can be extracted from the property. Metanor bought 100% of the rights from Murgor.

For the last Claim, Metanor bought 100% of the rights from Murgor and Freewest.

Following the terms of the agreement between Murgor and Metanor, the counterpart payable for the acquisition of this property includes a $200,000 cash payment on January 15th, 2007 as well as a Royalty on the proceeds of sales of gold produced from the property. This royalty is established to 9 % of the sale's price of gold produced. A first advance on this royalty of production forthcoming, corresponding to $250,000 will be payable to Murgor by the issuing of 416,666 common shares of Metanor, as soon as this transaction will be accepted by the regulatory authorities. A second advance of $250,000 cash will be payable, on the first of the two dates hereinafter mentioned: 30 days after the issuance of the exploitation permit or on January 1st, 2008. The reimbursement of the advances will be made by a reduction of 50% of the amounts of royalty due to Murgor, after the beginning of the commercial exploitation of the Barry deposit.

The agreement is subject to the approval of regulatory authorities.

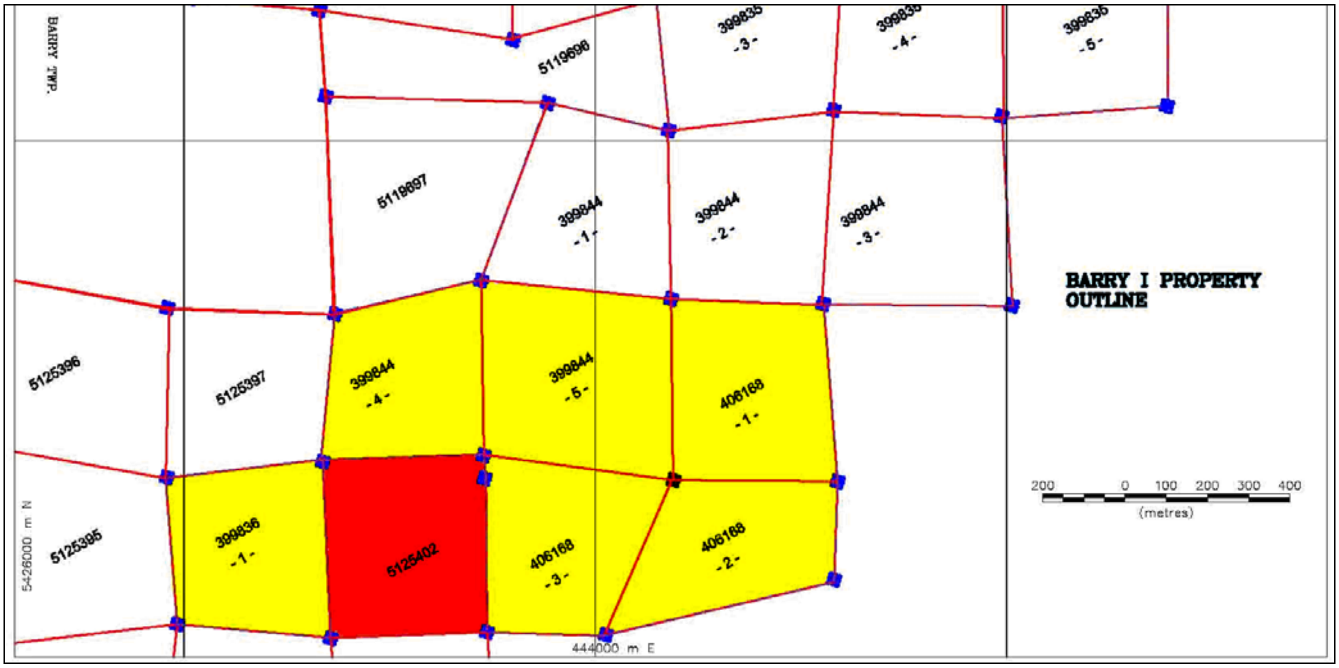


Figure 3.2: Claim map of the Barry I property with claims aquired from Murgor in yellow and claims aquired from Murgor and Freewest in red.

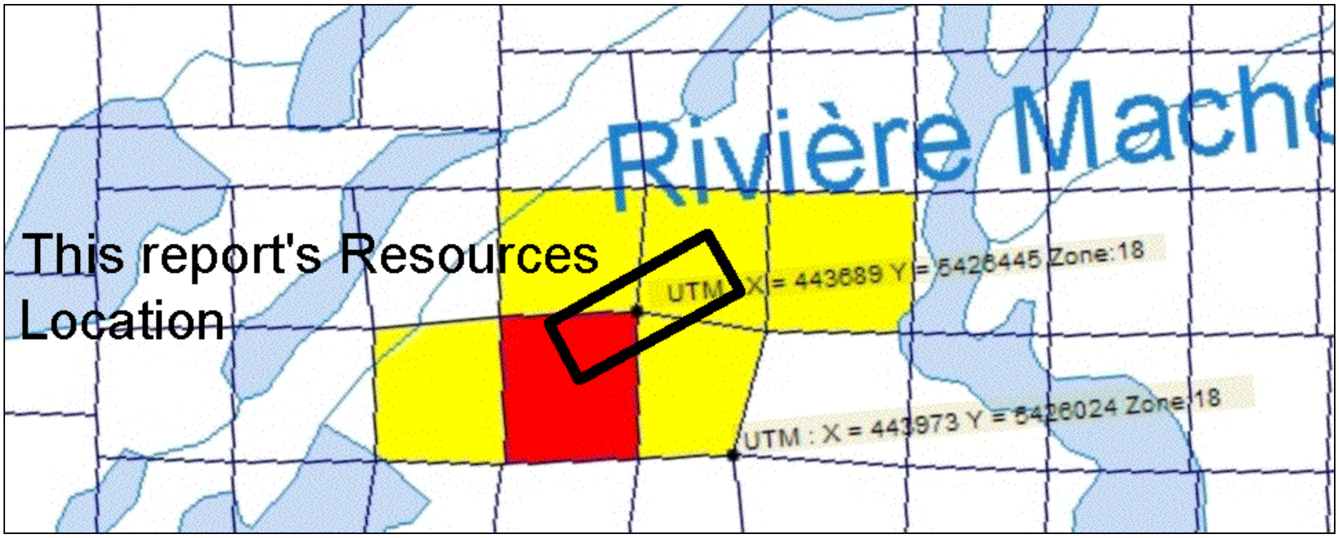


Figure 3.3: Claim map of the Barry I property with location of the resources

Metanor owns sufficient surface rights for the development of the Barry I Main Zone. Operational permits and environmental authorization certificates are required for the mining of the open pit but it is expected that these approvals will be obtained normally when needed.

Mining titles of the Barry I claim block

# Claims	Title holder	Expiration date	Area of the claim in the property (ha)
399836-1	SDBJ	2009/05/31	15.69
399844-4	SDBJ	2009/05/29	15.14
399844-5	SDBJ	2009/05/29	16
406168-1	SDBJ	2008/11/02	16
406168-3	SDBJ	2008/11/02	16
406168-2	SDBJ	2008/11/02	16
512540-2	Murgor	2008/05/31	16.75
Total: 7 claims			**111.58 ha**

Table 3.1: Mining titles of the Barry I claim block.

3.2 Bachelor Lake Property

Most of the informations of this item are extracted from the InnovExplo Technical Report of December 2005 for the Bachelor Lake Property and have been updated.

Location, claims status and royalties

The Bachelor Property was formerly owned by BLJV, a joint venture in which Metanor Resources and Halo Resources each had a 50% interest in The Bachelor Lake Property, the Hewfran Property and the MJL-Hansen Property (collectively “Bachelor Property”)

In November 2006, Metanor Resources signed an agreement to purchase the 50% interest of Halo Resources. When the terms of this agreement will be fulfilled, Metanor will own 100% of the Bachelor Lake Property. Metanor has the complete financial capacity needed to realize the acquisition of the Bachelor lake property.(ref: Press release November 20, 2006)

The property is located within the Abitibi Greenstone Belt (Northwestern Québec, Canada) in the Township of Le Sueur (CL740), approximately 225 km north of the town of Val-d’Or. It lies within the NTS sheets 32F08 and 32F09, and the Bachelor headframe is located at the latitude of 76º 8' 78" North and longitude 49º 29' 56" West. The mine site is situated 3.5 km east of the village of Desmaraisville and 30 km south of the Cree community of Waswanipi.

Desmaraisville is serviced by bus and truck transport, and is connected to the 113 Provincial highway, railroad, power grid and telecommunication systems. An experienced labour force in the mining industry is available within a 240-km radius of the project site (Val-d’Or, Lebel-sur-Quevillon, Chapais, Chibougamau). Val-d’Or is a major full service centre for exploration, mining and economic activity in Northwestern Québec.

The known resources extension limits and the headframe of the Bachelor property of Metanor Resources Inc has been located on the claims map and is shown under figure 3.4

As seen on that figure the resources extension and the surface installations are all within a group of 9 claims and one mining concession having a total area of 184.73 Ha. The list of these Mining Titles is reproduced under the table 3.1.

Surrounding that claim group, Metanor Resources Inc has 168 additional claims and another mining concession, all contiguous, for a total of 4653.12 Ha. The listing of these Mining Titles is shown in the table 3.2

The validity of Mining Titles has been verified at the official site of the Quebec government using GESTMIN, on May 27th 2007.

Royalties

The royalties applying to the Bachelor Property are all of the NSR type and will apply on the mining and milling of the ore, they are summarized in the following

Summary of milling and mining royalties

Milling Royalties				
On all ore that will be shipped and milled at the Bachelor mill and belongs				
to Metanor (Bachelor, Hewfran, Barry)				
	NSR	USD	Price of gold	USD
	0.25%	425$	Au	450$
	0.50%	450$	Au	485$
	0.75%	485$	Au	560$
	1.00%	Over 560$/oz		
Payable to CONCOPPER Enterprise (Maximum of 1.75M$)				
Ore Royalties				
Bachelor ore	**NSR**	USD	Price of gold	USD
	0.50%	425$	Au	450$
	1.00%	450$	Au	485$
	1.50%	485$	Au	560$
	2.00%	Over 560$/oz		
	Payable to CONCOPPER Entreprise, and			
	1% payable to Halo Resources			
Hewfran ore	**NSR**			
	2%	Payable to AUR Resources		
	1%	Payable to HALO Resources		

Table 3.2: Royalties applying to the ore milled and mined by Metanor

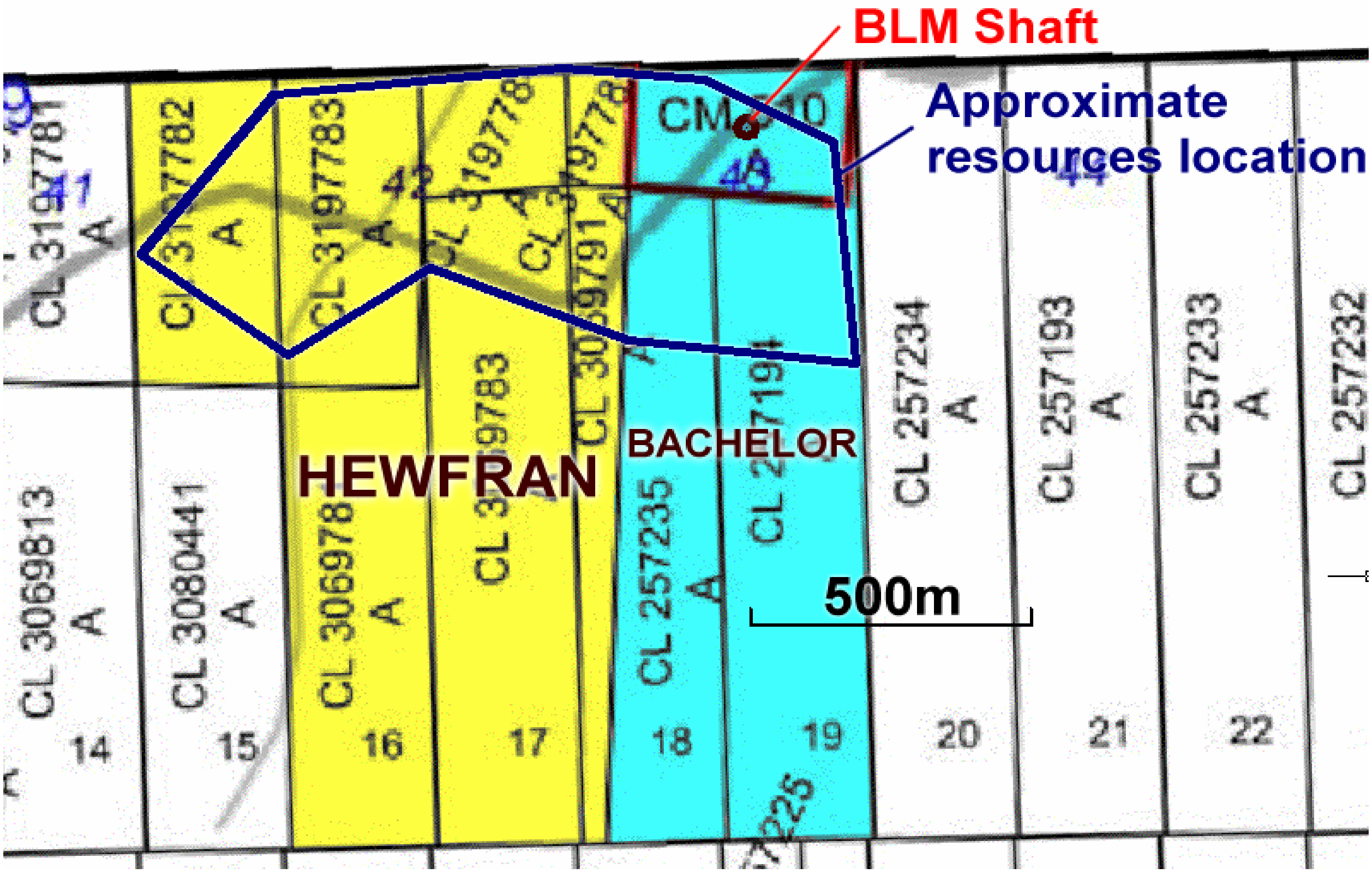


Figure 3.4: Bachelor Claims map showing the known resources extension limit

Bachelor Lake Property - List of Claims

#	NTS Sheet	Type of Title	Title No	Status	Date of Staking	Registration Date	Expiry Date	Annual Terms	Renewals	Area (Ha)	Intruments	Excess Work	Required Work	Required Fees	Titleholder(s)	Past Property
1	NTS 32F08	CL	3069781	Active	1970-05-15	1970-06-03	2009-05-14	0	9	20	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
2	NTS 32F08	CL	3069783	Active	1970-05-15	1970-06-03	2009-05-14	0	9	30	Yes	0	2500	50	RESSOURCES AUR INC	Hewfran
3	NTS 32F08	CL	3069791	Active	1970-05-15	1970-06-03	2009-05-14	0	9	10	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
4	NTS 32F08,32F09	CL	3197782	Active	1971-10-15	1971-12-30	2009-01-07	0	9	17.2	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
5	NTS 32F08,32F09	CL	3197783	Active	1971-10-15	1971-12-30	2009-01-07	0	9	20	Yes	636,734	1000	25	RESSOURCES AUR INC	Hewfran
6	NTS 32F08,32F09	CL	3197784	Active	1971-10-15	1971-12-30	2009-01-07	0	9	10	Yes	185,896	1000	25	RESSOURCES AUR INC	Hewfran
7	NTS 32F08,32F09	CL	3197785	Active	1971-10-15	1971-12-30	2009-01-07	0	9	4.8	Yes	472,096	1000	25	RESSOURCES AUR INC	Hewfran
									Area	112.00						
1	NTS 32F08,32F09	CM	510	Active		1964-04-13		17	0	16.13	Yes		35		RESSOURCES METANOR INC. (20103) 100 % (responsible)	
									Area	16.13						
1	NTS 32F08,32F09	CL	257194	Active	1946-09-21	1946-10-17	2007-09-20	0	8	29.6	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)	
2	NTS 32F08	CL	257235	Active	1946-09-23	1946-10-17	2007-09-20	0	8	27	Yes	54573,27	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)	
									Area	56.6						
									Total Area	**184.73**						

Table 3.5: Bachelor Claims List showing the known resources extension

Bachelor Lake Property - List of Claims

#	NTS Sheet	Type of Title	Title No	Status	Date of Staking	Date of Registration	Expiry Date	Annual Terms	Number of Renewals	Area (Ha)	Intruments	Excess Work	Required Work	Required Fees	Titleholder(s)	Past Property
1	NTS 32F09	CL	3011591	Active	1970-06-16	1970-07-06	2009-06-15	0	9	33.2	Yes	0	2500	50	RESSOURCES AUR INC	Hewfran
2	NTS 32F09	CL	3011592	Active	1970-06-16	1970-07-06	2009-06-15	0	9	34	Yes	0	2500	50	RESSOURCES AUR INC	Hewfran
3	NTS 32F08	CL	3069812	Active	1970-05-16	1970-06-03	2009-05-14	0	9	4.4	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
4	NTS 32F08	CL	3069813	Active	1970-05-16	1970-06-03	2009-05-14	0	9	22	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
5	NTS 32F08	CL	3080441	Active	1970-05-15	1970-06-03	2009-05-14	0	9	20	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
6	NTS 32F09	CL	3083911	Active	1970-06-16	1970-07-06	2009-06-15	0	9	34.4	Yes	0	2500	50	RESSOURCES AUR INC	Hewfran
7	NTS 32F09	CL	3083912	Active	1970-06-16	1970-07-06	2009-06-15	0	9	9.6	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
8	NTS 32F09	CL	3083913	Active	1970-06-16	1970-07-06	2009-06-15	0	9	9.6	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
9	NTS 32F09	CL	3083914	Active	1970-06-16	1970-07-06	2009-06-15	0	9	4.8	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
10	NTS 32F09	CL	3083921	Active	1970-06-16	1970-07-06	2009-06-15	0	9	32.8	Yes	0	2500	50	RESSOURCES AUR INC	Hewfran
11	NTS 32F09	CL	3083922	Active	1970-06-16	1970-07-06	2009-06-15	0	9	12.4	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
12	NTS 32F09	CL	3083923	Active	1970-06-16	1970-07-06	2009-06-15	0	9	12	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
13	NTS 32F09	CL	3083924	Active	1970-06-16	1970-07-06	2009-06-15	0	9	11.6	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
14	NTS 32F09	CL	3083925	Active	1970-06-16	1970-07-06	2009-06-15	0	9	11.6	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
15	NTS 32F09	CL	3087671	Active	1970-08-19	1970-09-08	2009-08-18	0	9	34.4	Yes	0	2500	50	RESSOURCES AUR INC	Hewfran
16	NTS 32F09	CL	3087672	Active	1970-08-19	1970-09-08	2009-08-18	0	9	34.4	Yes	0	2500	50	RESSOURCES AUR INC	Hewfran
17	NTS 32F09	CL	3087681	Active	1970-08-19	1970-09-08	2009-08-18	0	9	34.4	Yes	0	2500	50	RESSOURCES AUR INC	Hewfran
18	NTS 32F09	CL	3087682	Active	1970-08-19	1970-09-08	2009-08-18	0	9	34.4	Yes	0	2500	50	RESSOURCES AUR INC	Hewfran
19	NTS 32F09	CL	3145911	Active	1971-02-09	1971-04-13	2009-02-08	0	9	3.2	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
20	NTS 32F09	CL	3145912	Active	1971-02-09	1971-04-13	2009-02-08	0	9	2.7	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
21	NTS 32F08,32F09	CL	3197781	Active	1971-10-15	1971-12-30	2009-01-07	0	9	14.8	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
22	NTS 32F08,32F09	CL	3197791	Active	1971-11-01	1971-11-18	2009-01-07	0	9	0.8	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
23	NTS 32F08,32F09	CL	3252153	Active	1972-10-17	1972-11-06	2007-10-16	0	8	35.6	Yes	0	2500	50	RESSOURCES AUR INC	Hewfran
24	NTS 32F08,32F09	CL	3252154	Active	1972-10-17	1972-11-06	2007-10-16	0	8	18.8	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
25	NTS 32F08,32F09	CL	3252155	Active	1972-10-17	1972-11-06	2007-10-16	0	8	9.6	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
26	NTS 32F09	CL	3645604	Active	1977-02-10	1977-03-03	2009-02-09	0	9	16	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
27	NTS 32F09	CL	3645605	Active	1977-02-10	1977-03-03	2009-02-09	0	9	16	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
28	NTS 32F09	CL	3645611	Active	1977-02-10	1977-03-03	2009-02-09	0	9	16	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
29	NTS 32F09	CL	3645694	Active	1977-02-12	1977-03-03	2009-02-11	0	9	16	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
30	NTS 32F09	CL	3645762	Active	1977-02-09	1977-03-03	2009-02-08	0	9	16	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
31	NTS 32F09	CL	3645873	Active	1977-02-07	1977-03-03	2009-02-06	0	9	16	Yes	0	1000	25	RESSOURCES AUR INC	Hewfran
									Total area	**571.50**						

BACHELOR LAKE PROPERTY - List of Claims

#	NTS Sheet	Type of Title	Title No	Status	Date of Staking	Registration	Expiry Date	Annual Terms	Renewals	Area (Ha)	Intruments	Excess Work	Required Work	Required Fees	Titleholder(s)
1	NTS 32F09	CDC	3206	Active		2003-09-15	2007-09-14	0	1	42.32	No	1774,43	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
2	NTS 32F09	CDC	3207	Active		2003-09-15	2007-09-14	0	1	42.30	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
3	NTS 32F08	CDC	3208	Active		2003-09-15	2007-09-14	0	1	27.51	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
4	NTS 32F08	CDC	3209	Active		2003-09-15	2007-09-14	0	1	22.36	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
5	NTS 32F08	CDC	3210	Active		2003-09-15	2007-09-14	0	1	17.87	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
6	NTS 32F08	CDC	3211	Active		2003-09-15	2007-09-14	0	1	15.80	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
7	NTS 32F08	CDC	3212	Active		2003-09-15	2007-09-14	0	1	9.92	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
8	NTS 32F08	CDC	3213	Active		2003-09-15	2007-09-14	0	1	19.96	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
9	NTS 32F08	CDC	3214	Active		2003-09-15	2007-09-14	0	1	27.08	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
10	NTS 32F08	CDC	3215	Active		2003-09-15	2007-09-14	0	1	29.10	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
11	NTS 32F08	CDC	3216	Active		2003-09-15	2007-09-14	0	1	25.20	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
12	NTS 32F08	CDC	3217	Active		2003-09-15	2007-09-14	0	1	31.91	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
13	NTS 32F08	CDC	3270	Active		2003-09-15	2007-09-14	0	1	31.44	No	1774,42	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
14	NTS 32F08	CDC	3271	Active		2003-09-15	2007-09-14	0	1	42.31	No	1774,42	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
15	NTS 32F08	CDC	3272	Active		2003-09-15	2007-09-14	0	1	42.31	No	1774,42	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
16	NTS 32F08	CDC	3273	Active		2003-09-15	2007-09-14	0	1	42.31	No	1774,42	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
17	NTS 32F08	CDC	3274	Active		2003-09-15	2007-09-14	0	1	42.32	No	1774,42	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
18	NTS 32F08	CDC	3275	Active		2003-09-15	2007-09-14	0	1	42.32	No	1774,42	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
19	NTS 32F08	CDC	3276	Active		2003-09-15	2007-09-14	0	1	42.32	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
20	NTS 32F08	CDC	3277	Active		2003-09-15	2007-09-14	0	1	42.32	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
21	NTS 32F08	CDC	3278	Active		2003-09-15	2007-09-14	0	1	42.32	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
22	NTS 32F08	CDC	3279	Active		2003-09-15	2007-09-14	0	1	43.02	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
23	NTS 32F08	CDC	3280	Active		2003-09-15	2007-09-14	0	1	43.00	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
24	NTS 32F08	CDC	3281	Active		2003-09-15	2007-09-14	0	1	42.98	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
25	NTS 32F08	CDC	3282	Active		2003-09-15	2007-09-14	0	1	10.32	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
26	NTS 32F08,32F09	CDC	3283	Active		2003-09-15	2007-09-14	0	1	41.85	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
27	NTS 32F08,32F09	CDC	3284	Active		2003-09-15	2007-09-14	0	1	38.35	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
28	NTS 32F08	CDC	3285	Active		2003-09-15	2007-09-14	0	1	55.96	No	1774,42	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
29	NTS 32F08	CDC	3286	Active		2003-09-15	2007-09-14	0	1	55.96	No	1774,42	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
30	NTS 32F08	CDC	3287	Active		2003-09-15	2007-09-14	0	1	40.52	No	1774,42	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
31	NTS 32F08	CDC	3288	Active		2003-09-15	2007-09-14	0	1	55.96	No	1774,43	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
32	NTS 32F08	CDC	3289	Active		2003-09-15	2007-09-14	0	1	55.96	No	1774,43	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
33	NTS 32F08	CDC	3290	Active		2003-09-15	2007-09-14	0	1	40.44	No	1774,43	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
34	NTS 32F09	CDC	3291	Active		2003-09-15	2007-09-14	0	1	15.63	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
35	NTS 32F09	CDC	3292	Active		2003-09-15	2007-09-14	0	1	15.64	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)

BACHELOR LAKE PROPERTY - List of Claims

#	NTS Sheet	Type of Title	Title No	Status	Date of Staking	Registration	Expiry Date	Annual Terms	Renewals	Area (Ha)	Intruments	Excess Work	Required Work	Required Fees	Titleholder(s)
36	NTS 32F09	CDC	3293	Active		2003-09-15	2007-09-14	0	1	36.69	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
37	NTS 32F09	CDC	3294	Active		2003-09-15	2007-09-14	0	1	36.69	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
38	NTS 32F09	CDC	3295	Active		2003-09-15	2007-09-14	0	1	42.58	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
39	NTS 32F09	CDC	3296	Active		2003-09-15	2007-09-14	0	1	41.84	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
40	NTS 32F08	CDC	8367	Active		2003-12-09	2007-12-08	0	1	18.70	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
41	NTS 32F08	CDC	8368	Active		2003-12-09	2007-12-08	0	1	19.22	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
42	NTS 32F09	CDC	13637	Active		2004-02-17	2008-02-16	0	1	16.59	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
43	NTS 32F09	CDC	13638	Active		2004-02-17	2008-02-16	0	1	17.24	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
44	NTS 32F09	CDC	13639	Active		2004-02-17	2008-02-16	0	1	53.29	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
45	NTS 32F09	CDC	13640	Active		2004-02-17	2008-02-16	0	1	55.89	No	0	1200	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
46	NTS 32F09	CDC	13641	Active		2004-02-17	2008-02-16	0	1	23.16	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
47	NTS 32F09	CDC	13642	Active		2004-02-17	2008-02-16	0	1	24.44	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
48	NTS 32F08	CDC	15481	Active		2004-03-05	2008-03-04	0	1	23.46	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
49	NTS 32F08,32F09	CL	257181	Active	1946-09-23	1946-10-17	2007-09-22	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
50	NTS 32F09	CL	257182	Active	1946-09-23	1946-10-17	2007-09-22	0	8	29.60	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
51	NTS 32F09	CL	257183	Active	1946-09-23	1946-10-17	2007-09-22	0	8	29.60	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
52	NTS 32F09	CL	257184	Active	1946-09-23	1946-10-17	2007-09-22	0	8	27.50	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
53	NTS 32F09	CL	257185	Active	1946-09-23	1946-10-17	2007-09-20	0	8	16.90	Yes	0	1000	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
54	NTS 32F08,32F09	CL	257191	Active	1946-09-21	1946-10-17	2007-09-20	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
55	NTS 32F08,32F09	CL	257192	Active	1946-09-21	1946-10-17	2007-09-20	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
56	NTS 32F08,32F09	CL	257193	Active	1946-09-21	1946-10-17	2007-09-20	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
57	NTS 32F08	CL	257205	Active	1946-09-24	1946-10-17	2007-09-20	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
58	NTS 32F08	CL	257225	Active	1946-10-03	1946-10-17	2007-09-20	0	8	1.40	Yes	981795,7	1000	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
59	NTS 32F08,32F09	CL	257231	Active	1946-09-23	1946-10-17	2007-09-22	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
60	NTS 32F08,32F09	CL	257232	Active	1946-09-23	1946-10-17	2007-09-22	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
61	NTS 32F08,32F09	CL	257233	Active	1946-09-23	1946-10-17	2007-09-22	0	8	40.00	Yes	43102,45	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
62	NTS 32F08,32F09	CL	257234	Active	1946-09-23	1946-10-17	2007-09-22	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
63	NTS 32F09	CL	267941	Active	1946-10-06	1946-10-19	2007-10-05	0	8	18.40	Yes	0	1000	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
64	NTS 32F09	CL	267942	Active	1946-10-06	1946-10-19	2007-10-05	0	8	17.60	Yes	0	1000	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
65	NTS 32F09	CL	267955	Active	1946-10-06	1946-10-19	2007-10-05	0	8	20.00	Yes	0	1000	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
66	NTS 32F08	CL	3734191	Active	1978-11-11	1978-12-04	2008-11-10	0	9	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
67	NTS 32F08	CL	3734192	Active	1978-11-11	1978-12-04	2008-11-10	0	9	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
68	NTS 32F08	CL	3734201	Active	1978-11-11	1978-12-04	2008-11-10	0	9	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
69	NTS 32F08	CL	3734202	Active	1978-11-11	1978-12-04	2008-11-10	0	9	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
70	NTS 32F08	CL	3734211	Active	1978-11-11	1978-12-04	2008-11-10	0	9	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)

BACHELOR LAKE PROPERTY - List of Claims

#	NTS Sheet	Type of Title	Title No	Status	Date of Staking	Registration	Expiry Date	Annual Terms	Renewals	Area (Ha)	Intruments	Excess Work	Required Work	Required Fees	Titleholder(s)
71	NTS 32F08	CL	3734212	Active	1978-11-12	1978-12-04	2008-11-11	0	9	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
72	NTS 32F08	CL	3734221	Active	1978-11-12	1978-12-04	2008-11-11	0	9	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
73	NTS 32F08	CL	3742541	Active	1978-09-16	1978-10-10	2007-09-15	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
74	NTS 32F08	CL	3742542	Active	1978-09-16	1978-10-10	2007-09-15	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
75	NTS 32F08	CL	3742551	Active	1978-09-16	1978-10-10	2007-09-15	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
76	NTS 32F08	CL	3742552	Active	1978-09-16	1978-10-10	2007-09-15	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
77	NTS 32F08	CL	3742561	Active	1978-09-17	1978-10-10	2007-09-16	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
78	NTS 32F08	CL	3742562	Active	1978-09-17	1978-10-10	2007-09-16	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
79	NTS 32F08	CL	3742571	Active	1978-09-17	1978-10-10	2007-09-16	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
80	NTS 32F08	CL	3742572	Active	1978-09-17	1978-10-10	2007-09-16	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
81	NTS 32F08	CL	3742581	Active	1978-09-18	1978-10-10	2007-09-17	0	8	26.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
82	NTS 32F08	CL	3742582	Active	1978-09-18	1978-10-10	2007-09-17	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
83	NTS 32F08	CL	3742583	Active	1978-09-18	1978-10-10	2007-09-17	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
84	NTS 32F08	CL	3742584	Active	1978-09-18	1978-10-10	2007-09-17	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
85	NTS 32F08	CL	3742651	Active	1978-11-06	1978-12-04	2008-11-05	0	9	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
86	NTS 32F08	CL	3742652	Active	1978-11-06	1978-12-04	2008-11-05	0	9	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
87	NTS 32F08	CL	3742661	Active	1978-11-06	1978-12-04	2008-11-05	0	9	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
88	NTS 32F08	CL	3742662	Active	1978-11-06	1978-12-04	2008-11-05	0	9	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
89	NTS 32F08	CL	3742712	Active	1978-09-16	1978-10-10	2007-09-15	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
90	NTS 32F08	CL	3742721	Active	1978-09-16	1978-10-10	2007-09-15	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
91	NTS 32F08	CL	3742722	Active	1978-09-16	1978-10-10	2007-09-15	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
92	NTS 32F08	CL	3742731	Active	1978-09-17	1978-10-10	2007-09-16	0	8	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
93	NTS 32F08	CL	3742771	Active	1978-11-06	1978-12-04	2008-11-05	0	9	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
94	NTS 32F08	CL	3742772	Active	1978-11-06	1978-12-04	2008-11-05	0	9	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
95	NTS 32F08	CL	3742781	Active	1978-11-06	1978-12-04	2008-11-05	0	9	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
96	NTS 32F08	CL	3742782	Active	1978-11-06	1978-12-04	2008-11-05	0	9	40.00	Yes	0	2500	50	RESSOURCES METANOR INC. (20103) 100 % (responsible)
97	NTS 32F08	CL	3812031	Active	1979-03-12	1979-04-02	2009-03-11	0	9	15.00	Yes	0	1000	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
98	NTS 32F09	CL	5268741	Active	2003-07-24	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
99	NTS 32F09	CL	5268742	Active	2003-07-24	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
100	NTS 32F09	CL	5268743	Active	2003-07-24	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
101	NTS 32F09	CL	5268744	Active	2003-07-24	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
102	NTS 32F09	CL	5268745	Active	2003-07-24	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
103	NTS 32F09	CL	5268746	Active	2003-07-25	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
104	NTS 32F09	CL	5268747	Active	2003-07-25	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
105	NTS 32F09	CL	5268748	Active	2003-07-25	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)

BACHELOR LAKE PROPERTY - List of Claims

#	NTS Sheet	Type of Title	Title No	Status	Date of Staking	Registration	Expiry Date	Annual Terms	Renewals	Area (Ha)	Intruments	Excess Work	Required Work	Required Fees	Titleholder(s)
106	NTS 32F09	CL	5268749	Active	2003-07-25	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
107	NTS 32F09	CL	5268750	Active	2003-07-25	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
108	NTS 32F09	CL	5268751	Active	2003-07-26	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
109	NTS 32F09	CL	5268752	Active	2003-07-26	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
110	NTS 32F09	CL	5268753	Active	2003-07-26	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
111	NTS 32F09	CL	5268754	Active	2003-07-26	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
112	NTS 32F09	CL	5268755	Active	2003-07-26	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
113	NTS 32F09	CL	5268756	Active	2003-07-27	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
114	NTS 32F09	CL	5268757	Active	2003-07-27	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
115	NTS 32F09	CL	5268758	Active	2003-07-27	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
116	NTS 32F09	CL	5268759	Active	2003-07-27	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
117	NTS 32F09	CL	5268760	Active	2003-07-27	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
118	NTS 32F09	CL	5268761	Active	2003-07-28	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
119	NTS 32F09	CL	5268762	Active	2003-07-28	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
120	NTS 32F09	CL	5268763	Active	2003-07-28	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
121	NTS 32F09	CL	5268764	Active	2003-07-28	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
122	NTS 32F09	CL	5268765	Active	2003-07-28	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
123	NTS 32F09	CL	5268766	Active	2003-07-29	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
124	NTS 32F09	CL	5268767	Active	2003-07-29	2004-02-20	2008-02-19	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
125	NTS 32F09	CL	5268768	Active	2003-07-29	2004-02-20	2008-02-19	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
126	NTS 32F09	CL	5268769	Active	2003-07-29	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
127	NTS 32F09	CL	5268770	Active	2003-07-29	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
128	NTS 32F09	CL	5268771	Active	2003-07-30	2004-02-20	2008-02-19	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
129	NTS 32F09	CL	5268772	Active	2003-07-30	2004-02-20	2008-02-19	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
130	NTS 32F09	CL	5268773	Active	2003-07-30	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
131	NTS 32F09	CL	5268774	Active	2003-07-30	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
132	NTS 32F09	CL	5268775	Active	2003-07-30	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
133	NTS 32F09	CL	5268776	Active	2003-07-31	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
134	NTS 32F09	CL	5268777	Active	2003-07-31	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
135	NTS 32F09	CL	5268778	Active	2003-07-31	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
136	NTS 32F09	CL	5268779	Active	2003-07-31	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
137	NTS 32F09	CL	5268780	Active	2003-07-31	2003-11-07	2007-11-06	0	1	16.00	No	0	500	25	RESSOURCES METANOR INC. (20103) 100 % (responsible)
1	NTS 32F08	CM	478	Active		1961-02-12				32.94	Yes		35		RESSOURCES METANOR INC. (20103) 100 % (responsible)
									Total Area	**4,081.62**					

Table 3.6: Bachelor List of Claims contiguous to the known resources extension limits

Figure 3.5: Location Map of Properties

3.2.1 Environment

The following item is extracted from the InnovExplo report of December 2005.

Following operation at the Bachelor Lake underground mine, the owner at that time, Espalau Mining, gave to Géospex the mandate to prepare a restoration plan proposal (phase 1) for the Bachelor Lake mining site.

The restoration plan proposal was described in a report dated January 1997 and submitted for approval to the "Service du développement et du milieu miniers" of Quebec government for which no offical response was given because of the "temporary closing" status of the site. However, the property was considered conform in the conclusion of a preliminary report (personal communication with Mr. Louis Marcoux, P.Eng. from this department). This report (February 1998) also mentioned that the "Ministère de l'Environnement" did not add any recommendations.

The mine is now kept dewatered by the owners. The water is pumped from the underground workings into the tailing pond in conformity with the Certificate of Authorization delivered by the "Ministère de l'Environnement du Québec" on June 28th, 2004 (personal communication with Louis Jalbert from the "Ministère de l'Environnement du Québec", in Quebec city). Exploration activities on the property conform to the Québec regulations.

No change in this situation was reported or observed during the site visit in May 2007.

4. Accessibility, Climate, Local Resources, Infrastructures and Physiography (Item 7)

This following information is part of the April 2007 Technical Report on Resources Evaluation of the Barry-1 Property by Geostat.

4.1 Barry-1 Property

Most of the following information is from the 2007 NI43-101 Report from M. Yann Camus Eng from Geostat Systems Int'l Inc.

Accessibility

The Barry I Area project is easily accessible by the provincial paved highway 113, a major regional road linking the town of Senneterre to Chapais, and by a 120 km all-weather gravel road linking the property to the town of Lebel-sur-Quevillon. Many forest roads give access to the different sectors of the property. This property is located 65 km to the south of the Bachelor Lake Mine and a good quality gravel road exists between the two sites.

The topography is generally flat; the bedrock is covered by a relatively thin layer of till, and, in the majority of the surface property, by fir trees and black spruces. The thickness of the overburden varies between zero in the area already stripped to 30 metres. Only a few natural outcrops are present on the property.

Climate

The climatologic data used to characterize the sector under study comes from the meteorological station of Chapais, Quebec. These observations were carried out during 1961-1990.

Precipitation

On average 919 mm of water falls annually in the area. The most abundant precipitation falls in September, with 120 mm of water. Average monthly precipitation ranges from 40 mm in February to 120 mm in September.

It is in June, however, that the strongest precipitation, for one 24-hours period, was registered with 101 mm of water. Snow falls from October to April, but is much more significant from November to March. The average for these five months is 23 mm, expressed in mm of water. The pH of the precipitation measured at the Joutel station in 1991 varies from 4.30 in November to 4.78 in June (MEF, 1993).

Temperature

In the area of Chapais, the average daily temperature is slightly over the freezing point, i.e. 0.5°C. The average temperature during July reaches 16°C, while the temperature in January falls to -17°C.

Winds

The anemometric data collected in Val d'Or between 1952 and 1980 show that from June to January the southwest winds are dominant, whereas from February to May the winds coming from the northwest are more frequent. Furthermore, in this sector, the winds have an average velocity varying between 11 and 14 km/h for an average of 13 km/h during the year.

Local Resources

The regional resources concerning labour force, supplies and equipment are sufficient, the area being well served by geological and mining service firms. The closest town, Lebel-sur-Quevillon provides the workforce for minor services and the town of Val d'Or and Chibougamau for the possible mine exploitation.

A camp on the property, built in 1996 by Murgor Resources Inc., provides living facilities for a small group and core logging and splitting facilities, as for storage. All major services are available in Val d'Or, Chibougamau, and minor ones in Lebel-sur-Quevillon.

Infrastructures

The access road, the camp and the stripping of the overburden were realised by Murgor between 1995 and 2005 on Barry and are all kept in good condition. Installations with catering and sleeping facilities can accommodate up to six workers during a stay at the site.

It is also important to mention the availability of sand and gravel from an esker crossing the Barry I property, if additional material is required. A major hydroelectric power line crosses the eastern part of the property.

The Quebec government has encouraged, in the past, natural resources development through the granting of permits, title security and financial incentives. Politically, the province and the county of the Municipalité de la Baie James are supportive of mining activities.

Note: The road conditions during last May site visit were allowing any regular vehicle (2x4) to travel from Lebel-sur-Quévillon to Barry and to Bachelor site.

Physiography

The overburden depth on the Murgor property is variable, ranging from zero metre to 5 metres thick in the area of the "Main Showing Zone" to over 30 metres in other areas of the property. It is often made up of gravel, large boulders and till.

Topographic relief is weak to moderate, locally up to 50 metres in the northwest part of the property due to outcrop ridges and eskers trending in a NE-SW direction. The southeast part of the property is of very low relief and is poorly drained. Fir trees and black spruces characterize the vegetation in the well-drained part of the property. The more poorly drained parts to the south are covered with spruce, balsam and Labrador-tea. The site of the Barry I Area project presents low relief topography. Primarily black spruce forests, swamps, eskers and small lakes cover the property area. The vertical relief in the area is very low with a mean altitude of 400 metres above sea level. Very few outcrops occur on the property.

Figure 4.1: View looking northeast of the topography and the typical vegetation of the Barry I Area property.

Figure 4.2: View looking east-northeast of the topography and the typical vegetation of the Barry I Area property.

Most of the overburden covering the Barry I central area has been removed and is stored on the property. The remaining overburden in the Barry I Area shows a thickness smaller than 5 metres, according to the present drilling information.

There is plenty of room for potential waste rock and tailing storage to the northwest of the actual deposit. A processing plant could be built on the property or a stockpile pad close of the actual access road.

4.2 Bachelor Lake Mine Property

Some of the informations of this item are extracted from the InnovExplo Technical Report of December 2005 for the Bachelor Lake Property and have been updated.

Accessibility

The Property is easily accessible by a 3-hour drive (225 km) on the Provincial highway 113 from Val-d'Or to Chibougamau (Province of Quebec). From Val-d'Or, the access is via the 117 Provincial highway heading east and then heading north on the highway 113 and by driving through to Senneterre, Lebel-sur-Quévillon and Desmaraisville. At Desmaraisville, a 3.5-km gravel road heads east to the Bachelor Mine.

The area is relatively flat (maximum elevation variation of 20 m) and lies at a general elevation of 295 masl. Coarse and sandy glacial deposits cover the area. Outcrop exposure is less than 2% and swampy areas are prevalent in the central and southern portions of the property.

Figure 4.3: Part of the surface infrastructures at Bachelor Lake Mine (May 2007)

Climate, Precipitation & Temperatures

This area is characterized by a continental climate. Winter temperatures range from -10°C to -35°C with an average snow accumulation of 83 cm. Summer temperatures range from 10°C to 22°C with an average rainfall of 115 mm. No rivers cross the immediate mining site area and the closest water source is Lake Bachelor approximately 3 km to the north. Access to a drinking water source is possible with a well and pumping station located on the mining concession #478 (previously Coniagas' property also owned by Metanor).

Local Resources

Production at the Bachelor mine terminated in 1989 after seven years of operation. Desmaraisville significantly decreased in population and in services after the mine closure. However, the area is still well provided with public services as it lies directly on a regional highway with power and telephone lines. The proximity of an active mining centre such as Val-d'Or still guarantees the availability of material and manpower for exploration and mining.

Infrastructures

The mine site includes surface infrastructure, hoist room, shaft house, mill (500 tonnes per day), tailing pond, and core shack. The infrastructures are in good conditions but need to be upgraded and refurbished.

4.2.1 Concentrator (Mill)

All informations about the concentrator are from a report prepared by Gilbert Rousseau, a consultant eng who visited the Bachelor Property on May 14th 2007.

This report refers to the actual general condition of the mill, the amount of work and cost involve resuming operations at 500 metric tonnes per day, the amount of work and costing necessary to raise tonnage to 750 metric tonnes par day, the manpower required to operate the mill and the operation costs.

Reliance on other experts

This report deals only with what was visually apparent. No equipment was opened and thoroughly inspected. No metallurgical tests were performed on the ore from Barry or Bachelor Lake. All information came from reports submitted by others.

Location and past operation

The Bachelor Lake Mine concentrator was built in the years 1981-1982 and is located 3.5 km south-east of Desmaraisville in the James Bay municipality some 225 kilometres north-east of the town of Val d'Or, more or less adjacent to provincial road 113.

The mill was in operation from 1982 to 1989. During this period, 869,400 metric tonnes of ore were milled at an average grade of 4.7 grams per tonne. To the best of the author of this report knowledge, mill average recovery was in the 93.0 % range. The mine and the mill were shut down due to higher than expected operation costs.

Mill description summary

As built, the mill process and equipments were very conventional. Crushing, grinding, leaching, zinc precipitation (Merrill-Crowe process) and bullion furnaces constituted the bulk of the mill equipments.

Crushing

The underground ore is hoisted into a 500 ton live load coarse ore bin. The main equipments of the crusher room are : a 36" x 24" Kue Ken jaw crusher, a 5' x 12' Dillon double deck screen and a 4.25 foot short head Symons cone crusher.

Grinding

The Dillon screen under size (-½") is conveyed to a 1,200 tons live load bin which feeds the grinding circuit. Main grinding equipments comprise a 10.6' x 10.0' x 9.0' Hardinge ball mill followed by a 6.0' x 10.0' of what seems to be an Allis Chalmers ball mill (no name plate). At the Hardinge ball mill, classification is done through a series of 15" Kreb cyclones whereas at the 6' x 10' ball mill sorting is done by a series of 10" Kreb cyclones. The 15" cyclone overflow is pumped to the 10" cyclones whereas the underflow goes by gravity back to the Hardinge ball mill. The 10" cyclone underflow feeds the 6.0' x 10.0' ball mill while the overflow is pumped to a 45' double tray thickener.

Thickening and Leaching

The 10" cyclone overflows are pumped into a 40' double tray thickener. Thickener underflow is pumped to a bank of five 20.0' x 20.0' agitator tanks connected in series whereas thickener overflow (pregnant solution) goes by gravity into two 20' Dia x 12'6" reservoirs also connected in series.

Product from last agitator tank is pumped to a first 12.0' Dia x 14.0' drum filter. Filter cake is repulped and pumped to the second 12.0' Dia x 14.0' drum filter. Solid product from this last filter is also repulped and then pumped to the tailings pond. Filtrate from both filters is pumped back to the thickener. A third drum filter was installed in the mill sometimes after the 1982 start-up. It is not known if this filter was ever put in operation.

Precipitation (Merrill-Crowe)

Pregnant solution from the two above mentioned reservoirs is pumped to a clarifier. Clarifier clear solution is literally sucked via a vacuum pump into a Crowe vacuum tank which in turn reports to a zinc precipitation dust feeder. Precipitate is then pumped to two Perrin filter presses. Clear solution from the presses (barren solution) is pumped into two

20.0’ Dia x 21.0’ reservoirs to be used for pump glands, repulpers, filters, etc, whereas excess goes to the tailings pond.

Refinery

Precipitate from the gold presses is dried and then melted in one of the two Wabi gold bullion furnaces. Hot liquid gold is then poured in special mold into doré bars.

Today’s state of the Mill (500 tpd)

The Bachelor Lake mill has been dormant since 1989. This partly explains that upon the author’s visit to the property, the mill was far to be ready to resume operation even at 500 tonnes per day not to mention 750 tpd.

Aside from badly worn milling equipments, the mill was also vandalized. Some conveyor belts were cut, the mill electrical room was stripped from most if not all of its easily reachable copper. Electric wires were cut here and there again for the copper, mill and crusher house were stolen from part of their exterior wall cladding, etc.

As for the milling machinery to be repaired one can mention the Symons short head cone crusher, the conveyors # 4 and 5, the 40’ thickener and the two wooden drum filters.

On the other hand, even at 500 tpd the mill is short of one filter presse, one bullion furnace, one system of destruction of cyanide, some instrumentation devices, spare parts, most of the mill and assay laboratories apparatus and glass ware and consumables inventory.

Moreover the entire wooden floor in the filter area is to be replaced by steel grating, all equipments and protection devices should be painted and as mentioned before, a good part of the mill electrical is to be restored and the roof along with some of the walls are to be repaired. Finally new toilet and shower facilities will have to be added to the one existent to accommodate feminine personal and for the ease of the operations, spare pumps should be purchased and installed wherever sole pumps are already in place.

Metallurgy

500 tonnes per day operation

There should be no problem to restart the mill at 500 tonnes per day. After all, the mill operated at 500 short tons for many years. There are unfortunately two drawbacks. If the underground goes to long hole mining instead of shrinkage, more than probably crushing will have to be done on a two 8 hour shift basis as the muck will probably be much coarser than before. Second if the 40’ double tray thickener is modified to a single tray, at least the feed well will have to be rearranged in order to increase thickener capacity. Here again, there is no guarantee that this will be enough. Since the use of flocculants is never a good idea with a Merrill Crowe process as they have a tendency to gum the clarifier leaves, experience and especially prudence suggest the immediate installation of a new 40’ thickener (see next chapter, 750 tpd operation).

Even if historical mill recovery was only in the 93 % range, a mill run made at the Lake Short concentrator before the shut down of all operations in 1989 reported recovery of +95 %. If the mill machinery is in good operating order, there is no reason that such **gold recovery at 95%** could not be achieved nowadays.

750 tonnes per day operation

With its 24" x 36" Kue-Ken jaw crusher, its 4.25' short head cone crusher and its 5' x 12' Dillon vibrating screen, the entire crushing plant capacity will be very marginal especially if long holes mining method is used. If Metanor does not want to invest on a bigger crushing plant, more than probably, crushing will have to be done on three shifts or at least 19 shifts per week leaving two shifts for clean up and maintenance. The short head crusher may have to be opened to 5/8" or even ¾" at the expense of installing more grinding capacity.

Along with what is needed to resume operation at 500 tpd, to increase the tonnage to 750 tonnes par day, a new 350 to 400 HP rod mill should be bought and installed as primary grinding along with a new 12.0' Dia x 14.0' drum filter. All pumps will have to be resized especially at the grinding.

There might be a lack of agitator capacity. As designed, at 500 st/d and 58 % solid, retention time in the agitators was calculated to be around 48 hours. If the tonnage to the mill is increased by 40 %, retention time will be decreased by the same amount or more or less by 19 hours. Even if 29 hours of leaching time seems to be enough for the Barry ore, without any tests as to the amenability of the new Bachelor Lake ore to cyanide leaching, a retention time of 29 hours is definitely on the low side.

Finally, the 40' single tray thickener will definitely be too small even if the feed well is rearrange. If it is possible to cheat a little whit the agitators retention time at the expense of some gold recovery, the same cannot be done with the thickening capacity especially with a Merrill Crowe process. As it is the intention of Metanor to modify the thickener from a double tray to a single one, at 750 tpd, the new thickener capacity will only have 1.52 pi^2/ton/24 hours whereas in the old days Leslie Engineering designed, at least for the Bachelor Lake ore, a 5.0 pi^2/ton/24 hours thickener. Therefore a new 60' diameter thickener will have to be installed in parallel to the 40' one.

Better still, the actual 40' thickener should be dismantled and a new 75' thickener should be installed outside thus leaving room in the mill to install two new 20' Dia x 22' agitators.

In order not to cause any delays in the mill operation, if it is the firm intention of Metanor to eventually increase the mill feed rate to 750 tpd, all these new pieces of equipment should be installed before the start up of the mill.

Mill Refurbishing Costs (500 tonnes per day)

The refurbishing cost to resume milling operation at 500 tonnes per day is based on recent similar projects (Resources Jake and Beaverbrook Antimony Mines), from Bachelor Lake mill superintendent own estimate and from the author's personal communications with used equipment dealers. The mill refurbishing excluded the indirect costs, the tailings pond and all environmental costs. As no formal quotations were asked, all costs are orders of magnitude and total cost is felt to be within a 25 % margin of error.

Roof and walls	200,000
Electricity	700,000
Cone crusher	50,000
Drum filters	200,000
Conveyors	50,000
Gold presse	20,000
New thickener (40’ Dia.)	150,000
Refinery	50,000
Pumps and motors	100,000
Propane installation (refinery)	20,000
Cyanide destruction system	100,000
Painting, clean up, house keeping	35,000
Floors (filter area)	50,000
New toilet and shower facilities (feminine personal)	50,000
Overhead Crane	30,000
Instrumentation	100,000
Laboratory equipment	175,000
Consumables	100,000
Spare parts	100,000
Other items (ball mils, cyclones, agitators, piping, etc)	100,000
Sub total	$2,380,000
Contingencies 10 %	$238,000
Total cost to start the mill at 500 tpd	**$2,618,000**

Table 4.1: Cost of mill refurbishing to 500 mt per day

Cost to Increase Tonnage to 750 tpd

Along with the cost of refurbishing the mill at 500 tpd, to insure a smooth operation, a new rod mill, a new thickener, two new agitators, a new drum filter and more than probably new pumps will have to be bought and installed. Inventory of spare parts and consumables will also have to be increased more or less in proportion to the new tonnage. The following costs include the equipment installation.

Rod mill (8’ Dia x 12’, 350 HP)	300,000
Cyclones and piping	50,000

Pumps
50,000
Thickener (75' Dia.)
250,000
Agitators (2 x 22' Dia. X 20')
150,000
Spare parts and consumables (above what is needed for 500 tpd)
100,000
Drum filter (12' Dia x 14')
250,000
(Thickener 40')
(150,000)
Sub total
$1,000,000
Contingencies 10 %
$100,000
Total
$1,100,000
Total cost to start mill at 750 tpd
$**3,718,000**

Table 4.2: Cost of mill increase from 500 to 750 mt per day

Operating Personnal

At 500 tpd, a mill a crew of 28 persons will be required. One more operator shall be hired if the mill goes to 750 tpd to add one shift at the crusher. This work force excludes the contactors' personal that will be required for at least the first six months of operation.
Personal repartition is as follows :

Mill Superintendent	1	
General foreman	1	
Crushing (one more at 750 t)	3	
Grinding	4	
Solution (leaders)	4	
Labourers	4	
Millwrights	4	
Electricians	2	
Gold presses and refinery	1	
Mill clerk (met. technician)	1	
Sampler (sample prep)	1	
Assayers	2	
Total	28	(29 at 750 tpd)

Labor cost at (500 tpd)*

		$/Hour	$/Year	F.B 34 %	Total $	$/Tonne
Mill super.			90000	30600	120600	0,77
Gen. Foreman			70000	23800	93800	0,60

Crushing		25	150000	51000	201000	1,28
Grinding		25	200000	68000	268000	1,70
Solution (leader)		27	216000	73440	289440	1,84
Labourers		23	184000	62560	246560	1,57
Millwrights		27	216000	73440	289440	1,84
Electricians		27	108000	36720	144720	0,92
Refinery		25	50000	17000	67000	0,43
Technician			60000	20400	80400	0,51
Sampler		23	46000	15640	61640	0,39
Assayers		27	108000	36720	144720	0,92
Total			1 498 000	509 320	2 007 320	12,74

Table 4.3: Cost of mill labor at 500 mt per day

It is assumed that the mill will operate 350 days per year at 90 % availability for a total of 157,000 tonnes per year.

Labor Cost at (750 tpd)

	$/Hour	$/Year	F.B 34 %	Total $	$/Tonne
Mill Super.		90000	30600	120600	0,51
Gen. Foreman		70000	23800	93800	0,40
Crushing	25	200000	68000	268000	1,13
Grinding	25	200000	68000	268000	1,13
Solution (leader)	27	216000	73440	289440	1,23
Labourers	23	184000	62560	246560	1,04
Millwrights	27	216000	73440	289440	1,23
Electricians	27	108000	36720	144720	0,61
Refinery	25	50000	17000	67000	0,28
Technician		60000	20400	80400	0,34
Sampler	23	46000	15640	61640	0,26
Assayers	27	108000	36720	144720	0,61
Total		1 548 000	526 320	2 074 320	8,78

Table 4.4: Cost of mill labor at 750 mt per day

Same as above for a total of 236,250 metric tonnes per year

Consumables (500 tpd)

	Kg/tonne	Unit cost	Cost/tonne
Jaw crusher wear plates			0.10
Cone crusher bowl and mantle			0.20
Ball mills liners (primary)		100,000.00 $/mill	0.21
(secondary)		50,000.00 $/mill	0.11
Steel balls (slugs)	0.30	1,000.00 $/tonne	0.30
Sodium cyanide	1.50	2,287.00 $/tonne	3,43
Lime	1.00	185.00 $/tonne	0.19
Electrical power		0.08 $/kWh	3.00
Sub total			7.54
Flocculent		5.76 $/kg	
Zinc dust		5.76 $/kg	
Lead nitrate		1.00 $/kg	
Super cell		35.90 $/bag	
Acid		1.17 $/kg	
Borax		65.13 $/bag	
Mg dioxide		89.32 $/pail	
Soda ash		18.37 $/bag	
Silica		6.03 $/bag	
Clarifier bag		130.75 $ each	
S.S. wire filter		945.00 $ each	
Filter cloth		945.00 $ each	
Propane		1.00 $/pound	
Assay office supplies		20,000.00 $/year	
Spare parts		50,000.00 $/year	
Other (building, tailings, etc.)		10.000.00 $/year	
Sub total			2.00 $
Total consumables			**9.54 $**

Table 4.5: Consumables at 500 mt per day

The total direct cost of labor and consumables at (500 tpd) is **22.28 $/t**

Consumables (750 tpd)

Even if more consumables will be used at 750 compare to what will be employed at 500 tpd, the cost per tonne for these consumables will remain in the same order of magnitude. The main difference comes in the manpower, as only one operator will have to be added

Total direct cost labor and consumables (750 tpd) **18.32 $**

Mill Overhead Costs (indirects costs)

Past experience has shown that the overhead costs (main office, engineering, insurance, surface and road maintenance, guard house, dormitories, etc), are in the order of 10 % of the direct costs.

Total Milling Costs

The total milling cost at 500 metric tonnes per day should be the direct costs plus 10% overhead, that is $24.51 ($22.28 plus 10%) and $20.15 ($18.32 plus 10%) in the case of 750 metric tonnes per day.

End of Gilbert Rousseau Report

4.2.1.1. Mill Operating Costs per imperial tons

Following a meeting with Metanor's staff it was agreed to have all production data and costs in imperial units. The following table is the summary of the concentrator capacities and costs expressed in imperial units obtained from Gilbert Rousseau report.

BACHELOR LAKE MINE CONCENTRATOR OPERATING COSTS

Description	**500 st/day**	**750 st/day**
	350 days/year	350 days/year
	90% availability	90% availability
	157,500 st/year	236,250 mt/year
	13,125 st/month	19,687 st/month
Rounded to	**13,125**	**20,000 st/month**
Labour (incl 34% F.B.)	$ 14.03	$ 9.65
Consumables	$ 8.67	$ 8.67
Sub-total	$ 22.70	$ 18.32
Overhead at 10%	$ 2.27	$ 1.83
Total	**$ 24.97**	**$ 20.15**

Other Infrastructures

The Bachelor gold deposit was mined by underground mining methods, mainly by shrinkage stoping. The mine was dewatered during the winter of 2004-2005 for the realization of an underground drilling program in 2005. The mine is currently accessible by a three-compartment shaft to the 7th Level and a four-compartment shaft beyond the 7th Level. An important aspect regarding the underground infrastructures is the decision to install a 10 ft hoist already available on site.

The shaft sump is at a depth of 562.66 m (1 846'). Twelve levels, with ventilation and egress, have been developed. Underground access from the Bachelor mine on the Hewfran claims already exists on the 4th, 6th and 8th levels.

A contract has been awarded to A. Golder & Ass to conduct geotechnical workings to define the work necessary to rehabilitate the tailings pond to the Government requirements. (ref: Press releases of March 14th and April 12th 2007)

5. History (Item 8)

5.1 Barry I Property

This following information is part of the April 2007 Technical Report on Resources Evaluation of the Barry-1 Property by Geostat.

Summary of previous work

Previous Work

1943	Area mapped by Mimer.
1946-47	Area mapped by Fairbairn and Graham.
1958	Geological survey performed by Geological Survey of Canada.
1961	An airborne MAG-EM survey was performed by Claims Ostiguy.
1962-65	Geology, geophysics and 5 ddh were completed by Fab Metal Mines LTD.
1981-84	An airborne MAG-EM survey was performed by Questor Surveys LTD. for the Quebec Ministry of Energy and Resources.
1981-83	Prospecting and Geological Mapping was carried out by SDBJ followed by three drill holes.
1983	Mines Camchib completed one hole of 146 metres (MB-83-1 1) at the western edge of the property. No significant assays were reported.
1988-89	Ground MAG and EM surveys were completed by Cominco-Agnico Eagle. Nine drill holes followed.
1990	An evaluation of this property was carried out by Albanel Minerals LTD. and Somine Inc.
1995	Overburden stripping, trench and channel sampling by Murgor.
1995	Detailed mapping and geophysical works realized on the discovery showing.
1995-6	Murgor drilled 56 holes on the property and sent 167 channel samples for assay.
1997	IP survey, geological mapping, lithogeochemical sampling and drilling of 4,456 metres of core by Teck Exploration, mainly on the Barry I Main property.
2004-2005	Geological interpretation and drilling (61 holes) on the property by Osisko Resources Inc. Report deposit still pending.
2005	Writing of a preliminary assessment study on the Barry property by George McIsaac, eng., M. eng.
2005	Murgor realised one drilling campaign of six holes for 225 m. and a new geological interpretation of the Barry deposit by Murgor's staff.
2006	Drilling by Murgor of 32 drill holes for 1,409 m. and survey of the visible drill holes collars of the Main Zone.
2006-2007	Drilling of 58 drill holes totalling 5,076 m.

Fab Metal Mines	1962-65	5 holes	114 m
SDBJ	1981-83	3 holes	264 m
Mines Camchib	1983	1 hole	146 m
Cominco-Agnico Eagle	1988-89	9 holes	1,461 m
Murgor Resources	1995-96	74 holes	7,703 m
Murgor Resources	1995	167 channels	1,203 m
Teck Exploration	1997	15 holes	4,456 m
Osisko	2004-05	61 holes	2,580 m
Murgor Resources	2005	6 holes	225 m
Murgor Resources	2006	32 holes	1,409 m
Murgor Resources	2006-2007	58 holes	5,076 m

Table 5.1: Summary of the previous exploration drilling work on the Barry property.

Details of previous work on Barry

The area surrounding the Murgor property was first mapped in the 1940's, but it was not until 1962 that exploration work on the property was first recorded. Exploration in the area has progressed significantly in the last 10 years due to the increased access provided by the expanding network of logging roads.

Work by Fab Metals Mines in 1962-1964

Seven shallow drill holes (458 m) were drilled outside of the "Main Showing" area. In 1962, Fab Metal Mines, owned by Fred A. Boylen, drilled three short holes totalling 87 metres on the eastern shore of the Macho River. Basalts and feldspar porphyry were intersected, which contained sparse pyrite mineralization and the odd quartz veins. These holes were drilled outside of the "Main Showing" area.

In 1964, Boylen drilled two additional short holes totalling 37 metres on a zone of strong quartz veining on the west shore of the Macho River. Boylen's drill logs referred to sheared volcanics with quartz tourmaline veins and visible gold. No follow-up work has been done to date on that area.

Work done by Questor Suveys Ltd in 1981-1984

In 1981 and 1984, Questor Surveys Ltd. completed an airborne EM-INPUT and magnetometer survey over the area for the Quebec Ministry of Energy and Resources. This survey (DP 83-08 and DP 85-19A and B) identified several EM anomalies on the Murgor property associated with a strong magnetic conductor.

Work done by SDBJ in 1982-1984

The discovery of the "Main Showing" dates back to 1982 when grab samples taken by SDBJ assayed up to 35 g/t Au. Between 1982 and 1983, SDBJ completed prospecting, line cutting, geological mapping, magnetometer and horizontal loop EM surveys. Three diamond drill holes (83-9, 83-10 and 83-11) totalling 264.5 metres were drilled in the area of the "Main Zone" to test geophysical targets. All the drill holes intersected anomalous gold mineralization, with drill hole 83-9 assaying 4.l g/t over 1.4 metres.

Work done by Mines Camchib in 1983

In 1983, Mines Camchib completed one hole of 146 metres (MB-83-1 1) at the western edge of the property. No significant assays were reported.

5.1.6 Work done by Cominco-Agnico Eagle in 1989-89

In 1988-89, a Cominco-Agnico Eagle joint venture completed magnetic, EM, IP and soil geochemical surveys along with overburden trenching. Nine diamond drill holes (LON-88-l, -2, -3 & LON-89-4, -5, -6, -7, -8 and -9), totalling 1,461 metres, were drilled on the property. The best assay was from drill hole LON-88-3 with an assay of 6.45 g/t over 1.8 metres.

Work done by Murgor resources in 1994

In November of 1994, Murgor optioned the SDBJ claim block as well as the Duval and Boudreault claim blocks. The property was surveyed with magnetic, IP and basal till surveys along with an extensive overburden stripping and channel-sampling program. Diamond drilling completed by Murgor concentrated on the Barry I Main Zone Area and totalled 56 holes (MB-1 to 56) for 5,918 metres. The Barry I Main Zone Area had been drilled over a strike length of 800 metres and down to a vertical depth of 250 metres. Multiple gold bearing zones were identified with intersections as high as 9.7 g/t Au over 7.7 metres. A mineral inventory was calculated on the Barry I Main by Murgor, which totalled 610,000 mt grading 6.8 g/t Au (Tessier, 1996).

Work done by Murgor in 1995

A program of 18 drill holes was completed on the Barry I property between February 20 to April 2 1995. A total of 1,785 metres of NQ core were drilled with 1,516 samples were assayed for gold.

The drilling confirmed the presence of gold. A typical gold zone is composed of alternating sections of auriferous altered volcanics and unaltered volcanics.

The drill results proved the mineralized zones to be very complicated, where it was impossible to tie together the mineralization on strike and on section. Some features, which may be localizing the gold mineralization, could be the folding, contacts, fractures, flexures or intersecting structures.

The conclusions of these works are the following:

1) The Barry I property is located within a major deformation zone created by overlapping strain aureoles related to the emplacement of two large plutons. The two large plutons flank the greenstone rocks to the northwest and southeast.

2) The strike orientation of the gold associated deformation zone is 060º (east-northeast). Several gold showings in this area are also associated with this orientation. The dip of the units on the property is 60° south, whereas the plunge is 45° - 50° to the east.

3) The gold mineralization is typical Archean lode gold style with auriferous quartz-carbonate-albite veinlets hosted within highly carbonatized pillowed basalts and basaltic flows. The gold usually occurs as the native element or as inclusions within the pyrite.

4) Hydrothermal fluids have been deposited within fractures rather than shear zones. Very little shearing is evident.

5) 90% of the veinlets have the same dip as the foliation, which is 060° to the south.

6) Broad zones of Fe carbonate exist, zoned away from the veinlets. Biotite alteration also exists at the immediate contact with the volcanics and sometimes along fractures at right angles within the veinlets. The presence of biotite and the hornfelsic appearance of the volcanics locally suggest a very high temperature deposition of the fluids.

7) Some drill holes did not encounter the expected gold mineralization, as the result of previous surface works, suggesting a possible plunge of the main showing.

8) The same style of veinlets and sulphides observed in the quartz feldspar porphyries did not carry gold mineralization even though they did in the volcanics. This suggested that the QFP was not chemically correct to allow for gold precipitation.

9) The showing corresponds to a coincident MAG high and IP anomaly.

10) The greater the vein frequency, the stronger the alteration, the higher the percentage of pyrite and therefore the higher the gold assays.

11) The veinlets are bulged suggesting a stretching deformation, while the pillows are flattened suggesting a compression deformation.

Work done by Teck option during 1997

A total of 4,456 metres of diamond drilling in 15 drill holes were completed on the Murgor property between June and August of 1997. This drilling tested the extensions of the auriferous Barry I Main Zone and parallel or faulted off structures to the north.

Drilling

Teck had the holes MB-57 to 62 and MB-68 to 71 on the property. These holes tested the extension of the gold mineralization hosted in the Barry I Main Zone, along a strike of 800 metres and down to a vertical depth of 325 metres below surface. The gold mineralization was intersected in mineralized corridors in a variety of stratigraphic units. The most significant areas in order of importance include:

1. Altered basalts at the hanging wall contact of the quartz-feldspar-porphyry.
2. Basalts at the footwall contact of the quartz-feldspar-porphyry.
3. Basalt-gabbro to the north of the quartz-feldspar porphyry.
4. Quartz-feldspar porphyry.
5. Massive basalt unit to the south of the quartz-feldspar porphyry.
6. Brecciated basalt unit.

The best gold intersections were from the altered basalts located to the south of the hanging wall contact of the main quartz-feldspar-porphyry sill. The altered and mineralized basalts were intersected over thicknesses of up to 85 metres and contained several gold bearing zones. The best intersection assayed 6.38 g/t Au over 7.7 metres from hole MB-62. It is composed of several narrow higher-grade mineralized intervals with unaltered basalt located in-between. Other significant mineralized sections include 3.95 g/t Au over 5.0 metres and 3.39 g/t Au over 4.6 metres from holes MB-58 and MB-68 respectively. Although the major stratigraphic units in the area are continuous, the gold bearing sections are not as large and appear much more discontinuous.

The location of the gold mineralization on the footwall side of the quartz-feldspar- porphyry is not as well defined as that found in the hanging wall due to the varying thickness of the quartz-feldspar-porphyry unit. Best intersections include 8.48 g/t Au over 2.2 metres in drill hole MB-58 and 6.47 g/t Au over 2.9 metres and from drill hole MB-70.

Sections with anomalous gold mineralization were also identified in the quartz-feldspar porphyry unit, the brecciated basalt unit, the more massive basalt unit to the south of the quartz- feldspar-porphyry and in the massive basalt-gabbro unit to the north of the quartz-feldspar- porphyry. Assay results for these zones were as high as 3.49 g/t Au over 1.8 metres. The gold mineralization in these corridors was commonly present as sheared and altered zones close to small quartz-feldspar-porphyry sills.

The diamond drilling did confirm that the mineralized system at the Barry I Main Zone Area is large, and the zone was intersected in virtually every hole. Although the mineralization remains open in all directions, the drilling shows that on a detailed scale the gold bearing zones are represented by numerous smaller lenses. Based on previous surface stripping and closed spaced shallow drilling the size of individual mineralized lenses may only be in the order of 45 metres in strike.

The diamond drill holes MB-63 to 67 targeted a chargeability anomaly and associated magnetic high parallel and to the north of the Barry I Main Zone. The only significant assay from this shallow diamond drilling was from hole MB-64, which assayed 1.73 g/t Au over a core length of 1.6 metres. The gold mineralization encountered in this area is similar in style to that encountered at the Barry I Main Zone, and is associated with biotite-carbonate alteration, quartz-carbonate veining and disseminated pyrite. The assay quoted above in drill hole MB-64 is from the contact of a small quartz-feldspar-porphyry unit.

Surface Mapping and Sampling

A program of surface mapping and outcrop sampling was completed on the property concurrently with the diamond-drilling program in the summer of 1997. A total of 52 samples were analyzed for gold, of these, 27 samples were also analyzed for major and minor elements. The highest gold assay from a surface grab sample outside of the Barry I Main Zone Area was 2.01 g/t Au. This sample was taken from a small pit, located approximately 150 metres to the north of the Barry I Main Zone, which corresponds to the northern IP conductor drill tested with holes MB-63 to 67. The IP anomalies are due to the presence of disseminated pyrite and local stringers of magnetite.

A significant amount of quartz veining with rare pyrite mineralization was located in outcrops close to IP chargeability anomalies in the northern part of the property at L23+85E, l2+75N and in the eastern part of the property at L4l+85E, 7+105. The quartz veins in the northern part of the property on L23+85E were also found to contain up to 5% of a mineral identified as geikielite ($MgTiO_3$), which has been found to be locally associated with gold mineralization in the Val d'Or mining camp.

Geophysical IP survey

A dipole-dipole array IP survey with a totalling 53 km covered portions of the property not covered by previous surveys was realized. Several moderate to strong chargeability anomalies were outlined in the northern and eastern parts of the property.

Two of the 12 anomalies defined by previous surveys correspond to the known sulphide mineralization; i.e. the Barry I Main Zone Area and the zone 150-200 metres to the north. These 17 anomalies are characterized by strong chargeability, background resistivity signatures and are associated with magnetic highs. Both of these anomalies, each approximately 1,000 metres in length, appear to have been offset by an E-W trending structure with a sinistral movement. The chargeability highs are due to finely disseminated pyrite (3-7%) and lesser pyrrhotite and magnetite.

Based on the recent IP survey, there exist up to six separate IP (chargeability) anomalies in the northern and eastern part of the Murgor property. Individual IP anomalies can be traced over strike lengths of up to 2,000 metres. All are untested by diamond drilling and no outcrops are present in the area of the anomalies.

IP surveying has proven to be the most useful geophysical technique in the Urban-Barry Volcanic Belt. It works well in identifying and locating the disseminated style of the sulphide mineralization associated with the gold mineralization.

Litho geochemistry results

Systematic core sampling at 30 metres intervals, for 160 samples, was completed on all drill holes. The samples were analyzed for 10 major oxides, loss on ignition and a 32 elements package by ICP. Alteration trends were appraised through bulk chemistry methods designed to monitor relative enrichment-depletion patterns of mobile elements typical of gold deposits

The basaltic rocks are of tholeiitic to transitional affinity as defined by immobile element plots. Three populations of chemically different rock units were identified from various X-Y plots using AL_2O_3, $Ti0_2$, and Zr concentrations. These included quartz-feldspar porphyry, basalts and plagioclase-phyric basalts or feldspar porphyries. No significant geochemical difference could be established amongst the various subunits of basalts and gabbros.

Though the most significant gold intersections were hosted within the basalts, the quartz-feldspar-porphyry unit commonly showed a higher background concentration of gold. Median gold levels in the basalts are 6 ppb while, in the quartz-feldspar-porphyry, the values were almost four times higher at 23 ppb. The mineralized zones within the basalts do not show any significantly large alteration halo identifiable by geochemical anomalous gold values or associated pathfinder elements. The gold mineralization is restricted to the quartz veins and their borders.

The conclusions on the work done by Teck option during 1997 are the following:

The continuity and size of these individual higher-grade zones is difficult to establish and appears erratic. No significant increase in the gold grade was observed along strike or at depth. The mineralized corridors do however remain open in all directions.

The Murgor property covers iron rich basalts intruded by quartz-feldspar porphyry, both of which are favourable hosts for gold mineralization. Mineralization at the Barry I Main consists mainly of sheeted auriferous quartz-carbonate-albite veins aligned parallel to the regional foliation at 060°. A second set of contemporaneous quartz-carbonate-albite veins is also present, oriented at 020° parallel to the Milner Shear Zone. This favourable geology and structural setting are interpreted to be present elsewhere on the property.

Work done by Osisko option during 2004-2005

A total of 61 drill holes, for 2,580 metres, were drilled mainly on the Barry I Main Zone Area by Osisko Resources Inc. during the June 2004 and February 2005 period. A partial survey of the drill holes collars was carried out during this period. Only the computerized version of the drill logs was available for this study. One database including all the computerized data on the Barry property was prepared and kept up to date. No other document prepared by Osisko was given to Murgor. The staff of Osisko did a new interpretation of the mineralized deposit according to the information retrieved from the new drill holes. Following their study of the gold potential for that deposit, they released their option to concentrate their efforts on another deposit of larger tonnage. The size of the Barry deposit does not fulfill their requirement for a large deposit to exploit.

The release of their report on the Barry property is still pending.

Work done by Murgor during 2005-2006

Six drill holes for 225 metres were drilled mainly on the Barry I Main Zone by Murgor during December 2005. A new geological model interpretation was developed according to the new data and tested by three drill holes required by Geostat. These drill holes confirmed the presence of gold. The three others aimed to add tonnage to the Barry I Main and to test a high-grade target in the southwest part of the Barry I Main Zone. One database was created and verify by Geostat's staff. The position of the collars had to be surveyed. The data of five of the previously drill holes were not found. All the assays greater then 1 g/t Au were checked when the assay certificates were available. A new resource estimate was calculated from the new geological interpretation and aimed at define resources possibly mined by open-pit. They were estimated by inverse distance using a maximum of 10 composites of 1.5 metres length. The results of the estimates of February 2006, according to different cut offs, are the following'

Total resources (no cut-off) Rounded

Category	Tonnage (mt)	Volume (m3)	Density	Au (g/t)	Oz Au
Indicated	**211,000**	**75,000**	**2.8**	**4.35**	**29,500**
Total	**211,000**	**75,000**	**2.8**	**4.35**	**29,500**
Inferred	150,000	54,000	2.8	4.18	20,200

Total resources (cut-off of 1 g/t Au) Rounded

Category	Tonnage (mt)	Volume (m3)	Density	Au (g/t)	Oz Au
Indicated	**208,000**	**74,000**	**2.8**	**4.41**	**29,400**
Total	**208,000**	**74,000**	**2.8**	**4.41**	**29,400**
Inferred	147,000	53,000	2.8	4.26	20,100

Total resources (cut-off of 2 g/t Au) Rounded

Category	Tonnage (mt)	Volume (m3)	Density	Au (g/t)	Oz Au
Indicated	**176,000**	**63,000**	**2.8**	**4.92**	**27,800**
Total	**176,000**	**63,000**	**2.8**	**4.92**	**27,800**
Inferred	118,000	42,000	2.8	4.90	18,700

Total resources (cut-off of 3 g/t Au) Rounded

Category	Tonnage (mt)	Volume (m3)	Density	Au (g/t)	Oz Au
Indicated	**129,000**	**46,000**	**2.8**	**5.77**	**24,000**
Total	**129,000**	**46,000**	**2.8**	**5.77**	**24,000**
Inferred	83,000	30,000	2.8	5.91	15,800

Total resources (cut-off of 4 g/t Au) Rounded

Category	Tonnage (mt)	Volume (m3)	Density	Au (g/t)	Oz Au
Indicated	**96,000**	**34,000**	**2.8**	**6.58**	**20,300**
Total	**96,000**	**34,000**	**2.8**	**6.58**	**20,300**
Inferred	55,000	20,000	2.8	7.18	12,600

Total resources (cut-off of 5 g/t Au) Rounded

Category	Tonnage (mt)	Volume (m3)	Density	Au (g/t)	Oz Au
Indicated	**70,000**	**25,000**	**2.8**	**7.34**	**16,500**
Total	**70,000**	**25,000**	**2.8**	**7.34**	**16,500**
Inferred	38,000	14,000	2.8	8.35	10,300

Table 5.2: Resources estimates in February 2006 on the Barry I Main Zone Area.

The Barry I Main Zone Area property, as per February 6 2006 and including holes drilled in December 2005, i.e. 162-167, contains a total of 27,800 ounces in the indicated category and 18,700 ounces in the inferred category, at a cut-off of 2 g/t Au.

Work done by Murgor during 2006

A second drilling campaign was executed in the first months of 2006. Some 32 drill holes for a total of 1,409 m were drilled on the Main Zone and tested the SW extension of the Main Zone Area and the Zone 43. A total of 1,279 samples were sent to the lab for gold assay. Twenty of these samples were for quality control.

Murgor performed a survey of the casings still presents and visible over the snow cover on the Main Zone Area that permitted to update their collar coordinates. The position of the surveyed drill holes moved slightly as their three coordinates changed. All the previous estimates were based on measured coordinated according to the cut line pattern.

This new drilling campaign permitted to better define the extension of the mineralized zone inside the Main Zone Area and to verify the southwest and northwest extensions of the Main Zone. Some of the holes drilled tested the extension of the Zone 43 located southwest of the Main Zone. They intersected this zone to a depth up to 50 meters and the known southwest northeast extension is 130 metres.

A new interpretation of the mineralized zones and an update of the previously estimated resources were performed. The resource estimate aimed to define mineralization exploitable by open-pit mining. This new design included the mineralized zones from the Main Zone, the zones 43, 45 and the southwest extension of the Main Zone.

Work done during 2006-2007

A new drilling campaign was executed. Some 58 drill holes for a total of 5,076 m were drilled on the Main Zone and tested the east, north and south deeper extensions of the Main Zone Area and the Zone 43. A total of 4,988 samples were sent to the lab for gold assay.

This new drilling campaign permitted to better define the extension of the mineralized zone inside the Main Zone Area and to verify the extensions of the Main Zone.

A new interpretation of the mineralized zones and an update of the previously estimated resources were performed. The resource estimate aimed to define mineralization exploitable by open-pit mining. This new design included the mineralized zones from the Main Zone, the zones 43, 45 and the southwest extension of the Main Zone.

The mineralization possibly exploitable by open-pit was not altered by this new drilling.

5.2 Bachelor Lake Mine Property

Most of the information of this item are extracted from the InnovExplo Technical Report of December 2005 for the Bachelor Lake Property and have been updated.

History

The property was originally staked by O'Brien Gold Mines Ltd. (O'Brien) in 1946 following to the discovery of the "Main" Zone on the eastern part of the O'Brien pluton. This discovery rapidly led to trenching, geophysical surveys and numerous drill holes.

In the sixties, Sturgeon River Mines Ltd. (Sturgeon) sank a shaft and drilled underground to the 7th Level (1961-64). From 1972 to 1975, **739 000 short tons at a grade of 0.18 oz/t Au** were outlined. In the 1980's, Bachelor Lake Gold Mines (subsidiary of Sturgeon) conducted several underground development work phases in order to start mining in 1982. They deepened the shaft to the 12th Level in 1987, and stopped production in 1989 (**958 368 short tons at a grade of 0.136 oz/t Au** were mined, for a total of 131 029 oz of refined gold).

Since the mine closure, several resource estimates were published on the Bachelor claims and the Hewfran claims.

1) <u>Bachelor claims</u>

Measured Resources: 204 454 short tons at 0.257 oz/t Au;
Indicated Resources: 216 685 short tons at 0.315 oz/t Au; and
Inferred Resources: 256 285 short tons at 0.304 oz/t Au.

Originally estimated by Harron (1990), the resources were cited in Géospex (1993) and modified by Géospex (1995), validated by SNC-Lavalin (1999), by Met-Chem (2001) and finally by Innovexplo (2004) in a NI 43-101 technical report, Table 10.

2) <u>Hewfran claims</u>

After several exploration drilling programs in June 1989, Aur (Y. Rougerie) yielded to an estimated gold resource of 594 000 st @ 0.170 oz/t Au for the West Zone (100 900 ounces of gold). The East Zone resource has been recently re-evaluated and downsized by Y. Buro (2005) from 120 000 st @ 0.210 oz/t to 68 000 st @ 0.259 oz/t.

The spacing interval of mine levels varies:

- Surface to 1st Level: 53.34 m (175');
- 2nd to 7th Levels: 45.72 m (150');
- 8th to 12th Levels: 38.10 m (125').

This reduced level interval indicates the difficulties encountered while extracting the ore tonnage and it also explains the higher production costs relative to the increased amount of development required to access ore. Operations were awarded to mining contractors and the production mining equipment was also supplied by the contractor (Loco, cars, jack legs, mucking machines, etc.). This also could explain the overall higher production costs. If the mining operations have been fully integrated and equipment included, then cost results would have improved.

In the last year of operation (1989), the operating costs were reported by Bachelor Lake Gold Mines to be as follows:

(prices are rounded off)

Bachelor Lake Gold Mine 1989 operating cost

Mining:	**$35.00 /st**
Milling:	**$12.50 /st**

Adm. & general:	**$9.00 /st**
Camp:	**$5.50 /st**
Total:	$62.00 /t

Table 5.3 - Bachelor Lake Gold Mine Historic (1989) Operating Costs

The calculated head grade was approximately 0.145 oz/t for the life of the mine, indicating a serious dilution problem especially when the estimated grade for the resources was 0.21 oz/t. Firstly, an ore pass system will have to be installed on the lower level if a production of 500 st per day is anticipated.

The mine was flooded in 1992, and dewatered in November 2004 in anticipation of the U/G drilling program. Metanor's employees are presently keeping the mine dewatered. In 1990, under a Joint Venture agreement with Acadia Mineral Venture Ltd. (Acadia) (controlled by Hecla Mining Company of Canada, "Hecla"), 34 drill holes were drilled from the 12th Level and 5 drill holes from the 11th Level. In 1994, Espalau mining acquired 100% of the property and 10 surface drill holes were drilled in 1995.

Since December 2004, the Bachelor claims have been registered 100% to Metanor. Metanor acquired the property from GéoNova/MSV/Campbell. Since September 2005, Halo satisfied its work agreement on the property and acquired a 50% interest which has led to the formation of the BLJV. In 2004-2005, Halo dewatered the Bachelor mine and initiated a 13 346 m (69 holes) underground drilling program in order to fulfill its option agreement.

The exploration history of the property, presented below, is partly based on compilation work previously provided by Innovexplo, as well as information from Aur internal report and from the SIGÉOM database, the "Ministère des Ressources Naturelles, de la Faune et des Parcs" database for reports and assessment work files (http://sigeom.mrnfp.gouv.qc.ca). Other validation and complementary verification was also done for the entire Bachelor claims and the Hewfran claims.
The history of the property is summarized below.

Bachelor Claims		
Date	Company	Work description
1946 – 1949	O'Brien Gold Mines Ltd.	DISCOVERY (Au) of the "Main Zone" on the eastern side of the O'Brien pluton. (GM 00972 A @ E). Mag survey and drilling program (more than 53 holes) (GM 25061).
1951	Quebec Depart-ment of Mines	Longley, W.W., geological mapping of the Bachelor Lake area.
1961 – 1964	Sturgeon River Mines Ltd	- 14 holes were drilled, - EM survey (GM 25061, GM 13632). Sinking of a three-compartment shaft of 338.63 m (1 111') and underground drilling (GM 25061). 3 surface holes and 17 underground holes were drilled (GM 13462). In 1964, the company was reorganized and the name changed to Quebec Sturgeon River Mines Ltd.

Hewfran Claims		
Date	Company	Work description
1946 - 1947	S-Francis Mining	Discovery of Agar #1 and #2 gold showings: geological mapping and trenching (GM 3553; 6602).
1948	Batch River Gold Mines	- Discovery of 2 zinc showings and a gold showing in the northern third of lots 12-19, RV. - Numerous geophysical surveys, mapping and trenching (GM 467; GM 10 879).
1948	Hewelt (Inspiration option)	Geological mapping (GM 7 091).
1948	Dome Exploration	Testing for Coniagas-type massive sulphide mineralization, and extension of Agar #2 Au showing: 34 DDH totalling 4 066 m (13 337 ft) and geology.
1957	Quebec Bachelor Mining Corp.	- Magnetometric survey in the northwestern part of the Hewfran property (lots 8-11). - Several drill holes from surface (GM 5 211 A-B).
1960	Roxford Mining	Geological survey (GM 10 172).

Bachelor Claims			Hewfran Claims		
Date	Company	Work description	Date	Company	Work description
			1965	**Sturgeon River Mines**	**IP survey and drilling program: covered entire region looking for Coniagas-type massive sulphides, several weak anomalies detected.**
			1970	**Valdex Mines**	**IP survey (Geoterrex) over several parts of the Hewfran property: gradient, pole-dipole arrays, lots 8-15, RV searching for VMS, several anomalies defined, two drilled in 1971.**
			1971	**Valdex Mines**	- **Magnetometric and EM surveys in the south-eastern part of the Hewfran property, covering five scattered blocks, partly following up on 1970 IP survey;** - **4 surface DDH, 323 m (1 060') (7.85% Cu and 1.1 oz/t Ag over 0.9 m (3.0') on a RIV/RV line).**
1972 – 1975	**Quebec Sturgeon River Mines Ltd**	**Surface and underground exploration: 13 holes were drilled in 1972 (GM 28460) and 6 more holes in 1973 (GM 29068).** In 1975, historic Mineral Resource estimate at 739 000 st @ 0.18 oz/t Au. **High grades were cut to 1.0 oz/t Au and 15% dilution was included.**	**1973**	**Valdex Mines**	**Geophysical surveys covering RIV, Lots 14-18 (north half) and RV, Lots 14-18 (south half). Searching for Coniagas- type massive sulphides and Bachelor-type gold. IP anomalies detected over power line, over Zn-Ag-Pb showing and over Agar #1 Au showing.**
			1974	**Valdex Mines**	- **Geological mapping by Picard & Mongeau of north half of lots 14-18, RIV.** - **3 holes tested Agar #1 showing (0.05 oz/t Au over 3.35 m (11.0')).** - **1 hole tested Agar #2 showing (0.9 m (3.0') of 0.12 oz/t Au);** - **One hole lost in overburden.**
			1975	**Valdex Mines**	**Geological mapping and lithogeochemical study by Descarreaux & Gaboury: Re-mapped north part of lots 14-18 RIV and mapped south half of lots 13-17 RV; Attempting to trace Coniagas horizon and search for alteration halos.**

Bachelor Claims			Hewfran Claims		
Date	Company	Work description	Date	Company	Work description
			1978	Brominco Inc.	IP survey in several parts of the property: re-testing Coniagas stratigraphy with new NE-SW grid. Also surveyed north half of lots 16-19 RV. No significant anomalies found. Surveyed (HLEM, Maxmin II) the NE corner of property and north half of lots 14, 15 RIV on NE-SW grid. Several weak (quadrature) anomalies detected in both areas.
1980	Bachelor Lake Gold Mines Inc.	- Bachelor Lake Gold Mines Inc. was created as a subsidiary of Quebec Sturgeon River Mines Limited. - Construction of the mine infrastructure initiated in 1980. - 20 holes were drilled (GM 40863). Beginning of the commercial production on July 1982 until 1989.			
			1983	Brominco Inc.	Detailed mapping from Kretschmar with VLF-EM survey and lithogeochemistry covering the NE corner of property (Lots 18-22, RV/RIV). Two zinc showings re-located, several VLF-EM anomalies of unknown significance detected.
1984 – 1985	Bachelor Lake Gold Mines Inc.	Humus geochemical survey (every 33 m (100') sampled on lines separated by 100 m (300') for the western part and with 180 m (600') for the eastern part). A total of 1 283 samples were taken.	1984	Brominco Inc.	Detailed geology, P.E.M., humus surveys and diamond drilling (11 holes, 1 350 m (4 428')) mainly exploring for Zn-Ag-Pb-Cu mineralization along Coniagas Horizon; limited work on Agar #1 and #2 Au showings.
1985 – 1987	Bachelor Lake Gold Mines Inc.	MAG, VLF and EMH survey on the northern third part of the property. Temporary suspension of operations in 1987, for shaft deepening (four compartments) from the 7th to the 12th level and development.	1986	Aur Resources Inc.	Program to explore for the extension of Bachelor Lake mine: - 76 m (250') of drifting on 4th level and 2 482 m (8 144') of UDD in 24 holes and 1 396 m (4 581') of SDD in 5 holes; 0.277 oz/t over 6.3 m (20.8') at vertical depth of 381 m (1 250'); - Geological mapping and stripping on Agar #2 Showing and channel sampling.
1989	Bachelor Lake Gold Mines Inc.	End of the production at BLGM, 958 368 st of ore produced @ 0.136 oz/t Au	May 1987 – May 1989	Aur Resources Inc.	Major exploration drilling program: - 14 255.5 m (46 770') SDD in 47 holes;

Bachelor Claims			Hewfran Claims		
Date	Company	Work description	Date	Company	Work description
		(total of 131 029 oz refined gold). Ore dilution has been excessive and was undoubtedly the main reason for the financial difficulties.			- 10 401 m (34 125') UDD in 96 holes; - 826 m (2 711') lateral U/G development from BLGM.
		The mine was placed on a care and maintenance basis, when costs exceeded revenues.	1988	Aur Resources Inc.	Mill test of 5 246 t (5 783 st) bulk sample from the Main Zone at 8th level (3 300 st) and the "Main" zone at the 6th level (2 300 st).
1990	Bachelor Lake Gold Mines Inc.	Resource estimate study by G.A. Harron: 839 500 st @ 0.230 oz/t (all categories) including 28 247 st @ 0.172 oz/t of broken ore left after production.			
1990	Hecla Mining Company of Canada – Acadia Mineral Venture Ltd.	In January 1990, a joint venture agreement was negotiated with Hecla Mining Company of Canada whereby Hecla could earn a 60% interest for placing the property back into production. After Hecla acquired control in Acadia Mineral Ventures Ltd, the Bachelor Lake property was assigned to that company. Acadia carried out 167.64 m (550') of underground drifting to establish 2 drill stations. 4 807.3 m (15 722') of DDH (in 34 holes) from various locations on the 11th and 12th levels. A number of significant gold intersections were cut, but establishing continuity between the intersections was difficult. The discrepancies were interpreted as being related to dykes and various faults. Summary Mineral Resources were estimated on the Main and B zones): 597 999 st @ 0.301 oz/t Au. Acadia cancelled the joint venture agreement. Only 25% of the planned program has been done.			
1992	Ross-Finlay	Before letting the U/G developments getting flooded, Ross-Finlay recovered broken ore as well as mining materials: there is no report of work carried out at this time. Representatives of Ross-Finlay indicated to Géospex (personal comm., 1993) that: ± 20 000 st @ ± 0.20 oz/t had been recovered.			

Bachelor Claims			Hewfran Claims		
Date	Company	Work description	Date	Company	Work description
			October 18th, 1993	QSR	4 companies have amalgamated into a new corporation called QSR. The companies involved are: Quebec Sturgeon River Mines (TSE), The Coniagas Mines (TSE), Anglo Dominion Gold Exploration (TSE) and Garrison Creek Consolidated Mines (CDN). Exchange ratios were: 18 QSR shares for 100 Anglo Dominion shares; 32 QSR shares for 100 Coniagas shares; seven (7) QSR shares for 1 000 Garrison Creek shares; and 20 QSR shares for 100 Quebec Sturgeon shares.
1994	Espalau Mining Corporation	Acquisition of 100% of the BLGM property.			
1995		Geospex realized the surface DDH (10 holes), 2 571.9 m (8 438’) as a follow-up program to a magnetometric and VLF surveys (GM 53978 and GM 53979).			
		Mineral Resources estimate of 204 454 short tons at 0.257 oz/t Au (Proven), 227 279 st at 0.315 oz/t Au (Probable) and 458 201 st at 0.289 oz/t Au (Inferred).			
1998	Ced-Or Corporation	Espalau Mining Corporation changes its name to Ced-Or Corporation.			
1999	Sabre capital partners inc.	Mineral Resource estimate of 204 454 tons at 0.257 oz/t Au (Measured), 216 685 st at 0.315 oz/t Au (Indicated), 256 285 st at 0.304 oz/t Au (Inferred). Mineral Resource estimate, mill conditions and environmental assessment by SNC-Lavalin.			
2001	*GéoNova Explorations Inc.* (subsidiary of Resources Campbell Inc.)	Acquisition of 100% interest in the BLGM property (March 2001), including buildings (offices, shops, dryers, compressor rooms, head frame, cyanidation plant and crusher room).			
		An audit of reserves/resources produced by Met-Chem for MSV resources, GéoNova and Campbell resources inc (identical to SNC-Lavalin).			
		Mineral Resource estimate of 204 454 st at 0.257 oz/t Au (Measured), 216 685 st at 0.315 oz/t Au (Indicated),			

Bachelor Claims			Hewfran Claims		
Date	Company	Work description	Date	Company	Work description
		256 285s t at 0.304 oz/t Au (Inferred).			
2003 – 2004	**Wolfden Resources Inc.**	**Wolfden signs an agreement to acquire a 50% interest in the property from GéoNova after incurring $3 000 000 in exploration over three (3) years.**			
		Dewatering started **(executed by CMAC). In October 2004, the dewatering level had reached mine levels 4 and 5 and should be completed by the end of February 2005.**			
October 2004	**Metanor Resources Inc.**	**Metanor is in the process of acquiring the 100% interest held by GéoNova for a total amount of $2.3M.**			
Oct.-Dec. 2004	Halo and Metanor	NI 43-101 Technical report by Innovexplo on the BLGM property. Resources were: - 204 454 st @ 0.257 oz/t (Measured), - 216 685 st @ 0.315 oz/t (Indicated), - 256 285 st @ 0.304 oz/t (Inferred).			
Nov.-Dec. 2004	**Halo Resources Inc.**	**Announcement of Letter of Intent (Heads of Agreement) with Wolfden to acquire a 50% interest in the Bachelor Lake property.**			
May-August 2005	Halo and Metanor	13 345.55 m (69 holes) were drilled from underground 12th level targeting "Main", "B" and "A" Zones **to test and increase resources below and laterally of 12th level.**			
Sept. 2005	**Halo and Metanor (50-50%)**	**Halo did satisfy its work commitments and other obligations to acquire a 50% undivided ownership interest in the property. The BLJV between Halo and Metanor is created.**			
October 2005	BLJV	New resource estimate and NI 43-101 technical report by Innovexplo (refer to item 19): Measured: 192 594 tonnes (212 299 st) at 8.81 g/t Au (0.256 oz/t Au). Indicated: 648 997 tonnes (715 397 st) at 7.51 g/t Au (0.218 oz/t Au). Inferred: 426 148 tonnes (469 748 st) at 6.51 g/t Au (0.189 oz/t Au).			

Date	Company	Work Description
Nov.2006	METANOR & HALO	Metanor signs a new agreement to acquire Halo's Resources Ltd 50% interest in the Bachelor Lake Property, Quebec. **Metanor announes the conclusion of a new purchase agreement whereby Metanor will purchase from Halo its 50% interest, the Hewfran property and the MJL-Hansen property (collectively, the 'Bachelor Property'). The new purchase provides for the payment by Metanor of $2,000,000 on November 20, 2006 of $500,000 on or before May 31, August and November 30, 2007 in cash or common share to Metanor's choice. Metanor also grants to Halo a 1% net smelter returns on all minerals or minerals products derived from The Bachelor Property.**
Nov.2006	Metanor	**Metanor begins a 2,500 meters surface drilling program on the Hewfran property. Following the completion of this program Metanor will be the sole owner of the Lac Bachelor property.**
Dec.2006	METANOR **and** MURGOR	Metanor signs an agreement to acquire Murgor's 100% of Barry open pit deposit**. The agreement includes a $200,000 cash payment on or before January 15th 2007 as well as a Royalty on the proceeds of sales of gold. Barry deposit resources were estimated in compliance with NI 43-101 by Systèmes Géostat International in April 2006 as,** **Indicated: 269,000 t at 4.10 g/t Au Inferred : 450,000 t at 4.68 g/t Au**
Jan.2007	METANOR **and** MURGOR	Metanor – Agreement to acquire Murgor's Barry property**. The royalty payable to Murgor is established to 9% of the sale's price of gold produced. The first advance on this royalty corresponding to $250,000 will be payable to Murgor by issuing 416,666 shares of Metanor. A second advance of $250,000 cash will be payable on the first of the two dates: 30 days after the issuance of the exploitation permit or on January 1st, 2008. The reimbursement of the advances will be made by a reduction of 50% of the amounts of royalty due to Murgor after the beginning of the commercial exploitation of the Barry deposit.**
Jan.2007	Metanor	Metanor increases the surface area of the Bachelor lake Property. **Following the acquisition of 63 new claims located easterly and jn continuity with the Bachelor lake property, Metanor will increased the area surface of the Bachelor Lake property of 2,787.2 hectares. The acquisition includes a $5,000 cash payment and the issuance of 200,000 common shares in favour of the vendor one year later. The counterpart includes a 2% net smelter revenues royalty redeemable as follows: 1% for $1 MCAD and the other 1% following the conditions which will be negotiated between the parties.**
Jan.2007	Metanor	Metanor starts refurbishing its Bachelor Lake gold mill**. This capital cost to restart the 500 tpd mill is estimated at $3.08 M by Genivar. A further capital investment of $3.09 M is estimated by Genivar to bring the mill capacity to 750 tpd**
Feb2007	Metanor	Metanor announces the first DDH results on the Bachelor deposit extension. **Preliminary results show that hole B06-132 intersected Hewfran's west zone grading 6.03 g/t Au over 5.15 m. More assays are expected from the laboratories.**
May 2007	Metanor	Metanor re-evaluates the Barry gold deposit **New resource re-evaluated by Geostat Systems International Inc. in compliance with NI 43-101 are now estimated at: 52,300 oz Au of Indicated Resources (385,000 t at 4.23 g/t Au) and,** **126,600 oz Au of Inferred Resources (966,000 t at 4.07 g/t Au) in zones 43, 45 and the southwest extension of the main zone.**

6. Geological Setting (Item 9)

This following information is part of the April 2007 NI 43-101 Technical Report on Resources Evaluation of the Barry-1 Property by Geostat.

6.1 Barry I Property

This following information is part of the April 2007 Technical Report on Resources Evaluation of the Barry-1 Property by Geostat.

Regional geology

The Barry property is located in the north segment of the Abitibi. It belongs to the Archean Abitibi Volcano-plutonic Sub-Province, part of the Superior Province of the Canadian Shield within the volcanogenic Harricana-Turgeon belt of the northwest part of the Sub-province of Abitibi. All rock types are of Archean age except the diabase dykes (Proterozoic). The Barry I property belongs to the metavolcanic metasedimentary belt, which extends from Wilson Lake to the Grenville Front. This sequence of rocks named the "Wilson-Marceau" is flanked to both the north and south by granitic massive intrusives. Greenschist-grade metamorphism is found throughout the region except in areas proximal to granitoid stocks where metamorphic grade is increased to amphibolite grade.

The Barry property covers largely the Casa Berardi Tectonic Zone, which includes several corridors of ductile E-W and ESE-WNW deformations. In the Harricana-Turgeon Belt, one can find the mining camps of Joutel, Matagami, Brouillan and Casa Berardi, where polymetallic volcanogenic clusters deposits (Estrades and Isle-Dieu), polymetallic veins deposits (Selbaie) and lode gold deposits (Casa Berardi, Vezza, Douay West and Détour) were discovered.

On a regional scale, the known mineralization of the Murgor property is hosted at the intersection of two major structures. These are the Murgor Shear Zone, a more intense portion of the 060° regional deformation, and the Mimer Shear, a 020° structure known to contain several small yet significant gold showings. Both of these larger scale structures associated with gold mineralization, are represented by quartz vein systems that can be seen in the Barry I Main Zone.

Other deposits in the Barry I deposit area

Murgor possess the Windfall property, located about 20 km northeast of the Barry I project. The mineralization found consists of a sericite-fuschite shear zone with 1-15% quartz-carbonate±tourmaline veins. These veins contain from traces up to 5% pyrite and locally some VG. The shear is between 2 to 5 metres thick, is oriented N060º and dipping NNW at 60-70º. Murgor is presently working on that property by geophysics surveys and drilling campaign. The mineralization appears close to the surface.

At least two other properties are close to Barry I present deposit and had resources evaluated. These deposits are not close to the surface and could be mined by underground operations. The published resources of these properties, which are not NI 430-101 compliant, are:

Property	Tonnage (mt)	Au g/t
Lac Rouleau deposit	544,000	7.0
Nubar deposit	564,000	6.2

Table 6.1: List of some of the deposits close to the Barry I property area.

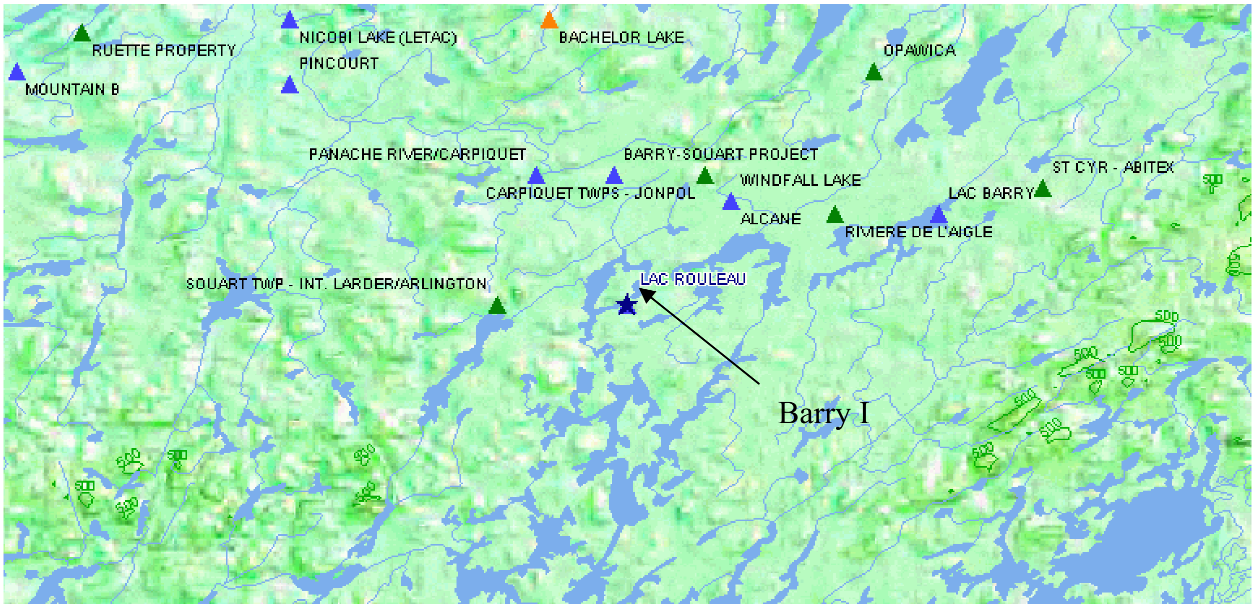


Figure 5.1: Aerial map showing some properties around the Barry I property (Infomine).

Local geology

Stratigraphy

The property is underlain by greenschist facies volcanic and intrusive rocks belonging to the Wilson-Marceau Volcanic Belt (Hocq, 1989). These rocks are of tholeiitic affinity. As there is limited outcrop exposure, the geology had to be deduced from drill holes data and geophysics. Geological mapping and diamond drilling identified a series of basaltic flows that are interpreted to cover over 90% of the property. The only intrusive bodies identified on the property were the quartz-feldspar porphyry in the area of the Barry I Main Zone Area and a series of gabbro sills to the north. An outcrop of siltstone was identified approximately 300 metres northeast of the Barry I Main Zone. Stratigraphic tops are to the southeast, as indicated by pillow facing directions. The rocks on the property are overprinted by a weak to moderate NE-SW trending foliation (S2) that parallels the regional shearing and the contacts of the large granitic intrusions.

The mafic volcanic rocks are the most common rocks on the property and consist of dark green, fine-grained, iron-rich tholeiitic basalts. In order of decreasing abundance, these flows vary from massive, amygdaloidal, brecciated, feldspar-phyric to locally pillow. Alteration varies from a regional chlorite alteration to locally carbonate, sericite, epidote plus minor silicification, hematization, biotite and actinolite alteration (Tessier 1996, Lariviere 1997). All these rocks vary from generally non-magnetic to locally strongly magnetic with up to 5% disseminated magnetite crystals and less commonly stringers of magnetite.

The mafic volcanic rocks in the area of the Barry I Main Zone Area are intruded by a series of porphyritic to granitic felsic dykes or sills. They are grey to pink in colour and contain up to 50% white feldspars, 15% blue quartz and 10% biotite phenocrysts ranging in size from two to l0 mm. The quartz-feldspar porphyry varies in colour from a fresh looking medium grey, to a reddish tint (due to hematization), to a bleached light grey (due to strong silicification). The quartz-feldspar-porphyry is “sill like”, maintaining a general stratigraphic position within the volcanic pile, while, at the same time, it can be seen crosscutting the volcanic stratigraphy on surface. The thickness of this unit varies from several metres to over 125 metres.

One can observe two sets of porphyritic to granitic felsic dykes or sills. The first set is foliated and shows 35% of feldspars and less than 5% of blue quartz-eyes. The second set of quartz-feldspar porphyry is not foliated and contains 8-12% of blue quartz-eyes and 50% of feldspars.

The gabbro is massive, medium to coarse-gained with a dark green colour. At times, the gabbro develops a finer gained gradational contact with the basalts and varies from moderately to non-magnetic. Drilling indicates that the gabbro is sill like and up to 20 metres thick.

Structure

The overburden had been removed in 1995 over most of the Barry I Main Zone Area. The bedrock had been mapped in 1995 and the maps help to understand the structure of the mineralized zones. The major aspects of the structure observed on the Barry I Main Zone Area can be summarised as follow:

- The impact of the major fault present at the northwest of the property, the Murgor Shear Zone, seems not very important, at least laterally.
- The displacement of one fault, mapped in 1995, occurring in the northwest part of the stripped zone, seems to be less than 100 metres laterally. The vertical movement is unknown.
- Occurrences of mapped folded zones suggest the presence of two major anticlines and one syncline. The orientation of the fold axes is southwest northeast. The plunges are variables, but generally sub-horizontal.
- Many deformational features are brittles (faults, fractures, veinlets, intrusives) to brittle-ductile (shear zones) and others are from the deformation of the ductile mafic formations (pillows deformation and boudinage).
- According to the interpretation from the 2006 drill holes, the limb of the southeast anticline extends deeper to the southeast to form the Zone 43. Some drill holes intersected the Zone 43.
- The Zone 43 can be interpreted as one side of a syncline, repeating the SW-NE undulating fold pattern.
- Minor north south faults, with displacement smaller than 10 metres, are developed on the mapped area.

- The main schistosity is 060°, dipping steeply to the southeast.

6.2 Bachelor Lake Property

Informations of this item are extracted from the InnovExplo Technical Report of December 2005 for the Bachelor Lake Property.

Geological Setting

The following sections describe the regional and local geological settings which have been adapted from the previous NI 43-101 technical report on the Bachelor Lake Gold Mine prepared by Innovexplo in December 2004.

Regional Geological Setting

The Bachelor Lake area is located within the Northern Volcanic Zone (NVZ) of the Abitibi Subprovince, Superior Province (Chown et al., 1992). The Bachelor Lake area is situated near the western limit of the Chibougamau-Chapais greenstone belt. The mafic to felsic volcanic and volcaniclastic rocks of the Bachelor Lake area are part of the basal mafic-dominated sequence referred to as the Volcanic Cycle I (Mueller et al., 1989). The Volcanic Cycle I formed between 2730 and 2720 Ma (Mortensen, 1993), and is composed of massive, pillowed and brecciated, tholeiitic basalt flows with local felsic and sedimentary units. The Northern Volcanic Zone of the Abitibi Subprovince is interpreted as a diffuse arc passing laterally into a back-arc environment with numerous felsic and mafic-felsic edifices (Chown et al., 1992) and intra-arc sedimentary basins (Mueller et al., 1996).

The Bachelor Lake property lies along a local northeast-trend which is deviated from the general east-west pattern of the Abitibi Subprovince due to significant synvolcanic pluton emplacement and the influence of the major northeast-trending Wedding-Lamarck fault in the Bachelor Lake area (Doucet et al., 1998). This general trend includes several mines as Agnico-Eagle's Telbel mine, Golden Hope's Estrades deposit and other deposits in Douay Township. Other deposits in this area include the Lac Shortt gold mine, the Joe Mann gold mine, the Zn-Pb-Ag massive sulphide Coniagas mine and the Cu-Zn massive sulphide deposit of the Gonzague-Langlois mine (Grevet).

Local Geological Setting

The property is underlain by Archean volcanic rocks of the Obatogamau Formation in a poorly known and poorly explored area of the Abitibi greenstone belt. Because of the absence of marker horizons and the paucity of outcrops, it is difficult to establish a well-defined rock sequence in the Coniagas-Bachelor Lake area (Doucet et al., 1998). The Obatogamau Formation includes mafic, intermediate and felsic flows and their synvolcanic intrusive equivalents which are the host for the volcanogenic massive sulphide occurrences (e.g. Coniagas). A local composite stratigraphic section shows a typical complex volcano-sedimentary assemblage (Figure 7.1). This stratigraphic sequence includes the 280 m thick Coniagas mine sequence represented by a mafic-dominated volcanoclastic sequence. Porphyritic lava flows, prominent in the immediate area of the Coniagas Zn-Pb-Ag deposit (1.5 km west of Bachelor Lake deposit), cover the volcanoclastic unit. A significant 500-700 m thick, lenticular and dome-shaped felsic unit composed of massive to brecciated rhyolitic to rhyodacitic lava flows occurs up-section. This felsic-dominated unit corresponds to the Bachelor Lake Au deposit host rocks. Mafic volcanic and volcanoclastic rocks make up the upper part of the sequence. The Auger Lake and Bachelor Lake sedimentary rocks remain enigmatic but probably mark the top of the sequence. The Late emplacement of several plutons (e.g. O'Brien granodiorititic pluton located east of the Bachelor Lake deposit), adds to the complexity of the region. Gold mineralization at Bachelor Lake has been interpreted to be related to the late granodioritic O'Brien pluton (Buro, 1984 and Lauzière, 1989). Intrusive rocks related to the O'Brien pluton include granitic porphyry and biotite-hornblende granodiorite. Post-tectonic lamprophyre dykes are also common at the Bachelor Lake mine and kimberlitic dykes were documented in the Desmaraisville area. This later intrusive phase (N030° and N110° lamprophyre and kimberlitic dykes) has recently been investigated for their diamond potential in the Desmaraisville area.The local northeast-trending sequence deviates from the general east-west pattern of the Abitibi Subprovince due to the presence of significant pluton emplacement and the influence of the major northeast-trending Wedding-Lamarck fault. The folded volcanic rock sequence (see Compilation Map in Appendix X) shows local changes in trend from N025° to N065°, with vertical to steep northwest dips (60° to 77°). Folding and faulting are responsible for stratigraphic repetition and disruption of the volcano-sedimentary sequence. Foliation relationships indicate a possible third phase of deformation (Sharma and Lauzière, 1983).

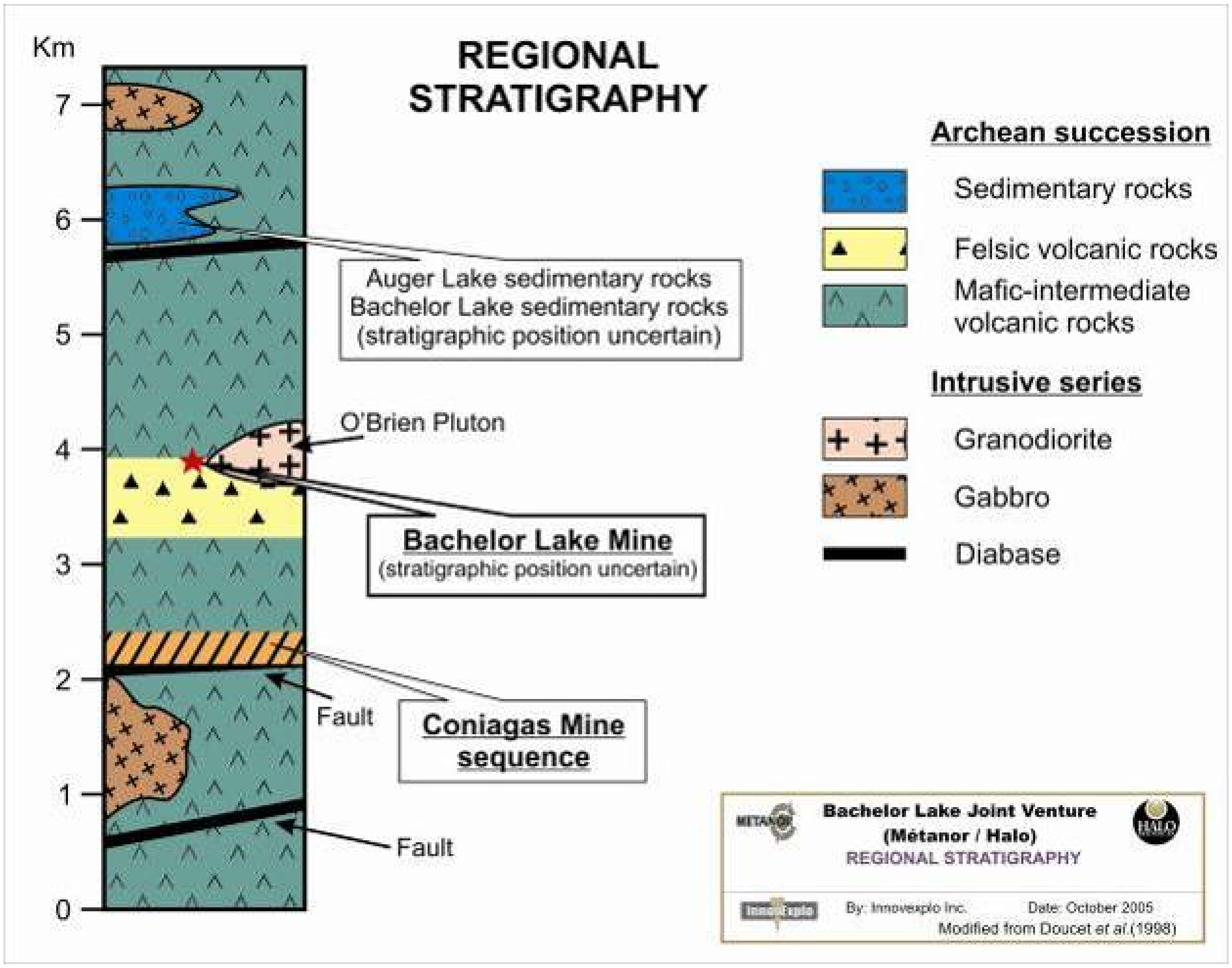


Figure 6.2: Composite stratigraphic column of the Desmaraisville area
(modified from Doucet *et al.*, 1998)

At the Bachelor Lake mine, most deformational features are brittle (faults, fractures, veinlets) to brittle-ductile (shear zone).

Based on Lauzière (1989) study and the last drilling program, five (5) post-ore fault systems striking N110° are recognized on the property and affect the gold-bearing zones at Bachelor Lake:

1) Flat-lying faults, generally small displacement which appears to be along strike of Main Zone. These veins are well illustrated on level plan views where mineralized zones show local metric discontinuities;
2) ENE brittle-ductile, rotational faults moderately dipping at 60°, namely the WAC (Waconichi faults system) and showing an oblique slip with dextral and reverse movement (Lauzière, 1989);
3) ENE brittle-ductile rotational faults steeply dipping at 80°, namely the WAC' (Waconichi faults system) and which could be interpreted as a conjugate to the WAC;
4) NNE to NE late brittle faults steeply dipping to the NW, transverse faults, namely the T. A good example of this fault is shown on the 12th level plan view between sections 50 W

and 1+00 E where the "Main", "B" and "A" zones are all dislocated as blocks in a late brittle fault corridor;

5) NW brittle faults moderately to steeply dipping (65° to 90°), namely the T', could be interpreted as conjugates to the T brittle fault system.

In the Waconichi fault system, the Big WAC fault is one of the most significant and, according to the upper description, should be related to the WAC fault system. The last underground drilling campaign demonstrates that the Big WAC may have two (2) major impacts at Bachelor: (i) by locally remobilizing gold (higher grade) and (ii) by dislocating or displacing the "Main" and "B" Zones (missing in the footwall of the Big WAC fault). The movement on it may be approximately 15 m (50'), it often tends to be adjacent to the "Main" Zone (10th to 12th Levels) and, at depth, on 13th to 15th Levels, it tends to occur between the "Main" and "B" zones. Interpretation also demonstrates that when faults cross the "Main" and "B" zones, thickening of the zones occurs.

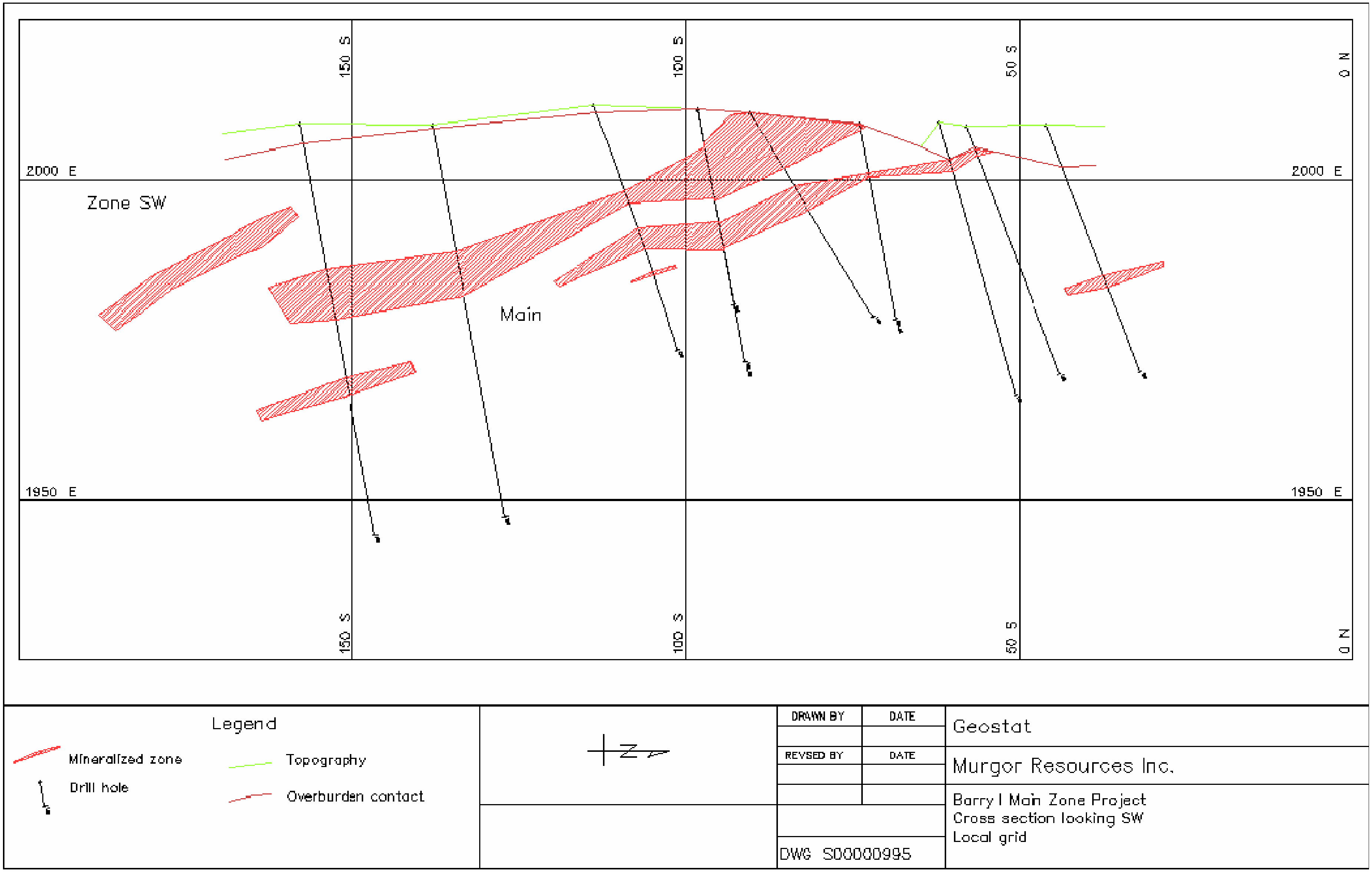
7. Deposit type (Item 10)

This following information is part of the April 2007 Technical Report on Resources Evaluation of the Barry-1 Property by Geostat.

7.1 Barry I Property

The Barry I Main Zone Area deposit is structurally complex. Its genesis seems to be from hydrothermal origin, with strong structural and chemical controls. The information acquired from the holes drilled between 2004 and 2006 offers a new perspective and a better understanding of the Barry I Main Zone Area mineralization and the Zones 43 and 45. It becomes obvious that the presence of gold-bearing mineralization is not only possible in the limbs of the fold, as previously taught, but also at the top of the fold.

The next figures present the spatial organization of the mineralization, which we can find in the Barry I Main Zone Area and Zone 43 and 45. The sections are section 995 E and 1111 E.



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Geostat

Murgor Resources Inc.

Barry I Main Zone Project
Cross section looking SW
Local grid

Figure 7.1: Schematic section 995 E of the spatial organization of the mineralization – Barry I Main Zone.

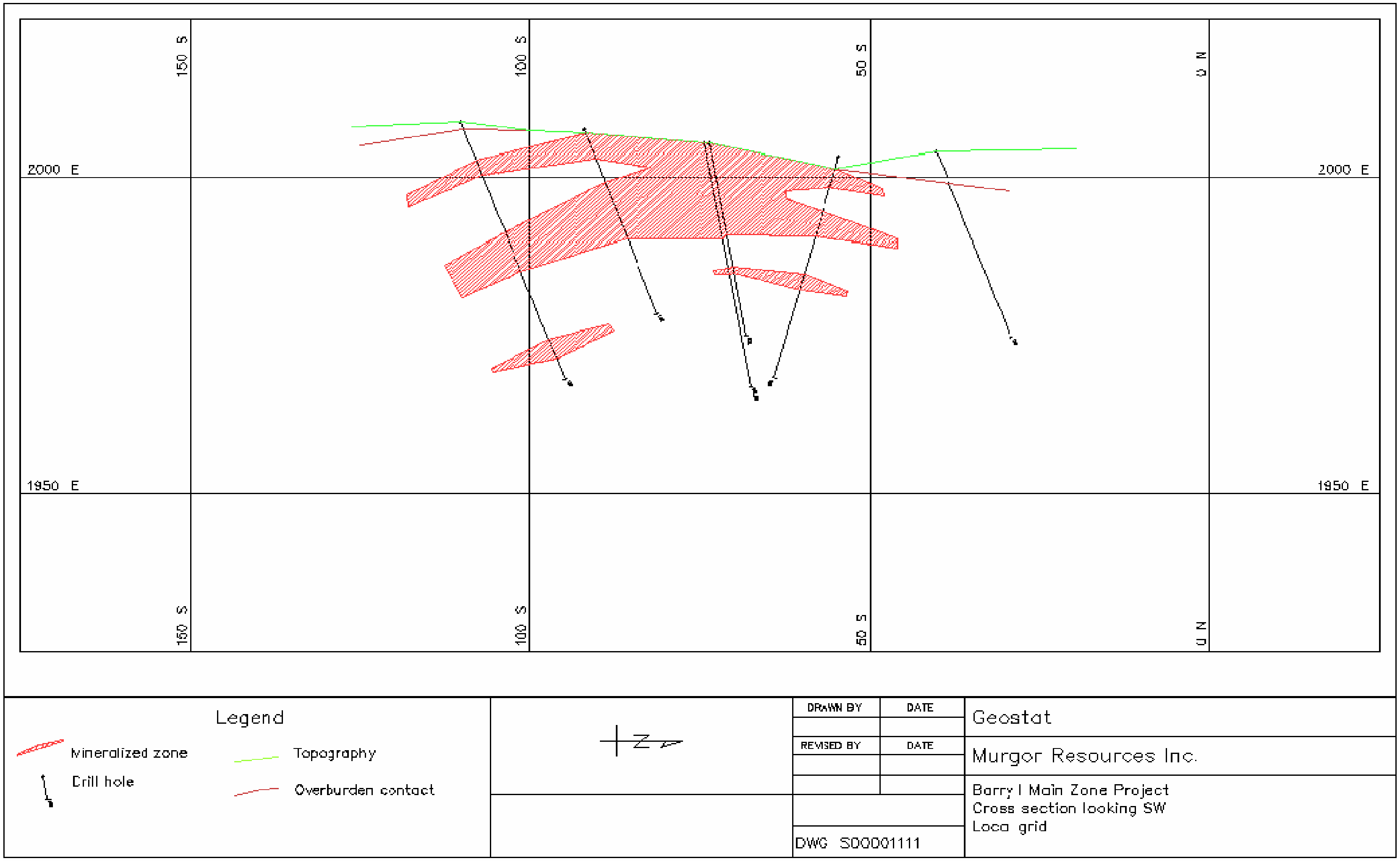


Figure 7.2: Schematic section 1111 E of the spatial organization of the mineralization – Barry I Main Zone.

The Barry I Main Zone Area is composed of steep southeast dipping (60°) quartz-carbonates, biotite and ankerite veins. The ankerite alteration extends outside the quartz veinlets. This alteration halo is very proximal and may extend for up to 30 cm on either side of the narrow 1 to 5 cm quartz carbonates veinlets. Where there is a high frequency of veins, the alteration veins merge, producing alteration zones of over 20 metres in thickness in certain areas. The biotite and ankerite quartz carbonates veinlets extend in depth, as they can be followed in the deepest drill holes. Their gold contain is variable.

The Main Zone Area gold mineralized envelopes are located almost flat lying at the top of a dome shaped fold. The vertical thickness of the gold bearing envelopes varies between one up to 20 metres. Their length is variable because they are pinching and swelling. The southwest northeast extension (up to 300 metres) of the envelopes zone is more important than the southeast northwest extension (60 to 85 metres). The first top 35 metres of rock contains most of the continuous gold mineralized envelopes. One small extension toward the southeast of the main zone, along the side of the anticline and in the top of the anticline, is defined by drilling in the southwest part of the main zone, further southwest of the quartz-porphyry intrusion.

The dome shaped design of the mineralized zone seems to repeat itself in the northwest part of the Main Zone. A new interpretation of the mapped stratigraphy suggests the presence of another anticline in the northwest part of the exposed rock of the Main Zone Area (Figure 16).

The Zone 43, identified by drilling, is located on the southeast limb of the Main Zone Area anticline, higher in the stratigraphy than the main zone. Six drill holes were drilled in 2005-2006 to better understand and explore this zone. The northeast part of the Zone 43, located south of the Main Zone, seems to be cut or displaced by a quartz-porphyry intrusion located just at the southeast of the Main Zone. The type of mineralization needs to be study.

The Zone 45 is the possible extension of the Main Zone, to the southeast. The zone is present on the sections 700 and 750. It is possible to join this zone with the Main Zone at depth. The zone is possibly open to the east. The collars of the 107, 120 and 156 need to be surveyed to assure their local position on the section 800E

The presumed sequence leading to the presence of gold mineralization is the following:

- Lavas deposition
- First alteration period– silicification and set-up of the first sequence of quartz-carbonate-fuschite veins. The gold was distributed within the quartz veins and in the host rocks. The gold is present in disseminated and in nugget form. The nuggets can reach up to 1 mm.
- Intrusion, after the alteration period, of the quartz-porphyry complexes.
- Shear period, set-up of the presently visible foliation.
- Second alteration period – Silicification and set-up of the second set of milky quartz veins, none neither folded nor sheared.

The next figure presents a typical cross-section of the Barry I Main Zone.

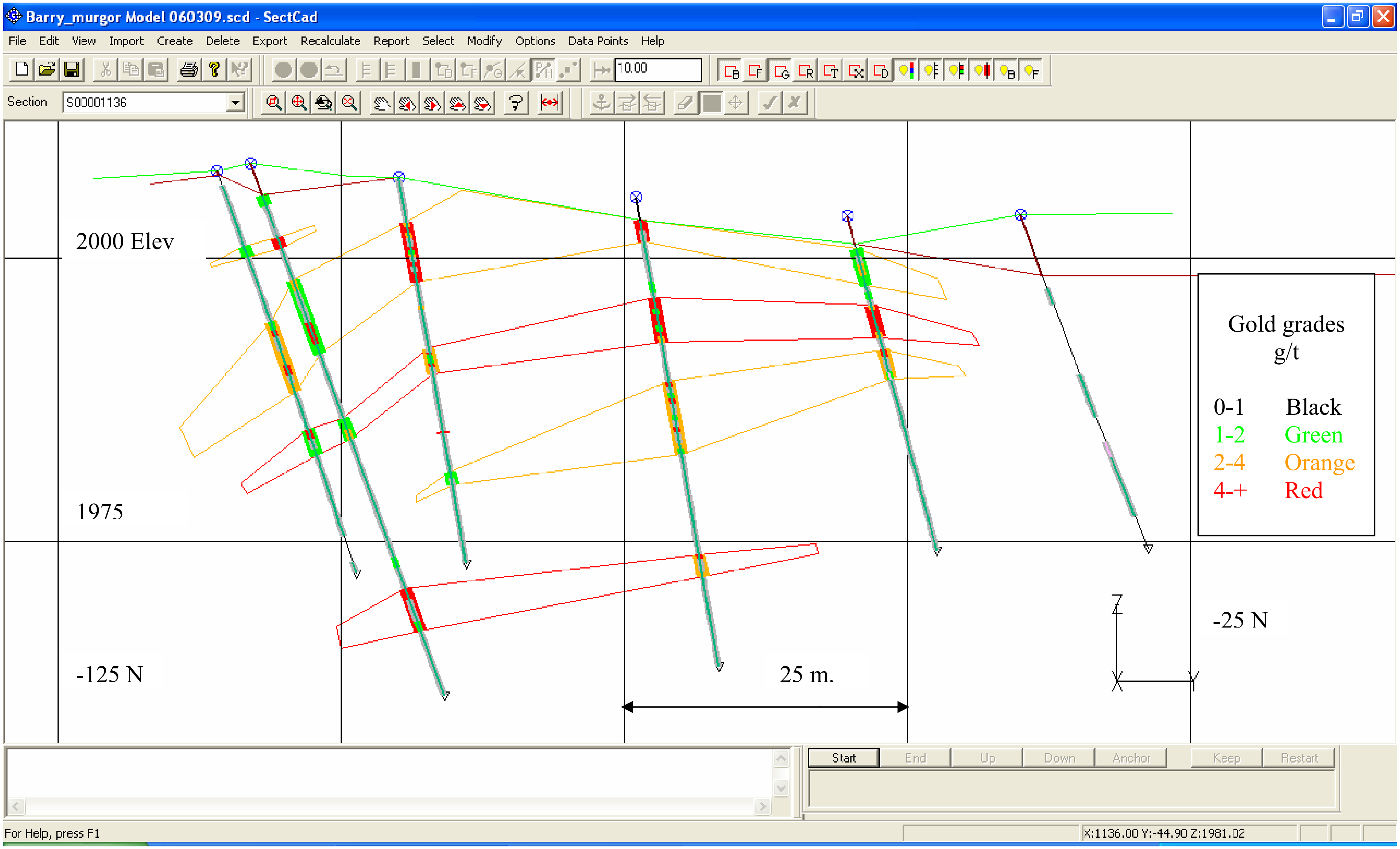


Figure7.3: Cross-section looking west of the Main mineralized zone (grid lines are at 25 m).

7.2 Bachelor Lake Property

Informations of this item are extracted from the InnovExplo Technical Report of December 2005 for the Bachelor Lake Property.

Deposit Types

The property hosts a wide variety of deposit types from volcanogenic polymetallic type to syn- to late-orogenic gold mineralization (Fig. 7.4). On the property, volcanic-hosted massive sulphide potential is illustrated by the Coniagas Horizon, Zinc Showing #1 and #2, Area-Opawica showings and by the Coniagas deposit located on the adjacent property.

The Bachelor Lake gold mineralization is related to brittle deformational features and dilatational zones (stockwork) and to brittle-ductile shear zones. The Bachelor gold deposit can be either classified an "*orogenic lode gold deposit*" or an *"intrusion-related gold deposit"*. The gold distribution appears to be controlled by both structural and lithological features (e.g. the rhyolite being more fractured compared to the agglomerate) (Y. Buro, personal communication, 2004).

The Bachelor Lake gold mineralization has also been interpreted to be associated with the late-tectonic granitic to granodioritic intrusion (O'Brien pluton located east of the deposit and associated dykes documented at the mine). The link between the late intrusive rocks and the gold distribution can be interpreted as either: the result of a litho-structural relationship (i.e. lithological contrasts), or, as a magmatic process (intrusion-related, oxydized magma). According to Buro (1984) and Lauzière (1989), the O'Brien granodioritic stock probably provided the concentrating mechanisms through heat and hydrothermal solutions. The late phase dykes related to the O'Brien stock were introduced later than the shearing event, and the gold mineralization event has been bracketed between the occurrence of these late dykes and the earlier granodioritic phase (Lauzière, 1989). The high fluorine content of the hydrothermal biotite in the ore zone alteration correlates with that of magmatic biotite within the intrusive phases. There is probably a direct genetic link between the O'Brien stock and the gold mineralization (Fig. 7.4 B.).

In this perspective, the Bachelor Lake gold deposit may well correspond to the new class of gold deposits introduced by Robert (1997) in the southern Abitibi Belt: « Syenite-Associated Disseminated Gold Deposit ». In this class of deposits, the ore bodies usually consist of zones of **disseminated sulphides** and variably developed **stockworks** associated with **intensely altered wallrocks**. The mineralization, with sharp to diffuse limits, is defined by a decrease in sulphide content, gold grades and intensity of stockwork fracturing (Robert, 2001). « Intrusion-Related Disseminated Gold Deposit », rather than « Syenite-Associated », may be more appropriate class heading to describe the Bachelor Lake gold deposit. Gold remobilization along the "A" shear and mineralized zone may well represent another event as illustrated in Figure 7.4 C.).

From a descriptive point of view, Brisson and Guha (1993) have documented two (2) main types of gold mineralization occurring in the Bachelor Lake area and in the Wachigamau Member:

1) Gold-bearing quartz veins with gold disseminated sulphides in wallrocks, and
2) Disseminated gold-bearing sulphide zones.

During the last drilling program, these differences have been recognized and can be illustrated as the "B" and "Main" zones. They were interpreted as contemporaneous disseminated gold-bearing sulphide zones, the B Zone just superseding the Main Zone formation.

In both types, the mineralization is characterized and dominated by pyrite. Gold is: (i) native and is in close association with pyrite, or, (ii) free in quartz predominant veins. The mineralization is found in close association with hydrothermal alteration zones (silica-hematite alteration) which have been superimposed on the regional metamorphic minerals.

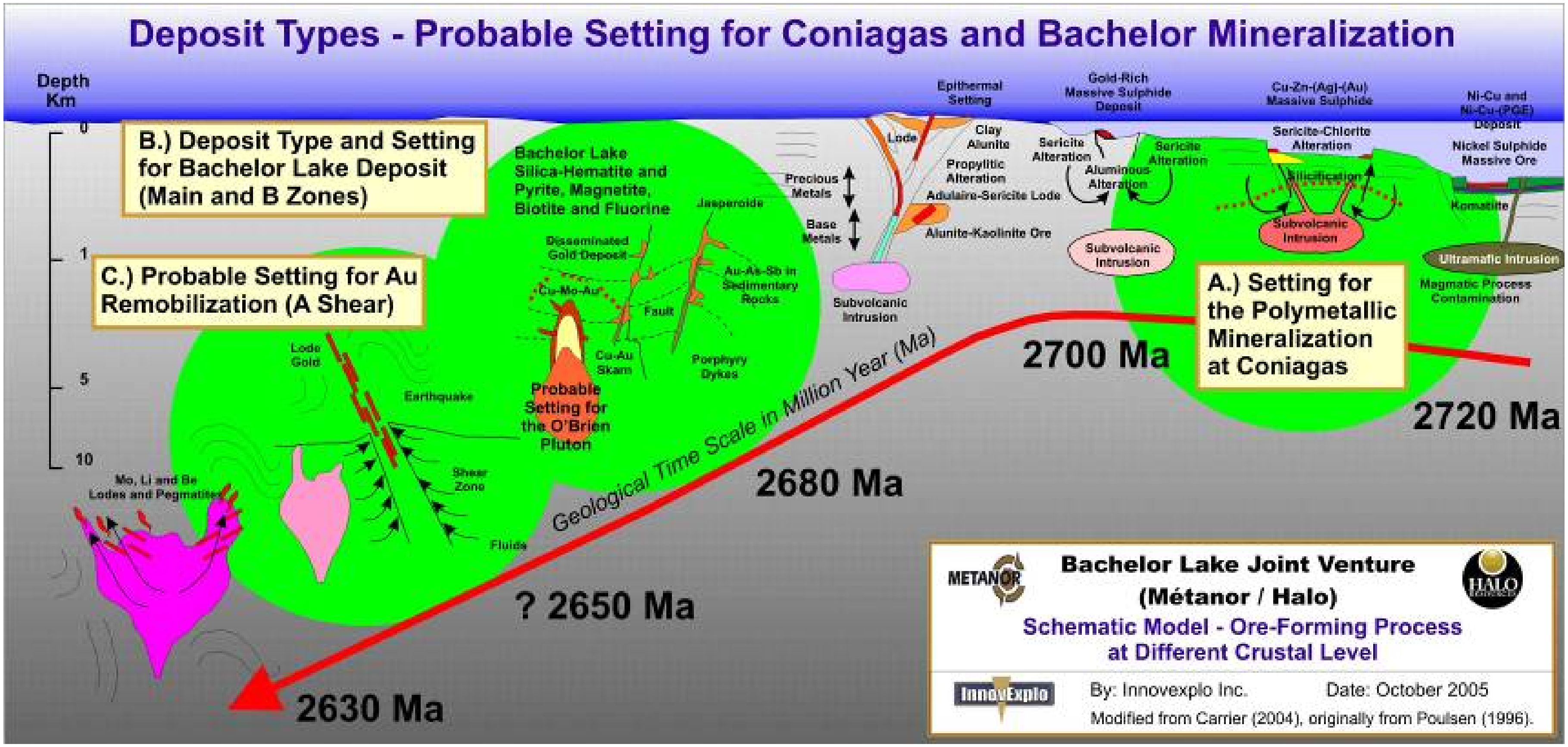


Figure 7.4 - Setting for Coniagas polymetallic massive sulphide mineralization and Bachelor Lake silica-hematite disseminated gold mineralization.

8. Mineralization (Item 11)

This following information is part of the April 2007 Technical Report on Resources Evaluation of the Barry-1 Property by Geostat.

8.1 Barry I Property

The Barry I Main Zone Area type mineralization

Gold mineralization on the property occurs for the most part in a system of sheeted quartz-carbonate (ankerite) - albite veins, veinlets in the associated proximal alteration halos of biotite-carbonate and disseminated pyrite. The gold occurs as free gold in gangue minerals within veins and altered wall rocks, as well as along micro fractures in pyrite (Lariviere, 1997).

Quartz Veining

From the mapping of the showing, one can observe that the main quartz veins system, which accounts for approximately 90% of the gold bearing veins, consist of sheeted veins. The dominant veins are oriented at 040° to 060°, parallel to the region foliation, and dip 62° to the SE (Tessier, 1996). The veins are surprisingly continuous for their thickness, which generally does not exceed 5 cm, yet at times, they extend for over 50 metres along strike. They are not folded. Their sulphides contain is low and they seems to appear lately in the sequence.

The second set of quartz veins is less developed, oriented at 020° on the showing and at an angle to the drill core very variable, and is parallel to the Milner Shear and dip 73° to the east. These veins can be distinguished by their well-developed crenulations, or folding, bulging and is continuous over several metres at best. One can find the association quartz-carbonates-(ankerite)-fuschite in these folded veins. They show a brownish color and contain a variable amount of pyrite. They are found at the contact of the alteration zones and are possibly concordant with the schistosity.

Although both sets of quartz veins crosscut each other and are composed of the same sugary quartz-ankerite-albite mixture, they suggest an asynchronous time of injection. They are post depositional but some are anterior to the folding and the others posterior. Their gold mineralization seems to come from the remobilization of the gold during the folding period.

Alteration

Alteration mineralogy includes.

The alteration of the basalts most commonly associated with the gold mineralization on the property consists of a very fine mixture of albite, carbonate, biotite, ankerite, epidote, chlorite, sericite and garnet. This alteration halo is very proximal and may extend for up to 30 cm on either side of the narrow 1 to 5 cm quartz veins. Where there is a high frequency of veins, the alteration veins amalgamate together producing alteration zones of over 20 metres in thickness. In fresh core samples, the alteration may be distinguished by its darker grey colour due to the biotite, in contrast to the green more chloritic, non-mineralized sections. On surface, the oxidization of iron in the ankerite produces a very noticeable rusty brown colour.

A less common form of proximal alteration, often associated with the better gold mineralization, is an intense bleaching due to albitization and a local reddish colour produced due to hematization.

A larger, broader alteration package that generally encompasses the mineralized basalts is characterized by the presence of chlorite-sericite-calcite-magnetite crosscut by a magnetite destructive proximal halo of biotite-ankerite-albite and pyrite. This alteration, locally referred to as "texture destructive alteration", may be as much as 80 metres thick and is commonly found close to the quartz-feldspar-porphyry basalt contact (Tessier, 1996).

The alteration associated with gold mineralization in the quartz-feldspar-porphyry unit is generally a strong bleaching and texture destructive alteration caused by silicification. This alteration is most commonly found within the quartz-feldspar-porphyry at the contact with the basalts.

Sulphides

Sulphide mineralization is mainly pyrite, which can account for 3 to 5% of the rock, but can be as high as 10%. It is commonly finely disseminated in the host rock as sulphide haloes to the quartz veins. Locally coarse cubic pyrite, up to 10 mm in size, is present. Both the fine and coarse pyrite are found associated with visible gold. Pyrrhotite is also present, although less common than pyrite, it varies from one to 5%. Chalcopyrite and sphalerite have also been identified but are not common.

Gold mineralization

The gold mineralization on the property is closely related to the amount of veining, intensity of alteration and percentage of sulphides. All are key factors and generally all three of these elements are needed in order to obtain significant gold mineralization. This style of mineralization produces sections with significant gold concentrations but they are commonly narrow with widths in the order of 0.3 to 1.5 metres. The thicker mineralized sections represent a higher density of these narrow zones.

The presence of the gold mineralization located at the top of the dome shaped is associated with silicified volcanites. At depth greater than 30 metres, the volcanic rocks are more mafic, massive and show the presence of vacuoles.

The majority of the mineralized sections are located within the silicified-carbonated basalts close to the contacts with the quartz-feldspar-porphyry. It is thought that the emplacement of the quartz-feldspar-porphyry is significant in the ground preparation. The emplacement of the porphyry body is thought to have increased the fracture-induced permeability of the basalts and created the conducts necessary for gold bearing hydrothermal fluids to circulate (Lariviere, 1997). The motor of the leaching process can be the intrusion of the first sequence of quartz-feldspar-porphyry. The hydrothermal fluids leached surrounding carbonated rocks and created carbon-based acid that had dissolved their gold contain. As the oxide-reduction potential of the environment changed, the carbonated-gold rich solution precipitated in the fracture zone presents at the top of the dome-shaped folds. The mineralized quartz-carbonate-albite veins, when extending from the basalts into the quartz-feldspar-porphyry, rarely contain any significant gold mineralization.

The folding periods having happened after the intrusion of the first porphyry body might have remobilized the gold mineralisation in the fold noses.

Figure 8.1: NQ Core of well-mineralized zone in the hole MB-05-162 (162) of December 2005.

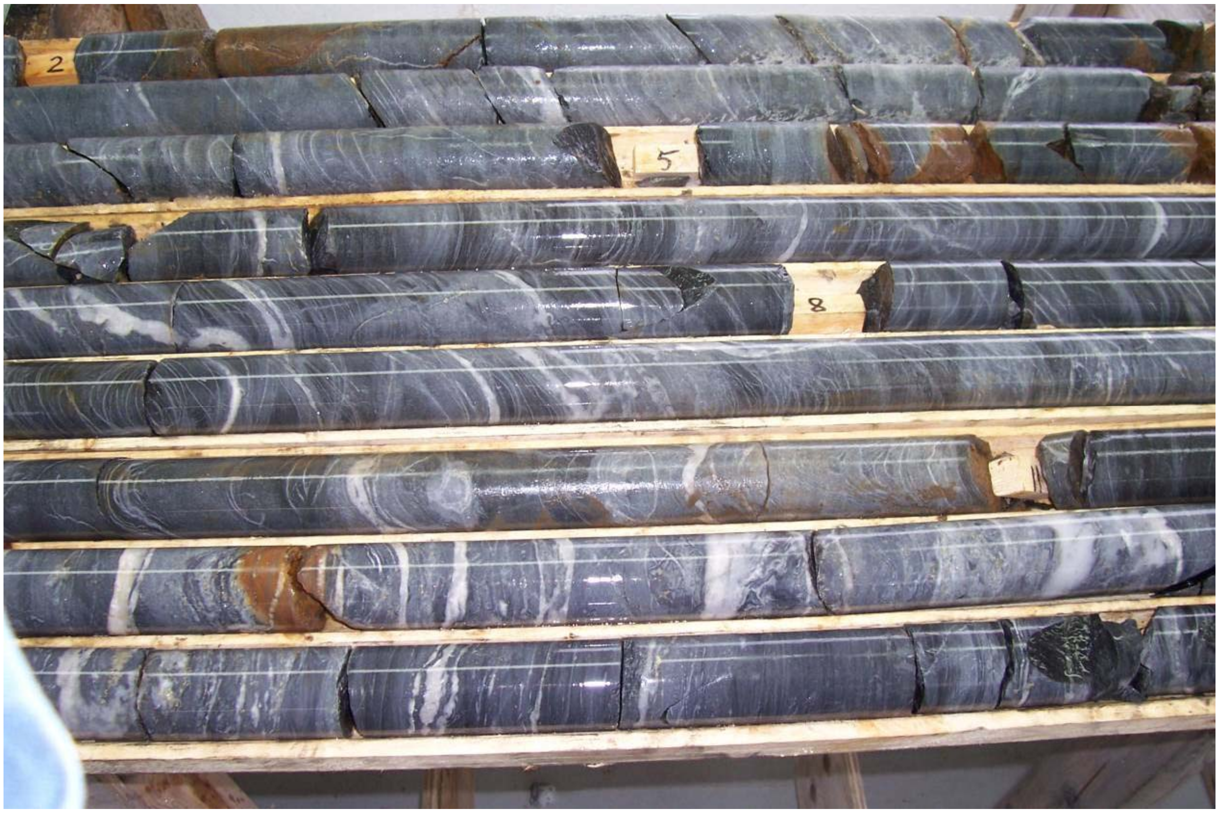


Figure 6: NQ Core of well-mineralized zone in the hole MB-05-162 (162) of Dec. 2005.

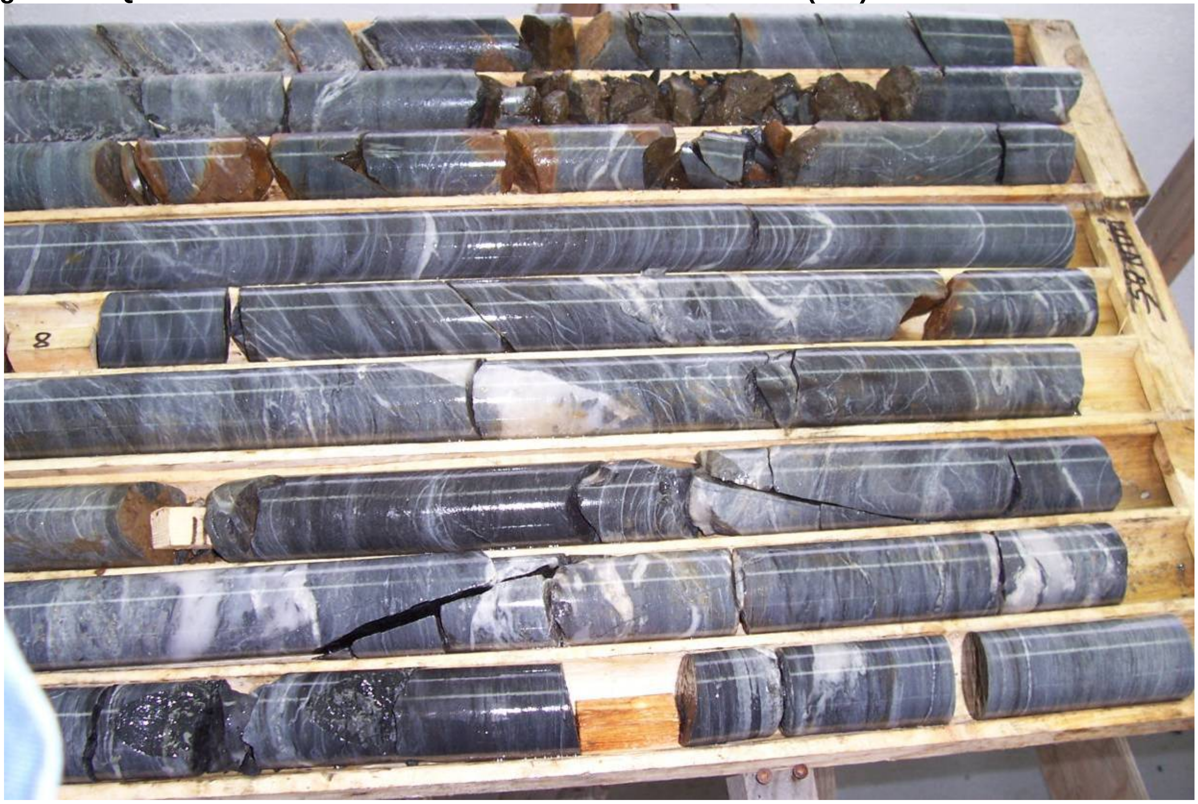


Figure 8.3: NQ Core of well-mineralized zone in the hole MB-05-162 (162) of Dec. 2005.

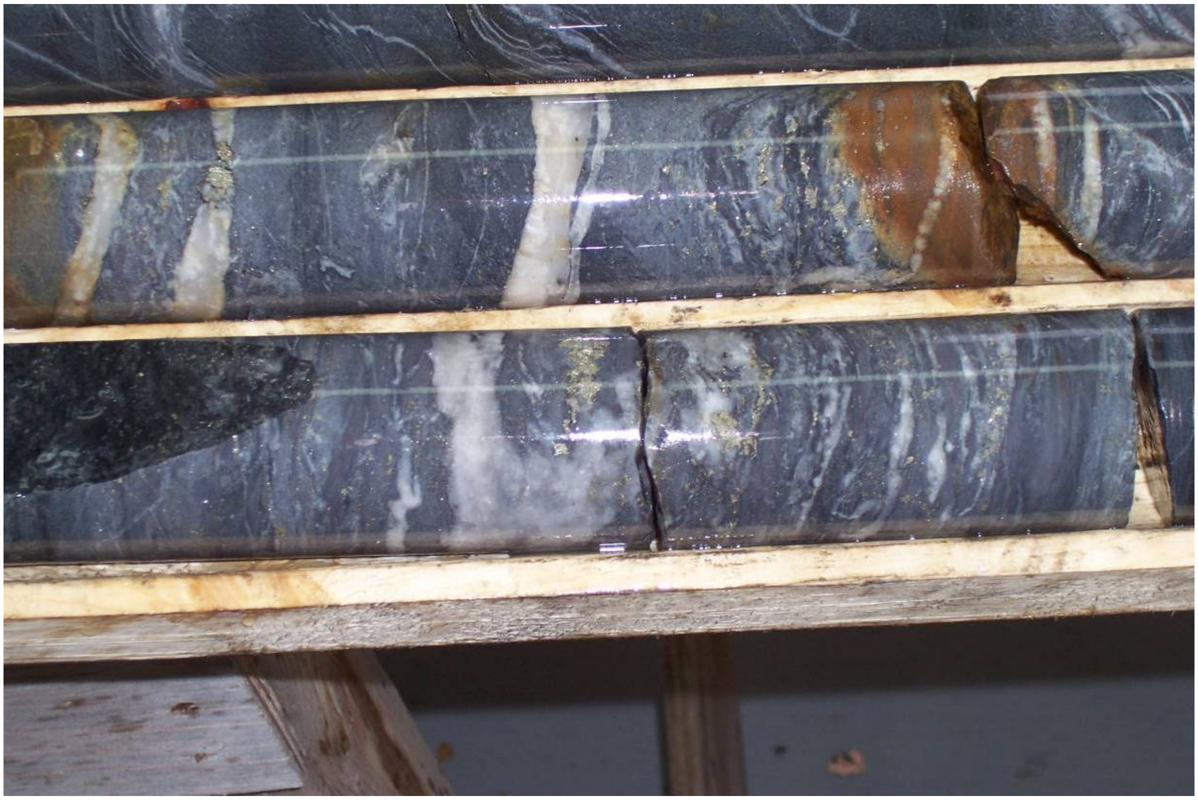


Figure 8.4: Close view of the mineralized zone in the hole MB-05-162 (162) of Dec.2005.

Figure 8.5: Presence of visible gold in the hole MB-05-162 (162) of December 2005.

Other mineralized zones on the property

The following is a brief description of the different mineralized zones on the Barry I property. Geostat did not recalculate the resources of these other mineralized zones

Numerous other gold showings are present on the Barry I property. These principal zones are the SW extension, the zones 45 and 48. These zones were defined by geophysics and tested by diamond drill holes. The core shows interesting mineralized intersections and they need further drilling. The zone 51 was interpreted by geophysics and need to be drilled.

The best intersection is located on the section 700 E in the Zone 45. The grade is 9.7 g/t Au over 7.7 metres. This intersection can be connected with two holes present on the section 750E.

More than 50 different gold occurrences are widespread over a surface of 1,000 x 350 metres, with many porphyry dykes intersecting the volcanics on the property.

The next two figures present a schematic view of the Barry I Main Zone Area and a cross section of the mineralized zones.

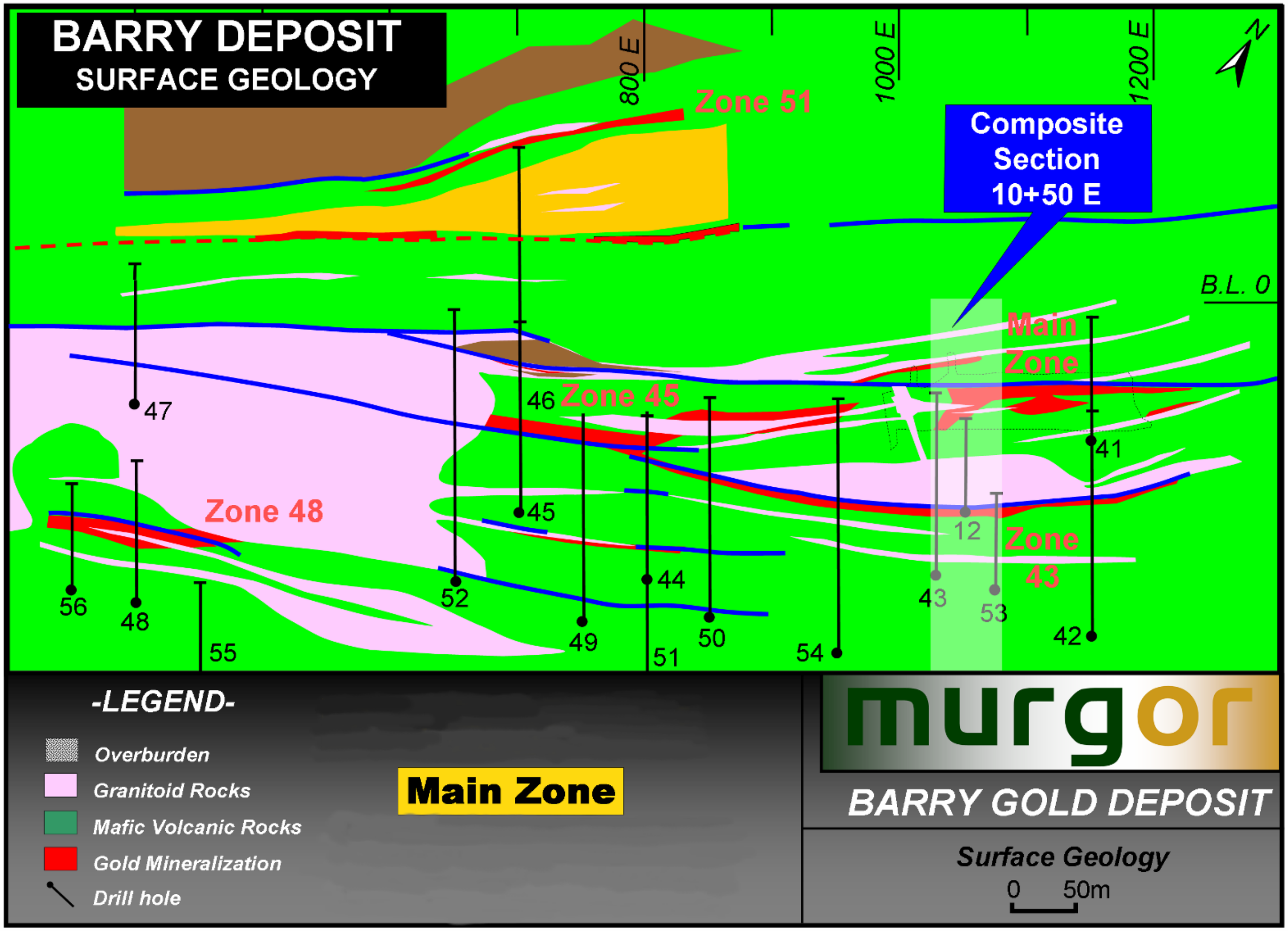
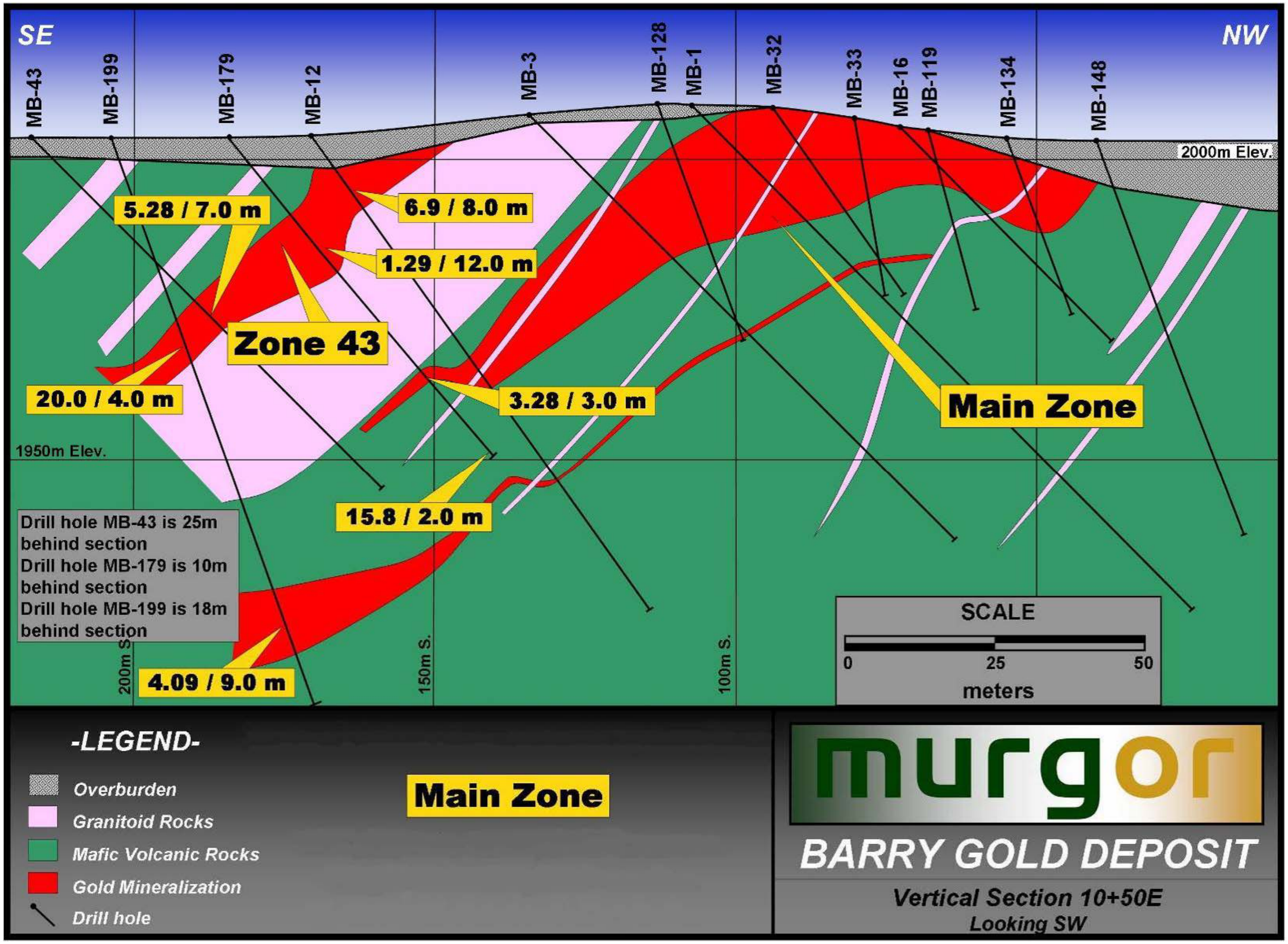


Figure 8.6: Schematic plan view of the Barry I property showing only a few holes.



Drill hole MB-43 is 25m behind section
Drill hole MB-179 is 10m behind section
Drill hole MB-199 is 18m behind section

Figure 8.7: Schematic & composite vertical cross-section 10+50 E showing the geological interpretation of the main and 43 zones.

8.2 Bachelor Lake Property

Informations extracted from the InnovExplo NI 43-101 report of December 2005

Property surface showings

Mineralization on the property was discovered from surface exploration in 1946. The property hosts several gold and base metal showings occurring on surface and illustrated by numerous showings: Agar #1 (Au-Zn), Agar #2 (Au), Area-Opawica (Zn-Cu-Ag), O'Brien showing (Au) which is also the original discovery at Bachelor, Terri and Middle showings (Au), Valdex (Au), Zinc showing #1 (Zn), Zinc showing #2 (Zn), and Hole 19501-52 occurrence (Zn-Au). The property also hosts the eastern extension of the Coniagas marker horizon (Zn-Pb-Ag). Illustrations and geological description of these showings can be found in Appendix IV and their location is plotted on the geological Compilation map in Appendix X.

Bachelor-type gold-bearing zones

The property hosts six (6) gold-bearing zones ("Main", "A", "B", "C", "A West" and "B West" zones) which were all included in the 2005 resource estimate.

For illustrations of the mineralization and description of the alteration, refer to Appendix IV.

The Bachelor Lake gold deposit is located along an ESE-trending, SW-dipping, silicified shear zone with hematitic alteration (Buro, 1984). It transects NE-trending, steeply dipping volcanic rocks and the O'Brien granitic to granodioritic pluton. Major W-SW and N-NE trending faults have affected the ore zone and the emplacement of the granite intrusions. Movements along the WSW set may have opened the fractures filled by mineralization (Buro, 1984).

Two (2) types of gold-bearing zones have been identified at Bachelor Lake: silica-flooding and hematite-altered ± stockwork zones, illustrated by the "**Main Zone**" and the "**B Zone**". In both cases, gold is spatially associated with pyrite and the gold content correlates well with the pyrite content as illustrated in the Figure IV-10 in Appendix IV.

Gold mineralization at Bachelor occurs predominately within the pyrite (>70%), as grains attached to the pyrite (-18%) or as free gold enclosed in the gangue (-10%). This was demonstrated in a polished-thin section examination done on the Hewfran claims (Table IX-2 in Appendix IX). The gold is fine grained with an average diameter between 6 to 8 mm (Table IX-2 in Appendix IX), and visible gold (VG) is more characteristic of the "B Zone". Pyrite is usually finely disseminated (2 to 10%) hosted in strongly altered rocks, often brecciated and occasionally injected by quartz/carbonate veins and veinlets. At surface, traces of gold, chalcopyrite and ilmenite occurrences have been observed. Gold has been introduced late in the paragenetic sequence as were fluorite and some of the carbonates (Lauzière, 1989).

Geometry, strike, and dip of the six (6) zones (“Main”, “A”, “B”, “C”, “A” West and “B” West) interpreted for the 2005 resource estimate are described below and illustrated on a schematic vertical section (Figure 9.1). They are also illustrated on plan views at different elevations on the Figures IV-3, IV-4, IV-5 in Appendix IV.

Figure 9.2 and Figure IV-7 (Appendix IV) illustrate the 3D projections of the wire frame model for the Bachelor gold-bearing zones. Figures IV-8 and IV-9 (Appendix IV) illustrate characteristic geological features of the “Main” and “B” zones (pictures of core samples). The relationship between gold values and alteration zonation is illustrated on the Figure IV-10 in Appendix IV.

The “Main” Zone:

The “Main” Zone has contributed 90% of the ore derived from the Bachelor Lake Gold Mine.

The “Main” zone is characterized by pervasive moderate to strong silicification and hematitization with 2-10% pyrite generally associated with hematite alteration. It is cross-cut by quartz-carbonate veinlets usually less than 2 cm, and some local narrow late siliceous hydraulic breccias are described. Some intense altered zone intersections show association with ankeritisation. The “Main Zone” contains also minor amounts of epidote, chlorite, amethyst, micas, magnetite and base metal sulphides. A distinctive deep brick red hematite alteration characterizes the “Main Zone”.

The “Main Zone” trends N110°, dipping at 55° southwest near the surface, steepens to near vertical at the 12th Level, and changes to 60° to 75° at depth. The “Main Zone” alteration envelop increases in width with depth (below the 12th Level), while ore values are not uniformly distributed within the zone, which results in an anastomosing mineralized pattern. The last drilling program also demonstrated the recurrent presence of a weaker and narrower alteration zone of 3 to 5 m in the foot wall of the Main Zone. This “northern branch” is clearly related to the same event but rarely shows economic interest.

The average width of the “Main Zone”, above the 6th Level, was 1.82 m (6’), and increased to an average of 2.44 m (8’) below this level. The 2005 drilling program below the 12th Level has confirmed that the average width of the “Main” zone increased. Based on the 2005 resource database, the “Main” zone has an average horizontal width of 2.8 m (9.2’) (median at 2.1 m (7’)) and reached a maximum horizontal width of 12.8 m (42’).

This alteration system, which constitutes the main mineralized zone, is recognized over 1 150 m (N110°- N290° trend) and was mined over 335 m from the western limit of the Bachelor claims to the western contact of the O’Brien pluton. The new interpretation proved the Main Zone continuity, with the drill hole intersections, to be over 488 m (from section 1 000’ W to 600’ E) horizontally and 900 m vertically (from surface to the elevation 7 000’).

The “B” Zone:

The "B" Zone was recognized on the 11th and the 12th Levels and may also represent a potential for additional resources, but until now very little mining has taken place in this zone. It was previously described as being similar to the "Main Zone" but the last drilling campaign illustrated their differences. Test mining has also indicated that the "B" Zone has competent walls.

Based on the resource database, the "B" Zone has an average horizontal width of 3.1 m (10.3') (median at 2.1 m (7')) and reaches a maximum horizontal width of 10.5 m (34.5'). The "B" zone was previously considered to be generally narrower than the "Main" zone but the 2005 drill program confirmed that this zone has similar width. The "B" zone dips generally steeper than the "Main" Zone, at about 75° to 85° to the south-southwest.

The "B" zone is interpreted to be the result of a younger geological event and formed after the "Main" zone mineralization. It is characterized by a hydraulic glassy to white silica breccia with angular fragments of the altered unit and cut by quartz veins. Its alteration is similar to the "Main" zone and is represented by strong to intense silicification and hematitization and generally by moderate ankeritization. Mineralization is characterized by 2% to 7% pyrite generally associated with the late quartz breccias. The presence of visible gold (VG) is often seen in this alteration zone and especially in the sections east of the T1 fault.

The "A" Zone:

The "A" Zone was discovered by drilling from the 9th Level and has been traced up to the 4th Level. Test mining at the Bachelor Lake Gold Mine, using shrinkage techniques, has shown an unacceptable level of dilution on this zone.

The "**A Zone**" is visually distinct from the "Main" and "B" zones. It is a highly altered and sheared zone which strikes N060-070° and dips 45° to 50° to the southeast and cross-cuts the "Main" and "B" Zones. It has previously been interpreted as a gold-bearing "Zone" as well, but the last U/G drilling campaign demonstrated a poor grade development of this zone when alone. Table IV-1 (Appendix IV) shows all the "A" Zone intercepts greater than 1 g/t Au described in the last drilling campaign. It demonstrates that best values in the "A" Zone are related to its junction with other zones. Significant intersections have been documented while crossing the "Main" or "B" zones, probably due to gold remobilization. The last interpretation showed increases in thickness at these junctions, especially around section 100' E.

The general aspect and trend of the "A" zone could lead to a correlation interpretation between the "A" zone and the Waconichi fault system.

The "C Zone":

The newly interpreted C Zone has been incorporated into the 2005 Bachelor Lake Resource Estimate. This zone has similar characteristics to the "Main" zone and it appears that it can be a branch of the "Main" zone. The "C" zone actually seems to be less continuous than the "Main" Zone. The "C" zone has been documented in the Bachelor mine area between the sections 150' E and 600' E, in the eastern portion of the 2005 interpretation.

The “A” West and “B” West zones:

The “A” West and “B” West zones have been delineated in the West zone area of the Hewfran claims. These zones are interpreted to be the continuity of the “A” and “B” zones identified at the Bachelor mine area.

The “A” West zone lies within the western extension of the “A” shear and the mineralized zone documented at the Bachelor mine. The discovery hole (19501-39, 0.168 oz/t Au over 6 ft) intersected the zone, 487 m (1 600’) west of the last encountered ore grade within the “A” Zone at 13 500 E. The hole was drilled to test the eastern extension of the mineralized shear structure (N080°) identified in the Agar #1 outcrop which had been mechanically stripped, washed and channel sampled during the summer of 1987 (Rougerie, 1989).

As read in Y. Rougerie report from 1989:

> *most of the ore grade intersects occur along two converging subhorizontally plunging ore shoots. Several spectacular intersections were encountered within these ore shoots including 0.295/29.0’ in HU-6-9; 0.276/84.2’ in HU-6-24; and 0.280/45.0’ in HU-6-30 which indicate that the West Zone is laterally continuous for more than 800 ft, and remains untested to the west. However, sub-economic intersections above and below the ore shoots suggest the ore lenses are vertically discontinuous.*

Our last interpretation illustrates these thicknesses in several holes. Lateral continuity of the structure from section to section is obvious but gold mineralization appears sporadic and essentially concentrated in the vicinity of Sections 12 100 E and 12 300E.

The “B” West zone seems to be the extension of the “B “zone documented at the Bachelor mine. The zone dips at about 80 to 85° (almost vertically) and shows only very sporadic grades over a cut-off grade of 3.43 g/t Au (0.10 oz/t Au). This zone is characterized by a strong silica and hematite alteration, and by local brecciation.

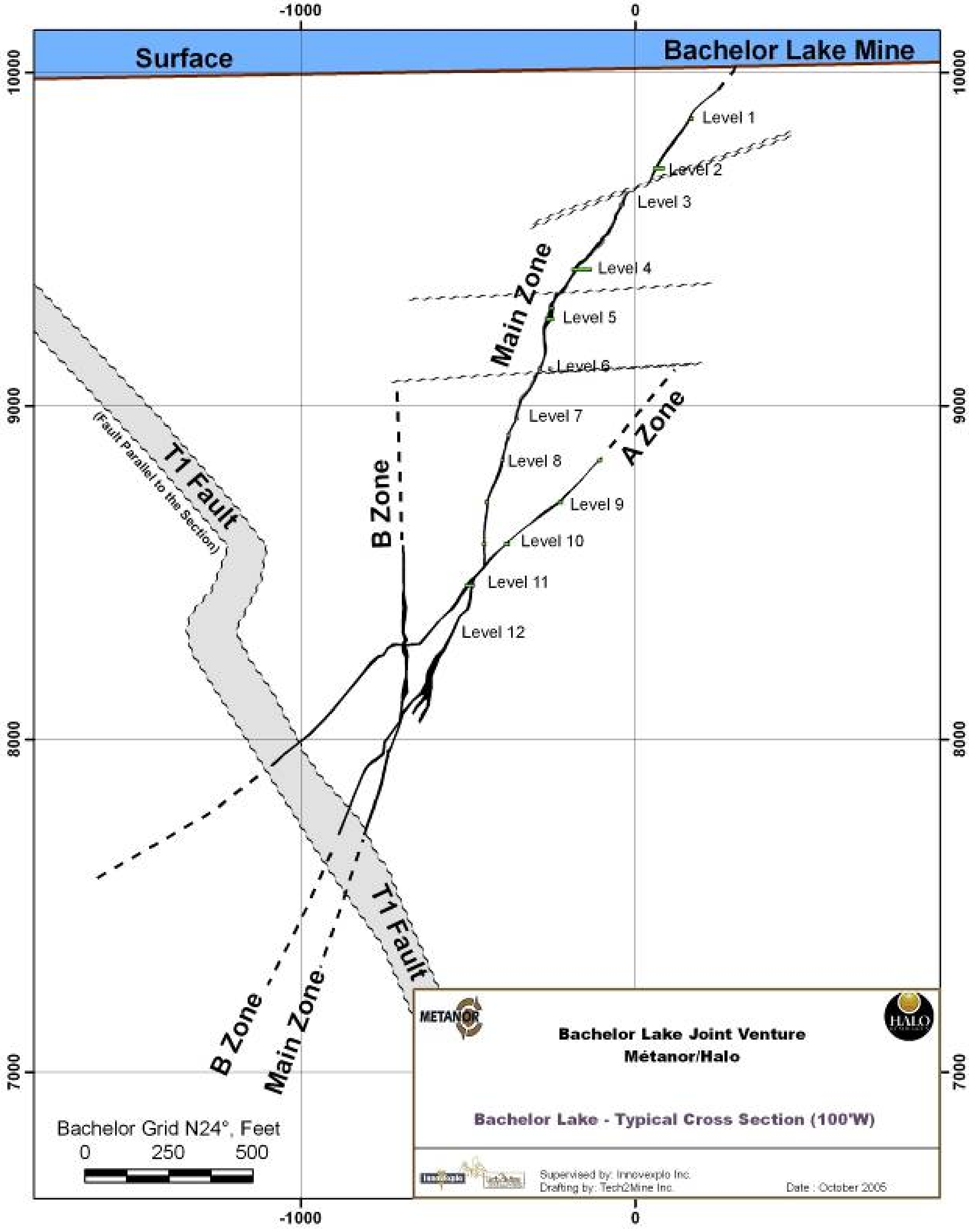


Figure 8.8: Schematic cross-section of the Bachelor Lake gold deposit (100’W)

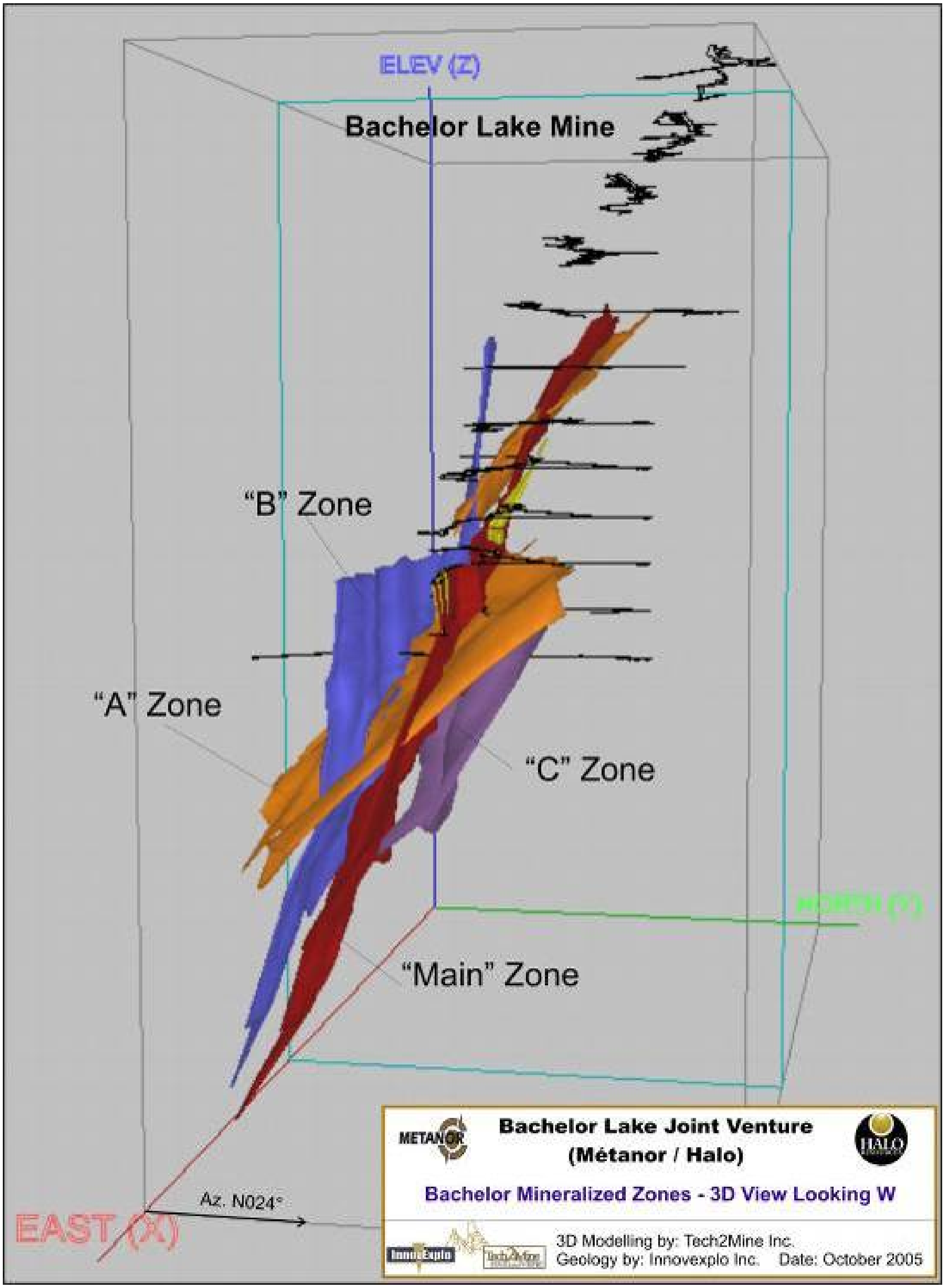


Figure 8.9: 3D view looking west of the "Main", "A", "B", and "C" zones illustrating their relationship.

9. Exploration work (Item 12)

This following information is part of the April 2007 Technical Report on Resources Evaluation of the Barry-1 Property by Geostat.

9.1 Barry I Property

Exploration history of the property is directly linked to the history of the discovery and development of the Barry I property and other mineralized zones previously discussed in this report.

Geophysics

A large part of the Barry I property was surveyed by induced polarization and MAG ground surveys. Murgor still explore the property using geophysics methods. The mineralized zones on the Barry I property responded very well to geophysics.

The next two figures show the compilation maps of the geophysical anomalies detected on the Barry I property.

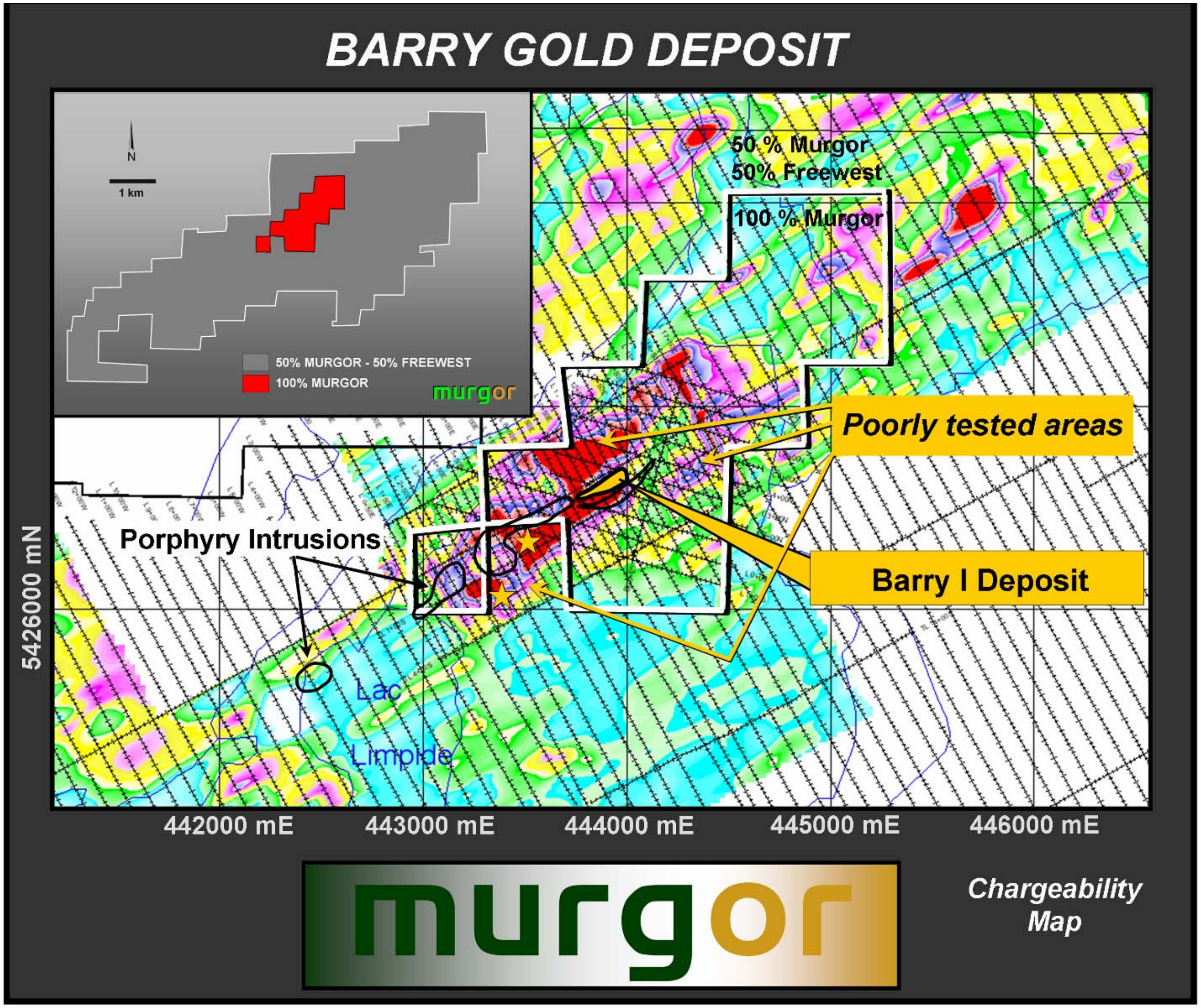


Figure 9.1: Composite map of the chargeability geophysical anomalies on the Barry I property.

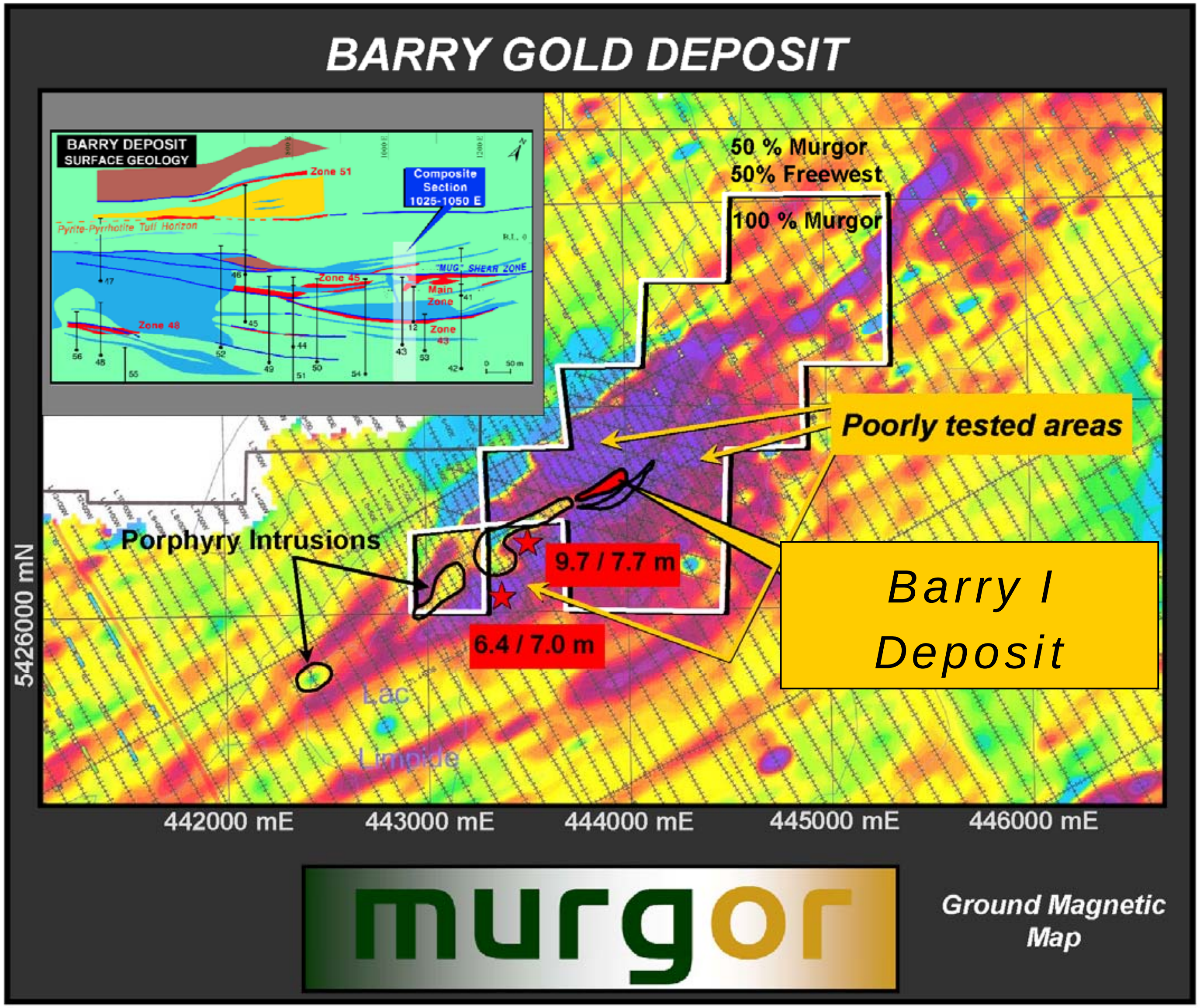


Figure 9.2: Composite map of the MAG geophysical anomalies on the Barry I property.

Survey

Several exploration campaigns have taken place on the Barry I property since its discovery. Three survey campaigns were performed since the beginning of the exploration works. Some of the holes were surveyed in 1996 according to the mine grid. In October 2004, a professional surveyor surveyed the position of most of the collars of the drill holes in the stripped area of the Barry I Main Zone Area according to the UTM Nad83 Zone 18 grid. In regards to the holes drilled in 2005-2006, the position of the remaining collars of the precedent campaign, including the latest holes drilled had been survey by the same professional surveyor as for the 2004 campaign. Some of the remaining drill holes, further away from the Main Zone drilling area, have also been surveyed.

Grids used on the property

One local grid system, in metres, is used on the property. All the surveys and other information are related or transferred to that grid system. The MTM and UTM NAD 83 coordinate systems are also used for survey, exploration and reporting purposes. The drill holes database includes MTM and local grids for the holes surveyed in 2004 and 2006 and only the local grid for the others.

As the relative data from the 2006 survey in UTM coordinates and the values already loaded in the database does not correspond, the coordinates of the holes had been corrected according to the surveyed coordinate instead of the measured one. The common point chosen to link the local grid to the UTM grid is the collar of the hole MB95-05. The coordinates of all the drill holes surveyed according to the UTM grid were transferred to the local grid according to the common point. This modification of the drill holes coordinates changed slightly the position of some drill holes according to the others. The easting, the northing and the elevation of all the drill holes changed slightly. There was no major errors in the tape measured coordinates in regard of the surveyed coordinated. The March 2006 interpretation was done according the surveyed coordinates transferred in the local grid.

The estimation of the resources is relative to local grid. The north of the local grid is oriented N330°. The next figure shows the position of the local grid over the claims position.

Recent verification of the database shows an estimated convertion from the mining coordinates to UTMs of:
Rotation: -28.8512936227465°
Translation:
X: +442886.77 m
Y: +5426103.98 m
Z: -1604.92 m

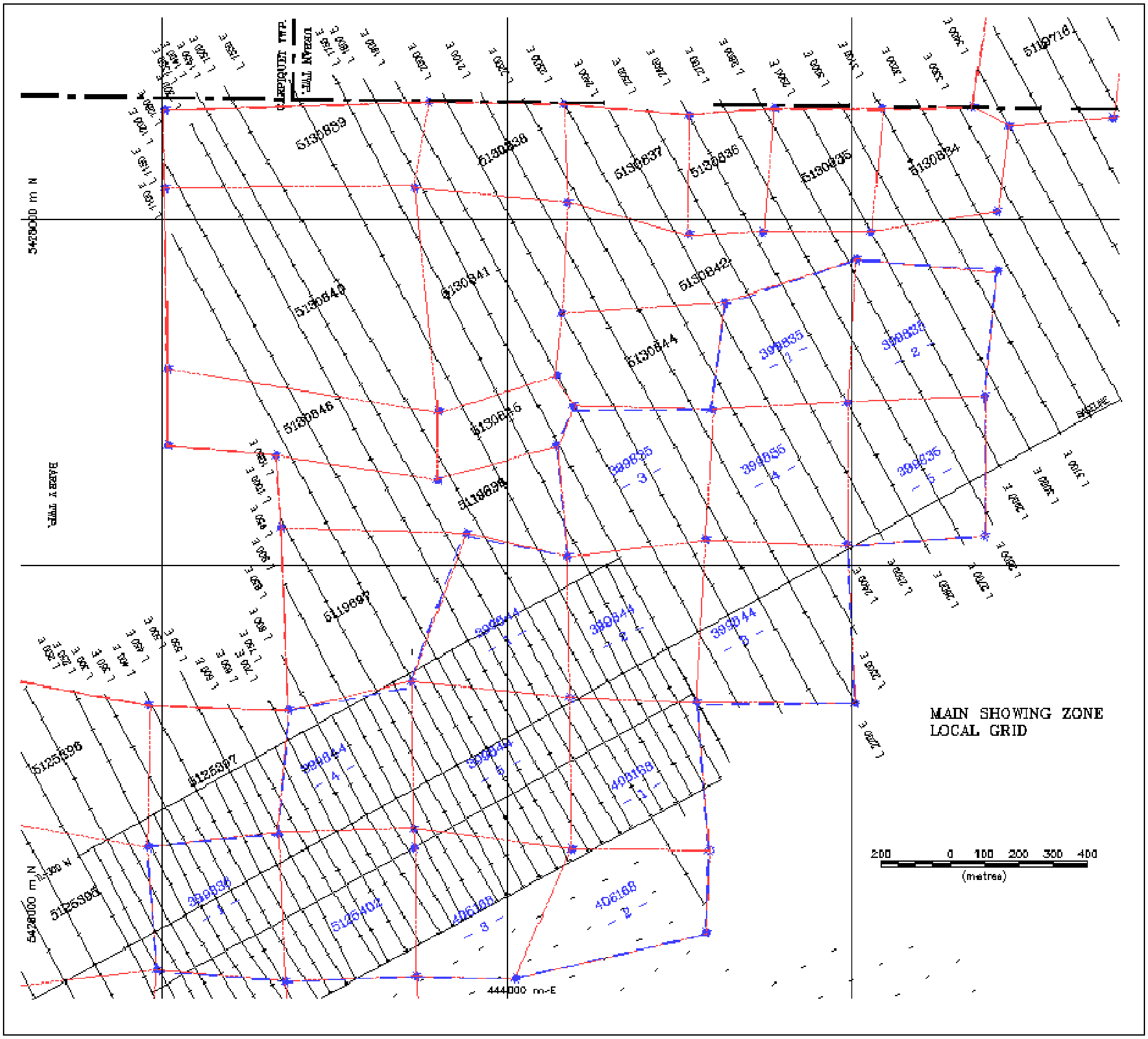


Figure 9.3: Claim map of the Barry I property and Barry I local grid.

9.2 Bachelor Lake Property

Informations extracted from the InnovExplo NI 43-101 report of December 2005

Exploration

The most recent exploration work program was executed from May to August 2005, by Innovexplo for Halo and then for the BLJV. The program consisted in 13 345 m of underground drilling, described in detail in the Drilling section (Item 13).

The issuers (Metanor and Halo, BLJV) have not conducted any other exploration program on the property. Previous exploration programs are discussed in the History section (Item 8).

10. Drilling, mapping and trenches (Item 13)

This following information is part of the April 2007 Technical Report on Resources Evaluation of the Barry-1 Property by Geostat.

10.1 Barry I Property

Metanor possess a voluminous drill holes database for the Barry property. A total of 427 drill holes and for 26,380 metres, have been drilled on the property and the information loaded in the database. The information regarding old five drill holes is not found. All these holes were drilled over several years, from 1962 up to 2006. During 1995, a total of 1,203 metres of channels samples had been collected and sent for gold assay.

An important quantity of core is stored on the project site. Some of the drill holes core intersecting the most interesting mineralized intercepts has been removed from the core boxes over the years by Murgor and various partners.

The Barry I mineralized body has been interpreted using North 330° cross-sections of different spacing from the coordinate 650 E to 1,275 E.

Trenches and mapping of the Barry I project by Murgor

Given the thickness of the overburden and the few outcrops, there are only trenches in the Barry I Main Zone area.

Murgor had stripped the overburden over the Barry I Main Zone Area in 1995. All the stripped areas were mapped and channel sampled. It is important to note that the channel samples are not used in this estimation of the property resources. They could have been used if surveyed in 3D to follow the topography.

The detailed hard copy on the mapping is not available in electronic format. A scan on the simplified copy of the mapping is presented in Appendix.

Recent drilling and evaluation of the Barry project by Murgor

A new drilling campaign was executed. Some 58 drill holes for a total of 5,076 m were drilled on the Main Zone and tested the east, north and south deeper extensions of the Main Zone Area and the Zone 43. A total of 4,988 samples were sent to the lab for gold assay.

This new drilling campaign permitted to better define the extension of the mineralized zone inside the Main Zone Area and to verify the extensions of the Main Zone.

A new interpretation of the mineralized zones and an update of the previously estimated resources were performed. The resource estimate aimed to define mineralization exploitable by open-pit mining. This new design included the mineralized zones from the Main Zone, the zones 43, 45 and the southwest extension of the Main Zone.

The mineralization possibly exploitable by open-pit was not altered by this new drilling.

The next table shows the parameters of the 58 holes drilled in 2006-2007 by Murgor on the Barry I Main Zone area

Hole Name	Easting (m)	Northing (m)	Elevation (m)	Azimuth	Dip	Length (m)
MB06-200	1147	-200	2000	0	-55	206
MB06-201	1124	-174	2002	0	-50	83
MB06-202	1096	-180	2001.94	0	-50	89
MB06-203	1074	-175	2002.72	0	-50	74
MB06-204	1056	-205	2001.71	0	-60	119
MB06-205	1185	-125	2002	0	-65	71
MB06-206	1158	-123	2003.67	0	-70	65
MB06-207	1111	-140	2004.31	0	-65	56
MB06-208	1088	-145	2004.84	0	-60	70
MB06-209	1067	-138	2006.68	0	-55	70
MB06-210	1037	-143	2006.72	0	-65	77
MB06-211	995	-186	2004.76	0	-75	107
MB06-212	1056	-240	2000.45	0	-60	80
MB06-213	800	-187	2008.16	0	-80	108
MB06-214	800	-128	2012.32	0	-75	176
MB06-215	800	-114	2013.28	0	-75	77
MB06-216	800	-98	2014.94	0	-75	50
MB06-217	800	-78	2016.33	0	-80	65
MB06-218	850	-143	2010.79	0	-80	122
MB06-219	850	-119	2012.98	0	-80	116
MB06-220	850	-94	2015.57	0	-80	62
MB06-221	850	-68	2014.58	0	-80	50
MB06-222	900	-145	2010.04	0	-75	181
MB06-223	900	-165	2008.58	0	-80	128
MB06-224	900	-191	2006.52	0	-80	128

MB06-225	900	-42	2009.6	0	-80	47
MB06-226	950	-163	2007.75	0	-84	95
MB06-227	950	-231	2004.92	0	-80	107.3
MB06-228	650	-190	2010.57	0	-80	57
MB06-229	650	-164	2014.19	0	-80	62
MB06-230	650	-137	2014.36	0	-80	68
MB06-231	650	-104	2015.65	0	-80	50
MB06-232	650	-79	2014.55	0	-80	55
MB06-233	700	-195	2009.04	0	-50	95
MB06-234	770	-77	2014.11	0	-80	50
MB06-235	700	-38	2017.49	0	-80	62
MB06-236	750	-188	2008.75	0	-80	71
MB06-237	750	-163	2010.29	0	-80	116
MB06-238	750	-142	2011.25	0	-80	92
MB06-239	750	-93	2013.45	0	-80	71
MB06-240	750	-68	2015.22	0	-80	80
MB06-241	700	50	2017.6	180	-85	65
MB06-242	650	25	2016.64	180	-85	101
MB06-243	1000	0	2007.38	180	-80	62
MB06-244	1000	100	2009.39	180	-80	80
MB06-245	1000	200	2000.78	180	-80	80
MB06-246	1200	100	1998.64	180	-80	80
MB06-247	1200	200	1996.37	180	-80	90
MB06-248	1300	100	2003.01	180	-80	86
MB06-249	1074	-232	2002	0	-60	92
MB06-250	1096	-204	2002	0	-70	80
MB06-251	1037	-233	2002	0	-65	111.28
MB06-252	1016	-231	2005	0	-60	92
MB06-253	995	-205	2005	0	-74	101
MB06-254	900	-212	2007	0	-70	56
MB06-255	750	-120	2012	0	-87	86
MB06-256	1056	-220	2002	0	-65	104
MB06-257	1147	-170	2002	0	-55	101

Table 10.1: Parameters of the 58 holes drilled in 2006-2007 by Murgor.

10.2 Bachelor Lake Property

Most of the informations of this item are extracted from the InnovExplo Technical Report of December 2005 for the Bachelor Lake Property.

DRILLING

On the Hewfran claims, the last drilling campaign was done between 1987 and 1989 by Aur. Their program has included 47 surface holes for 14 255.5 m (46 770') and 96 underground holes for 10 401 m (34 125'). Between 1990 and the 2005 underground drilling program, two drilling programs were completed on the Bachelor claims: one program in 1990 and one in 1995. In 1990, Acadia Mineral Ventures Ltd (subsidiary of Hecla Mining Company of Canada) drilled 34 holes for a total of 4 807 m (15 722') from the underground workings at various locations on the 11th and 12th Levels. In 1995, Espalau Mining Corporation did 10 drill holes from surface for a total of 2 572 m (8 438'). This surface drilling program was executed by Géospex. From 1987 to 1989, the western block of the property (Hewfran claims) was the site of a major drilling program: 47 holes drilled from surface for a total of 14 259 m (46 770'), and 96 holes drilled from underground for a total of 10 404 m (34 125'). These drilling programs are detailed in the History section 6.0.

In 2005, the issuers Metanor and Halo (BLJV) did a major underground drilling program (69 holes for a total of 13 345 m). This program has been initiated by Halo and later followed by Metanor and the BLJV. The 2005 underground drilling campaign is described in detail in this section. It should also be noted that a surface drilling program is presently underway by the BLJV on the property. Illustrations and drill holes summary Table are presented in Appendix V.

Scope of the 2005 underground drilling program

The main goals of the 2005 underground drilling program were: (1) upgrading the resources and (2) increasing the resources. The drilling program was originally designed by Yves A. Buro, geological consultant for Wolfden. At the beginning, Yves A. Buro and Mitch Dumoulin proposed to test the depth extension of the Bachelor Lake gold deposit -300 m (-1 000') under the 12th Level. In the first planning, new underground developments were required in order to have new drilling access, adequate angle and a regular spacing of drill intercepts on the extension at depth of the mineralized zones. Afterwards, the planning changed and no underground developments were accomplished. The program was initiated by Y. Buro for Halo and the complete 2005 drilling program (69 holes) was finally performed from two (2) fixed drill stations located on the 12th Level by performing azimuth drilling. The drill program was initiated by Halo with a clear objective of upgrading the resources by performing 20-25 m (75') drill centers on the "Main" zone and to some extents on the "B" and "A" zones, which are located closer to the two (2) drill stations.

The 2005 drilling program was designed to further define and build tonnage, and to improve our understanding of the geological setting and the continuity of the ore lenses. Despite the fact that this program was performed in restricted area, it successfully:

- filled the central gap between the T1 fault and the "A" zone and also between the two main ore shoots with seventeen (17) holes;

- infilled the gaps left with the previous exploration programs (the 1990 Hecla program was not accomplished) with twenty-four (24) drill holes;
- extended the mineralized zones laterally to the West, on the footwall of the Waconichi fault and at depth with nineteen (19) holes;
- extended and connected the Bachelor resources to the West with the Hewfran claims with six (6) holes;
- extended the mineralized zones to the East side with three (3) holes.

Innovexplo involvement during the 2005 drilling program

Initially, the underground program (dewatering and drilling) was carried out to complete the work requirement of Wolfden (later transferred to Halo) for the acquisition of 50% of the property. During the fall of 2004, the dewatering of the Bachelor Lake mine was initiated by Wolfden in order to facilitate the underground drilling program. The dewatering of the mine was performed by CMAC (formerly Talpa) and was completed during the winter of 2005. From April 6th to July 26th, 2005, sixty-nine (69) holes (BQ size) were drilled by Forage Orbit of Val-d'Or for a total of 13 345.55 m (44 977.36').

At the beginning and for the period from April 6th to May 2nd 2005, the drilling program was completed, planned and logged by Yves A. Buro (for Halo). At the end of April, Halo mandated Innovexplo to continue the drilling program already in progress. A meeting between Y. Buro, A. Carrier and J. Davy (Innovexplo) was organized at BLGM site on April 27th. The local geology, main geological features, sampling protocol and drilling parameters were then transmitted. These parameters could be summarized as follows:

- Core logging, sample intervals, RQD are in meters on Excel spreadsheets;
- Deviation tests are obtained from a Flex-It instrument;
- Planning is done on the Bachelor Lake local grid in feet and oriented 24° east from the geographic North;
- Drill holes were planned using quick logs (brief descriptive follow-up), plotted on plan views and azimuth sections (usually without assay results);
- Holes were stopped generally 6 m (20') after the targeted alteration zones. No exploration holes were then attempted;
- Pictures of the entire core were taken systematically;
- Three (3) standards were used and inserted in each batch of 20 samples;
- Blanks were taken from an "assumed" barren local rock source (homogeneous intermediate volcanic tuff between 108 and 132 m in hole 12-41). Blanks were inserted into regular sample sequence every 20 to 30 samples, preferentially after a mineralized zone;
- No visible gold documented and no special treatment for sample with visible gold;
- Samples were sawed in halves and sent by bus to ALS Chemex laboratory in Val d'Or.

On May 2nd, 2005, more than 25% of the entire program was already drilled (3 500 m with twenty-two (22) holes). Some of these holes were twinning historic drill holes and others were following a 20-25 m (75') centers infill program with the "Main" zone as the principal target. The three (3) mineralized zones ("Main", "A" and "B" zones) were usually intersected in each hole. Nine (9) holes out of twenty-two (22) were already logged by Yves Buro. Thirteen (13) holes were not described (back log) and fifteen (15) holes were not sampled. Logging and sampling of these holes were performed by Innovexplo.

From May 2nd, 2005, to July 26th, 2005, Innovexplo's geologists and qualified people, Julien Davy, P.Geo, M.Sc. and Eddy Canova, P.Geo, B.Sc., were on site, on scheduled rotations of 7 days in and 7 days out (12 hours per day) for planning, interpretation, follow-up and core logging. During this period, Alain Carrier, P.Geo, M.Sc. (Innovexplo) made about ten (10) visits on site to follow up and supervise as a Qualified Person for the program. At Bachelor, sampling, core moving and technical support were performed by Innovexplo's exploration technicians, Michel Lachance and Christian Paquin, also on a rotation schedule of 7 days in and 7 days out (12 hours per day). On site, Innovexplo had the support of Halo's geologist, Patrick McLaughlin, B.Sc., to catch up with the logging. Christine Beausoleil, P.Geo., B.Sc. (Innovexplo) also logged drill holes during P. McLaughlin's vacation. To catch up with the sampling, some of the core was sent to Val-d'Or and sampled by Cindy St-Amand (at Metanor's core shack) and by Marcel Naud (at Innovexplo's core shack). Data management (core logging database, assay tables, data entry and validation for 2005 drill holes) was performed by Julien Davy and Eddy Canova (Innovexplo) in collaboration with Robert Duchesne, Josette Boucher, Louise Charbonneau and Mélanie Benoit (Tech2Mine inc.). During the drilling program, discussions were held periodically with Tom Healy (Halo). Follow-ups and Press Releases were accomplished in collaboration with Tom Healy and Marc Cernovitch (Halo). Denis Blais, Yves Buro (consultants for Halo) and André Tremblay (Metanor) were also involved in the drilling program.

Some logistical aspects of the drilling program were changed. Core logging was done using an "Access format" logging software (Géotic Log). Sampling, security and sample shipping and laboratory protocols were established or changed during May 2005. Deviation tests obtained from the Flex-It instrument and assay results from AlS-Chemex were electronically transferred in the Géotic Log database. By June 23rd, 2005, all the holes in the back log were completed and the geologists were following the production of the two (2) rigs. By the end of the program, in July 2005, all the collar locations were surveyed and all the assay results were received within fifteen (15) days after the end of the program.

From April to July, 2005, a total of 13 345.55 m (44 977.36') was drilled from sixty-nine (69) holes (BQ size). A total of 3 555 samples were taken from these holes. Twenty-five percent (24.8%) of the total drilled length was sampled for an amount of 3 307.63 m. The area covered by the program is illustrated in Figure 11.1.

Drilling was performed by Forage Orbit of Val-d'Or (Fig. V-1, Appendix V), on a basis of two 12-hour shifts per day using two (2) rotating crews per drill to ensure non-stop drilling during the program. Table V-2 (Appendix V) provides a list of all diamond drill holes statistics for the entire program drilled by Forage Orbit. Drill hole locations are shown on the map in the Appendix X.

The program was performed with azimuth holes drilled from two (2) fixed drill stations located on the 12th Level of the Bachelor Lake. The two (2) drill stations, #1 and #2 were located ± 45 m (150') apart. From the total of 13 345.55 m, 6 854.55 m were drilled from drill station #1 and 6 496.00 m from drill station #2. The whole program was drilled from the deepest level of the Bachelor Lake mine, 12th Level (level at 8 328' elevation). The 12th Level is at -516 meters (1 692') below the surface level.

The entire drill campaign was logged (geology and sampling) using Géotic Log core logging software. All sample results were regularly imported to this database which also contains collar location, deviation test, assay results, RQD and recovery information that were measured during logging by the geologists.

The program was conducted while taking into account all the recent MRNFP environmental standards and procedures. All holes (collars) were identified using aluminium ID tags.

It should be noted that neither the recent underground holes nor the historic underground drill holes have been cemented. All the **underground diamond drill holes have to be cemented** before any further underground works are performed.

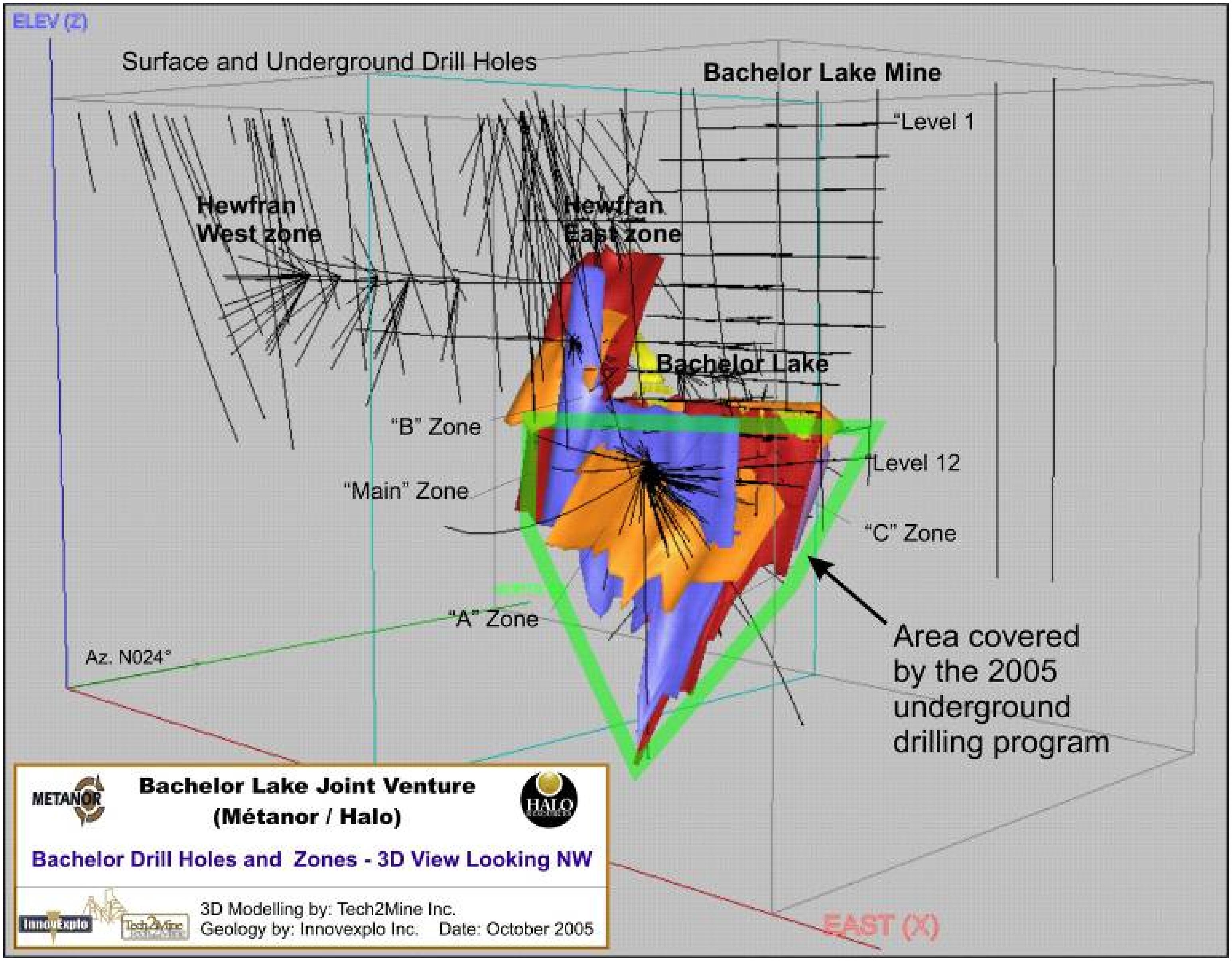


Figure 10.1 - Bachelor 3D view illustrating historic drill holes coverage and the 2005 underground drilling program

Azimuth drilling, target, estimated width and deviation tests

At Bachelor, plan views, azimuth cross-section and longitudinal views were used as follow-up and planning purposes, using final logs when available or more often quick logs results. Regular north-south cross-sections (grid 24°E) were drafted occasionally during the program in order to help with the planning aspect. Final interpretation was realized on north-south cross-sections.

Planning and follow-up of azimuth holes from two (2) fixed underground drill stations are one thing, but reaching a specific target (every 20-25 m (75')) with an azimuth hole could be a real challenge especially when the mineralized zones show changes in strike and dip. With azimuth drilling, thicknesses are apparent and the real picture of the geology is difficult to establish (Fig. 11.2). Core lengths of a zone can be four (4) times its true width. Estimated horizontal widths for the mineralized zones were obtained on the north-south cross-sections.

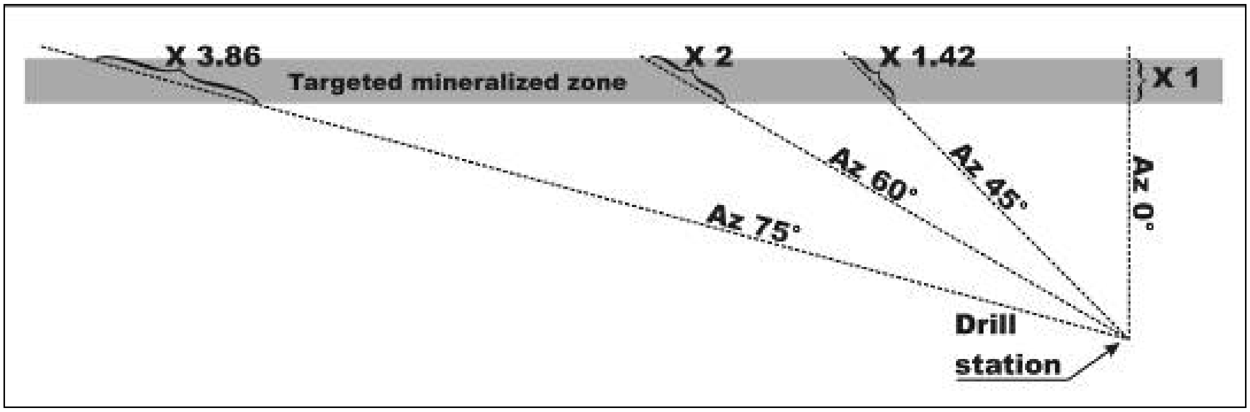


Figure 10.2 - Schematic plan view illustrating the strong influence of azimuth drilling on the apparent width of the mineralized zones.

Slight deviation of 2° has a real impact on the final "x", "y","z" locations of a drill intercept when the hole is drilled with an azimuth in the range 60° to 75°. Furthermore, the mineralized zones at Bachelor Lake are discordant (in plan and in section) to the volcano-sedimentary sequence (stratigraphy is folded and overturned); in this particular case, the deviation of a hole is hard to predict.

Underground at Bachelor, the planned holes were spotted using front sight and back sight aluminium ID tag ("spade") each 10° by a surveyor crew of Jean-Luc Corriveau (using the Bachelor mine grid at 24°E of the true north). Drill hole planning (collar azimuth and plunge) was prepared to fit either these 10° surveyed tags or their mid-distance.

The planning indications were transferred to the drilling team using a single information sheet per drill rig with the hole number, azimuth and plunge at the collar and planned length. Holes were spotted by Forage Orbit's driller under the supervision of François Faucher (foreman). The method was to bring the drill rig parallel to a rope attached from the front to the back sight tags. No central point was used as a rotation point for the drill rig which resulted in the variation of the collar location in the drill stations. Each collar location was later surveyed by Jean-Luc Corriveau's surveyor crew.

Deviation tests for the drilled holes were obtained from down-hole surveys with Flex-It ™ instrument rented from Fordia Canada and used by the drilling contractor. Measurements (azimuth, dip, and magnetism) were taken every 3 m when the hole was completed while pulling out the rods. In some long holes, measurements were taken several times to be able to follow the deviation of the hole during its progression and to ensure that the target would be attained.

At Bachelor, strong magnetism associated with some of the volcanic units may have influenced the electronic multi-shot instrument. A statistical mean for non-magnetic rocks at Bachelor has been determined to calibrate the instrument. Twenty-three (23) multi-shot surveys representing 379 non-magnetic measurements have been used for the mean of 56 279 nt. (nano tesla) with outer limits of ± 1 500 nt. This value was used for all down-hole surveys completed during the 2005 drilling program. Details of calculations are shown in Appendix X. The azimuth and dip surveys for each hole have been compiled, checked and transferred in the database.

Results and highlights of the 2005 drill program

Out of the sixty-nine **(69) holes** drilled during the 2005 drill campaign, forty **(40) holes** have intercepted composite grades over a cut-off of 3.43 g/t Au (0.10 oz/t Au) on a minimum horizontal width of 1.5 m (5') (Table V-1).

Eight (8) holes intercepted a mineralized interval having a horizontal width over 6 m (20'): **6.97 g/t Au over 7.92 m; 13.08 g/t Au over 7.62 m; 7.03 g/t Au over 6.40 m; 9.72 g/t Au over 7.92 m; 12.62 over 6.10 m; 9.88 g/t Au over 7.92 m; 10.35 g/t Au over 8.53 m; and 7.40 g/t Au over 8.23 m**.

Fourteen (14) composite mineralized intervals have a grade higher than 10 g/t Au (0.29 oz/t Au): **14.84 g/t Au over 1.98 m; 14.31 g/t Au over 5.64 m; 17.76 g/t Au over 3.66 m; 16.83 g/t Au over 3.05 m; 14.22 g/t Au over 4.57 m; 16.36 g/t Au over 1.52 m; 16.00 g/t Au over 2.29 m; 10.35 g/t Au over 8.53 m; and 26.47 g/t Au over 1.52 m**.

For the overall program and from the twenty-nine (29) holes that did not intercept a significant grade, nine (9) holes do not seem to have reached the targeted area. From these nine (9) holes: two (2) holes ended in the O'Brien granite (holes 12-43 and 12-47); one (1) hole was dyked out (porphyritic monzonite intercepted at the location of the "Main" zone); one (1) hole was planned to reach the O'Brien granite contact at depth and not the "Main" zone (hole 12-116); two (2) holes were planned using a natural deviation which did not happen (holes 12-94 and 12-96); finally, three (3) holes appeared to be too short according to the final geological interpretation (holes 12-71, 12-77, and 12-91).

The drill program has greatly enhanced the understanding of the geological structure at depth and has led to the generation of significant new drill targets. This knowledge has the potential to significantly increase resource tonnage. Geological review has demonstrated that significant increase in both gold grade and thicknesses appeared particularly at the intersection between major structures.

11. Sampling methodology (Item 14)

This following information is part of the April 2007 Technical Report on Resources Evaluation of the Barry-1 Property by Geostat.

11.1 Barry I Property

We do not have much information on the detailed methodology of sampling used before the exploration work on the property by Murgor in 2005. All we can state is that half core samples were sawed and then sent to an analytical laboratory to assay the gold content.

Assay validation of the results from Osisko 2004 campaign

Osisko Exploration Inc. drilled 61 holes on the property in 2004-2005. Most but not all the drill core of the mineralized zones were sawed and sent to the laboratory. They sent 2,001 core samples to the laboratory for gold assay. Some 55 standards were sent to the laboratory for quality control. A total of 2.7% of all the samples sent to the laboratory were for quality control.

A total of 195 samples from the holes 101 to 108 were assayed by Osisko by the metallic sieve method and Au 50g-FA-AA and Au 50g-FA-GRAV.

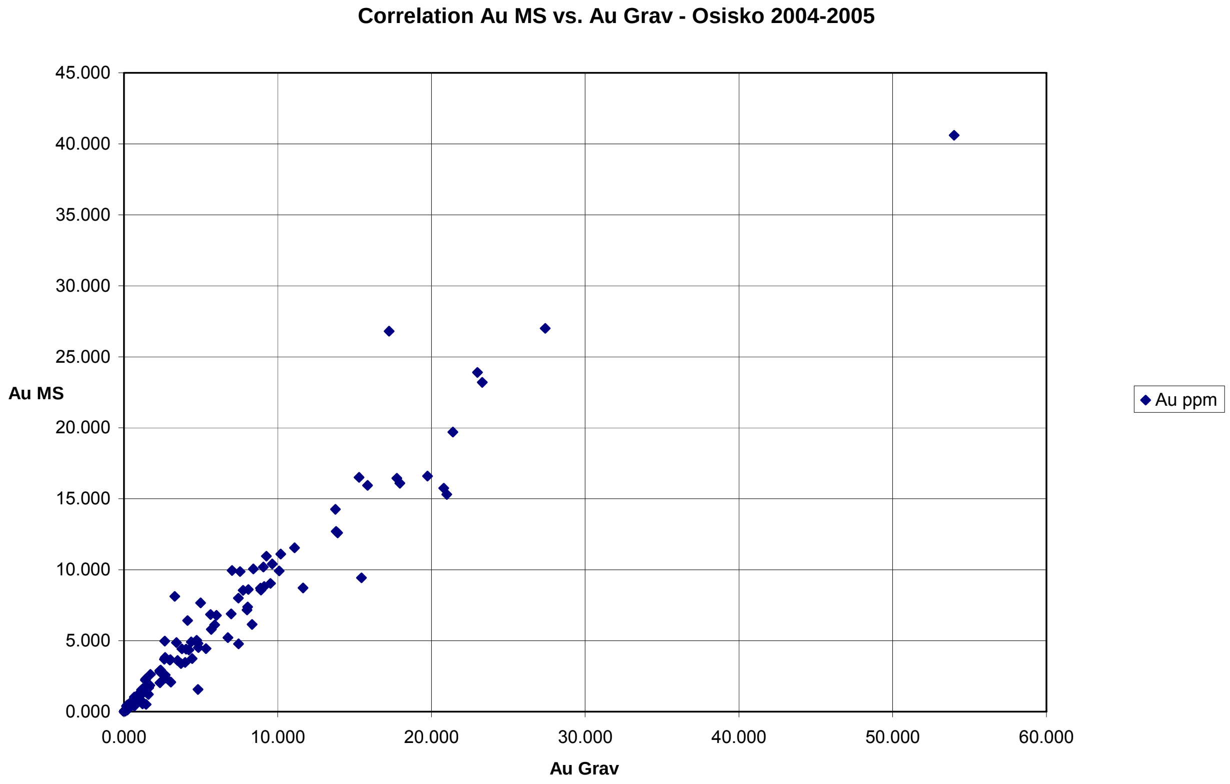


Figure 11.1: Correlation of the assays Au-MS vs. Au-50 g-FA-GRAV for the 195 core samples sent by Osisko in 2004-2005.

The correlation between the results from the metallic sieve and the 50g-FA-GRAV method, for 195 core samples sent by Osisko in 2004-2005, is strong. The value is 0.97.

Murgor initiated a quality control and assurance protocol for its gold exploration programs on the Barry I deposit for the samples of the drilling campaign in 2006, This procedure includes the systematic addition of a certified standard to approximately every holes drilled. Twenty 20 standards samples were sent for gold analysis at commercial certified laboratories.

The next table shows the correlation between the standards values and laboratory results.

Hole Name	Sample Number	Laboratory	Standard	Std. Error	Measured Error	% Error
	Au_g/t	Au_g/t	Au_g/t	Au_g/t	Au_g/t	Au_g/t
174	187635	0.053	0.049	0.001	0.00	8.16%
177	187718	0.051	0.049	0.001	0.00	4.08%
195	186492	0.051	0.049	0.001	0.00	4.08%
170	167899	0.105	0.091	0.003	0.01	15.38%
172	187550	0.093	0.091	0.003	0.00	2.20%
173	187596	0.095	0.091	0.003	0.00	4.40%
176	187697	2.88	2.77	0.02	0.11	3.97%
189	186263	2.78	2.77	0.02	0.01	0.36%
190	186237	2.85	2.77	0.02	0.08	2.89%
193	186422	2.7	2.77	0.02	-0.07	-2.53%
197	186547	2.88	2.77	0.02	0.11	3.97%
198	186569	2.8	2.77	0.02	0.03	1.08%
168	167823	3.55	3.36	0.05	0.19	5.65%
175	187650	3.46	3.36	0.05	0.10	2.98%
178	187757	3.37	3.36	0.05	0.01	0.30%
194	186448	3.4	3.36	0.05	0.04	1.19%
199	186592	3.19	3.36	0.05	-0.17	-5.06%
169	167860	9.72	9.64	0.14	0.08	0.83%
171	167950	9.58	9.64	0.14	-0.06	-0.62%
196	186521	11.3	11.33	0.17	-0.03	-0.26%
Average					0.02	2.65%

Table 11.1: Results of the quality control for the holes drilled in 2006.

If we consider the values greater than 2 g/t Au, the maximum difference between the standard value and the assayed value is 0.19 g/t Au for a standard of 2.77 g/t Au ± 0.05 g/t Au. The results show that the error is less than 6% on the gold value. The average error is positive, with an augmentation of the gold contains of 0.02 g/t Au, for an average error of 2.65%. Geostat considers these values within the average for that type of metal determination.

Samples coming from half cut NQ cores and lengths up to one metre are sent for analysis to ALS Chemex in Val d'Or, a certified laboratory. Samples are assayed by fire-assay followed by atomic absorption or gravimetry according to industry standards. Their methodology is well documented and a quality control is in place. A chemist signs their analysis certificates.

We do not have reason to believe that the methodology used by the different laboratories was not adequate for the results in the Barry I project. Geostat carried out analytical checks of a series of core samples. The results are presented in the data validation section of this report.

11.2 Bachelor Lake Property

Most of the informations of this item are extracted from the InnovExplo Technical Report of December 2005 for the Bachelor Lake Property and have been updated.

Sampling Method and Approach

Sampling method and approach for both historic and new exploration are considered to be appropriate and accurate. A list of composites with estimated horizontal width is presented in Table V-1.

Sampling method before 2005 underground drilling program

Met-Chem completed a review of the sampling protocol employed at the Bachelor Lake Mine during the production period. **Sampling** of the drill holes (BQ and AQ size) is very regular with geology being the first criteria to determine the sample length and that did not exceed 1.5 m (5'). During the Hecla drilling program, Buro (2004, personal communication) noted some gaps in the sampling.

Except for the Met-Chem review, there was no systematic review of the sampling method and approach in the historical assessment works. However, it can be stated that the sampling method and approach used were essentially core samples, chip samples and muck samples from underground development at Bachelor Lake mine and the Hewfran East area. During the Aur drilling program (Hewfran claims), similar sampling protocols were used with the AQ and BQ size core. For Hewfran, the zones were either sawed or split in half for the sampling. Aur has also used chip and muck samples from underground levels. Moreover, Aur did a bulk sample test on the Hewfran claims.

Prior to the NI 43-101 standards, it was generally assumed that the data provided was accurate and reproducible.

Controls (rock types), sampling intervals, zone width and treatment of higher grade area have been described in a Technical report prepared by Carrier (2004b):

> *[… Mineralized portion of drilled holes at Bachelor Lake are characterized by a red alteration (silica-hematite) and by higher pyrite concentration which facilitated their recognition and sampling. Weighted average and horizontal width calculation were obtained from these zones. High grade samples (over 0.6 oz/t Au) are usually encompassed within a series of significant assay results (over 0.3 oz/t Au), see for example assay results from hole 12-04 for the Main Vein interval (710.7' to 729.5'). Also in hole 12-04, results between 0.1 oz/t Au and 0.3 oz/t Au located before (710.0' to 710.7') and after (729.5' to 739.0') the Main Vein interval where not included within the weighted average and horizontal width calculation of the Main Vein. Hole 12-09 is also an example of marginal mineralized fringes not included within the calculate interval.*

> *Other high grade sample results are usually encompassed within a continuous series of mineralized samples, see for example for the Main Vein: holes 11-14, 12-02, 12-31 and 12-33; and for the B Vein: holes 12-11, 12-15 and 12-23.*
>
> *Weighted average and horizontal width calculation from mineralized intervals may also included high grade assay (over 1 oz/t Au) and marginal result (0.01 oz/t Au) in the same interval. For example in hole 11-10, the Main Vein interval (0.253 oz/t Au over 17.0') included one (1) high grade assay (over 1 oz/t Au) and two (2) marginal assay result of 0.01 and 0.05 oz/t Au. Although, this interval is also supported by four (4) other samples with assay results between 0.25 and 0.50 oz/t Au. …]*

Met-Chem (2001) stated the processing of samples with visible gold and high grade sample:

> *[… It appears that the mine operation did not feel much concerned toward sampling and check assays due, in part, to the absence of visible gold. This is repeatedly mentioned in most reports.*
>
> *Visible gold is considered to be an indicator of the presence of nuggets (coarse particles of gold) associated with disturbance of grade determination in gold mines. It is labelled "contamination" and withdrawn from sample bags, usually. There is wide spread misconceptions about this natural phenomena most commonly associated with gold. The staff at the mine may have comforted themselves with the apparent absence of VG, meaning no coarse gold is observed. What may have escaped their attention is that the sudden change of grade observed in the drill logs which continue to manifest itself on the polygonal section of the resources, amounts to the same results. In other words, this strong variation of grades from, say, 0.1 to 0.6 oz/t Au is equivalent to an erratic behaviour of gold distribution, even if it appears relatively smooth. Gold grade continually shifts in audited data. The potential negative impact of a variable thickness is comparable, although it is usually less significant.*
>
> *To complicate matters, the cut-off grade (1.0 and 0.65 oz/t) used by the mine to 'cap' the high grade samples for respectively the "Main Zone" and the "B Zone" is reportedly ineffective. This comes from the fact that assays have very few, if any values above this cut-off (no VG), as it can be seen in drill hole S-95-08 results (table below). Again, this situation arise from a peculiar characteristic of the ore at this site which makes it escape the usual visual (VG) and statistical (1.0 oz/t cut) quality controls. The excessively high grade samples are noticeable, but left untouched. If they were clustered together, they would cause minimal disturbance, but they occur somewhat randomly, surrounded by steeply lower grades, which makes them less representative of their environment. They must be weighed down fairly in the calculation of the resource average grade. …].*

Table 11.2: Details of the Main Zone in drill hole S-95-08

From (ft)	To (ft)	Interval (ft)	Gold (oz/t)	Silver (oz/t) 196
1904.53	1909.45	4.92	0.021	
1909.45	1914.7	5.25	0.088	0.076
1914.7	1919.62	4.92	0.084	0.075
1919.62	1924.21	4.59	0.631	0.574
1924.21	1929.79	5.58	0.238	0.226
1929.79	1933.4	3.61	0.064	0.073
1933.4	1938.98	5.58	0.042	0.025
1938.98	1943.9	4.92	0.256	0.317

1943.9	1948.82	4.92	0.181	0.151
1948.82	1953.74	4.92	0.019	0.016

2005 Drilling sampling method

From April 6th to July 26th, 2005, sixty-nine (69) BQ size (36.5 mm diameter) drill holes were performed by Forage Orbit Inc., for a total of 13 345.55 m (44 977.36') using the industry standard wire line methods. All of them were drilled from two (2) underground drill stations at the 12th Level. Fifty-two (52) reached the mineralized zones in the Bachelor claims, while seventeen (17) reached the mineralized zones in the Hewfran claims. Holes were planned using the "Main" Zone longitudinal section with intercepts every 22.8 m (75'). The 2005 drill hole database contains a total of 3 555 samples. One hundred percent (100%) of the 2005 drilling program was stored and categorized for future reference purposes in the core library located at the Bachelor Lake site (Desmaraisville, Quebec) (Figure VI-2).

Due to limited access, the 2005 drilling was performed with azimuth holes from two drill stations. For azimuth holes, the difference between the core length and true thickness could be considerable (core length can be 10 times the true thickness for drilled holes at 85° azimuth). All thicknesses are horizontal width and were calculated on sections.

For the 2005 drilling program, the core sampling protocol was established by Innovexplo and is described in Appendix VI.

Core sample quality and representativeness:

During the 2005 drilling, 3 251 samples were submitted for gold analysis, representing 3 347.63 m (24.4% of total drilled length). Inserted throughout these samples, 304 blanks and standards (8.55%) were also shipped for a controlled follow-up for a total of 3 555 samples.

Every altered zone (especially hematization and silicification) containing pyrite or every wide altered zone was considered potentially mineralized and therefore sampled. This systematic exploration sampling allowed to confirm the attitude of mineralization within the altered zones as well as other lateral small mineralized zones.

At Bachelor, samples collected through the diamond drilling are of good quality (the mineralization in the core is generally intact with no possibility of loss due to wash out). The hardness nature of the mineralized zones (hematization and silicification) explains the excellent recovery for the mineralized zones.

The core was rarely ground on short distances (less than 0.5 m). Overall, the drill core sample recovery from the mineralized zones can be considered to be representative.

12. Sample preparation, analysis and security (Item 15)

This following information is part of the April 2007 Technical Report on Resources Evaluation of the Barry-1 Property by Geostat.

12.1 Barry I Property

As mentioned in the previous section, the method of preparation and analysis of the samples is not available for the core samples assayed before the work done in 2005. However, the assay certificates from the samples sent to the laboratory in 1995, 1997, in 2005 by Murgor and Osisko and in 2006 by Murgor are available and a complete verification of the values higher than 1 g/t Au shows an excellent correspondence between these certificates and the values in the database. Only 158 of the 206 holes drilled in the Barry I project area are intersecting the Main mineralized zone.

Percentage of assay certificates available for the holes drilled in the Barry project

Company	Year	Holes drilled	Results with assay certificates
Fab Metal Mines	1962-65	5 holes	0
SDBJ	1981-83	3 holes	0
Mines Camchib	1983	1 hole	0
Cominco-Agnico Eagle	1988-89	9 holes	0
Murgor Resources	1995-96	74 holes	5,537
Murgor Resources	1995	167 channels	3,406
Teck Exploration	1997	15 holes	2,105
Osisko	2004-05	61 holes	1,951
Murgor Resources	2005	6 holes	143
Murgor Resources	2006	32 holes	1,278

Table 12.1: History of the drilling and sampling on the Barry I Zone.

There are 188 holes out of the 206 holes (91%) with assay certificates to support the values found in the Barry I project database.

The core was taken from the drill rig and carried to the core shack by the employees of Murgor. Afterwards, core boxes are opened for drill hole logging and identification of the intersection to be sampled by Murgor geologist and consultants. Core is described directly in the drill holes description software. Sections of the core to be analyzed are marked with a marker. Then, Murgor technicians prepare the sample books, sample bags and tags accordingly. After cutting the core in half for one core box, the samples are then inserted into the sample bags. The bags are sealed and put into a large bag for transportation to the laboratory. The splitting of the core is done with a rock saw. Murgor is well organized for core description and sample preparation.

Their own staff has handled all the 2006 samples taken by Murgor. Logging of the core drilled in 2006 by Murgor has been done by Mr. Robert Gagnon, Bsc. Geo., Project geologist, Consultant for Murgor, Mr. Jean-Philippe Desrochers, Ph. D., P. Geo., V.P. Exploration and Mr. André C. Tessier, P. Geo. (On), P. Eng. (QC), President and CEO of Murgor Resources Inc.

In Geostat opinion, the sample preparation, security and analytical procedures are adequate and are done according to the industry standards. Geostat has no reason to believe that samples or analytical results are tempered with at any point during the sampling, test work and assaying procedures for the holes drilled in the Main Zone. The values of the other drill holes were not validated.

12.2 Bachelor Lake Property

Most of the informations of this item are extracted from the InnovExplo Technical Report of December 2005 for the Bachelor Lake Property.

SAMPLE PREPARATION, ANALYSES AND SECURITY

Sampling, preparation, security and analytical procedures used on the property were judged to be adequate. Results from the pre-2005 sampling and assaying are considered to be good. The performance of the laboratory during the 2005 drilling program was good.

Sample preparation and analyses before 2005

For sample preparation and analyses before 2005, Horvath and Carrier (2005) stated:

> *[… No details were provided with regards to the pre-2005 drill hole sampling and assaying protocols; although more recent historic conventions in the gold mining/exploration industry generally followed similar protocols excepting sample splits after initial crushing were generally smaller (i.e. only 250 g instead of 1 kg) and fusion was most often completed on a 30 g split of the sample pulp (sometimes 15 g) instead of the 50 g split used in the 2005 protocols. Historically, samples identified with visible gold were also treated by the metallic screen method as in the 2005 program although similarly smaller sized sample splits may have been used at the time. …]*

During the mine operation, assays were performed at the local laboratory of the Bachelor Lake mill. This local laboratory did not have any accreditation and the method used to determine the primary gold assays was by atomic absorption (AA) and not by fire assay (FA). Check assays were occasionally made by fire assay in an independent and accredited laboratory (Bourlamaque Laboratory in Val-d'Or, Québec). There was clearly a positive correlation between the AA assay values and the FA check assay values (Buro 2004, personal communication). Furthermore, a total of forty-six (46) samples selected from holes 12-37, 12-27, 12-1, 12-2 and 12-4 showed a direct correlation between original and check assay values from the Hecla drilling program. Met-Chem cited that no problems were reported for the in situ analyses.

Sampling protocol employed at the mine during the production period was reviewed by Met-Chem (2001). For adequate referral, the author has reproduced its content below:

> *[… Of all the reports consulted, none mentioned assaying as a potential source of the grade discrepancies. Met-Chem has no reason to doubt the quality of the assaying, but no details were provided concerning the methodology used.*
>
> *In the mine, it was reported that muck and chip sampling was taken in development and presumably in the faces of shrinkage stopes. Our scope of work does not include this area, even though reconciliation of mine grade is a good indicator of the quality of the methods used in sampling and assaying. Indeed, the Bachelor Lake Gold Mine is notorious for its grade problems, mostly blamed on excessive mine dilution. Dilution and grade control is part of sampling and assaying. …]*

Sample preparation and analyses during 2005 underground drilling

Sampling and laboratory protocol for the 2005 drilling program were defined by Innovexplo. During the program, core samples were sent to ALS Chemex Chimitec in Val-d'Or, certified ISO 9001:2000. At the laboratory, all the bags were opened and conformed to the laboratory protocols. Horvath and Carrier (2005) have summarized these steps as follows:

> *[… Sample batches received by Chimitec Laboratories were processed by completing initial inventory, drying and primary crushing (jaw crushers) of the as received samples to 90% passing 10 mesh (i.e. 2mm). Samples were then riffle split (Jones riffle splitters) to reduce sample size for pulverisation to a maximum of 1 kg. Samples were pulverised (ring & puck) to 90% passing 200 mesh (i.e. 75 µm). For a limited number of samples at the start of the 2005 program, fusion of each sample utilized a 30 g split from the pulp; however, this protocol was modified and the majority of samples were assayed using a 50 g split from the pulp for fusion.*
>
> *Analytical protocols required all samples to be finished using acid digestion-AAS finish. All results with initial AAS results reporting greater than 5 g/t Au were re-assayed from the same pulp using gravimetric finish.*
>
> *For specific samples, most commonly those identified with visible gold, the 2005 sampling and assaying protocols required that the split BQ core samples were processed using the metallic screen method. This method required the laboratory to prepare an initial 1 kg split of the coarse crushed sample and pulverise the entire split to 90% passing 150 mesh. The entire pulverised 1 kg samples are screened at 150 mesh and the small (<50g) +Fraction is fire assayed to extinction. The remainder of the sample (approximately 950 g) is the –150 mesh Fraction. Duplicate 50 g fire assays with gravimetric finish are completed on the –150 mesh Fraction of the pulp. The final gold grade is calculated by a weighted average of the +Fraction result and the average of the two –Fraction results.*
>
> *In addition, to the regular sampling and assaying of samples, additional quality control protocols initiated externally by InnovExplo (Halo/Metanor Resources) and those internal to Chimitec's quality control protocols required the preparation of various duplicate samples to evaluate the precision (i.e. reproducibility) and accuracy (i.e. correctness) of the values reported. …]*

Sampling and laboratory protocol are illustrated. (in Appendix VI).

For the 2005 program, the number of samples, standards and blanks per hole are presented in the Table VI-2. The laboratory delivered results in electronic format through the ALS Chemex webtrieve Internet access as well as an e-mail sent to the data manager. Assay results were reported in grams per tonne (g/t) and transferred directly in the central assay data base (GeoticLog and Gems).

Quality Assurance / Quality Control (QA/QC) program

Results from the QA/QC program are detailed in Appendix VI.

No contamination was discovered during the 2005 drill program. The good performance of the laboratory for external standards (field standard) is an evidence of accurate determinations being made by the laboratory.

The QA/QC analysis of the pulp duplicate demonstrates a reasonable level of precision with overall approximate errors of 12%. This level of error is not uncommon for Archean gold deposits where the principal component of the ore if often "freely" liberated gold. In fact, many coarse "nuggety" gold deposits demonstrate much poorer levels of precision in pulp duplicate sample results (Horvath and Carrier, 2005). Precision of metallic screen assay (150 mesh pulp duplicate) was analyzed. The metallic sieve method incorporates duplicate fire assay determinations of the –150 mesh fraction of the screened pulp. The results demonstrate that precision levels of the screened pulp duplicate assays are overall approximate 6.5%. A 5% residual "nugget" effect at 150 mesh is quite acceptable for this type of gold mineralization (Horvath and Carrier, 2005).

The result for the coarse duplicate was not that good. The extremely large introduction of error between coarse and pulp duplicates is clearly indicative of unrepresentative 1 kg coarse crush sample splits. The cause may be inappropriate crush/splitting specifications or related to original field sample size, while this type of error may not result in any global change in resource estimation (Horvath and Carrier, 2005).

13. Data validation (Item 16)

This following information is part of the April 2007 Technical Report on Resources Evaluation of the Barry-1 Property by Geostat.

13.1 Barry I Property

Within the framework of our visit to the site, Geostat carried out an independent sampling program and an analytical check of the samples for the holes drilled at the end of 2005.

The objective of this validation was to confirm the presence of the high gold values, especially on the few sections responsible for the majority of the mineralization found in the Barry I Main Zone Area project. Geostat had three holes drilled to verify the model interpretation of the deposit. One of the holes was drilled 30 metres further west of the desired coordinates, due to error location in the field. These holes were sampled and the core transported directly to the laboratory by Geostat's staff. We verified and sampled again some other holes. These holes are MB_29, MB_31, MB_40, 108, 103, 134, 139 and 140. We selected a set of nine mineralized intersections corresponding to samples already analyzed in the past to verify the gold contain. Previous partners of Murgor already retrieved some of the selected mineralized zones from the core boxes. The core of the hole MB_26, from 8.1 to 11.11, hole 103 from 5 to 12.8, hole 108 from 16 to 24.8 and hole 140 from 4.5 to 29 was missing in the core boxes. Geostat selected all the samples and supervised their extraction from the core boxes. For the samples of which remained a half-core of sufficient size, a quarter of the core was taken. Geostat photographed in detail the core boxes before the assay sample removal.

Core sampling

Figure 13.1: Core logging facilities on the Barry I property.

Figure 7: Ghislain Deschênes, a former Geostat's geologist, checking 2005 drill core before sampling of the freshly drilled core.

Figure 13.48: Description and re sampling of older drill holes core from the Barry I property.

Figure 13.6: View of a core box with sampling tag before cutting the core half in 2 quarters.

The core samples were first sent to the ALS Chemex laboratory in Val d'Or for preparation, gold and for metallic sieve analysis. At our request, the ALS Chemex Laboratory also sent pulps of each sample to the Bourlamaque laboratory in Val-d'Or for gold check analysis. The rejects were kept by the laboratory for validation purpose.

A total of 41 control samples and 102 core samples from freshly drill holes were assayed for the Barry I projects.

The next table presents the assay results on the samples taken by Geostat.

			Murgor		Murgor		ALS Chemex	ALS Chemex	ALS Chemex	Lab. Bourlamaque
Hole Name	From	To	Sample No	Length	Au g/t	No Geostat	Au g/t GRAV 30g	Re-assays	Au g/t MS Total 50 m	Au g/t GRAV 30g
103	1.50	2.10	25169	0.60	9.27	263998	9.960			8.47
103	2.10	2.90	25170	0.80	6.01	263999	4.430			3.97
103	2.90	4.00	25171	1.10	9.07	264000	15.200			13.37
134	3.50	4.30	30126	0.80	8.99	263991	9.470			12.77
134	6.36	7.13	30130	0.77	18.85	263992	16.900			15.93
134	7.50	8.00	30133	0.50	11.85	263993	10.900			14.20
134	9.81	10.30	30137	0.49	7.75	263994	11.800			10.83
134	10.30	10.84	30138	0.54	4.27	263995	3.570			3.77
134	10.84	11.34	30139	0.50	16.60	263996	7.910	12.300		11.93
134	11.34	11.97	30140	0.63	8.87	263997	11.150			10.90
139	8.50	9.00	41936	0.50	54.70	263990	68.3			62.00
162	1.76	3		1.24		53502	3.140			4.73
162	3	4		1		53503	1.035			0.73
162	4	5		1		53504	5.190			4.77
162	5	6		1		53505	5.970			4.10
162	6	7		1		53506	8.530			8.07
162	7	8		1		53507	5.870			7.03
162	8	9		1		53508	4.560			3.77
162	9	10		1		53509	4.370			4.87
162	10	11		1		53510	5.840			5.17
162	11	12		1		53511	8.330			7.63
162	12	13		1		53512	8.770			10.77
162	13	14		1		53513	18.100			17.73
162	14	15		1		53514	0.702			0.60
162	15	16		1		53515	2.160			1.97
162	16	17		1		53516	0.848			0.80
162	17	18		1		53517	0.706			0.90
162	18	19		1		53518	0.615			0.63
162	19	20		1		53519	0.347			0.43
162	20	21		1		53520	1.275			1.10

162	21	22		1		53521	0.059			0.10
162	22	23		1		53522	0.164			0.10
162	23	24		1		53523	0.460			0.40
162	24	25		1		53524	0.085			0.10
162	25	26		1		53525	0.003			0.05
162	26	27		1		53526	0.116			0.10
162	27	28		1		53527	0.154			0.17
162	28	29		1		53528	0.003			0.05
162	29	30		1		53529	0.003			0.05
162	30	31		1		53530	0.013			0.05
162	31	32		1		53531	0.149			0.17
162	32	33		1		53532	0.347			0.33
162	33	34		1		53533	0.010			0.05
162	34	35		1		53534	0.573			0.77
163	0.9	2		1.1		53569	0.012		-0.05	0.05
163	2	3		1		53570	0.01		-0.05	0.05
163	3	4		1		53571	0.056		0.09	0.05
163	4	5		1		53572	0.087		0.07	0.05
163	5	6		1		53573	0.115		0.16	0.10
163	6	7		1		53574	1.75		1.81	1.43
163	7	8		1		53575	0.014		-0.05	0.05
163	8	9		1		53576	0.03		-0.05	0.05
163	9	10		1		53577	0.349		0.42	0.30
163	10	11		1		53578	0.007		-0.05	0.05
163	11	12		1		53579	0.07		0.06	0.05
163	12	13		1		53580	0.05		0.05	0.05
163	13	14		1		53581	0.156		0.19	0.10
163	14	15		1		53582	0.16		0.17	0.17
163	15	16		1		53583	1.36		1.84	1.47
163	16	17		1		53584	0.106		0.1	0.05
163	17	18		1		53585	0.058		0.08	0.05
163	18	19		1		53586	0.03		-0.05	0.05
163	19	20		1		53587	0.188		0.12	0.27
163	20	21		1		53588	0.762		0.29	0.53

163	21	22		1		53589	1.465		1.17	1.37
163	22	23		1		53590	1.33		1.17	1.67
163	23	24		1		53591	0.268		0.32	0.43
163	24	25		1		53592	0.066		0.07	0.05
163	25	26		1		53593	0.347		0.4	0.43
163	26	27		1		53594	0.02		-0.05	0.05
163	27	28		1		53595	0.007		-0.05	0.05
163	28	29		1		53596	0.044		-0.05	0.05
163	29	30		1		53597	0.016		-0.05	0.05
163	30	31		1		53598	0.009		-0.05	0.05
163	31	32		1		53599	0.34		0.2	0.37
163	32	33		1		53600	0.393		0.48	0.50
163	33	34		1		53601	0.02		-0.05	0.05
163	34	35		1		53602	0.605		0.26	0.87
164	0.65	2		1.35		53535	3.64		2.68	2.37
164	2	3		1		53536	2.23		1.74	2.17
164	3	4		1		53537	1.295		0.79	0.93
164	4	5		1		53538	0.394		0.22	0.53
164	5	6		1		53539	0.028		-0.05	0.05
164	6	7		1		53540	3.21		2.25	2.47
164	7	8		1		53541	1.635		1.06	1.03
164	8	9		1		53542	0.24		0.23	0.27
164	9	10		1		53543	0.727		0.54	0.80
164	10	11		1		53544	1.405		1.52	1.33
164	11	12		1		53545	0.089		0.07	0.05
164	12	13		1		53546	0.793		0.54	0.70
164	13	14		1		53547	0.639		0.77	1.07
164	14	15		1		53548	0.208		0.18	0.20
164	15	16		1		53549	0.005		-0.05	0.05
164	16	17		1		53550	0.016		-0.05	0.05
164	17	18		1		53551	0.021		-0.05	0.10
164	18	19		0.96		53552	0.007		-0.05	0.05
164	19	20		1.04		53553	0.005		-0.05	0.05
164	20	21		1		53554	15.1		23.5	10.30

164	21	21.9		0.86		53555	0.013		-0.05	0.05
164	21.9	23		1.14		53556	0.124		0.12	0.17
164	23	23.8		0.75		53557	0.129		0.09	0.13
164	23.8	25		1.25		53558	0.006		-0.05	0.05
164	25	26		1		53559	0.01		-0.05	0.05
164	26	27		1		53560	0.003		-0.05	0.05
164	27	28		1		53561	0.007		-0.05	0.05
164	28	29		1		53562	0.005		-0.05	0.05
164	29	30		1		53563	0.008		-0.05	0.05
164	30	31		1		53564	0.007		-0.05	0.05
164	31	32		1		53565	0.005		-0.05	0.05
164	32	33		1		53566	0.003		-0.05	0.05
164	33	34		1		53567	0.003		-0.05	0.05
164	34	35		1		53568	0.003		-0.05	0.05
MB_29	5.10	5.60	585717	0.50	10.46	263967	0.213			0.30
MB_29	6.60	7.10	585720	0.50	16.33	263968	1.275			1.80
MB_29	7.10	7.60	585721	0.50	5.23	263969	1.390			1.53
MB_29	7.60	8.10	585722	0.50	3.09	263970	2.830			2.73
MB_29	12.60	13.10	585732	0.50	10.88	263971	0.908			1.03
MB_29	13.10	13.60	585733	0.50	7.72	263972	14.750			13.73
MB_31	1.30	1.65	585761	0.35	5.57	263973	37.800			34.17
MB_31	1.65	2.15	585762	0.50	17.52	263974	19.150			17.77
MB_31	3.40	4.00	585766	0.60	9.55	263975	8.810			7.97
MB_31	4.00	4.40	585767	0.40	10.93	263976	14.600			14.43
MB_31	4.40	5.00	585768	0.60	11.30	263977	12.600			23.00
MB_31	5.00	5.50	585769	0.50	13.35	263978	8.300			9.00
MB_31	5.50	5.90	585770	0.40	6.50	263979	7.750			6.20
MB_31	6.30	7.00	585773	0.70	6.39	263980	8.380			9.23
MB_31	7.00	7.50	585774	0.50	6.86	263981	2.250			2.40
MB_31	7.50	8.00	585775	0.50	8.50	263982	9.820			10.63
MB_31	8.00	8.50	585776	0.50	5.28	263983	5.960			6.13
MB_31	8.50	8.90	585777	0.40	2.27	263984	3.030			3.33
MB_31	8.90	9.60	585778	0.70	8.92	263985	13.350			13.77
MB_31	11.30	12.00	585781	0.70	33.83	263986	4.800			4.70

MB_31	12.65	13.15	585784	0.50	15.79	263987	15.750			14.60
MB_31	13.15	13.55	585785	0.40	13.92	263988	20.000			21.43
MB_31	13.55	14.00	585786	0.45	12.12	263989	0.392			0.40
MB_40	10.70	11.50	586245	0.80	5.68	263960	0.049			0.05
MB_40	11.50	12.00	586246	0.50	0.03	263961	0.010			0.05
MB_40	14.00	14.50	586251	0.50	1.29	263962	1.020			1.07
MB_40	15.00	15.50	586253	0.50	4.54	263963	4.430			4.20
MB_40	15.50	16.00	586254	0.50	3.31	263964	2.970			2.43
MB_40	16.00	16.50	586255	0.50	1.65	263965	22.500			21.13
MB_40	16.50	17.30	586256	0.80	8.92	263966	8.770			9.80

Table 13.1: Assay results of the core sampling by Geostat from the 2005 drilling campaign.

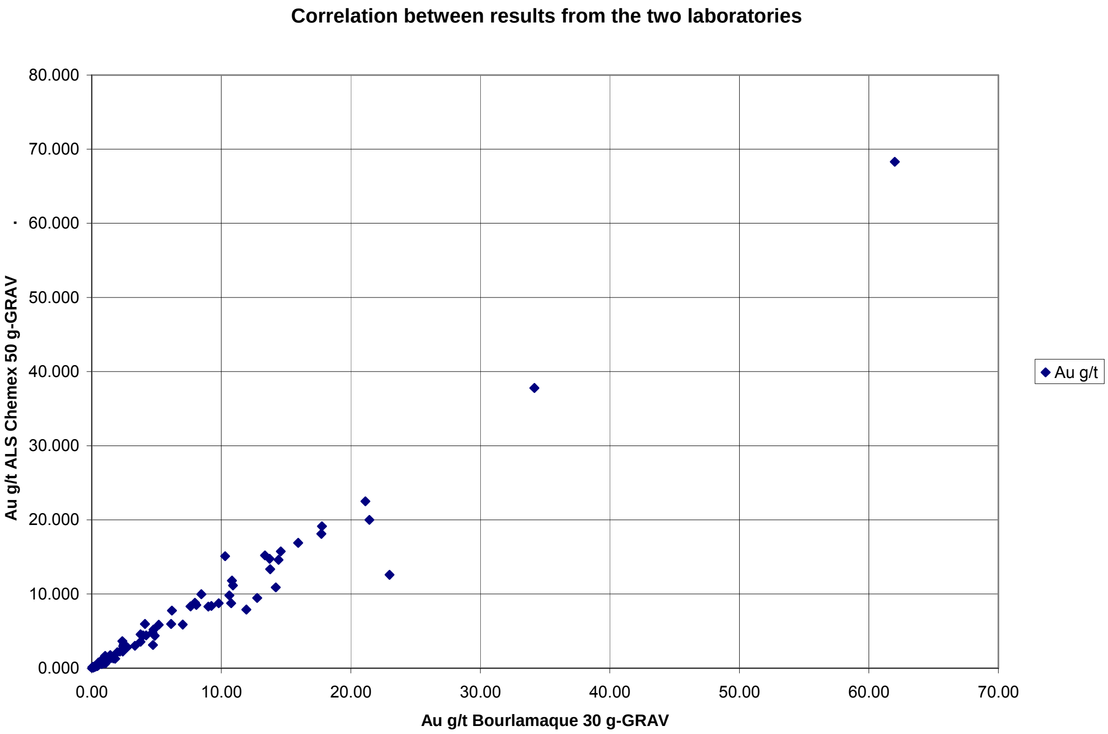


Figure 13.4: Correlation of the gold values between the Bourlamaque and ALS Chemex laboratories.

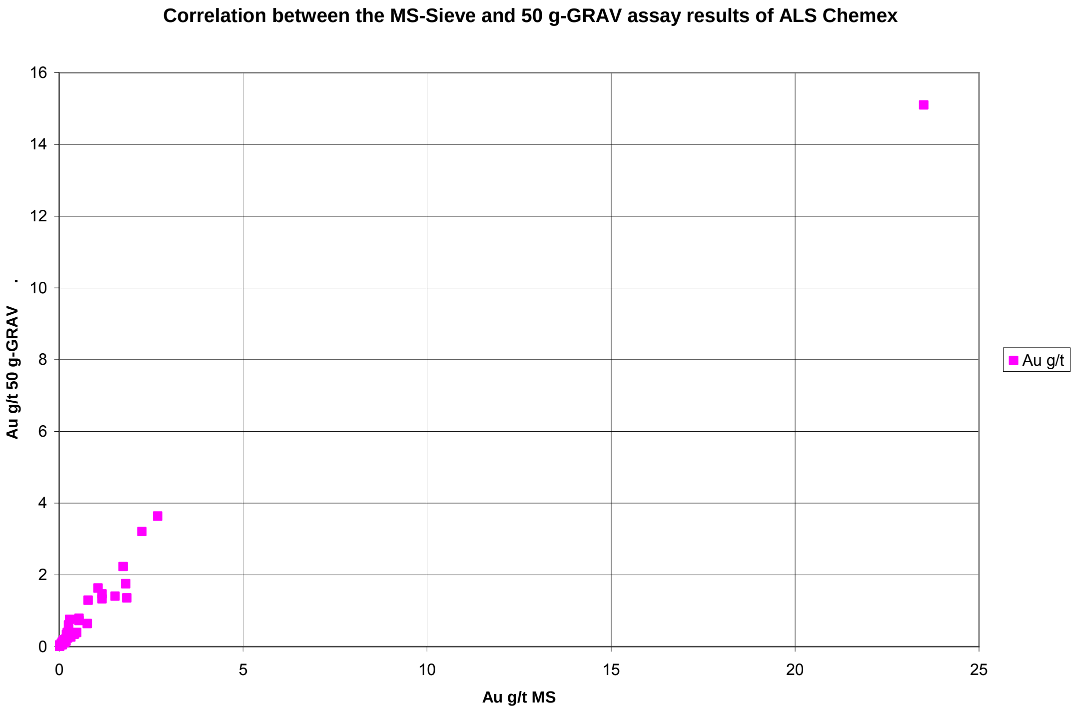


Figure 13.5: Correlation of the gold values between the MS-GRAV and 30 g-GRAV of ALS Chemex laboratories.

Most of the samples present a good correlation between the three results. There are only 11 samples out of 68 containing more than 1 g/t of gold in the results obtained from the ALS Chemex laboratory for the gold assay by MS-sieve and 30g-GRAV in the samples taken by Geostat. The sample of the hole 164, from 20 to 21 metres, shows a certain variation between the 30g-GRAV and the MS-GRAV methods. The results from the gravimetric method, from the two different laboratories, are 15.1 and 10.3, while the metallic sieve result is 23.5. There are not sufficient high gold values assayed to reach any conclusion.

Calculations

The correlation between the results from the Bourlamaque and the ALS Chemex laboratories is 0.986.

We applied the "signs test" to the results to compare the values from the re-assays of the different samples. This test is non-parametric for paired values, i.e. it does not imply the calculation of statistical parameters as the average or the standard deviation. It consists in counting the proportion of the samples whose value of group 1 is higher than that of group 2 and by adding the half-proportion to it where the samples are equal.

If the difference is only random, the proportion should range between $0.5\text{-}1/n^{½}$ and $0.5+1/n^{½}$, where n is the number of pairs implied in the comparison with a probability of 95%. This means that with only 10 pairs, the proportion can be as high as 81% and as low as 19%. With 21 pairs, the result must lie between 28 and 72%. With 100 pairs, it must be between 40 and 60% while with 1000 pairs, it must be between 47 and 53%

In the case of the results from the two laboratories, we got 143 pairs. The result should be between 0.5± 0.0836, i.e. 58% and 42%. The result is 67/142, 47%. Then, there is no bias observed in these results.

The correlation between the results from the MS-Sieve and the 30g-GRAV assay from the ALS Chemex laboratory is 0.981.

The result of the "sign test", for 68 pairs, should be between 0.5± 0.12, i.e. 38% and 62%. The result is 37/68, 54%. There is no bias observed in these results.

Sample preparation and assay

The procedure used for the sample preparation and the assay can be illustrated as follows

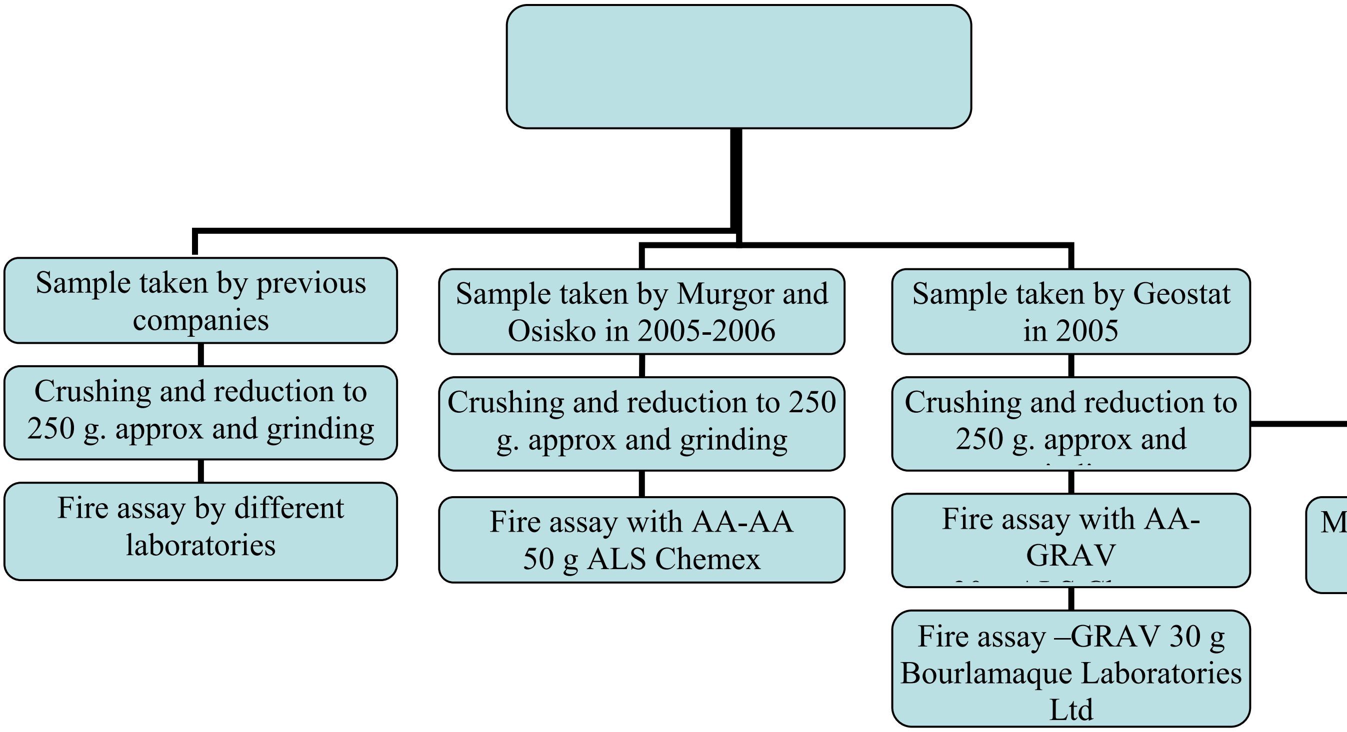


Figure 13.6: Diagram of the analytical checking procedure of the core samples.

13.2 Bachelor Lake Property

Most of the informations of this item are extracted from the InnovExplo Technical Report of December 2005 for the Bachelor Lake Property.

Data Verification

The Gemcom (GEMS 5.51) database used for the 2005 resource estimation included **15 192 assay results** from **394 diamond drill hole** records (each having hole ID, collar location, deviation test, geology, assay result, etc…). From the total, 325 were historical holes that were compiled and 69 holes were new (2005 program). Both the historical and the new data acquired were validated. Illustrations and Tables are provided in Appendix VI.

Data entry and validation

On the Bachelor claims, eighty (80) underground drill holes from the Bachelor Lake mine and 2 315 assays have been compiled by Y. Buro. No original assay certificates were available but 100% of the assay results were checked against the original logs by Tech2Mine (an independent firm in database management).

Tech2Mine has compiled and entered some surface holes and underground drill holes located below the 9th Level of the Bachelor Lake mine. Sixty (60) drill holes and 1 978 assay results were added to the database. Forty-two (42) assays have been checked against original assay certificates and all the others were checked against the original logs.

INNOVEXPLO (A. Carrier, M.Sc., P.Geo.) reviewed the geological setting of the gold mineralization at Bachelor Lake. Selected intervals from the hole 12-33 were examined. Core from mineralized intervals in the holes 12-13, 12-22, 12-23, 12-4, 11-11, 11-14 and 12-15 were reviewed (Refer to Figure VI-12).

On the Hewfran claims, one hundred eighty-five (185) holes and 7 650 assay results were transferred from Aur Gemcom database into the new database. Verification by Tech2Mine has included the verification of assay results against original assay certificates (for 2 557 results) and all other results were checked against the original logs.

Verification has also included the 2005 drilling program, nine (9) holes were checked (12-38; 12-39; 12-40; 12-41; 12-42; 12-43; 12-44; 12-46 and 12-48) and transferred from logging electronic supported Excel to the new logging software GeoticLog. All the assay results obtained during the 2005 campaign were checked against the original assay certificates for the sixty-nine (69) new holes.

Existing maps with stopes and drifts were used by Tech2Mine to support new drilling information. This data was provided to Innovexplo by Génivar (formerly Léandre Gervais & Associé(e)s inc.) in AutoCad format. Tech2Mine has also validated collar location and surveys when available for the data from 9th to 12th Levels. Numerous holes did not have deviation data and were then plotted linearly.

Bachelor grid orientation appears on several plans at 24° east of true north, which also corresponds to a communication from Y. Buro on May 2nd, 2005. A grid orientation of 24° east was used for data entry and during the whole 2005 program. Some deviation, typically related to grid variation as collar holes in the drift wall, appears. Until now, no adjustment has been made to correct this deviation, but the author recommended fixing this situation by rotating the whole database 0.5° to the west. The real Bachelor Lake grid orientation is more likely to be at 23.5° east and not 24° east.

2004-2005 check assaying results

In 2004, Wolfden, in the course of their due diligence, took some check samples and assaying of selected intervals from drill core. Table VI-4 in Appendix VI provides the results of the due diligence sampling from Wolfden (note that only portions of the drill holes and intersections were sampled).

In October, 2005, Innovexplo did re-sample 24 samples within the "A West" mineralized zones from six (6) drill holes of **Hewfran claims**. Fifteen (15) samples were coming from the Hewfran West area and nine (9) from the Hewfran east area. Core boxes containing mineralized zones intersections were already in Val d'Or, at the Alexis Mineral core shack. Selected cores were transported at Metanor's core shack where they were examined and re-sampled by Innovexplo's team. Quarter splitting was then performed by Metanor's technician for the fifteen (15) Hewfran

West BQ core samples while the other nine (9) were entirely sampled because of their AQ size. Two (2) high grade certified standards were also inserted into sequences, and samples were sent to ALS Chemex laboratory in Val d'Or. The same analytical package as the last underground drilling program was requested. Focus was made on several high grade assay results obtained by Aur.

All check samples were assembled and separated into 4 groups as described hereunder and detailed in Table VI-5:

1) 2 samples **below the cut-off grade** (under 0.1 oz/t Au) have a difference of **0.003 oz/t Au**;
2) 6 samples **close to the cut-off grade** (from 0.1 to 0.15 oz/t Au) have an average difference of **0.004 oz/t Au**. This important verification minimized the risk associated to misclassification of Ore and Waste block material;
3) 10 samples **close to the resource average grade** (from 0.15 to 0.3 oz/t Au) have an average difference of **0.015 oz/t Au**. This significant low difference also means that the overall average may not change drastically. Although some absolute difference can be as high as 0.284 oz/t Au, meaning that on a local basis, some ore blocks may have been overestimated or underestimated;
4) 10 samples with **high grade assay results** (over 0.3 oz/t Au) have a greater average grade difference (**0.043 oz/t Au)**. Locally, some grade can be either over or underestimated.

Confirmation drill hole from current BLJV exploration program

The BLJV is presently realizing a surface exploration program at Bachelor (October 2005). The current drilling exploration program includes one (1) confirmation drill hole located in the Hewfran East area in order to confirm Aur results. The drilling program is performed by the geologist, Patrick McLaughlin (Halo), under the supervision of Kevin Leonard (Halo).

The mineralized zone has been confirmed in the hole B05-117A and the "Main" zone was intercepted between 366.1m and 373.2m (at 10 397'N, 13 902'E and 8 893' Elev.) with results of 9.27 g/t Au over 1.8 m (0.27 oz/t Au over 5.9') contained within 3.56 g/t Au over 7.1 m (0.10 oz/t Au over 23.62').

14. Adjacent properties (Item 17)

This following information is part of the April 2007 Technical Report on Resources Evaluation of the Barry-1 Property by Geostat.

14.1 Barry I Property

We can only find little information on the properties adjacent to the Barry I Zone and Barry claims.

Murgor possess the Windfall property, located about 20 km northeast of the Barry I project. The mineralization found consists of a sericite-fuschite shear zone with 1-15% quartz-carbonate±tourmaline veins. These veins contain from traces up to 5% pyrite and locally some VG. The shear zone is between 2 to 5 metres thick, oriented N060° and dipping NNW at 60-70°. Murgor is presently working on that property by geophysics surveys and drilling campaign. The mineralization appears close to the surface.

At least two other known properties are close to Barry I present gold deposits and had resources evaluated. These deposits are not close to the surface and should probably be mined by underground operations. These resources are not NI 43-101 compliant. The resources of these properties published by the different owners are:

Property	Tonnage (mt)	Au g/t
Lac Rouleau deposit	544,000	7.0
Nubar deposit	564,000	6.2

Table 16.1: List of some of the deposits close to the Barry I property area.

As presented in figure 6.1, many projects and prospects are worked near the Barry I project.

Geostat's staffs have no mining interest in the sector.

14.2 Bachelor Lake Property

Informations extracted from InnovExplo December 2005 report

Adjacent Properties

The BLJV (Metanor/Halo) also owns claims adjacent and contiguous to the Bachelor property (refer "MJL Explorations and J. Hansen claims" description).

Several showings (gold and base metal occurrences) are located in the vicinity of the Bachelor and the Hewfran claims, these are detailed in the Table VII-1 in Appendix VII. Comparison between geological setting and features of the Bachelor Lake deposit with Coniagas and Lac Shortt is also presented in Appendix VII.

MJL Explorations and J. Hansen claims

Since August 10th, 2005, Metanor acquired two (2) claim blocks respectively located adjacent to the north and to the west of its property from MJL Explorations and J. Hansen. It includes 88 mining claims and covers a 2 287.69 ha area. Seventy four (74) claims (1 976.36 ha) are from MJL Explorations and 14 claims (311.33 ha) from J. Hansen.

In counterpart of the acquisition of this property, Metanor paid a sum of $10,000 and issued 50,000 Common Shares from its capital stock to the vendor. Metanor will pay the same amount to the vendor every year over a three-year period (2006 to 2008 inclusively). The acquisition amounts to

$40,000 and 200,000 Shares. Furthermore, the transaction includes a 2% NSR, redeemable under certain conditions.

The **Coniagas volcanic-hosted, massive sulphide deposit** lies on the J. Hansen claims, on the same road to the Bachelor Lake Gold Mine. The discovery of this past mine was made in 1947 by Dome Exploration Co. (Quebec) Ltd., and the Coniagas Mine was operated from 1961 to 1967.

The Coniagas Mine has been classified as a volcanic-hosted, massive sulphide deposit (VHMS) rich in Zn-Pb-Ag. It comprised four small massive sulphide lenses restricted to the felsic massive lapilli tuff unit (Allard et al., 1972; Doucet et al., 1994; Doucet et al., 1998). These four thin lenses combined reserve of 718 465 t grading 10.77% Zn, 1% Pb, 0.05% Cu and 183 g/t Ag (Allard *et al.,* 1972, *in* Doucet *et al.*, 1998). At the end of the operation in 1967, the potential for extensions at depth of the north and southwest lenses had been suggested, and further drilling had intersected 5.5% Zn and 1.4 oz/t Ag over 5.15 m and 4.6% Zn and 0.4 oz/t Ag over 5.85 m (G. Riverin, pers. commun., 1991 in Doucet *et al.*, 1998). The Lemoine and Scott Lake VHMS deposits in the Chibougamau region with 728 000 t (4.2% Cu, 9.6% Zn, 4.5 g/t Au, 83.85 g/t Ag) and 680 000 t (0.55% Cu, 6.9% Zn, 13.3 g/t Ag), respectively (Pilote and Guha 1995), have comparable dimensions, tonnage, and sulphide phases. Both deposits are part of volcanic cycle 1.

The exposed Main lens is 183 m wide, with an average thickness of 3.5 m, tapering to a width of 50 m at 340 m depth (G. Riverin, pers. Commun., 1991 in Doucet *et al.*, 1998). The remaining lenses are smaller and thinner, except for local thickening up to 10 m due to folding (Allard *et al.*, 1972). Numerous faults, first mapped by Allard et al., (1972), may be responsible for the disruption of a single, initially continuous, massive lens into four lenses. Folding in the vertical plane is attributed to shortening during regional deformation (Chown et al., 1992), in which the volcano-sedimentary succession in this region molded around pre-existing synvolcanic plutons.

Doucet et al. (1998) characterized the mineralization as an assemblage of sphalerite + pyrite + galena ± chalcopyrite, sulfides which have selectively replaced the porous felsic lapilli tuff unit. The well-laminated mineralization displays millimetre- to centimetre-scale diffuse bands of alternating sphalerite and pyrite which are accentuated by trains of milky quartz recrystallized from the matrix of the felsic lapilli tuff. An absence of massive sulphide fragments in the lapilli tuff suggested in situ mineralization without subsequent fragmentation, reworking and redeposition. This subsurface replacement massive sulfide deposit had features common to both Mattabi- and Noranda-type deposits.

> *A limited 5-10 m wide chloritic and sericitic hydrothermal Mn rich, alteration halo is discernible in the footwall and a meter-thick silicified zone overlies the mineralization (Doucet et al., 1995). The sphalerite + pyrite + galena ± chalcopyrite sulfide mineral assemblage in the Main lens differs significantly from the pyrite + chalcopyrite + sphalerite + pyrrhotite ± galena assemblage in the stringer zone. Chlorite compositions are Fe rich close to the mineralized zone, with an Fe/(Fe + Mg) ratio of 0.38-0.48 in the hanging wall and 0.65-0.70 below the ore. Delicate sulfide textures including colloform pyrite and concentric sphalerite are consistent with a low temperature of formation, whereas higher temperatures are inferred for the stockwork zone. Electron probe microanalysis of sphalerite supports inferred hydrothermal fluid temperatures. The low Fe contents (6.7-10.8 mol% FeS) in sphalerite associated with colloform pyrite of the Main lens contrast with the elevated Fe content (12.7-14.1 mol% FeS) in sphalerite from the stockwork (Doucet et al., 1998).*

The following description is reproduced from "Summary Report on the 1987-89 exploration program on the adjacent Aur Resources Inc. ("AUR") Hewfran property" prepared by Y. Rougerie (1989):

15. Mineral processing and metallurgical testing (item 18)

15.1 Barry-1 Property

During the spring of 2006, two composites samples of gold bearing ore were submitted to the mineral processing laboratories of Queen's University in Ontario by Murgor Resources, actually belonging to Metanor Resources Inc. The objective was to perform a bench-scale testwork for gold recovery. The conclusions presented by S. Kelebek are stating that conventional cyanidation provided the highest extraction of gold yielding results of 94.2% to 97.5%. A copy of the report was transmitted by Metanor and is reproduced in Appendix A

Description of samples sent to Queen's

A copy of the original document prepared by Ghyslain Deschênes geologist and a former Geostat's employee is attached to qualify that the samples are from Barry-1 site.

Murgor Échantillon 1 - métallurgie Barry Main Zone (March 2, 06)

Hole Name	From	To	Sample Number	Length	Assay1
MB-05-162	1.7	3	53502	1.3	3.14
MB-05-162	3	4	53503	1	1.035
MB-05-162	4	5	53504	1	5.19
MB-05-162	5	6	53505	1	5.97
MB-05-162	6	7	53506	1	8.53
MB-05-162	7	8	53507	1	5.87
MB-05-162	8	9	53508	1	4.56
MB-05-162	9	10	53509	1	4.37
MB-05-162	11	12	53511	1	8.33
MB-05-162	12	13	53512	1	8.77
MB 06 176	9	10	187671	1	0.83
MB 06 180	9	10	187831	1	1.48
MB 06 180	10	11	187832	1	5.255
MB 06 181	5	6	187861	1	1.445
				Average	4.63

Note: Scier tous les echantillons de cette serie (14 echantillons) et les grouper dans une meme chaudiere ou 2 chaudieres pour l'envoi. L'ensemble des 14 echantillons seront pour **l'echantillon composite 1**. SVP mettre chacun des echantillons dans des sacs avec les numeros de chaque echantillon sur le sac. Les echantillons seront des ¼ split de carottes.

Murgor Échantillon 2 - métallurgie Barry Main Zone (March 2, 06)

Hole Name	From	To	Sample Number	Length	Assay1
MB 06 175	4	5	187637	1	2.07
MB 06 175	5	6	187638	1	9.19
MB 06 175	8	9	187641	1	3.59

MB 06 175	9	10	187642	1	23.7
MB 06 175	10	11	187643	1	2.71
MB 06 175	12	13	187645	1	1.25
MB 06 175	14	15	187647	1	4.6
MB 06 176	7	8	187669	1	1.51
MB 06 176	10	11	187672	1	4.77
MB 06 180	3	4	187825	1	3.85
MB 06 180	4	5	187826	1	4.19
MB 06 180	20	21	187842	1	7.9
MB 06 181	4	5	187860	1	4.76
				Average	5.699

Note: Scier tous les echantillons de cette serie (13 echantillons) et les grouper dans une meme chaudiere ou 2 chaudieres pour l'envoi. L'ensemble des 14 echantillons seront pour **l'echantillon composite 2**. SVP mettre chacun des echantillons dans des sacs avec les numeros de chaque echantillon sur le sac. Les echantillons seront des ¼ split de carottes.

NOTE: Ces echantillons seront envoyes a l'adresse suivante :

Queen's University
Department of Mining Engineering
c/o George McIsaac
Goodwin Hall, Room 459
Kingston, Ontario
K7L 3N6
Tel : 613 533 2230

VOIR AUSSI PAGE SUIVANTE

Liste d'echantillons pour test de generation acide

Note : preparer chaque echantillon et le mettre dans un sac a part avec le numero d'echantillon sur chaque sac. Mettre ensuite les sacs dans une canne d'envoi qui mentionne test de generation acide et les envoyer a Ghislain Deschenes a l'adresse suivante :

Systèmes Géostat International Inc.
10 Boul. de la Seigneurie Est, Suite 203
Blainville, Québec, CAN
J7C 3V5
Tel. 450-433-1050

Amygdular andesite :	0-15% rounded calcite amygdules. Weak to mod. Reaction to HCl. 3-4% thin calcite veinlets. Traces of diss. Py	MB05-162	24	25	53524	85
Massive Andesite	Strong albitization. Strong to moderate biotite local hematite Traces to 5% diss. Py	MB05-163	11	12	53579	70
Porphyry	Coarse grained	MB05-166	6.6	7.6	167738	36

ORE					
2-3 g/t	MB05-162	15	16	53515	2160
4-6 g/t	MB05-162	9	10	53509	4370
	Note: ¼ de la carotte va a l'echantillon de metallurgie et le ¼ restant est pour le test acide. Indiquer dans la boite de carotte la raison pour laquelle il n'y a plus de carotte.				
6-8 g/t	MB-31	6.3	7.0	585773	6390
	MB-31	7.0	7.5	585774	6860
	Note: Grouper ces 2 echantillons pour n'en faire qu'un seul				

The results from Queen's University are indicative that the overall mill recovery estimated at +95% by Gilbert Rousseau (ref. Item 6) is obtainable at the Bachelor mill where a regular cyanidation circuit is in place.

15.2 Bachelor Lake Property

MINERAL PROCESSING AND METALLURGICAL TESTING

Bachelor Lake historic mill recovery

recovery rate of 93.0% obtained from historic milling at Bachelor Lake may be an appropriate Results from metallurgical testing were not available for Bachelor. It can be presumed that metallurgical tests were certainly being done prior to the mill opening. However, at Bachelor Lake mill facilities, the **historic mill recovery rate** during a period of seven (7) years (between 1982 and 1989) ranges from **91.8% to 93.7%** (refer to the Table below). The average gold estimate for the Bachelor Lake resources.

Table 16.1: Bachelor Mill Operating Statistic

Bachelor Mill Operating Statistic					
Year	Milled (short tons)	Head grade (oz/t)	Recovery rate (%)	Mill avail. (%)	Ounces produced
1982	73 178	0.124	-	-	8 077
1983	166 894	0.166	92.0	91.0	25 627
1984	156 086	0.140	92.4	85.5	20 104
1985	164 081	0.141	93.6	87.1	21 729
1986	136 520	0.158	93.7	83.3	20 140

1987	31 650	0.151	91.8	92.2	4 391
1988	144 298	0.146	92.7	86.9	19 516
1989	85 661	0.141	92.7	93.1	11 445
TOTAL	**958 368**	**0.147**	**93.0**	**88.4**	**131 029**

The BLJV is presently revising the actual conditions of the Bachelor Lake mill. This mandate has been given to Génivar (formerly Léandre Gervais & associés engineering firm) and will include a review of adequate mineral processing for the Bachelor Lake gold mineralization.
Further descriptions of the Bachelor Lake milling circuit from SNC-Lavalin in 1999 are reproduced in Appendix VIII.

Hewfran metallurgical test

From September 12 to 25, 1988, Aur performed a mill test from a bulk sampling from their 6th and 8th Level in the Hewfran east area. According to Aur memorandums, ± 3 300 tons (784 skips @ 4.0 tons per skip) were taken from the "Main" zone at the 8th Level and ± 2 300 tons (601 skips at 4.0 tons per skip) were taken from the "A" zone at the 6th Level. Bulk samples were milled at the Bachelor mill under Aur's engineer supervision, André Tardif.

No official final report on the mill test was found by Innovexplo. However, several memorandums indicated that they first milled the "A" zone but also mixed it with the "Main" zone muck samples. This mix was proposed to avoid documented dilution problems on the "A" zone (probably due to the presence of a chlorite rich footwall as suggested by Aur's geologists).

Before the mill test, Aur anticipated:
2 800 to 3 300 st @ 0.148 oz/t Au (from the "Main" zone at the 8th Level) and,
2 300 st @ 0.091 oz/t Au (from the "A" zone at the 6th Level)
For a total of:
5 600 st @ 0.123 oz/t Au (underground mucking sampling average)

The last daily report from the Bachelor mill (September 25, 1988) stated that **5 783 short dry tons were milled at an average of 0.1094 oz/t Au**, and yielded a bullion bar of 736.46 ounces, of which **632.84 ounces were gold**. The calculated mill recovery was 91.96 %.

The results correspond to a discrepancy with the Aur calculated stock pile grades (0.123 oz/t Au). The extra 175 to 200 tons of waste (0.001 oz/t Au) was explained by surface pad scraping, in view of the large surface area covered by the muck and was even considered as an excellent execution by scoop operators (Y. Rougerie, Aur's memorandum, October 1988). However, these added waste tons did not explain the discrepancy (11%).

In the same memorandum, Y Rougerie stated that after a planimetric measurement of the 6th and 8th level drifts, average tons and grade for each drift were calculated as follows:

8th Level	"Main" zone	± 3 300 tons @ 0.1455 oz/t Au
6th Level	"A" zone	± 2 300 tons @ 0.091 oz/t Au

16. Mineral resource and mineral reserve estimates (Item 19)

This following information is part of the April 2007 Technical Report on Resources Evaluation of the Barry-1 Property by Geostat.

16.1 Barry-1 Property

Geostat carried out the update of the resources estimation of the Barry I Main Zone Area project. This section presents the methodology used and the results of the resource estimation.

Data used

The data from the drill holes core used for the estimation comes from the drill holes database managed by Metanor. We added the results of the most recent drilling campaign (holes MB06-200 to MB06-257) and the hole LON88_3 to the database. We added a total of 59 holes. A total of 189 holes intersects the Barry I Main Zone in 437 instances. 48 of these holes are in the most recent drilling campaign. The surface topography still has to be surveyed by a certified surveyor. The topography used for the estimation is actually derived from the drill holes collars. Some of the drill holes was surveyed, others not. All the interpretation is done according to the local grid, which is roughly N330°. Most of the holes were drilled along the N330° orientation, roughly perpendicular to the general direction of the mineralized zones.

It was noted that mine coordinates and UTMs were not concordant in some of the older holes and some of the newer holes. Here are lists of missmatches between Mine and UTM coordinates. Recent holes have been tested in two dimentions.

The estimated convertion from the mining coordinates to UTM is:
Rotation: -28.8512936227465°
Translation:
X: +442886.77
Y: +5426103.98
Z: -1604.92

Hole Name	Mine-X	Mine-Y	Mine-Z	UTM-X	UTM-Y	UTM-Z	Error (m)
165	1011.05	-92	2009.98	443855.6	5426555	400.06	58.7
MB_36	1017.45	-74.46	2007.63	443815	5426527	402.71	2.6
179	1042.05	-185	2002.4	443895.3	5426434	397.48	12.7
164	1066.05	-72	2009.72	443883.4	5426538	404.8	33.3
163	1082.05	-101	2007.49	443896.3	5426558	402.57	24.3
MB06-229	650	-164	409.27	443535.52	5426269.02	409.27	5.0
MB06-234	770	-77	409.19	443539.6	5426374.12	409.19	67.9
MB06-213	800	-187	403.24	443682.02	5426329.29	403.24	5.3

MB06-214	800	-128	407.4	443653.52	5426378.59	407.4	4.3
MB06-218	850	-143	405.87	443706.34	5426388.11	405.87	6.1
MB06-219	850	-119	408.06	443692.93	5426412.58	408.06	5.0
MB06-224	900	-191	401.6	443771.16	5426372.54	401.6	4.2
MB06-223	900	-165	403.66	443758.84	5426393.49	403.66	4.2
MB06-227	950	-231	400	443815.47	5426382.16	400	26.6
MB06-210	1037	-143	401.8	443873.23	5426483.75	401.8	10.3
MB06-212	1056	-240	395.53	443933.04	5426407.76	395.53	7.1
MB06-204	1056	-205	396.79	443917.98	5426437.27	396.79	8.1
MB06-209	1067	-138	401.76	443892.46	5426498.16	401.76	4.5

The Mine coordinates were used for the calculations in this report. The incertainty on the location of the holes leads us to classify all the resource as inferred.

For the hole LON88_3 (Hole not used in the estimation because it does not intersect the Main zone), the mine coordinates have been calculated from the UTMs to be incorporated in the database.

Hole Name	Azimuth	Dip	Length	UTM-X	UTM-Y	UTM-Z	Mine-X	Mine-Y	Mine-Z
LON88_3	330	-90	100	443454.12	5426327.2	409.2	604.67	-78.26	2014.12

The next table lists the drill holes intersections used for the Barry I Main Zone Area resource estimation.

Hole Name	From	To	Au g/t	Hole Name	From	To	Au g/t
101	0	17.2	3.93	126	13.2	17	1.12
101	18.9	19.9	0.06	126	21.29	22.87	3.48
102	19.8	20.8	7.63	127	9.97	11.96	3.84
102	0	14.6	5.22	127	16.3	20.06	2.15
103	1.5	22	8.48	127	43.3	44.3	1.56
103	25	28	0.14	128	10.88	13.7	1.84
104	17.7	22.2	3.9	128	17.35	19.26	2.49
104	6	14	5.25	128	23.37	27.3	1.77
105	2	5	5.11	130	16.82	17.33	5.79
105	19.2	21.7	1.15	130	22.88	23.58	1.12
105	9	13	1.78	130	30.52	31.07	1.57
106	19.5	21.8	3.47	132	12.08	12.67	1.41
106	26.3	28.5	3.78	133	15	16.3	2.17
106	31.5	35	4.77	133	5.3	6.39	3.21
106	42	43.4	1.82	134	3.5	16.5	4.92
107	34	35.6	1.26	135	3.71	13.5	6.93
108	4.2	7	2.66	136	11.8	19.62	6.82
108	19	28.5	8.82	136	0	9.45	5.23
108	30.2	31.5	1.74	137	14.1	20.74	2.87
111	8.5	10.3	2.08	137	24.21	26.62	1.81
112	11.98	14.07	9.62	137	7.1	8.09	1.5
112	15.3	15.86	3.82	138	4.3	7.3	6.35

113	2.85	4.39	1.23	138	21	23.59	2.15
113	15.91	17.29	3.09	139	6.52	9	13.87
113	8.5	9.11	1.15	139	18.52	25.2	2.36
113	11	11.53	1.9	139	36.57	40	2.36
114	3	11.16	2.68	140	0.6	4.8	1.46
114	12.22	14.94	2.5	140	8.82	18.21	7.37
115	6.5	8.55	5.73	141	22.91	28	3.2
116	8.02	19.68	6.44	141	32.69	33.47	4.65
117	10.14	18.04	5.6	141	17	20.07	1.16
118	1.27	2.56	2.22	142	25.55	26.38	4
118	4.5	9.49	3.4	142	1	7.07	4.05
118	19.08	20.08	2.38	142	12.56	20.49	3.09
119	1.31	9.08	2.4	142	8.76	10.97	1.32
120	38.04	38.6	6.76	143	22	23	2.98
120	25.65	26.21	2.57	143	0	19.34	4.82
120	30.53	31.66	3.22	144	24.3	26.55	2.1
121	14.33	19.93	3.57	144	9.66	13.3	1.2
122	27.83	30	3.44	144	19.89	20.4	1.36
123	26.09	28	1.24	144	36.86	37.43	1.37
123	13.13	17.93	1.61	145	13.2	17.68	1.58
123	6.57	6.93	4.56	145	0	10.27	5.37
124	34.76	38.74	1.99	146	21.74	22.77	1.11
124	16.65	23.88	2.82	146	4	9.48	2.39
125	20.21	23.7	7.93	147	35.43	36.35	2.06
125	13.98	16.1	2.05	147	23.33	24	1.02
125	27.36	27.86	2.43	147	9.1	17.7	1.15

Hole Name	From	To	Au g/t	Hole Name	From	To	Au g/t
150	5.4	8.14	2.12	179	50	52	4.18
150	32.78	33.81	4.7	180	1	14	5.4
151	0.7	4.2	3.51	180	20	26	5.61
151	7.89	12.96	3.74	181	1.2	7	2.84
152	75.77	78.03	3.93	181	10	12	1.14
152	20.5	27.52	3.76	181	18.2	19.7	0
154	33.94	39.78	6.53	182	0.6	7	1.94
154	17	18.14	1.11	182	11	12	1.12
154	10.7	11.86	1.49	182	15	16	1.51
158	34.2	35.57	2.93	182	18	19	1.31
158	44.8	52	4.44	183	20	22	5.01
159	20.4	27.5	0	183	14	15	2.78
162	1.7	21	4.52	183	6	8	1.39
163	15	16	1.36	184	5	6	1.55
163	6	7	1.75	184	9	11	1.66
163	21	23	1.4	184	18	19	6.29
164	20	21	15.1	184	24	25	5.08
164	0.5	8	1.9	184	21	22	1.7
164	10	11	1.41	185	8	12	2.21
165	4.5	5.5	3.59	185	22	23	2.82
165	28.5	29.5	1.71	186	21	23	2.57
165	15.5	16.5	0.01	186	3	8	1.59

166	45.4	48.4	9.08	186	16	17	3.95
166	23.4	24.4	1.2	187	11	18	1.96
166	51.4	53	1.63	187	24	26	1.92
167	26	27	2.29	187	40	44	5.66
168	4	5	1.89	187	7	8	5.31
169	25	27	3.82	188	12	13	4.78
170	6	8	2.73	188	22	29	1.57
171	31	32	2.28	188	45	46	4.27
172	8	11	1.77	189	23	31	2.26
172	28	31	3.85	189	40	43	2.91
172	40	42	1.15	190	20	27	7.21
172	43	44	1.09	190	56	60	1.8
173	14	18	3.29	191	17	26	2.37
173	26	28	6.15	192	36	39	3.94
174	23	25	2.72	192	48	53	6.4
174	32	34	2.92	192	25	26	1.76
174	39	41	1.08	193	11	12	2.77
175	3	15	4.5	193	20	26	1.08
176	4	13	5.86	194	12	15	3.47
177	11	12	8.27	194	18	20	1.48
178	3	8	2.29	195	29	30	2.42
179	23	24	6.15	195	19	20	1.41
179	68	70	15.84	197	3	4	3.01
179	32	35	1.9	197	14	19	6.91
179	6	7	4.85	197	8	11	1.09
179	44	46	1.69	199	35	39	20
199	81	83	2.71	MB_18	60.6	64.15	3.63

Hole Name	From	To	Au g/t	Hole Name	From	To	Au g/t
199	89	90	31.1	MB_18	39.9	42.38	2.51
BA4_83_10	54.8	58	1.86	MB_18	8.2	10.5	4.05
BA4_83_10	15.2	16.3	0.52	MB_18	31.15	31.45	3.15
LON88_2	50	51.6	2.8	MB_18	36.4	37.55	2.81
LON89_5	28.6	29.7	1.85	MB_19	14.7	15.6	1.48
LON89_6	71.3	72.8	3.6	MB_19	19.5	21.5	0.4
LON89_6	60.5	62	1.1	MB_19	1.5	4	11.1
LON89_6	45	46	0	MB_19	8	9.2	11.9
LON89_9	29.4	31.2	1.3	MB_20	18.5	21.3	1.4
MB_01	27.98	31.97	1.69	MB_20	6.7	11.4	3.46
MB_01	14.65	25.7	3.11	MB_20	2.7	3.95	3.59
MB_02	15.9	18.2	4.75	MB_21	0	16.5	4.43
MB_02	28.95	29.92	4.13	MB_21	18.85	19.55	7.64
MB_02	33.45	36.9	3.49	MB_22	7.3	8.3	1.78
MB_03	20	27.1	4.47	MB_22	13.3	15.8	1.56
MB_03	51.5	52.7	2.15	MB_22	19.5	21.3	1.27
MB_04	63.8	65.85	6.3	MB_22	35	36	4.37
MB_04	4	5	3.05	MB_22	29	30	2.53
MB_06	26	27.1	2.41	MB_22	24	25	1.78
MB_06	33.4	37.3	4.46	MB_23	21.2	23.3	7.81
MB_06	53.8	54.1	2.47	MB_23	16.5	17.2	1.03

MB_07	61.6	65.9	6.36	MB_23	3	7.45	1.96
MB_07	25.5	27	1.14	MB_23	0.2	1.5	1.48
MB_07	34.8	35.4	3.82	MB_24	0.3	7	2.39
MB_08	35	37.3	2.49	MB_24	18	20.6	2.67
MB_08	13.1	13.5	1.44	MB_24	9	9.5	1.25
MB_08	52.5	54	1.67	MB_24	15.3	16.4	5.77
MB_09	49.7	54.3	2.4	MB_25	2	4	5.45
MB_10	27.6	29.6	2.1	MB_25	7.5	13	5.67
MB_10	64.2	65.3	1.02	MB_25	16.5	23	3.2
MB_11	64.1	66.4	4.29	MB_25	32	34	3.7
MB_11	40.4	42.55	3.94	MB_26	4	9.35	6.18
MB_11	52.7	60.6	0.83	MB_26	15.25	17.5	3.73
MB_12	7	15	9.51	MB_26	26.3	27.5	1.38
MB_12	68.9	71.6	1.75	MB_27	9.8	16	3.57
MB_12	37.9	40.6	0.18	MB_27	1	3.65	4.05
MB_13	22.7	31.65	2.06	MB_27	26	28.2	0.99
MB_13	6.4	7.2	1.09	MB_27	7.2	7.7	2.12
MB_13	32.5	33.7	1.83	MB_28	0	15.25	3.99
MB_13	19	20.2	1.08	MB_28	20	21	0.75
MB_14	13.2	14.4	4.87	MB_29	0.2	20.6	6.06
MB_14	25.9	29.05	7.9	MB_30	0	4	2.6
MB_14	21.2	21.6	3.09	MB_30	9.7	11	2.38
MB_15	64.4	69	2.48	MB_30	0	0.5	4.32
MB_15	79.2	80.2	2.78	MB_30	14	22.5	3.85
MB_15	49.9	50.5	7.82	MB_31	25	28.1	16.53
MB_16	4.5	13.6	3.03	MB_31	1.3	22.5	7.01
MB_32	17	21.2	5.62	MB06-202	13	14	23.6

Hole Name	From	To	Au g/t	Hole Name	From	To	Au g/t
MB_32	5.3	15	5.7	MB06-202	54.5	55	7.04
MB_33	12.5	14.6	4.5	MB06-203	65.2	66.4	3.82
MB_33	1	7.5	3.09	MB06-203	42.1	43.2	0.44
MB_34	0	3.5	2.17	MB06-204	20.6	31.5	0.74
MB_34	10.4	14.4	3.01	MB06-204	48.6	49.6	1.6
MB_35	18.7	19.7	1.12	MB06-204	64.5	66	0.02
MB_35	0.4	6	2.32	MB06-205	21.5	22.7	1.15
MB_36	24	24.5	3.39	MB06-205	27.9	30.2	1.78
MB_36	3	7.5	2.1	MB06-205	40	41.4	3
MB_37	3	4.6	2.8	MB06-205	54.7	56	3.02
MB_37	13.3	14.4	2.19	MB06-206	39.2	39.7	6.31
MB_37	29.1	30	3.16	MB06-206	24.5	29.1	1.54
MB_38	8	8.5	1.01	MB06-207	52.6	54.4	3.93
MB_38	0.3	1	2.28	MB06-207	30.9	36.7	1.69
MB_39	5.2	14.5	3.66	MB06-208	18.3	21.4	3.1
MB_39	18	22	0.79	MB06-208	49.5	50.5	2.28
MB_40	0	20.35	4.09	MB06-209	16.8	18.7	1.79
MB_40	22	22.9	0.35	MB06-209	30	43.5	2.16
MB_41	9.9	13	1.18	MB06-209	48	49	1.13
MB_41	3	5	1.13	MB06-210	21.8	26.1	2.18
MB_41	27	29	6.13	MB06-210	41.2	43.1	3.91

MB_42	140	141.5	5.35
MB_42	184.5	187	3.06
MB_43	37.8	43.8	6.14
MB_44	64.7	66.5	4.84
MB_44	72.7	80.5	3.7
MB_44	20	25.5	3.52
MB_44	102.4	106.5	1.49
MB_44	37.3	37.6	2.44
MB_45	41	48.7	9.72
MB_46	12.1	14	8.11
MB_49	101.5	112	2.81
MB_49	52.6	53.6	3.63
MB_49	65.3	67.5	1.38
MB_50	115	123.5	2.66
MB_51	172.6	174.6	1.08
MB_51	142.9	144.45	1.99
MB_52	55.4	56.4	0.03
MB_53	39.7	41.8	7.96
MB_53	75.7	76.8	2.51
MB_54	124.8	135	3.67
MB_54	143.1	149.5	2.66
MB_54	99	101	1.4
MB_62	149.6	157.3	6.38
MB06-200	70.2	75.5	5.85
MB06-200	110	112.2	1.99
MB06-201	46.5	49.7	6.14
MB06-201	33.9	34.9	5.62
MB06-201	68.1	70.6	14.95
MB06-211	61.7	72.7	2.66
MB06-211	11.3	19.5	5.28
MB06-212	59.6	63.8	5.12
MB06-213	57.5	64	3.42
MB06-213	72.7	78.1	4.24
MB06-213	101.4	103.7	2.01
MB06-213	6	7	2.09
MB06-213	21.4	23.5	2.41
MB06-214	42.6	47.6	4.73
MB06-214	33.6	34.6	3.17
MB06-215	35	39.5	3.8
MB06-215	19.6	20.1	15.25
MB06-215	24.6	25.2	4.96
MB06-215	28.9	29.9	1.45
MB06-216	19.5	20.2	2.06
MB06-217	29.3	31	2.11
MB06-218	40	41	28
MB06-218	61.6	67.5	1.69
MB06-218	79.2	81.4	7.32
MB06-219	31	34	3.29
MB06-220	31	33	3.3
MB06-220	34	35	1.06
MB06-220	56	58	2.32
MB06-221	16	19	4.11
MB06-221	24	25	3.43
MB06-221	41.7	42.7	2.01
MB06-222	42	44	1.73
MB06-222	57	59	1.33

Hole Name	From	To	Au g/t
MB06-222	68	71	1.87
MB06-223	104	106	3.31
MB06-223	111.2	113.3	3.36
MB06-223	117.7	119.7	4.58
MB06-223	87	88	1.21
MB06-223	81	83	0.94
MB06-224	20.2	29.1	3.14
MB06-224	117	119	1.21
MB06-224	122.1	124	0.98
MB06-225	8.3	9.5	1.02
MB06-225	26.4	27.2	7.46
MB06-225	34.9	35.9	6.99
MB06-226	14	15	2.51
MB06-226	27.3	28.3	23.9
MB06-226	59.3	67.5	5.02
MB06-226	80	86	10.1
MB06-227	89.6	94.6	2.83
MB06-228	49	50	4.37
MB06-228	9.5	10.5	2.9
MB06-229	27.5	34.8	2.27
MB06-230	20.2	26	2.74

MB06-231	7.2	16	3.18
MB06-231	38.1	42	6.41
MB06-232	8.2	14.7	0.97
MB06-233	50.1	53.8	4.29
MB06-235	12	14	3.58
MB06-236	22.8	23.3	9.29
MB06-236	17.5	19	1.64
MB06-237	44.4	46.4	3.88
MB06-237	72.2	79.7	6.77
MB06-238	38.5	42.5	18.68
MB06-238	65.5	71.5	1.57
MB06-239	60.4	61.4	6.09
MB06-239	35.8	36.8	4.52
MB06-239	10.2	11.2	2.71
MB06-249	51	57	2.65
MB06-249	91	92	1.27
MB06-250	27.3	29.3	1.79
MB06-251	61.7	63.4	3.05
MB06-252	56.5	59	2.84
MB06-253	28.1	29.7	2.44
MB06-254	36.4	44.6	1.84
MB06-255	36	41	2.62
MB06-255	62	66.5	2.38
MB06-255	24	25	1.14
MB06-256	44.2	51	5.89
MB06-256	73	75	2.53
MB06-256	80	80.8	6.64
MB06-257	62.2	64.4	5.88

Table 2: List of the mineralized intersections used in Barry I Main Zone Ares resources estimation.

Mineralized zones

Geostat, based on the previous interpretation provided by the geologists of Murgor, carried out the geological interpretation of the mineralized zone. We checked the agreement between geological interpretation and the mineralized intersections defined from the drill holes. We designed envelopes around the composites formed of the mineralized zones estimated according to the geological interpretation between the sections and our knowledge of the deposit. The correlation with the rock types was not validated but for the quartz-porphyry, as the core description need to be standardized according to the new geological model and to the lack of information regarding the geology in the database.

The gold mineralization on the property is closely related to the amount of veining, intensity of alteration and percentage of sulphides. All are key factors and generally all three of these elements are needed in order to obtain significant gold mineralization. This style of mineralization produces sections with significant gold concentrations but they are commonly narrow with widths in the order of 0.3 to 1.5 metres. The thicker mineralized sections represent a higher density of these narrow zones.

Gold-bearing mineralization lies in pyritized and moderately altered volcanic flows near contacts with quartz-feldspar porphyry dykes and plugs. Mineralization is characterized by a system of east-northeast and north-northeast sheeted quartz-carbonate veins dipping at 40-60 degrees to the south-southeast and the east-southeast respectively. The known mineralization is up to 500 metres long, up to 150 metres wide and tested, for the majority of the drill holes.

The Barry I gold mineralized envelopes are located south of a major shear and fold zone. The mineralized envelopes represent elongated dome shapes even if the mineralized veins are dipping moderately. The mineralized envelopes show variable thicknesses, which vary from a few centimetres to more than 30 metres. The actual mineralized envelopes of the Main Zone Area occur mainly in the first 30 metres of rock below surface. The presence of the gold mineralization located at the top of the dome shaped is associated with silicified volcanites. At depth greater than 30 metres, the volcanic rocks are more mafic, massive and show the presence of vacuoles.

The gold grades vary within the envelopes and visible gold is frequently observed in the core.

The folding periods having happened after the intrusion of the first porphyry body might have remobilized the gold mineralisation in the fold noses.

Composites

The method used to estimate the resources is to evaluate the grades of regular blocks inside the mineralized envelopes. This method requires the use of samples of regular length. Composites are then created starting from the original samples. We used a composite length of 1.5-m with a minimum of 0.8 meter sampled with some dillution. We consider this length suitable compared to the dimension of the blocks of the model (3 metres E by 3 metres N by 3 metres Z) and to smooth the effect of the high grade samples with shorter lengths. The highest grades used are 60.9 g/t Au over 1 metre (Hole MB06-238) and 54.7 g/t Au over 0.5 metre (Hole 139). Moreover, we consider the average thickness of the mineralized zones at 6 metres.

Analysis of the gold grades distribution

The grades of the 1.5-m composites used in the calculation show a distribution approaching the lognormal law. There is presence of high values. In the Barry project, the maximum gold content is 39.8 g/t for a 1m composite and 26.03 g/t for a 1.5m composite. The following figures present the histograms and cumulative frequency plot of the 1.5-m composites in the Barry I Main Zone Area project.

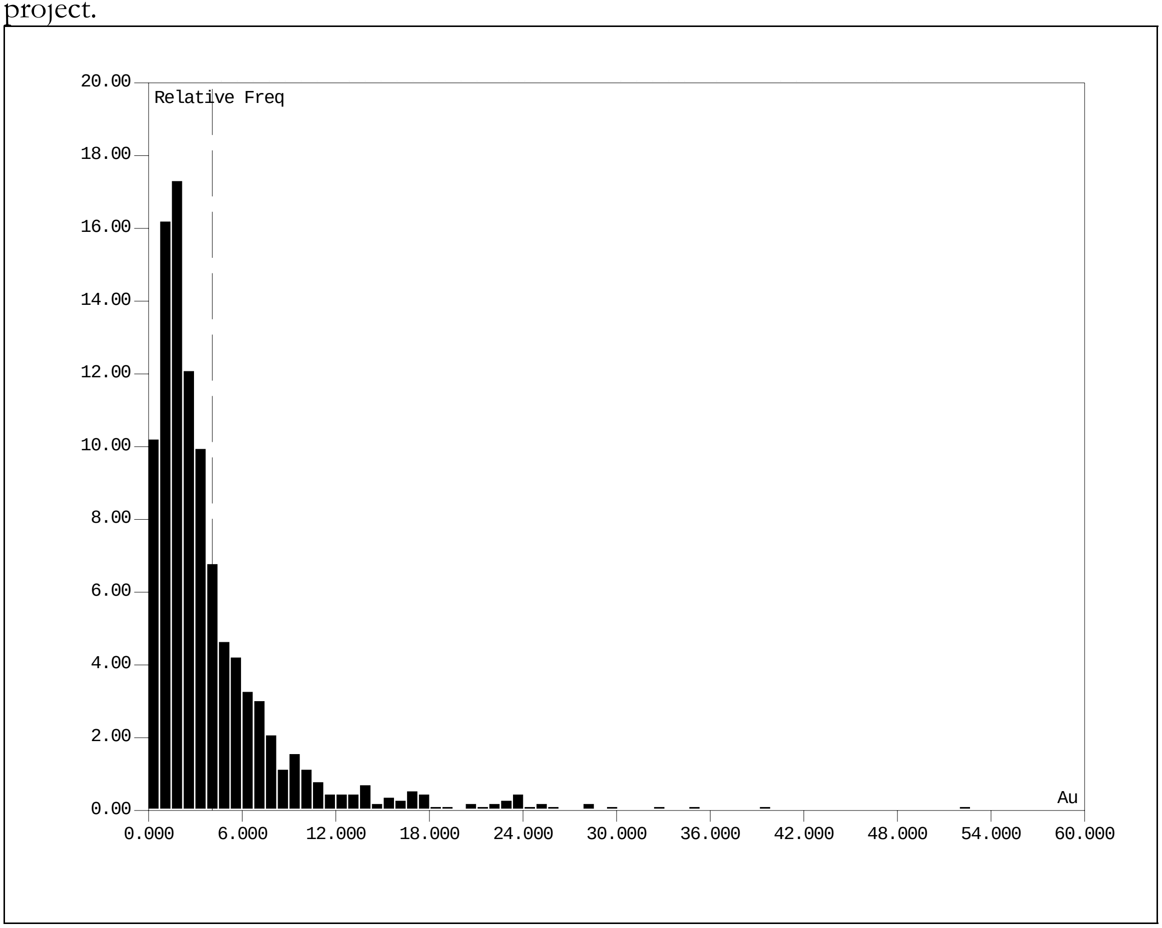


Figure 29: Regular distribution of the 1.5-m composites intersectin the Main Zone.

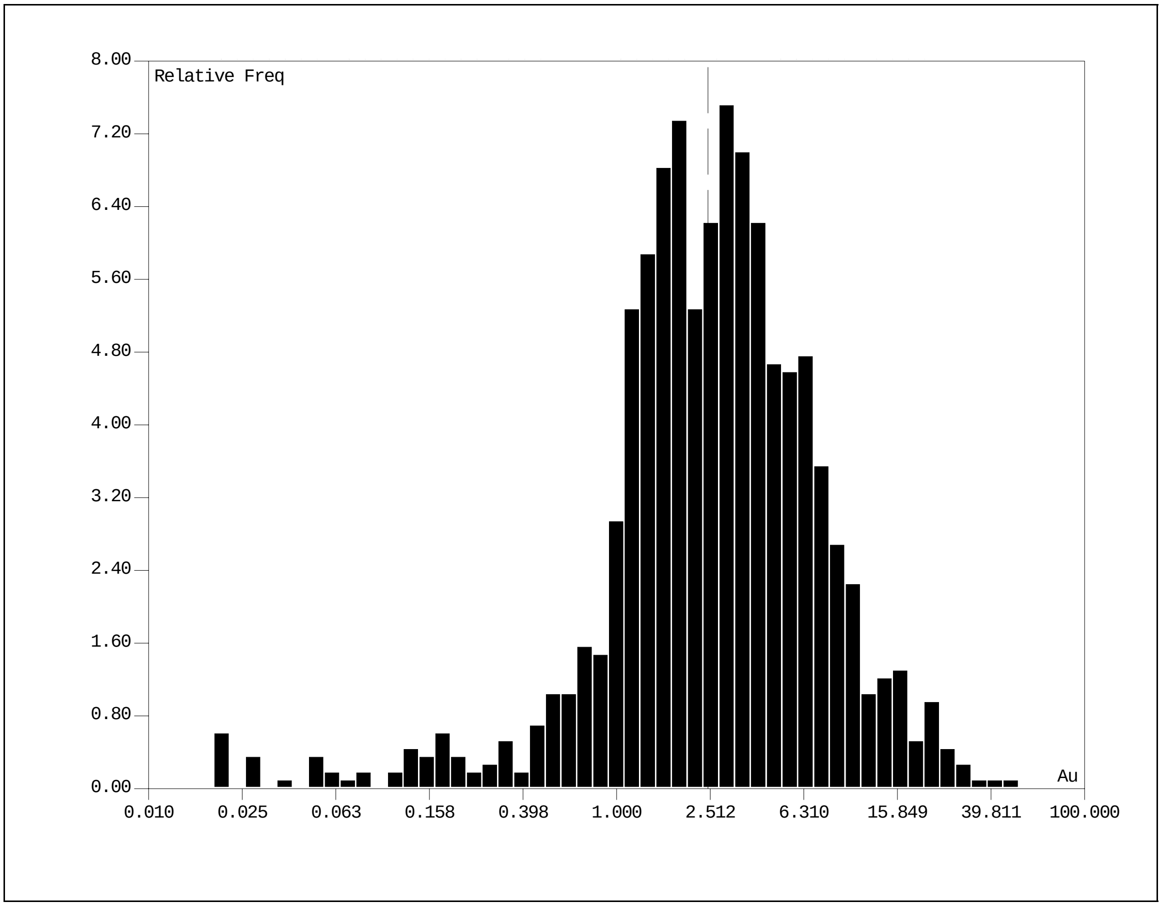


Figure 9: Log histogram of the 1.5-m composites of Barry I Main Zone Area.

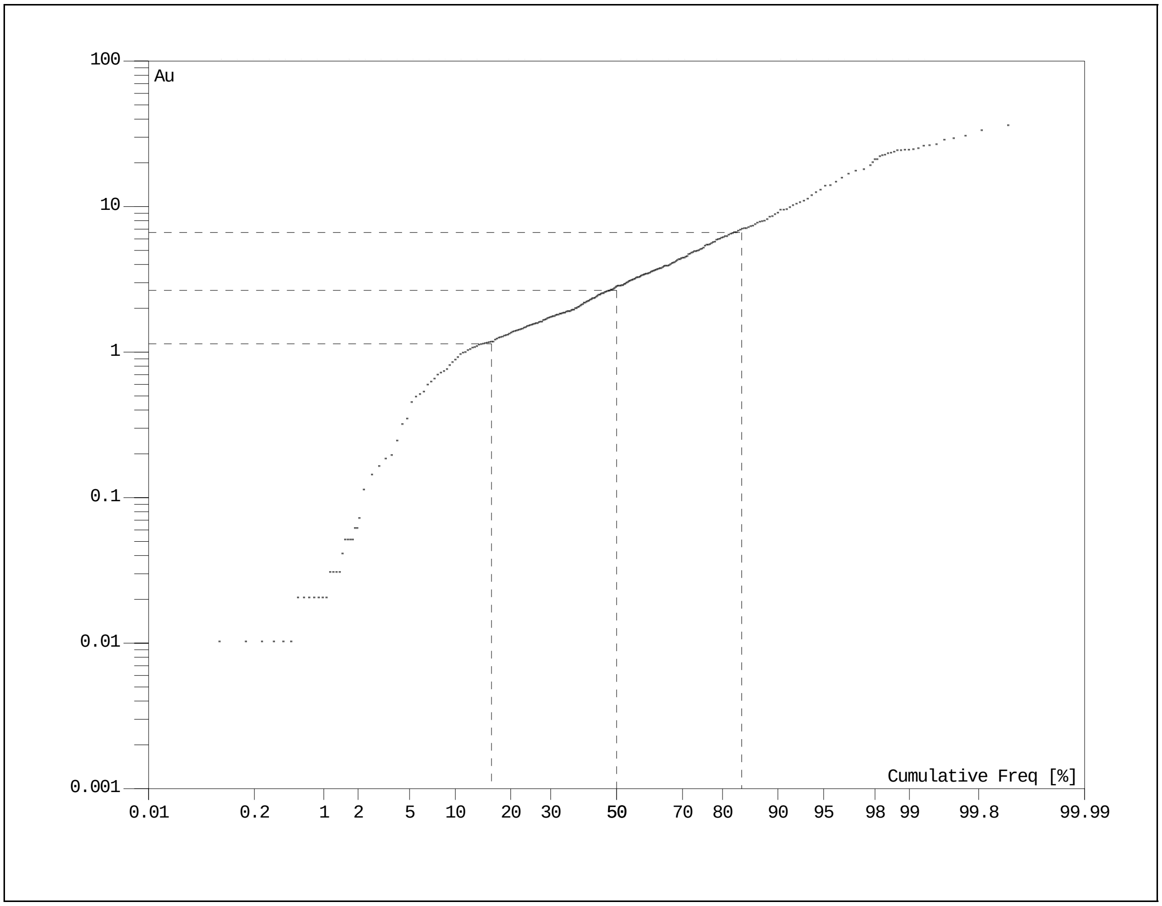


Figure 10: Curve of cumulated frequencies for 1.5-m composites of Barry I Main Zone Area.

As the histogram of the precedent figure shows, the high gold values do not deviate significantly from the lognormal curve, except for the very low grades. Therefore, according to this test, it is not necessary to cut the high values.

It is also interesting to consider the contribution of the gold contained in the high-grade samples proportionally to their number in the data set. We consider an anomaly the situation when more than 10% of the gold contained in the high grades is found in less than 1% of the set of composites. In this case, 10% of the gold is contained in 1.55% of the composites. The following graph shows the gold contribution of the high-grade gold values according to the corresponding quantity of data (expressed as a percentage.) We conclude that the ratio 10:1 is not exceeded and that, according to this test, it is not necessary to cut the high gold values of the 1.5-m composites in the Barry I Main Zone Area project.

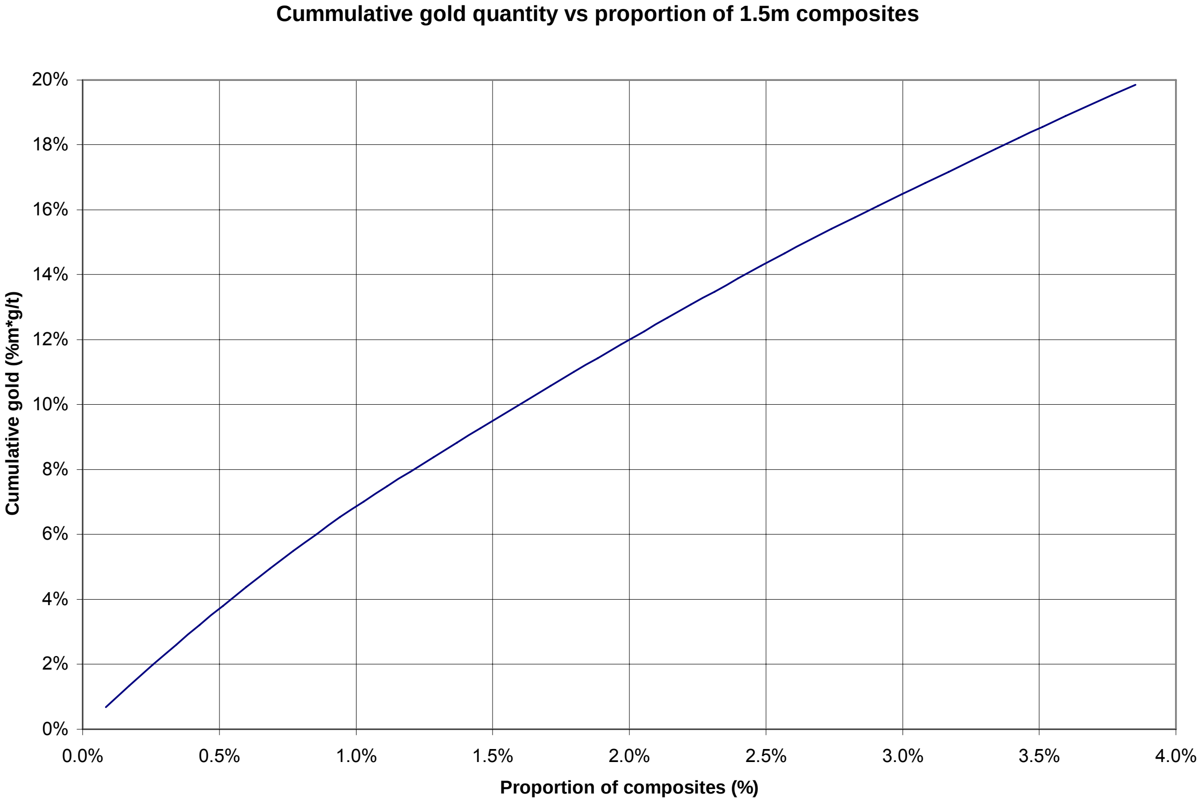


Figure 11: Relation between the cumulative contributions of the gold found in the 1.5-meter composite samples, Barry I Main Zone Area.

Spatial continuity of the gold distribution

The spatial continuity of gold was calculated in the 2006 report. All the calculated variograms suggested a short distance of influence of the samples, at maximum to 13.5 metres. The drilling pattern should be smaller than 13.5 m to be able to define a good anisotropic variogram.

Resource estimation

Definitions:

The classification of Mineral Resources and Mineral Reserves used in this report relies with the definitions provided in National Instrument 43-101, which came into effect on February 1, 2001. We further confirm that we have followed the guidelines adopted by the Council of the Canadian Institute of Mining Metallurgy and Petroleum. The relevant definitions for the CIM Standards/Nl 43-101 are as follows:

1. **Preliminary Feasibility Study**
   A Preliminary Feasibility Study is a comprehensive study of the viability of a mineral project that has advanced to a stage where the mining method, in the case of underground mining, or the pit

configuration, in the case of an open pit, has been established, where an effective method of mineral processing has been determined, and includes a financial analysis based on reasonable assumptions of technical, engineering, legal, operating, and economic factors and evaluation of other relevant factors which are sufficient for a Qualified Person, acting reasonably, to determine if all or part of the Mineral Resource may be classified as a Mineral Reserve.

2. **Exploration Information**
   Exploration information means geological, geophysical, geochemical, sampling, drilling, trenching, analytical testing, assaying, mineralogical, metallurgical and other similar information concerning a particular property that is derived from activities undertaken to locate, investigate, define or delineate a mineral prospect or mineral deposit.

3. **Mineral Resource**
   A Mineral Resource is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

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

5. **Indicated Mineral Resource**
   An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

6. **Measured Mineral Resource**
   A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

7. **Mineral Reserve**

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

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

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

Resources evaluation

The resources of the Main Zone Area were evaluated by the section and plan method. The calculation method used the average of the composites grades within the entire estimated envelope to calculate a grade. No individual envelopes for every composite were created for these evaluations. The minimum grade for a composite to be used for the calculation is 1 g/t Au over 1 metre. The results are the following:

Method	Tonnage, t.	Grade, g/t	Vertical Extents
Sections 700E-1200^{E}	1,747,000	3.81	1,825m to top
Plan 1925-2009	1,526,000	3.90	1925m to top

Table 3: Preliminary resources evaluation of the Main Zone Area by the average section and plan methods.

Block modelling of the Main Zone Area resources

A block model of 3 metres by 3 metres by 3 metres is used. The parameters of the block model are the following:

Origin of the model	East	North	Elevation
Dimension of the blocks	3	3	-3
Minimum coordinate	602m (1)	-250m (1)	1904m (38)
Maximum coordinate	1,250m (217)	0m (85)	2015m (1)

Table 4: Geometric parameters used for the Main Zone block model estimation.

16.1.1 Block modelling of the Main Zone Area resources by inverse distance

The settings in BlkCad for the estimations were:
Search Ellipsoid: 80m x 80m x 40m Orientation: 180º x -40º x 0º
Minimum number of samples to use: 1
Maximum number of samples to use: 12
Maximum number of samples from one hole: 4
Block discretisation: 3 x 3 x 3
Use ellipsoid influenced distances in calculation: Yes

The estimated resources were classified in accordance with the specifications of the 43-101 Policy, namely in measured, indicated, and inferred resources. In spite of the close distance between the actual drill holes and a survey by a certified surveyor of most of their collars in the main Zone, no measured resources were defined in the Main Zone. This is due to the absence of a detailed topographic survey of the actual surface, the topographic surface acting as a mineralization contact boundary, the incertitude regarding the position of the hole collars not surveyed. The mineralized envelopes beginning close to the surface, this parameter is important for measured resources. The classification criterion is based on a scheme of proximity and the parameters are as follows:

Category	**Search ellipse** (oriented according to the lenses, 90ºx0ºx40º)	Minimum number of composites	Maximum number of composites per hole
Indicated	25 m, 12.5 m, 6m	4	2
Inferred	Inside the mineralized envelope, not indicated		

Table 5: Parameters used for the classification of the resources.

16.1.2 Total Resources by Category

The following table presents the results of the resources estimated by inverse distance and rounded:

Total resources inverse distance method(No cut-off) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**Density**	**Au (g/t)**	**Oz Au**
Indicated	**415,000**	**148,000**	**2.80**	**4.05**	**54,000**
Total	**415,000**	**148,000**	**2.80**	**4.05**	**54,000**
Inferred	1,102,000	394,000	2.80	3.78	133,800

Table 6: Resources evaluation by inverse distance of the Main Zone Area.

The following table presents the results of the estimated resources using different cut-offs. The resource evaluation was done according to the possible scenario of open pit mining of the upper part of the deposit.

The first section presents the total resources calculated by inverse distance and rounded, and the second one the resources according to different cut-offs.

Total resources inverse distance (No cut-off) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**Density**	**Au (g/t)**	**Oz Au**
Indicated	**415,000**	**148,000**	**2.8**	**4.05**	**54,000**
Total	**415,000**	**148,000**	**2.8**	**4.05**	**54,000**
Inferred	1,102,000	394,000	2.8	3.78	133,800
Total resources inverse distance (Cut-off of 1 g/t) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**2.80**	**4.00**	**36,100**
Indicated	**415,000**	**148,000**	**2.8**	**4.05**	**54,000**
Total	**415,000**	**148,000**	**2.8**	**4.05**	**54,000**
Inferred	1,102,000	394,000	2.8	3.78	133,800
Total resources inverse distance (Cut-off of 2 g/t) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**Density**	**Au (g/t)**	**Oz Au**
Indicated	**385,000**	**138,000**	**2.8**	**4.23**	**52,300**
Total	**385,000**	**138,000**	**2.8**	**4.23**	**52,300**
Inferred	966,000	345,000	2.8	4.07	126,600
Total resources inverse distance (Cut-off of 3 g/t) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**Density**	**Au (g/t)**	**Oz Au**
Indicated	**277,000**	**99,000**	**2.8**	**4.89**	**43,600**
Total	**277,000**	**99,000**	**2.8**	**4.89**	**43,600**
Inferred	690,000	246,000	2.8	4.70	104,300
Total resources inverse distance (Cut-off of 4 g/t) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**Density**	**Au (g/t)**	**Oz Au**
Indicated	**174,000**	**62,000**	**2.8**	**5.74**	**32,100**
Total	**174,000**	**62,000**	**2.8**	**5.74**	**32,100**
Inferred	404,000	144,000	2.8	5.59	72,600
Total resources inverse distance (Cut-off of 5 g/t) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**Density**	**Au (g/t)**	**Oz Au**
Indicated	**109,000**	**39,000**	**2.8**	**6.49**	**22,800**
Total	**109,000**	**39,000**	**2.8**	**6.49**	**22,800**
Inferred	225,000	80,000	2.8	6.46	46,700

Table 7: Estimated and classified resources, undiluted, Barry I Main Zone Area project.

16.1.3 Specific gravity

This item was taken from the 2006 NI43-101 Report from M. Ghislain Dechêsnes, P. Geo. from Geostat Systems Int'l Inc.

Specific gravity measurements were done for the 2006 NI43-101 report in order to confirm and validate the previous values used by other companies that have worked on the property. Murgor prepared five samples. Geostat measured the volume and weighed the core samples in order to measure the specific gravity.

The S.G. ranges from 2.79 up to 2.99. The average for the mineralized rock is 2.89.

Measured density - Measured volume				
Hole	Measured	From	To	Vol cm3
MB-05-A	Ore	Casing		2.79
MB-134	Waste	26.09	26.24	2.99
MB-05-A	Ore	14.6	14.8	2.86
MB-134	Waste	28.3	28.47	2.82
MB-134	Waste	21.87	22.02	2.99
Average				2.89

Table 8: 2005 results of the measured specific gravity of the Barry I Main Zone.

A second estimation of the S.G. was realized by adding the cumulative error of ± 10 ml due to the reading error on the volume of the rock piece in the graduated cylinder. 10 ml were removed to the water value read on the cylinder with only water and 10 ml were added to the value read for the volume of water and sample. We added the two possible errors of reading on the cylinder to minimize the possibility of over-estimating the density. In that case, the S.G. ranges from 2.73 up to 2.88. The average for the mineralized rock is 2.79.

Measured density- Measured volume + 20 ml of cumulative error				
Hole		From	To	Vol cm3
MB-05-A	Ore	Casing		2.75
MB-134	Waste	26.09	26.24	2.88
MB-05-A	Ore	14.6	14.8	2.73
MB-134	Waste	28.3	28.47	2.74
MB-134	Waste	21.87	22.02	2.88
Average				2.79

Table 9: 2005 results of the measured specific gravity of the Barry I Main Zone Area with cumulative errors.

The S.G. used for the calculation of the resources for the Barry I Main Zone Area was fixed to 2.8.

The following picture shows Geostat personal measuring the sample density.

Figure 12: Measure of the volume of the rock samples for determination of the density.

Environment

The environmental aspects are not addressed in this report.

Waste dump and proposed site configuration

The aspects of waste dump and proposed site configuration aspects are not addressed in this report

16.2 Bachelor Lake Property

16.2.1 Previous resource estimation by InnovExplo

The 2005 NI 43-101 report from InnovExplo shows resources for geological zones Main, A, B, C, AW and BW, see table 16.2.7.

For Indicated and Inferred resources, the geological zones were meshed in 3D. Composites of 2.5 ft calculated in the geological zones were cut and used to calculate a block model in the meshes. Resources were calculated with an arbitrary 0.1 opt Au (oz per short ton) cut-off.

Measured Resources were evaluated from underground mapping and faces sampling results. The measured resources represent 18% of the gold (approx. 54,000 oz Au). We looked at the level maps that were used for this calculation. We believe that the evaluation of these resources is sufficiently accurate and usable for this study.

16.2.2 Indicated and Inferred resources by Geostat for stope designs

In order to be able to calculate stopes in the indicated and inferred resource, all the zones listed in the NI-43-101 report from InnovExplo were recalculated for this report on East-West longitudinals. The following procedure was used:

To get an equivalent to the 2.5 ft composite cutting by InnovExplo, we did some cutting directly on the assays. The details are explained later in this Title (Item 19) .

For every vein:

- Geological intervals from InnovExplo were used.
- Two or three intervals in one hole were summed to make one unique intersection.
- Thicknesses of geological intercepts were calculated for the longitudinal projection. (Horizontal thicknesses in the North-South direction were calculated).
- A two dimensional longitudinal block model was then interpolated from these intercepts.

For the Main and B zones, the limit between the Bachelor and Hewfran were digitized from InnovExplo longitudinals.

16.2.3 Interpolation method and parameters

The interpolation of thickness and grade were made separately on 6 different longitudinals for zones A, B, C, Main, AW and BW. The method used for interpolation was the one used by InnovExplo and was the inverse distance squared method.

The tables used as geological intersections came from InovExplo's Gemcom files: ddh_LONGZONEP (Main), ddh_FROMSOLIDA, ddh_LONGZONEB, ddh_FROMSOLIDAW, ddh_LONGZONEBW, ddh_FROMSOLIDC. Spot checks on these intersections showed that they were adequate. There are no intersections overlaps so no gold was counted twice.

For calculations of the Main zone, blocks in the measured outlines were not counted in the results to respect the use of tonnages and grades used in InnovExplo's Report.

Block model settings were:
Center of the first block (zones A, C, AW, BW): X=-2500ft Y=7500ft
Center of the first block (zones B, Main): X=-1250ft Y=7500ft
Size of blocks for all zones: 10ft x 10ft

A first interpolation was made with an anisotropic search 2D ellipsoid of 150ft x 95ft with an orientation of 145° with 2 to 6 composites and then with another ellipsoid of 65ft x 40ft with an orientation of 145° with 1 to 6 composites.

16.2.4 Density used

The density used by InnovExplo was 2.755. We used a rounded value of 2.75. The difference has no significance.

16.2.5 Details of the cutting method used

The previous NI-43-101 report cut the 2.5 ft composites at 1.5 opt Au in the Main vein and 1.0 opt Au in the veins A and B.

Because we did our estimates on longitudinal sections, we had to do an equivalent:

Cut for assays under 2.5 ft long: The equivalent quantity of gold represented by a composite of 1.5 opt over 2.5 ft is 3.75 opt*ft. For every assay interval in the Main vein that has over 3.75 opt*ft, we cut the Au to the value required to meet the 3.75 opt*ft.

Cut for assays over 2.5 ft long: 1.5 opt for assays in the Main zone and 1 opt for assays in other zones. This method is slightly more conservative then the one used by InnovExplo. This cutting is found adequate.

Hole Name	From	To	length	AUOPT_MOY	Q	Vein	AuCUT
12-106	259.35	261.81	2.46	1.19	2.93	B	1.016
12-112	325.95	327.59	1.64	2.4004	3.94	Main	2.287
12-40	150.59	154.95	4.36	0.8225	3.59	B	No Cut
12-40	242.19	246.06	3.87	0.8837	3.42	Main	No Cut
12-44	457.71	461.45	3.74	0.7787	2.91	Main	No Cut
12-46	219.78	224.02	4.24	0.9158	3.88	B	No Cut
12-46	214.9	219.78	4.88	0.7029	3.43	B	No Cut
12-46	197.11	200.36	3.25	0.7933	2.58	B	No Cut
12-48	265.75	269.69	3.94	0.7787	3.07	B	No Cut
12-48	269.69	273.75	4.06	0.6912	2.81	B	No Cut
12-50	196.85	201.77	4.92	0.5483	2.70	B	No Cut
12-50	201.77	203.58	1.81	1.4408	2.61	B	1.381
12-59	373.2	377.62	4.42	0.8312	3.67	B	No Cut
19501-12	1281	1285.6	4.6	0.734	3.38	Main	No Cut
BLM-11-34	128.2	132	3.8	0.811	3.08	Main	No Cut
BLM-12-04	788.9	790.1	1.2	2.615	3.14	Main	No Cut
BLM-12-04	764.5	766.2	1.7	1.668	2.84	Main	No Cut
BLM-12-04	715	719	4	0.64	2.56	Main	No Cut
BLM-12-22	163	167	4	0.998	3.99	B	No Cut
BLM-12-22	126.8	130	3.2	1.21	3.87	?	1.000
BLM-12-26	358.2	360.5	2.3	1.584	3.64	B	1.087
HU-6-24	417.49	420.01	2.52	3.432	8.65	?	1.000
HU-8-41	208.01	212.01	4	0.928	3.71	Main	No Cut
S-95-08	1919.6	1924.2	4.59	0.631	2.90	?	No Cut

Table 16.2.1: Cut values in the DDH database

16.2.6 Details of the intervals used for the calculations

The following tables are the geological intervals used for the estimation for each zone. The true horizontal thickness has been calculated according to the orientation of each of the 3 most important zones.

The true horizontal thickness is obtained by multiplying the horizontal North South thickness by the following factors:
B Zone dip vector : 182° azimuth / 78° dip / factor = 1.00
Main Zone dip vector : 171° azimuth / 67° dip / factor = 0.99
AW Zone dip vector : 144° azimuth / 72° dip / factor = 0.81

Hole Name	From	To
12-100	168.27	187.47
12-102	147.59	158.25
12-104	205.36	217.5
12-106	255.9	264.92
12-108	330.33	349.85
12-110	330.71	348.57
12-112	186.95	203.18
12-114	394.61	443.41
12-38	109.83	119.99
12-39	305.74	312.11
12-40	142.18	165.6
12-41	319.63	328.31
12-42	163.55	177.9
12-43	354.06	367.61
12-44	240.38	274.63
12-45	336.3	347.94
12-46	194.71	224.01
12-47	338.74	348.64
12-48	261.48	278.11
12-49	347.45	353.36
12-50	164.82	203.53
12-51	363.6	369.96
12-52	271.61	285.35
12-53	319.15	327.97
12-53	386.62	405.96
12-54	194.04	223.56
12-55	319.61	357.54
12-55	369.83	396.57
12-56	104.32	122.38
12-57	361.5	380.04
12-57	325.91	341.99
12-58	360.24	379.55
12-59	370.1	396.89
12-59	319.77	357.62
12-60	359.57	369.75
12-61	322.79	346.56
12-61	384.24	399.37
12-62	346.58	353.97
12-63	377.59	395.97
12-64	199.32	240.79
12-65	421.73	436.5
12-66	148.42	156.46
12-67	533.32	540.76
12-68	113.18	152.55
12-69	384.12	414.29
12-69	435.12	447.43
12-70	190.14	196.02
12-71	486.31	495.39
12-72	294.29	300.2
12-73	558.19	568.69
12-74	329.2	336.57
12-75	552.61	559.89
12-76	234.82	244.71
12-77	527.85	535.35
12-78	224.49	233.83
12-79	1032.32	1044.78
12-80	228.8	239.63
12-83	356.05	382.28
12-84	347.07	347.23
12-85	306.08	311.17
12-86	238.98	276.52
12-87	333.64	353.98
12-87	364.82	379.06
12-88	344.46	359.52
12-89	351.18	358.5
12-90	165.55	176.39
12-91	752.35	760.84
12-92	209.78	221.42
12-93	366.6	392.19
12-95	541.28	566.55
12-98	534.73	574.09
19501-12	952.98	968
19501-29	1633.5	1643
19501-37	843.01	856.01
BLM-11-08	38.82	44.91
BLM-11-09	28.27	34.27
BLM-11-33	221.9	226.7
BLM-11-34	0	5.36
BLM-11-36	0	5.95
BLM-11A-24	182.79	195.6
BLM-11A-25	84.95	103.46
BLM-11A-26	32.84	47.7
BLM-12-04	833.58	852.01
BLM-12-07	1079.51	1090.25
BLM-12-09	373.18	379.37
BLM-12-11	98.7	141
BLM-12-13	280	291.3
BLM-12-14	108	118.8
BLM-12-15	186	194
BLM-12-16	350.93	378.42
BLM-12-17	154.59	155.49
BLM-12-18	182.99	204.79
BLM-12-18	251.98	265.17
BLM-12-19	240.06	249.99
BLM-12-20	348.29	358.99
BLM-12-21	340.4	347.58
BLM-12-22	153.56	188.96
BLM-12-22	200.96	236.96
BLM-12-23	194.97	208.97
BLM-12-25	318.8	324.5
BLM-12-26	342.98	388.78
BLM-12-27	400	413.29
BLM-12-28	430.49	443.79
BLM-12-29	452.68	461.48
BLM-12-30	310.98	317.58
BLM-12-31	320.29	325.38
BLM-12-32	389.89	449.88
BLM-12-33	362.76	369.73
BLM-12-37	485.5	495.79
HU-10-2	32.51	43.01
HU-10-3	20.41	25.96
HU-8-12	47.74	53.51
HU-8-13	58.94	65.49
HU-8-14	72.01	83.01
HU-8-15	42.85	48.22
HU-8-16	42.9	49.02
HU-8-17	50	57.12
HU-8-18	112.48	127.98
HU-8-19	31.32	36.18
HU-8-20	54.5	63.05
HU-8-21	45.1	53.08
HU-8-22	64.01	71
HU-8-23	86.41	98.63
HU-8-24	54.32	60.74
HU-8-25	72.01	79.83
HU-8-26	99.97	112.37
HU-8-27	58.29	61.14
HU-8-28	55.01	62
HU-8-29	49.56	55.65
HU-8-30	33.74	38.52
HU-8-31	38.19	43.38
HU-8-32	33.02	38.24
HU-8-33	34.16	40.22
HU-8-34	41.43	49.32
HU-8-35	41.99	50.98
HU-8-37	33.69	38.46
HU-8-38	64.05	71.65
HU-8-39	66.96	73.97
HU-8-40	53	59.36
HU-8-41	35.35	42.06
HU-8-42	39.45	47.41
HU-8-7	124.94	130.17
HU-9-1	106	112.01
HU-9-2	217.91	234.24
HU-9-3	112.99	120.01
S-95-08	1938.97	1948.81

Table 16.2.2: Longitudinal intercepts for Zone B

Hole Name	From	To
BLM-11-01	350.98	359.98
BLM-11-02	48.36	61.5
BLM-11-03	124	132.27
BLM-11-04	34.6	41.95
BLM-11-05	148	156
BLM-11-06	137.09	142.3
BLM-11-07	163.78	176
BLM-11-10	172.86	192.29
BLM-11-11	209.19	232.52
BLM-11-11	235.99	264.1
BLM-11-12	172.44	186.26
BLM-11-13	215	224.8
BLM-11-14	263.1	350.15
BLM-11-15	242	254.5
BLM-11-16	120	125.6
BLM-11-17	124.99	130.89
BLM-11-21	0	2.22
BLM-11-24	0	2.76
BLM-11-27	16.41	23.29
BLM-11-28	13.71	19.45
BLM-11-29	1.76	10.44
BLM-11-32	0	6.7
BLM-11-34	116	136
BLM-11-35	198.46	214.95
BLM-11-36	230.95	241.33
BLM-11-37	92.39	99.84
BLM-11-38	210.44	237.03
BLM-11-38	239.31	270.88
BLM-12-01	499.98	516.48
BLM-12-02	640.98	650.98
BLM-12-03	537.57	545.86
BLM-12-03	546.7	549.94
BLM-12-04	709.99	790.08
BLM-12-06	748.96	762.8
BLM-12-07	994.46	1004.84
BLM-12-08	672.51	679.92
BLM-12-09	232.99	271.79
BLM-12-10	721.7	737.85
BLM-12-12	7.98	24.98
BLM-12-13	351.99	365
BLM-12-14	231.8	240.57
BLM-12-15	258.4	265
BLM-12-15	275.1	285
BLM-12-15	303.5	310
BLM-12-16	387.81	398.92
BLM-12-17	250	258.08
BLM-12-18	463.47	473.38
BLM-12-19	343.08	353.63
BLM-12-22	359.96	368.16
BLM-12-23	294.4	298.54
BLM-12-25	418.49	425.1
BLM-12-26	414.28	420.51
BLM-12-27	489.8	496.59
BLM-12-28	562.54	567.59
BLM-12-30	574.16	585.8
BLM-12-31	555.95	594.07
BLM-12-32	534.06	542.5
BLM-12-33	570.17	585.16
BLM-12-36	0	1.95
BLM-12-37	628.95	641.65
12-38	227.36	239.5
12-39	603.51	613.34
12-40	231.78	259.88
12-41	670.15	683.92
12-42	323.89	340.97
12-44	452.28	468.8
12-45	677.64	687.17
12-46	251.2	256.13
12-46	273.77	280.01
12-46	302.22	313.4
12-48	282.47	294.46
12-49	554.79	564.2
12-49	572.64	581.5
12-50	341.16	353.61
12-51	577.99	584.35
12-52	527.15	541.37
12-53	436.61	471.21
12-54	367.35	377.74
12-55	409.19	414.94
12-56	226.36	242.43
12-57	403.8	416.42
12-58	552.06	560.98
12-59	482.09	487.68
12-60	373.03	387.62
12-61	513.71	521.8
12-62	441.62	452.43
12-63	488.64	499.14
12-64	298.52	308.48
12-65	546.72	553.78
12-66	267.01	283.58
12-67	619.25	627.94
12-68	218.22	234.56
12-69	569.86	576.6
12-70	283.44	284.16
12-72	300.2	308.07
12-73	735.03	744.88
12-74	365.64	378.44
12-75	697.6	706.5
12-76	313.44	328.03
12-78	317.41	326.6
12-80	400.04	413.99
12-79	1158.21	1166.34
12-82	445.55	458.18
12-88	534.25	550.13
12-90	339.08	347.61
12-84	349.73	360.55
12-86	416.29	425.96
12-92	464.53	475.83
12-98	679.06	705.34
12-102	229.93	259.77
12-104	442.88	450.47
12-85	528.52	534.27
12-106	270.34	279.53
12-110	513.42	528.17
12-83	524	532.88
12-87	476.83	482.24
12-89	573.21	589.77
12-112	321.3	341.63
12-114	469.88	519.23
12-93	459.78	467.98
12-95	703.68	721.72
19501-12	1269.69	1285.6
19501-16	1131.5	1150.98
19501-26	928.51	942.49
19501-34	1033.19	1040.07
19501-35	974.47	985.99
19501-37	1131	1145.01
19501-45	917.99	924.74
19501-50	947.01	952.99
HU-10-1	15.84	24.64
HU-10-2	144.85	152.22
HU-10-3	27.49	47.01
HU-4-10	345.39	347.99
HU-4-11	272.98	283.08
HU-4-19	275.98	281.25
HU-4-20	258.08	264.05
HU-4-23	316.96	325.46
HU-4-24	402.79	427
HU-4-8	287.99	291.99
HU-6-2	0	5.51
HU-8-1	10.11	29.76
HU-8-12	175.97	181.98
HU-8-13	197.01	200.98
HU-8-14	206	212.01
HU-8-15	174.99	180.99
HU-8-16	231.98	237.98
HU-8-18	306	329
HU-8-19	179.96	187.96
HU-8-2	14.3	50.2
HU-8-20	262.98	268.79
HU-8-21	191.99	196
HU-8-22	221.96	229.43
HU-8-23	247.99	260
HU-8-24	198	206
HU-8-25	195.99	206.49
HU-8-26	310.7	325.59
HU-8-27	226.5	239
HU-8-28	183.99	189.01
HU-8-29	213.44	219.25
HU-8-3	8.91	27.01
HU-8-30	231	255.51
HU-8-31	275.48	286.39
HU-8-32	178.4	187
HU-8-33	225.97	232.5
HU-8-34	180.97	187.96
HU-8-35	243.5	256
HU-8-37	237	268.49
HU-8-38	249.47	256.46
HU-8-4	2.99	19
HU-8-40	214.01	230.51
HU-8-41	207.99	219.99
HU-8-42	224.01	234.5
HU-8-5	4	36.52
HU-8-6	2	8.2
HU-8-8	2.99	8.99
HU-8-9	3.15	8.74
HU-9-2	133.26	145.02
HU-9-2	170.39	179.64
S-95-08	2024.27	2032.47
11A-1	363.18	368.68
11A-13	174.99	183.99
11A-21	0	2.9
11A-23	0	6.3
11A-36	0	13.17
11A-4	362.51	364.98
82-11	1863.76	1873.34
84-15	1696.35	1705.09
84-17	1391.67	1448.32

Table 16.2.3: Longitudinal intercepts for Zone Main

Hole Name	From	To
19501-18	512.5	521
19501-19	1214.98	1226.5
19501-20	836.51	847.99
19501-23	436.51	447.01
19501-26	928.51	942.49
19501-32	703.98	713.46
19501-33	708.97	717.47
19501-34	1012	1022
19501-36	331.99	337.01
19501-39	1356.98	1366.98
19501-40	1247	1252.98
19501-41	986.94	1002.92
19501-44	1041.75	1060.36
19501-45	875.53	884.84
19501-46	872	881.49
19501-48	628.98	633.97
19501-51	911.21	943.19
19501-52	693.3	700.48
19501-53	912.01	929.99
19501-55	749.02	763
19501-58	975.51	1062.01
19501-59	1005.26	1026.83
19501-60	1115.99	1131.98
D-14	356.02	362.56
HU-4-12	356.18	363.02
HU-4-2	475.95	479.61
HU-6-10	433.99	447
HU-6-11	547.01	723.98
HU-6-12	238.65	246.17
HU-6-14	218.48	224.98
HU-6-15	237.99	246
HU-6-16	291.99	300.97
HU-6-17	210.01	216.01
HU-6-18	220.01	227.99
HU-6-19	323.87	333.06
HU-6-20	230.19	236.94
HU-6-21	221	229.99
HU-6-22	217.14	224.23
HU-6-23	245.6	265.41
HU-6-24	383.5	430.18
HU-6-25	319.8	329.06
HU-6-26	529	537.44
HU-6-27	221.12	229.59
HU-6-28	462.01	474.61
HU-6-29	252.49	261.52
HU-6-30	327.99	373
HU-6-31	543.5	550.98
HU-6-32	248.49	256
HU-6-33	363.46	376.36
HU-6-34	492.26	504
HU-6-35	489.99	500.97
HU-6-36	437	451.98
HU-6-37	435.01	442.79
HU-6-38	539.99	549.51
HU-6-39	310.8	319.6
HU-6-4	92.49	99.02
HU-6-41	283.01	287.99
HU-6-42	336.83	349.18
HU-6-43	249.99	266.99
HU-6-5	196.48	212
HU-6-6	241.05	257.6
HU-6-7	202.98	208.98
HU-6-8	271.78	277.99
HU-6-9	381	396
HU-8-10	0.01	6.42
HU-8-47	742.69	794.95
19501-59	1082.29	1114.99
HU-6-8	308.99	319.74

Table 16.2.4 Longitudinal intercepts for Zone AW

16.2.7 Stopes design and detailled results

On longitudinals, it was possible to design shapes for stopes. This design has been done on the major zones (Main, B and AW) and not on the smaller zones because smaller zones cannot be mined at comparable costs. Furthermore, they would not significantly add value to the project at this stage. The next figures shows stopes designed on the longitudinals.

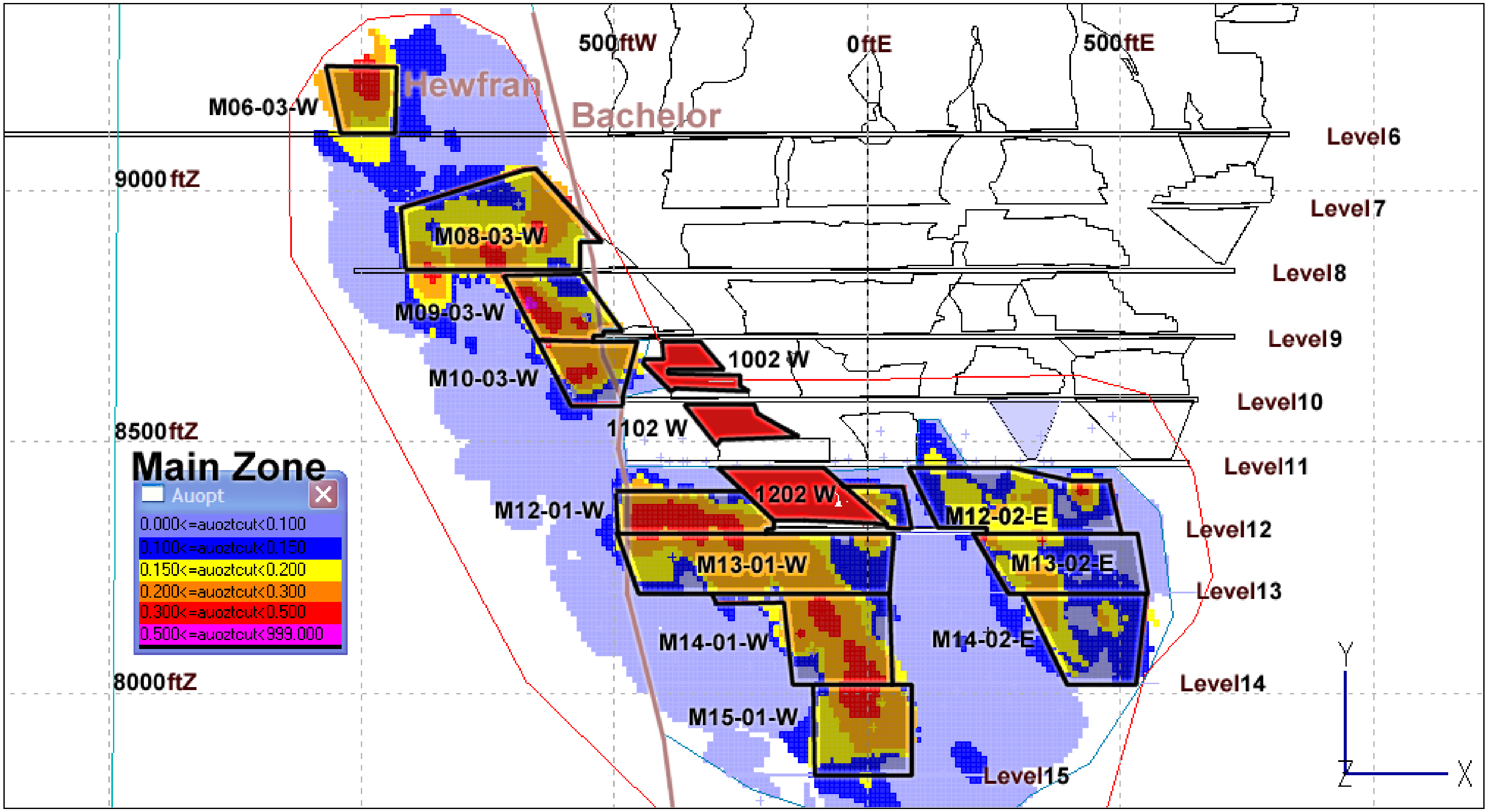


Figure 16.2.1: Longitudinal for Zone Main – Color for Au (opt) with stopes designs

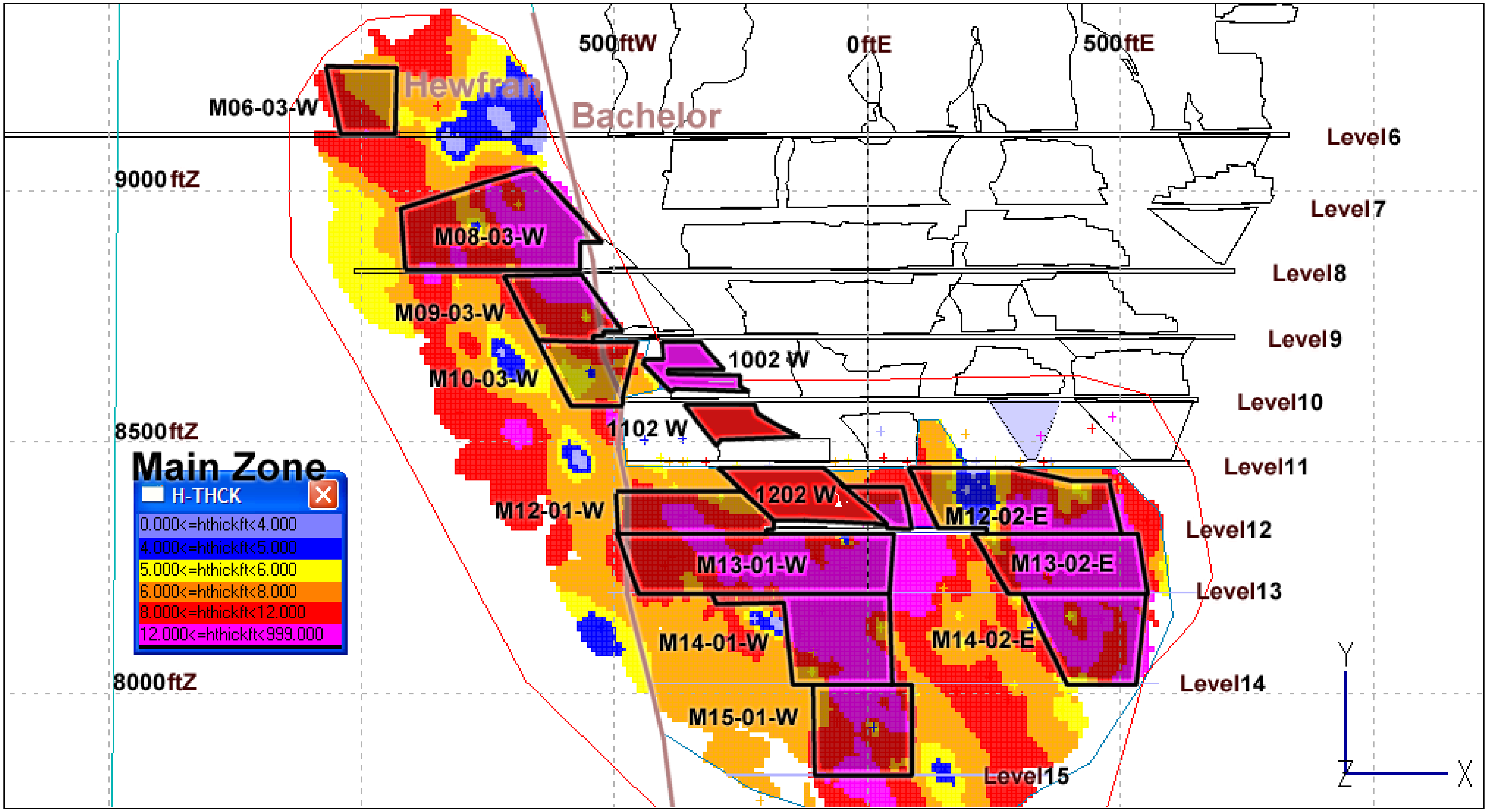


Figure 16.2.2: Longitudinal for Zone Main – Color for Horizontal NS thickness (ft) with stopes designs

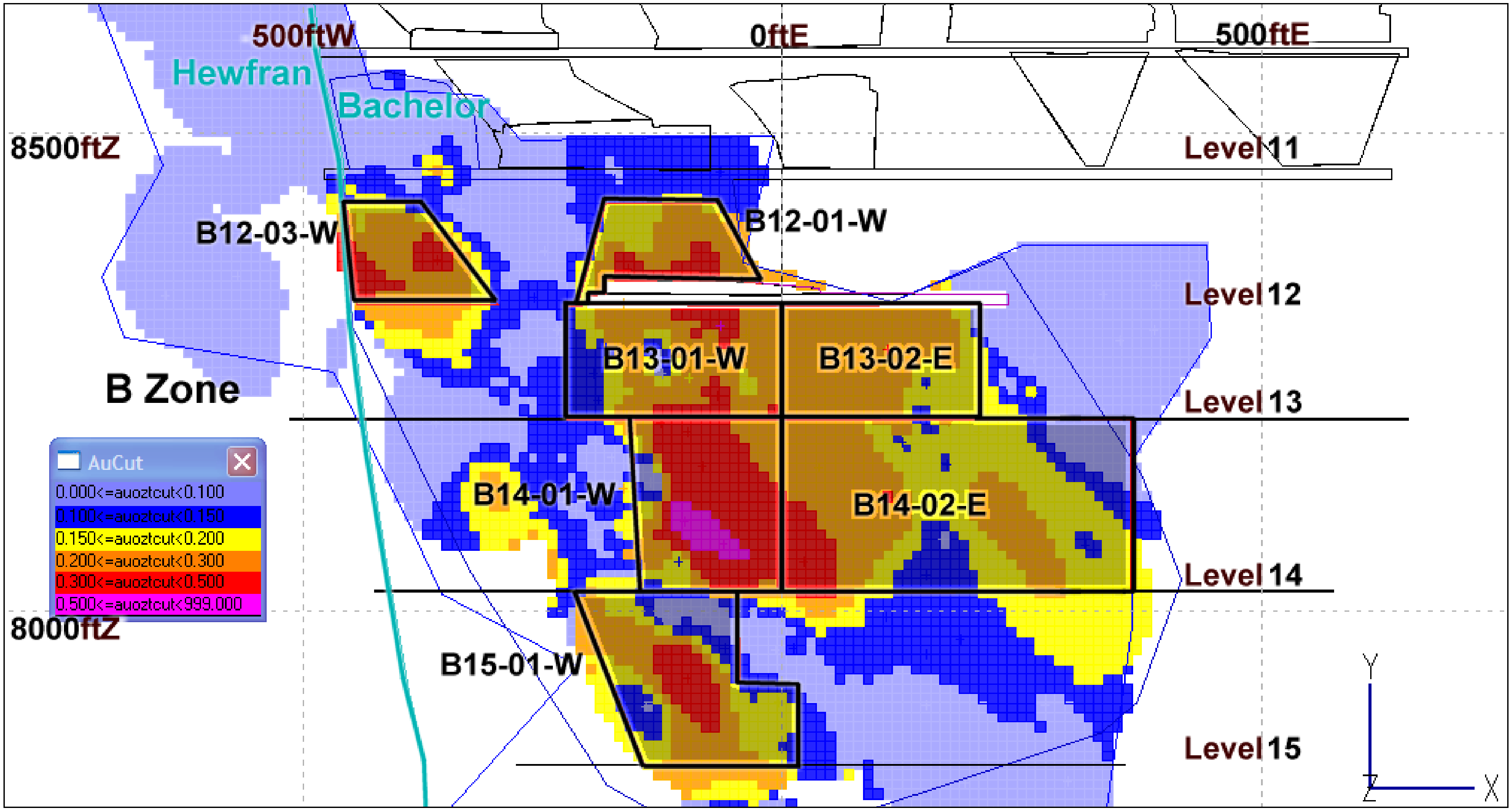


Figure 16.2.3: Longitudinal for Zone B – Color for Au (opt) with stopes designs

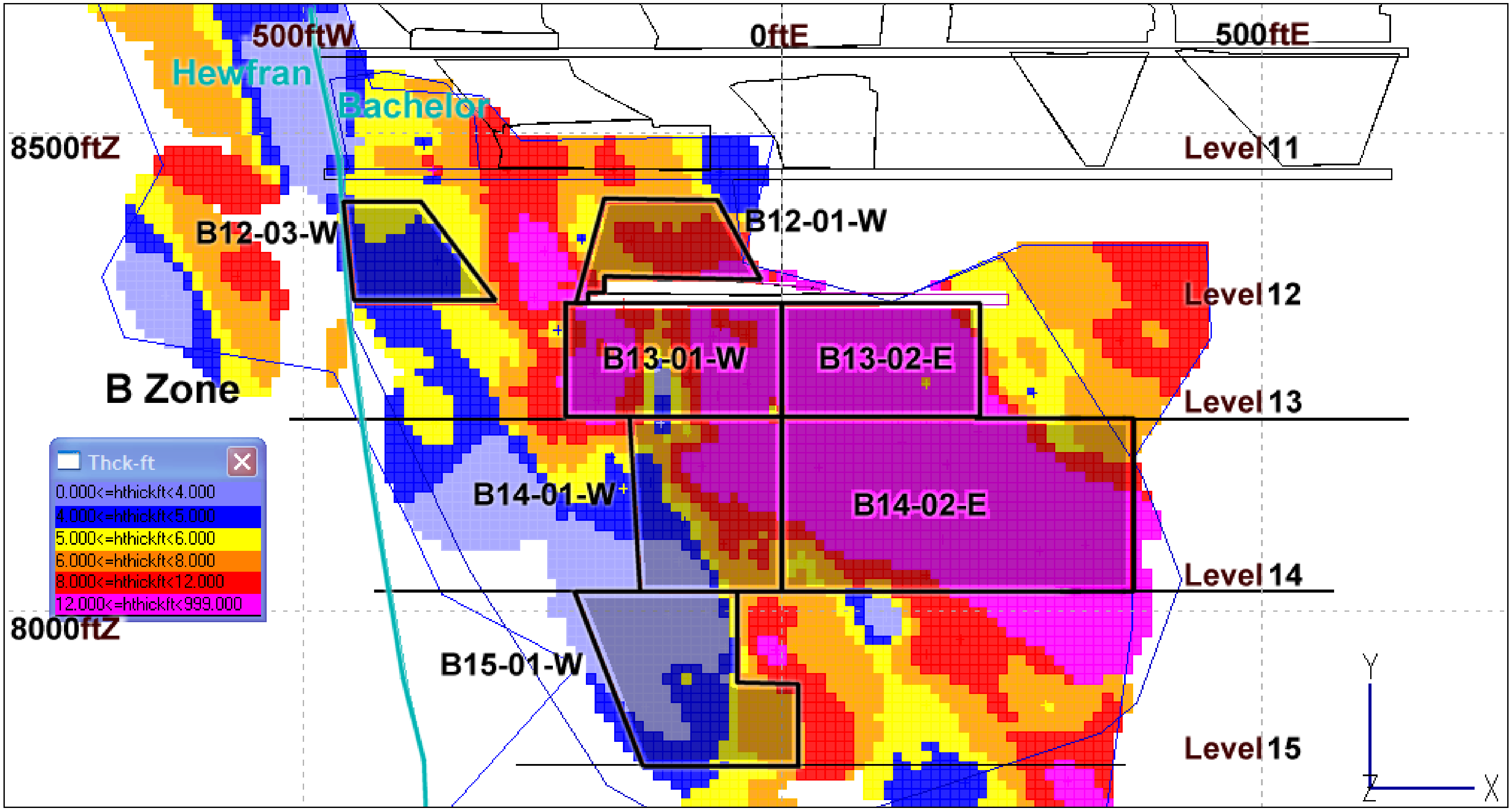


Figure 16.2.4: Longitudinal for Zone B – Color for Horizontal NS thickness (ft) with stopes designs

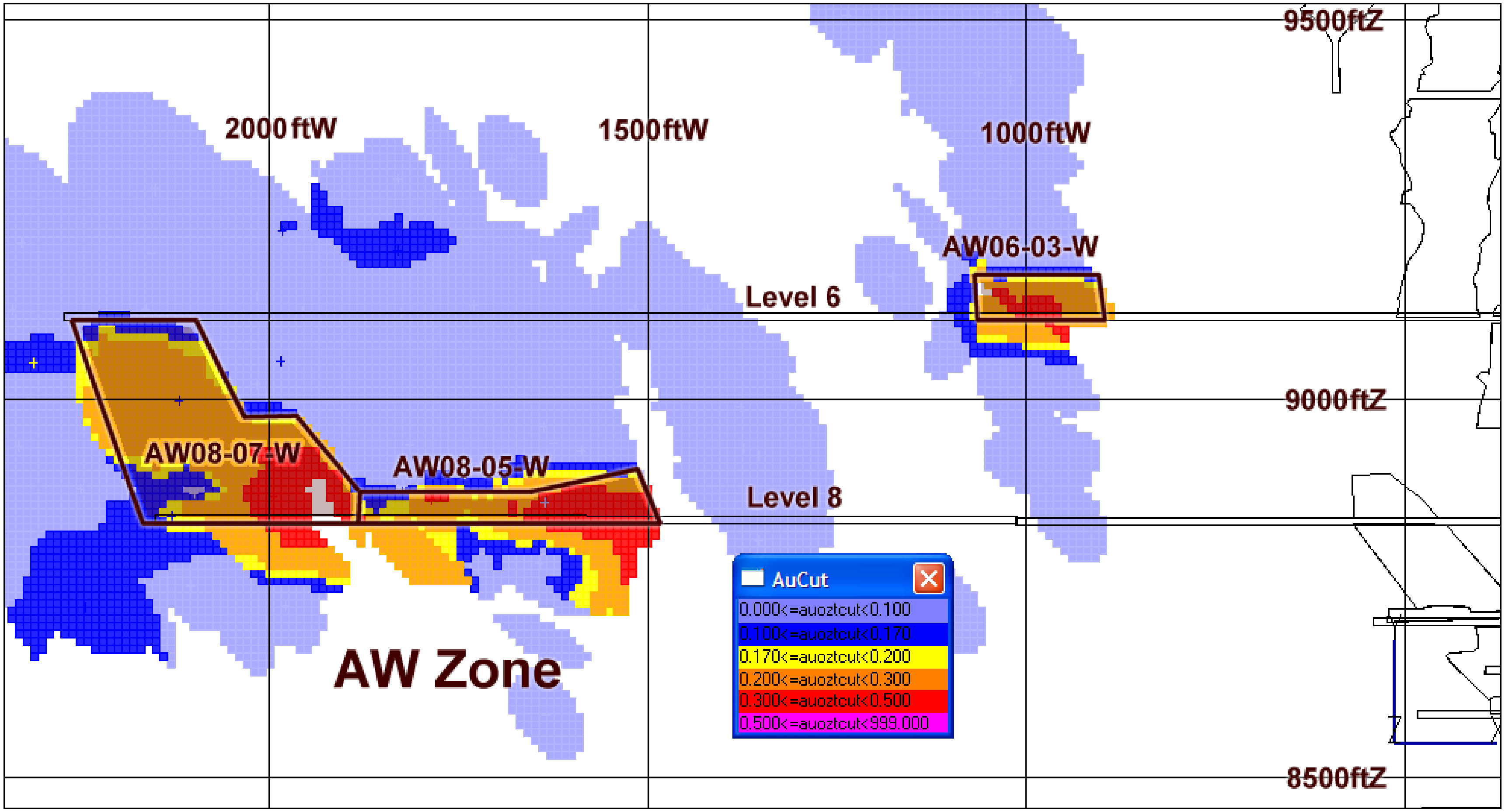


Figure 16.2.5: Longitudinal for Zone AW – Color for Au (opt) with stopes designs

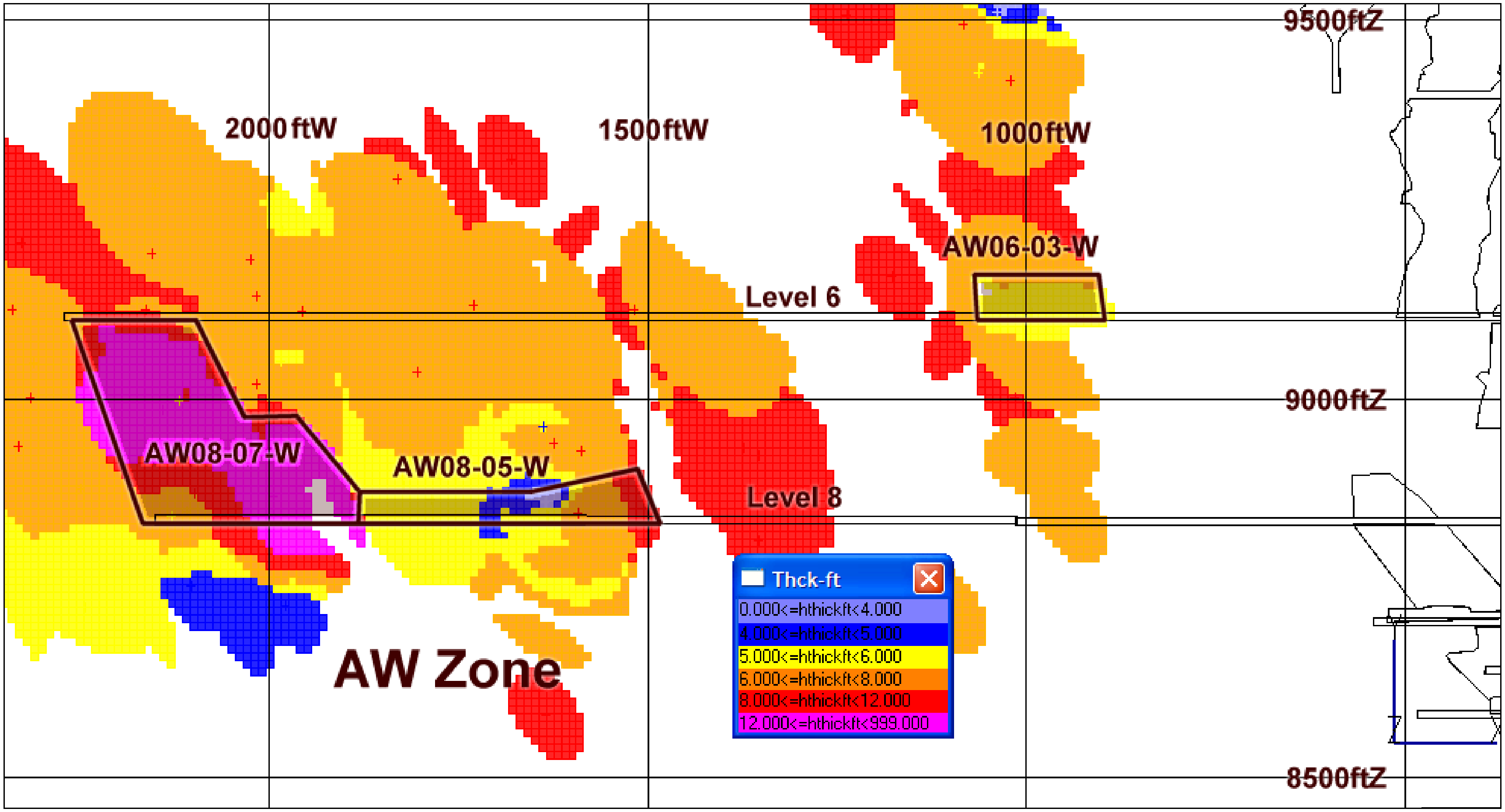


Figure 16.2.6: Longitudinal for Zone AW – Color for Horizontal NS thickness (ft) with stopes designs

16.2.8 Geostat total resources

The following table is showing the undiluted total resources that have been “retained for mining” in the present study.

UNDILUTED STOPE RESOURCES - FROM CONTACT TO CONTACT

Ref	Stope Name	Thickness (ft)	Au opt	Volume ft3	T/m3	Tonage (short tons)	Au (oz)	Category
1	M06-03W	8.2	0.24	131,212	2.75	11,239	2,699	Ind. & Inf.
2	M08-03W	13.3	0.20	753,507	2.75	64,544	12,973	Ind. & Inf.
3	M09-03W	11.7	0.22	240,832	2.75	20,629	4,464	Ind. & Inf.
4	M10-03W	7.2	0.21	139,120	2.75	11,917	2,529	Ind. & Inf.
5	M12-01W	10.2	0.25	314,468	2.75	26,937	6,635	Ind. & Inf.
6	M12-04E	9.2	0.15	396,070	2.75	33,927	4,930	Ind. & Inf.
7	M13-01W	13.5	0.20	798,713	2.75	68,417	13,458	Ind. & Inf.
8	M13-02E	17.1	0.13	614,622	2.75	52,648	6,713	Ind. & Inf.
9	M14-01W	16.3	0.25	634,521	2.75	54,352	13,441	Ind. & Inf.
10	M14-02E	13.2	0.15	456,390	2.75	39,094	5,758	Ind. & Inf.
11	M15-01W	13.2	0.24	460,281	2.75	39,427	9,541	Ind. & Inf.
12	8-02-E	8.5	0.36	18,223	2.75	1,561	568	Measured
13	8-03-E	8.5	0.25	90,567	2.75	7,758	1,932	Measured
14	1002W	12.5	0.32	253,910	2.75	21,750	7,000	Measured
15	11-02-E-1	8.3	0.38	85,115	2.75	7,291	2,741	Measured
16	11-02-E-2	7.8	0.22	151,110	2.75	12,944	2,783	Measured
17	1102W	9.7	0.34	146,112	2.75	12,516	4,270	Measured
18	1202W	9.9	0.32	241,150	2.75	20,657	6,630	Measured
	TOTAL Main	**12.8**	**0.202**			**507,607**	**102,435**	
19	B12-01W	8.7	0.25	108,465	2.75	9,291	2,298	Ind. & Inf.
20	B12-03W	4.8	0.26	54,208	2.75	4,643	1,193	Ind. & Inf.
21	B13-01W	12.8	0.23	345,237	2.75	29,572	6,781	Ind. & Inf.
22	B13-02E	27.1	0.22	596,232	2.75	51,072	11,113	Ind. & Inf.
23	B14-01W	9.1	0.36	250,547	2.75	21,461	7,713	Ind. & Inf.
24	B14-02E	26.2	0.20	1,737,160	2.75	148,802	29,091	Ind. & Inf.
25	B15-01W	4.2	0.20	129,232	2.75	11,070	2,255	Ind. & Inf.
	TOTAL B	**21.8**	**0.219**			**275,913**	**60,444**	
26	AW06-03-W	6.1	0.190	81,718	2.75	7,000	1,400	Ind. & Inf.
27	AW08-05-W	16.3	0.220	175,110	2.75	15,000	3,200	Ind. & Inf.
28	AW08-07-W	14.7	0.200	1,330,836	2.75	114,000	22,400	Ind. & Inf.
	Total AW	**14.70**	**0.199**			**136,000**	**27,000**	
	TOTAL: B+Main+AW		**0.212**			**919,520**	**189,878**	

Table 16.2.5 Total undiluted resources for the economic study

The total undiluted resources “retained for mining” of 919, 520 tons as presented in the above table is obtained after selecting parts of the resources that we assumed to be easily accessible and that can be exploited by the three proposed mining methods mentioned in the Title 17.2. This “retained for mining” total is containing 835,043 tons of indicated and inferred resources plus 84,477 tons of measured resources coming from a total of 208,300 tons, as published by InnovExplo. This low percentage of the measured resources “retained for mining” resources is explained below.

16.2.8.1 Measured resources of the Main Zone

Some measured resources are reported into the previous workings. These resources are mainly on the West side of the mine between level 12th and level 8th. Some of these measured resources were taken out of the mining calculation for the following reasons. On most cases no evaluation is available as to the cost and the possibilities of rehabilitating the level accesses. For example, in the case of the 101 east there would be the need of rehabilitating over 400 ft of cross cut. On some of the level plans that we obtained during the site visit, it is obvious that the mining method used at that time of the first operation (1982-1989) was the TDB (Take Down Back) with timbered backs and ore chutes that created dangerous conditions unacceptable under today's recognized safety standards.

On the first level the west stope does not meet the mining cut off grade. On the second level the 201 E#2 as enough gold content but the tonnage of just over 3,000 tons would have to pay the stope preparation and the loading pocket or ore pass rehabilitation, the stope resources are deleted. The same applies for the 3rd and 4th levels. As for the Eastern part of the mine, from level 6 down to the 11th the rehabilitation, we assumed that a portion of these resources can be mined out. A total of 84,477 st at 0.306 opt is included in seven (7) stopes as the "retained for mining" resources and shown in the table 16.2.6.

There is an obligation of placing a ventilation-emergency raise between the 6th and the 12th level. Placing this raise on the east side would connect with the actual ventilation on the 6th level and enhance the possibility of mining the measured resources in that area. Nonetheless, the raise could also be placed on the Western portion of the mine from the 12th to the 6th. There seems to be no doubt about the advantages of placing this raise on the Western portion. The raise would be placed near the 500 W section to give access to the Bachelor Main West and the Hewfran Main East.

From short sub levels from the ventilation raise both the measured resources of the Main lens on the Bachelor between the 12th and the 9th levels and on the Hewfran between the10th and 7th levels would be accessible to mining. The mucking raises required to mine the Bachelor and the Hewfran Main lens would be converted into ore and waste passes at the end of the mining cycle and serves for any existing ore material or newly discovered ore above the 6th level or on any level above the 11th.

16.2.9. Total "retained for mining" diluted resources

Recovery and dilution

It is assumed that 10% of the estimated ore will be left in place, either as pillars or losses, a dilution of 10% is added to the remaining tonnage at a grade of 0.03 opt as it is shown in the following table.

Estimation of the Bachelor Property "retained for mining" Resources of all Categories

Stope Name	Thickness (ft)*	Grade opt	Density	Total tons (st)	Total ounces in place (oz)	10% losses and pillars (st)	10% losses and pillars (oz)	Tons before dilution (st)	Ounces before dilution (oz)	Tons of dilution at 10%	Oz of dilution at 0.03 opt	Ounces (oz)	Mill feed diluted tons (st)	Mill feed grade opt	Class
M06-03-W	8.2	0.24	2.75	11,239	2,699	1,124	270	10,115	2,429	1,012	30	2,459	11,127	0.221	II
M08-03-W	13.3	0.20	2.75	64,544	12,973	6,454	1,297	58,090	11,676	5,809	174	11,850	63,899	0.185	II
M09-03-W	11.7	0.22	2.75	20,629	4,464	2,063	446	18,566	4,018	1,857	56	4,073	20,423	0.199	II
M10-03-W	7.2	0.21	2.75	11,917	2,529	1,192	253	10,725	2,276	1,073	32	2,308	11,798	0.196	II
M12-01-W	10.2	0.25	2.75	26,937	6,635	2,694	663	24,243	5,971	2,424	73	6,044	26,667	0.227	II
M12-02-E	9.2	0.15	2.75	33,927	4,930	3,393	493	30,534	4,437	3,053	92	4,528	33,587	0.135	II
M13-01-W	13.5	0.20	2.75	68,417	13,458	6,842	1,346	61,575	12,112	6,157	185	12,297	67,732	0.182	II
M13-02-E	17.1	0.13	2.75	52,648	6,713	5,265	671	47,383	6,041	4,738	142	6,183	52,121	0.119	II
M14-01-W	16.3	0.25	2.75	54,352	13,441	5,435	1,344	48,917	12,097	4,892	147	12,244	53,809	0.228	II
M14-02-E	13.2	0.15	2.75	39,094	5,758	3,909	576	35,184	5,183	3,518	106	5,288	38,703	0.137	II
M15-01-W	13.2	0.24	2.75	39,427	9,541	3,943	954	35,484	8,587	3,548	106	8,694	39,033	0.223	II
8-02-E	8.5	0.36	2.75	1,561	568	156	57	1,405	511	140	4	516	1,545	0.334	Measured
8-03-E	8.5	0.25	2.75	7,758	1,932	776	193	6,982	1,739	698	21	1,760	7,680	0.229	Measured
1002W	12.5	0.320	2.75	21,750	6,960	2,175	696	19,575	6,264	1,958	59	6,323	21,533	0.294	Measured
11-02-E-1	8.3	0.376	2.75	7,291	2,741	729	274	6,562	2,467	656	20	2,487	7,218	0.345	Measured
11-02-E-2	7.8	0.215	2.75	12,944	2,783	1,294	278	11,650	2,505	1,165	35	2,540	12,815	0.198	Measured
1102W	9.7	0.340	2.75	12,516	4,255	1,252	426	11,264	3,830	1,126	34	3,864	12,391	0.312	Measured
1202W	9.9	0.320	2.75	20,657	6,610	2,066	661	18,591	5,949	1,859	56	6,005	20,450	0.294	Measured
TOTAL Main	**13.0**	0.215	2.75	507,607	108,990	50,761	10,899	456,846	98,091	45,685	1,371	99,462	502,531	0.198	
B12-01-W	8.7	0.247	2.75	9,291	2,298	929	230	8,362	2,068	836	25	2,093	9,198	0.228	II
B12-03-W	4.8	0.257	2.75	4,643	1,193	464	119	4,179	1,074	418	13	1,086	4,597	0.236	II
B13-01-W	12.8	0.229	2.75	29,572	6,781	2,957	678	26,615	6,103	2,662	80	6,183	29,277	0.211	II
B13-02-E	27.1	0.218	2.75	51,072	11,113	5,107	1,111	45,965	10,002	4,597	138	10,140	50,562	0.201	II
B14-01-W	9.1	0.359	2.75	21,461	7,713	2,146	771	19,315	6,942	1,932	58	7,000	21,247	0.329	II
B14-02-E	26.2	0.196	2.75	148,802	29,091	14,880	2,909	133,922	26,182	13,392	402	26,584	147,314	0.180	II
B15-01-W	4.2	0.204	2.75	11,070	2,255	1,107	225	9,963	2,029	996	30	2,059	10,959	0.188	II
TOTAL B	**20.8**	0.219	2.75	275,913	60,444	27,591	6,044	248,322	54,399	24,832	745	55,144	273,154	0.202	
TOTAL AW		0.199	2.75	135,611	26,954	13,561	2,695	122,050	24,259	12,205	366	24,625	134,255	0.183	
GRAND TOTAL		**0.214**	**2.75**	**919,131**	**196,388**	**91,913**	**19,639**	**827,218**	**176,749**	**82,722**	**2,482**	**179,231**	**909,940**	**0.197**	

Table 16.2.6: Mill feed tonnage and grade from Bachelor property

Other Zones

The other zones (A, C and BW) may contain additional "retained for mining" resources According to their location, thicknesses and grades, we assumed that half of the A zone could probably be mined, 20% of the C zone and 60% of the BW zone. This would add around 30,000 tons at 0.27 opt. More detailed studies are needed on these zones before they can be added to the "retained for mining" resources.

16.2.10 InnovExplo's resources

The following informations of this item are extracted from the 43-101 InnovExplo Technical Report of December 2005 for the Bachelor Lake Property.

MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

The section described below highlights the results of a new resource estimate done subsequent to the 2005 drilling program.

Historical Mineral Resources estimates are discussed in Appendix IX.

Resource parameters for the 2005 estimation

Geological model

The geological interpretation of the mineralized zones was done on transversal section and on plan views. Results of the interpretation were digitized and linked as 3D solids using Gems software. Wire frame solids were built for the six (6) estimated zones: "Main", "A", "B", "C", "A" West, and "B" West (Fig. 17.1). The area covered by geological interpretation and the resource block model is illustrated in the Figure 17.2.

Several of the pertinent features of the Bachelor geological model and characteristics reviewed are described in the "Mineralization" section of this report and were also discussed in Horvath and Carrier (2005).

The current geological model is re-interpreting the "zones" intersection areas as potential shoots or pockets of high-grade mineralization.
In addition, to the "zones" intersection, wider zones within the individual "zones" often demonstrate an apparent zoning from lower grades at the contacts to high grade cores with similar multi-gram gold grades in the cores.

		InnovExplo - Resources - >0.1 opt								
		indicated+inferred					measured			
		Zone	Thickness (ft)*	Au (oz/short ton)	Tonage (short tons)	Au (oz)	Thickness (ft)*	Au (oz/short ton)	Tonage (short tons)	Au (oz)
Bachelor		A Zone	NA	0.19	79,300	14,970	NA	0.19	4,100	759
		B Zone	NA	0.21	312,000	65,487				
		C Zone	NA	0.21	53,850	11,466				
		Main Zone	NA	0.23	297,200	67,490	NA	0.26	192,000	49,728
		TOTAL Bachelor	**NA**	**0.21**	**742,350**	**159,413**	**NA**	**0.26**	**196,100**	**50,487**
Hewfran	**East**	A Zone	NA	0.27	3,300	879				
		B Zone	NA	0.19	2,800	522				
		Main Zone	NA	0.20	119,700	24,431	NA	0.25	16,200	4,018
	West	AW Zone	NA	0.20	229,200	46,680				
		BW Zone	NA	0.16	87,800	13,796				
		TOTAL Hewfran	**NA**	**0.19**	**442,800**	**86,308**	**NA**	**0.25**	**16,200**	**4,018**
BACHELOR & HEWFRAN TOTAL			**NA**	**0.21**	**1,185,150**	**245,721**	**NA**	**0.26**	**212,300**	**54,504**
			Thickness (ft)*	Au (oz/short ton)	Tonage (short tons)	Au (oz)				
TOTAL MEASURED+INDICATED+INFERRED			**NA**	**0.21**	**1,397,450**	**300,225**				

NA: Not Available
PA: Partly available

Table 16.2.7 Resources from InnovExplo

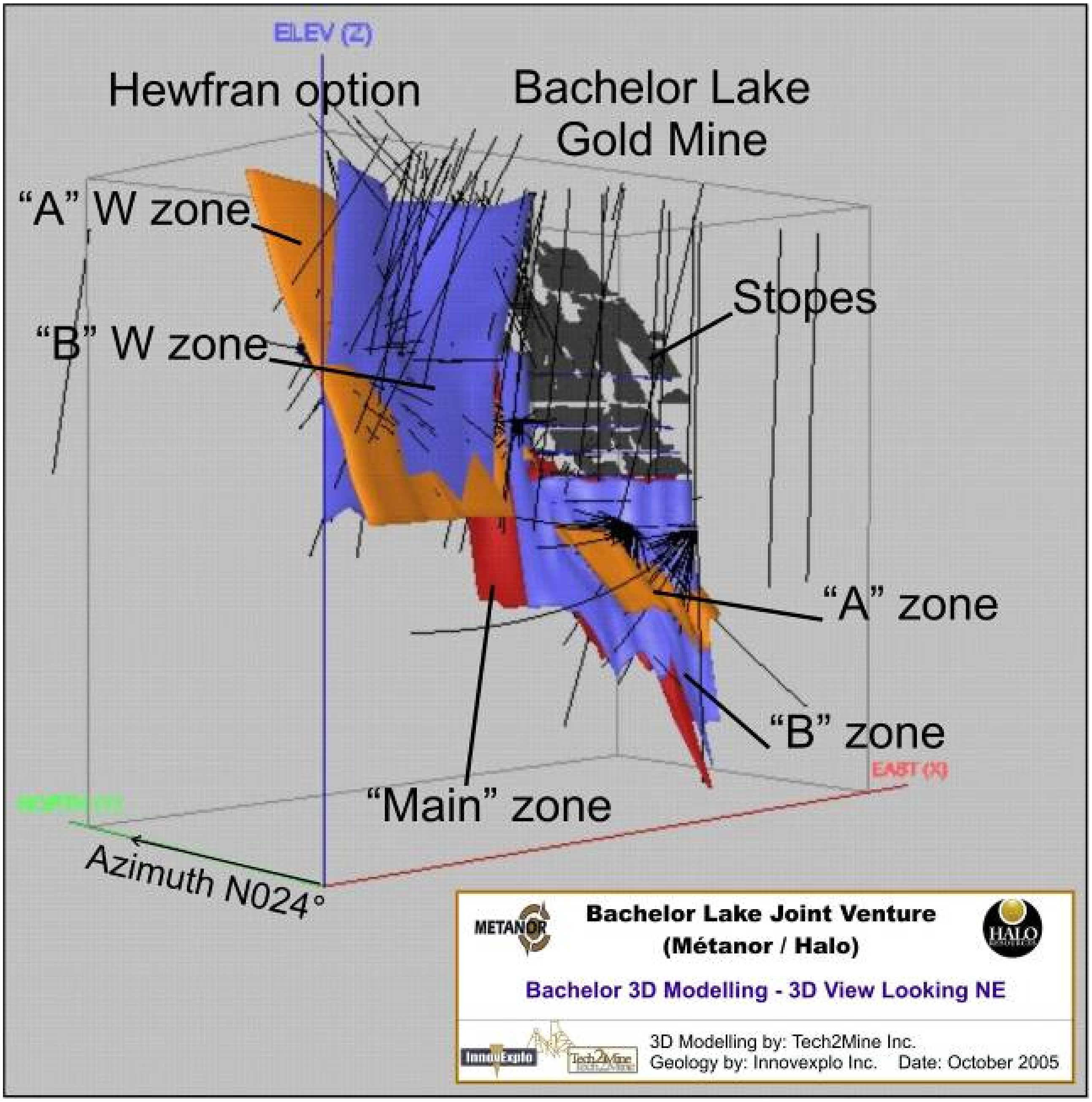


Figure 17.1 - Wire frame model used for the 2005 resource estimate

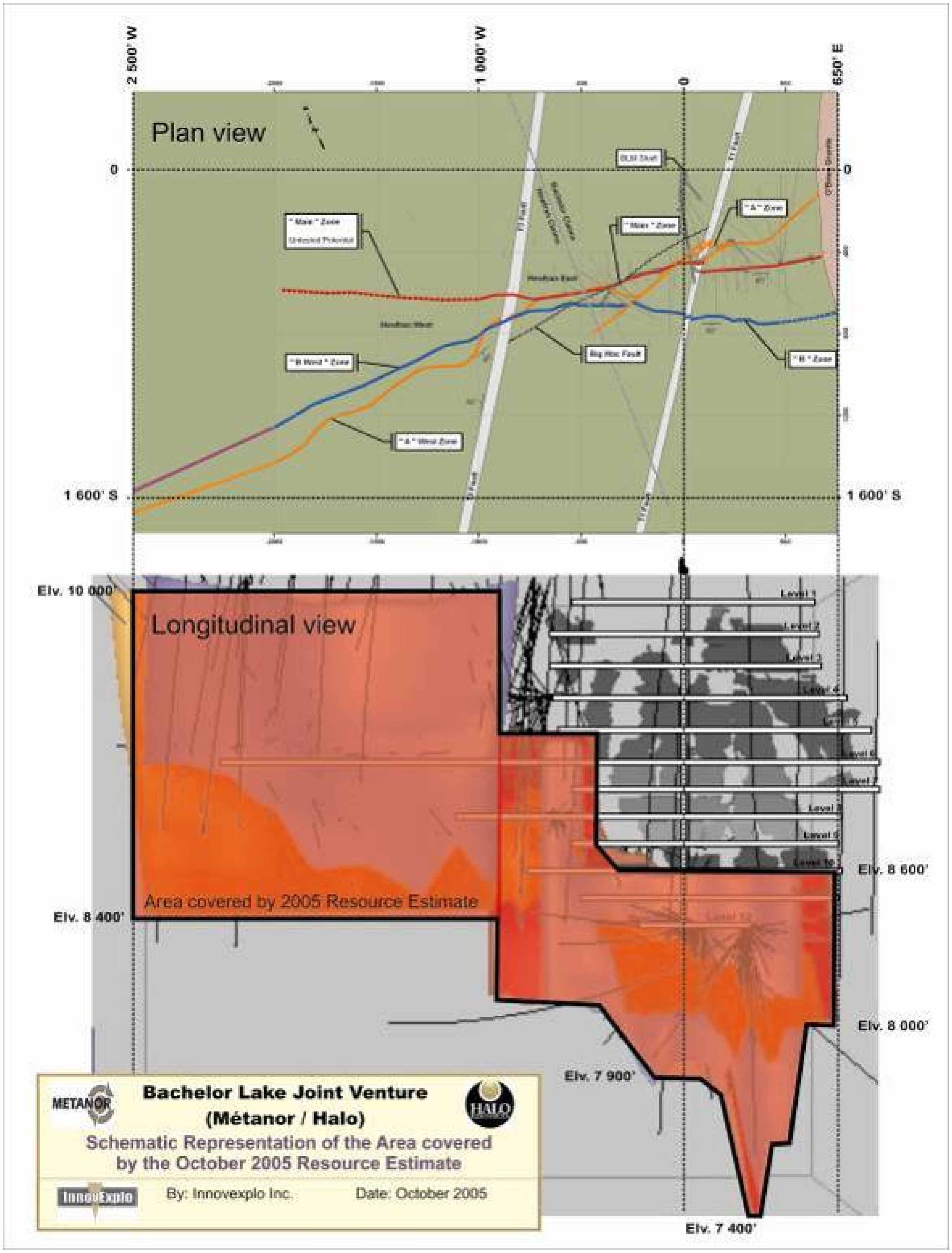


Figure 17.2 - Area covered by the 2005 resource estimate

Drill holes and assay database

The Gemcom (GEMS 5.51) database used for the 2005 resource estimates comprise **15 192 assay analyses** from **394 diamond drill hole** records. The quality of the database has been discussed in detail in previous sections ("Sample preparation, analyses and security" and "Data verification").

In summary, results from check sampling on historic holes and the performance of the laboratory during the 2005 drilling program were good. As stated in Horvath and Carrier (2005): no contamination was identified, the accuracy of results was very good and the precision (i.e. reproducibility) was quite good for a gold deposit.

The 2005 estimation relies upon **3 684 composite** intervals of ±0.75 m (2.5'): 1 005 composites for the "Main" zone; 874 composites for the "B" zone; 889 composites for the "A" zone; 395 composites for the "B" West zone; 458 composites for the "A" West zone and 63 composites for the "C" zone.

Geostatistical evaluation

Horvath and Carrier (2005) did a geostatistical evaluation of the Bachelor Lake drill hole database used for resource estimate and demonstrated that the geological model correlated very well with the assay grades. The mineralized zones have been defined by their characteristic geological features which also demonstrated zone specific geostatistical features. The geostatistical features of these individual zone assay sub-populations have been evaluated by univariate statistics and variography to provide the recommended treatment of the raw assay data and values for the required parameters necessary to complete a block model resource estimate.

Details on the geostatistical analysis for both univariate statistics and variography have been reproduced from Horvath and Carrier (2005) in Appendix IX. Recommendations for block modelling are summarized in the following table.

TABLE 9.1 from the geostatiscal evaluation of Horvath and Carrier (2005)

Bachelor Lake Mines Summary Block Modelling Recommendations for DDH Assays

	Recommended Procedure
Data Preparation	
Final Au results	Average all Fire Assays (ie. AAS & Grav.) except use Met. Screen if present
Assay Compositing	Composite final Au assays on 2.5 feet equal lengths
	Exclude composites with <50% of composite interval assayed
Composite Cutting	Cut composites grades >1.5 opt Au within Vein M to 1.5 opt Au
	Cut all other composite grades >1.0 opt Au within Veins A & B to 1.0 opt Au
Block Model Parameters	
Block Size	Recommend blocks dimensions with similar anisotropy to ellipse with 2.5 feet for minimum direction
Ore Loss/Dilution Considerations	Estimate volume% of vein wire frame solids within blocks
Ore loss	For blocks with >0% and <50% vein solids (ie. waste side of contacts), estimate volume of vein wire frames and calculate ore loss
Dilution	Similarly for blocks with >50% and <100% vein solids (ie. ore side of contacts), estimate volume of vein wire frames and calculate dilution
	Optimize block dimensions to minimize vein loss and dilution
Sample Search Parameters	Anisotropic search as defined by Azimuth (principal axis of ellipse) - Dip (principal axis of ellipse) - Azimuth (2ndary axis) Method
Search Ellipse Orientations	
Blocks within Veins M and B	
ellipse principal axis (x) Az	180 deg
ellipse principal axis (x) Dip & Range	-75 deg , 125 feet
ellipse intermediate axis (y) Az & Range	090 deg , 75 feet
ellipse tertiary axis (z) Range	15 feet
Blocks within Vein A	
ellipse principal axis (x) Az	140 deg
ellipse principal axis (x) Dip & Range	-65 deg , 115 feet
ellipse intermediate axis (y) Az & Range	050 deg , 75 feet
ellipse tertiary axis (z) Range	15 feet
Ellipse SubSearch Type	Octant - (subdivides ellipse into 8 octants, recommended for declustering clustered data especially for ID interpolation)
Max. samples per Octant	12
Min. number of Octants with samples	1
Max. samples per Hole	7 = 17.5 feet downhole range/2.5 feet composites
High Grade Transition	none
Krigging	Use nugget, sill, range and variogram models as defined in tables provided

Specific gravity, minimum width, cut-off grade, compositing and capping

Historically at Bachelor Lake, a **specific gravity (SG)** of 12 ft^3/t was used for tonnage estimate. No studies were available on the determination of that SG value. In 2005, this value was revised and calculated on 26 samples from the "Main", "B" and "A" zones. Results indicated that a fixed density of **2.755 g/cm^3** per metric ton (***11.636 ft^3/t***) should be used. The results, shown in Appendix IX, did not show major differences between the three zones and their host rocks.

The **minimum horizontal width** used during the interpretation was established at **1.5 m** (***5'***). 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.

The **arbitrary cut-off grade** was established at **3.43 g/t Au (0.10 oz/t Au)**. A cut-off grade of 3.43 g/t Au roughly represents a value of US$47/t at the current gold price. This cut-off grade must be re-evaluated in light of the present market conditions: gold price, exchange rate and mining cost.

Compositing of the final gold assay results was done on **2.5' (± 0.75 m) equal lengths** and constraint within the interpreted wire frame for each zone. Composites having lengths of 50% of the composite interval were excluded for the interpolation calculations.

Historically and during mining at Bachelor Lake, **high grade assays were capped** at 34.3 g/t Au (1.0 oz/t Au) for the "Main" zone, and to 22.3 g/t Au (0.65 oz/t Au) for the "B" zone. In the Bachelor archives, no statistical studies were available to support this threshold.

For the 2005 estimation, capping was done on the composite interval. The last statistical study reveals the following **capping values**:

"Main" zone	51.43 g/t Au (1.5 oz/t Au)
"A" zone	34.29 g/t Au (1 oz/t Au)
"B" zone	34.29 g/t Au (1 oz/t Au)
"C" zone	34.29 g/t Au (1 oz/t Au)
"A" West zone	34.29 g/t Au (1 oz/t Au)
"B" East zone	34.29 g/t Au (1 oz/t Au)

Capping and compositing is supported by Horvath and Carrier (2005) statistical study as reproduced below:

[… Figure 6.3 demonstrates that each of the vein sub-populations display similar trends to assay higher grade when shorter sample intervals are used. As stated earlier however, these higher grade shorter intervals can be correlated with geological features within the vein zones. It is notable that all assays that exceed the cutting-values indicated from the log-probability plots in each sub-population are all less than the average sample length (i.e. 2.73 feet) for samples within the wire frame solids.

The issue of how to treat these few high grade assay values that remain within these wire frames is not too significant since only 4 assays occur within the Main vein that are greater than the indicated cut value of 1.5 opt Au (max. 2.62 opt Au) and similar only 8 assays occur within the B vein that are greater than the indicated cut-value of 1.0 opt Au (max. 1.584). No assays occur with the A vein that are greater than 1.0 opt (max. 0.81 opt Au). Figure 6.3 also indicates the effect that cutting the values greater than the indicated cut-values would produce versus that of weighted average composites for the assay values within the individual wire frames on equal 2.5 feet lengths.

Compositing the assays on short 2.5 feet sample intervals has several effects on the assay population as can be seen from the from Figure 6.1 between the resulting statistics from all assays in the Bachelor Lake database versus 2.5 feet equal length composites of all the assays. The results demonstrate that compositing the data has the beneficial effects of smoothing the somewhat erratic nature of individual values as indicated by the lower variance and co-efficient of variance for the composite values while not impacting the median and actually lowering the mean grade slightly. The compositing of the high grade assay values on 2.5 feet equal lengths is having the effect of normalising the sample population by smoothing any bias possibly introduced by variable sample lengths. Further evidence of this smoothing effect is clearly demonstrated in the better correlation of points in the variography of the composite values versus those of the raw data and is a direct result of the lower overall variance for the composite value population. The composite length of 2.5 feet was selected based on normalizing the data near to the average sample lengths within the veins.

Alternatively, and more commonly the accepted practice is to cut the assay values to the indicated threshold value from the log-probability plots (i.e. Main vein at 1.5 opt Au and B vein at 1.0 opt Au) in this case affecting only 12 assays. The effect of cutting the high values is also demonstrated in Figure 6.3 and demonstrates that all grade (i.e. gold content) greater than the cut-value is simply discounted to have no value (i.e. same sample length at lower grade).

Recall that the review of the geological model and features of the deposits with InnovExplo and Halo/Metanor Resource geologists included re-interpreting "vein" intersection areas as potential shoots or pockets of high-grade mineralization. In addition, to the "vein" intersection zones, wider zones within the individual "veins" often demonstrated an apparent zoning from lower grades at the contacts to high grade cores that might be modelled as potential high grade shoots or pockets.

Table 6.4 includes five examples from the 2005 ddh program of high grade intersections of veins M, B and/or vein B and A intersection zones of which three contain values exceeding 1.0 opt Au.

Table 6.4 Example High Grade Vein Intersections and Grade Zoning From Horvath and Carrier (2005) statistical study

Hole ID	Sample	From (m)	To (m)	Length (m)	Au (g/t)	Au (oz/t)	Geology
12-59	108,756	97.50	98.90	1.40	5.83	0.17	Vn B/A? Intersection
12-59	108,757	98.90	99.40	0.50	4.82	0.14	Vn B/A? Intersection
12-59	108,758	99.40	99.75	0.35	15.75	0.46	Vn B/A? Intersection
12-59	108,759	99.75	100.40	0.65	0.22	0.01	Vn B/A? Intersection
12-59	108,760	100.40	100.70	0.30	34.30	1.00	Vn B/A? Intersection
12-59	108,761	100.70	101.05	0.35	16.50	0.48	Vn B/A? Intersection
12-59	108,762	101.05	101.35	0.30	5.48	0.16	Vn B/A? Intersection
12-59	108,763	101.35	102.15	0.80	5.21	0.15	Vn B/A? Intersection
12-59	108,764	102.15	102.45	0.30	8.77	0.26	Vn B/A? Intersection
12-59	108,765	102.45	103.50	1.05	4.89	0.14	Vn B/A? Intersection
12-59	108,766	103.50	105.00	1.50	5.65	0.16	Vn B/A? Intersection
12-88	110181	103.50	105.00	1.50	2.40	0.07	Vn B
12-88	110182	105.00	105.85	0.85	9.57	0.28	Vn B
12-88	110183	105.85	106.70	0.85	12.55	0.37	Vn B
12-88	110184	106.70	107.30	0.60	18.85	0.55	Vn B
12-88	110185	107.30	107.80	0.50	42.30	1.23	Vn B
12-88	110186	107.80	108.90	1.10	15.15	0.44	Vn B
12-88	110187	108.90	109.60	0.70	10.52	0.31	Vn B
12-106	122523	78.00	78.50	0.50	12.55	0.37	Vn M
12-106	122524	78.50	79.05	0.55	34.00	0.99	Vn M
12-106	122526	79.05	79.80	0.75	40.80	1.19	Vn M
12-106	122527	79.80	80.75	0.95	18.20	0.53	Vn M
12-106	122528	80.75	81.50	0.75	2.62	0.08	Vn M
12-112	122663	97.95	98.25	0.30	6.51	0.19	Vn M
12-112	122664	98.85	99.35	0.50	4.23	0.12	Vn M
12-112	110536	99.35	99.85	0.50	82.30	2.40	Vn M
12-112	122665	99.85	100.50	0.65	5.80	0.17	Vn M
12-112	122667	100.50	102.00	1.50	1.83	0.05	Vn M
12-112	122668	102.00	102.90	0.90	4.48	0.13	Vn M
12-83	110767	108.55	109.55	1.00	7.84	0.23	Vn B/A Intersection
12-83	110768	109.55	110.30	0.75	14.85	0.43	Vn B/A Intersection
12-83	110517	110.30	110.60	0.30	26.90	0.78	Vn B/A Intersection
12-83	110519	110.60	110.90	0.30	21.90	0.64	Vn B/A Intersection
12-83	110769	110.90	111.20	0.30	23.60	0.69	Vn B/A Intersection
12-83	110771	111.20	111.75	0.55	22.60	0.66	Vn B/A Intersection
12-83	110772	111.75	112.65	0.90	5.93	0.17	Vn B/A Intersection

The results demonstrate that locally high grade cores occur at some locations within the veins and at intersections of veins. Simply bracketing the multi-gram grades in the examples presented indicates zones up to 2.5 feet and more of multi-gram grade.

The ability to identify and potentially model these "high grade shoots" within the vein zones or at intersections will likely require significant in-fill and additional drilling however, the benefits would clearly be significant. Cutting the assay values prior to compositing may unnecessarily discount gold grades that are actually reasonably accurate resulting in an inability to define or underestimate potential "high grade" resources.

The purpose of cutting assay values is to prevent the impact of values considered erratic and potentially misrepresentative of grade so as not to overestimate resource and resource grade. In consideration that the 2.5 feet composite intervals represent only 30% of a minimum width for mining, and all other factors reviewed above, the effect of not cutting these few possibly representative rather than erratic assays prior to compositing would not be misrepresent grade and/or resource, however may assist and more realistically assign grades to potentially "high grade" shoots as suggested by the current geological model. It would be recommended however, that the composites eventually be capped at the indicated cut-off of grades prior to resource estimation so that no possibility would exist for severe overestimation by limiting any impact to 1/3rd of a probable minimum mining width. …]

Methodology

In previous Mineral Resource estimates, the methodologies used were extrapolation methods (polygonal on longitudinal section or on transversal section - Fig. 17.3). For the 2005 resource estimate, both methods were used: (1) **extrapolation** for the **Measured Resources** from underground sampling results (faces and lifts); and (2) **interpolation** for the **Indicated and Inferred Resources** from a drill hole database.

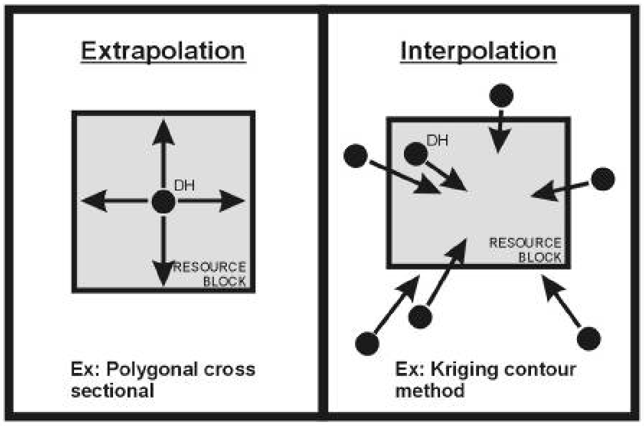


Figure 17.3 - Type of resource estimation method (modified from RPA; Roscoe and Clow, 2004)

Measured Resources were modified from Innovexplo (2004) re-classification. Originally evaluated by Harron (1990), and modified afterwards by Géospex (P. Gagnon, 1993; and Y. Gagnon, 1995), the following modifications were made for the 2005 estimation: correction of the volume for block 12E (11-02-E1); subtraction of block 12E (955 t); modification of block associated with the "A" zone; and calculation of the Measured Resources for the Hewfran East area. Tonnage and grade for the Measured Resources were obtained from ore outlines of the zones supported by geological mapping and calculations on underground sampling results.

Indicated and Inferred Resources were obtained from block modelling constraint by wire frame. The **block size** was established at 76 cm (2.5') (N-S) x 2.28 m (7.5') (vertical) x 1.5 m (5.0') (E-W) reflecting anisotropic nature of the mineralized zones.

The interpolation method retained for the estimation of **Indicated and Inferred** Resources was the **inverse distance squared method (ID^2)** illustrated in Figure 17.4. The method was chosen after the realization of tests at different power increments (ID^0, ID^2, ID^4, and ID^8). No tests were realized using ordinary kriging or indicator kriging.

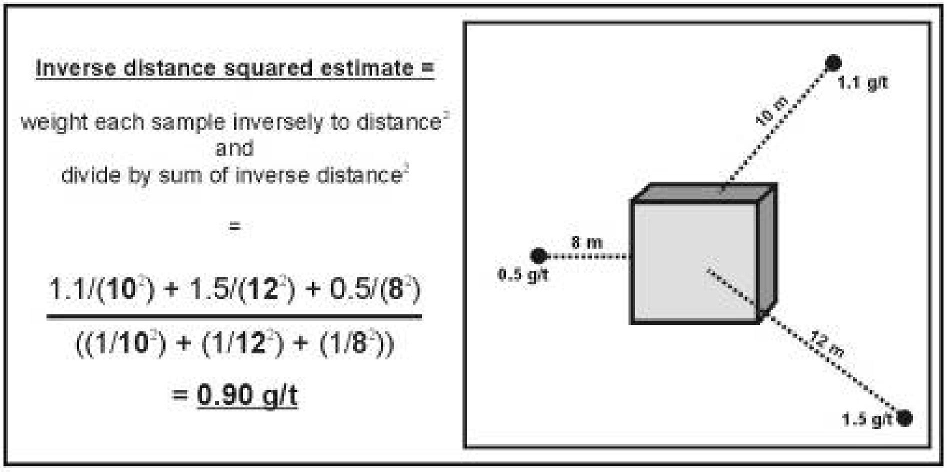


Figure 17.4 - Inverse distance squared method (modified from SNOWDEN; Glacken, 1999)

Grade interpolation for the whole block model was done in two (2) phases: (1) 2.5 X range for the Inferred Resources; and (2) 1 X range for the Indicated Category and a portion of Inferred.

Grade estimation was obtained from anisotropic search ellipse (defined from the variography study) using: a minimum of 1 composite interval per octant; a maximum of 12 composites per octant; and a maximum of 7 composites per drill hole.

After the grade estimation, the **Resource categories** were defined using the range value as follows: (1) blocks within ½ range are **Indicated**; (2) blocks between ½ the range and 2.5 X the range are **Inferred**.

New resource estimate summary

The effective date of the resource estimate is October 5, 2005. The resource estimate has been established within reasonable parameters. These parameters were defined by Alain Carrier, M.Sc., P.Geo. and Julien Davy, M.Sc., P.Geo. from Innovexplo, Qualified and Independent people, and were based on recommendations of the **CIM Standing Committee on Ore Reserves and Resources** and are compliant to regulation of the **National Instrument 43-101**. The Qualified people were involved from the data acquisition phase, validation, geological interpretation, 3D modeling, establishment of key assumptions, and resource calculation.

Mineral Resources which are not mineral reserves do not have demonstrated economic viability. Results from the resource estimate are presented **undiluted** and **in situ** and some resource blocks may be locked in pillars. The resources were compiled using a **cut-off grade of 3.43 g/t Au** (0.10 oz/t Au). A **fixed density of 2.755 g/cm^3** (11.636 ft^3/t) was used.

Measured Resources were evaluated from underground mapping and faces sampling results. Indicated and Inferred Resources were evaluated from drill hole results using a block model approach. In the block model, each zone was constrained within narrow wire-frames with a **minimum of 1.5 m (5.0’) horizontal width**, using the grade of the adjacent material when assayed, or a value of zero when not assayed. **High grade capping** were fixed at **51.4 g/t Au (1.5 oz/t Au)** for the “Main” zone, and to **34.3 g/t Au (1.0 oz/t Au)** for the “A”, “B”, “C”, “A” West and “B” West zones.

Calculations were done using Imperial units (feet, short tons and oz/t Au) and results were rounded to reflect their “estimate” nature. These results were after converted in Metric using a factor of 0.90178 for the conversion of short tons into tonnes and a factor of 34.2865 for the conversion of oz/t Au into g/t Au.

Table 17.1: Results of the 2005 Mineral Resources estimate

BACHELOR LAKE MINERAL RESOURCES (METRIC UNITS)

		BACHELOR	HEWFRAN	TOTAL
Measured	**Metric Tons (t)**	**177 898**	**14 696**	**192 594**
	Grade (g/t)	**8.83**	**8.50**	**8.80**
	Oz of Gold	*50 487*	*4 018*	***54 504***
	kg of Gold	*1 570*	*125*	***1 695***
Indicated	**Metric Tons (t)**	**465 928**	**183 069**	**648 997**
	Grade (g/t)	**7.63**	**7.14**	**7.49**
	Oz of Gold	114 329	42 024	**156 352**
	kg of Gold	*3 556*	*1 307*	***4 861***
Measured + Indicated	**Metric Tons (t)**	**643 826**	**197 765**	**841 591**
	Grade (g/t)	**7.96**	**7.24**	**7.79**
	Oz of Gold	*164 815*	*46 042*	***210 857***
	kg of Gold	*5 126*	*1 432*	***6 556***
Inferred	**Metric Tons (t)**	**207 517**	**218 630**	**426 148**
	Grade (g/t)	**6.76**	**6.30**	**6.52**
	Oz of Gold	*45 083*	*44 283*	***89 366***
	kg of Gold	*1 402*	*1 377*	***2 778***

1.) The Qualified People for the Mineral Resource estimates as defined by National Instrument 43-101 were Alain Carrier, M.Sc., P.Geo. and Julien Davy, M.Sc., P.Geo. (Innovexplo Inc.), and the effective date of the estimate is October 5, 2005.
2.) Mineral Resources are not Mineral Reserves having demonstrated economic viability.
3.) Results are presented undiluted and in situ, and some resource blocks may be locked in pillars. The estimate included six (6) gold-bearing zones (“Main”, “A”, “B”, “C”, “A West” and “B West”) and covers the Bachelor Lake, Hewfran East and West areas.
4.) The resources were compiled using a cut-off grade of 3.43 g/t Au. This cut-off must be re-evaluated in the light of the present market conditions: gold price, exchange rate and mining cost. A fixed density of 2.755 g/cm^3 was used. A minimum of 1.5 m horizontal width was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed. High grade capping were fixed at 51.4 g/t Au for the “Main” zone, and to 34.3 g/t Au for the “A”, “B”, “C”, “A” West and “B” West zones and were done on 0.75 m drill hole composite interval.
5.) Measured Resources were evaluated from a polygonal method using underground geological mapping and face sampling assay results.
6.) Indicated and Inferred Resources were evaluated from drill hole results using a block model approach (inverse distance squared interpolation) constrained within six (6) individual 3D wire frames (“Main”, “A”, “B”, “C”, “A West” and “B West” zones).
7.) Calculations used Imperial units (feet, short tons and oz/short ton Au) and results were rounded to reflect their “estimate” nature. These results were later converted into Metric using a factor of 0.90178 for the conversion of short tons into tonnes and a factor of 34.2865 for the conversion of oz/t Au into g/t Au.
8.) The companies are not aware of any known environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues that could materially affect the Mineral Resource estimates.

17. Preliminary economic assessment

This item is describing the mining approach of both properties, their capital and production costs and the preliminary Cash Flow.

17.1 Barry 1

17.1.1 Mining Operation Proposal

The gold mineralization at the Barry-1 deposit is exposed at surface and favourable to be mined in the first phase by open pit that was designed using the following parameters in metric units.

Parameters Description				
Ore mining cost	$6.34/t			
Crushing cost	$1.44/t			
Transport	$18.22/t			
Processing (milling)	$18/st			
Gen+Adm cost	$5/t			
Waste mining cost	$4.36/t			
OVB mining cost	$3.00/t			
Metallurgical recovery	95%			
Gold price (US$/oz)	600			
Exchange rate	1.1			
Royalties on net revenue	11%			
SG	2.8 (rock), 2.0 (overburden)			

Table 17.1: List of global open-pit parameters

From the above parameters the cut off grade is 2.89 g/t Au. The optimization results of the total pit design including the dilution and the ramp are the followings:

Bench	Floor Elevation	Ore tonnage	Waste tonnage	OVB tonnage	g/t Au
1	2,014	0	18,226	17,400	
2	2,011	216	31,459	99,580	3.52
3	2,008	4,623	89,755	121,700	4.15
4	2,005	20,724	203,786	82,450	4.35
5	2,002	45,378	264,204	63,653	4.50
6	1,999	58,928	252,540	39,826	4.70
7	1,996	58,048	225,550	18,031	4.83
8	1,993	58,163	185,493	2,936	4.63
9	1,990	48,597	138,729	14	4.81
10	1,987	30,456	119,209	0	4.93
11	1,984	23,741	69,599	0	5.68
12	1,981	22,212	44,132	0	6.15
13	1,978	18,650	29,569	0	6.74
14	1,975	10,412	19,558	0	7.40
15	1,972	9,184	6,066	0	7.60
16	1,969	4,840	832	0	8.92
Total		414,171	1,698,708	445,591	5.10
		W/O ratio:	5.18		

Table 17.2: Results of the global Barry-1 open-pit optimization

The outline of this pit which is illustrated in the figure 17.1 is clearly showing two main ore zones that were designated as the East and West Pits.

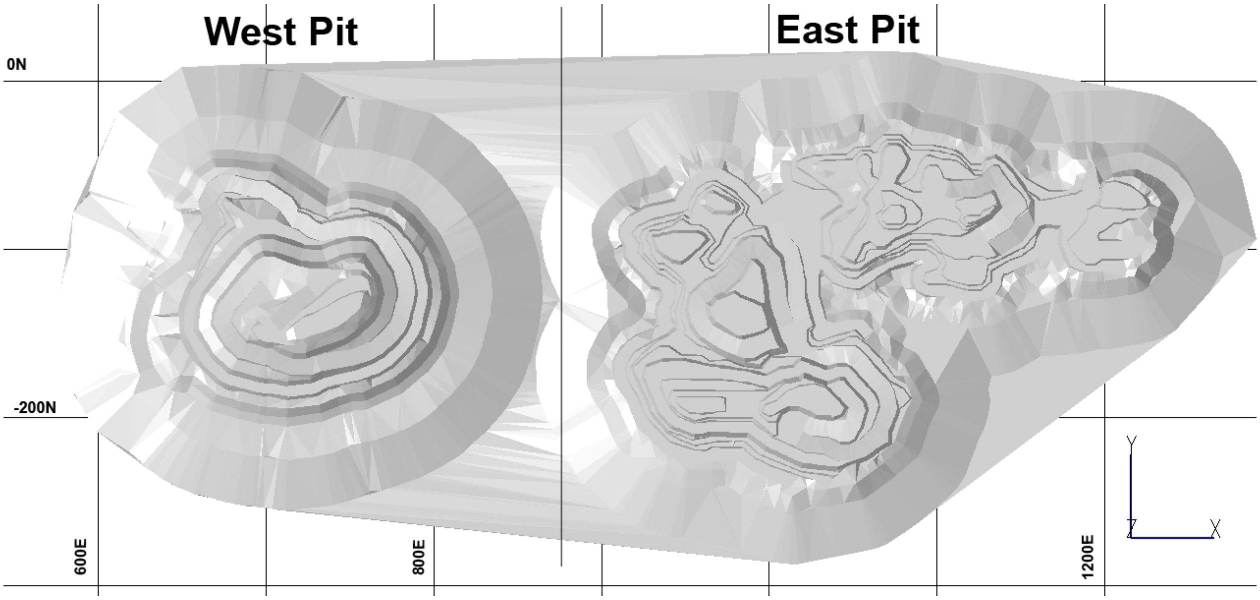


Figure 17.1: Outline of the global Barry-1 open-pit

The total mineralized material at the Barry-1 project appeared to be economically mineable through the open pit mining method. From the beginning, the combined East and West pits represented some economical weaknesses on account of the 1,698,000 metric tonnes of waste that would have to be mined out to access the 414,171 mt of resources at an average grade of 5.10 g/t Au, before dilution. After a tonnage dilution of 20% at a grade of 0.5 g/t Au, the average grade to the mill drops to 4.33 g/t Au.

To total waste material to be moved to access the ore material is increased by 445,591 tonnes when taking the overburden into consideration. The Barry pit(s) being at near 100 km away by road from milling facilities automatically increases the cost of placing a tonne at the mill site substantially.

To decrease the waste and the overburden tonnage to a more acceptable level while improving the economical viability of the project **a combination of open pitting and one underground mining option was scrutinized.**

17.1.2 Open-pit resources

An evaluation of the resources of the two possible separate pits gave the following results:

East Pit – Diluted Resources in metric units

Bench	Floor Elevation	Ore tonnage	Waste tonnage	OVB tonnage	Ore g/t
1	2,014	0	5,756	3,129	
2	2,011	260	13,536	34,014	3.02
3	2,008	5,548	50,455	48,100	3.54
4	2,005	21,600	117,617	52,707	3.83
5	2,002	49,512	155,354	57,033	3.89
6	1,999	67,728	139,500	39,031	4.03
7	1,996	66,225	138,562	18,031	4.13
8	1,993	64,559	105,781	2,936	3.97
9	1,990	52,334	67,501	14	4.10
10	1,987	28,431	65,725	0	4.17
11	1,984	17,719	31,577	0	4.96
12	1,981	14,475	12,868	0	5.37
13	1,978	8,520	5,807	0	5.56
14	1,975	2,517	2,350	0	6.46
Total		**399,427**	**912,390**	**254,995**	**4.16**
		W/O ratio:	2.92		

Table 17.3: Results of Barry-1 East open-pit

West Pit – Diluted metric tonnes

Bench	Floor Elevation	Ore tonnage	Waste tonnage	OVB tonnage	Ore Au
1	2,014	0	12,470	14,272	
2	2,011	0	17,880	65,566	
3	2,008	0	38,375	73,600	
4	2,005	3,269	82,024	29,742	2.90
5	2,002	4,942	99,774	6,621	3.26
6	1,999	2,986	101,255	795	3.29
7	1,996	3,433	75,378	0	3.79
8	1,993	5,236	68,080	0	3.60
9	1,990	5,982	61,509	0	3.95
10	1,987	8,117	47,393	0	4.28
11	1,984	10,769	33,274	0	4.58
12	1,981	12,179	26,821	0	5.01
13	1,978	13,860	20,031	0	5.78
14	1,975	9,976	15,126	0	6.19
15	1,972	11,021	4,229	0	6.42
16	1,969	5,672	0	0	7.68
Total		97,442	703,619	190,596	5.05
		W/O ratio:	9.18		

Table 17.4: Results of Barry-1 West open-pit

During this evaluation it was noted that the West pit was adding only a Net Value of $43,850 when ramp and dilution are taken in account. The West pit while having only 20% of the potential ore resources (97,442 t), has 42% of the waste and 43% of the overburden that has to be extracted to access these resources. Although the grade of the West pit is somewhat superior to the East pit, the highest grade portion is situated at the last five benches of the 16 proposed benches.

It is suggested to mine by open pit method the East portion of the Barry resources while ramping to the West from the wall of the East pit in order to access the high grade resources starting at bench no 12.

17.1.3 Final East Open-Pit Proposal

The final proposed East Open-Pit containing 440,000 diluted short tons (399,427 mt) at a grade of 0.121 opt (4.16 g/t Au) is illustrated on the next page.

Figure 17.2: Barry-1 East Pit horizontal projection with ramp

An aerial view of the East Open-Pit is shown below.

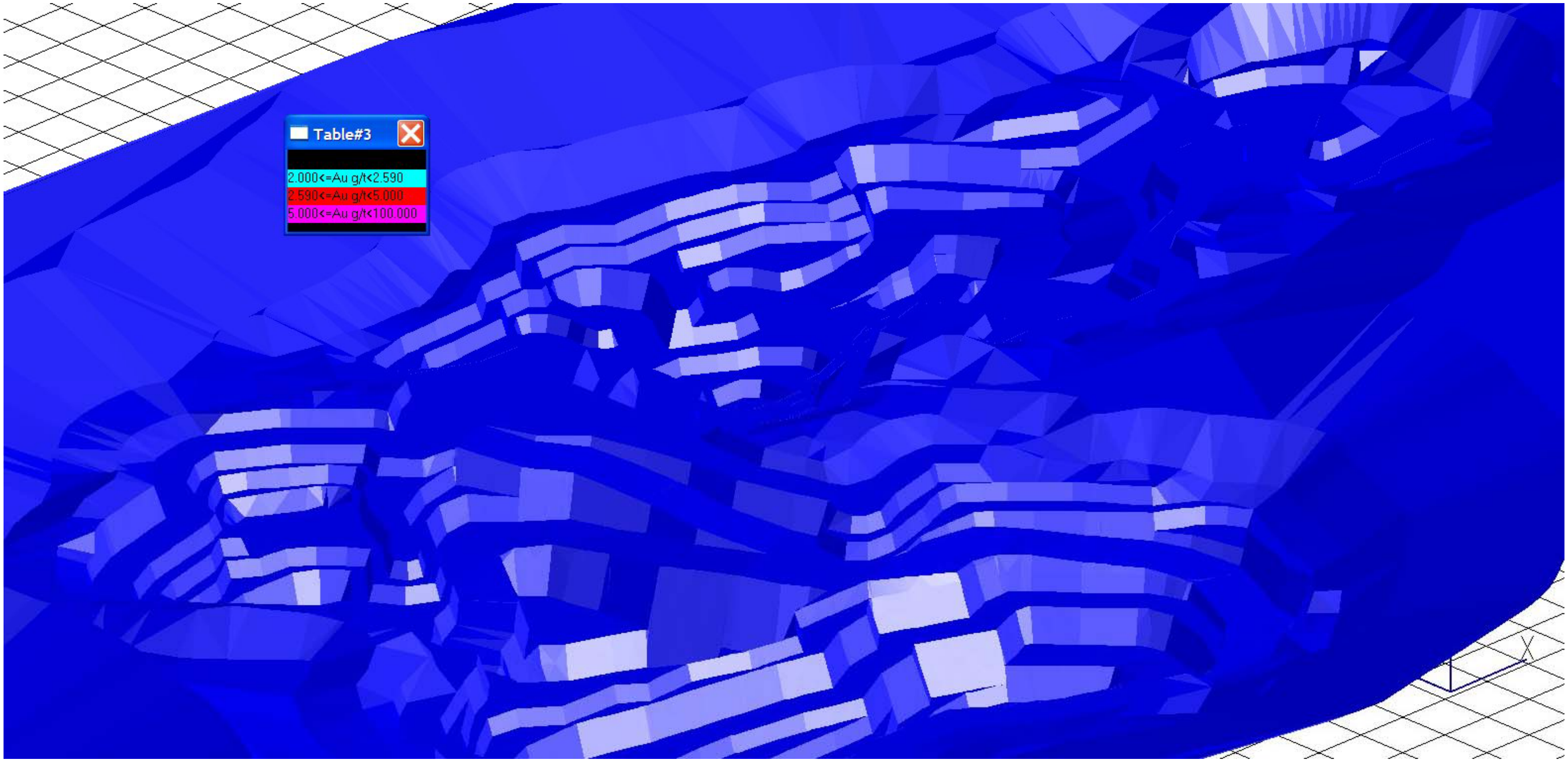


Figure 17.3: Aerial view of Barry-1 East Open-Pit

17.1.3.1 Ore Scheduling by month and bench

The next table is giving the design ore scheduling as produced by the open pit computer

Barry East Pit Ore Schedule

		Mill feed (tonnes)		13608	13608	13608	13608	19051	19051	19051	19051	19051	19051	19051	19051	19051	19051	19051	19051	19051	19051	19051	19051	19051	19051	19051	
Bench	Ore tonnage	g/t Au	Month	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	
1	0																										0
2	260	3.02		260																							260
3	5,548	3.54		4,616	932																						5,548
4	21,600	3.83		8,732	12,676	192																					21,600
5	49,512	3.89				13,416	13,608	19,051	3,437																		49,512
6	67,728	4.03							15,614	19,051	19,051	14,012															67,728
7	66,225	4.13										5,039	19,051	19,051	19,051	4,032											66,225
8	64,559	3.97														15,019	19,051	19,051	11,439								64,559
9	52,334	4.10																	7,612	19,051	19,051	6,619					52,334
10	28,431	4.17																				12,432	15,999				28,431
11	17,719	4.96																					3,052	14,667			17,719
12	14,475	5.37																						4,384	10,092		14,475
13	8,520	5.56																							8,520		8,520
14	2,517	6.46																							440	2,078	2,517
	399,427		Mill feed	13,608	13,608	13,608	13,608	19,051	19,051	19,051	19,051	19,051	19,051	19,051	19,051	19,051	19,051	19,051	19,051	19,051	19,051	19,051	19,051	19,051	19,051	2,078	399,428
			Av Grade	3.72	3.81	3.89	3.89	3.89	4.01	4.03	4.03	4.06	4.13	4.13	4.13	4.01	3.97	3.97	4.03	4.10	4.10	4.15	4.29	5.05	5.48	6.46	4.16

Table 17.5: East open pit ore scheduling

17.1.4 West Zone Underground Mining

The proposed resources assumed to be exploited by underground method are estimated below.

West Zone diluted resources in imperial tons			
Bench	Tons	Grade	Ounces
12	11,184	0.172	1,928
13	12,728	0.200	2,540
14	9,161	0.214	1,959
15	10,120	0.222	2,243
16	5,333	0.260	1,388
11	9,889	0.157	1,555
10	7,453	0.147	1,094
Sub-Total	65,868	0.193	12,706
Dilution 15% at 0.015 opt	9,880	0.015	144
Total	**75,748**	**0.170**	**12,850**

Table 17.6: West zone UG diluted resources in short tons

It is suggested that a ramp be initiated at elevation 2000 (bench no 5) to intersect the West zone at bench elevation no 12. This suggestion is done on **the assumption that the rock is competent enough to implement low cost open stope mining** with local pillars left in accordance to a future detailed mining sequence.

The dip (inclination) of the material at the West pit do represents a mining disadvantage. It's, more or less inclined at 33^0, a situation often mined with room and pillar mining method with slushers. However, as a ramp is nearby, the mining method can be the horizontal down dip method and the ramp used to access the consecutive benches. Pillars are left were ground condition indicates by going into the foot wall for the distance required for the pillar.

The proposed layout of the ramp is shown below on the longitudinal section of the next page.

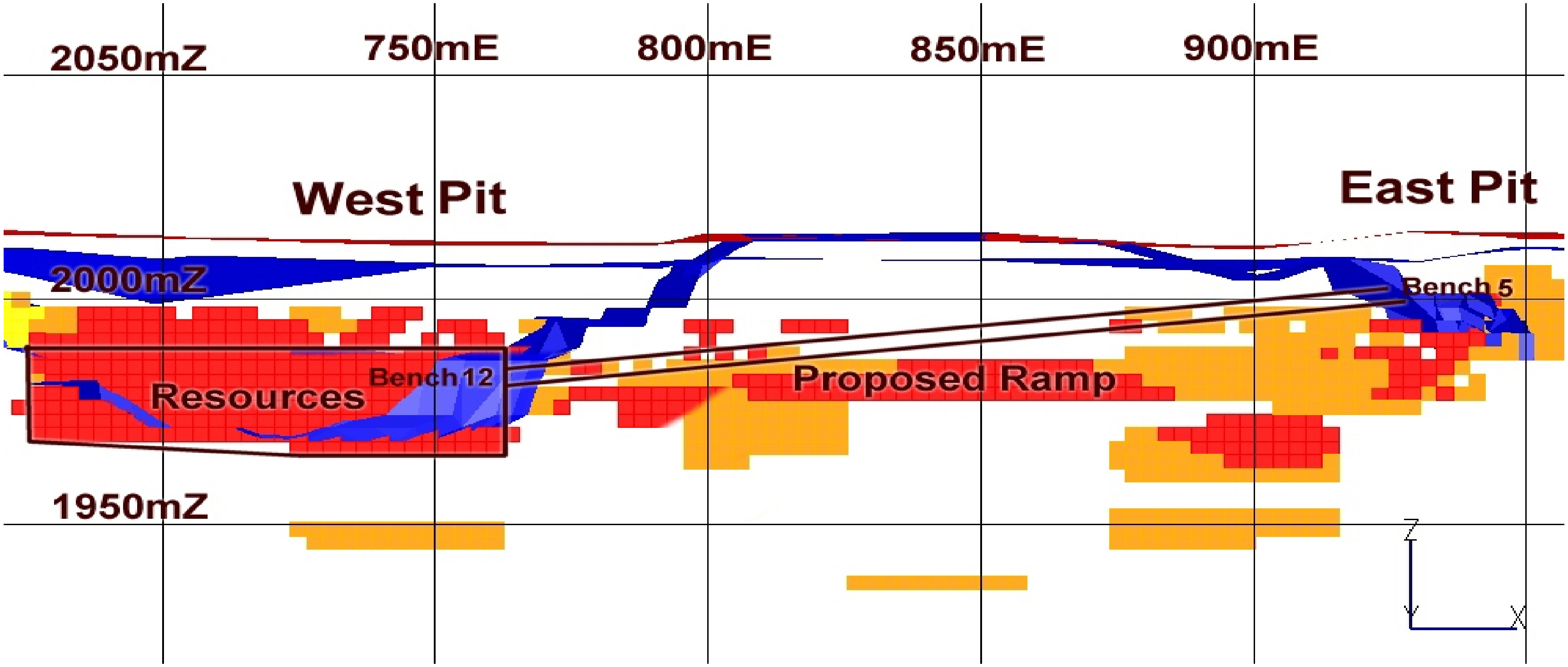


Figure 17.4: Long section showing West zone proposed UG mining

17.1.5 Capital and Operating Cost Estimates of East Open-Pit

East Pit

The unit operating costs are the same as those presented as the design parameters in table 17.1 plus the ore selectivity that is estimated at $1.25/t, the exception being the milling costs that are varying. The owners are forecasting to start the concentrator at 500 short tons per day and increase the tonnage to 750 tons per day after 4 months of operation.

East Pit Production Costs - imperial tonnes

Description	$/t	Q (tons)	$ total	$/t of ore
Overburden: $/ton of overburden	$ 2.72	281,000	$ 764,320	$ 1.74
Waste: $/ton of waste	$ 3.95	1,012,500	$ 3,999,375	$ 9.09
Ore: $/ton of ore	$ 5.75	440,000	$ 2,530,000	$ 5.75
Crushing: $/ton of ore	$ 1.28	440,000	$ 563,200	$ 1.28
Transport: $/ton of ore	$ 16.53	440,000	$ 7,273,200	$ 16.53
Ore selectivity	$ 1.25	440,000	$ 550,000	$ 1.25
Gen + Administration	$ 4.55	440,000	$ 2,002,000	$ 4.55
Sub-total			*$ 17,682,095*	*$ 40.19*
(milling at 500 tpd) Total	$ 24.97	52,500	$ 1,310,925	$ 65.16
(milling at 750 tpd) Total	$ 20.15	387,500	$ 7,808,125	$ 60.34
Average total cost for East Pit		**440,000**	**$ 26,801,145**	**$ 60.91**

Table 17.7: East Open pit production costs

There is no capital cost estimate for the pit exploitation itself as it is considered that all mining will be done by contractors. The capital costs are shown in the Cash Flow where the mill refurbishing, the upgrade to 750 tons, the tailing rehabilitation and the closure operation provision are involved.

It is proposed to first mine out a bulk sample to better evaluate and define the characteristics of the Barry-1 deposit. A sample in the vincinity of 50,000 tons at 0.128opt (4.39 g/t Au) is recommended. This bulk sample can be mined without any overburden removal as the ore is already exposed to surface and with a minimum of ±7,000 tons of waste. This operation will necessitate four months of mill operation at 500 st per day, or 13,125 st per month assuming an availability of 90%. The remaining resources are estimated to last for an additional 20 months.

The capital costs are therefore those of the refurbishing of the concentrator, the tonnage increase to 750 tons per day, the tailing pond rehabilitation and an estimate for the open pit closure provision for total of **$6,208,000.**

All operating and capital costs, with revenues estimate are presented in the following tables

BARRY: 4 m at 500 stpd, after 750 stpd

Description	Preproduction	Unit costs	Month-1	Month-2	Month-3	Month-4	**Bulk-total**	Month-5	Month-6	Month-7	Month-8	Month-9	Month-10	Month-11	Month-12	Year-1
CAPEX		$/ st (ore)														
Mill refurbishing to 500 stpd	$ 2,618,000															
Mill upgrading to 750 stpd	$ 1,100,000															
Tailing pond study & rehabilitation	$ 2,190,000															
Mine closure provision	$ 300,000															
Total CAPEX	**$ 6,208,000**															
East Pit Production Costs																
Overburden (st)	281,000		0	0	0	0	0	35,125	35,125	35,125	35,125	35,125	35,125	35,125	35,125	281,000
Waste (st)	1,012,500		1,763	1,763	1,763	1,763	7,052	84,303	84,303	84,303	84,303	84,303	84,303	84,303	84,303	681,476
Ore mining (st)	440,000		13,125	13,125	13,125	13,125	52,500	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	212,500
Overburden $/t	$ 2.72	$ 1.74	$ -	$ -	$ -	$ -	$ -	$ 95,540	$ 95,540	$ 95,540	$ 95,540	$ 95,540	$ 95,540	$ 95,540	$ 95,540	$ 764,320
Waste $/t	$ 3.95	$ 9.09	$ 6,964	$ 6,964	$ 6,964	$ 6,964	$ 27,855	$ 332,997	$ 332,997	$ 332,997	$ 332,997	$ 332,997	$ 332,997	$ 332,997	$ 332,997	$ 2,691,830
Ore $/t	$ 5.75	$ 5.75	$ 75,469	$ 75,469	$ 75,469	$ 75,469	$ 301,875	$ 115,000	$ 115,000	$ 115,000	$ 115,000	$ 115,000	$ 115,000	$ 115,000	$ 115,000	$ 1,221,875
Ore selectivity $/t	$ 1.25	$ 1.25	$ 16,406	$ 16,406	$ 16,406	$ 16,406	$ 65,625	$ 25,000	$ 25,000	$ 25,000	$ 25,000	$ 25,000	$ 25,000	$ 25,000	$ 25,000	$ 265,625
Crushing $/t	$ 1.28	$ 1.28	$ 16,800	$ 16,800	$ 16,800	$ 16,800	$ 67,200	$ 25,600	$ 25,600	$ 25,600	$ 25,600	$ 25,600	$ 25,600	$ 25,600	$ 25,600	$ 272,000
Transport $/t	$ 16.53	$ 16.53	$ 216,956	$ 216,956	$ 216,956	$ 216,956	$ 867,825	$ 330,600	$ 330,600	$ 330,600	$ 330,600	$ 330,600	$ 330,600	$ 330,600	$ 330,600	$ 3,512,625
Gen + Administration $/t	$ 4.55	$ 4.55	$ 59,719	$ 59,719	$ 59,719	$ 59,719	$ 238,875	$ 91,000	$ 91,000	$ 91,000	$ 91,000	$ 91,000	$ 91,000	$ 91,000	$ 91,000	$ 966,875
Milling 500 tpd $/t	$ 24.97	$ 24.97	$ 327,731	$ 327,731	$ 327,731	$ 327,731	$ 1,310,925									$ 1,310,925
Milling 750 tpd $/t		$ 20.15						$ 403,000	$ 403,000	$ 403,000	$ 403,000	$ 403,000	$ 403,000	$ 403,000	$ 403,000	$ 3,224,000
Total Expenses		$ 85.31	$ 720,045	$ 720,045	$ 720,045	$ 720,045	**$ 2,880,180**	$ 1,418,737	$ 1,418,737	$ 1,418,737	$ 1,418,737	$ 1,418,737	$ 1,418,737	$ 1,418,737	$ 1,418,737	**$ 14,230,075**
REVENUES	440,000		13,125	13,125	13,125	13,125	52,500	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	212,500
Mill feed diluted grade	f=0.02917		0.12806	0.12806	0.12806	0.12806	0.12806	0.12018	0.12018	0.12018	0.12018	0.12018	0.12018	0.12018	0.12018	
Mill recovery	95-96%		0.95	0.95	0.95	0.95		0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	
Ounces produced			1,597	1,597	1,597	1,597		2,307	2,307	2,307	2,307	2,307	2,307	2,307	2,307	
Gross revenue @ $CDN / oz	$ 660		$ 1,053,854	$ 1,053,854	$ 1,053,854	$ 1,053,854	$ 4,215,415	$ 1,522,921	$ 1,522,921	$ 1,522,921	$ 1,522,921	$ 1,522,921	$ 1,522,921	$ 1,522,921	$ 1,522,921	**$ 16,398,783**
Gross profit before royalties			**$ 333,809**	**$ 333,809**	**$ 333,809**	**$ 333,809**	**$ 1,335,235**	**$ 104,184**	**$ 104,184**	**$ 104,184**	**$ 104,184**	**$ 104,184**	**$ 104,184**	**$ 104,184**	**$ 104,184**	**$ 2,168,708**
Ore NSR royalties - 10%			$ 105,385	$ 105,385	$ 105,385	$ 105,385	$ 421,542	$ 152,292	$ 152,292	$ 152,292	$ 152,292	$ 152,292	$ 152,292	$ 152,292	$ 152,292	$ 1,639,878
Milling NSR royalty - 1%			$ 10,539	$ 10,539	$ 10,539	$ 10,539	$ 42,154	$ 15,229	$ 15,229	$ 15,229	$ 15,229	$ 15,229	$ 15,229	$ 15,229	$ 15,229	$ 163,988
(East Pit) EBITDA			$ 217,885	$ 217,885	$ 217,885	$ 217,885	**$ 871,539**	-$ 63,337	-$ 63,337	-$ 63,337	-$ 63,337	-$ 63,337	-$ 63,337	-$ 63,337	-$ 63,337	**$ 364,841**

Table 17.8: Operating costs & revenues from East open-pit, first 12 months

BARRY: 4 m at 500 stpd, after 750 stpd

Description	Preproduction	Unit costs	Month-13	Month-14	Month-15	Month-16	Month-17	Month-18	Month-19	Month-20	Month-21	Month-22	Month-23	Month-24	Year-2	TOTAL
CAPEX		$/ st (ore)														
Mill refurbishing to 500 stpd	$ 2,618,000															
Mill upgrading to 750 stpd	$ 1,100,000															
Tailing pond study & rehabilitation	$ 2,190,000															
Mine closure provision	$ 300,000															
Total CAPEX	**$ 6,208,000**															
East Pit Production Costs																
Overburden (st)	281,000															281,000
Waste (st)	1,012,500		84,303	84,303	84,303	78,115									331,024	1,012,500
Ore mining (st)	440,000		20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	11,500	231,500	444,000
Overburden $/t	$ 2.72	$ 1.74													$ -	$ 764,320
Waste $/t	$ 3.95	$ 9.09	$ 332,997	$ 332,997	$ 332,997	$ 308,554									$ 1,307,545	$ 3,999,375
Ore $/t	$ 5.75	$ 5.75	$ 115,000	$ 115,000	$ 115,000	$ 115,000	$ 115,000	$ 115,000	$ 115,000	$ 115,000	$ 115,000	$ 115,000	$ 115,000	$ 66,125	$ 1,331,125	$ 2,553,000
Ore selectivity $/t	$ 1.25	$ 1.25	$ 25,000	$ 25,000	$ 25,000	$ 25,000	$ 25,000	$ 25,000	$ 25,000	$ 25,000	$ 25,000	$ 25,000	$ 25,000	$ 14,375	$ 289,375	$ 555,000
Crushing $/t	$ 1.28	$ 1.28	$ 25,600	$ 25,600	$ 25,600	$ 25,600	$ 25,600	$ 25,600	$ 25,600	$ 25,600	$ 25,600	$ 25,600	$ 25,600	$ 14,720	$ 296,320	$ 568,320
Transport $/t	$ 16.53	$ 16.53	$ 330,600	$ 330,600	$ 330,600	$ 330,600	$ 330,600	$ 330,600	$ 330,600	$ 330,600	$ 330,600	$ 330,600	$ 330,600	$ 190,095	$ 3,826,695	$ 7,339,320
Gen + Administration $/t	$ 4.55	$ 4.55	$ 91,000	$ 91,000	$ 91,000	$ 91,000	$ 91,000	$ 91,000	$ 91,000	$ 91,000	$ 91,000	$ 91,000	$ 91,000	$ 52,325	$ 1,053,325	
Milling 500 tpd $/t	$ 24.97	$ 24.97														$ 1,310,925
Milling 750 tpd $/t		$ 20.15	$ 403,000	$ 403,000	$ 403,000	$ 403,000	$ 403,000	$ 403,000	$ 403,000	$ 403,000	$ 403,000	$ 403,000	$ 403,000	$ 231,725	$ 4,664,725	$ 7,888,725
Total Expenses		$ 85.31	$ 1,323,197	$ 1,323,197	$ 1,323,197	$ 1,298,754	$ 990,200	$ 990,200	$ 990,200	$ 990,200	$ 990,200	$ 990,200	$ 990,200	$ 569,365	**$ 12,769,110**	**$ 26,999,185**
REVENUES	440,000		20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	11,500	231,500	444,000
Mill feed diluted grade	f=0.02917		0.12018	0.12018	0.12018	0.12018	0.12018	0.12018	0.12018	0.12018	0.12018	0.12018	0.12018	0.12018	0.12018	
Mill recovery	95-96%		0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	
Ounces produced			2,307	2,307	2,307	2,307	2,307	2,307	2,307	2,307	2,307	2,307	2,307	1,327		
Gross revenue @ $CDN / oz	$ 660		$ 1,522,921	$ 1,522,921	$ 1,522,921	$ 1,522,921	$ 1,522,921	$ 1,522,921	$ 1,522,921	$ 1,522,921	$ 1,522,921	$ 1,522,921	$ 1,522,921	$ 875,680	**$ 17,627,810**	**$ 34,026,593**
Gross profit before royalties			**$ 199,724**	**$ 199,724**	**$ 199,724**	**$ 224,167**	**$ 532,721**	**$ 532,721**	**$ 532,721**	**$ 532,721**	**$ 532,721**	**$ 532,721**	**$ 532,721**	**$ 306,315**	**$ 4,858,700**	$ 7,027,408
Ore NSR royalties - 10%			$ 152,292	$ 152,292	$ 152,292	$ 152,292	$ 152,292	$ 152,292	$ 152,292	$ 152,292	$ 152,292	$ 152,292	$ 152,292	$ 87,568	$ 1,762,781	$ 3,402,659
Milling NSR royalty - 1%			$ 15,229	$ 15,229	$ 15,229	$ 15,229	$ 15,229	$ 15,229	$ 15,229	$ 15,229	$ 15,229	$ 15,229	$ 15,229	$ 8,757	$ 176,278	$ 340,266
(East Pit) EBITDA			$ 32,203	$ 32,203	$ 32,203	$ 56,645	$ 365,200	$ 365,200	$ 365,200	$ 365,200	$ 365,200	$ 365,200	$ 365,200	$ 209,990	**$ 2,919,641**	**$ 3,284,483**

Table 17.9: Operating costs & revenues from East Pit: months 12 to 24

17.1.6 Underground Mining of West Zone

Mining approach

When accessing the material at bench no 12, an ore drift is driven along the full length of the economically mineralized zone. The back is bolted and screened. During the diving of this production drift the ramp continues downwards towards the no 16 bench and upwards to the no 11 bench and also on the 12th bench elevation to access the parallel ore material structures situated in the foot wall that exist above the no 12 bench. Once this drift is completed the mining moves down the ramp to slash the next bench and so on until the no 16 bench. It must be noted that the bench numbering used is for the description of the mining process only and the size of the benches used during the actual mining can be much thicker than the present 3 meter benches. There are at least two options for the benching implementation; one is the Jumbo where a slash is taken and the other is by Long Holing. The Long Hole approach gives a near continuous mining possibility. The Jumbo has some mucking downtime as the next slash is being drilled off. The development ventilation is assured through fans and fan tubing while the production ventilation is through a ventilation-emergency escape way driven at the end of the mineralized zones. Mine water pumping can probably be through the ventilation raise. The summary of the west zone resources is shown below.

West Zone diluted resources in short tons

Bench	Tons	Grade	Ounces
12	11,184	0.172	1,928
13	12,728	0.200	2,540
14	9,161	0.214	1,959
15	10,120	0.222	2,243
16	5,333	0.260	1,388
11	9,889	0.157	1,555
10	7,453	0.147	1,094
Sub-Total	65,868	0.193	12,706
Dilution 15% at 0.015 opt	9,880	0.015	144
Total	**75,748**	**0.170**	**12,850**

Table 17.10: West zone diluted UG resources

17.1.7 West Zone Development and Mining

The estimated quantities and costs of the west zone development are summarized below.

West Zone Development

Access & Development Costs			
Main Ramp from East pit	Meters	$/m	$ (total)
Level 5 to 12	185	$ 3,500	$ 647,500
Level 12 to 11	50	$ 3,500	$ 175,000
Level 12 to 16	125	$ 3,500	$ 437,500
Ramp access to bench	50	$ 3,500	$ 175,000
Ventilation and Emergency Exit Raise	45	$ 3,500	$ 157,500
Total	455	$ 3,500	$ 1,592,500

Table 17.11: Summary of West zone development costs

The unit cost of mining is estimated to be $28.13/ short ton ($31.00/ metric tonne). This rather low cost is achievable only if the ground conditions are of a quality to realize **fully mechanized large open stope mining**.

Here is the estimated total costs of production for the West Zone

West Zone Production Costs - in imperial tons

Description	$/t	Tons of ore	$ total
Development	$ 20.95	76,000	$ 1,592,500
Ore Mining: $/ton of ore	$ 28.13	76,000	$ 2,137,880
Crushing: $/ton of ore	$ 1.28	76,000	$ 97,280
Transport: $/ton of ore	$ 16.53	76,000	$ 1,256,280
Gen + Administration	$ 4.55	76,000	$ 345,800
Sub	$ 71.44	76,000	$ 5,429,740
Milling at 750 st per day	$ 20.15	76,000	$ 1,531,400
Average total cost for West UG Zone	**$ 91.59**	76,000	**$ 6,961,140**

Table 17.12: West Zone production costs

The following table is the summary of costs and revenues of the West Zone exploitation.

BARRY-1 West Zone - UG Mining		$/ st(ore)	Month-24		Month-25	Month-26	Month-27	Month-28	Year-3	**TOTAL**
	Tons	76000	8000	8000	20000	20000	20000	8000	68,000	76,000
Development: 1492 ft @ $1,067/ft	$ 1,592,500	$ 20.95	$ 167,632	$ 167,632	$ 419,079	$ 419,079	$ 419,079	$ 167,632	$ 1,424,868	1,592,500
Production costs - Ore										
Mining $/t	$ 28.13	$ 28.13	$ 225,040	$ 225,040	$ 562,600	$ 562,600	$ 562,600	$ 225,040	$ 1,912,840	2,137,880
Crushing $/t	$ 1.28	$ 1.28	$ 10,240	$ 10,240	$ 25,600	$ 25,600	$ 25,600	$ 10,240	$ 87,040	97,280
Transport $/t	$ 16.53	$ 16.53	$ 132,240	$ 132,240	$ 330,600	$ 330,600	$ 330,600	$ 132,240	$ 1,124,040	1,256,280
Milling 750 tpd $/t	$ 20.15	$ 20.15	$ 161,200	$ 161,200	$ 403,000	$ 403,000	$ 403,000	$ 161,200	$ 1,370,200	1,531,400
Gen + Administration $/t	$ 4.55	$ 4.55	$ 36,400	$ 36,400	$ 91,000	$ 91,000	$ 91,000	$ 36,400	$ 309,400	345,800
Total Expenses		**$ 91.59**	$ 732,752	**$ 732,752**	$ 1,831,879	$ 1,831,879	$ 1,831,879	$ 732,752	$ 6,228,388	**$ 6,961,140**
REVENUES										
Ore tonnage: st	76,000		8,000	8,000	20,000	20,000	20,000	8,000	68,000	76,000
Grade optg/t	0.17	0.170	0.17	0.17	0.17	0.17	0.17	0.17	0.17	0.17
Mill recovery	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96
Ounces produced			1,306	1,306	3,264	3,264	3,264	1,306	11,098	11,098
Gross revenue @ $CDN / oz	$ 660	$ 660	$ 861,696	$ 861,696	$ 2,154,240	$ 2,154,240	$ 2,154,240	$ 861,696	$ 7,324,416	**$ 8,186,112**
Gross profit (loss) before royalties			$ 128,944	$ 128,944	$ 322,361	$ 322,361	$ 322,361	$ 128,944	$ 1,096,028	**$ 1,224,972**
Ore NSR royalties - 10%			$ 86,170	$ 86,170	$ 215,424	$ 215,424	$ 215,424	$ 86,170	$ 732,442	818,611
Milling NSR royalty - 1%			$ 8,617	$ 8,617	$ 21,542	$ 21,542	$ 21,542	$ 8,617	$ 73,244	81,861
(Barry-1 UG) EBITDA			**$ 34,158**	**$ 34,158**	**$ 85,395**	**$ 85,395**	**$ 85,395**	**$ 34,158**	**$ 290,342**	**324,500**

Table 17.13: Operating costs & revenues from West Zone – 5 months

Cash Flow of Barry-1

The table below is showing the final Cash Flow of the exploitation of the East Pit and the West

SUMMARY of BARRY-1 CASH FLOW

BARRY: 4 m at 500 stpd, after 750 stpd

Description	Preproduction	Unit costs	Year-1	Year-2	Year-3	**TOTAL**
CAPEX		$/ st (ore)				
Mill refurbishing to 500 stpd	$ 2,618,000					
Mill upgrading to 750 stpd	$ 1,100,000					
Tailing pond study & rehabilitation	$ 2,190,000					
Mine closure provision	$ 300,000					
Total CAPEX	**$ 6,208,000**					
East Pit Production Costs						
Overburden (st)	281,000		281,000			281,000
Waste (st)	1,012,500		681,476	331,024		1,012,500
Ore mining (st)	440,000		212,500	231,500		444,000
Overburden $/t	$ 2.72	$ 1.74	$ 764,320	$ -		$ 764,320
Waste $/t	$ 3.95	$ 9.09	$ 2,691,830	$ 1,307,545		$ 3,999,375
Ore $/t	$ 5.75	$ 5.75	$ 1,221,875	$ 1,331,125		$ 2,553,000
Ore selectivity $/t	$ 1.25	$ 1.25	$ 265,625	$ 289,375		$ 555,000
Crushing $/t	$ 1.28	$ 1.28	$ 272,000	$ 296,320		$ 568,320
Transport $/t	$ 16.53	$ 16.53	$ 3,512,625	$ 3,826,695		$ 7,339,320
Gen + Administration $/t	$ 4.55	$ 4.55	$ 966,875	$ 1,053,325		
Milling 500 tpd $/t	$ 24.97	$ 24.97	$ 1,310,925			$ 1,310,925
Milling 750 tpd $/t		$ 20.15	$ 3,224,000	$ 4,664,725		$ 7,888,725
Total Expenses		$ 85.31	**$ 14,230,075**	**$ 12,769,110**		**$ 26,999,185**
REVENUES	440,000		212,500	231,500		444,000
Mill feed diluted grade	f=0.02917			0.12018		
Mill recovery	95-96%			0.96		
Ounces produced						
Gross revenue @ $CDN / oz	$ 660		**$ 16,398,783**	**$ 17,627,810**		**$ 34,026,593**
Gross profit before royalties			**$ 2,168,708**	**$ 4,858,700**		$ 7,027,408
Ore NSR royalties - 10%			$ 1,639,878	$ 1,762,781		$ 3,402,659
Milling NSR royalty - 1%			$ 163,988	$ 176,278		$ 340,266
(East Pit) EBITDA			**$ 364,841**	**$ 2,919,641**		**$ 3,284,483**

BARRY-1 West Zone - UG Mining		$/st(ore)	Year-1	Year-2	Year-3	**TOTAL**
	Tons	76,000		8000	68,000	76,000
Development: 1492 ft @ $1,067/ft	$ 1,592,500	$ 20.95		$ 167,632	$ 1,424,868	1,592,500
Production costs - Ore						
Mining $/t	$ 28.13	$ 28.13		$ 225,040	$ 1,912,840	2,137,880
Crushing $/t	$ 1.28	$ 1.28		$ 10,240	$ 87,040	97,280
Transport $/t	$ 16.53	$ 16.53		$ 132,240	$ 1,124,040	1,256,280
Milling 750 tpd $/t	$ 20.15	$ 20.15		$ 161,200	$ 1,370,200	1,531,400
Gen + Administration $/t	$ 4.55	$ 4.55		$ 36,400	$ 309,400	345,800
Total Expenses		**$ 91.59**		**$ 732,752**	$ 6,228,388	**$ 6,961,140**
REVENUES						
Ore tonnage: st	76,000			8,000	68,000	76,000
Grade optg/t	0.17	0.170		0.17	0.17	0.17
Mill recovery	0.96	0.96	0.96	0.96	0.96	0.96
Ounces produced				1,306	11,098	11,098
Gross revenue @ $CDN / oz	$ 660	$ 660		**$ 861,696**	**$ 7,324,416**	**$ 8,186,112**
Gross profit (loss) before royalties				**$ 128,944**	**$ 1,096,028**	**$ 1,224,972**
Ore NSR royalties - 10%				$ 86,170	$ 732,442	818,611
Milling NSR royalty - 1%				$ 8,617	$ 73,244	81,861
(Barry-1 UG) EBITDA				**$ 34,158**	**$ 290,342**	**324,500**

BARRY-1 : Annual Cash Flow			**$ 364,841**	**$ 2,953,799**	**$ 290,342**	**$ 3,608,982**
BARRY-1 : Cumulative Cash Flow	**-$ 6,208,000**		**-$ 5,843,159**	**-$ 2,889,360**	**-$ 2,599,018**	

Table 17.14: Barry-1 total Cash Flow

Comments about Barry-1 exploitation

As seen in the previous tables the total Barry-1 exploitation will last for 28 months and will generate a positive Cash Flow of $3,609,000. This is 58% of the preproduction capital of $6,208,000 required to refurbish and upgrade the mill at 750 short tons per day .

Based on the tonnage prorata, Barry-1 having only 516,000 tons of ore compared to Bachelor with 910,000 tons, should have only 36% of the preproduction capital,or $2,234,880. Under this arrangement, Barry-1 would show a cumulative Cash Flow of $1,374,102.

The practical effect of this situation is only theorical since the Barry-1 and Bachelor exploitations are consolidated in the economic analysis.

17.2 Bachelor Lake Mine

Mine life

The last estimate of the "retained for mining" resources of all categories is amounting to 909,400 short tonnes at a diluted grade of 0.197 opt. By assuming that the mill will run at a monthly tonnage of 20,000 st per month, the Bachelor property has a projected **mine life of over 45 months.**

17.2.1 Mining Method Selection

As a general local mining guideline the ore at Bachelor Lake Mine is sitting in ground conditions that are allowing open stoping in most areas. Both the footwalls and hanging walls are of good quality and the dip, being around 70° is favourable. The core recovery is generally excellent and confirms the good rock quality. These conditions were verified when the author and co-author visited the property in mid-May.

Three (3) mining methods were first proposed for the exploitation: the Long Hole for 52% of all stopes, the Alimak vein mining for 38% and the Shrinkage for 10%. After reviewing the last resources estimate done for 28 stopes, we observed that only four (4) of them are narrower than 8 ft, clearly indicating that the percentage of Long Hole mining will be over the 52% that we first estimated and was retained in this study. The methods are described below.

Long Hole Mining

To obtain the best dilution control possible and the lowest unit costs the modified Long Hole method is the mining method of choice, this method is basically a long hole design involving internal pillars.

Although the Golders Associates Ltd. rock mechanic study states the excellent competency of the host rock and the possibility of large opened mining areas in their September 15, 2005 report by François Chabot and Jean Sébastien Houle it is suggested that internal pillars be left in places and some additional wall support be employed in the form of cable bolting between stope pillars. These pillars do not have to be on a set pattern but can be left, when possible, in faulted, narrow or lower gold content areas.

The proposed Long Hole method is done on the assumption that sub levels will be done at a maximum vertical spacing of less than thirty (30) meters and that all blast holes will be done downward allowing the operators to verify the breakthrough of the contact holes for a better selectivity.

Another positive aspect of the modified Long Hole mining method is the cost efficiency that can be obtained through the just about unlimited stope sizes by eliminating the necessity of the expensive service raises on a set pattern due to rock mechanic constrains. Mucking raises can be cut between subs when the hauling distances starts hindering efficiency.

Alimak vein mining:

In areas where the ore lenses are only some 100ft on strike which does not warrant the expensive mine development of the Long Hole method the Alimak vein mining method could be used. It is more expensive than the Long Hole method but has the feature of being a mechanized mining method. The length of these stopes is control by the rock mechanic constrains and the length can be increased, to some extent, by cable bolting the hanging wall area at the Alimak raise site.

Shrinkage mining

Where the ore vein is very narrow 4.5 to 5ft it will be compulsory to employ the conventional shrinkage method providing the gold content justifies the extra mining costs.

17.2.2 Mining and Administration Costs

The unit mining costs are estimated to be the followings:

- **Long Hole at $61.44/st**
- **Alimak vein mining at $66.65/st**
- **Shrinkage at $83.10/st**

The average direct mining cost, including the stope preparation costs is therefore **$37.51/st** for a total direct mining average unit cost of **$66.54/st.** These costs are including the general administration cost that is fixed for all 3 methods at $**29.03/st.**

Stoping Unit Cost

The largest stopes will be mined using Long Hole method. We are reproducing below a breakdown of the unit costs of a typical Long Hole stope of 55,000 st having 6 sub-levels. This arrangement means an average of 22 meters for the down holes.

Stoping Unit Costs for Long-hole - 6 subs

Cost of a stope of 55,000 tons

Description	$ total	$/st	Distribution
Mobilisation	$ 2,200	$ 0.04	0.07%
Alimak set-up	$ 9,900	$ 0.18	0.29%
Alimak cut-out	$ 28,288	$ 0.51	0.84%
Alimak raise	$ 265,200	$ 4.82	7.85%
Mucking raise	$ 75,000	$ 1.36	2.22%
Sub-level drifts	$ 256,850	$ 4.67	7.60%
Slot raise	$ 75,000	$ 2.31	3.76%
Manway services	$ 76,500	$ 2.36	3.84%
Long-hole drilling	$ 195,250	$ 3.55	5.78%
Long-hole blasting	$ 86,350	$ 1.57	2.56%
Cable bolting	$ 24,320	$ 0.75	1.22%
Raise for pillar	$ 37,500	$ 1.16	1.88%
Sub	$ 1,132,358	$ 20.59	33.51%
Miucking	$ 199,100	$ 3.62	5.89%
Hoisting	$ 143,550	$ 2.61	4.25%
Electrical mainenance	$ 78,650	$ 1.43	2.33%
Mechanical maintenance	$ 157,300	$ 2.86	4.66%
Surface costs	$ 71,500	$ 1.30	2.12%
Sub	$ 650,100	$ 11.82	19.24%
Total before administration	$ 1,782,458	**$ 32.41**	52.75%
Administration	$ 1,596,650	$ 29.03	47.25%
Grand Total	**$ 3,379,108**	**$ 61.44**	100.00%

Table 17.15: Breakdown of the unit costs of a long-hole stope

Mining Manpower

The total direct estimated manpower for 750 st per day is 75 men, excluding the preproduction development and milling.

Salaries and Bonus

The underground miner's hourly salaries and bonus are estimated to be $45/hr plus 34% fringe benefits for a total of **$60/hr, or $120,000 per year.**

Administration Costs

The following table is a breakdown of the estimated mine administration costs.

General Mine Administration Costs (at 750 tpm)				
Description	Quantity	Salary/m	Total	$/st
Mine Management	1	$ 10,000	$ 10,000	$ 0.48
Mine accoutant	2	$ 6,000	$ 12,000	$ 0.57
Clerk	2	$ 5,000	$ 10,000	$ 0.48
Receptionist and secretary	1	$ 4,000	$ 4,000	$ 0.19
Nurse	1	$ 6,000	$ 6,000	$ 0.29
Safety and personnel	1	$ 7,000	$ 7,000	$ 0.33
Buyer and store	2	$ 6,000	$ 12,000	$ 0.57
Security			$ -	$ -
Sub-Total	10	$ 44,000	$ 61,000	$ 2.90
Engineering		$/month		
Chief engineer	1	$ 8,000	$ 8,000	$ 0.38
Chief geologist	1	$ 8,000	$ 8,000	$ 0.38
Mine engineer	1	$ 7,000	$ 7,000	$ 0.33
Mine suveyor and technician	2	$ 5,000	$ 10,000	$ 0.48
Draft person	1	$ 4,500	$ 4,500	$ 0.21
Mine geologist	2	$ 7,000	$ 14,000	$ 0.67
Samplers	2	$ 4,500	$ 9,000	$ 0.43
Sub-Total	10		$ 60,500	$ 2.88
Surface Services		$/day		
Loader and truck operator	2	$ 160	$ 320	$ 0.43
Surface maintenance	1	$ 150	$ 160	$ 0.29
Dry and lamps	2	$ 150	$ 150	$ 0.60
Sub-Total	5			$ 1.32
Mining		$/month		
Mine Superintendent	1	$ 9,000	$ 9,000	$ 0.43
Assistant Mine super	1	$ 8,500	$ 8,500	$ 0.40
Mine Captain	2	$ 8,000	$ 8,500	$ 0.76
Mine Shift boss	6	$ 7,000	$ 42,000	$ 2.00
Chief mechanic	1	$ 7,000	$ 7,000	$ 0.33
Chief electrician	1	$ 7,000	$ 7,000	$ 0.33
Sub-Total	12		$ 82,000	$ 3.90
Total	**36**			**$ 11.01**
Benefits at 34%				$ 3.74
Total				**$ 14.75**
General	$/month			
Power heating/monthy	$ 85,000			$ 4.05
Inssurance, QMMA, Mine rescue: $/month	$ 22,000			$ 1.05
Tel. office material: $/month	$ 25,000			$ 1.19
Board and room/monthly	$ 168,000			$ 8.00
Grand Total				**$ 29.03**

Table 17.16: Breakdown of general mine administrative costs

17.2.3 Ongoing Capital Cost Estimates

Before resuming production in the mine the owners of Bachelor have to install a larger hoist, sink the shaft for 675 feet, excavate ore and waste passes and do all the preproduction developments on five (5) levels. A summary of these estimated costs is presented below.

Bachelor Mine CAPEX

Description	CDN$
Hoist installation	$ 1,020,750
Service building and Warehouse	$ 600,000
Compressors and generators repairs	$ 385,000
Shaft sinking, ore & waste passes	$ 9,196,131
Camp	$ 600,000
Explosive & detonators magazines	$ 85,600
Level developments (12-13-14-15-16)	$ 3,031,248
Equipment acquisition	$ 2,343,000
Ventilation study	$ 16,050
Mine closure provision	$ 1,500,000
Total	$ 18,777,779

Table 17.17: Bachelor preproduction capital costs

Shaft deepening and primary services

The estimated costs of required installations and workings related to the shaft deepening, ore and waste passes, ventilation and escape way raises, water sump and electrical installations are summarized in the following table.

Shaft deepening & primary services

Description	$ Total
Actual shaft inspection and rehab.	$ 215,070
Shaft sinking to 16th level: 675 feet	$ 5,421,337
Ventilation raises	$ 954,750
Ore & waste passes	$ 1,449,374
Water sump	$ 85,600
Electrical installation	$ 1,070,000
Total	**$ 9,196,131**

Table 17.18: Bachelor shaft deepening and services costs

The estimated cost of $5,421,337 is a total lump sum price proposed by a contractor, it includes the mobilization, the sinking itself, the level stations, the skip loading station, a spill pocket, piping, etc.

The actual estimate is based on the sinking of 675 feet to give access to four new levels: 13-14-15 and 16.

Breakdown of proposed level development costs

Primary developments that are including the cross-cuts, the hauling drifts with draw points, the undercut drifts and all required services, mainly the ore and waste dumping stations, explosives and detonators magazines and material storage areas are summarized in the following table. The secondary developments, or stope preparation costs are included in the direct mining production costs.

Estimation of preproduction level developments

Description	Levels					Total	$/ft	$ Total
	12	13	14	15	16			
Main X-cut		600	700	800	840	2,940	$356	$ 1,046,640
Haulage drift	400	400	400	400	300	1,900	$356	$ 676,400
Undercut drift	400	400	400	400	300	1,900	$356	$ 676,400
Draw points		326	282	326	100	1,034	$356	$ 368,104
Services	146	148	148	148	148	738	$356	$ 262,728
Total ft	946	1,874	1,930	2,074	1,688	8,512	$356	$ 3,030,272
$/ft	$ 392	$ 350	$ 356	$ 346	$ 355			
$ Total	**$370,568**	**$656,392**	**$687,192**	**$717,992**	**$599,104**			**$ 3,031,248**

Table 17.19: Summary of Bachelor shat deepening and services costs

Underground equipment list and costs

Underground Equipment Summary

Description	Quantity	Cost $
Locos, batteries, chargers	28	$ 592,900
Mucking machines & Cavos	21	$ 577,500
Mucking cars - 5 ton	28	$ 154,000
Jacklegs and stopers	45	$ 198,000
Longtom	1	$ 77,000
Main pumps - all types	4	$ 104,500
Radio communication - lot	1	$ 33,000
Miscellaneous tools - lot	1	$ 441,100
Inventory - lot	1	$ 165,000
Total		**$ 2,343,000**

Table 17.20: Mining equipment costs

17.2.4 Operating costs and revenues summary

The Bachelor's capital costs are spread over a period of 18 months before mine production and are considered as ongoing capital costs. The development of the mine will be done while the Barry-1 ore is being treated at the Bachelor concentrator. The summary of the operating costs and revenues is shown in the following tables.

BACHELOR LAKE MINE 750 st/day		Years														Year - 1
		Months		Month - 1	Month - 2	Month - 3	Month - 4	Month - 5	Month - 6	Month - 7	Month - 8	Month - 9	Month - 10	Month - 11	Month - 12	
		Tons														
CAPEX (CDN$)	CAPEX		$/ st of ore													
Hoist & Headframe	*$ 1,020,750*															$ 1,020,750
Compressors & Generators	*$ 385,000*															$ 385,000
Explosive Magazines	*$ 85,600*															$ 85,600
Camp	*$ 600,000*															$ 600,000
Service buildings & warehouse	*$ 600,000*															$ 600,000
Sub-total	***$ 2,691,350***	8 months						$ 336,500	$ 336,500	$ 336,500	$ 336,500	$ 336,500	$ 336,500	$ 336,500	$ 336,500	**$ 2,691,350**
Shaft sinking to 16th level	*$ 9,196,131*															
Preproduction & Equipment Costs	*$ 5,390,298*															
Sub-total	***$ 14,586,429***	16 months													$ 911,650	$ 911,650
Mine closure provision	*$ 1,500,000*	5 months														
Total on going CAPEX	***$ 18,777,779***							$ 336,500	$ 336,500	$ 336,500	$ 336,500	$ 336,500	$ 336,500	$ 336,500	$ 1,248,150	**$ 3,603,000**
Production Costs																
Zones: Main + B + AW (st)		910,000														
Stope development & mining: $/st		$ 37.51	$ 37.51													
Ore drilling definition - $/st		$ 2.50	$ 2.50													
General Administration - $/st		$ 29.03	$ 29.03													
Milling cost - $/st		$ 20.15	$ 20.15													
Sub-total			$ 89.19													
Total Production Costs																
Total CAPEX & Production Costs								**$ 336,500**	**$ 336,500**	**$ 336,500**	**$ 336,500**	**$ 336,500**	**$ 336,500**	**$ 336,500**	**$ 1,248,150**	**$ 3,603,000**
Revenues																
Milling rate - st/month	910,000	20,000	20,000													
Diluted average grade of all Zones - opt		0.197	0.197													
Milled recovery - %		0.96	0.96													
Ounces produced																
Gross revenue at $CDN/oz		$ 660.00	$ 660.00													
Gross profit (loss) before royalties								-$ 336,500	-$ 336,500	-$ 336,500	-$ 336,500	-$ 336,500	-$ 336,500	-$ 336,500	-$ 1,248,150	-$ 3,603,000
Ore NSR royalty - 3 %																
Milling NSR royalty - 1%																
(BACHELOR LAKE MINE) EBITDA*								-$ 336,500	-$ 336,500	-$ 336,500	-$ 336,500	-$ 336,500	-$ 336,500	-$ 336,500	-$ 1,248,150	**-$ 3,603,000**

Table 17.21: Operating costs & revenues from Bachelor property, months 1 to 12

BACHELOR LAKE MINE 750 st/day		Years														Year - 2
		Months		Month - 13	Month - 14	Month - 15	Month - 16	Month - 17	Month - 18	Month - 19	Month - 20	Month - 21	Month - 22	Month - 23	Month - 24	
		Tons														
CAPEX (CDN$)	CAPEX		$/ st of ore													
Hoist & Headframe	*$ 1,020,750*															
Compressors & Generators	*$ 385,000*															
Explosive Magazines	*$ 85,600*															
Camp	*$ 600,000*															
Service buildings & warehouse	*$ 600,000*															
Sub-total	***$ 2,691,350***	8 months														
Shaft sinking to 16th level	*$ 9,196,131*															
Preproduction & Equipment Costs	*$ 5,390,298*															
Sub-total	***$ 14,586,429***	16 months		$ 911,650	$ 911,650	$ 911,650	$ 911,650	$ 911,650	$ 911,650	$ 911,650	$ 911,650	$ 911,650	$ 911,650	$ 911,650	$ 911,650	$ 10,939,800
Mine closure provision	*$ 1,500,000*	5 months														
Total on going CAPEX	***$ 18,777,779***			$ 911,650	$ 911,650	$ 911,650	$ 911,650	$ 911,650	$ 911,650	$ 911,650	$ 911,650	$ 911,650	$ 911,650	$ 911,650	$ 911,650	**$ 10,939,800**
Production Costs																
Zones: Main + B + AW (st)		910,000														
Stope development & mining: $/st		$ 37.51	$ 37.51													
Ore drilling definition - $/st		$ 2.50	$ 2.50													
General Administration - $/st		$ 29.03	$ 29.03													
Milling cost - $/st		$ 20.15	$ 20.15													
Sub-total			$ 89.19													
Total Production Costs																
Total CAPEX & Production Costs				**$ 911,650**	**$ 911,650**	**$ 911,650**	**$ 911,650**	**$ 911,650**	**$ 911,650**	**$ 911,650**	**$ 911,650**	**$ 911,650**	**$ 911,650**	**$ 911,650**	**$ 911,650**	**$ 10,939,800**
Revenues																
Milling rate - st/month	910,000	20,000	20,000													
Diluted average grade of all Zones - opt		0.197	0.197													
Milled recovery - %		0.96	0.96													
Ounces produced																
Gross revenue at $CDN/oz		$ 660.00	$ 660.00													0
Gross profit (loss) before royalties				-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	-$ 10,939,800
Ore NSR royalty - 3 %																
Milling NSR royalty - 1%																
(BACHELOR LAKE MINE) EBITDA*				-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	-$ 911,650	**-$ 10,939,800**

Table 17.22: Operating costs & revenues from Bachelor property, months 13 to 24

BACHELOR LAKE MINE 750 st/day		Years														Year - 3
		Months		Month - 25	Month - 26	Month - 27	Month - 28	Month - 29	Month - 30	Month - 31	Month - 32	Month - 33	Month - 34	Month - 35	Month - 36	
		Tons												5,893	15,000	20,893
CAPEX (CDN$)	CAPEX		$/ st of ore													
Hoist & Headframe	*$ 1,020,750*															
Compressors & Generators	*$ 385,000*															
Explosive Magazines	*$ 85,600*															
Camp	*$ 600,000*															
Service buildings & warehouse	*$ 600,000*															
Sub-total	***$ 2,691,350***	8 months														
Shaft sinking to 16th level	*$ 9,196,131*															
Preproduction & Equipment Costs	*$ 5,390,298*															
Sub-total	***$ 14,586,429***	16 months		$ 911,650	$ 911,650	$ 911,650										$ 2,734,950
Mine closure provision	*$ 1,500,000*	5 months														
Total on going CAPEX	***$ 18,777,779***			$ 911,650	$ 911,650	$ 911,650										**$ 2,734,950**
Production Costs																
Zones: Main + B + AW (st)		910,000					12,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	172,000
Stope development & mining: $/st		$ 37.51	$ 37.51													
Ore drilling definition - $/st		$ 2.50	$ 2.50													
General Administration - $/st		$ 29.03	$ 29.03													
Milling cost - $/st		$ 20.15	$ 20.15													
Sub-total			$ 89.19				$ 89.19	$ 89.19	$ 89.19	$ 89.19	$ 89.19	$ 89.19	$ 89.19	$ 89.19	$ 89.19	$ 89.19
Total Production Costs							**$ 1,070,280**	**$ 1,783,800**	**$ 1,783,800**	**$ 1,783,800**	**$ 1,783,800**	**$ 1,783,800**	**$ 1,783,800**	**$ 1,783,800**	**$ 1,783,800**	**$ 15,340,680**
Total CAPEX & Production Costs				**$ 911,650**	**$ 911,650**	**$ 911,650**	**$ 1,070,280**	**$ 1,783,800**	**$ 1,783,800**	**$ 1,783,800**	**$ 1,783,800**	**$ 1,783,800**	**$ 1,783,800**	**$ 1,783,800**	**$ 1,783,800**	**$ 18,075,630**
Revenues																
Milling rate - st/month	910,000	20,000	20,000				12,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	172,000
Diluted average grade of all Zones - opt		0.197	0.197				0.197	0.197	0.197	0.197	0.197	0.197	0.197	0.197	0.197	0.197
Milled recovery - %		0.96	0.96				0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96
Ounces produced							2,269	3,782	3,782	3,782	3,782	3,782	3,782	3,782	3,782	32,529
Gross revenue at $CDN/oz		$ 660.00	$ 660.00				$ 1,497,830	$ 2,496,384	$ 2,496,384	$ 2,496,384	$ 2,496,384	$ 2,496,384	$ 2,496,384	$ 2,496,384	$ 2,496,384	**$ 21,468,902**
Gross profit (loss) before royalties				-$ 911,650	-$ 911,650	-$ 911,650	$ 427,550	$ 712,584	$ 712,584	$ 712,584	$ 712,584	$ 712,584	$ 712,584	$ 712,584	$ 712,584	$ 3,393,272
Ore NSR royalty - 3 %							$ 44,935	$ 74,892	$ 74,892	$ 74,892	$ 74,892	$ 74,892	$ 74,892	$ 74,892	$ 74,892	$ 644,067
Milling NSR royalty - 1%							$ 14,978	$ 24,964	$ 24,964	$ 24,964	$ 24,964	$ 24,964	$ 24,964	$ 24,964	$ 24,964	$ 214,689
(BACHELOR LAKE MINE) EBITDA*				-$ 911,650	-$ 911,650	-$ 911,650	$ 367,637	$ 612,729	$ 612,729	$ 612,729	$ 612,729	$ 612,729	$ 612,729	$ 612,729	$ 612,729	**$ 2,534,516**
* EBITDA = Estimated benfit before tax, depreciation and amortization																Year - 3
Bachelor - Annual Cash Flow																$ 2,534,516
Bachelor - Cumulative Cash Flow																-$ 12,008,284

Table 17.23: Operating costs & revenues from Bachelor property, months 25 to 36

BACHELOR LAKE MINE 750 st/day		Years		Year - 1	Year - 2	Year - 3	Year-4	Year-5	Year-6	Year-7	Total
		Months									
		Tons				20,893	180,000	180,000	145,318		526,211
CAPEX (CDN$)	CAPEX		$/ st of ore								
Hoist & Headframe	*$ 1,020,750*			$ 1,020,750							
Compressors & Generators	*$ 385,000*			$ 385,000							
Explosive Magazines	*$ 85,600*			$ 85,600							
Camp	*$ 600,000*			$ 600,000							
Service buildings & warehouse	*$ 600,000*			$ 600,000							
Sub-total	***$ 2,691,350***	8 months		**$ 2,691,350**							**$ 2,691,350**
Shaft sinking to 16th level	*$ 9,196,131*										
Preproduction & Equipment Costs	*$ 5,390,298*										
Sub-total	***$ 14,586,429***	16 months		$ 911,650	$ 10,939,800	$ 2,734,950					**$ 14,586,400**
Mine closure provision	*$ 1,500,000*	5 months							$ 1,500,000		$ 1,500,000
Total on going CAPEX	***$ 18,777,779***			**$ 3,603,000**	**$ 10,939,800**	**$ 2,734,950**			**$ 1,500,000**		**$ 18,777,750**
Production Costs											
Zones: Main + B + AW (st)		910,000				172,000	240,000	240,000	240,000	18,000	910,000
Stope development & mining: $/st		$ 37.51	$ 37.51								
Ore drilling definition - $/st		$ 2.50	$ 2.50								
General Administration - $/st		$ 29.03	$ 29.03								
Milling cost - $/st		$ 20.15	$ 20.15								
Sub-total			$ 89.19			$ 89.19	$ 89.19	$ 89.19	$ 89.19	$ 89.19	$ 89.19
Total Production Costs						**$ 15,340,680**	**$ 21,405,600**	**$ 21,405,600**	**$ 21,405,600**	**$ 1,605,420**	**$ 81,162,900**
Total CAPEX & Production Costs				**$ 3,603,000**	**$ 10,939,800**	**$ 18,075,630**	**$ 21,405,600**	**$ 21,405,600**	**$ 22,905,600**	**$ 1,605,420**	**$ 99,940,650**
Revenues											
Milling rate - st/month	910,000	20,000	20,000			172,000	240,000	240,000	240,000	18,000	910,000
Diluted average grade of all Zones - opt		0.197	0.197			0.197	0.197	0.197	0.197	0.197	0.197
Milled recovery - %		0.96	0.96			0.96	0.96	0.96	0.96	0.96	0.96
Ounces produced						32,529	45,389	45,389	45,389	3,404	172,099
Gross revenue at $CDN/oz		$ 660.00	$ 660.00		0	**$ 21,468,902**	**$ 29,956,608**	**$ 29,956,608**	**$ 29,956,608**	**$ 2,246,746**	**$ 113,585,472**
Gross profit (loss) before royalties				-$ 3,603,000	-$ 10,939,800	$ 3,393,272	$ 8,551,008	$ 8,551,008	$ 7,051,008	$ 641,326	$ 13,644,822
Ore NSR royalty - 3 %						$ 644,067	$ 898,698	$ 898,698	$ 898,698	$ 67,402	$ 3,407,564
Milling NSR royalty - 1%						$ 214,689	$ 299,566	$ 299,566	$ 299,566	$ 22,467	$ 1,135,855
(BACHELOR LAKE MINE) EBITDA*				**-$ 3,603,000**	**-$ 10,939,800**	**$ 2,534,516**	**$ 7,352,744**	**$ 7,352,744**	**$ 5,852,744**	**$ 551,456**	**$ 9,101,403**
* EBITDA = Estimated benfit before tax, depreciation and amortization					Year - 2	Year - 3	Year-4	Year-5	Year-6	Year-7	Total
Bachelor - Annual Cash Flow				-$ 3,603,000	-$ 10,939,800	$ 2,534,516	$ 7,352,744	$ 7,352,744	$ 5,852,744	$ 551,456	**$ 9,101,403**
Bachelor - Cumulative Cash Flow				-$ 3,603,000	-$ 14,542,800	-$ 12,008,284	-$ 4,655,540	$ 2,697,204	$ 8,549,947	$ 9,101,403	
Discounted Cash Flow at 10%											**$ 2,761,811**

Table 17.24: Bachelor Lake operating costs & revenues: total production.

17.3 Consolidated costs and revenues of Barry-1 and Bachelor Mine

When adding the results of the exploitation of Barry-1 and Bachelor lake Mine, the final cumulative cash flow before tax, interest, depreciation and amortization at the end of the operations is $6,502,385, as seen in the following table.

SUMMARY of BARRY-1 & BACHELOR LAKE MINE CASH FLOW

BARRY: 4 months at 500 stpd, after 750 stpd										
Description	Preproduction	Unit costs	Year-1	Year-2	Year-3	Year-4	Year-5	Year-6	Year-7	**TOTAL**
CAPEX		$/ st (ore)								
Total CAPEX	**$ 6,208,000**									
East Pit Production Costs										
Total Expenses		$ 85.31	$ 11,349,895	**$ 12,769,110**						**$ 24,119,005**
REVENUES	440,000		212,500	231,500						444,000
Ounces produced			24,847							
Gross revenue @ $CDN / oz	$ 660		16,398,783	**$ 17,627,810**						**$ 34,026,593**
Gross profit before royalties			2,168,708	**$ 4,858,700**						$ 9,907,588
Ore NSR royalties - 10%			1,639,878	$ 1,762,781						$ 3,402,659
Milling NSR royalty - 1%			163,988	$ 176,278						$ 340,266
(East Pit) EBITDA			364,841	**$ 2,919,641**						**$ 3,284,483**
BARRY-1 West Zone - UG Mining		$/st(ore)	Year-1	Year-2	Year-3					**TOTAL**
Total Expenses		**$ 91.59**		**$ 732,752**	$ 6,228,388					**$ 6,961,140**
REVENUES										
Ounces produced				1,306	11,098					11,098
Gross revenue @ $CDN / oz	$ 660	$ 660		**$ 861,696**	**$ 7,324,416**					**$ 8,186,112**
Gross profit (loss) before royalties				**$ 128,944**	**$ 1,096,028**					**$ 1,224,972**
Ore NSR royalties - 10%				$ 86,170	$ 732,442					$ 818,611
Milling NSR royalty - 1%				$ 8,617	$ 73,244					$ 81,861
(Barry-1 UG) EBITDA				**$ 34,158**	**$ 290,342**					**324,500**
BARRY-1 : Annual Cash Flow			**$ 364,841**	**$ 2,953,799**	**$ 290,342**					**$ 3,608,982**
BARRY-1 : Cumulative Cash Flow	**-$ 6,208,000**		**-$ 5,843,159**	**-$ 2,889,360**	**-$ 2,599,018**					
BACHELOR LAKE MINE 750 st/day			Year - 1	Year - 2	Year - 3	Year-4	Year-5	Year-6	Year-7	Total
					172,000	240,000	240,000	240,000	18,000	910,000
CAPEX (CDN$)	CAPEX	$/ st of ore								
Surface Sub-total	***$ 2,691,350***	8 months	**$ 2,691,350**							**$ 2,691,350**
Mine Development Sub-total	***$ 14,586,429***	16 months	$ 911,650	$ 10,939,800	$ 2,734,950					**$ 14,586,400**
Mine closure provision	*$ 1,500,000*	5 months						$ 1,500,000		$ 1,500,000
Total on going CAPEX	***$ 18,777,779***		**$ 3,603,000**	**$ 10,939,800**	**$ 2,734,950**			**$ 1,500,000**		**$ 18,777,750**
Production Costs										
Zones: Main + B + AW (st)		910,000			172,000	240,000	240,000	240,000	18,000	910,000
Sub-total		$ 89.19			$ 89.19	$ 89.19	$ 89.19	$ 89.19	$ 89.19	$ 89.19
Total Production Costs					**$ 15,340,680**	**$ 21,405,600**	**$ 21,405,600**	**$ 21,405,600**	**$ 1,605,420**	**$ 81,162,900**
Total CAPEX & Production Costs			**$ 3,603,000**	**$ 10,939,800**	**$ 18,075,630**	**$ 21,405,600**	**$ 21,405,600**	**$ 22,905,600**	**$ 1,605,420**	**$ 99,940,650**
Revenues										
Milling rate - st/month	910,000	20,000			172,000	240,000	240,000	240,000	18,000	910,000
Diluted average grade of all Zones - opt		0.197			0.197	0.197	0.197	0.197	0.197	0.197
Milled recovery - %		0.96			0.96	0.96	0.96	0.96	0.96	0.96
Ounces produced					32,529	45,389	45,389	45,389	3,404	172,099
Gross revenue at $CDN/oz		$ 660.00		0	**$ 21,468,902**	**$ 29,956,608**	**$ 29,956,608**	**$ 29,956,608**	**$ 2,246,746**	**$ 113,585,472**
Gross profit (loss) before royalties			-$ 3,603,000	-$ 10,939,800	$ 3,393,272	$ 8,551,008	$ 8,551,008	$ 7,051,008	$ 641,326	$ 13,644,822
Ore NSR royalty - 3 %					$ 644,067	$ 898,698	$ 898,698	$ 898,698	$ 67,402	$ 3,407,564
Milling NSR royalty - 1%					$ 214,689	$ 299,566	$ 299,566	$ 299,566	$ 22,467	$ 1,135,855
(BACHELOR LAKE MINE) EBITDA*			**-$ 3,603,000**	**-$ 10,939,800**	**$ 2,534,516**	**$ 7,352,744**	**$ 7,352,744**	**$ 5,852,744**	**$ 551,456**	**$ 9,101,403**
Bachelor - Cumulative Cash Flow			-$ 3,603,000	-$ 14,542,800	-$ 12,008,284	-$ 4,655,540	$ 2,697,204	$ 8,549,947	$ 9,101,403	
METANOR ANNUAL CASH FLOW			**-$ 3,238,159**	**-$ 7,986,001**	**$ 2,824,858**	**$ 7,352,744**	**$ 7,352,744**	**$ 5,852,744**	**$ 551,456**	**$ 12,710,385**
METANOR CUMULATIVE CASH FLOW	-$ 6,208,000		-$ 9,446,159	-$ 17,432,160	-$ 14,607,301	-$ 7,254,558	$ 98,186	$ 5,950,930	**$ 6,502,385**	
Discounted Metanor Cash Flow at 10%										$ 137,442
Discounted Metanor Cash Flow at 7.5%										$ 1,353,258
Discounted Metanor Cash Flow at 5.0%										$ 2,789,439

Table 17.25: Total Cash Flow of Barry-1 and Bachelor properties

17.4 Economic Analysis Results

As shown in the above table the exploitation of the two properties is generating a **Net Cash Flow of $6,502,385 for the expected 73 months of operation**. This Cash Flow is shown as EBITDA, (Estimated Benefit Before Tax Depreciation & Amortization), in other words this is a Pre-Tax Undiscounted Cash Flow.

The situation at the end of this period will leave Metanor Resources Inc with two properties that most likely will not be exhausted, plus a running concentrator.

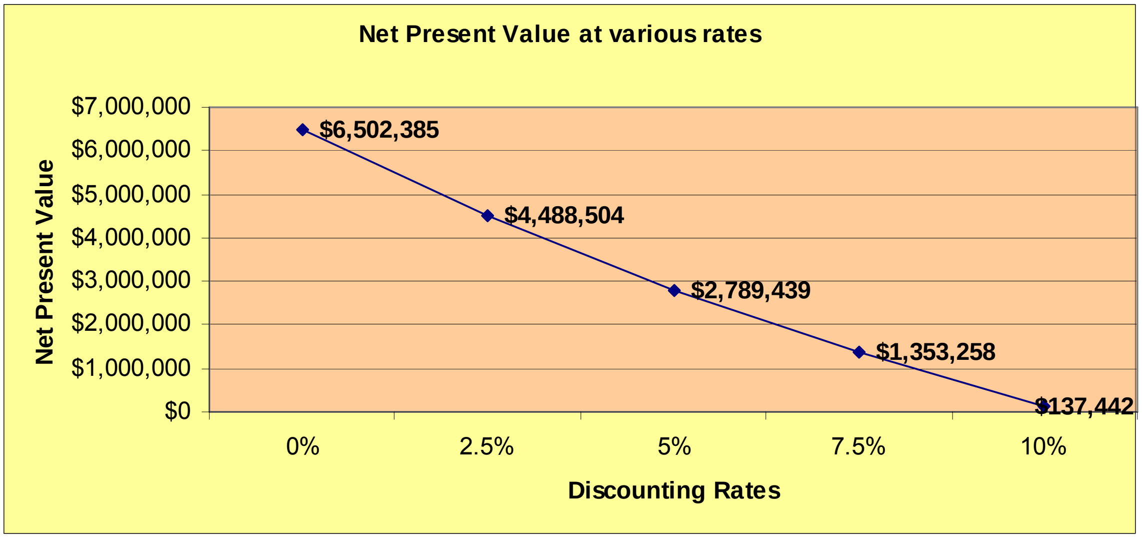
It is also important to note that no salvage values have been given to the assets in the Cash Flow estimate.

17.5 Discounted Cash Flow

The effect of discounting the Base Case result is illustrated in the following table.

Undiscounted Cash Flow	$	6,502,385
Discounted at 2.5 %	$	4,488,504
Discounted at 5.0 %	$	2,789,439
Discounted at 7.5%	$	1,353,258
Discounted at 10%	$	137,442

The same variation is shown in graphic mode



Estimated Mine Life

From the above economic results, the total mining life of both operations **is seventy three (73) months.**

Both properties are showing good possibilities of extending their mine life, but as more geological detailed information is needed, it is not possible to evaluate the order of magnitude of this extension.

18. Other relevant data and information (Item 20)

18.1 Barry I Property

There is no other relevant data and information for this report.

18.2 Bachelor Lake Property

Other relevant data and information

There is no other relevant data and information for this report.

19. Interpretation and conclusions (Item 21)

19.1 Barry-1 Property

The resources reported in this document are compliant with current standards as outlined in the National Instrument 43-101.

Geostat can confirm that most of the gold content of the database are corroborated by check analyses and that no statistical bias was observed.

Specific gravity measurement of the core was performed and it ranged from 2.79 to 2.99, the average value for the mineralized rock used in the calculation being 2.80.

The absence of a detailed surveyed topography, the incertitude regarding the position of the drill holes not surveyed and the difficulty to calculate an anisotropic variogram does not allow us to declare measured resources at this stage.

The resources estimated by block modelling (inverse distance), 3 metres along the all the directions (north, south, elevation) could be established as follows for the Barry I Main Zone Area project:

Total resources inverse distance (No cut-off) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**Density**	**Au (g/t)**	**Oz Au**
Indicated	**415,000**	**148,000**	**2.8**	**4.05**	**54,000**
Total	**415,000**	**148,000**	**2.8**	**4.05**	**54,000**
Inferred	1,102,000	394,000	2.8	3.78	133,800
Total resources inverse distance (Cut-off of 1 g/t) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**2.80**	**4.00**	**36,100**
Indicated	**415,000**	**148,000**	**2.8**	**4.05**	**54,000**
Total	**415,000**	**148,000**	**2.8**	**4.05**	**54,000**
Inferred	1,102,000	394,000	2.8	3.78	133,800
Total resources inverse distance (Cut-off of 2 g/t) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**Density**	**Au (g/t)**	**Oz Au**
Indicated	**385,000**	**138,000**	**2.8**	**4.23**	**52,300**
Total	**385,000**	**138,000**	**2.8**	**4.23**	**52,300**
Inferred	966,000	345,000	2.8	4.07	126,600
Total resources inverse distance (Cut-off of 3 g/t) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**Density**	**Au (g/t)**	**Oz Au**
Indicated	**277,000**	**99,000**	**2.8**	**4.89**	**43,600**
Total	**277,000**	**99,000**	**2.8**	**4.89**	**43,600**
Inferred	690,000	246,000	2.8	4.70	104,300
Total resources inverse distance (Cut-off of 4 g/t) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**Density**	**Au (g/t)**	**Oz Au**
Indicated	**174,000**	**62,000**	**2.8**	**5.74**	**32,100**
Total	**174,000**	**62,000**	**2.8**	**5.74**	**32,100**
Inferred	404,000	144,000	2.8	5.59	72,600
Total resources inverse distance (Cut-off of 5 g/t) Rounded					
Category	**Tonnage (mt)**	**Volume (m3)**	**Density**	**Au (g/t)**	**Oz Au**
Indicated	**109,000**	**39,000**	**2.8**	**6.49**	**22,800**
Total	**109,000**	**39,000**	**2.8**	**6.49**	**22,800**
Inferred	225,000	80,000	2.8	6.46	46,700

The Barry I project contains enough indicated and inferred gold resources to justify additional work on the west side of the Main Zone, in the 43 and 45 zones and exploration work on the rest of the property.

19.2 Bachelor Lake Property

Extracted from InnoExplo 43-101 report of December 2005

Property status and potential:

The property consists of two blocks of claims: (i) Bachelor claims which are already registered 100% to Metanor and part of the BLJV; and (ii) Hewfran claims where the BLJV has the right to acquire a 100% from Aur. Acquisition of the Hewfran claims are underway and are conditional to a work commitment of $ 1.6M. Exploration activity (drilling) has been carried out on the Hewfran claims and a portion of the 2005 underground drilling program has also been done on these claims. The claims are in good standing and there are no land claim issues or ownership disputes pending with the property. There is no environmental issue, and exploration activities (including mine dewatering) are being carried out according to regulations set out by the Government of Quebec.

The Bachelor Lake mine site includes surface infrastructures, hoist room, shaft house, mill (500 tons per day), tailing pond, and core shack. The infrastructure is generally in good condition but will require modifications and rehabilitation if the BLJV needs to work underground for future exploration program. The BLJV is currently keeping the mine dewatered which is currently accessible to the 12th Level (shaft sump is at a depth of 562.66 m (1 846')).

The Bachelor property (Bachelor and Hewfran claims) and the adjacent property (MJL and Hansen claims) cover two past producing mines: (i) Bachelor Lake gold mine (Prod. 131 029 ounces of refined gold from 869 412 tonnes of ore grading 4.70 g/t Au) and (ii) Coniagas base metals mine (Prod. 718 465 tonnes of ore grading 10.77% Zn, 1.0% Pb, and 183 g/t Ag). The compilation of mineral occurrences on the property clearly indicated that the project has additional surface potential for both type of mineral deposit: lode gold mineralization (Bachelor-type and lode gold mineralization) and polymetallic (Zn-Cu-Au-Ag) massive sulphide mineralization (felsic volcanoclastic rocks, zinc showings no.1 and no.2; Coniagas marker horizon; hole 19501-52). The surface exploration potential has been preliminary compiled and some target areas are illustrated on the Compilation Map (1:10 000) in Appendix X (refer to the area of original gold discovery east of the O'Brien granodiorite; induced polarization (IP) anomaly within O'Brien granodiorite about 305 m (1 000') south of the eastward projection of the "Main" zone; hole 82-11 area, approximately 305 m (1 000') south of the "Main" zone; area of drill holes 45, 51, 53 approximately 457 m (1 500') south southwest of the Bachelor shaft; the southwest contact of the O'Brien granodiorite; alteration and low gold values in trenches 122 m (400') north of the shaft; and pyritic shear zone 760 m (2 500') north of the shaft).

Results of the 2005 drilling program:

In 2005, a major underground exploration drilling program has been initiated by Halo and completed by the BLJV. This drilling program (13 345 m in 69 holes) had a significant impact on the geological understanding of the deposit. Highlighted geological features from the 2005 drilling program showed:

1. The continuity of the "Main Zone" has been extended substantially (over a total strike length of 450 m (1 500') from the Bachelor Lake to the "East" zone onto the Hewfran claims; hence,

opening the potential for additional resources. The “Main” zone has been intersected on the Hewfran claims, some 107 m (350’) west of the old Hewfran / Bachelor Lake property boundary and it is still open westward as drilling has not been performed west of the 850’W section. Furthermore, the mineralization is known to exist as far as Hewfran “West” zone, some 300 m (1 000’) further West.

2. Dilational widening of the mineralized zones appeared at the junction of several major structural features, and created the potential for high productivity, lower cost mining methods. This bulging is described as follows:

    a. At the junction of the “B” zone” with the “A” zone on the sections 0’E, 50’W, 100’W, 150’W, 200’W and 250’W: This thicker zone has a potential strike length of 45 m, a down-dip length of 35 m and an estimated true width of 10 m. This is also the area where the increased presence of visible gold has been noted.

    b. At the junction of the “B” zone with the “Main” zone (hole BLM12-04 has an estimated true width intersection of 12 m).

3. The O’Brien granite contact opened at depth. Hole 12-116 drilled towards the O’Brien late granitic stock has put the granite-volcanic contact further east, opening the possibility of extending the mineralized zone to the east and, in addition, opens new areas for additional resources.

4. The “Main” zone has been documented in the footwall of the Waconichi fault (Big Wac fault).

5. The "gap zone" (between the T1 fault and "A" Zone) has the potential to positively impact the project. This area was previously interpreted to contain granite dykes but the latest drill results (such as hole 12-57) showed that the "Main" and "B" zones continue through this area. Furthermore, this area appears to be the locus for late stage gold emplacement which is associated with strong hematite alterations, silica-flooding with the occurrence of visible gold.

Significant assay results were obtained during the 2005 drilling program and out of the sixty-nine (69) holes drilled, forty (40) holes have intercepted composite grades over a cut-off of 3.43 g/t Au on a minimum horizontal width of 1.5 m or higher. From these, eight (8) holes intercepted a mineralized interval having a horizontal width over 6 m.

<u>2005 Mineral Resource estimates (NI 43-101 compliant):</u>

Results from the 2005 geological interpretation and data validation have allowed to include Hewfran East and West areas in the 2005 Mineral Resource estimates. The area covered by historical resources of Hewfran East and West areas are now part of the Bachelor NI 43-101 compliant Mineral Resource estimates.

Another impact of the 2005 underground drilling program and new geological interpretation on the Mineral Resource estimates has been by upgrading resource category for the Bachelor Lake, Hewfran East and West areas (88 131 ounces were added in the Indicated category). The Mineral Resource estimates (compliant to NI 43-10) on the property are now of:

- **Measured Resources**: 192 594 tonnes grading at 8.80 g/t Au (54 504 oz Au);
- **Indicated Resources**: 648 997 tonnes grading at 7.49 g/t Au (156 352 oz Au);
- **Inferred Resources**: 426 148 tonnes grading at 6.52 g/t Au (89 366 oz Au).

The Bachelor Lake property remains an advanced-stage exploration project but with more than 2/3 of the Mineral Resources within Measured and Indicated categories (210 857 ounces of gold: 841 591 tonnes grading at 7.79 g/t Au) and considering underground access. Measured resources are already accessible from the actual underground infrastructure (at Bachelor and Hewfran), although some blocks may be locked up in pillars. The indicated resources are also located below the footprint of the existing underground development. Engineering studies (scoping and feasibility studies) have to be carried out to determine the economic potential of both the Measured and Indicated resources.

The property has a significant potential for the discovery of additional gold mineralization located in the vicinity of the Bachelor Mine and surrounding deposits. There is potential for more Inferred resources which total at 89 366 ounces of gold: 426 148 tonnes grading at 6.52 g/t Au. Additional underground exploration under the Bachelor Lake and East Zone areas may increase the Mineral Resources especially down-plunge extension at depth and at the site of structural junctions (local widening of the zones). The new geological interpretation indicated that the projection of the “Main” zone remained untested on the Hewfran claims and that the “A West” and “B West” (Hewfran) are probably connected with the “A” and “B” zones (Bachelor). Further exploration work is required on the “A West” and “B West” zones. These two “zones” are defined by fewer and wider spaced drill holes. Additional drilling will likely be required to verify grade continuity for the Hewfran “zones”. However, geologically the zones demonstrate similar characteristics to their respective counterparts on the Bachelor Lake side.

Comparison with previous Mineral Resources estimates:

The comparison between previous resources compliant to NI 43-101 with the new results for the property indicates a significant increase in both tonnage and contained gold for all categories (Table 19.1). For this particular comparison, note that the Hewfran historical resources were not considered. Measured resource increased in tonnage by 4% (7 846 short tons), its grade did not changed and the total ounces increased by 4% (2 000 ounces). Indicated resource increased in tonnage by 230% (498 715 short tons), its grade decreased by 31% (-0.096 oz/t Au), and the total ounces increased by 129% (88 131 ounces). Inferred resource increased by 83% (213 465 short tons), its grade decreased by 37% (-0.113 oz/t Au), and the total ounces increased by 15% (11 542 ounces).

It is difficult to compare historical resources results with the new Mineral Resource estimates and to make conclusions on these comparisons mainly because the key assumptions and basic parameters used in the historic estimate are not known (such as minimum width, cut-off grade, capping, radius of influence). However, there is an apparent lack of Inferred resources when the historical resources of Hewfran are included in the previous inferred category for comparison purposes. Note that, following the National Instrument 43-101, the Hewfran historical resources should not be added to previous Inferred category. All historical resource estimates for Hewfran were not compliant with National Instrument 43-101. Consequently, they are neither in compliance with this current standard nor with the CIM Committee on Ore Reserves and that appropriate actions were not fulfilled by Qualified Person in order to ascertain the classification of resources and that no action should be taken on the strength of historical estimate.

Table 19.1: Comparison with previous resource estimate (Imperial Units).

BACHELOR LAKE RESOURCES SUMMARY (IMPERIAL UNITS)

NI 43-101 COMPLIANT RESOURCES ESTIMATE BEFORE 2005 U/G DIAMOND DRILLING

		Zone	Measured			Indicated			Measured + Indicated			Inferred		
			Short Tons (t)	Grade (oz/t)	Oz Gold	Short Tons (t)	Grade (oz/t)	Oz Gold	Short Tons (t)	Grade (oz/t)	Oz Gold	Short Tons (t)	Grade (oz/t)	Oz Gold
Bachelor		"A" Zone	11 722	0.270	3 165	15 395	0.301	4 634	27 117	0.288	7 799			
		"B" Zone				78 913	0.339	26 752	78 913	0.339	26 752	71 990	0.354	25 484
		"Main" Zone	192 732	0.256	49 339	122 377	0.301	36 835	315 109	0.273	86 175	184 295	0.284	52 340
		BACHELOR TOTAL	**204 454**	**0.257**	**52 504**	**216 685**	**0.315**	**68 221**	**421 139**	**0.287**	**120 725**	**256 285**	**0.304**	**77 824**
Hewfran	East	"A" Zone	No resource compliant to 43-101(1)									No resource compliant to 43-101		
		"Main" Zone												
	West	"A West" Zone	No resource compliant to 43-101(2)									No resource compliant to 43-101		
		"B West" Zone												

Note: Historical and preliminary geological resources: (1) 68 000 t @ 0.259 oz/t by Buro (2005) and (2) 450 000 t @ 0.17 oz/t reported by Rougerie (1989).

NEW RESOURCES ESTIMATE - SEPTEMBER 2005 (revised October 2005)

		Zone	Measured			Indicated			Measured + Indicated			Inferred		
			Short Tons (t)	Grade (oz/t)	Oz Gold	Short Tons (t)	Grade (oz/t)	Oz Gold	Short Tons (t)	Grade (oz/t)	Oz Gold	Short Tons (t)	Grade (oz/t)	Oz Gold
BACHELOR		"A" Zone	4 100	0.185	759	58 700	0.195	11 447	62 800	0.194	12 205	20 600	0.171	3 523
		"B" Zone				225 700	0.224	50 557	225 700	0.224	50 557	86 300	0.173	14 930
		"C" Zone				18 600	0.209	3 887	18 600	0.209	3 887	35 250	0.215	7 579
		"Main" Zone	192 000	0.259	49 728	210 600	0.230	48 438	402 600	0.244	98 166	86 600	0.220	19 052
		BACHELOR TOTAL	**196 100**	**0.257**	**50 487**	**513 600**	**0.223**	**114 329**	**709 700**	**0.232**	**164 815**	**228 750**	**0.197**	**45 083**
HEWFRAN	East	"A" Zone				1 900	0.181	344	1 900	0.181	344	1 400	0.382	535
		"B" Zone				2 200	0.196	431	2 200	0.196	431	600	0.151	91
		"Main" Zone	16 200	0.248	4 018	87 600	0.210	18 396	103 800	0.216	22 414	32 100	0.188	6 035
	West	"A West" Zone				85 200	0.220	18 744	85 200	0.220	18 744	144 000	0.194	27 936
		"B West" Zone				24 900	0.165	4 109	24 900	0.165	4 109	62 900	0.154	9 687
		HEWFRAN TOTAL	**16 200**	**0.248**	**4 018**	**201 800**	**0.208**	**42 024**	**218 000**	**0.211**	**46 042**	**241 000**	**0.184**	**44 283**
BACHELOR & HEWFRAN TOTAL			**212 300**	**0.257**	**54 504**	**715 400**	**0.219**	**156 352**	**927 700**	**0.227**	**210 857**	**469 750**	**0.190**	**89 366**
		DIFFERENCE	***7 846***	***0.000***	***2 000***	***498 715***	***-0.096***	***88 131***	***506 561***	***-0.059***	***90 132***	***213 465***	***-0.113***	***11 542***
		(%)	***(+ 4%)***	***(0%)***	***(+ 4%)***	***(+ 230%)***	***(-31%)***	***(+ 129%)***	***(+ 120%)***	***(-21%)***	***(+ 75%)***	***(+83%)***	***(-37%)***	***(+15%)***

By comparison, the Hewfran historical resources (East Zone: 68 000 short tons at 0.259 oz/t Au (Buro, 2005); West Zone: 450 000 short tons at 0.17 oz/t Au (Rougerie, 1989)) were added in the previous Inferred category. Using this comparison, the Inferred resources have decreased by 119% (-304 535 short tons), its grade decreased by 10% (-0.032 oz/t Au) and the contained ounces decreased by 106% (-82 570 oz). This apparent lack of Inferred resources is not the result of a lack in exploration potential but the result of a lack of information at depth but also the result of upgrade resource blocks from Inferred to Indicated category. Further exploration on the property may significantly increase the Inferred resources.

For a global comparison, the sum of Measured, Indicated and Inferred resources was made: it can be stated that the overall tonnage has increased by 17% (202 026 short tons), that the grade decreased by 12% (-0.030 oz/t Au) and that the contained gold increased by 3% (7 561 oz Au) (note that NI 43-101 does not allow the sum of Measured, Indicated and Inferred resources which has been made here only for global comparison and that no action should be taken on the strength of these sums).

The main impact of the 2005 underground drilling program has been in upgrading resource category (88131 ounces were added in the indicated category), although the overall grade of the resources has decreased. This may be explained in part by the methodology used for the estimate (block model generally having lower grade than extrapolation method (such as polygonal estimate using a longitudinal), but also by the fact that more data is added to the model, more the final grade will be closer to the true average grade of the deposit.

Comments on grade reliability of the 2005 resource estimates:

The quality of a resource estimate is only as good as the data used to create the estimate. Following the 2005 drilling program, the QA/QC analysis (Horvath and Carrier, 2005) indicated that the assay results were accurate and that the assay protocol was adequate and must be used in future program at Bachelor. The performance of the laboratory during the 2005 drilling campaign was good. No contamination was identified during processing and the accuracy of results, as indicated by the certified reference standards used to monitor accuracy, both internally and externally to the laboratory, was deemed very good.

Precision (i.e. reproducibility) of pulp duplicate sample assays demonstrate a fair level of precision of 12% which is quite good for a gold deposit. The precision of the coarse duplicate was not that good and has a 90% indicated error. While this type of error may not result in any global change in resource estimation, if locally assays are imprecise, than locally block model grade estimates will also be imprecise. While the global results may remain unchanged, poor mine planning and ore development will result from imprecise assays and grade estimation (Horvath and Carrier, 2005).

Data acquired in 2005 represented twenty-three percent (23%) of the entire resource database (3 555 samples from 69 holes in 2005 versus 15 192 assay results from 394 holes for the entire database). Conclusion of the QA/QC analysis indicated a similar geostatiscal behavior between the 2005 and historic assay results which led to an increased level of confidence on the entire database. The database used for the new Mineral Resource estimates included historical and new drill holes: (i) Bachelor Lake (63 new holes and validated historical holes; i.e. location, assay certificate, and check assays); Hewfran East (6 new holes and validated historical holes; i.e. location, assay certificate, check assays and 1 confirmation hole); and West areas (validated historical holes; i.e. location, assay certificate, and check assays). Geostatistical analysis (univariate statistic and variography) done in

2005 had allowed to establish parameters and key assumptions for the 2005 Mineral Resource estimate which relies upon 3 684 composite intervals of 0.75 m.

Results obtained from data verification for both the historic and the new data were good. Check sampling results showed that the precision around the cut-off grade was really good and block misclassification should not be a problem. Precision on higher grade sample (check sampling results) was good. Results of the confirmation hole B05-117A done within the Hewfran East area have been positive for both locations of the "Main" zone and its grade.

Comments on resource blocks location:

The 2005 drilling program was entirely done with azimuth oriented holes executed from two underground drill stations located on the 12th Level of the Bachelor mine. Azimuth holes having only a slight deviation (1° or 2°) can have a significant impact on the exact "x"-"y"-"z" location of a mineralized zone. However, during the 2005 drilling program, all holes were systematically surveyed down-the-hole using a Flex-It instrument. All collar locations were surveyed by a professional surveyor (J.L. Corriveau Surveyor). Location of the mineralized zones obtained from the 2005 drilling program and from the historical data is considered to be good.

Comments following actual resources estimate by Geostat

Please refer to the title 16 (Item 19)

20. Recommendations (Item 22)

- Following the positive results of the estimated cash-flow, Geostat is recommending to the owners to advance the properties in the direction of a commercial production.
- Geostat also recommends that Metanor should proceeds with a pre-feasibility study in order to confirm our recommendation.
- Before proceeding to the next pre-feasibility phase, Geostat recommends that Metanor should prepare the followings:

At the Barry property

- Better evaluate the full economic benefit of the treatment of a bulk sample
- Define the cost saving resulting from the ore crushing at Barry-1 before sending it to Bachelor; in our cash-flow no cost reduction has been applied.
- Reassess the economic impact of the royalties, specially those of the Barry-1 property
- Perform additional fill-in drilling to better define the known mineralized zones
- Explore the vincinities of the proposed open-pit to avoid stockpiling waste or overburden over possible mineralized areas
- Complete the survey of the topography and all the drill holes that have not already been surveyed in the area of the Barry-1 property
- Realize a detailed new description of some of the old drill core to better understand the correlation between the mineralized envelopes and the geology of Barry.
- Continue the exploration around the proposed East Pit and West Zone where the presence of mineralized zones could add resources to the exiting ones.

The costs of the recommended works at Barry works is summary below

Barry Property

Description			Cost
Ore definition at the Bulk Sample area - lump sum			$35,000
Exploration under the stockpiling areas	500 m	$120/m	$60,000
Resources in-fill drilling	2,000 m	$120/m	$240,000
General expansing drilling	2 000 m	$120/m	$240,000
Pre-feasibility study - lump sum			$35,000
Bulk sample exploitation			$3,106,000
Total			**$3,716,000**

At the Bachelor property

- Proceed to replace the existing hoist
- Initiate the shaft deepening to give acccess to the ore portion that is below the twelve level.
- Proceeds to the development of the proposed ore undercuts to have a full understanding of the of the geology and to assay the mineralized zones
- Realize an infill drilling program estimated at 20,000 ft

The total of all these workings is shown below.

Bachelor Property

Description			Cost
Hoist & headframe repair			$ 1,020,750
Shaft sinking with services - 650 ft			$ 9,196,150
Excavation of undercuts (50% of all level developments)	4,000 ft	$356/ft	$ 1,424,000
In-fill drilling, c/w assays	20,000 ft	$35/ft	$ 700,000
Pre-feasibility study - lump sum			$ 35,000
Total			**$ 12,375,900**

The following are other recommendations that were included in the InnovExplo report of December 2005

Structural geology and metallogenic study:

Investigate: why, when, how and where the mineralization was emplaced by completing geological and structural geology studies of the "Bachelor mineralized corridor". Determination of the relationship between gold grades, width of mineralization, alteration haloes, late to syn-tectonic intrusive rocks and structural geological features (relationship with the Wedding-Lamarck regional fault and the O'Brien stock, geometry of dilational zones, etc…). This may be accomplished through different studies.

Resource model:

Develop a geostatistical resource model implementing economic mining parameters based on verified data from previous exploration programs and Phase I drilling.

Regional compilation review and target generation:

Compile regional assessment data and identify targets for follow-up geophysical surveying and/or exploration drilling. Identify the vectors (geophysical, geochemistry, structural, lithologies, alteration) for lode gold and polymetallic VHMS mineralization.

VHMS polymetallic (Zn-Cu-Au-Ag) potential:

In a near future, the potential for VHMS polymetallic (Zn-Cu-Au-Ag) should be re-assessed on the property. This potential is illustrated by surface showings (i.e. Agar #2), geophysical anomalies and drill hole 19501-52 (which returned a significant Zn-Cu-Ag interception) which confirmed the east extension of the Coniagas marker horizon on the property (Refer to Compilation map in Appendix X). For VHMS potential, it has been proposed by Gagnon (1995) to systematically sample all felsic volcanic rocks on the property for whole-rock geochemistry (primary features and alterations) and to cover the property with new geophysical surveys.

Geophysical coverage of the property:

The last geophysical survey performed on the property in 1985 was carried out on only a portion of the Bachelor claims and the last survey on the Hewfran claims was in 1984. No recent geophysical surveys were accomplished and Innovexplo recommends a uniform and complete survey of the new Bachelor property.

Information program:

Develop an information program to facilitate communications with the Waswanipi Cree Nation and other public interest groups.

Cementing of underground drill holes:

It is also important to note that, as mentioned in the Drilling section (Item 13), no holes were cemented during the last 2005 underground drilling program neither were the old drill holes. Innovexplo recommends before any further underground works to **cement all drill holes**.

QA/QC recommendations for future program:

Horvath and Carrier (2005) stated that two major points should be noted for future QA/QC sampling program:

*1)*The unacceptably poor performance for the **field blank samples** (i.e. frequency of assays greater than 3 x the detection limit) suggests that source material used for the standard is not suitable. No characterization studies have been completed for the field blank material (mean grade and expected levels of deviation).

[… Characterization of a field blank normally includes isolating up to several tonnes of the proposed field blank material selected from a barren source local to the project. The material should be devoid of alteration, mineralization and veining. A minimum of 10 samples should be randomly collected from the several tonnes. Each sample weighing 2-4 kg (typical sample size) should be forwarded to a commercial laboratory for preparation of 1 kg pulp samples as per the standard preparation protocols used in the 2005 program. The 1 kg pulps should be split into 3 equal sized samples and packaged in pulp bags, so that each original sample has 3 approximately 330 g pulp bags. One of the pulp bags for each sample should be retained by the original lab for duplicate ppb level Au determinations by geochemical methods for each of the samples (i.e. 10x2=20 assays). The other two pulp bags for each sample should be forwarded to separate independent commercial laboratories for similar duplicate ppb level geochemical Au determinations for each sample. The resulting 60 assays from this "round robin" characterization of the field blank can be used to determine accepted mean grades and levels of deviation for the field blank in sampling and assaying programs. Normally one of the laboratories selected is also the laboratory that will be selected for the ddh sampling/assaying program. …]

2) A program of **field duplicates** sampling should be introduced in future programs. The results should be evaluated early to determine if a minimum sample interval or larger whole core samples might be required to obtain more precise results and to determine the overall precision of final sample assays.

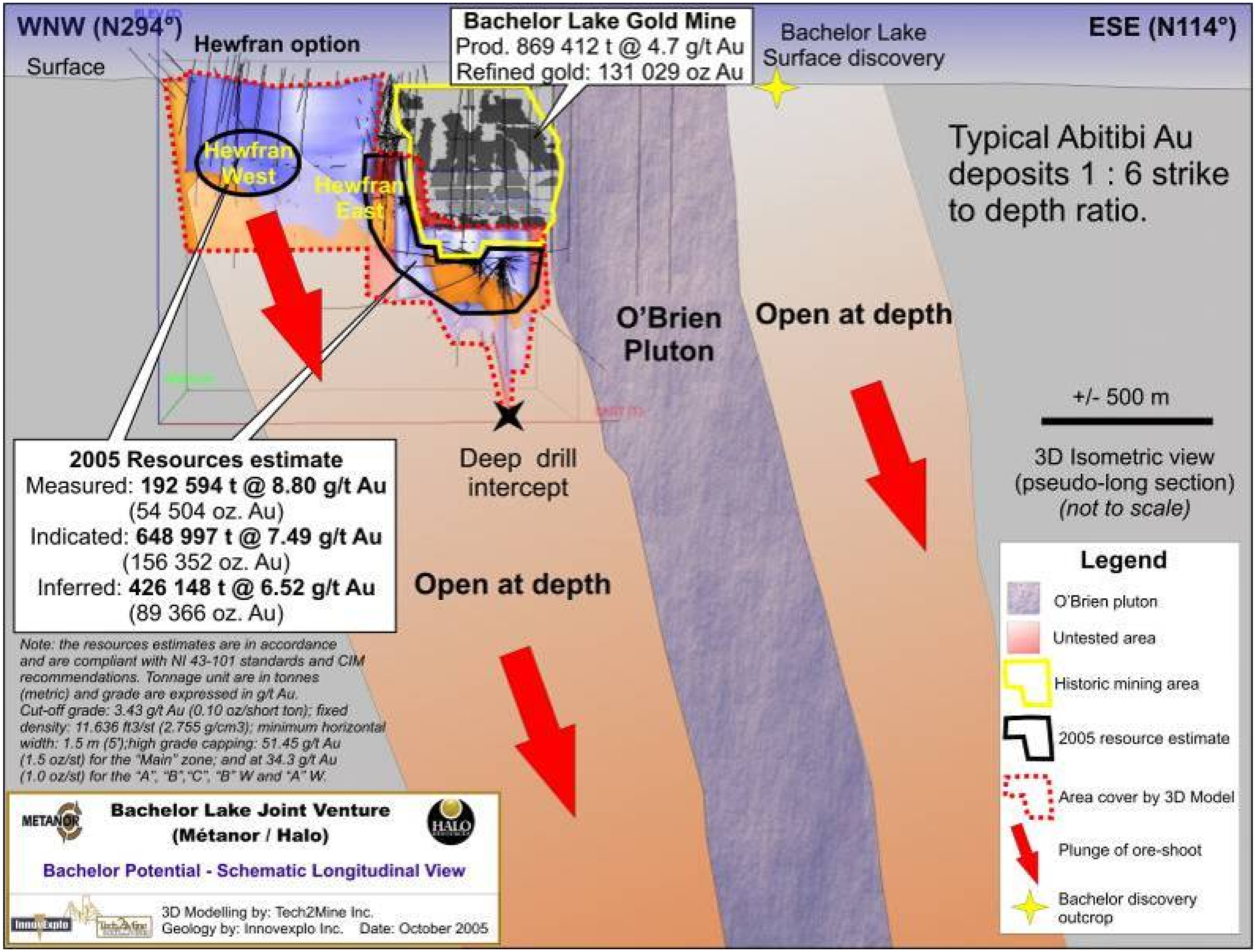


Figure 20.1 - Bachelor potential – Schematic longitudinal view

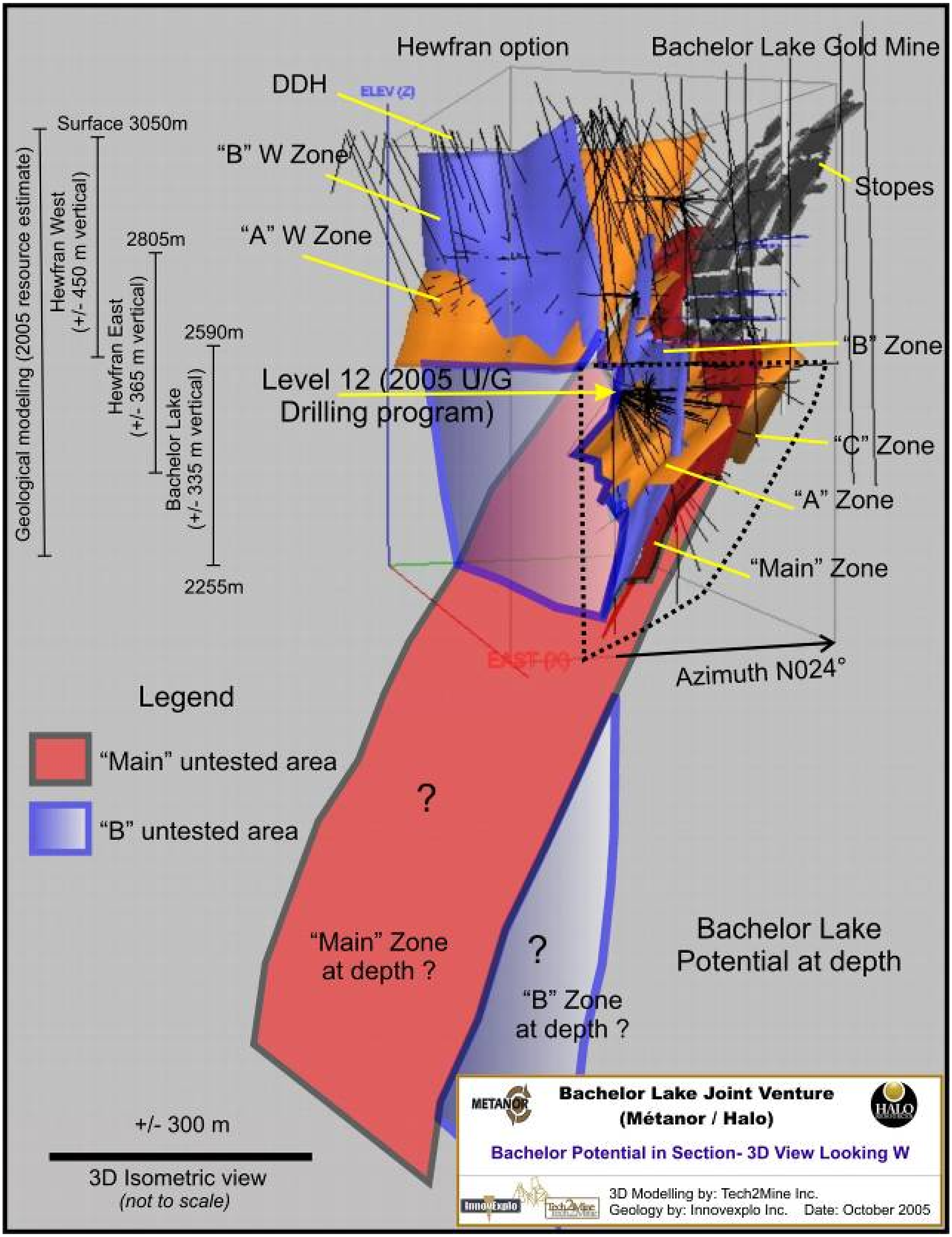


Figure 20.2 - Bachelor potential – Schematic 3D view

21. References (Item 23)

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INTERNET SITE:

Metanor Resources: http://www.Metanor.ca

22. Date and Signature of the report (Item 24)

PRELIMINARY ASSESSMENT REPORT

ON THE BARRY-1 AND THE BACHELOR LAKE PROPERTIES

Prepared for

Metanor Resources Inc.
V'al-d'Or, Québec, Canada

Gaston Gagnon

Gaston Gagnon
15918

Gaston Gagnon, eng

Signed on August 16, 2007

Systèmes Géostat International, Inc.
10 boul. de la Seigneurie est
Blainville, Québec Canada J7C 3V5

Yann Camus

Yann Camus
125443

Yann Camus, eng

Signed on August 16, 2007

Systèmes Géostat International, Inc.
10 boul. de la Seigneurie est
Blainville, Québec Canada J7C 3V5

Gilbert Rousseau

20268

Gilbert Rousseau, eng

Signed on August 16, 2007

Systèmes Géostat International, Inc.
10 boul. de la Seigneurie est
Blainville, Québec Canada J7C 3V5

23. Qualification Certificates of Authors

CERTIFICATE OF AUTHOR

1. I am working for:
   Systèmes Géostat International Inc.(Geostat)
   10 boul de la Seigneurie Est, suite 203
   Blainville (Québec) J7C 3V5

2. I graduated with a mining engineer degree from Laval University in 1964.

3. I am a member of the l'Ordre des Ingénieurs du Québec (#15918)

4. I have worked as a mining engineer for Geostat since June 2006 and mainly as an underground mining engineer before that date.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association, as defined in NI43-101 and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI43-101.

6. I am responsible for the preparation of all items except sections 7 and 19 of the technical report titled "Preliminary Assessment" of Barry-1 and Bachelor Lake Properties of Metanor Resources Inc. of Val d'Or, (Quebec) Canada.

7. I have not 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 of 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 the instrument and form.

10. I consent to the public 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 16th Day of August 2007

(signed)

Gaston Gagnon, Eng

CERTIFICATE OF AUTHOR

1. I am working as a consultant for:

Systèmes Géostat International Inc.(Geostat)
10 boul de la Seigneurie Est, suite 203
Blainville (Québec) J7C 3V5

2. I graduated with a mining engineer degree from the Ecole Polytechnique of the University of Montreal in 1969.

3. I am a member of the l'Ordre des Ingénieurs du Québec (#20288)

4. I have worked as a mining engineer since my graduation, being mainly involved in metallurgy and milling.

5. I visited the Bachelor property on May 14th of 2007 for the preparation of this technical report.

6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association, as defined in NI43-101 and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI43-101.

7. I am responsible for the preparation of item 7 (infrastructures' - concentrator) of the technical report titled "Preliminary Assessment" of Barry-1 and Bachelor Lake Properties of Metanor Resources Inc. of Val d'Or, (Quebec) Canada.

8. I have not had prior involvement with the property that is the subject of the Technical Report.

9. I am not aware of any material fact or material change with respect of 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 the instrument and form.

11. I consent to the public 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 17th Day of August 2007

signed

Gilbert Rousseau, Eng

Qualification Certificate of Yann Camus, Eng.

CERTIFICATE OF AUTHOR

a) Yann Camus, Eng.
6285 Chambord
Montréal (Québec) H2G 3B8
Email : ycamus@geostat.com
I work for:
Systèmes Géostat International Inc.
10, boul. de la Seigneurie Est, Suite 203
Blainville (Québec) J7C 3V5

b) This certificate applies to the report titled “Preliminary Assessment” of Barry-1 and Bachelor Lake Resources of Metanor Resources.

c) I have worked as a geological engineer for over 6 years with Geostat and did mineral resource estimations since then. I graduated with a geological engineer degree from the “École Polytechnique de Montréal” in 2000. I am a member of the Ordre des ingénieurs du Québec. 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 purpose of NI 43 -101.

d) I visited the Barry property on may 14th of 2007 and the Bachelor property on May 15th of 2007 for the preparation of this technical report.

e) I am responsible for the preparation of the item 19.

f) I certify that there is no circumstance that could interfere with my judgment regarding the preparation of this technical report.

g) I have had no prior involvement with the properties studied in this report.

h) 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.

i) To my best knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 17th Day of August 2007

________signed_____________________

Yann Camus, Eng

INNOVEXPLO APPENDICES

Appendix I : Abbreviations and Conversion Factors

TERMS AND DEFINITIONS

APPENDIX I - ABBREVIATIONS, CONVERSION FACTORS, TERMS AND DEFINITIONS

Abbreviations Used and Conversion factors

For the compilation of this technical report, information from different sources was taken. Either the Bachelor Lake or the Hewfran previous properties were previously worked using the imperial system (various grids (N-S local grid, A Vein grid, N024° Bachelor grid), units (Imperial and Metric) and scale (1”=20’, 1”=40’, 1”=100’, 1:10 000). In order to maintain uniformity and to allow comparison, the new information from the 2005 drilling program has been plotted using the imperial system (oz/t Au and 1”/20’ scale sections and plans).

The Bachelor Lake grid oriented at 24°E to the geographic North was used during drilling and project evaluation. Grid orientation at 24°E has been provided by Yves Buro (Consultant to Halo) at the beginning of the 2005 underground drilling program. In this grid, the Bachelor Lake is located at 0,0 at the elevation of 3054 m (10 020 ft). All historical and new results have been merged into one database using the Bachelor Lake mine grid as the reference grid (grid in feet and oriented at N024°). However, measurements on historical plans indicated that the grid orientation is not perfectly at 24°E to the geographic North, our tests indicated that the grid is probably at 23.5°E rather than at 24°E.

Units in this report are metric unless otherwise specified and historical data are repeated in the imperial system, which are written between brackets to avoid any confusion. Precious metal content is reported in gram of metal per metric ton (g/t Au or Ag) except otherwise stated. Tonnage figures are dry metric tons unless otherwise stated. The ounces are in Troy ounces.

Abbreviations

°C	Degrees Celsius
g	Grams
ha	Hectares
kg	Kilograms
km	Kilometre
masl	Meters above sea level
m	Meters
mm	Millimetre
'	Foot
"	Inch

oz	Troy ounces
oz/t	Ounces per short tons
g/t	Grams per metric tons
ppb	Part per billion
ppm	Part per million
st	Short tons
t	Metric tonnes
$	Canadian dollars

Conversion Factors Used for Measurements

1 inch →	25.4	mm	1 mm →	0.3937	inch
1 foot →	0.3048	m	1 m →	3.28083	foot
1 mile →	1.6093	km	1 km →	0.6214	mile
1 acre →	0.4047	ha	1 ha →	2.4711	acre
1 acre →	4046.825	m^2	1 ha →	0.01	km^2
1 oz →	31.1035	g	1 g →	0.03215	oz
1 oz →	1.0971	oz (avdp)	oz (avdp) →	0.9115	1 oz
1 oz/t →	34.2857	g/t	g/t →	0.0291	1 oz/t
1 pound (avdp) (lb) →	0.454	kg	kg →	2.2046	Lb (avdp)
1 pound (avdp) (lb) →	1.215	pound (troy)	kg →	2.6792	pound (troy)
1 ton (short) →	0.9071	t	t →	1.1023	1 ton (short)

Terms and Definitions

Abbreviation	Reference
Innovexplo	InnovExplo Inc. – Geological Services
Metanor or MTO	Metanor Resources Inc.
Halo or HLO	Halo Resources Ltd.
BLJV	Bachelor Lake Joint Venture
MRNFP	Ministère des Ressources naturelles, de la Faune et des Parcs du Québec
BLGM	Bachelor Lake Gold Mines
DDH	Diamond drill holes
UDD	Underground diamond drilling
SDD	Surface diamond drilling
Espalau	Espalau Mining
Ced-Or	Ced-Or corporation
GéoNova	Exploration GéoNova Inc.
CMAC	CMAC Mining Contractor Inc. (formerly Talpa Mining Contractor Inc.)
U/G	Underground

Bachelor Lake Geological Legend

Code	Définition française	English definition
AGG	Agglomérat	Agglomerate
am	Amygdalaire	Amygdaloid
aph	Aphanitique	Aphanitic
AZ	Altération Hydrothermale indéterminée	Undetermined hydrothermal alteration
bo	Boudiné	Boudinage
BQ	Brèches de lave	Brecciated flow
br	Brèchifié	Brecciated
cb	Carbonate	Carbonate
cg	Grain grossier	Coarse grained
CK	Coulées massives, fines	Massive flow, fine grained
cln	Chloritisation	Chloritization
cnr	Carotte non récupérée	Core not recovered (CNR)
CO	Coulées coussinées	Pillow flows
cp	Chalcopyrite	Chalcopyrite
CW	Coulées massives, grenues	Massive flow, coarse grained
ds	Disséminé	Disseminated
EP-CL	Alteration Épidote-chlorite	Epidote-chlorite alteration
EP-CL-CB-HM	Alteration Épidote-chlorite-carb-hém	Epidote-chlorite- carbonate-hematite alteration
epn	Épidotisation	Epidotization
fa	Fracture	Fracture
fe	Fracturé	Fractured
fg	Grain fin	Fine grained
FL	Faille indéterminée	Undetermined fault
fln	Foliation	Foliation
flu	Fluorine	Fluorite
fol	Folié	Foliated

Code	**Définition française**	**English definition**
FT	Faille Type transverse	Transverse fault
FT1	Faille	Fault T1
FT2	Faille	Fault T2
FT3	Faille	Fault T3
gl	Galène	Galena
HM-	Alteration hématite-silicification	Hematite-silica alteration
I	Roche Magmatique intrusive	Magmatic intrusive rocks
I1B	Granite	Granite
I1C	Granodiorite	Granodiorite
I1D	Aplite	Aplite
I1G	Pegmatite	Pegmatite
I1H	Granophyre (Porphyre granitique à grain fin)	Granophyre
I2D	Syénite	Syenite
I2G	Diorite	Diorite
I3	Mafique intrusive	Mafic intrusive
I3A	Gabbro	Gabbro
I3B	Diabase	Diabase
I4	Dyke Ultramafique	Ultramafic dyke
I4A	Hornblendite	Hornblendite
I4B	Pyroxénite	Pyroxenite
I4O	Lamprophyre	Lamprophyre
ir	Irrégulier	Irregular
lo	Localement	Locally
M	Roche Métamorphique	Metamorphic rock
M1	Gneiss	Gneiss
M16	Amphibolite	Amphibolite
M8	Schiste	Schist
mass	Massif	Massive
mg	Grain moyen	Medium grained
mm	Monogénique	Monogenic
mod	Moyen, modéré	Moderate
MZ	Minéralisation de type Indéterminé	Undetermined mineralized-type
MZA	Minéralisation de type veine A	A vein- type mineralization
MZB	Minéralisation de type veine B	B vein- type mineralization
MZP	Minéralisation de type veine Principale	Main vein- type mineralization
ob	Oblique	Oblic
pa	Parallèle, en direction	Parallel
pe	Perpendiculaire	Perpendicular
pg	Pegmatitique	Pegmatitic
pm	Polygénique	Polygenic
po	Pyrrhotine	Pyrrhotite
por	Porphyritique	Porphyritic
py	Pyrite	Pyrite
qcb	Quartz-carbonate	Quartz-carbonate
qz	Quartz	Quartz
ru	Rubané	Banded
S	Roche Sédimentaire	Sedimentary rock
S10	Chert	Chert
S10D	Chert sulfureux	Sulfurous chert
S10J	Jaspe	Jasper

Code	**Définition française**	**English definition**
sa	Subparallèle	Sub-parallel
sc	Schisteux	Schistous
sd	Saccharoïde	Saccharoid
se	Sub-perpendiculaire	Sub-perpendicular
SH	Cisaillement	Shear
shd	Cisaillé	Sheared
SHW	Cisaillement de type Waconichi	Waconichi-type shear
sin	Silicification	Silicification
str	Fort	Strong
stw	Stockwerk	Stockwerk
TC	Tuf Cherteux	Cherty tuff
TCB	Brèche de faille (cohésive)	Fault breccia (cohesion)
TCM	Mylonite	Mylonite
TCT	Pseudo-tachylite	Pseudo-tachylite
TCU	Cataclasite, indéterminée	Undetermined cataclasite
TD	Tuf À cendres	Ash tuff
TG	Tuf Graphiteux	Graphitic tuff
TI	Tuf Lithique	Bedded tuff
TL	Tuf À lapillis	Lapillis tuff
TLB	Brèche de faille (non cohésive, meuble)	Fault breccia (loose)
TLG	Argile de faille	Gouge
TM	Tuf À blocs	Block tuff
to	Tourmaline	Tourmaline
tr	Traces	Traces
TU	Tuf	Tuff
TX	Tuf À cristaux	Crystal tuff
TY	Tuf À lapillis et blocs	Lapillis and block tuff
V	R magmatiques volc	Volcanic magmatic rock
V1	Rhyolite	Rhyolite
V2	Andésite	Andesite
V3	Basalte	Basalte
va	Variolitique	Variolitic
vd	Veine- dilatation/tension	Dilatational/tension vein
ve	Vésiculé	Vesicule
vld	Veinule- dilatation/tension	Dilatational/tension veinlets
vls	Veinule - cisaillement	Shear-veinlets
VNA	Veine A	A Vein (A Zone)
VNB	Veine B	B Vein (B Zone)
VNP	Veine principale	Main Zone
vs	Veine cisaillement	Shear Vein (Zone)
vs	Veine - cisaillement	Shear - Vein (Zone)
WA1	Cisaillement de type Waconichi 1	Waconichi 1-type shear
WA2	Cisaillement de type Waconichi 2	Waconichi 2-type shear
WA3	Cisaillement de type Waconichi 3	Waconichi 3-type shear
WA4	Cisaillement de type Waconichi 4	Waconichi 4-type shear
wk	Faible - léger	Weak

CLASSIFICATION – ROCKS

Rocks	Origin	Code
Magmatic	intrusives	I
	volcanics	V
Metamorphic		M
Tectonite		T
Sedimentary		S

Magmatic rocks

Composition	Code
Felsic	1
Intermediate	2
Mafic	3
Ultramafic	4

Intrusives rocks

	Detailled classification	Code
Felsics	Granite	I1B
	Granodiorite	I1C
	Granophyre	I1H
	Aplite	I1D
	Pegmatite	I1G
Intermediate	Syenite	I2D
	Diorite	I2G
Mafics	Mafic intrusive	I3
	Gabbro	I3A
	Diabase	I3B
Ultramafic	Undeterminate	I4
	Hornblendite	I4A
	Pyroxenite	I4B
	Lamprophyre	I4O

Volcanic rocks

Classification	Code
Rhyolite	V1
Andesite	V2
Basalte	V3

Pyroclastic rocks		**EX.: V2TL**
Tuff	Undifferentiated	TU
	Crystal	TX
	Lithic	TI
	Lapillis	TL
	Blocks	TM
	Lapillis and blocks	TY
	Ash	TD
	Cherty	TC
	Graphitic	TG

Effusive Rocks

	Code
Massive flow, fine	CK
Massive flow, coarse	CW
Pillowed flow	CO
Brecciated flow	BQ

Sedimentary rocks

	Code
Agglomerate	AGG

Metamorphic rocks		
	Detailled classification	**Code**
	Gneiss	M1
	Schist	M8
	Amphibolite	M16
Tectonites		
Coherent	Cataclasite	TCU
	Fault breccia	TCB
	mylonite	TCM
	pseudotachylite	TCT
Loose (Late)	Fault breccia	TLB
Loose, running, uncons'd	Gouge	TLG

Appendix III: Assessment of Bachelor Mine Facilities

(LESLIE ENGINEERING, 1989)

APPENDIX III – Assessment of Bachelor mine facilities (Leslie Engineering, 1989)

Leslie Engineering Ltd. documented and carried out a detailed assessment of the Bachelor mine facilities and equipment on a fully installed basis in February 1989. This study was made at the end of the mine production and, as such, is based on a completely operational mine. Innovexplo can not guarantee the accuracy of this estimation nor attest to its compliance with a modern detailed engineering appraisal. It should be noted that prior to flooding of the underground developments (1992), Ross-Finlay recovered mining equipment and material. There is no report of the work carried out at this time.

The infrastructures can be listed as follows:
The 500 short tons per day mine facilities consist of the following (SNC-Lavalin report, March 1999):
- *An office, warehouse and shop complex;*
- *A head frame, bins, hoist and air compressor complex and substation;*
- *An underground mine;*
- *An ore processing complex;*
- *A tailing disposal area.*

The ore processing complex and tailings disposal area will be described in section 15.0 "Mineral Processing and metallurgical testing". The pictures (Fig. III-1, A to D) below were taken during the Innovexplo visit on October 12th, 2004.

Office, warehouse and shop complex:

This complex consists of three (3) prefabricated buildings, erected in 1982. These buildings have been used during the last underground drilling program and are in relatively good condition. Core racks are generally well preserved and available mineralized intersections easy to reach.

Headframe, bins, hoist and air compressors buildings:

This complex is in good condition and has been used for the underground drilling program during 3 months (equipment for the electricity power of the hoist has been renewed by CMAC). The current hoist can be used only for exploration and not for production (drum is cracked).

Underground mine:

According to the SNC-LAVALIN report from March 1999:

The Bachelor Lake Mine deposit was mined by underground mining methods, mainly by shrinkage stopping but the mine is now flooded. The mine was accessible by a three-compartment shaft to the 7th Level and a four-compartment shaft beyond the 7th Level. The shaft sump is at a depth of 562.66 m (1 846'). Twelve levels, with ventilation and egress, have been developed.

Ore passes have been driven from the 1st to the 6th Levels. A separate ore pass system joins the 7th and 8th Levels. A pneumatic rock breaker was located on the 8th Level and the loading pocket for this second ore pass system is below the 8th Level. Waste and ore from the 9th to the 12th Level, was loaded into the skips by means of lip pockets located at each of the level stations. All lip pockets were protected from passing oversize muck by means of 1" X 1" square section grizzly. Oversize muck was broken manually.

There is no ore pass and waste pass at the Bachelor Lake mine.

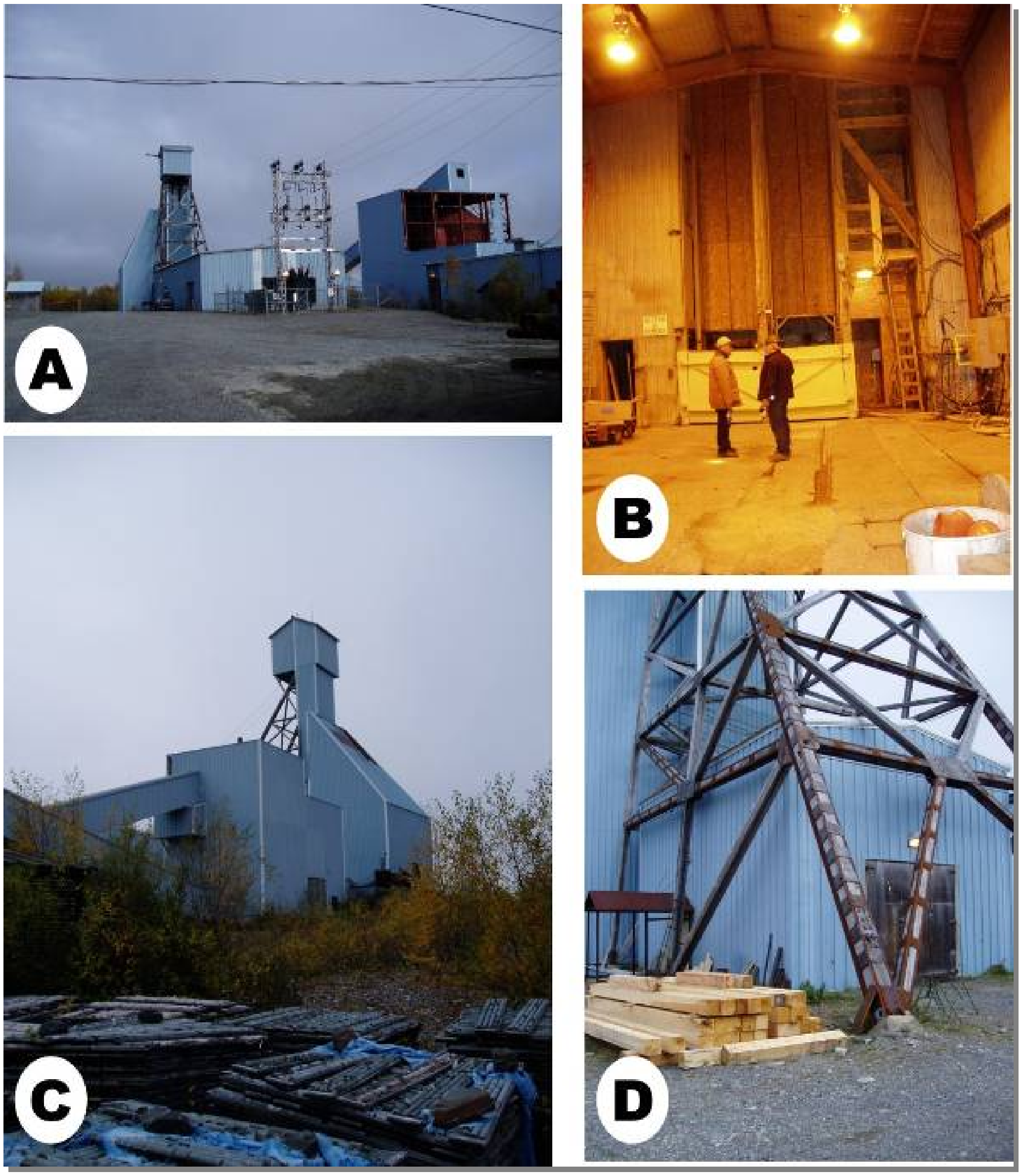


Figure III-1: Bachelor Lake surface infrastructures. A.) View looking East towards the Bachelor Lake surface infrastructures. Mill is in the right corner, the hoist room in the center and the headframe is in the left corner. B.) View from inside the headframe, the access to the shaft is clean and functional. C.) View from the core rack on the Bachelor Lake headframe. D.) The headframe structure has been recently reinforced with steel (work executed by CMAC).

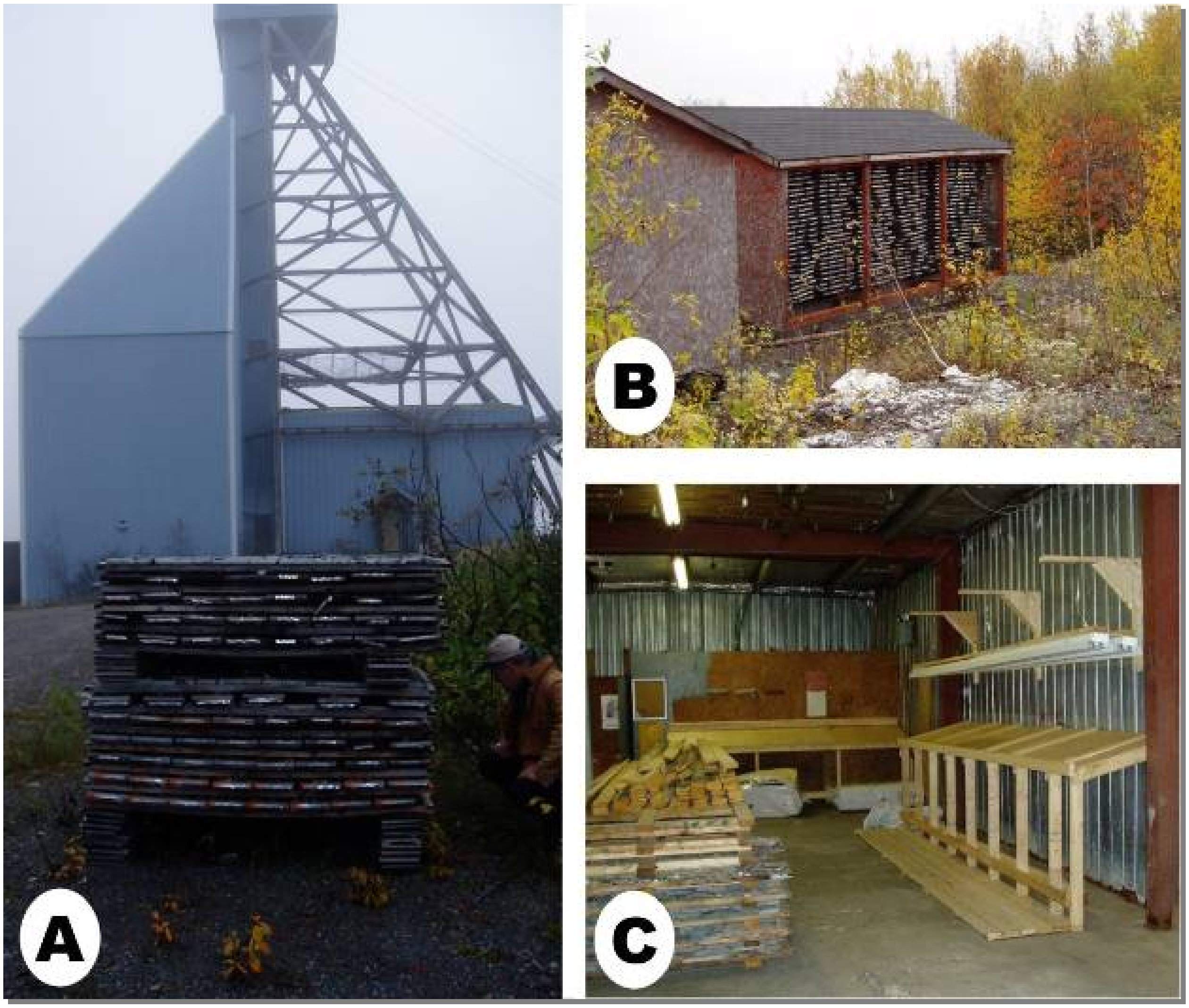


Figure III-2: Bachelor Lake core facilities. A.) View looking south on the headframe from the core shack. Note that the core pile is from a mineralized interval below 12th Level (DDH hole no. 12-37, 11-34, 12-23, 12-33, 12-14, 12-21, S-95-04, S-95-05, 12-22, 12-19, 12-12). B.) Core rack located south of the headframe. C.) View from inside the new core shack recently installed by Wolfden within the previous mine dry room.

Figure III-3: Bachelor Lake hoist room. A.) View from inside the hoist room; everything is presently functional for the dewatering process and for exploration purposes. B.) Equipment for the electricity power of the hoist has been renewed (work executed by CMAC).

Camp:

Further to the visit of the old camp site, it is obvious that this old camp (120 men) was demolished. Therefore, it will require a new camp or workers will have to travel to and from Lebel-sur-Quévillon, for example.

Concentrator Plant:

The buildings are in relatively good condition.

Records of an SNC-Lavalin Inc. personnel visit, in March 1999, report that:

The concentrator building is more damaged and requires a significant amount of reparation. One exterior wall in the ore storage area has been completely torn off and needs to be redone. The exterior south-east wall behind the no. 3 drum filter is also significantly damaged. There is a 60-90 cm (2-3') open gap in the south corner which requires to be repaired.

Appendix IV : Illustrations Of The Mineralization

AND DESCRIPTION OF THE ALTERATION

APPENDIX IV – Illustrations of the mineralization and description of the alteration

DESCRIPTION OF THE SURFACE SHOWINGS

Agar #1 (Au - Zn):

This surface showing, discovered in 1947 by Denis R. Agar, is located on the Hewfran claims and on the south side of the Bachelor Lake road (L16+00E, 14+00S). Stratigraphically located close to a fragmental basic (greywacke?) / rhyolite contact (Descarreaux, 1975), this showing could be described as fine grained disseminated pyrite (4 to 6%) in a 30-cm (1') wide quartz vein striking N083° and dipping 63° to the south. The vein is located in a silica and hematitized alteration zone (200' strike length and over 20 to 30' wide). The showing is exposed in a series of 6 trenches.

Historically, the best assays returned **0.12 oz/t Au** in the mineralized zones and Y. Rougerie also reported in 1989 significant sphalerite identified on the Agar #1 which led him to interpret this showing as the northeast extension of the Coniagas sulphide horizon.

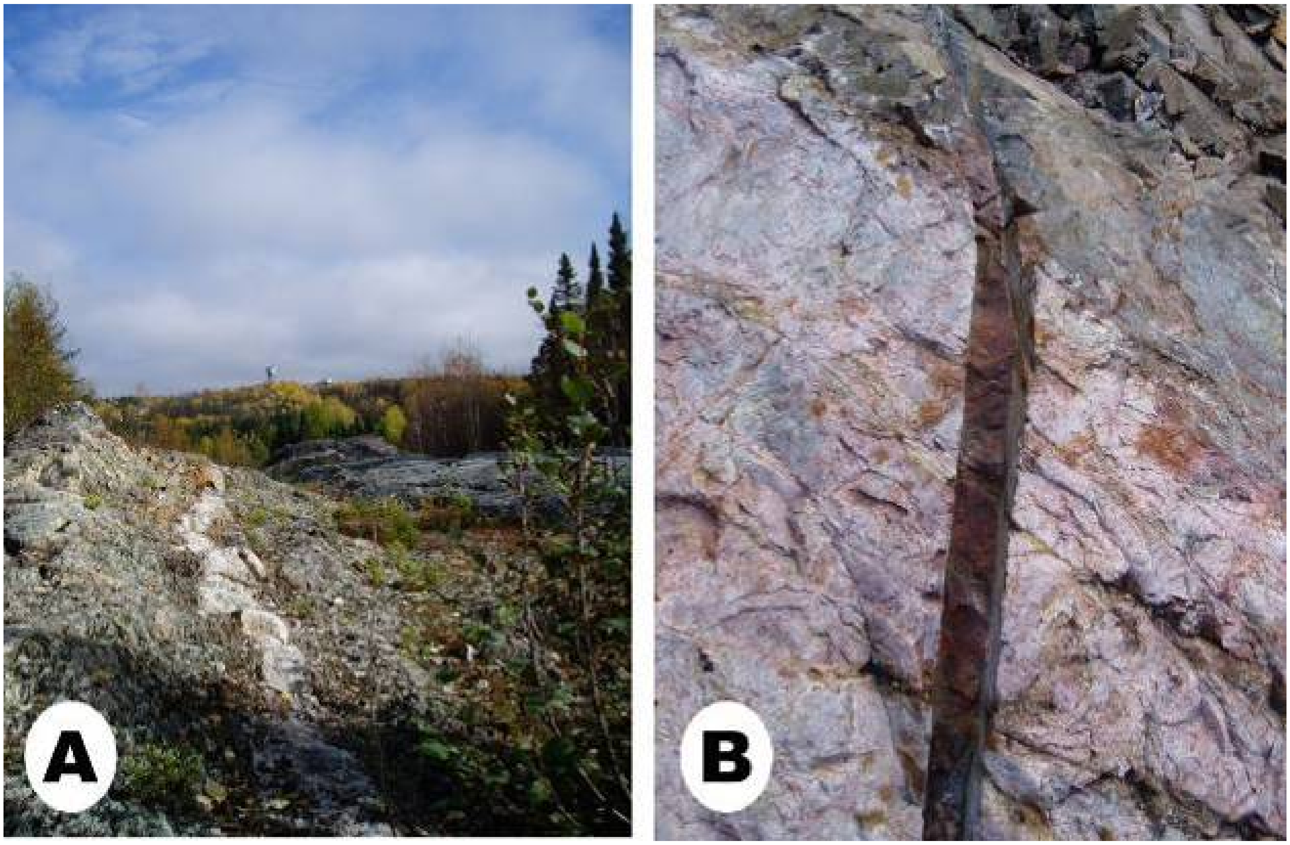


Figure IV-1: Hewfran showing (Agar #1). A.) Quartz vein on Hewfran showing with the Bachelor Lake headframe in the background. View looking East. B.) Channel sample within hematitized and pyritized gold mineralization at Hewfran.

Agar #2 (Au):

The Agar #2 surface showing is located on the eastern side of the Hewfran claims (L38+00N, 8+00S) and it was discovered in 1947 by Denis R. Agar. This showing lies in more basic rocks than the Agar #1 showing (Descarreaux, 1975) and it could be described as fine grained disseminated pyrite in a felsic dyke striking N070° and dipping 68° to the south. The walls are described as being well sheared. This showing was originally exposed in an east-northeast-trending trench and drilling has indicated to be the western extension of the Bachelor Lake “Main” Zone. It is located along the footwall contact of the Waconichi #4 fault (Fig. IV-2) and consists of 5 to 10% very fine grained pyrite with 10% fractures and stockwork controlled hematite alteration. Fine specks of gold were observed. The vein is located in a silica and hematitized alteration zone (200’ in strike length over 20 to 30’ in width). Several trenches have been dug across the structure.

Five pounds, selected as a sample of the representative better looking material returned an assay of **0.29 oz/t Au**. A small grab sample selected for pyrite mineralization assayed **0.48 oz/t Au**.

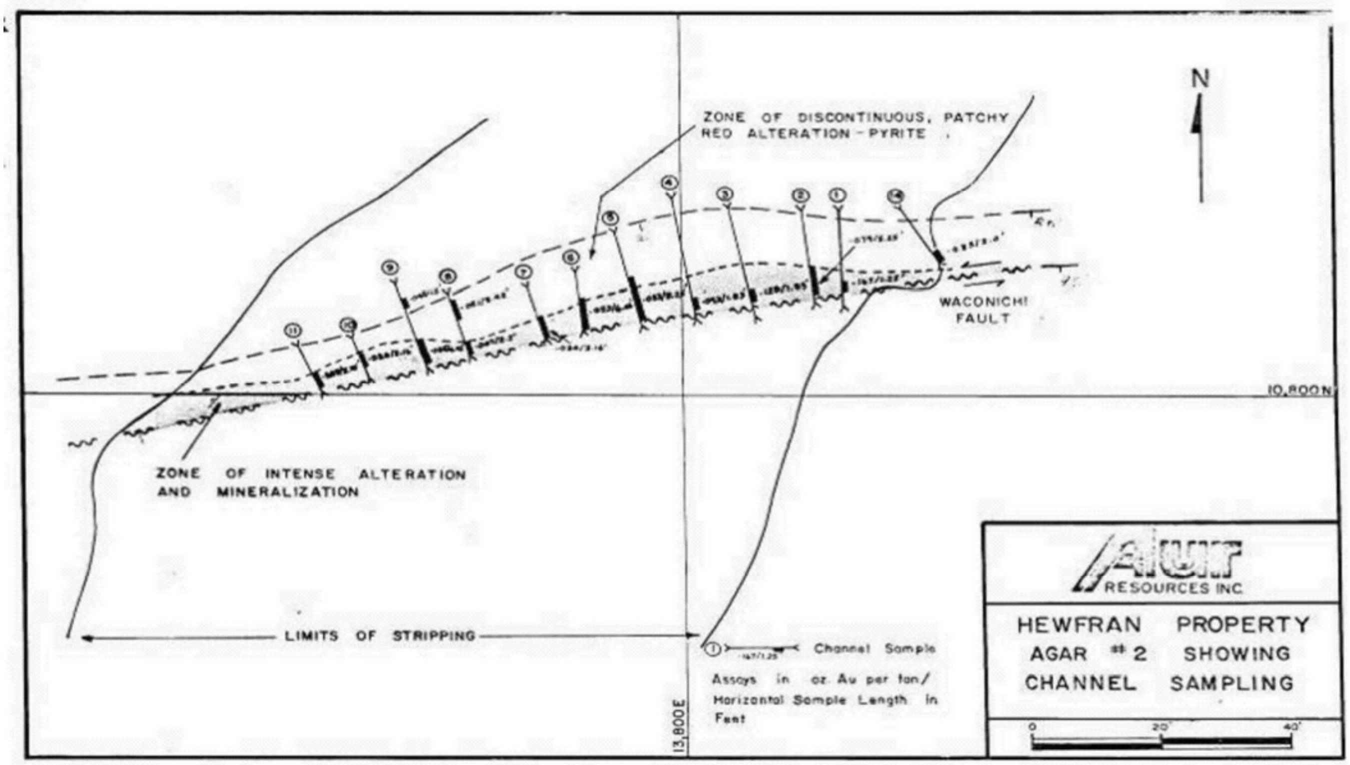


Figure IV-2: Agar #2 showing on the Hewfran claims illustrating channel samples location (from Bulman, V Draft report – 1986 exploration program)

Area- Opawica (Zn, Cu, Ag):

Located on the Hewfran claims, this showing is at 1.6 km from the Coniagas mine. The mineralization (chalcopyrite, sphalerite, massive and disseminated silver) has been described as a volcanogenic hosted in felsic pyroclastic environment. A surface sample returned 1.80% Zn, 0.75% Cu and 25.26 g/t Ag.

O’Brien showing (Au):

Located on the Bachelor claims, this showing corresponds to the original gold discovery made at surface on the Bachelor claims on the eastern side of the O’Brien granitic stock. Several trenches and drill holes followed the discovery.

Terri and Middle showings (Au):

The Terri and Middle showings are located on the Hewfran claims and discovered while mapping in the vicinity of the Agar #2 showing in June 1982. They are located 10.6 m (35’) (Middle) and 15 m (50’) (Terri) south of the Agar #2 showing at L48+00W, 34+70S. They are showings weak in hematization and silicification alteration with 5% pyrite; the best results obtained in the channel sampling were 0.055 and 0.012 oz/t Au over 15 and 20 cm (0.5’ and 0.67’) respectively. One grab sample assayed 0.08 oz/t Au.

Valdex (Au):

The Valdex is located on the Hewfran claims and discovered in 1947. It is hosted in a quartz vein cross cutting volcanic rocks. The occurrence is associated with the Bachelor-type mineralization. A surface sample returned an assay of 7.5 g/t Au over 0.15 m.

Zinc showing #1 (Zn):

The Zinc showing #1 is located on the Hewfran claims and on the north part of the road leading to the Bachelor Lake mine (claim 3 083 925). This mineralization has been described as a volcanogenic environment, in contrast to the Agar showings, which is a gold mineralization associated with hydrothermal and epigenetic sources.

Minor sphalerite mineralization is hosted by massive fine grained grey to black conchoidally fractured rhyolite or rhyolitic lapilli tuffs that locally contain disseminated pyrite. The light brown sphalerite and associated calcite occurs as fracture fillings in apparently tectonically brecciated rhyolite. Fractures generally are parallel to the bedding and locally may crosscut the bedding. Assays from Malouf (1948) are as follows:

Width (ft)	Ag (oz/t)	Zn (%)	Width (ft)	Ag (oz/t)	Zn (%)
1.0	0.02	Tr.	3.0	0.04	6.31
4.0	0.06	0.05	3.5	0.03	Nil
4.0	-	**6.85**	4.6	0.04	**5.28**
4.0	-	**6.60**	grab	0.04	**11.8**
5.0	0.04	**5.68**	Grab	0.28	**8.58**
3.0	0.02	Tr.	grab	0.30	**30.1**

Zinc showing #2 (Zn):

The Zinc showing #2 is located on the Hewfran claims and on the north part of the road leading to the Bachelor Lake Mine (claim 3 083 925). This mineralization has been described as volcanogenic.

Trenches and stripped area expose mainly rhyolites and rhyolitic lapilli tuff similar to that in the #1 showing. Assays from Malouf (1948) are as follows:

Width (ft)	Pb (%)	Zn (%)	Au (oz/t)	Ag (oz/t)	Width (ft)	Pb (%)	Zn (%)	Au (oz/t)	Ag (oz/t)
2.3	Nil	Tr.		0.02	1.5	0.05	**3.53**	tr	-
2.3	Nil	Nil		0.03	1.0	Nil	1.31		0.03
2.0	Nil	Nil		0.03	1.5	0.31	Tr.		0.11
-	-	1.51		0.04					

Hole 19501-52 – Zn-Au occurrence:

The hole 19501-52 – Zn-Au occurrence is located on the Hewfran claims. This hole has been drilled by Aur toward the north on the 12 100’ E section while prospecting for the Bachelor’s western extension of the mineralized zones. A significant base metal intercept returned 7% Zn and 5.45 g/t Au over 2.13 m. Projected on surface, this occurrence was interpreted by Y. Rougerie (1989) (as well as the Agar #1 showing) as the northeast extension of the Coniagas base metal horizon.

The Coniagas marker horizon (Zn-Pb-Ag):

The Bachelor property (Hewfran claims) is also the host of the north-east extension of the Coniagas horizon. Hosted within a felsic volcanic rock sequence, this marker horizon represents a favourable contact for polymetallic massive sulphide mineralization. Significant results were obtained in this horizon on the Bachelor property.

As read in Y. Rougerie report from 1989:

Significant sphalerite mineralization assaying 7.0% Zn and 0.159 oz/t Au over 7.0 ft was intersected in hole 19501-52. Significant sphalerite was also identified in the Agar #1 outcrop and these two occurrences are interpreted to represent the northeast extension of the Coniagas sulphide horizon. Two surface holes totalling 1260 ft were drilled in March and April, 1989, to follow-up on this new discovery. Although both holes 19501-61 and 62 encountered favourable lithologies, only weakly anomalous Zn assays were returned. However, the data does confirm that excellent potential exists for discovery of new Coniagas-type economic massive sulphide deposits on the north-eastern part of the Hewfran property.

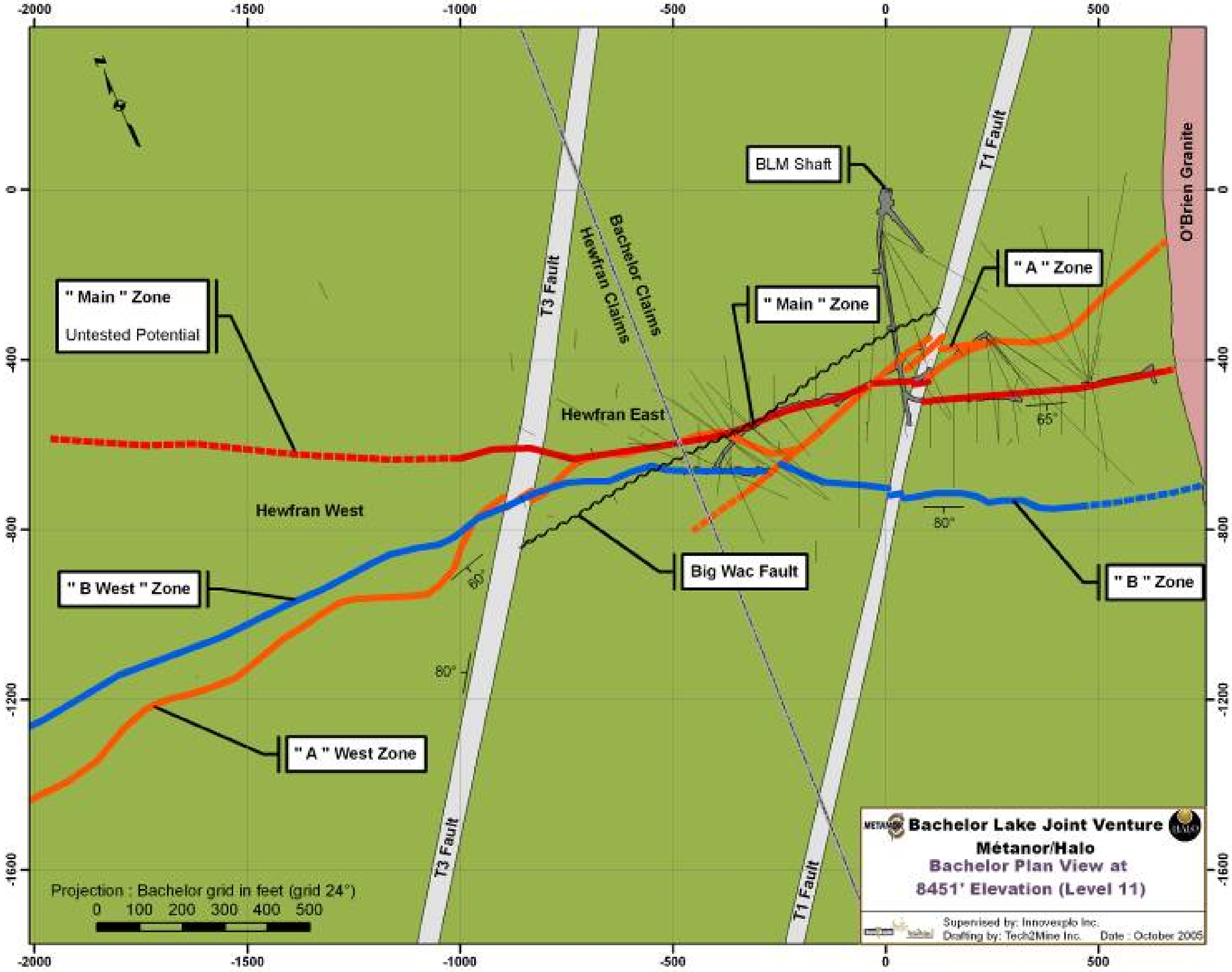


Figure IV-3: Schematic plan view of the Bachelor Lake gold deposit at 8451' elev. (11th Level)

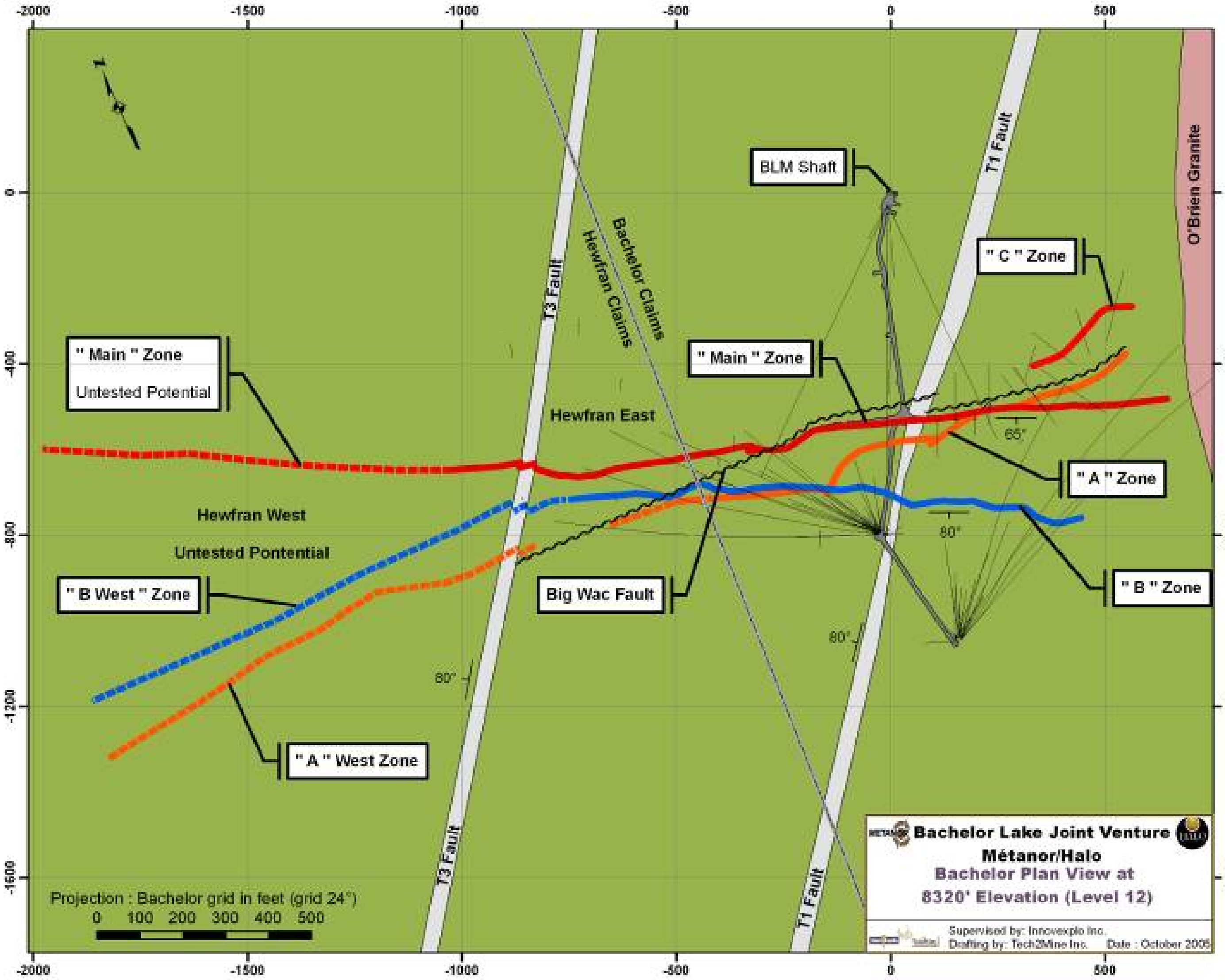


Figure IV-4: Schematic plan view of the Bachelor Lake gold deposit at 8320' elev. (12th Level)

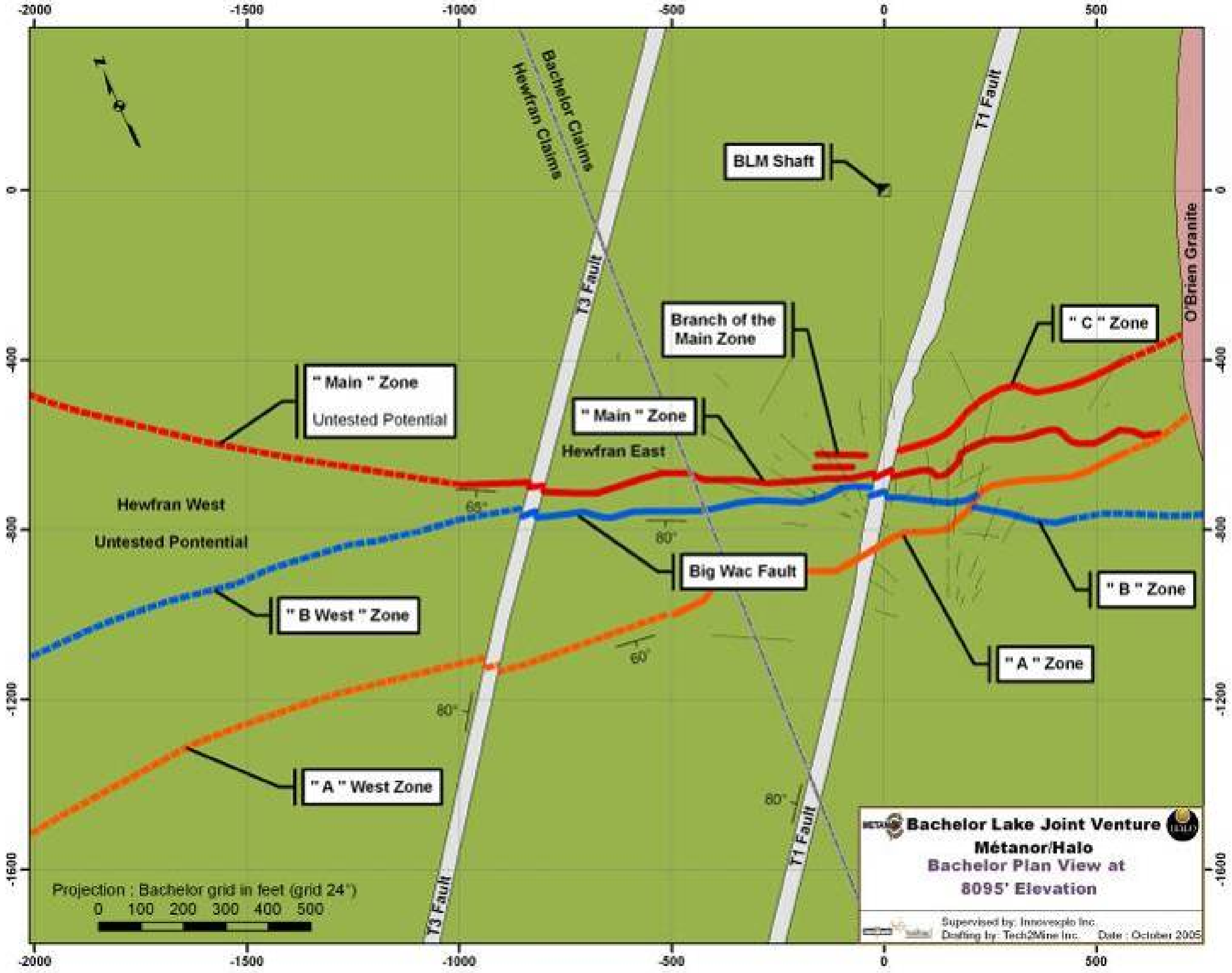


Figure IV-5: Schematic plan view of the Bachelor Lake gold deposit at 8095' elev. (15th Level)

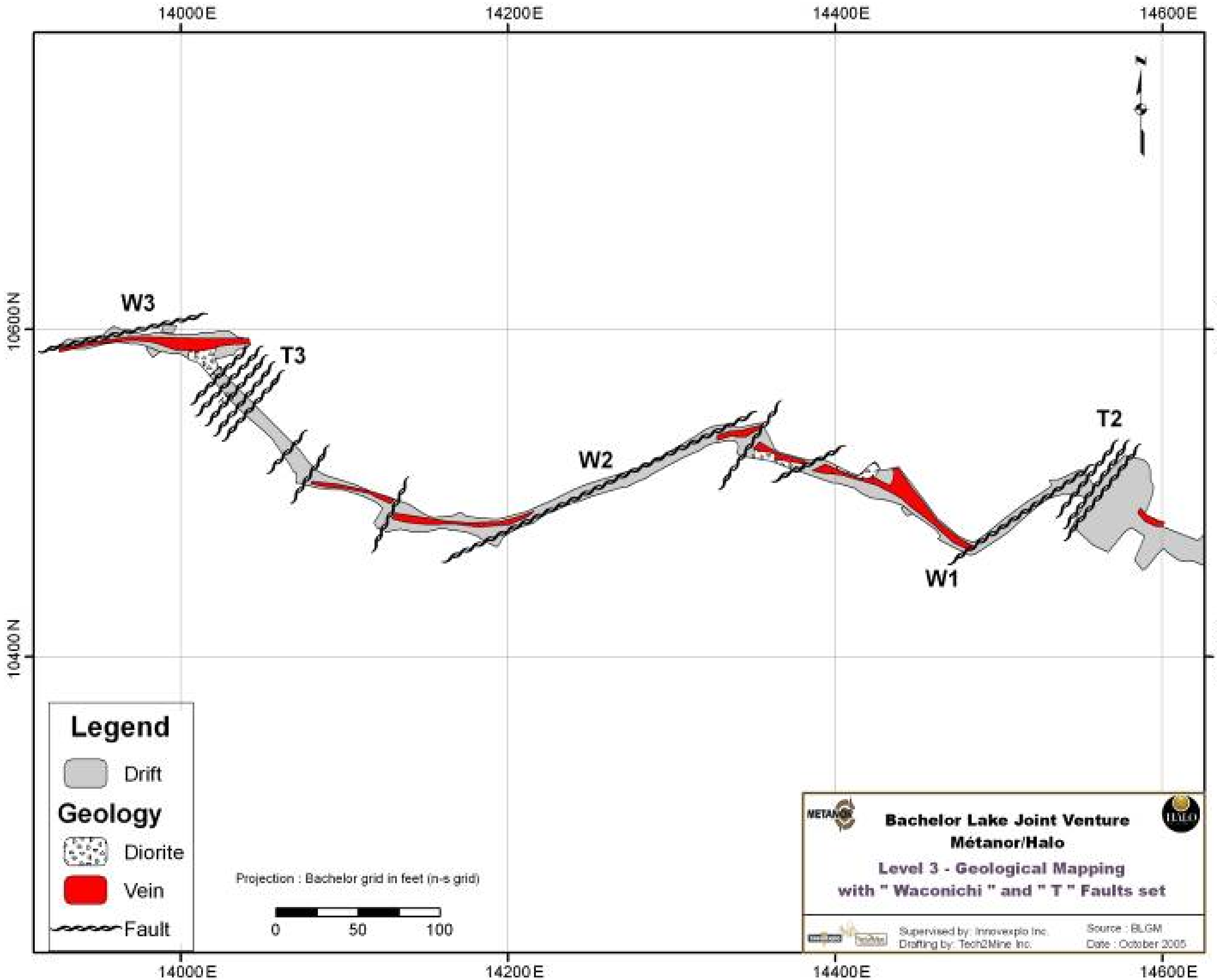


Figure IV-6: Effect of late faulting on the "Main" zone, Bachelor Lake gold deposit (3rd Level)

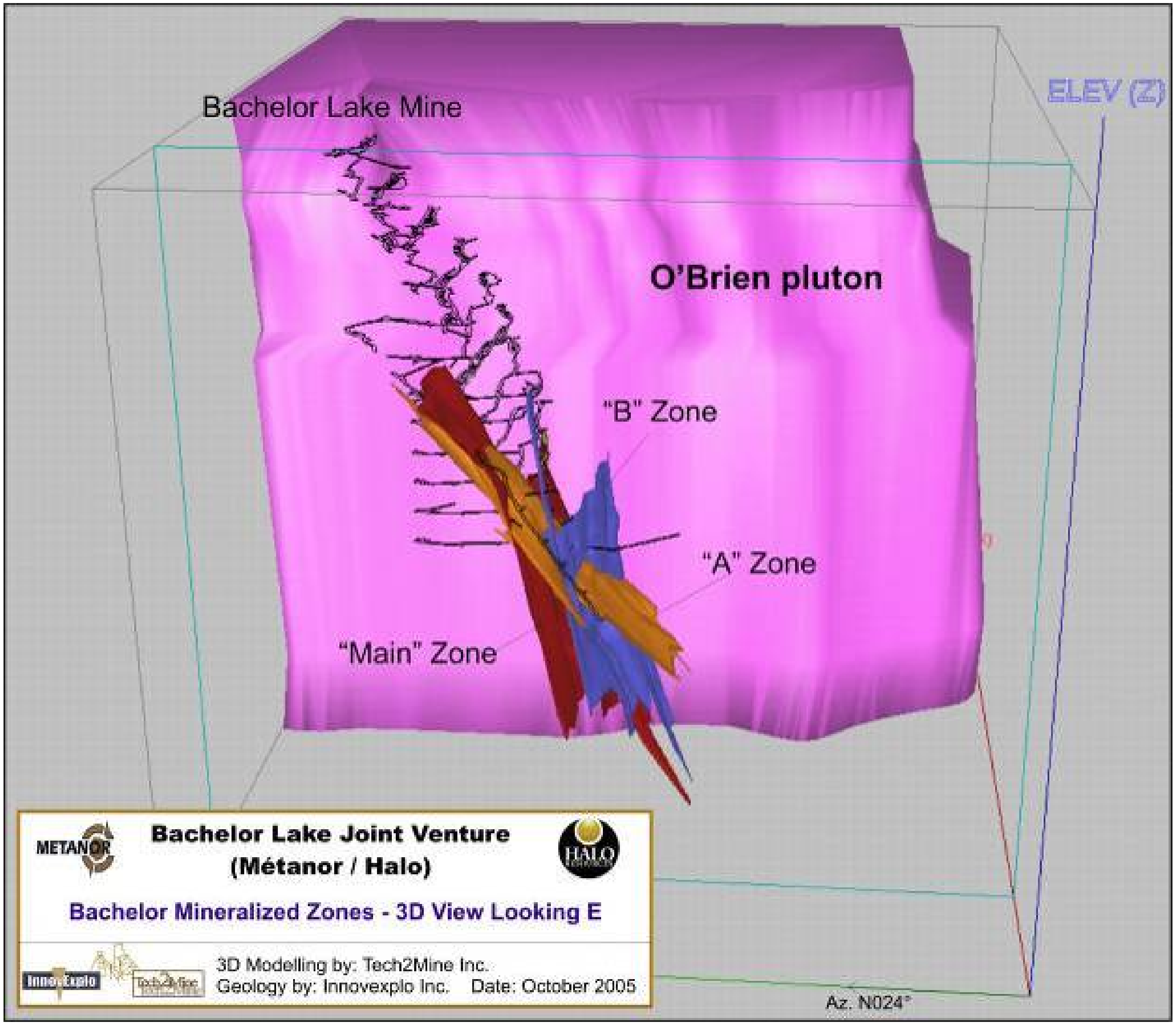


Figure IV-7: 3D view looking east of the "Main", "A", "B", and "C" zones illustrating their relationship

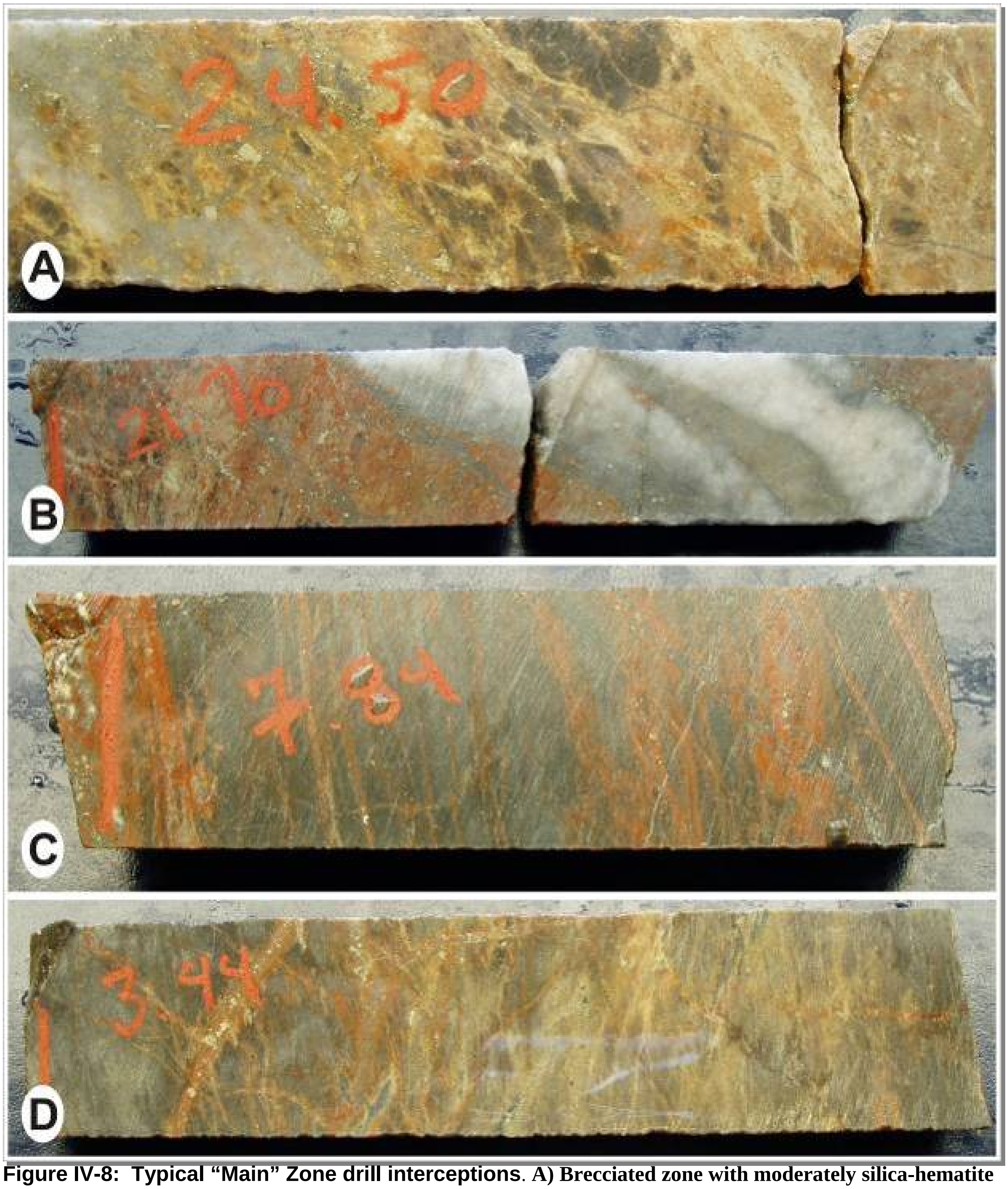


Figure IV-8: Typical "Main" Zone drill interceptions. A) Brecciated zone with moderately silica-hematite altered fragments with disseminated pyrite and grading 24.5 g/t Au (12-42, sample # 108 055 @ 100.72 m); B) Late quartz vein cross-cutting typical moderate pervasive hematitized matrix with disseminated pyrite grading 21.7 g/t Au (12-42, sample # 108 056 @ 101.8 m); C) Volcanic tuff altered and cross-cut by 20% of hematitized veinlets illustrating the alteration zones wallrocks and grading 7.83 g/t Au (12-42, sample # 108 058 @ 103 m); D) Same as C) grading 3.44 g/t Au (12-42, sample # 108 054 @ 100.8 m).

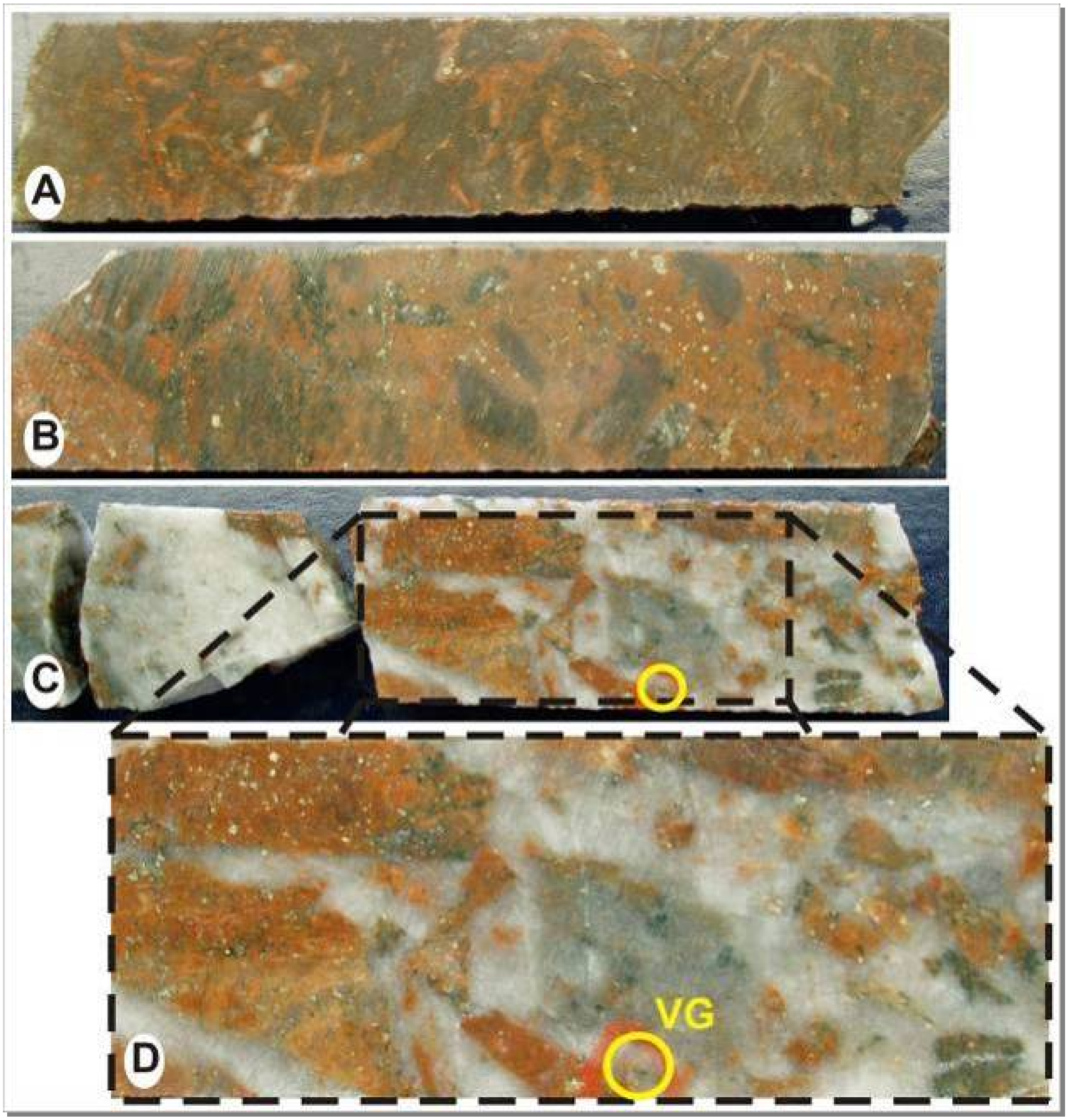


Figure IV-9: Typical “B” Zone drill interceptions. A) brecciated zone with strong hematitization over 45 cm grading 1.82 g/t Au, (12-42, sample # 108 037 @ 45.38 m); B) Breccia zone with remnant fragments in a strongly hematitized matrix with disseminated pyrite grading 6.83 g/t Au (12-42, sample # 108 044 @ 49.86 m); C) Quartz vein breccia with hematitized angular fragments illustrating the late brecciation event with visible gold grading 26.5 g/t Au (12-42, sample # 108 048 @ 53.38 m) with a zoom D);

Table IV-1 Grade of the "A" Zone at its junction with other zones (complete results over 1 g/t Au)

Hole ID	From	To	Zone	Au values (g/t over core length in meters)	Interpretation
12-42	42.17	45.38	«A»	**2.06 / 1.83**	Junction between «A» and «B» Zones
12-44	80.22	83.72	«A»	**5.19 / 3.50**	Weak shear 5 m of «B» Zone
12-50	49.50	56.40	«A»	**8.38 / 2.27**	Junction between «A» and «B» Zones
12-51	155.00	156.40	«A»	**2.25 / 1.40**	
12-53	112.35	120.75	«A»	**3.92 / 1.55**	Junction between «A» and «B» Zones
12-54	71.50	75.25	«A»	**1.92 / 3.75**	Brecciated zones interpreted as "B" Zone
12-55	103.45	110.00	«A»	**5.80 / 5.80**	Junction between «A» and «B» Zones
12-55	117.60	121.45	«A»	**1.50 / 3.95**	Brecciated zones interpreted as "B" Zone
12-57	115.81	127.00	«A»	**3.30 / 3.85**	Brecciated zones interpreted as "B" Zone. Presence of VG
12-59	104.90	110.70	«A»	**1.19 / 1.30**	Not typical "A" Zone type mineralization
12-59	132.40	136.45	«A»	**12.38 / 1.50**	Local «B» Zone lenses
12-61	121.25	123.80	«A»	**1.19 / 2.55**	Brecciated zones interpreted as "B" Zone with some local shearing
12-66	32.38	35.60	«A»	**1.11 / 3.22**	
12-68	45.00	51.70	«A»	**3.25 / 3.00**	«B» Zone description
12-70	32.60	36.50	«A»	**1.14 / 1.50**	Brecciated zones interpreted as "B" Zone
12-80	70.95	76.35	«A»	**1.14 / 2.01**	
12-83	112.45	114.00	«A»	**1.48 / 1.35**	Brecciated zones interpreted as "B" Zone. Presence of VG
12-83	116.55	125.00	«A»	**1.49 / 1.15**	Not typical "A" Zone type mineralization
12-84	33.80	39.80	«A»	**1.40 / 0.80**	
12-86	94.00	96.20	«A»	**1.01 / 0.70**	
12-88	128.60	129.95	«A»	**2.62 / 1.35**	Weak brecciated zones interpreted as "B" Zone

Alteration related to Bachelor-type gold mineralization

The Bachelor Lake alteration assemblage is superimposed on the regional metamorphic greenschist facies assemblage. Locally, metamorphism has reached the lower amphibolite facies at the contact zone of the O'Brien granitic stock (Lauzière, 1989). Alterations zones are metric to decametric in width and forms discordant zones relative to the stratigraphic sequences.

The alteration consists of an epidote-carbonate-chlorite-pyrite/magnetite assemblage adjacent to the gold mineralized zone and has two assemblages within it (Lauzière, 1989). These two (2) alteration assemblages are characterized by: (1) white mica-quartz-pyrite-magnetite and (2) K feldspar-hematite-pyrite-carbonate. The gold mineralization occurs either as a massive hematitized rock with disseminated pyrite, or more frequently as a stockwork of carbonate and/or K feldspar and hematite veinlets.

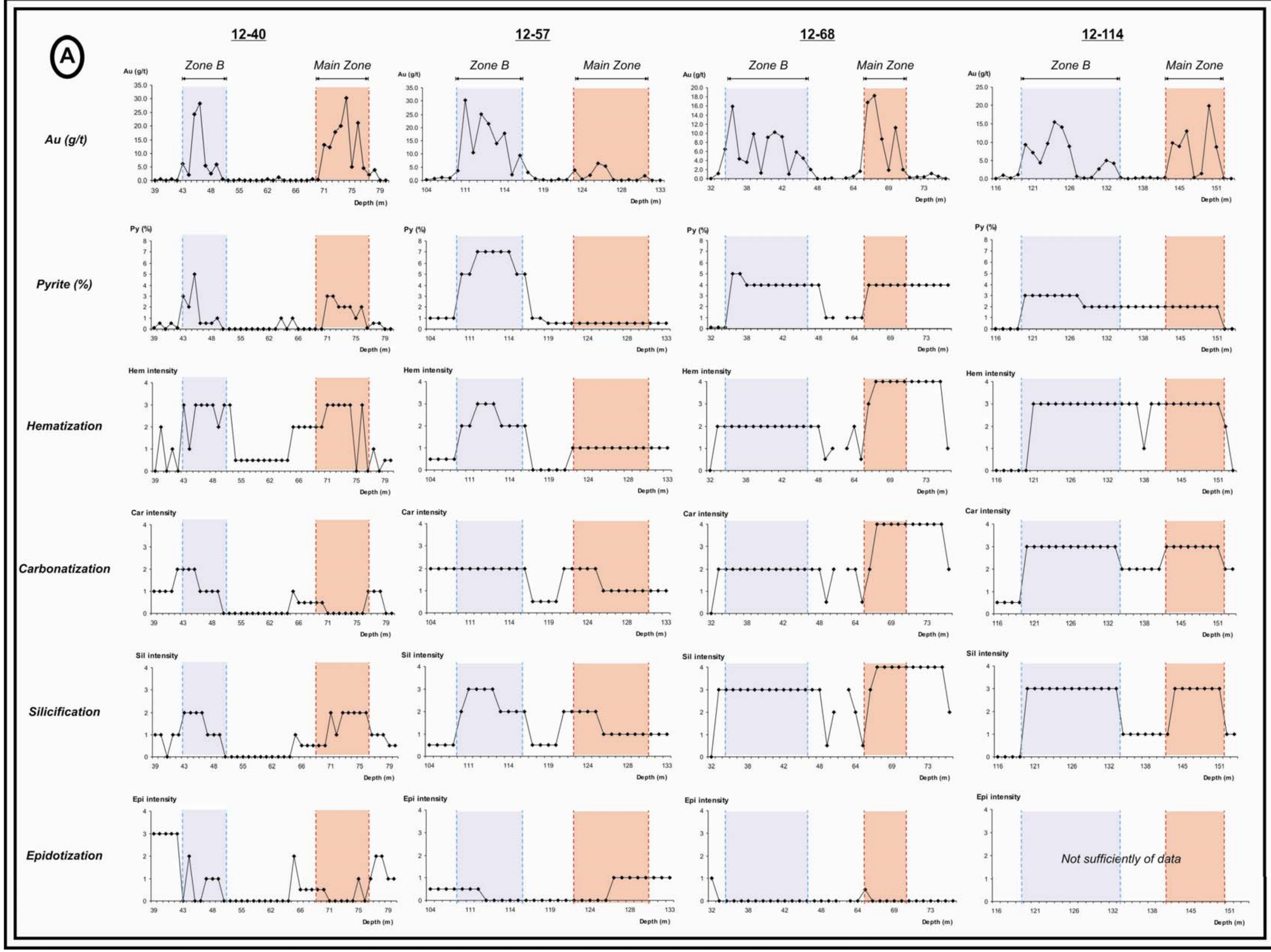
Zoning of the alteration zones and its relationship with gold mineralization at Bachelor is illustrated in Figures IV-10 (correlation) and IV-11 (core pictures). During the 2005 program, alteration has been described with more details and is presented below:

Table IV-2 The alteration zoning and haloes at Bachelor

toward gold mineralized core

MT $^{+}$	- weak magnetism (magnetite) ; - host rocks looks unaltered;
MT $^{+}$ ± Vn EP	- weak magnetism (magnetite); and - epidote veinlets cross cutting the host rock; - protolith can still be recognized;
Mt $^{++}$ ± Vn EP ↑ Vn CB	- strong magnetism (magnetite); - carbonates and epidote veinlets cross cutting the host rocks; - protolith can still be recognized;
MT $^{+++}$ CB $^{+}$	- very strong magnetism (magnetite); - pervasive carbonatization (mixed with epidote veinlets) - protolith can still be recognized;
MT $^{++}$ CB $^{++}$ ↑ SIL	- strong magnetism (magnetite) - silicification (lighter colour); - protolith is still recognized but primary features are affected;
SIL $^{++}$ ↑ Vn HM	- magnetism decreased but still present; - hematite veinlets cross cutting the host rock; - strong silicification and weak carbonatization; - disseminated pyrite (weak)
SIL $^{+++}$ ↑ HM	- no magnetite but strong and pervasive hematitization overlapping carbonatization; - strong silicification; - disseminated pyrite (moderate)
HM $^{+++}$ SIL $^{+++}$ PY $^{++}$ ± ANK $^{+++}$	- strong and pervasive hematitization, silicification and ankerite; - disseminated pyrite (strong) - intense hematitization is characterized by a characteristic deep red brick colour;

Legend: Vn: veins or veinlets; MT: magnetite (Fe_3O_4); EP: epidote; CB: carbonates (mainly calcite, $CaCO_3$); SIL: silicification (SiO_2); HM: hematite (Fe_2O_3); PY: pyrite (FeS_2); ANK: ankerite ($CaFe(CO_3)_2$.



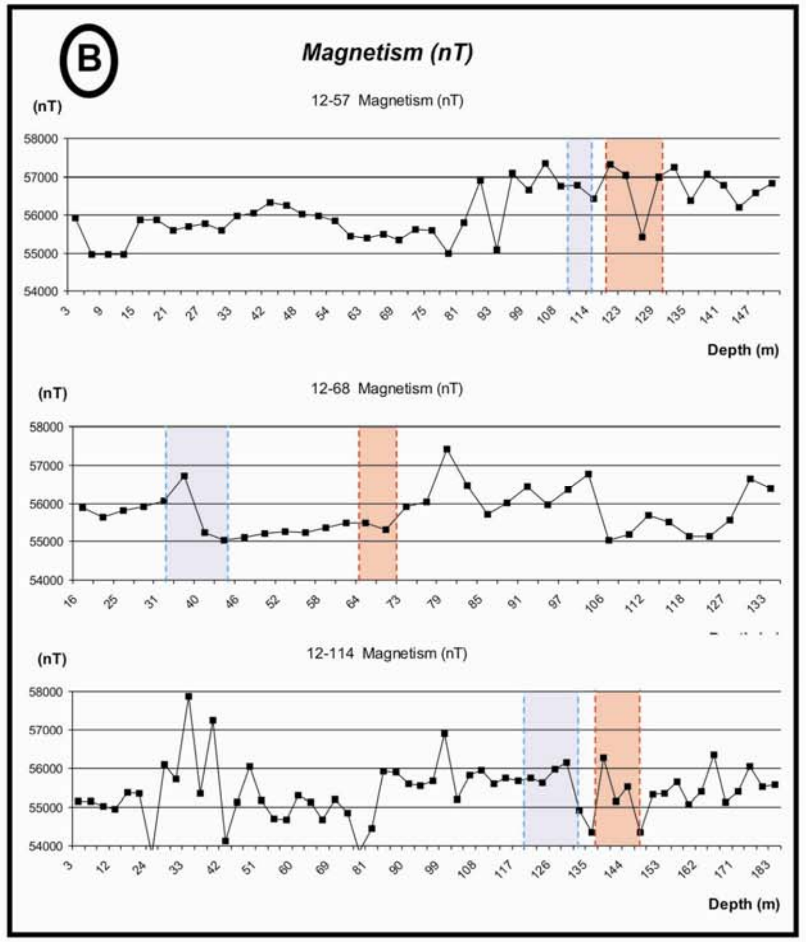


Figure IV-10 – A) Correlation between high gold values, pyrite content with hematite and silica alteration zones illustrate by four typical holes at Bachelor Lake. Mineralized zones are also edged by carbonate and epidote haloes. Opposite correlation also exists as illustrated within hole 12-57, where low gold values from the "Main Zone" could be explained by low content of pyrite even if alterations are present. **B) Local decreasing of the magnetism within the mineralized zone.**

Note:

- *Alteration intensity is expressed in four descriptive levels which are 1 for weak, 2 for moderate, 3 for strong and 4 for intense;*
- *Magnetism is expressed in nT, nano Tesla (surveyed by multi-shot electronic Flex-it instrument).*

Figure IV-10: A) Correlation between high gold values, pyrite content with hematite and silica alteration zones illustrate by four typical holes at Bachelor Lake. B) Local decreasing of the magnetism within the mineralized zone.

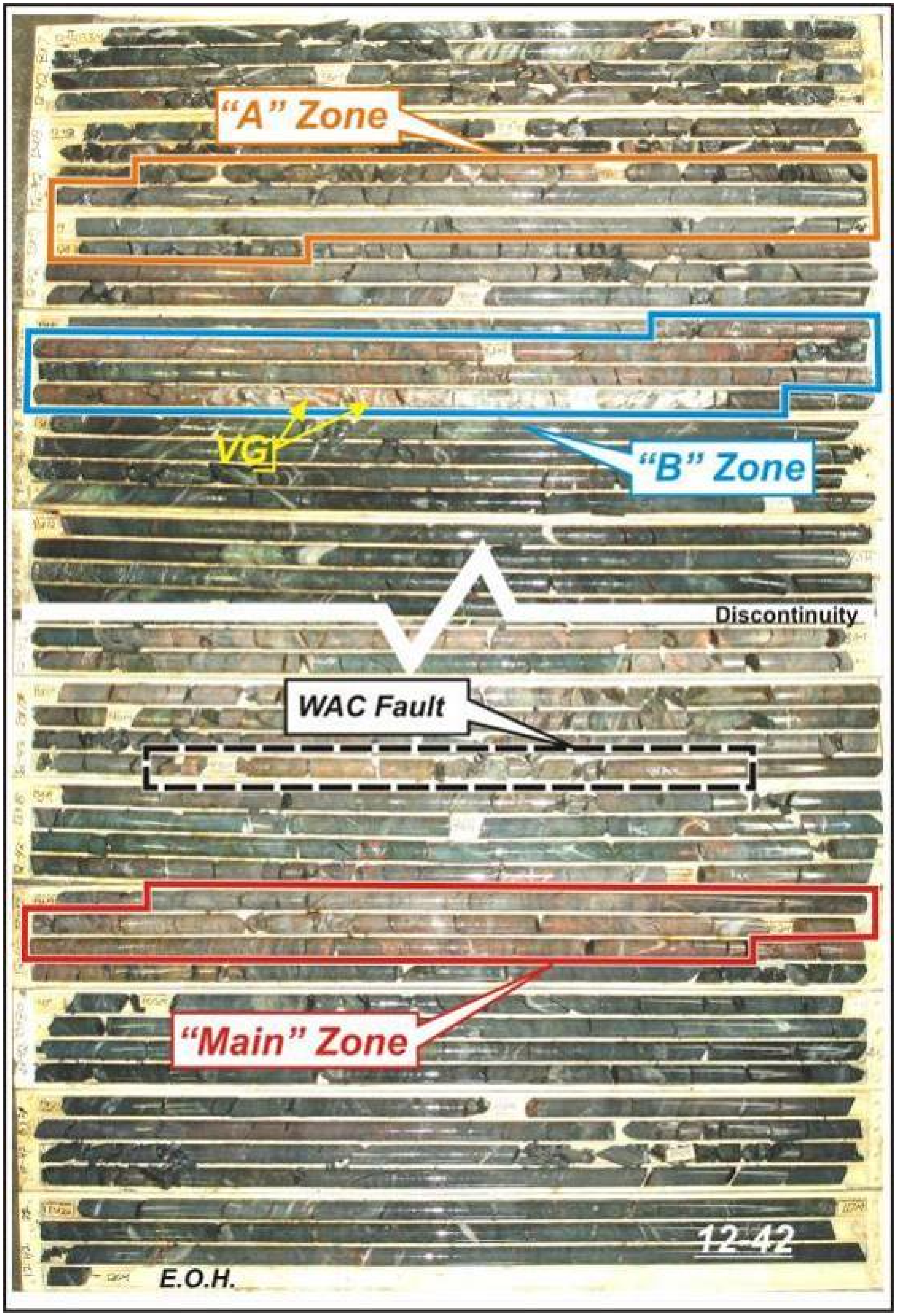


Figure IV-11: Zones ("A", "B" and "Main" zones), alteration and Waconichi fault (Hole 12-42)

Appendix V : 2005 Underground Drilling Program

Table V-1 Best results for the "Main" and "B" zones obtained during the 2005 underground drilling program (all composites over cut-off grade of 3.43 g/t Au (0.10 oz/t) on a minimum horizontal width of 1.5 m (5')

Hole ID	Zone ID	Section	From (ft)	To (ft)	Au Grade (oz/t) over horizontal width (ft)			Au Grade (g/t) over horizontal width (m)		
12-38	"B" Zone	50W	109,83	119,99	0,203	over	26,00	6,97	over	7,92
	"Main" Zone	50W	227,36	239,50	0,112	over	12,00	3,85	over	3,66
12-40	"B" Zone	50W	142,18	165,60	0,340	over	15,50	11,66	over	4,72
	"Main" Zone	100W	231,78	259,88	0,381	over	25,00	13,08	over	7,62
12-42	"B" Zone	150W	163,55	177,90	0,232	over	9,00	7,95	over	2,74
	"Main" Zone	300W	323,89	340,97	0,254	over	11,50	8,71	over	3,51
12-44	"B" Zone	250W	240,38	274,63	0,108	over	16,00	3,72	over	4,88
	"Main" Zone	450W	452,28	468,80	0,433	over	6,50	14,84	over	1,98
12-46	"B" Zone	100W	194,71	224,01	0,417	over	18,50	14,31	over	5,64
	"Main" Zone	100W	251,20	256,13	0,132	over	10,00	4,52	over	3,05
	"Main" Zone	100W	273,77	280,01	0,399	over	7,50	13,68	over	2,29
	"Main" Zone	100W	302,22	313,40	0,125	over	8,00	4,28	over	2,44
12-48	"B" Zone	50W	261,48	278,11	0,518	over	12,00	17,76	over	3,66
	"Main" Zone	50W	282,47	294,46	0,254	over	8,50	8,72	over	2,59
12-49	"Main" Zone	400E	572,64	581,50	0,204	over	7,50	7,01	over	2,29
12-50	"B" Zone	150W	164,82	203,53	0,273	over	17,50	9,37	over	5,33
	"Main" Zone	300W	341,16	353,61	0,235	over	16,00	8,07	over	4,88
12-53	"B" Zone	100E	319,15	327,97	0,192	over	8,50	6,59	over	2,59
	"B" Zone	100E	386,62	405,96	0,205	over	21,00	7,03	over	6,40
12-54	"B" Zone	200W	194,04	223,56	0,125	over	14,00	4,27	over	4,27
12-55	"B" Zone	150E	319,61	357,54	0,111	over	34,50	3,80	over	10,52
	"B" Zone	150E	369,83	396,57	0,150	over	26,50	5,16	over	8,08
12-56	"B" Zone	50W	104,32	122,38	0,283	over	26,00	9,72	over	7,92
	"Main" Zone	100W	226,36	242,43	0,207	over	16,00	7,10	over	4,88
12-57	"B" Zone	150E	325,91	341,99	0,129	over	16,00	4,41	over	4,88
	"B" Zone	100E	361,50	380,04	0,368	over	20,00	12,62	over	6,10
12-59	"B" Zone	150E	319,77	357,62	0,146	over	34,00	4,99	over	10,36
	"B" Zone	200E	370,10	396,89	0,288	over	26,00	9,88	over	7,92
12-60	"B" Zone	50W	359,57	369,75	0,133	over	8,00	4,55	over	2,44
	"Main" Zone	50W	373,03	387,62	0,196	over	9,00	6,71	over	2,74
12-61	"B" Zone	200E	322,79	346,56	0,134	over	20,50	4,59	over	6,25
	"B" Zone	200E	384,24	399,37	0,229	over	19,00	7,84	over	5,79
12-62	"B" Zone	300W	346,58	353,97	0,223	over	5,00	7,63	over	1,52
12-64	"B" Zone	200W	199,32	240,79	0,143	over	15,00	4,89	over	4,57
12-65	"B" Zone	200E	421,73	436,50	0,109	over	14,50	3,73	over	4,42
	"Main" Zone	200E	546,72	553,78	0,132	over	7,00	4,52	over	2,13
12-66	"Main" Zone	200W	267,01	283,58	0,271	over	14,50	9,28	over	4,42
12-68	"B" Zone	50W	113,18	152,55	0,178	over	15,00	6,10	over	4,57
	"Main" Zone	50W	218,22	234,56	0,216	over	16,50	7,40	over	5,03
12-69	"B" Zone	300E	384,12	414,29	0,143	over	20,00	4,89	over	6,10
	"B" Zone	350E	435,12	447,43	0,148	over	11,50	5,08	over	3,51
12-70	"B" Zone	150W	190,14	196,02	0,377	over	10,00	12,93	over	3,05

Table V-1 (cont.) - Best results for the “Main” and “B” zones obtained during the 2005 underground drilling program (all composites over cut-off grade of 3,43 g/t Au (0.10 oz/t) on a minimum horizontal width of 1,5 m (5’)

Hole ID	Zone ID	Section	From (ft)	To (ft)	Au Grade (oz/t) over horizontal width (ft)			Au Grade (g/t) over horizontal width (m)		
12-72	"B" Zone	100W	294.29	300.20	0.149	over	6.00	5.11	over	1.83
12-74	"Main" Zone	100W	365.64	378.44	0.190	over	8.00	6.53	over	2.44
12-77	**"B" Zone**	**100W**	**527.85**	**535.35**	**0.491**	**over**	**10.00**	**16.83**	**over**	**3.05**
12-80	"Main" Zone	350W	400.04	413.99	0.149	over	7.50	5.13	over	2.29
12-83	**"B" Zone**	**200E**	**356.05**	**369.49**	**0.415**	**over**	**15.00**	**14.22**	**over**	**4.57**
	"B" Zone	200E	372.83	382.28	0.137	over	10.50	4.71	over	3.20
12-87	"B" Zone	150E	333.64	353.98	0.120	over	24.00	4.13	over	7.32
	"B" Zone	200E	364.82	379.06	0.180	over	17.50	6.17	over	5.33
	"Main" Zone	200E	476.83	482.24	0.112	over	9.00	3.85	over	2.74
12-88	**"B" Zone**	**450W**	**344.46**	**359.52**	**0.477**	**over**	**5.00**	**16.36**	**over**	**1.52**
12-89	**"Main" Zone**	**450E**	**573.21**	**589.77**	**0.235**	**over**	**8.50**	**8.06**	**over**	**2.59**
12-90	**"B" Zone**	**150W**	**165.55**	**176.39**	**0.467**	**over**	**7.50**	**16.00**	**over**	**2.29**
12-93	**"B" Zone**	**100E**	**366.60**	**392.19**	**0.302**	**over**	**28.00**	**10.35**	**over**	**8.53**
12-100	"B" Zone	150W	168.27	187.47	0.161	over	12.00	5.52	over	3.66
12-102	"B" Zone	100W	147.59	158.25	0.198	over	10.50	6.78	over	3.20
12-102	**"Main" Zone**	**150W**	**229.93**	**259.77**	**0.216**	**over**	**27.00**	**7.40**	**over**	**8.23**
12-104	"B" Zone	200W	205.36	217.50	0.125	over	7.00	4.30	over	2.13
12-106	**"B" Zone**	**100W**	**255.90**	**264.92**	**0.772**	**over**	**5.00**	**26.47**	**over**	**1.52**
12-110	**"B" Zone**	**350W**	**330.71**	**348.57**	**0.213**	**over**	**6.00**	**7.30**	**over**	**1.83**
12-112	"B" Zone	200W	186.95	203.18	0.122	over	22.00	4.18	over	6.71
	"Main" Zone	**300W**	**321.30**	**341.63**	**0.297**	**over**	**13.50**	**10.17**	**over**	**4.11**
12-114	"B" Zone	0+00	394.61	443.41	0.174	over	18.00	5.95	over	5.49
	"Main" Zone	0+00	469.88	519.23	0.152	over	13.50	5.21	over	4.11

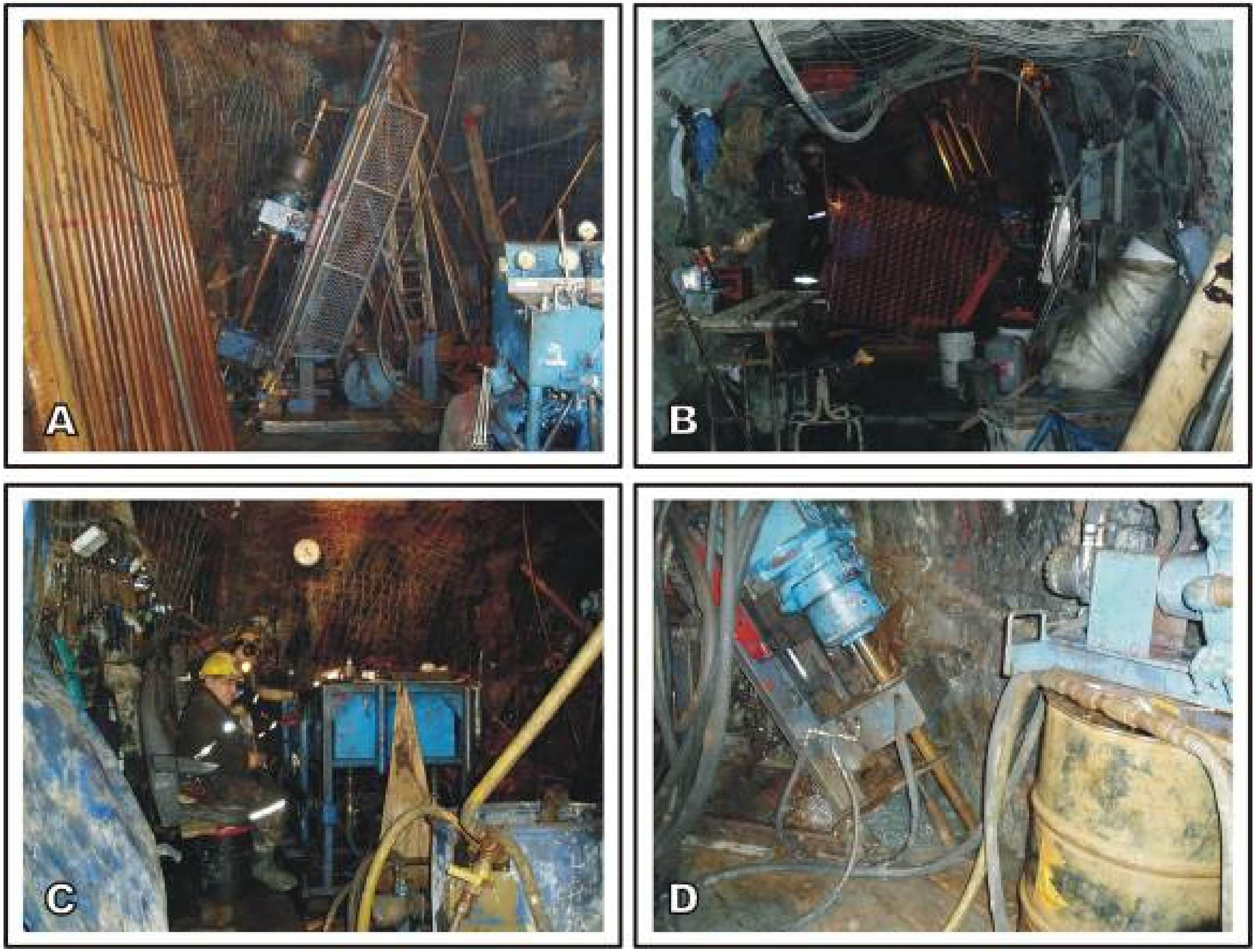


Figure V-1: Pictures from underground drill stations (Bachelor 12th Level). Electric drill rigs used during the 2005 drilling program

Table V-2 Bachelor 2005 underground drilling program – Drilled holes summary table

DDH & STATION		Survey at collar (Flex-it)			Length (m)	Length (ft)	Date		*"MAIN ZONE"* TARGET				*"B ZONE"* TARGET
		Az(Mine Grid)	**Az**(True North)	**Dip**			**Start**	**Finish**	**Easting**	**Northing**	**Elev.** (2)	**Description**	**Description**
12-38	Drill #1	10W	N350	-19.0	87.0	285.4	April 6	April 8	0+55 W	5+65 S	-75	Infill	Infill
12-39	Drill #2	15E	N015	0.0	198.0	649.6	April 13	April15	2+90 E	5+00 S	0	Infill	Infill
12-40	Drill #1	25W	N335	-40.0	89.0	292.0	April 8	April 10	1+00 W	6+20 S	-150	Infill	Infill
12-41	Drill #2	30E	N030	0.0	195.0	639.8	April 15	April 17	4+10 E	5+00 S	0	Infill	East ext.
12-41ext	Drill #2	30E	N030	0.0	66.0	216.5	July 7	July 8	5+50 E	4+75 S	+25	Infill	East ext.
12-42	Drill #1	55W	N305	-1.4	120.0	393.7	April 10	April 12	3+00 W	5+80 S	-0	Infill	Infill
12-43	Drill #2	38E	N038	-0.6	210.0	689.0	April 17	April 19	5+50 E	5+00 S	0	East ext.	East ext.
12-43ext	Drill #2	38E	N038	-0.6	99.0	324.8	May 22	May 24	5+50 E	5+00 S	0	East ext.	East ext.
12-44	Drill #1	65W	N295	-1.2	171.0	561.0	April 12	April 14	4+40 W	5+80 S	-0	West ext.	Infill
12-45	Drill #2	34E	N034	-8.0	215.0	705.4	April 19	April 22	4+50 E	5+40 S	-75	East ext.	Infill
12-46	Drill #1	35W	N325	-55.8	114.0	374.0	April 15	April 16	1+00 W	6+50 S	-225	Infill	Infill
12-47	Drill #2	40.5E	N041	-5.4	213.0	698.8	April 22	April 25	5+30 E	5+40 S	-75	East ext.	East ext.
12-48	Drill #1	27W	N333	-70.3	141.0	462.6	April 17	April 19	1+00 W	7+00 S	-300	Infill	Infill
12-49	Drill #2	22E	N022	-8.7	189.0	620.1	April 25	April 27	3+75 E	5+35 S	-75	Infill	Infill
12-50	Drill #1	58W	N302	-13.4	130.0	426.5	April 19	April 21	3+00 W	6+00 S	-75	Infill	Infill
12-51	Drill #2	20E	N020	-18.5	190.0	623.4	April 27	April 29	3+75 E	5+30 S	-150	Infill	Infill
12-52	Drill #1	73W	N287	-11.5	166.5	546.3	April 21	April 23	4+40 W	6+20 S	-75	West ext.	West ext.
12-53	Drill #2	11W	N349	-12.0	162.0	531.5	April 29	April 30	0+70 E	6+20 S	-75	Central portion	Infill
12-54	Drill #1	65W	N295	-22.0	140.0	459.3	April 23	April 25	3+00 W	6+20 S	-150	West ext.	Infill
12-55	Drill #2	7W	N353	-20.6	156.0	511.8	April 30	May 1	1+30 E	6+50 S	-150	Central portion	Infill
12-56	Drill #1	30.4W	N330	-19.8	90.0	295.3	April 25	April 26	1+30 W	5+75 S	-75	Infill	Infill
12-57	Drill #2	10.2W	N350	-21.3	150.0	492.1	May 2	May 3	0+55 E	6+50 S	-150	Central portion	Infill
12-58	Drill #1	76.3W	N284	-20.7	175.0	574.1	April 26	April 30	4+40 W	6+60 S	-150	West ext.	West ext.
12-59	Drill #2	1.7W	N358	-21.0	171.0	561.0	May 4	May 7	1+47 E	6+95 S	-125	Central portion	Infill
12-60	Drill #1	33W	N327	-79.4	180.0	590.6	April 30	May 2	+60 W	7+20 S	-375	Infill	Infill
12-61	Drill #2	6.2E	N006	-21.9	184.0	603.7	May 7	May 10	1+82 E	6+95 S	-128	Central portion	Infill
12-62	Drill #1	72W	N288	-34.5	180.0	590.6	May 3	May 4	3+50 W	7+00 S	-225	West ext.	Infill
12-63	Drill #2	1.5W	N349	-39.8	204.0	669.3	May 11	May 13	1+15 E	6+70 S	-315	Central portion	Central portion
12-64	Drill #1	58.6W	N301	-28.5	217.5	713.6	May 5	May 7	4+50 W	7+00 S	-225	West ext.	Infill
12-65	Drill #2	1.4E	N001	-55.0	178.0	584.0	May 13	May 16	1+21 E	7+30 S	-450	Central portion	Infill
12-65ext	Drill #2	1.4E	N001	-55.0	56.0	183.7	May 21	May 22	1+21 E	7+30 S	-450	Central portion	Infill
DDH & STATION		**Survey at collar (Flex-it)**			**Length (m)**	**Length (ft)**	**Date**		**MAIN ZONE TARGET**				**B ZONE TARGET**
		Az (Mine Grid)	**Az** (True North)	**Dip**			**Start**	**Finish**	**Easting**	**Northing**	**Elev.** (2)	**Description**	**Description**

12-66	Drill #1	47W	N313	-22.0	170.0	557.7	May 7	May 9	2+30 W	5+90 S	-120	West ext.	Infill
12-67	Drill #2	6.5E	N007	-69.0	235.0	771.0	May 17	May 21	1+36 E	9+05 S	-376	Central portion at depth	Central portion at depth
12-68	Drill #1	10.5W	N350	-41.0	144.0	472.4	May 9	May 10	+52 W	6+10 S	-150	Infill	Infill
12-69	Drill #2	24.5E	N025	-38.3	183.0	600.4	May 25	May 26	2+64 E	7+15 S	-295	Central portion	Infill
12-69ext	Drill #2	24.5E	N025	-38.3	69.0	226.4	June 24	June 25	2+64 E	7+15 S	-295	Central portion	Infill
12-70	Drill #1	46W	N314	-41.0	135.0	442.9	May 11	May 12	1+15 W	7+50 S	-195	Infill	Infill
12-71	Drill #2	20.2W	N340	-58.8	186.0	610.2	May 27	May 28	2+25 E	8+26 S	415	Central portion at depth	Infill
12-72	Drill #1	44.2W	N316	-65.8	174.0	570.9	May 15	May 18	1+44 W	6+76 S	-345	Infill	Infill
12-73	Drill #2	11W	N349	-70.8	229.0	751.3	May 28	June 2	1+96 E	10+09 S	-420	Central portion at depth	Central portion at depth
12-74	Drill #1	48W	N312	-77.0	160.5	526.6	May 18	May 21	1+78 W	6+44 S	-470	Infill	Infill
12-75	Drill #2	60W	N300	-49.6	225.0	738.2	June 4	June 6	2+70 W	8+00 S	-420	West ext.	West ext. at depth
12-76	Drill #1	58.6W	N301	-43.6	156.3	512.8	May 21	May 23	2+23 W	7+70 S	-215	West ext.	Infill
12-77	Drill #2	52W	N308	-48.2	177.0	580.7	June 7	June 8	1+55 W	7+90 S	-370	West ext.	Infill
12-78	Drill #1	63W	N297	-55.6	153.0	502.0	May 23	May 25	1+85 W	7+24 S	-255	Infill	Infill
12-79	Drill #2	98E	N098	-87.6	69.0	226.4	June 8	June 10	1+50 E	10+50 S	-1080	Central portion at depth	Central portion at depth
12-79ext	Drill #2	98E	N098	-87.6	315.0	1033.5	July 9	July 15	1+50 E	10+50 S	-1080	Central portion at depth	Central portion at depth
12-80	Drill #1	66W	N294	-9.3	162.0	531.5	May 25	May 27	3+75 W	6+37 S	-60	West ext.	West ext.
12-81	Drill #2	91.2W	N269	-30.7	530.0	1738.8	June 10	June 22	9+10 W	9+50 S	-370	West ext.	West ext.
12-82	Drill #1	88W	N272	-51.0	171.0	561.0	May 27	May 29	3+46 W	8+00 S	-332	West ext. at depth	Infill
12-82ext	Drill #1	88W	N272	-51.0	46.0	150.9	June 6	June 7	3+46 W	8+00 S	-332	West ext. at depth	Infill
12-83	Drill #2	1.3W	N359	-34.9	195.0	639.8	June 26	June 29	2+50 E	5+80 S	-289	Central portion	Infill
12-84	Drill #1	72.4W	N288	-71.8	183.0	600.4	May 29	June 1	3+46 W	8+00 S	-332	Infill	Infill
12-85	Drill #2	13.6E	N014	-18.2	213.0	698.8	June 29	July 1	3+20 E	5+40 S	-219	Infill	Infill
12-86	Drill #1	66.4W	N294	8.9	190.4	624.7	June 1	June 4	3+87 W	6+31 S	+68	Connection with Hewfran	West ext.
12-87	Drill #2	5.4W	N355	-30.7	165.0	541.3	July 1	July 4	1+55 E	6+30 S	-234	Central portion	Infill

DDH & STATION		Survey at collar (Flex-it)			Length (m)	Length (ft)	Date		MAIN ZONE TARGET				B ZONE TARGET
		Az (Mine Grid)	Az (True North)	Dip			Start	Finish	Easting	Northing	Elev. (2)	Description	Description
12-88	Drill #1	78.4W	N282	4.6	189.8	622.5	June 4	June 6	5+14 W	6+70 S	+40	West ext.	West ext.
12-89	Drill #2	27.3E	N027	-22.6	209.0	685.7	July 4	July 7	4+45 E	4+95 S	-204	Infill	East ext.
12-90	Drill #1	55.5W	N305	8.3	162.0	531.5	June 7	June 10	3+46 W	8+00 S	+82	Infill	West ext.
12-91	Drill #2	66.4W	N294	-52.7	255.0	836.6	July 17	July 21	2+00 W	9+40 S	-546	West ext. at depth	West ext. at depth
12-92	Drill #1	65W	N295	14.4	208.5	684.1	June 14	June 19	4+95 W	5+32 S	+132	Connection with Hewfran	West ext.
12-93	Drill #2	11.7W	N348	-34.5	157.0	515.1	July 15	July 17	0+50 E	7+62 S	-236	Central portion	Infill
12-94	Drill #1	94.8W	N265	0.0	235.0	771.0	June 10	June 14	6+80 W	7+95 S	0	West ext.	West ext.
12-95	Drill #2	22.1E	N022	-64.9	243.0	797.2	July 22	July 25	1+75 E	5+55 S	-76	Central portion at depth	Central portion at depth
12-96	Drill #1	95.7W	N264	-14.7	201.0	659.4	June 19	June 21	4+75 W	7+95 S	-123	West ext.	West ext.
12-98	Drill #1	88.2W	N272	-4.5	234.0	767.7	June 22	June 24	5+10 W	7+36 S	-32	West ext.	West ext.
12-100	Drill #1	55.5W	N305	18.5	153.0	502.0	June 25	June 26	3+65 W	5+65 S	+145	Connection with Hewfran	West ext.
12-102	Drill #1	38.2W	N322	-32.7	111.0	364.2	June 26	June 28	1+60 W	5+95 S	-148	Infill	Infill
12-104	Drill #1	61.9W	N298	-22.4	174.0	570.9	June 28	July 1	4+25 W	5+50 S	+207	Connection with Hewfran	West ext.
12-106	Drill #1	41.5W	N319	-62.4	115.0	377.3	July 1	July 6	1+50 W	6+45 S	-328	Infill	Infill
12-108	Drill #1	69.5W	N291	13.4	201.0	659.4	July 6	July 8	5+75 W	6+15 S	+137	Connection with Hewfran	West ext.
12-110	Drill #1	73.4W	N287	2.2	247.1	810.7	July 8	July 12	5+70 W	6+25 S	0	West ext.	West ext.
12-112	Drill #1	66W	N294	-15.5	150.0	492.1	July 12	July 14	2+50 W	6+00 S	-60	Infill	Infill
12-114	Drill #1	145E	N145	-86.0	183.0	600.4	July 14	July 17	0+00 E	8+00 S	-626	Central portion at depth	Central portion at depth
12-116	Drill #1	88.7E	N089	-44.0	444.0	1456.7	July 18	July 26	5+40 E	8+07 S	-571	West ext.	West ext.

Total drilled from station #1	**6 854.55 m.**	**(22 488.68 ft)**
Total drilled from station #2	**6 491.00 m.**	**(22 488.68 ft)**
TOTAL	**13 345.55 m.**	**(44 977.36 ft)**

Table V-3 Deviation Test Statistics 1

	12-39	**12-41**	**12-42**	**12-43**	**12-44**	**12-45**	**12-46**	**12-47**	**12-48**	**12-49**	**12-50**	**12-51**	**12-52**	**12-53**	**12-54**	**12-55**	**12-56**	**12-57**	**12-58**	**12-59**	**12-63**	**12-65**	**12-67**
Non-magnetic intervals measurements	55 545	56 614	55 410	57 355	56 050	56 119	55 273	56 794	55 867	57 707	56 403	56 594	55 193	56 009	56 773	56 250	55 954	55 913	56 310	56 762	55 148	56 067	55 318
	55 840	56 898	55 406	57 319	56 371	55 812	55 544	57 750	56 089	56 861	57 219	56 363	55 197	55 100	56 773	55 953	56 810	54 954	56 196	56 448	55 600	56 076	55 322
	56 182	57 197	56 013	57 614	54 773	56 822	55 463	56 087	55 847	56 806	57 299	55 627	55 499	55 116	56 769	56 921	56 861	54 954	56 195	56 243	56 135	56 253	55 314
	56 652	59 177	56 205	57 313	55 108	56 582	54 733	56 994	56 305	56 695	56 988	55 777	55 686	55 269	56 773	56 595	56 539	54 951	56 211	56 226	55 909	55 146	55 320
	56 837	57 817	57 209	57 524	55 235		53 603	56 859	55 168	57 304	56 559	57 194	55 297	54 175	56 790	57 086		55 870	56 349	55 962	56 256	56 133	55 313
	57 402	56 391	56 827	56 536	55 491		54 408	57 233	56 208	56 695		56 033	54 793	55 757	55 652	58 315		55 865	55 395	56 085	56 157	56 139	54 802
	56 957	57 689		56 760	56 010		54 913	57 457	56 310	56 753			57 020	55 361	55 366	57 642		55 604	56 868	55 977	55 575	55 362	55 081
	57 154	57 655		57 576	55 727		55 834	56 465	55 823	56 689			55 744	55 177	56 139			55 690	55 869	56 122	55 907	56 719	55 280
	57 161	56 464		57 862	55 543		56 163	56 741	55 749	57 004			55 240	55 430	55 856			55 762	55 237	56 049	56 173	55 818	55 278
	57 156	57 488		57 343	55 864		56 159	56 793					55 372	56 189	56 163			55 605	55 298	56 919	56 107	55 536	55 399
	57 838	56 974		56 923	56 636								55 717	55 368	55 888			56 335	55 923	56 490	56 620	56 134	56 061
	58 271	55 488		56 987	56 414								55 788	55 002	55 699			56 257	55 683	55 196	56 550	55 519	56 047
	57 505	56 497		56 864	55 833								55 540	55 316	55 857			56 025	55 068	55 295	57 235	56 789	56 544
	57 694	57 597		57 000	56 545								56 405	55 486	55 800			55 971		55 329	56 855	56 383	56 121
	58 395			57 139	55 601								56 421		55 876			55 842		55 963	56 865	55 951	54 991
				57 407	56 390								56 213		55 199			55 568		57 477	56 162	55 598	56 452
				57 128	56 417								56 435		55 797			55 453		56 592	56 776	55 916	55 473
				56 911	57 051								56 770		56 486			55 403		56 536	55 663	56 034	55 694
				57 002	56 169								56 233		56 072			55 492		56 100	55 310	55 346	55 538
				56 982	56 904								56 701		56 286			55 352		56 236	56 462	56 624	55 294
				56 859	56 403								56 005		55 255			55 622		56 304	56 607	58 320	54 660
				56 911	56 790								56 554					55 585			55 285	56 723	56 252
				56 963	55 938								56 476								55 672	56 355	55 836
				56 400									56 630								55 274	55 127	57 240
				56 817									56 454								55 007	56 451	52 892
				56 367									55 301								54 769	55 713	56 489
				56 536									56 318								54 603	56 838	56 129
				56 517									56 260								55 026	55 857	55 781
				56 403									55 796								56 494	56 793	55 941
				57 618									56 064								56 767	56 297	55 839
				59 495									56 073								56 700		55 675
				56 403									55 817										55 983
																							56 082
																							55 988
																							54 037
																							56 515
																							56 311
																							55 389
																							55 300
																							55 540
																							55 587
AVERAGE	**57 106**	**57 139**	**56 178**	**57 089**	**56 055**	**56 334**	**55 209**	**56 917**	**55 930**	**56 946**	**56 894**	**56 265**	**55 969**	**55 340**	**56 060**	**56 966**	**56 541**	**55 640**	**55 892**	**56 205**	**55 989**	**56 134**	**55 612**
STDDEV	**817**	**878**	**735**	**596**	**589**	**454**	**811**	**477**	**358**	**348**	**397**	**580**	**556**	**479**	**513**	**813**	**416**	**377**	**530**	**533**	**697**	**643**	**733**
MEDIUM	57 156	57 086	56 109	56 985	56 050	56 351	55 368	56 827	55 867	56 806	56 988	56 198	56 035	55 339	55 888	56 921	56 675	55 614	55 923	56 226	56 135	56 105	55 587
MAX	58 395	59 177	57 209	59 495	57 051	56 822	56 163	57 750	56 310	57 707	57 299	57 194	57 020	56 189	56 790	58 315	56 861	56 335	56 868	57 477	57 235	58 320	57 240
MIN	55 545	55 488	55 406	56 367	54 773	55 812	53 603	56 087	55 168	56 689	56 403	55 627	54 793	54 175	55 199	55 953	55 954	54 951	55 068	55 196	54 603	55 127	52 892
CONDMAX	57 973	57 964	56 844	57 581	56 639	56 805	56 179	57 304	56 225	57 154	57 385	56 778	56 590	55 818	56 401	57 734	57 091	55 990	56 453	56 759	56 832	56 748	56 320
CONDMIN	56 339	56 207	55 374	56 388	55 461	55 896	54 557	56 349	55 509	56 458	56 591	55 618	55 479	54 859	55 375	56 108	56 258	55 237	55 393	55 693	55 438	55 461	54 854
# measurements	15	14	6	32	23	4	10	10	9	9	5	6	32	14	21	7	4	22	13	21	31	30	41

TOTAL AVERAGE	**56 279**		
STDDEV	**577**		
MEDIUM	56 178		
	56 855		
# holes used	23	MAX	59 495
# measurements	379	MIN	52 892

Appendix VI : 2005 Sampling, Laboratory Protocol,Qa/Qc Analysis

APPENDIX VI – Sampling, laboratory protocol, QA/QC analysis and data verification

Sampling protocol

For the 2005 drilling program, the core sampling protocol was established by Innovexplo. Once the drilling core was extracted, the sampling method was as follows:

1) Core was washed with water sprayed from a hose;
2) Before any core handling, pictures of the core boxes were taken in its entirety;
3) Once the geology and location of the samples were described, the geologist marks the start and end of the sample directly onto the core with a coloured wax crayon while the core is still intact in the core box;
4) The core is generally sampled over regular intervals varying between 50 cm minimum and 1.5 m maximum. Some exceptions, samples were taken on a 30-cm length and up to 2 m in azimuth holes;
5) Samples are measured to the nearest tenth of a centimetre, but sample intervals have to coincide with major lithological boundaries;
6) A sample tag, especially made of waterproof paper and indelible ink, is placed at the start of the sample interval. Each sample number is unique and entered in the database. Two distinct series are used: one for regular analysis and another for metallic sieve samples (where visible gold has been described);
7) Blanks and standards tags (as mentioned in Item 15) were at that time inserted by the geologist into core boxes;
8) Samples were cut at Bachelor Lake Mine site (2 726 samples). In order to accelerate the sampling rate, some samples were sent for cutting at Metanor’s core shack in Val d’Or (654 samples) and others were split at Innovexplo’s core shack in Val d’Or (175 samples). In all cases, samples were cut in half, lengthwise, using a diamond core saw (or split) in order to provide witness samples (Figure VI-1),
9) Half the sample (assay sample) is placed separately in a stapled plastic bag. The other half returns to the box according to its original orientation (the proper end of the core, up hole) and retained for future reference;
10) In the case of “grinded core”, samples are taken by hand with a scoop and a representative part is kept in the core box;
11) The other identical sample tag is stapled into the core box at the end of the marked sample interval;
12) Samples with visible gold (which are marked with a distinct series) are sent in separate smaller batches for metallic sieve assays;
13) For each shipment of 25 samples, a shipping memo was completed. The request form specifies the name of the laboratory, the person making the request, the date, the sample series, assaying method, the units for the results to be reported (g/t Au), the analytical method and any other special instructions;
14) One CRM sample and a blank sample were introduced within each batch of 23 core samples. Every core shipment to the ALS Chemex Chimitec laboratory included 3 batches of samples (75 samples).
15) A copy of the request form is made and kept by the geologist at the core shack’s office;
16) Batches of 25 samples are grouped together according to the sample numbers filled in the request. Each bag of 25 samples (“rice” bags) is marked by “Bachelor”, the laboratory name and a number (1/3, 2/3, etc…) according to the request form which is inserted in the first bag (“rice” bag);
17) Each bag of 25 samples are tied with a « tie wrap » and then sealed with an Innovexplo tag that needs to be ripped in order to open the bag;

Figure VI-1: Core saw used on the Bachelor Lake site during 2005 U/G drilling program

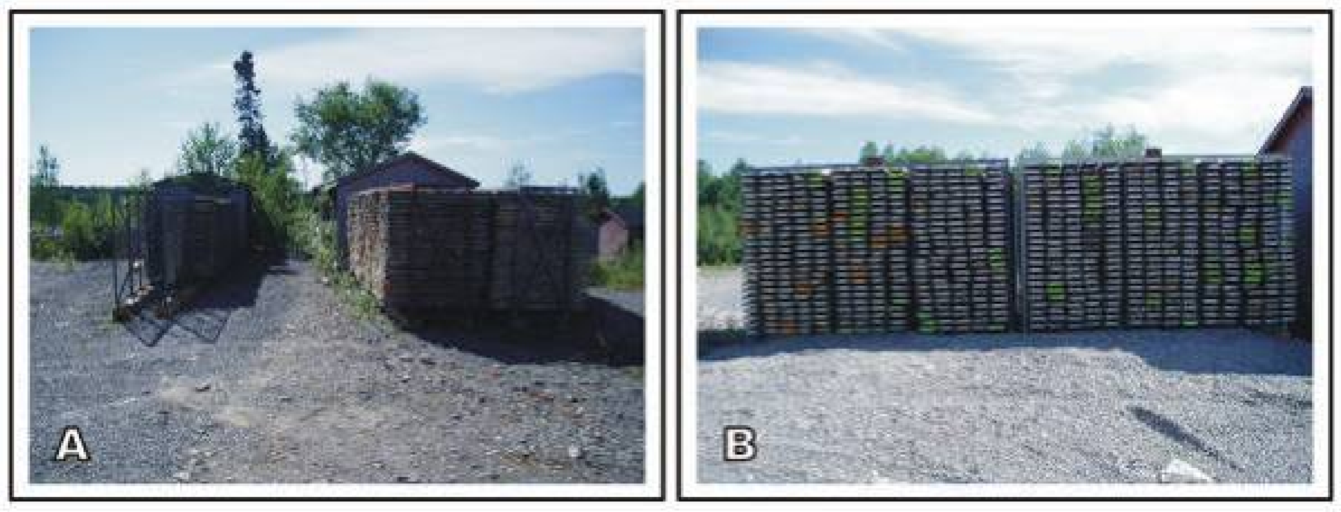


Figure VI-2: Core storage pictures of the 2005 drilling program at Bachelor Lake mine site

Sample shipment and security

All core samples from the 2005 underground drilling program were sent to ALS Chemex Chimitec in Val-d'Or, certified ISO 9001:2000. Each sawed core sample (or split) weighted ± 1 to 4 kg as shown in the Table VI-1. Once the core was sampled, the shipping method for the samples was as follows:

1) Samples were sent by bus to the laboratory on a regular basis (approximately every two (2) days);
2) The samples were brought to Desmaraisville's bus station by a team member of Innovexplo and placed in the luggage compartment of the bus (to avoid leaving the bags without any supervision in Desmaraisville). The bags remained in the luggage compartment until Val d'Or station where they were transferred to the parcel room facilities;
3) A call is made and a fax is sent to the laboratory in Val d'Or with the request form to pick up the samples;
4) Samples are picked up and transported to the laboratory by one of its team member;

5) For the samples cut in Val d'Or, the bags were transported directly to the laboratory and a copy of the request form was sent to the Bachelor's core shack office;

Table VI-1 Sample weight illustrated per length (half core)

Sample length (m)	Mean density(1) (g/cm^3)	Core volume (cm^3/m) (2)	Sample weight (half core) (g)
0.3	2.755	1 040	429.78
0.5	2.755	1 040	716.30
1.0	2.755	1 040	1 432.60
1.5	2.755	1 040	2 148.90
2.0	2.755	1 040	2 865.20

Laboratory protocol

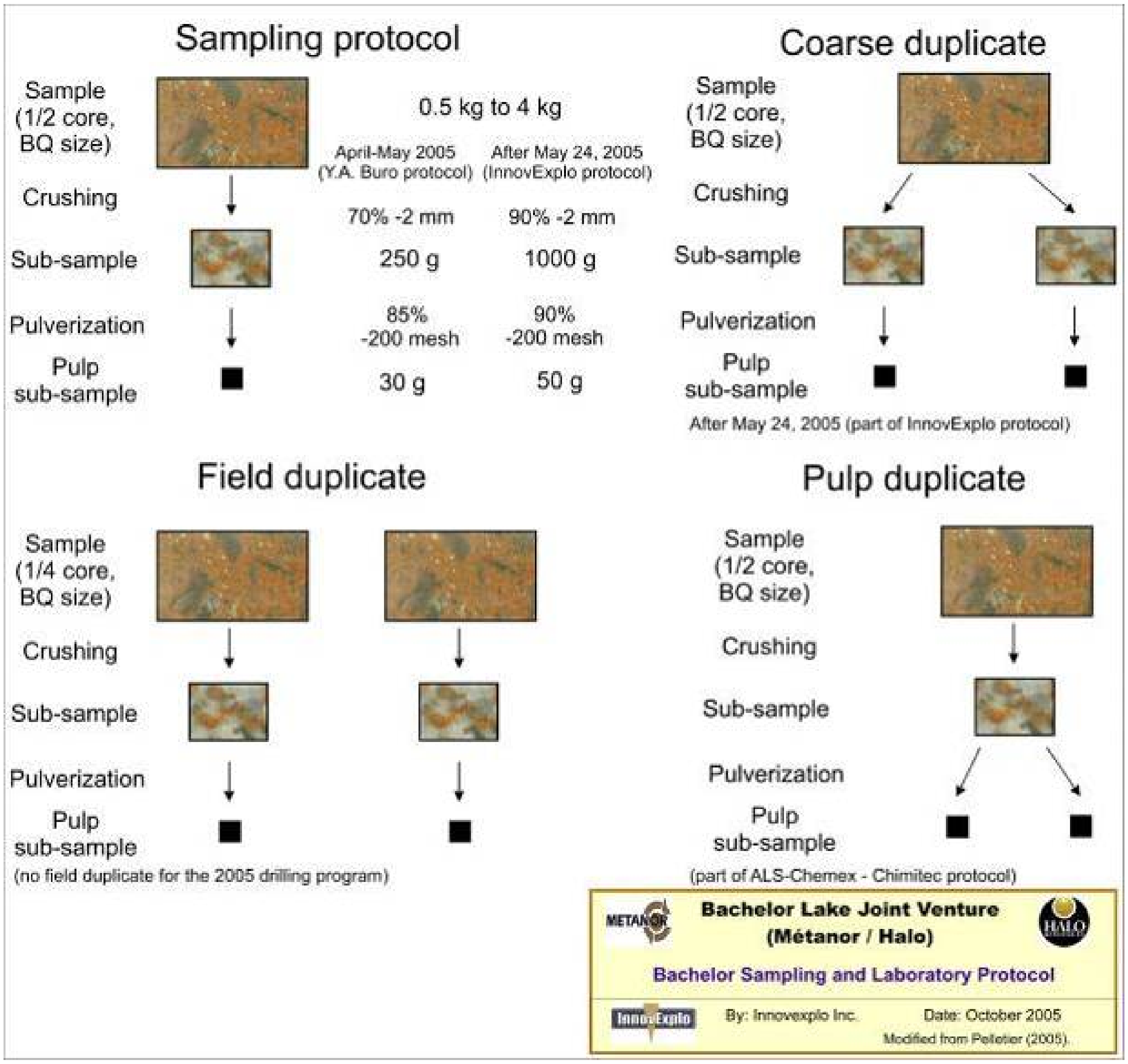


Figure VI-3: Summary of sampling and laboratory protocols used during the 2005 underground drilling program

Table VI-2 Number of sample, standards and blanks – 2005 drilling program

Hole ID	Nbr sample	*#STD + BLK*	Hole ID	Nbr sample	*#STD + BLK*	Hole ID	Nbr sample	*#STD + BLK*
12-38	31	*2*	**12-61**	37	*4*	**12-84**	27	*2*
12-39	9	*0*	**12-62**	56	*7*	**12-85**	20	*1*
12-40	46	*6*	**12-63**	51	*4*	**12-86**	63	*4*
12-41	43	*3*	**12-64**	60	*4*	**12-87**	46	*6*
12-42	31	*2*	**12-65**	60	*6*	**12-88**	43	*5*
12-43	32	*2*	**12-66**	54	*6*	**12-89**	49	*5*
12-44	25	*2*	**12-67**	39	*5*	**12-90**	29	*2*
12-45	55	*3*	**12-68**	67	*6*	**12-91**	27	*6*
12-46	42	*2*	**12-69**	42	*2*	**12-92**	24	*2*
12-47	26	*3*	**12-70**	58	*4*	**12-93**	42	*4*
12-48	36	*2*	**12-71**	28	*2*	**12-94**	42	*4*
12-49	28	*2*	**12-72**	36	*4*	**12-95**	50	*4*
12-50	53	*3*	**12-73**	31	*4*	**12-96**	53	*7*
12-51	26	*4*	**12-74**	49	*6*	**12-98**	66	*5*
12-52	80	*8*	**12-75**	59	*4*	**12-100**	35	*5*
12-53	73	*8*	**12-76**	47	*4*	**12-102**	37	*2*
12-54	67	*6*	**12-77**	57	*6*	**12-104**	42	*4*
12-55	78	*8*	**12-78**	44	*6*	**12-106**	35	*3*
12-56	60	*5*	**12-79**	38	*4*	**12-108**	37	*5*
12-57	68	*7*	**12-80**	58	*5*	**12-110**	91	*8*
12-58	57	*5*	**12-81**	48	*5*	**12-112**	56	*6*
12-59	46	*3*	**12-82**	71	*6*	**12-114**	85	*8*
12-60	36	*2*	**12-83**	45	*7*	**12-116**	69	*7*

	3 251	**Samples**	**24.4%**	(of total length drilled)
	154	***Standards***	***4.33%***	*(of total samples)*
	150	***Blanks***	***4.22%***	*(of total samples)*
GRAND TOTAL	**3 555**	**Samples**		

Table VI-3 Standard Reference Information

Geostats Round Robins (Australia)

STANDARD ID	Reference materiel	Au mean values (ppm)			
	#	Fire assay (50 g)	Dev.	Aqua Regia	Dev.
S1	G901-7	**1.52**	*0.06*	**1.53**	*0.11*
S2	G302-1	**0.43**	*0.03*	**0.42**	*0.08*
S3	G901-6	**21.9**	*0.91*	n.a.	*n.a.*
S4	G396-9	**3.43**	*0.17*	**3.25**	*0.29*
S5	G903-9	**11.26**	*0.41*	**11.15**	*0.77*
S6	G399-6	**2.52**	*0.12*	**2.39**	*0.16*
S7	G302-6	**0.99**	*0.05*	**0.99**	*0.09*

Rocklabs Ltd (New Zeland)

STANDARD ID	Reference materiel		Mean values (ppm)			
			Au		Ag	
	#	Jar Number	Fire assay (30 g)	Dev.	Aqua Regia	Dev.
S8	SK21	89828	**4.048**	*0.04*	n.a.	*n.a.*
S9	SK21	89831	**4.048**	*0.04*	n.a.	*n.a.*
S10	SP17	81163	**18.13**	*0.18*	**59.16**	*1.34*
S11	SP17	81165	**18.13**	*0.18*	**59.16**	*1.34*

QA/QC analysis

Contamination, accuracy and precision

From the total samples (3 555), 150 were blanks (4.22%) and 154 were standards (4.33%). More than 8% of the samples were control samples. No contamination was identified during the 2005 program. The good performance of the laboratory for external standards (field standard) is an evidence of accurate determinations being made by the laboratory. Horvath and Carrier (2005) have reviewed results from the QA/QC program:

[… The performance of the laboratory during the 2005 drilling campaign was good. No contamination was identified during processing and the accuracy of results as indicated by the certified reference standards used to monitor accuracy, both internally and externally to the lab, were also very good. Precision (i.e. reproducibility) of pulp duplicate sample assays is quite good for a gold deposit. However, additional work should be completed by Halo/Metanor Resources to identify whether larger field samples or finer crushing prior to splitting of samples for pulverisation are required to improve precision for the overall samples. Further details are provided in the Quality Control section of the report. …]

Contamination / Field blanks

Two (2) different blanks were introduced during the 2005 sampling procedure from “assumed” barren local rock source. They were core samples taken from units apparently without mineralization. The first one was collected by Yves A. Buro from an intermediate tuff homogeneous unit between 108.65 m and 132 m in hole 12-41. A homogeneous and massive granite unit from the hole 12-43 was later on chosen by Innovexplo for the second field blank. The blank cores were cut in half and put in bags of 100-150 g each with a tag also intercalated in the regular sample numbers. A blank sample was introduced every 25 samples. Adjacent to a mineralized zone, the blank was moved as close as possible to the end of the potential high grade sample. Figure VI-4 shows results from blank samples. Horvath and Carrier (2005) described the field blanks results:

[… Based on the results of the regular assaying of this field blank during the 2005 drill program, the author believes the material is not suitable for use as a field blank. The field blank has on numerous occasions returned values well above detection limits and clearly contains highly anomalous concentrations of gold. Due to these characteristics, another source of field blank material should be selected for future drilling and/or sampling programs. …]

[… The unacceptably poor performance (i.e. frequency of assays greater than 3 x the detection limit) for the field blank samples, especially in light of relatively good accuracy and precision of the other standards results, suggests the source material used for the standard is not suitable. With no characterization studies having been completed for the field blank material as to its mean grade and expected levels of deviation, it is not possible to conclusively comment on whether contamination may have occurred during sample processing. There is no indicated contamination in the analytical blank standard results and, similarly, the certified reference standard results neither demonstrate any obvious contamination. The lack of characterization studies for the field blank makes suspect the source material rather than contamination during sample preparation. …]

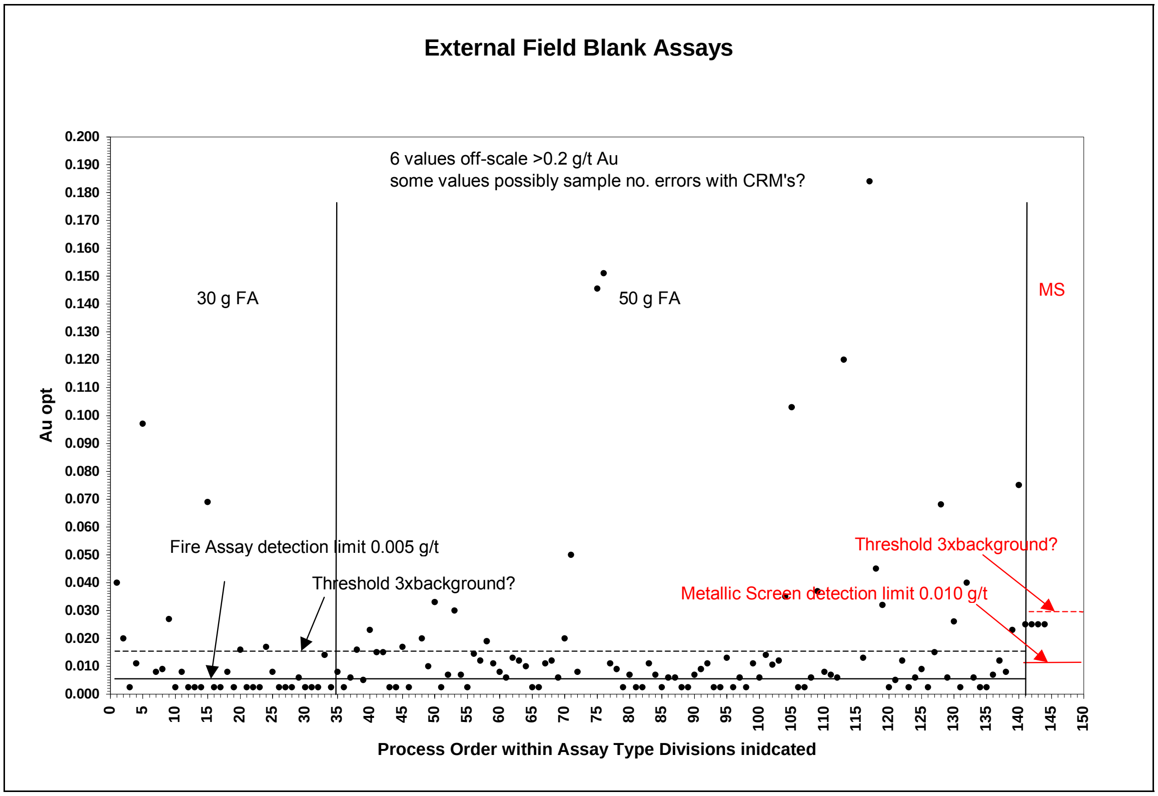


Figure VI-4: Plot of the field blank results obtained during the 2005 program (from Horvath and Carrier (2005)).

Analytical blank

As described in the QA/QC and geostatistical report of Horvath and Carrier (2005):

[… The laboratories internal analytical blank is usually a solution standard used to monitor contamination at the AAS; alternatively it may be a pulp standard introduced at fusion, however, in either case, the analytical blank *does not monitor for contamination* *that may have occurred at the most probable source namely, during sample preparation. …]*

Figure VI-5 presents the results of the internal analytical blank standards during the 2005 program.

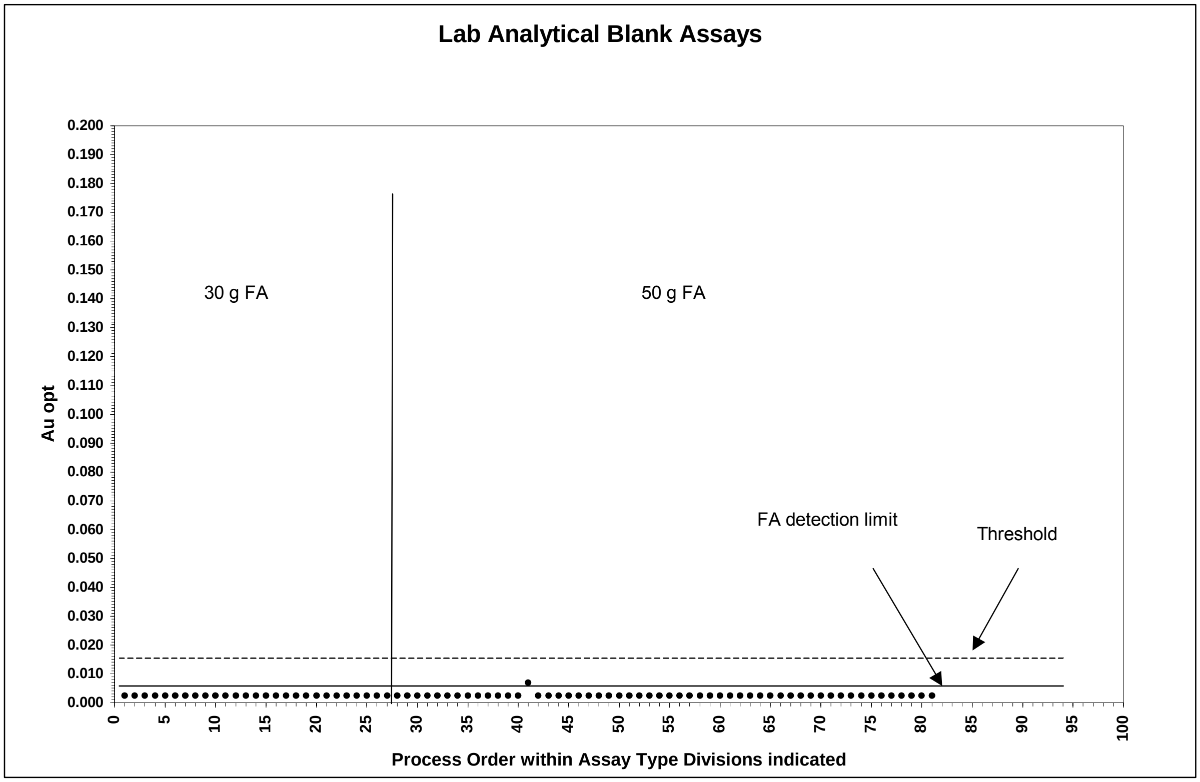


Figure VI-5: Results of the laboratories internal analytical blank standards during the 2005 drilling program (from Horvath and Carrier (2005)).

Accuracy and field standards (Certified Reference Materials)

Accuracy: the measure of an analytical determination to be the “true” value for the sample. Nine (9) certified reference materials (CRM) were bought and used throughout the drilling program to test the accuracy. Seven (7) from Geostats Round Robins: G-901-7 at 1.52 g/t Au, G302-1 at 0.43 g/t Au, G901-6 at 21.9 g/t Au, G396-9 at 3.43 g/t Au, G903-9 at 11.26 g/t Au, G399-6 at 2.52 g/t Au, G302-6 at 0.99 g/t Au. Two others were ordered from Rocklabs: SK21 (jar #: 89 828) at 4.048 g/t Au and SP17 (jar #: 81 163) at 18.13 g/t Au. Details of these references are available in the Appendix VI.

The CRM samples were chosen by the geologist during logging. Field standards were introduced every 25 samples. Adjacent to a mineralized zone, the standard was moved as close as possible to the potential high grade portion in order to test accuracy.

The field standards were packed by the technician under geologist supervision in packets of 30 g, with a tag intercalated in the regular sample numbers.

Conclusion of the QA/QC analysis revealed that generally a good performance of the field standards is an evidence of accurate determinations being made by the laboratory on the processed samples as illustrated in the Figure VI-6.

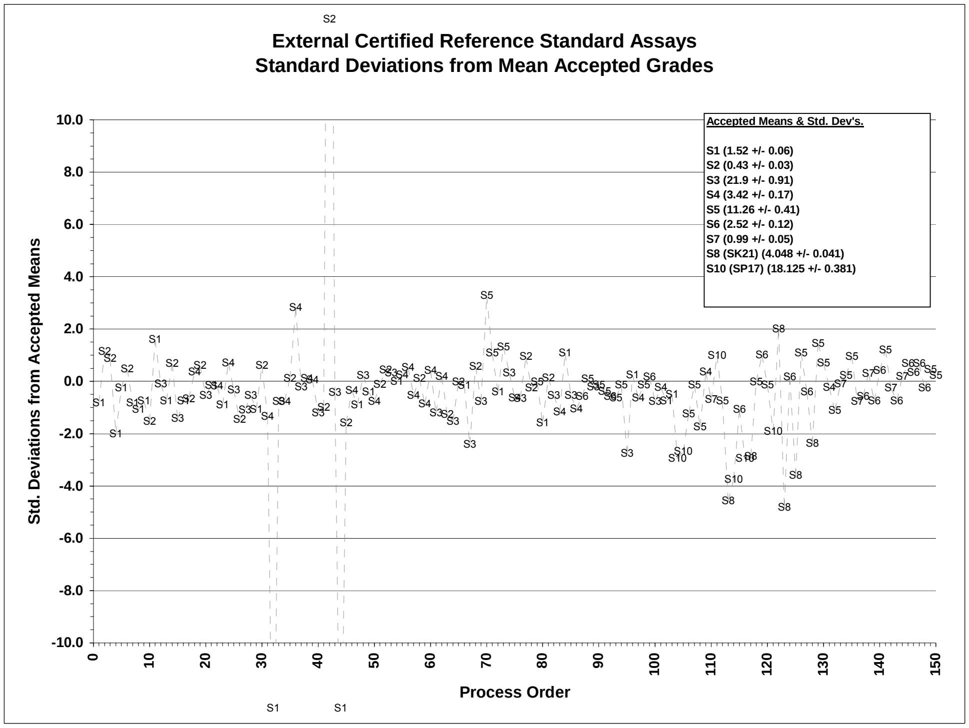


Figure VI-6: Plot of the externally submitted CRM assay results plotted in process order from the 2005 sampling program. The results are plotted in units of standard deviation from the accepted mean grades of the standards. The data is annotated with letter codes identifying the standard type (from Horvath and Carrier (2005)).

Accuracy / Laboratory Certified Reference Standard

As described in the geostatistical report of Horvath and Carrier (2005) and illustrated in Figure VI-7:

[… The plot of all CRM results internal to the lab plotted in process order demonstrates that the accuracy of the laboratory has been quite good over the course of the sampling/assaying program. The vast majority of results fall within the generally accepted 2 standard deviation envelope about the accepted mean grades of the standards. …]

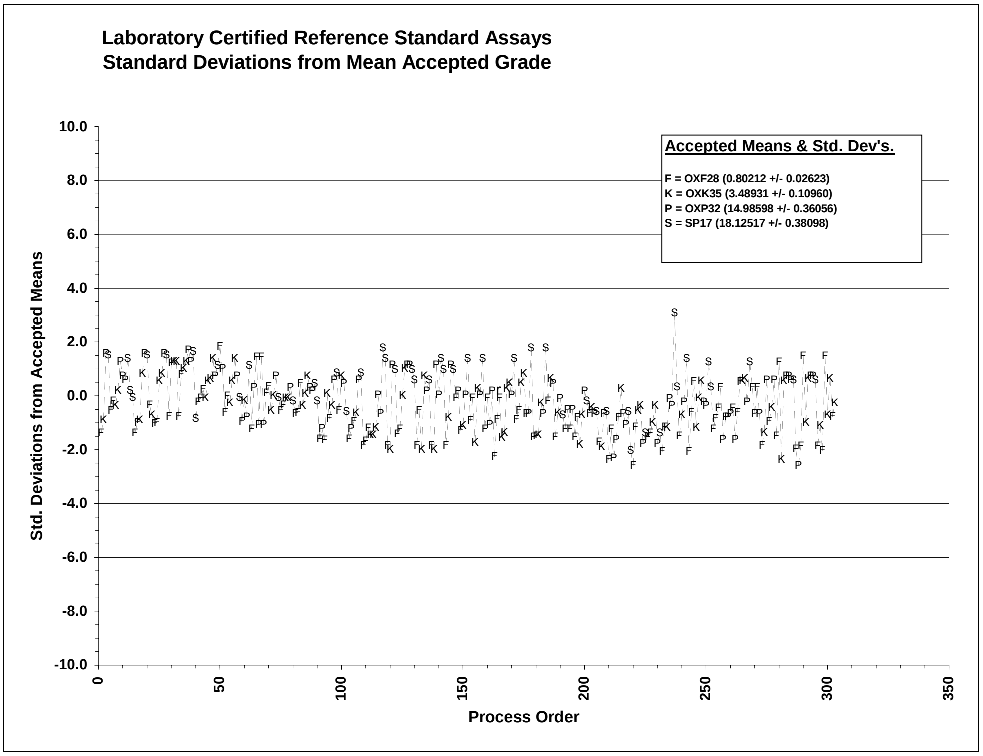


Figure VI-7: Results from all CRM's plotted in process order with symbols used to identify the standard type. All CRM results have been plotted in units of standard deviations from the mean accepted value for that particular standard (from Alex S. Horvath and Alain Carrier (October, 2005)).

Precision / reproducibility (duplicate sample)

Precision is the measure of reproducibility of any value determined for a sample. Innovexplo initiated a relatively simple program of duplicate sampling and assaying during its 2005 sampling and assaying program. Protocols were established with the laboratory to process duplicate samples at various sample process stages as well as for duplicate analytical determinations by differing methods.

In total, there were:
- 129 duplicate fire assays completed on coarse crush duplicate samples, and
- 148 duplicate fire assays from the same prepared pulp samples.

No field duplicate has been taken during the last 2005 drilling program.

QA/QC analysis of the pulp duplicate *[... demonstrates a fair level of precision with overall approximately 12% errors. This level of error is not uncommon of Archean gold deposits where the principal component of the ore is often "freely" liberated gold. In fact, many coarse "nuggety" gold deposits demonstrate much poorer levels of precision in pulp duplicate sample results. ...].*

Figure VI-8 illustrates all pulp duplicate sample fire assay results. All results greater than 10 g/t Au are gravimetric duplicate results since this was the upper limit used for determinations by AAS. The best fit linear regression of the data (shown in red) also demonstrates a near 1:1 correlation. Figure VI-9 illustrates that no bias is introduced by the analytical finish and, furthermore, is a likely indication that the current protocol used for re-assaying initial AAS determinations by gravimetric methods at 5 g/t Au is suitable.

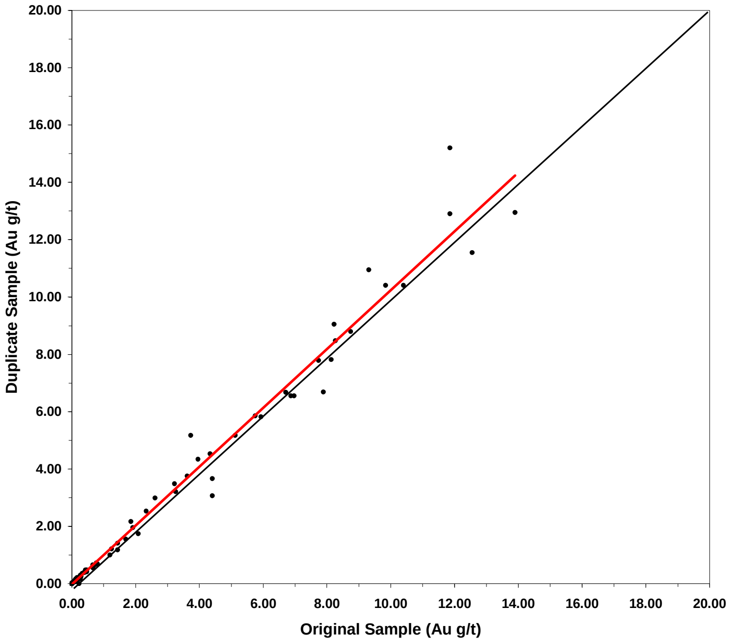


Figure VI-8: Bias Plot of all pulp duplicate sample fire assay results

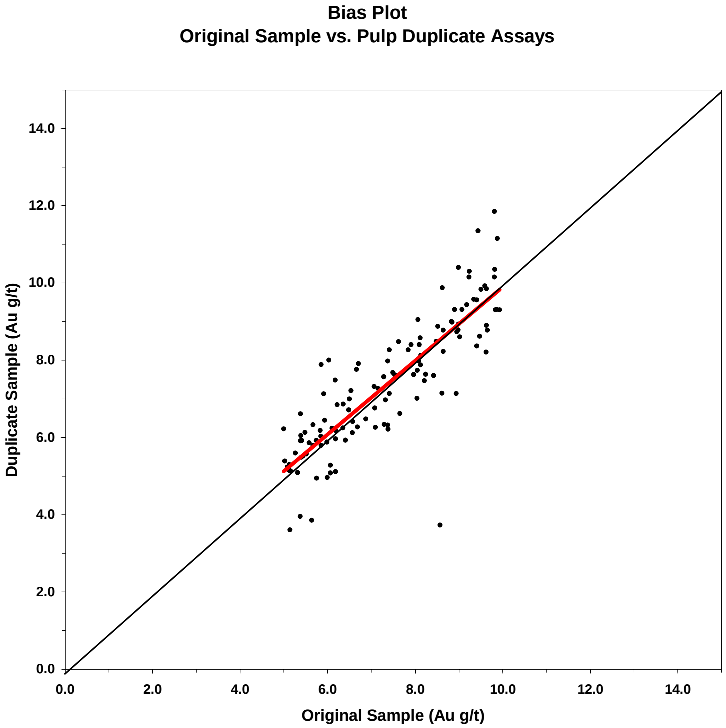


Figure VI-9: Bias plot of the sub-population of AAS vs gravimetric pulp duplicate

Precision of metallic screen – 150 mesh pulp duplicate

The metallic sieve method incorporates duplicate fire assay determinations of the –150 mesh fraction of the screened pulp. As illustrated in Figure VI-10 and described in the QA/QC analysis of Horvath and Carrier (2005):

[… The results demonstrate that precision levels of the screened pulp duplicate assays are overall approximately 6.5%. The better precision of the screened pulps is expected and indicates that the +150 Fraction (i.e. residual coarse gold) contributes approximately 5-6% of the error at the pulp level as demonstrated by the difference between screened pulp duplicate and unscreened pulp duplicate precision levels. A 5% residual "nugget" effect at 150 mesh is quite acceptable for an Archean gold deposit…]

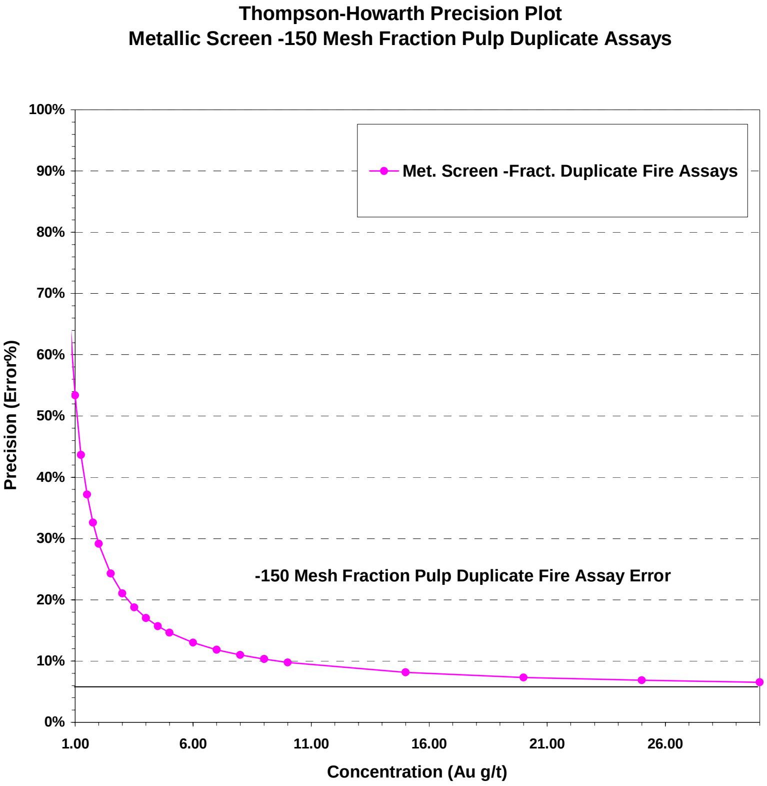


Figure VI-10: Thompson Howarth Precision Plot of the precision at varying concentrations for the metallic screen –150 mesh fraction duplicates

Precision of coarse crush duplicate sample

The result for the coarse duplicate was not that good, as illustrated in Figure VI-11 and described in the QA/QC analysis of Horvath and Carrier (2005):

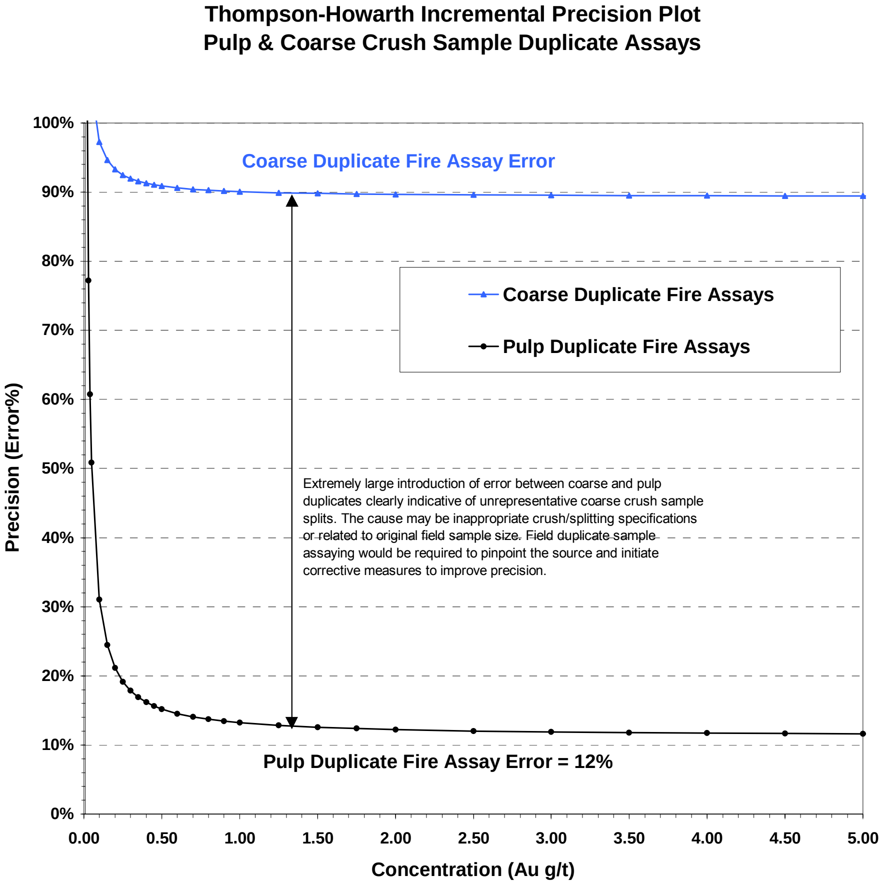
[… The extremely large introduction of error between coarse and pulp duplicates is clearly indicative of unrepresentative 1 kg coarse crush sample splits. The cause may be inappropriate crush/splitting

specifications or related to original field sample size. Field duplicate sampling and assaying would be required to pinpoint the source and initiate corrective measures to improve precision.

As discussed earlier, it is the field sample results that are the most important to be able to reproduce (i.e. have good precision). However, the results demonstrate that, at the coarse crush sample level, there is already virtually no precision in the assay results with some 90% indicated error. There would be no reproducibility (i.e. precision) of any field duplicate value using the existing protocols. Most probably, this error has been introduced by the relatively frequent small sample intervals (<1 feet) used within the mineral zones that produce the unrepresentative coarse crush products and high error in the coarse duplicate samples.

While this type of error may not result in any global change in resource estimation, if locally assays are imprecise, than locally block model grade estimates will also be imprecise. While the global results may remain unchanged, poor mine planning and ore development will result from imprecise assays and grade estimation.

A program of field duplicates sampling should be introduced in future programs. The results should be evaluated early to determine if a minimum sample interval or larger whole core samples might be required to obtain more precise results and to determine the overall precision of final sample assays. …]



Extremely large introduction of error between coarse and pulp duplicates clearly indicative of unrepresentative coarse crush sample splits. The cause may be inappropriate crush/splitting specifications or related to original field sample size. Field duplicate sample assaying would be required to pinpoint the source and initiate corrective measures to improve precision.

Figure VI-11: Thompson Howarth Precision Plot of the precision at varying concentrations for the coarse crush duplicate sample fire assays

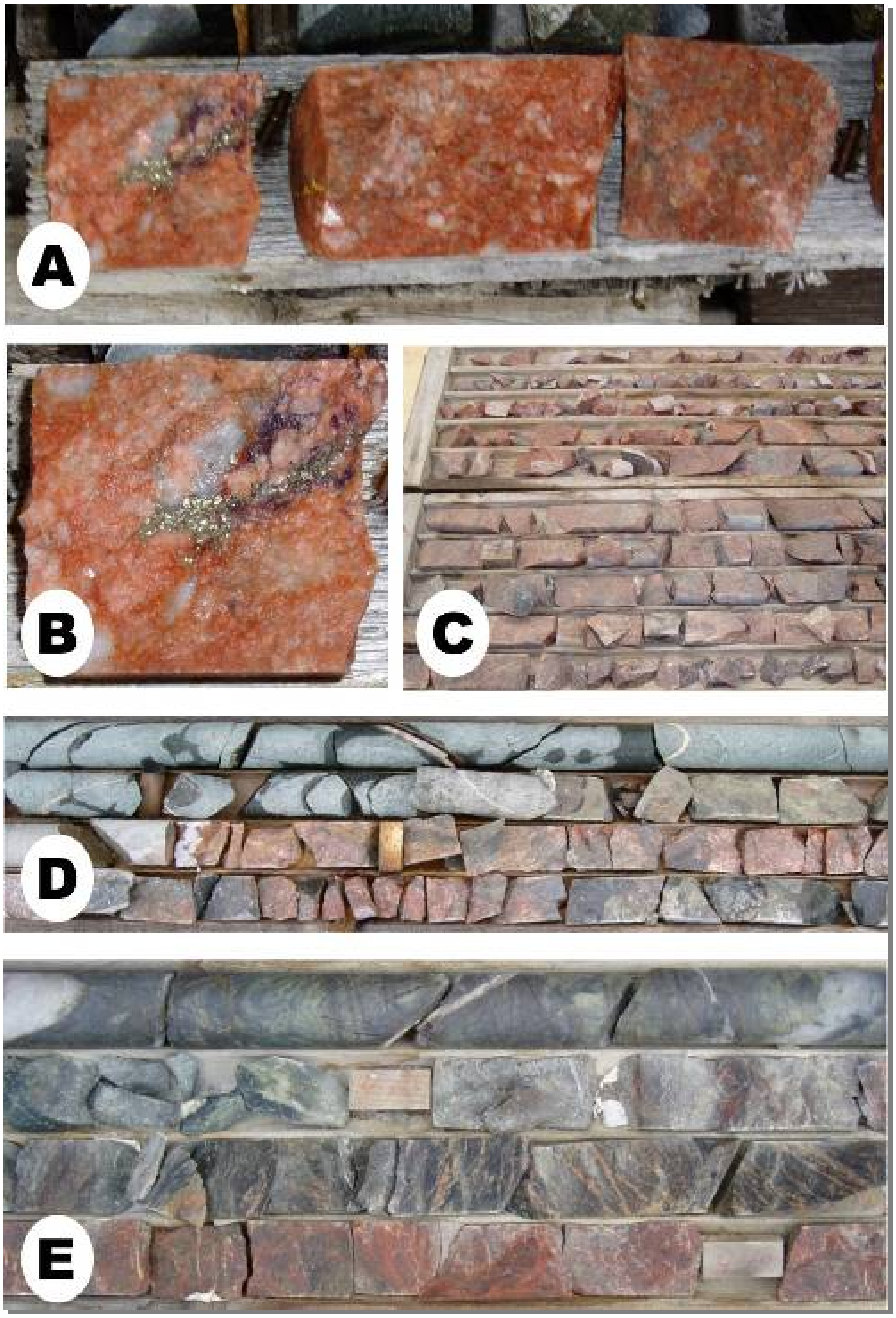

A
B
C
D
E

Figure VI-12: Gold mineralization from diamond drill core at Bachelor Lake property. A.) and B.) Highly hematitized and pyritized interval in hole 12-33 from the Main Zone (8.33 g/t Au over 7.01 m). C.) Hematitized and pyritized interval in hole 12-4 from the Main Zone (17.28 g/t Au over 3.66 m). D.) Hematitized and pyritized interval in hole 12-37 from the "B" Zone (6.69 g/t Au over 2.44 m). E.) Less altered and mineralized interval from hole 12-28 with insignificant gold results in the "B" zone.

Table VI-4 Check assays for the Bachelor and Hewfran resource area

Results of Wolfden check assays for the Bachelor resource area (BQ Hecla historic drill holes)

	Hole ID	From (ft)	To (ft)	Core length (ft)	Original sample	Assay result (oz/t Au)			New sample	Check assay result (WOLFDEN, 2004)						Diff.
						Original	Check	Average						Average (g/t)	Grade (oz/t)	
1	BL 12-15	180.0	193.9	13.90	-	0.260		**0.260**	-					8.23	**0.240**	-0.020
2	BL 12-15	186.0	193.9	7.90	-	0.430		**0.430**	-					13.03	**0.380**	-0.050
4	BL 12-4	708.7	729.0	20.30	-	0.460		**0.460**	-					16.11	**0.470**	0.010
5	BL 12-4	757.0	777.3	20.25	-	0.360		**0.360**	-					10.63	**0.310**	-0.050
						Average (Total)		**0.378**							**0.350**	**-0.028**

Note: One check assay for BL 11-11 (Hecla historic drill holes) gave 0.62 oz/t Au, but no original assay result was available for comparison purpose

Results of Innovexplo 1/2 split check assays for the Hewfran East resource area (AQ Aur Resources historic drill holes)

	Hole ID	From (ft)	To (ft)	Core length (ft)	Original sample	Assay result (oz/t Au)			New sample	Check assay result (g/t Au) INNOVEXPLO, 2005						Diff.
						Original	Check	Average		Au-AA24	Au-Gra22	Pulp Dup	Pulp Dup Check	Average (g/t)	Grade (oz/t)	
6	HU-8-30	233.0	236.0	3.00	53 758	0.317		**0.317**	123 117	>10.0	13.65	>10.0	12.70	13.18	**0.384**	0.067
7	HU-8-30	236.0	239.0	3.00	53 759	0.272		**0.272**	123 118	7.28	7.25			7.27	**0.212**	-0.060
8	HU-8-30	243.0	246.0	3.00	53 761	0.148		**0.148**	123 120	4		4		4.00	**0.117**	-0.031
9	HU-8-30	246.0	249.0	3.00	53 762	0.048		**0.048**	123 121	2.24				2.24	**0.065**	0.017
10	HU-8-37	241.0	242.5	1.50	53 865	0.298		**0.298**	123 122	>10.0	10.1	9.880	9.45	9.81	**0.286**	-0.012
11	HU-8-37	247.5	250.0	2.50	53 868	0.089		**0.089**	123 123	3.76				3.76	**0.110**	0.021
12	HU-8-37	250.0	252.0	2.00	53 869	0.166		**0.166**	123 124	3.34		3.110		3.23	**0.094**	-0.072
13	HU-8-37	254.0	256.0	2.00	53 871	0.297		**0.297**	123 125	>10.0	10.4			10.40	**0.303**	0.006
14	HU-8-37	260.0	262.0	2.00	53 874	1.110		**1.110**	123 126	>10.0	62.8	>10.0	62.70	62.75	**1.830**	0.720
						Average (Total)		**0.305**							**0.378**	**0.073**

Results of Innovexplo 1/4 split check assays for the Hewfran West resource area (BQ Aur Resources historic drill holes)

	Hole ID	From (ft)	To (ft)	Core length (ft)	Original sample	Assay result (oz/t Au)			New sample	Check assay result (g/t Au) INNOVEXPLO, 2005						Diff.
						Original	Check	Average		Au-AA24	Au-Gra22	Pulp Dup	Pulp Dup Check	Average (g/t)	Grade (oz/t)	
15	HU-6-24	348.0	351.0	3.00	61 643	0.196	0.225	**0.211**	123 001	7.13				7.13	**0.208**	-0.003
16	HU-6-24	351.0	353.5	2.50	61 644	0.105	0.094	**0.100**	123 002	1.43		1.505		1.47	**0.043**	-0.057
17	HU-6-24	359.0	361.0	2.00	61 647	0.242	0.270	**0.256**	123 003	>10.0	12.2			12.20	**0.356**	0.100
18	HU-6-24	380.5	382.0	1.50	61 656	0.262	0.324	**0.293**	123 004	3.31				3.31	**0.097**	-0.196
19	HU-6-24	382.0	383.5	1.50	61 657	0.028	0.033	**0.031**	123 005	0.69		0.669		0.68	**0.020**	-0.011
20	HU-6-30	356.0	359.0	3.00	61 785	0.614	0.620	**0.617**	123 006	>10.0	25.7			25.70	**0.750**	0.133
21	HU-6-30	359.0	361.0	2.00	61 786	0.599	0.710	**0.655**	123 008	>10.0	28.8	>10.0		28.80	**0.840**	0.186
22	HU-6-30	361.0	363.0	2.00	61 787	0.428	0.407	**0.418**	123 009	9.73				9.73	**0.284**	-0.134
23	HU-6-30	363.0	366.0	3.00	61 788	0.372	0.497	**0.435**	123 010	>10.0	13.05			13.05	**0.381**	-0.054
24	HU-6-36	437.0	438.0	1.00	63 834	0.225		**0.225**	123 011	1.55				1.55	**0.045**	-0.180
25	HU-6-36	438.0	439.5	1.50	63 835	0.122		**0.122**	123 012	4.76		4.56		4.66	**0.136**	0.014
26	HU-6-36	441.5	443.0	1.50	63 837	0.117		**0.117**	123 013	3.88				3.88	**0.113**	-0.004
27	19501-58	1 049.5	1 052.5	3.00	53 047	0.226		**0.226**	123 014	>10.0	17.5			17.50	**0.510**	0.284
28	19501-58	1 052.5	1 055.5	3.00	53 048	0.539		**0.539**	123 015	5.06		4.87		4.97	**0.145**	-0.394
29	19501-58	1 055.5	1 058.5	3.00	53 049	0.120		**0.120**	123 016	5.33				5.33	**0.155**	0.035
						Average (Total)		**0.291**							**0.272**	**-0.019**

Table VI-5 Check assay results on the Bachelor and Hewfran resource area per cut-off

	Original sample	Assay result (oz/t Au)			New sample	Check assay result						Diff.
		Original	*Check*	**Aver.**		*Au-AA24*	*Au-Gra22*	*Pulp Dup*	*Pulp Dup Check*	**Aver. (g/t)**	**Grade (oz/t)**	
Check on assay below cut-off (> 0.1 oz/t Au)	61 657	*0.028*	*0.033*	**0.031**	123 005	*0.69*		*0.669*		0.68	**0.020**	-0.011
	53 762	*0.048*		**0.048**	123 121	*2.24*				2.24	**0.065**	0.017
	Average			**0.039**							**0.043**	**0.003**
Check on assays close to the cut-off grade (from 0.1 to 0.15 oz/t Au)	61 644	*0.105*	*0.094*	**0.100**	123 002	*1.43*		*1.505*		1.47	**0.043**	-0.057
	63 835	*0.122*		**0.122**	123 012	*4.76*		*4.56*		4.66	**0.136**	0.014
	63 837	*0.117*		**0.117**	123 013	*3.88*				3.88	**0.113**	-0.004
	53 049	*0.120*		**0.120**	123 016	*5.33*				5.33	**0.155**	0.035
	53 761	*0.148*		**0.148**	123 120	*4.00*		*4.00*		4.00	**0.117**	-0.031
	53 868	*0.089*		**0.089**	123 123	*3.76*				3.76	**0.110**	0.021
	Average			**0.116**							**0.112**	**-0.004**
Check on assays close to resource average (from 0.15 to 0.3 oz/t Au)	61 643	*0.196*	*0.225*	**0.211**	123 001	*7.13*				7.13	**0.208**	-0.003
	61 647	*0.242*	*0.270*	**0.256**	123 003	*>10.0*	*12.2*			12.20	**0.356**	0.100
	61 656	*0.262*	*0.324*	**0.293**	123 004	*3.31*				3.31	**0.097**	-0.196
	63 834	*0.225*		**0.225**	123 011	*1.55*				1.55	**0.045**	-0.180
	53 047	*0.226*		**0.226**	123 014	*>10.0*	*17.5*			17.50	**0.510**	0.284
	53 869	*0.166*		**0.166**	123 124	*3.34*		*3.110*		3.23	**0.094**	-0.072
	53 759	*0.272*		**0.272**	123 118	*7.28*	*7.25*			7.27	**0.212**	-0.060
	53 865	*0.298*		**0.298**	123 122	*>10.0*	*10.10*	*9.880*	9.45	9.81	**0.286**	-0.012
	53 871	*0.297*		**0.297**	123 125	*>10.0*	*10.40*			10.40	**0.303**	0.006
	-	*0.260*		**0.260**	-						**0.240**	-0.020
	Average			**0.250**							**0.235**	**-0.015**
Check on high-grade assays (> 0.3 oz/t Au)	61 785	*0.614*	*0.620*	**0.617**	123 006	*>10.0*	*25.70*			25.70	**0.750**	0.133
	61 786	*0.599*	*0.710*	**0.655**	123 008	*>10.0*	*28.80*	*>10.0*		28.80	**0.840**	0.186
	61 787	*0.428*	*0.407*	**0.418**	123 009	*9.73*				9.73	**0.284**	-0.134
	61 788	*0.372*	*0.497*	**0.435**	123 010	*>10.0*	*13.05*			13.05	**0.381**	-0.054
	53 048	*0.539*		**0.539**	123 015	*5.06*		*4.87*		4.97	**0.145**	-0.394
	53 758	*0.317*		**0.317**	123 117	*>10.0*	*13.65*	*>10.0*	12.70	13.18	**0.384**	0.067
	53 874	*1.110*		**1.110**	123 126	*>10.0*	*62.80*	*>10.0*	62.70	62.75	**1.830**	0.720
	-	*0.430*		**0.430**	-						**0.380**	-0.050
	-	*0.460*		**0.460**	-						**0.470**	0.010
	-	*0.360*		**0.360**	-						**0.310**	-0.050
	Average			**0.534**							**0.577**	**0.043**

Figure VI-13: Drill core of the Hewfran claims used for the check assays validation

A) “A West” Zone intercept in the HU-6-30 (BQ size) drill hole from 324.6 ft to 361.2 ft; B) Detail of the HU-6-30 at 365 ft illustrating hydraulic brecciation with altered fragments and 3 to 4% pyrite (0.436 oz/t over 3 ft); C) “A West” zone intercept in the HU-6-24 (BQ size) drill hole illustrating a strongly altered zone with some local shearing.; D) Detail of the Main zone in HU-6-24 around 353 ft; E) Main Zone of Hewfran east HU-8-30 drill core (AQ size); F) Detail of the HU-8-30 around 240 ft; G) HU-8-37 (AQ size) around 256 ft illustrating disseminated pyrite (up to 5-6%) into a strong silicified and hematitized matrix.

Appendix VII : Showings And Deposits Of The Bachelor Area,

APPENDIX VII – Showings and deposits of the Bachelor area
Description and comparison

Table VII-1 Showings in the close vicinity of the Bachelor property

Name	Type	Coordinates (UTM, Nad 83, Zone 18)		Deposit Type	Geological comments	Best assays
		Easting	Northing			
Batch-River	showing	417 139	5 484 778	VHMS	Sulphide and gold rich veins cross cutting volcanic rocks	10.67 % Zn, 0.41 % Pb and 6.56 g/t Ag
Perry showing (Barbie-North Lake)	showing	425 529	5 484 628	Lode Gold	Metric wide hydrothermal breccia cross-cutting mafic rocks and containing pyrite ± chalcopyrite ± specularite	3.94 g/t Au and 3.0 g/t Ag
Bachelor-NE Lake	showing	424 679	5 488 628	VHMS	Disseminated pyrite and chalcopyrite in a gabbro	0.80 % Cu, 4.3 g/t Ag and 400 ppb Au
Bachelor-North Lake	DDH	420 479	5 489 579	VHMS	Silver rich pyrrhotite in graphitic schist and greywacke	0.27 % Zn over 1.53 m and 5.48 g/t Ag over 0.50 m
Billy-North Lake	showing	426 614	5 491 440	VHMS	Sulphide disseminated associated with a shear zone cross-cutting a felsic porphyry	6.5 g/t Ag, 160 ppb Au, 1200 ppm Cu and 510 ppm Zn
Billy-North Lake	DDH	426 875	5 491 862	Lode Gold	Sulphide disseminated associated with a shear zone cross-cutting a felsic porphyry	1253 ppb Au over 1.0
Le Sueur F (North Block)	DDH	416 804	5 486 403	VHMS	Base metal associated with quartz/carbonate veinlets cross-cutting cherty tuffs	42.4 g/t Ag over 1.28 m
LU-03	DDH	425 382	5 492 032	Lode Gold	Disseminated pyrite in strongly deformed and altered sediments	5.55 g/t Au over 1.52 m
LU-01 and LU-02	DDH	423 661	5 492 088	Lode Gold	Disseminated pyrite in strongly deformed and altered sediments	0.33 g/t Au over 17.62 m
Barry Exploration	DDH	411 404	5 481 278	Lode Gold	Strongly carbonatized diorite cross-cut by gold-bearing pyrite rich quartz veins	10.0 g/t Au over 0.30 m
Céré showing	DDH	404 394	5 478 167	Lode Gold	Fine grained disseminated pyrite in a strongly foliate sericitized schist	6.89 g/t Au
McIntyre-1	showing	412 104	5 482 678	Lode Gold	Fine grained disseminated pyrite in quartz/carbonate lenses	17.14 g/t Au
Narsillac Creek	showing	411 254	5 479 653	Lode Gold	Diorite hosting sulphide rich quartz/carbonate veins and veinlets associated with shear zones	5.83 g/t Au
Nelligan	DDH	403 081	5 478 086	Lode Gold	Basalts hosting pyrite rich quartz/carbonate veinlets in shear zones	4143 ppb Au over 1.3 m
Nel-92-02	DDH	406 509	5 478 541	Lode Gold	30% finely disseminated pyrite in quartz veinlets and wall rock cross-cutting sericitic and graphitic schists	1.0 g/t Au over 0.60 m
Castor	DDH	417 102	5 476 964	Lode Gold	Sheared mafic lavas hosting 60% mm to dm wide quartz/carbonate/sulphide	2.56 g/t Au, 16.4 g/t Ag, 2.53 % Zn and 0.11 % Cu over 13.0 m
Couloir Le Tac	DDH	417 757	5 476 693	Lode Gold	Basalts hosting meter wide sheared zones cross-cut by cm to dm wide quartz/ankerite/sulphide	0.17 g/t Au, 40.0 g/t Ag and 2.02 % Cu over 0.5 m
Empire - A Zone	Deposit	417 554	5 476 103	VHMS	Sphalerite, pyrite and pyrrhotite finely disseminated in andesite adjacent to pyroclastic rock and cross-cutting diabase dykes	Hist. Resources: 260 000 t @ 3% Zn
Gand-Bachelor	showing	416 079	5 477 053	VHMS	Chalcopyrite, pyrite and sphalerite in quartz/carbonates veinlets associated with shear zones cross-cutting a sericitic schist	2.09 % Cu, 8.95 % Zn, 6.18 g/t Au and 23.50 g/t Ag
Gand-Bachelor-SW: G Zone	DDH	415 592	5 476 341	VHMS	Pyrite ± sphalerite ± chalcopyrite in quartz/carbonates veinlets associated with shear zones cross-cutting a sericitic schist	27.6 g/t Ag, 1.63 % Cu, 234 ppb Au and 537 ppm Zn over 0.9 m
Soma Alta - C Zone	showing	417 510	5 475 008	VHMS	Sulphide veinlets associated with shear zone cross-cutting a felsic intrusion	1.3 % Zn over 0.3 m, 0.66 % Zn over 0.76 m and 5.14 g/t Au over 0.61 m
D Zone	DDH	416 679	5 475 003	VHMS	Stratiform disseminated sphalerite,	2.63 % Zn over 1.67 m

Name	Type	Coordinates (UTM, Nad 83, Zone 18)		Deposit Type	Geological comments	Best assays
		Easting	Northing			
					chalcopyrite and pyrite hosting by felsic pyroclastic rocks	
SW Zone	DDH	415 954	5 474 938	VHMS	Stratiform massive to disseminated sphalerite and pyrite hosting by felsic pyroclastic rocks	1.24 % Cu over 0.6 m
81-LS-F-1	DDH	412 679	5 480 153	VHMS	Stratiform fine pyrite and pyrrhotite lenses between lava flows	9.6 g/t Ag over 1.52 m; 33.0 g/t Ag over 1.39 m

Comparison of Bachelor (with the Hewfran option), Coniagas and Lac Shortt deposits

The Hewfran property is currently under option agreement by Metanor. Lac Shortt is not an adjacent property but it is cited in Table VII-2 because this deposit may share some similarities with the “A” Zone at Bachelor Lake. The Coniagas Zn-Pb-Ag is cited as an example of a volcanogenic hosted massive sulphide setting documented in the Bachelor Lake volcanic succession.

The Desmaraisville area hosts three (3) main types of mineral deposits:

1) Lac Shortt, structurally controlled **Au** deposits occurring along, or near, major northeast trending shear zones.
2) The Bachelor Lake mine and Hewfran **Au** occurrence. Silicified shear zone with hematitic alteration. Both deposits clearly cross-cut regional and local geology.
3) The Coniagas **Zn-Ag-Cu-Pb** deposit, located west of the property, is a deformed volcanogenic massive sulphide deposit conformable with the local geology.

Table VII-2 Comparison of Bachelor Lake with Hewfran, Coniagas and Lac Shortt

Mines	**BACHELOR CLAIMS**	**HEWFRAN (option)**	**LAC SHORTT**	**CONIAGAS**
Type of deposit	Lode gold in major shear zone ENE essentially disseminated		Lode gold in major shear zone ENE essentially disseminated	VMS
Location	• 225 km northeast of Val-d'Or, 5 km southeast of Desmaraisville • Hewfran is contiguous to Bachelor Lake Gold Mine.		• 90 km west of Chapais and crossed by the paved road linking the towns of Val d'Or and Chibougamau.	• West of Hewfran property, around 150 m of Bachelor Lake Gold Mine.
Capsule summary	• Deposit cross-cut regional and local geology. • The deposit consists of two main structures hosting hydrothermally emplaced pyritic gold mineralization.		Deposit cross-cut regional and local geology Obatogamau Formation	• Coniagas mine is a deformed VMS deposit conformable with local geology. • The Coniagas horizon has been traced for more than 609 m (2 000 ft) on the Hewfran property.
Metals	**Au**	**Au, Zn-Au**	**Au**	**Zn-Ag-Pb-Cu**
Host Rocks	• Felsic and mafic volcanites and granitoid intrusions	• Agar #1: rhyodacite & rhyolite • Agar #2: Massive and brecciated basalts	Massive and brecciated basalts Syenite and carbonatite intrusions	• Felsic lapilli tuffs
Nature of Ore	• Hydrothermally emplaced pyritic gold mineralization • Associated with a granitoid intrusion	• Hydrothermally emplaced pyritic gold mineralization	• Structural control in association with a syenitic and carbonatic intrusion	• The massive sulphides, a product of sub-surface replacement, have features common to both Mattabi- and Noranda-type deposits
Structure	• Structurally controlled deposit occurring along or near, major northeast trending ductile-brittle shear zones (N070°) - "Main Vein" and "B-Vein" : trend 110°, dips : 55° SW - "A-Vein" : trend 60-70°, dips : 45-70° SE	- Main zone: dilatant fracture zone oriented at 105°/80° SW. West extension of the Bachelor « Main » Zone - A West Zone : structurally controlled by a major regional ductile shear zone - B-Zone : trend N080°, dips 85° S	• Structurally controlled deposit occurring along or near, major northeast trending ductile-brittle shear zones. • Shear zone = 070°	• The massive sulphides form lenses which are apparently strongly folded in the vertical plane.
Alteration	Quartz + hematite + carbonate ± Feldspar ± albite	Quartz + hematite + carbonate ± Feldspar ± albite	Carbonate-Albite-Hematite-Sericite	Quartz + sericite ± epidote ± chlorite
Metamorphic grade	Green-schist	Green-schist	Green-schist	Green-schist
Age constraints	Archean	Archean	Archean	Archean
References	• Rougerie, Y., 1989 • ET 92-04	Rougerie, Y., 1989 MB89-66	• www.mrn.gouv.qc.ca/mines/quebec-mines/2004-06/urban-barry.jsp • Rougerie, Y., 1989, ET 92-04	•Doucet et al. 2004. • MB 95-14

Appendix IX : Mineral Resources Complements

APPENDIX IX – Mineral Resources complements

Historical resources estimation

All historical resource estimates for Bachelor Lake and Hewfran hereunder were calculated prior to National Instrument 43-101. Consequently, they are neither in compliance with this current standard nor with the CIM Committee on Ore Reserves. The resource estimate results from Harron (1990), Géospex (1993) and Géospex (1995) are only mentioned in this report as historical figures and should not be mentioned out of context. SNC-Lavalin (1999) first mentioned to change the term reserves used previously to resources in order to accommodate the new regulation of the NI 43-101 and Met-Chem (2001) audit the resource estimation. However, no NI 43-101 technical report was submitted at the time. The first technical report in compliance with the NI 43-101 regulations was submitted in October 2004 by Innovexplo.

Table IX-1 illustrates the results and parameters of several historical resource estimates before and after the NI 43-101 regulation.

Hewfran historic estimates are not mentioned in Table IX-1 because no details were found on the parameters used for these estimates. However, historical results were cited in Rougerie (1989). These results were of 594 000 short tons at 0.170 oz/t Au for the West Zone (100 900 ounces of gold) and of 120 000 short tons at 0.210 oz/t Au for the East zone. The East zone resource has been recently re-evaluated to 68 000 short tons at 0.259 oz/t Au by Buro (2005).

Table IX-1 Summary of methodology and parameters of previous resources estimations on the Bachelor property

<table>
<tr><th>Year</th><th>Author</th><th>Methodology, parameters and comments</th><th colspan="10">Summary of Historical[1] and Recent Resources Estimates</th></tr>
<tr><td rowspan="6">1990</td><td rowspan="6">Acadia Mineral Venture Ltd.
(Bachelor Property)</td><td rowspan="6">▪ 850’ horizontally X 250’ vertically tested area with underground drilling;
▪ Classic polygonal method using cross-sections plotted on longitudinal.</td><td colspan="10">Historical Resources[1]
Not in compliance with NI 43-101 standards</td></tr>
<tr><td rowspan="2">Zone</td><td colspan="2">Proven</td><td colspan="2">Probable</td><td>Possible</td><td colspan="2">Inferred</td><td colspan="2">Total</td></tr>
<tr><td>Und.</td><td>Dil. (25%)</td><td>Und.</td><td>Dil. (25%)</td><td>Dil. (25%)</td><td>Und.</td><td>Dil. (25%)</td><td colspan="2">(all categories)</td></tr>
<tr><td>Main</td><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td>344 487
0.283</td></tr>
<tr><td>B</td><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td>253 512
0.326</td></tr>
<tr><td>TOTAL</td><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td></td><td>597 999
0.301</td></tr>
<tr><td rowspan="9">1990</td><td rowspan="9">G.A. Harron
for:
Bachelor Lake Gold Mines Inc.
(Bachelor Property)</td><td rowspan="9">▪ 4 reserves categories:
- Proven;
- Probable;
- Possible reserves;
- Inferred.
▪ Parameters:
- Cut-off grade: 0.18 oz/t Au;
- High grade assay cutting values: 0.65 oz/t Au for “Main” Zone and 1 oz/t Au for “B” Zone;
- Minimum width: 5’;
- Fixed density: 12 ft^3/t;
- 25% @ zero grade dilution included.
- 28 247 t @ 0.172 oz/t of broken ore included (ore undoubtedly mined out by Ross-Finlay in 1992);
▪ Recommendations:
- Fill-in drilling along plunging ore shoots (around section 1+50 E);
- Very good potential at depth;
- Detailed exploration of the entire property;
- Acquisition of the Hewfran property.</td><td colspan="10">Historical Resources[1]
Not in compliance with NI 43-101 standards</td></tr>
<tr><td rowspan="2">Zone</td><td colspan="2">Proven</td><td colspan="2">Probable</td><td>Possible</td><td colspan="2">Inferred</td><td colspan="2">Total</td></tr>
<tr><td>Und.</td><td>Dil. (25%)</td><td>Und.</td><td>Dil. (25%)</td><td>Dil. (25%)</td><td>Und.</td><td>Dil. (25%)</td><td>Und.</td><td>Dil. (25%)</td></tr>
<tr><td>Main (0-12)</td><td></td><td>118 838
0.212</td><td></td><td>32 498
0.195</td><td>17 470
0.175</td><td></td><td>104 016
0.208</td><td></td><td>272 822
0.206</td></tr>
<tr><td>Main (13-15)</td><td></td><td>-</td><td></td><td>-</td><td>-</td><td></td><td>247 000
0.285</td><td></td><td>247 000
0.285</td></tr>
<tr><td>A</td><td></td><td>14 653
0.216</td><td></td><td>19 244
0.241</td><td>16 063
0.214</td><td></td><td>219 470
0.215</td><td></td><td>269 430
0.217</td></tr>
<tr><td>B</td><td></td><td>8 896
0.126</td><td></td><td>3 297
0.104</td><td>1 415
0.114</td><td></td><td>26 640
0.159</td><td></td><td>40 248
0.145</td></tr>
<tr><td>TOTAL</td><td></td><td>142 387
0.207</td><td></td><td>55 039
0.206</td><td>34 948
0.190</td><td></td><td>597 126
0.240</td><td></td><td>829 500
0.230</td></tr>
<tr><td colspan="10"></td></tr>
</table>

1: This historical resource has not been validated and is not compliant with NI 43-101 regulations and recommendations of the CIM Committee on Reserves and Resources. Appropriate actions were not fulfilled by Qualified people in order to ascertain the classification of resources and no action should be taken on the strength of historical estimate.

<table>
<tr><th>Year</th><th>Author</th><th>Methodology, parameters and comments</th><th colspan="10">Summary of Historical[1] and Recent Resources Estimates</th></tr>
<tr><td rowspan="8">1993</td><td rowspan="8">**Géospex**
for:
Ross-Finlay
(Bachelor Property)</td><td rowspan="8">▪ 4 reserves categories:
- Proven (validate samples plans defined on a minimum of 2 drifts);
- Non-validate stope (same as proven reserves but sample details inaccessible and based on previous works);
- Probable (50' radius around DDH);
- Possible (between 50' and 100' radius around DDH).
▪ Classic polygonal method using cross-sections plotted on longitudinal;
▪ Parameters (following previous criteria from Bachelor Lake Mine):
- Cut-off grade: 0.10 oz/t Au;
- High grade assay cutting values: 1.0 oz/t Au for "Main" Zone and 0.65 oz/t Au for "B" Zone;
- Minimum width: 5';
- Fixed density: 12 ft³/t.
▪ Recommendations:
- Drifting and 4 drilling station set-ups;
- U/G drilling program; and
- Sampling verification (methodology, distribution…) and pillar study.</td><td colspan="10">Historical Resources[1]
Not in compliance with NI 43-101 standards</td></tr>
<tr><td rowspan="2">Zone</td><td colspan="2">Proven</td><td colspan="2">Probable</td><td rowspan="2">Possible</td><td colspan="2">Possible + Inferred</td><td colspan="2">Total</td></tr>
<tr><td>Und.</td><td>Dil. (25%)</td><td>Und.</td><td>Dil. (25%)</td><td>Und.</td><td>Dil. (25%)</td><td>Und.</td><td>Dil. (25%)</td></tr>
<tr><td>**Main**</td><td>186 549
0.258</td><td>*233 187
0.207*</td><td>123 024
0.306</td><td>*153 780
0.245*</td><td></td><td>204 414
0.296</td><td>*255 518
0.236*</td><td>513 987
0.284</td><td>*642 485
0.227*</td></tr>
<tr><td>**A**</td><td>11 722
0.270</td><td>*14 653
0.216*</td><td>15 395
0.301</td><td>*19 244
0.241*</td><td></td><td>188 427
0.269</td><td>*235 533
0.215*</td><td>215 544
0.271</td><td>*269 430
0.217*</td></tr>
<tr><td>**B**</td><td>-</td><td>-</td><td>81 768
0.342</td><td>*102 211
0.274*</td><td></td><td>82 963
0.359</td><td>*103 704
0.287*</td><td>164 731
0.350</td><td>*205 915
0.281*</td></tr>
<tr><td>**TOTAL**</td><td>**198 271
0.259**</td><td>***247 840
0.208***</td><td>**220 187
0.319**</td><td>***275 235
0.255***</td><td></td><td>**475 804
0.296**</td><td>***594 755
0.237***</td><td>**894 262
0.295**</td><td>***1 117 830
0.236***</td></tr>
<tr><td colspan="10"></td></tr>
<tr><td rowspan="7">1995</td><td rowspan="7">**Géospex**
for:
Ressources Espalau Inc.
(Bachelor Property)</td><td rowspan="7">▪ Modification on the estimation to:
- Exclude "Main" Zone volume neighbouring the Aur resources property
- Include new DDH.
▪ Same parameters as the 1993 study;
▪ Recommendations:
- Exploration drilling program;
- Complete geophysical survey (Mag or TBF) on the whole property; and
- Surface geological mapping with lithogeochimical assaying.</td><td colspan="10">Historical Resources[1]
Not in compliance with NI 43-101 standards</td></tr>
<tr><td rowspan="2">Zone</td><td colspan="2">Proven</td><td colspan="2">Probable</td><td rowspan="2">Possible</td><td colspan="2">Possible + Inferred</td><td colspan="2">Total</td></tr>
<tr><td>Und.</td><td>Dil. (25%)</td><td>Und.</td><td>Dil. (25%)</td><td>Und.</td><td>Dil. (25%)</td><td>Und.</td><td>Dil. (25%)</td></tr>
<tr><td>**Main**</td><td>192 732
0.256</td><td>*240 915
0.205*</td><td>128 818
0.302</td><td>*161 023
0.242*</td><td></td><td>193 995
0.284</td><td>*242 494
0.227*</td><td>515 545
0.278</td><td>*644 431
0.222*</td></tr>
<tr><td>**A**</td><td>11 722
0.270</td><td>*14 653
0.216*</td><td>15 395
0.301</td><td>*19 244
0.241*</td><td></td><td>188 427
0.269</td><td>*235 533
0.215*</td><td>215 544
0.271</td><td>*269 430
0.217*</td></tr>
<tr><td>**B**</td><td></td><td></td><td>83 066
0.339</td><td>*103 833
0.271*</td><td></td><td>75 779
0.354</td><td>*94 724
0.283*</td><td>158 845
0.346</td><td>*198 556
0.227*</td></tr>
<tr><td>**TOTAL**</td><td>**204 454
0.257**</td><td>***255 568
0.206***</td><td>**227 279
0.315**</td><td>***284 100
0.252***</td><td></td><td>**458 201
0.289**</td><td>***572 751
0.231***</td><td>**889 934
0.288**</td><td>***1 112 419
0.230***</td></tr>
</table>

1: This historical resource has not been validated and is not compliant with NI 43-101 regulations and recommendations of the CIM Committee on Reserves and Resources. Appropriate actions were not fulfilled by Qualified people in order to ascertain the classification of resources and no action should be taken on the strength of historical estimate.

<table>
<tr><th>Year</th><th>Author</th><th>Methodology, parameters and comments</th><th>Summary of Historical[1] and Recent Resources Estimates</th></tr>
<tr>
<td>1999</td>
<td>**SNC-Lavalin**
for:
Sabre Capital Partners Inc.

(Bachelor Property)</td>
<td>SNC-Lavalin **changed “reserves” term to “resources”** and a 5% reduction factor on the projected indicated resources by drilling:
- Proven reserves change to measured resources;
- Probable reserves change to indicated resources;
- Possible reserves change to inferred resources.
▪ Audit of parameters used in the 1995 study: **no fundamental mistake found**;
▪ Recommendations:
- Exploration and definition program transferring inferred resources to indicated category;
- Sill drifts for continuity determination;
- Re-estimated mining dilution (estimate @ 40%);
- Feasibility study;
- Deep DDH.</td>
<td>
<table>
<tr><th colspan="10">Historical Resources[1]
Not in compliance with NI 43-101 standards</th></tr>
<tr><th>Zone</th><th colspan="2">Measured</th><th colspan="2">Indicated</th><th></th><th colspan="2">Inferred</th><th colspan="2">Total
(Measured + Indicated)</th></tr>
<tr><th></th><th>Und.</th><th>Dil. (25%)</th><th>Und.</th><th>Dil. (25%)</th><th></th><th>Und.</th><th>Dil. (25%)</th><th>Und.</th><th>Dil. (25%)</th></tr>
<tr><td>**Main**</td><td>192 732
0.256</td><td></td><td>122 377
0.301</td><td></td><td></td><td>184 295
0.284</td><td></td><td>315 109
0.273</td><td></td></tr>
<tr><td>**A**</td><td>11 722
0.270</td><td></td><td>15 395
0.301</td><td></td><td></td><td>-</td><td></td><td>27 117
0.288</td><td></td></tr>
<tr><td>**B**</td><td>-</td><td></td><td>78 913
0.339</td><td></td><td></td><td>71 990
0.354</td><td></td><td>78 913
0.339</td><td></td></tr>
<tr><td>**TOTAL**</td><td>**204 454
0.257**</td><td></td><td>**216 685
0.315**</td><td></td><td></td><td>**256 285
0.304**</td><td></td><td>**421 139
0.287**</td><td></td></tr>
</table>
</td>
</tr>
<tr>
<td>2001</td>
<td>**Met-Chem**
for:
MSV Ressources & GéoNova

(Bachelor Property)</td>
<td>▪ Audit of parameters used in 1995 study: **no fundamental mistake found**;
▪ Requisition on 40% of dilution factor.</td>
<td>
<table>
<tr><th colspan="10">Historical Resources[1]
Not in compliance with NI 43-101 standards</th></tr>
<tr><th>Zone</th><th colspan="2">Measured</th><th colspan="2">Indicated</th><th></th><th colspan="2">Inferred</th><th colspan="2">Total
(Measured + Indicated)</th></tr>
<tr><th></th><th>Und.</th><th>Dil. (25%)</th><th>Und.</th><th>Dil. (25%)</th><th></th><th>Und.</th><th>Dil. (25%)</th><th>Und.</th><th>Dil. (25%)</th></tr>
<tr><td>**Main**</td><td>192 732
0.256</td><td></td><td>122 377
0.301</td><td></td><td></td><td>184 295
0.284</td><td></td><td>315 109
0.273</td><td></td></tr>
<tr><td>**A**</td><td>11 722
0.270</td><td></td><td>15 395
0.301</td><td></td><td></td><td>-</td><td></td><td>27 117
0.288</td><td></td></tr>
<tr><td>**B**</td><td>-</td><td></td><td>78 913
0.339</td><td></td><td></td><td>71 990
0.354</td><td></td><td>78 913
0.339</td><td></td></tr>
<tr><td>**TOTAL**</td><td>**204 454
0.257**</td><td></td><td>**216 685
0.315**</td><td></td><td></td><td>**256 285
0.304**</td><td></td><td>**421 139
0.287**</td><td></td></tr>
</table>
</td>
</tr>
</table>

1: This historical resource has not been validated and is not compliant with NI 43-101 regulations and recommendations of the CIM Committee on Reserves and Resources. Appropriate actions were not fulfilled by Qualified people in order to ascertain the classification of resources and no action should be taken on the strength of historical estimate.

<table>
<tr><th>Year</th><th>Author</th><th>Methodology, parameters and comments</th><th>Summary of Historical[1] and Recent Resources Estimates</th></tr>
<tr>
<td>Oct. 2004</td>
<td>Innovexplo
for
Ressources Metanor Inc.

(Bachelor Property)</td>
<td>▪ Audit of parameters used in all previous studies: no fundamental mistake found;
▪ <u>Parameters:</u> Same as previously, judged reasonable;
▪ <u>Recommendations</u>:
- Option agreement (acquisition of Wolfden’s option);
- Exploration program by drilling and transferring inferred resources to indicated category;
- Scoping study;
- Property-scale and regional target generation;
- Claim acquisition.</td>
<td>
<table>
<tr><th colspan="10">Recent Resources
In compliance with NI 43-101 standards</th></tr>
<tr><th rowspan="2">Zone</th><th colspan="2">Measured</th><th colspan="2">Indicated</th><th rowspan="2"></th><th colspan="2">Inferred</th><th colspan="2">Total
(Measured + Indicated)</th></tr>
<tr><th>Und.</th><th>Dil. (25%)</th><th>Und.</th><th>Dil. (25%)</th><th>Und.</th><th>Dil. (25%)</th><th>Und.</th><th>Dil. (25%)</th></tr>
<tr><td>Main</td><td>192 732
0.256</td><td></td><td>122 377
0.301</td><td></td><td></td><td>184 295
0.284</td><td></td><td>315 109
0.273</td><td></td></tr>
<tr><td>A</td><td>11 722
0.270</td><td></td><td>15 395
0.301</td><td></td><td></td><td></td><td></td><td>27 117
0.288</td><td></td></tr>
<tr><td>B</td><td></td><td></td><td>78 913
0.339</td><td></td><td></td><td>71 990
0.354</td><td></td><td>78 913
0.339</td><td></td></tr>
<tr><td>TOTAL</td><td>204 454
0.257</td><td></td><td>216 685
0.315</td><td></td><td></td><td>256 285
0.304</td><td></td><td>421 139
0.287</td><td></td></tr>
</table>
</td>
</tr>
<tr>
<td>Dec. 2004</td>
<td>Innovexplo
for
Halo Resources Inc.

(Bachelor Property)</td>
<td>▪ Update of last report for the purchase intent;
▪ <u>Parameters:</u> Same as previously. judged reasonable;
▪ <u>Recommendations</u>:
- Purchase agreement (GéoNova’s option);
- Exploration program by drilling and transferring inferred resources to indicated category;
- Scoping study;
- Property-scale and regional target generation;
- Claim acquisition.</td>
<td>
<table>
<tr><th colspan="10">Recent Resources
In compliance with NI 43-101 standards</th></tr>
<tr><th rowspan="2">Zone</th><th colspan="2">Measured</th><th colspan="2">Indicated</th><th rowspan="2"></th><th colspan="2">Inferred</th><th colspan="2">Total
(Measured + Indicated)</th></tr>
<tr><th>Und.</th><th>Dil. (25%)</th><th>Und.</th><th>Dil. (25%)</th><th>Und.</th><th>Dil. (25%)</th><th>Und.</th><th>Dil. (25%)</th></tr>
<tr><td>Main</td><td>192 732
0.256</td><td></td><td>122 377
0.301</td><td></td><td></td><td>184 295
0.284</td><td></td><td>315 109
0.273</td><td></td></tr>
<tr><td>A</td><td>11 722
0.270</td><td></td><td>15 395
0.301</td><td></td><td></td><td></td><td></td><td>27 117
0.288</td><td></td></tr>
<tr><td>B</td><td></td><td></td><td>78 913
0.339</td><td></td><td></td><td>71 990
0.354</td><td></td><td>78 913
0.339</td><td></td></tr>
<tr><td>TOTAL</td><td>204 454
0.257</td><td></td><td>216 685
0.315</td><td></td><td></td><td>256 285
0.304</td><td></td><td>421 139
0.287</td><td></td></tr>
</table>
</td>
</tr>
</table>

1: This historical resource has not been validated and is not compliant with NI 43-101 regulations and recommendations of the CIM Committee on Reserves and Resources. Appropriate actions were not fulfilled by Qualified people in order to ascertain the classification of resources and no action should be taken on the strength of historical estimate.

Geostatistical evaluation

Univariate statistic

Univariate statistic results have been discussed in detailed in Horvath and Carrier (2005):
[… Univariate statistics and histogram distributions of the entire assay population of the database consist of over 15,180 assays ranging in grade from 0 to a maximum of 3.432 oz/t Au. The mean calculated grade is 0.036 opt Au with a median of 0.012 opt Au. The high mean value to the median is evidence of the lognormal distribution that can be observed in the plots provided in Appendix I. The high co-efficient of variation indicates that the extreme values in the tail of the log-normal population are contributing significantly to the high mean to median difference and may represent "flyers or nuggets" or another population of samples. This is also suggested by notable breaks in the linear that defines the population(s) on the cumulative frequency log probability plots.

A total of 2,168 assays occur within the wire frame solids constructed for the M, B and A veins. Additional assays will also occur within the C, Hewfran B-west and A-west vein wire frames however, construction of the wire frames was not yet complete at the time of the statistical evaluation. …]

[… The results from the sub-population evaluations are shown in Table 6.2 above and clearly demonstrate that the wire frames for the M, B, and A veins have been well constructed to isolate the assays that define the mineral zones/veins. The mean grades of the M and B vein sub-populations are both greater than 0.15 opt Au. The high mean to median values within the vein sub-populations are evidence the distributions are log normal and the plots provided in Appendix I clearly demonstrate this. Significantly, the lower correlation co-efficient for each of the M and B vein sub-populations also indicates that the few extreme values that remain in the upper tail of each log-normal sub-population contribute a less significant impact on the mean and in fact, may not be "flyers or nuggets" but the normal upper range values of these vein sub-population of assays. …]

Variography

Variography results have been discussed in Horvath and Carrier 2005 and are reproduced below:
[… A summary table of results from the initial variography of the entire raw assay database and the 2.5 equal length composites prior to vein coding are presented in Table 7.1.

Following completion of wire framing "M" , "B" and "A" veins of the Bachelor Lake area, a new table of the de-surveyed assay data up-dated with identification of sample assays that occurred within the respective wire frames was provided for geostatistical evaluation. Sub-populations of the assay data within the three principal veins provided an adequate number of points for variographic modelling. Variography on the three vein sub-population of assays was completed in an identical manner as the initial variography. A summary table of results from the vein specific assays variography are presented in Table 7.2.

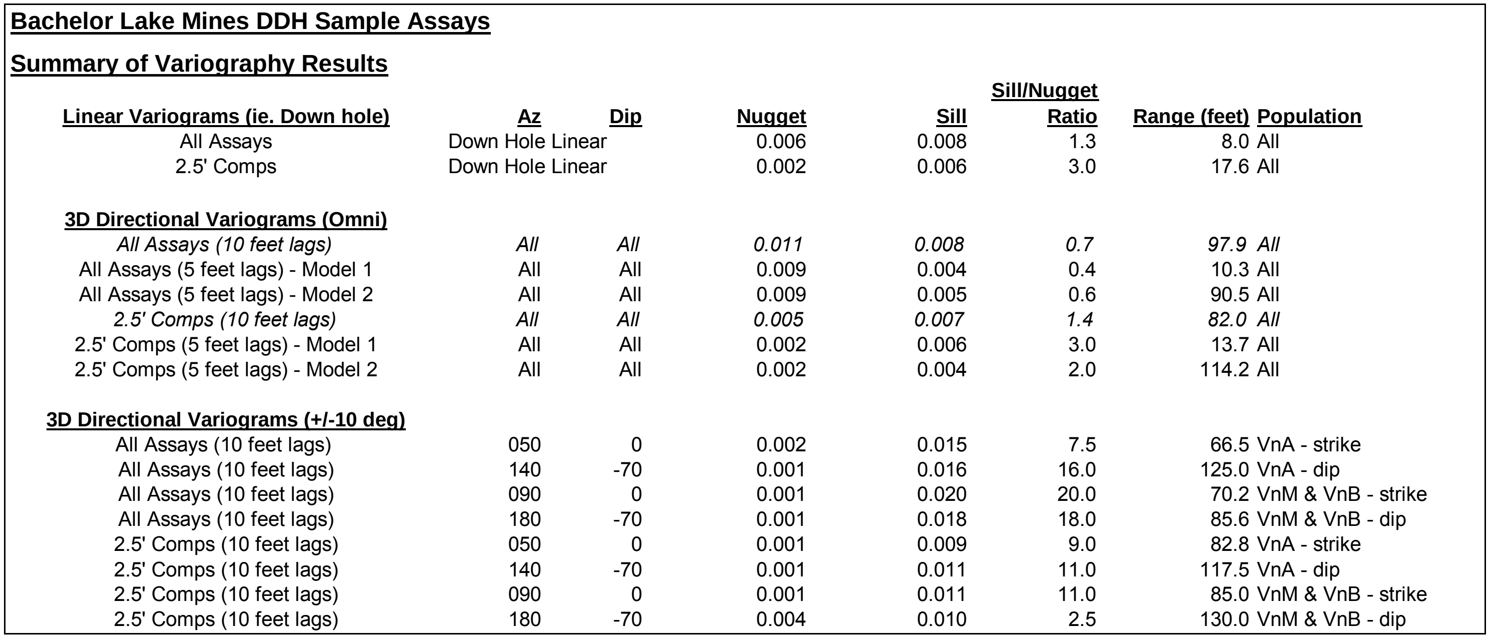
All relevant variograms generated and modelled for the various data sets are included in Appendix II. …] of Horvath and Carrier (2005) report.

[…The variography results tabulated in Tables 7.1 and 7.2 for each of the investigated vein populations provide the preferred orientations and ranges for search and interpolation ellipses during resource block modelling.

The results clearly demonstrate the different preferred orientations and ranges for the various vein sub-populations investigated and the importance of utilising the wire frame vein solids to sub-populate the assays. The wire frame vein solids should also be used to isolate sample selection and limit interpolation during resource block modelling for each of the vein wire frames.

The values from Tables 7.1 and 7.2 for the linear variograms investigated provide the nugget and sill values for each of the vein sub-populations, respectively. Similarly, the optimum orientation and ranges are obtained from the respective values for each sub-population investigated and indicated in the tables to define the most suitable search ellipses and interpolation ranges to use during resource modelling for each vein. …]

TABLE 7.1 from Horvath and Carrier 2005

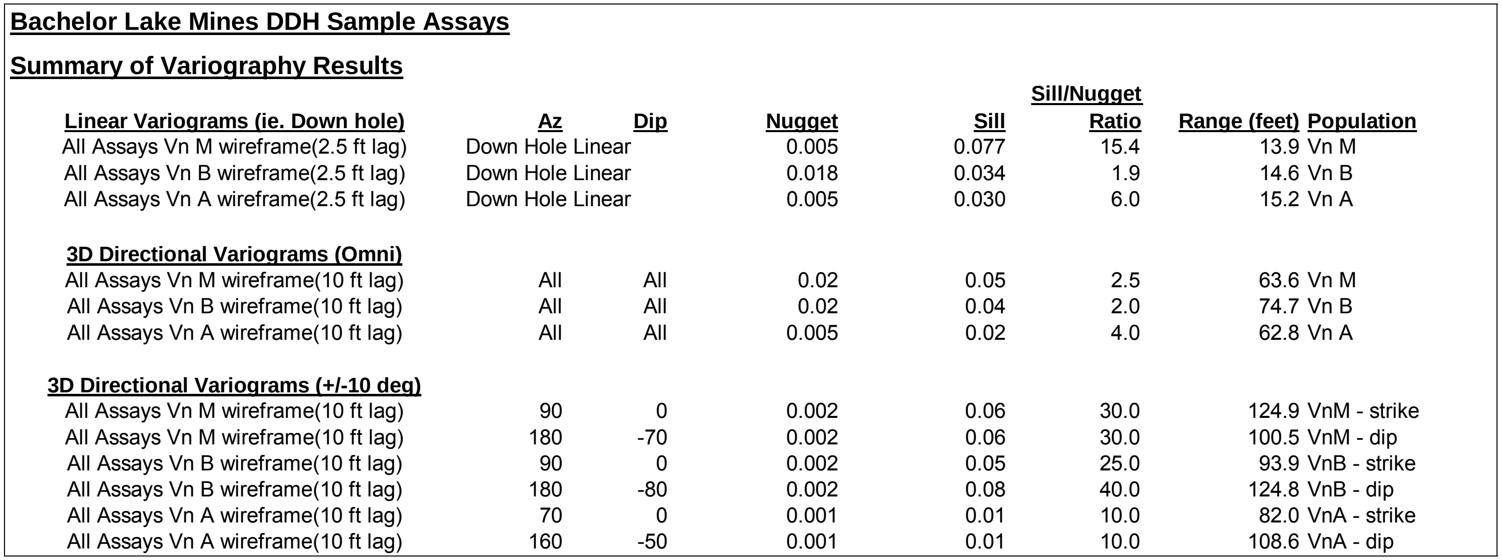


Bachelor Lake Mines DDH Sample Assays

Summary of Variography Results

Linear Variograms (ie. Down hole)	Az	Dip	Nugget	Sill	Sill/Nugget Ratio	Range (feet)	Population
All Assays	Down Hole Linear		0.006	0.008	1.3	8.0	All
2.5' Comps	Down Hole Linear		0.002	0.006	3.0	17.6	All
3D Directional Variograms (Omni)							
All Assays (10 feet lags)	*All*	*All*	*0.011*	*0.008*	*0.7*	*97.9*	*All*
All Assays (5 feet lags) - Model 1	All	All	0.009	0.004	0.4	10.3	All
All Assays (5 feet lags) - Model 2	All	All	0.009	0.005	0.6	90.5	All
2.5' Comps (10 feet lags)	*All*	*All*	*0.005*	*0.007*	*1.4*	*82.0*	*All*
2.5' Comps (5 feet lags) - Model 1	All	All	0.002	0.006	3.0	13.7	All
2.5' Comps (5 feet lags) - Model 2	All	All	0.002	0.004	2.0	114.2	All
3D Directional Variograms (+/-10 deg)							
All Assays (10 feet lags)	050	0	0.002	0.015	7.5	66.5	VnA - strike
All Assays (10 feet lags)	140	-70	0.001	0.016	16.0	125.0	VnA - dip
All Assays (10 feet lags)	090	0	0.001	0.020	20.0	70.2	VnM & VnB - strike
All Assays (10 feet lags)	180	-70	0.001	0.018	18.0	85.6	VnM & VnB - dip
2.5' Comps (10 feet lags)	050	0	0.001	0.009	9.0	82.8	VnA - strike
2.5' Comps (10 feet lags)	140	-70	0.001	0.011	11.0	117.5	VnA - dip
2.5' Comps (10 feet lags)	090	0	0.001	0.011	11.0	85.0	VnM & VnB - strike
2.5' Comps (10 feet lags)	180	-70	0.004	0.010	2.5	130.0	VnM & VnB - dip

TABLE 7.2 from Horvath and Carrier 2005



Bachelor Lake Mines DDH Sample Assays

Summary of Variography Results

Linear Variograms (ie. Down hole)	Az	Dip	Nugget	Sill	Sill/Nugget Ratio	Range (feet)	Population
All Assays Vn M wireframe(2.5 ft lag)	Down Hole Linear		0.005	0.077	15.4	13.9	Vn M
All Assays Vn B wireframe(2.5 ft lag)	Down Hole Linear		0.018	0.034	1.9	14.6	Vn B
All Assays Vn A wireframe(2.5 ft lag)	Down Hole Linear		0.005	0.030	6.0	15.2	Vn A
3D Directional Variograms (Omni)							
All Assays Vn M wireframe(10 ft lag)	All	All	0.02	0.05	2.5	63.6	Vn M
All Assays Vn B wireframe(10 ft lag)	All	All	0.02	0.04	2.0	74.7	Vn B
All Assays Vn A wireframe(10 ft lag)	All	All	0.005	0.02	4.0	62.8	Vn A
3D Directional Variograms (+/-10 deg)							
All Assays Vn M wireframe(10 ft lag)	90	0	0.002	0.06	30.0	124.9	VnM - strike
All Assays Vn M wireframe(10 ft lag)	180	-70	0.002	0.06	30.0	100.5	VnM - dip
All Assays Vn B wireframe(10 ft lag)	90	0	0.002	0.05	25.0	93.9	VnB - strike
All Assays Vn B wireframe(10 ft lag)	180	-80	0.002	0.08	40.0	124.8	VnB - dip
All Assays Vn A wireframe(10 ft lag)	70	0	0.001	0.01	10.0	82.0	VnA - strike
All Assays Vn A wireframe(10 ft lag)	160	-50	0.001	0.01	10.0	108.6	VnA - dip

[… Examination of the 2.5 feet composite variograms demonstrates the “smoother” trend of the variogram data points and generally better fit of the points to the modelled curves. As a result, the definition of the nugget, sill and range values are more precise with greater confidence in the modelled variograms. This is especially important for the data points located along the rising slope of the model variogram curve that defines the grade/distance relationship of the sample pairs and the critical values used in establishing parameters for resource modelling. As indicated earlier, the compositing produces the benefit of normalizing the data and erratic values to produce a clearer definition of the variogram (i.e. grade/distance relationship between sample points) and often indicates longer ranges as demonstrated in the results. …]

From the geostatiscal evaluation of Horvath and Carrier (2005), the following recommendations were provided for the resource modelling:

[… Table 9.1 provides the recommended procedures and parameters for block model grade and resource estimation including :
the treatment of raw assay data including compositing assay values for grade/resource estimation
block dimensions and a method of optimising block dimensions to the modelled veins in order to minimise ore loss and dilution effects
definition of anisotropic search ellipse orientations and ranges specific to vein type to be used to select samples and limit extents of grade interpolation
definition of other sample selection criteria for inverse distance grade interpolation to limit the effects of clustered data and if desired the
definition of variograms is provided in the tables of varographic results included in the report for kriging interpolation. …]

[… For the C, Hewfran B-west and A-west veins that were not wire framed at the time of the geostatistical evaluation, it would appear very reasonable to use those orientations (perhaps slightly modified to suit local variations as deemed necessary by geologist) and ranges that are indicated for their interpreted equivalents namely the M, B and A veins, respectively. Due to wider spaced drilling in the C and westerly extension veins on the Hewfran side, ranges may need to be extended beyond those indicated by the variograms in order interpolate grade in the blocks between the widely spaced holes. While these blocks may be considered to have geological continuity between holes the grade continuity is not considered established if a range beyond 100% of the indicated range is used. Hence, by definition blocks interpolated using ellipses beyond 100% of the range should be categorized as Inferred. It is recommended that 50% of the range define the limits of indicated resources and those resources generated by the nearest most reliably related sample grades at less than 50% of the indicated range could be classified as measured. …]

[… One exception can be made to the above recommendations. This relates to the tertiary direction of the search ellipse orientations or the cross-dip or width direction. InnvoExplo geologists have indicated that locally vein dips can be quite variable. A narrow search ellipse is suggested since the veins zones and results clearly demonstrate the limited extent (17 feet) in this direction. However, since the wire framing of the solids will control assays from being selected outside or interpolated outside the wire frames, a much larger width can be assigned as required to eliminate any potential problem of down-dip variation in vein dips without potential of spreading grade outside the limits of the vein boundaries. If wire frames were not available to restrict search/interpolation in this direction, increasing the range would not be a reasonable option. …]

TABLE 9.1 from the geostatiscal evaluation of Horvath and Carrier (2005)

Bachelor Lake Mines Summary Block Modelling Recommendations for DDH Assays

	Recommended Procedure
Data Preparation	
Final Au results	Average all Fire Assays (ie. AAS & Grav.) except use Met. Screen if present
Assay Compositing	Composite final Au assays on 2.5 feet equal lengths
	Exclude composites with <50% of composite interval assayed
Composite Cutting	Cut composites grades >1.5 opt Au within Vein M to 1.5 opt Au
	Cut all other composite grades >1.0 opt Au within Veins A & B to 1.0 opt Au
Block Model Parameters	
Block Size	Recommend blocks dimensions with similar anisotropy to ellipse with 2.5 feet for minimum direction
Ore Loss/Dilution Considerations	Estimate volume% of vein wire frame solids within blocks
Ore loss	For blocks with >0% and <50% vein solids (ie. waste side of contacts), estimate volume of vein wire frames and calculate ore loss
Dilution	Similarly for blocks with >50% and <100% vein solids (ie. ore side of contacts), estimate volume of vein wire frames and calculate dilution
	Optimize block dimensions to minimize vein loss and dilution
Sample Search Parameters	Anisotropic search as defined by Azimuth (principal axis of ellipse) - Dip (principal axis of ellipse) - Azimuth (2ndary axis) Method
Search Ellipse Orientations	
Blocks within Veins M and B	
ellipse principal axis (x) Az	180 deg
ellipse principal axis (x) Dip & Range	-75 deg , 125 feet
ellipse intermediate axis (y) Az & Range	090 deg , 75 feet
ellipse tertiary axis (z) Range	15 feet
Blocks within Vein A	
ellipse principal axis (x) Az	140 deg
ellipse principal axis (x) Dip & Range	-65 deg , 115 feet
ellipse intermediate axis (y) Az & Range	050 deg , 75 feet
ellipse tertiary axis (z) Range	15 feet
Ellipse SubSearch Type	Octant - (subdivides ellipse into 8 octants, recommended for declustering clustered data especially for ID interpolation)
Max. samples per Octant	12
Min. number of Octants with samples	1
Max. samples per Hole	7 = 17.5 feet downhole range/2.5 feet composites
High Grade Transition	none
Krigging	Use nugget, sill, range and variogram models as defined in tables provided

DENSITY SAMPLES ON BACHELOR LAKE GOLD MINE

		MAIN ZONE			B ZONE			A ZONE		
	SECTION	Hole ID	Sample	DENSITY	Hole ID	Sample	DENSITY	Hole ID	Sample	DENSITY
Mineralized zone	250 W	12-76	109 806	2.78	12-44	108 073	2.71	12-58	109 143	N.S.M.
	200 W	12-66	109 182	N.S.M.	12-64	109 330	2.71	12-64	109 317	N.S.M.
	159 W	12-102	110 806	2.78	12-90	110 266	2.70	12-42	108 034	2.84
	100 W	12-40	108 022	2.73	12-102	110 753	2.71	12-66	109 035	2.77
	50 W	12-68	109 426	N.S.M.	12-40	108 008	2.73	12-38	108 220	2.81
	0	12-114	122 858	2.75	12-114	122 736	2.73	12-114	122 719	2.75
	10 E	12-57	108 629	N.S.M.	12-57	108 599	N.S.M.	12-57	108 606	2.81
	10 E	12-93	122 833	2.66	12-93	122 828	2.70	12-93	122 814	N.S.M.
	150 E	12-55	108 929	2.76	12-55	108 886	N.S.M.	12-55	108 916	2.84
	200 E	12-67	108 249	2.72	12-65	108 811	N.S.M.	12-65	108 792	2.75
			MEAN :	**2.740**		**MEAN :**	**2.713**		**MEAN :**	**2.796**
Wall rock (adjacent to min. zones)	250 W	12-76	109 808	2.73	12-44	108 075	2.82	12-58	109 138	N.S.M.
	200 W	12-66	109 186	N.S.M.	12-64	109 332	N.S.M.	12-64	109 315	N.S.M.
	159 W	12-102	110 807	N.S.M.				12-42	108 032	2.75
	100 W	12-40	108 018	2.78						
	50 W	12-68	109 423	2.81						
			MEAN :	**2.773**		**MEAN :**	**2.820**		**MEAN :**	**2.750**

Zone's density average:
2.750 g/cm^3

Wall rock's density average:
2.778 g/cm^3

Nbr samples: 40 samples

Nbr of analyzis: 26 samples

TOTAL DENSITY AVERAGE: 2.755 g/cm^3
11.6355 ft^3/t

Core volume (BQ) 1 040.00 cm3/m (1)

Sample weigth per length (half core)

0.30 m	429.78 g
0.50 m	716.30 g
1.00 m	1 432.60 g
1.50 m	2 148.90 g
2.00 m	2 865.20 g

Notes:

N.S.M.: no sufficiant material

(1) Source: BERKMAN, D.A., 2001, Field Geologists' Manual, 4th ed., pp 403. The Australasian Institute of Mining and Metallurgy.

All density results are avalable in C.A.: VO05070152

Table IX-2 Density Samples on Bachelor Lake Gold Mine

Mineral Resource CIM classification

The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) guideline for resource classification includes the following definitions which are pertinent to the classification for the Bachelor resources:

*A **Mineral Resource** is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.*

*A **Measured Mineral Resource** is the estimated quantity and grade of that part of a deposit for which the size, configuration and grade have been well established by observations and sampling of outcrops, drill holes, trenches and mine workings. Those measured resources were defined from face and test hole sampling obtained from underground openings (drifts, sublevels, crosscuts, raises, sills and benches). The measured resource category extends to a maximum distance corresponding to the next opening (level, sublevel or raise). This area of influence over-imposes itself on any diamond drill hole result in the surrounding space.*

In the 2005 estimate, the Measured Resources were obtained from sampling of underground workings.

*An **Indicated Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.*

In the 2005 estimate, the Indicated Resources were obtained from block modelling and using ½ of the range (determined for each zone).

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

In the 2005 estimate, the Inferred Resources were obtained from block modelling and using 2 ½ times the range (determined for each zone).

Table IX-3 Bachelor 2005 resource summary (Imperial Units). **Summary table per category and detailed table with results per zone, category and claim block**

BACHELOR LAKE RESOURCES SUMMARY (IMPERIAL UNITS)

		BACHELOR	HEWFRAN	TOTAL
Measured	Short Tons (t)	196 100	16 200	**212 300**
	Grade (oz/t)	0.257	0.248	**0.257**
	Oz Gold	*50 487*	*4 018*	***54 504***
Indicated	Short Tons (t)	513 600	201 800	**715 400**
	Grade (oz/t)	0.223	0.208	**0.219**
	Oz Gold	*114 329*	*42 024*	***156 352***
Measured + Indicated	Short Tons (t)	**709 700**	**218 000**	**927 700**
	Grade (oz/t)	**0.232**	**0.211**	**0.223**
	Oz Gold	***164 815***	***46 042***	***210 857***
Inferred	Short Tons (t)	228 750	241 000	**469 750**
	Grade (oz/t)	0.197	0.184	**0.190**
	Oz Gold	*45 083*	*44 283*	***89 366***

Zone			**Measured**			**Indicated**			**Measured + Indicated**			**Inferred**		
			Short Tons (t)	**Grade (oz/t)**	**Oz Gold**	**Short Tons (t)**	**Grade (oz/t)**	**Oz Gold**	**Short Tons (t)**	**Grade (oz/t)**	**Oz Gold**	**Short Tons (t)**	**Grade (oz/t)**	**Oz Gold**
BACHELOR		**"A" Zone**	4 100	0.185	759	58 700	0.195	11 447	62 800	0.194	12 205	20 600	0.171	3 523
		"B" Zone				225 700	0.224	50 557	225 700	0.224	50 557	86 300	0.173	14 930
		"C" Zone				18 600	0.209	3 887	18 600	0.209	3 887	35 250	0.215	7 579
		"Main" Zone	192 000	0.259	49 728	210 600	0.230	48 438	402 600	0.244	98 166	86 600	0.220	19 052
		BACHELOR TOTAL	**196 100**	**0.257**	**50 487**	**513 600**	**0.223**	**114 329**	**709 700**	**0.232**	**164 815**	**228 750**	**0.197**	**45 083**
HEWFRAN	**East**	**"A" Zone**				1 900	0.181	344	1 900	0.181	344	1 400	0.382	535
		"B" Zone				2 200	0.196	431	2 200	0.196	431	600	0.151	91
		"Main" Zone	16 200	0.248	4 018	87 600	0.210	18 396	103 800	0.216	22 414	32 100	0.188	6 035
	West	**"A West" Zone**				85 200	0.220	18 744	85 200	0.220	18 744	144 000	0.194	27 936
		"B West" Zone				24 900	0.165	4 109	24 900	0.165	4 109	62 900	0.154	9 687
		HEWFRAN TOTAL	**16 200**	**0.248**	**4 018**	**201 800**	**0.208**	**42 024**	**218 000**	**0.211**	**46 042**	**241 000**	**0.184**	**44 283**
BACHELOR & HEWFRAN TOTAL			**212 300**	**0.257**	**54 504**	**715 400**	**0.219**	**156 352**	**927 700**	**0.227**	**210 857**	**469 750**	**0.190**	**89 366**

Table IX-4 Bachelor 2005 resource summary (Metric Units). **Summary table per category and detailed table with results per zone, category and claim block**

BACHELOR LAKE RESOURCES SUMMARY (METRIC UNITS)

		BACHELOR	**HEWFRAN**	**TOTAL**
Measured	Metric Tons (tm)	177 898	14 696	**192 594**
	Grade (g/t)	8.83	8.50	**8.80**
	kg of Gold	*1 570*	*125*	***1 695***
Indicated	Metric Tons (tm)	465 928	183 069	**648 997**
	Grade (g/t)	7.63	7.14	**7.49**
	kg of Gold	*3 556*	*1 307*	***4 861***
Measured + Indicated	Metric Tons (tm)	**643 826**	**197 765**	**841 591**
	Grade (g/t)	**7.96**	**7.24**	**7.79**
	kg of Gold	***5 126***	***1 432***	***6 556***
Inferred	Metric Tons (tm)	207 517	218 630	**426 148**
	Grade (g/t)	6.76	6.30	**6.52**
	kg of Gold	*1 402*	*1 377*	***2 778***

Zone			**Measured**			**Indicated**			**Measured + Indicated**			**Inferred**		
			Metric tons (tm)	**Grade (g/t)**	**kg of Gold**	**Metric tons (tm)**	**Grade (g/t)**	**kg of Gold**	**Metric tons (tm)**	**Grade (g/t)**	**kg of Gold**	**Metric tons (tm)**	**Grade (g/t)**	**kg of Gold**
BACHELOR		**"A" Zone**	3 719	6.34	24	53 251	6.69	356	56 971	6.66	380	18 688	5.86	110
		"B" Zone				204 751	7.68	1 572	204 751	7.68	1 572	78 290	5.93	464
		"C" Zone				16 874	7.17	121	16 874	7.17	121	31 978	7.37	236
		"Main" Zone	174 179	8.88	1 547	191 052	7.89	1 507	365 231	8.36	3 053	78 562	7.54	593
	BACHELOR TOTAL		**177 898**	**8.83**	**1 570**	**465 928**	**7.63**	**3 556**	**643 826**	**7.96**	**5 126**	**207 517**	**6.76**	**1 402**
HEWFRAN	**East**	**"A" Zone**				1 724	6.21	11	1 724	6.21	11	1 270	13.10	17
		"B" Zone				1 996	6.72	13	1 996	6.72	13	544	5.18	3
		"Main" Zone	14 696	8.50	125	79 469	7.20	572	94 165	7.41	697	29 120	6.45	188
	West	**"A West" Zone**				77 292	7.54	583	77 292	7.54	583	130 634	6.65	869
		"B West" Zone				22 589	5.66	128	22 589	5.66	128	57 062	5.28	301
	HEWFRAN TOTAL		**14 696**	**8.50**	**125**	**183 069**	**7.14**	**1 307**	**197 765**	**7.24**	**1 432**	**218 630**	**6.30**	**1 377**
BACHELOR & HEWFRAN TOTAL			**192 594**	**8.80**	**1 695**	**648 997**	**7.49**	**4 863**	**841 591**	**7.79**	**6 559**	**426 148**	**6.52**	**2 780**

Sensitivity study

Results from volumetric done on the block model for the Indicated and Inferred Resources are illustrated in Figures IX-1 and IX-2. These figures present the percentage of contained gold versus different cut-off grade increment from 0.05 oz/t Au to 0.25 oz/t Au. The results obtained from the Indicated resources show that a large proportion of the contained gold is coming from higher grade area. The different classes between cut-off of 0.1 oz/t Au and 0.2 oz/t Au show an equal distribution of the contained gold. For the Inferred Resource, a similar distribution of the contained gold is obtained but with lesser differences between each cut-off grade classes. Also note on these graphics that the tonnage and the grade below the 2005 cut-off grade (0.10 oz/t Au) are also illustrated (classes between 0.05 oz/t Au to 0.10 oz/t Au). This class is certainly below any economic cut-off and does not represent a significant amount of contained gold.

Detailed results for the "Main" and "B" zones show similar distribution with results from the whole resource estimate. These results are presented in Figures IX-3 and IX-4 (tonnage, Au grade and contained gold versus cut-off grade for the Indicated Resources (respectively for the "Main" and "B" zones)) and in Figures IX-5 and IX-6 (tonnage, Au grade and contained gold versus cut-off grade for the Inferred Resources (respectively for the "Main" and "B" zones)).

From these results, it can be stated that the total contained gold will be reduced by approximately -10% for each increment step of 0.025 oz/t Au of the cut-off grade.

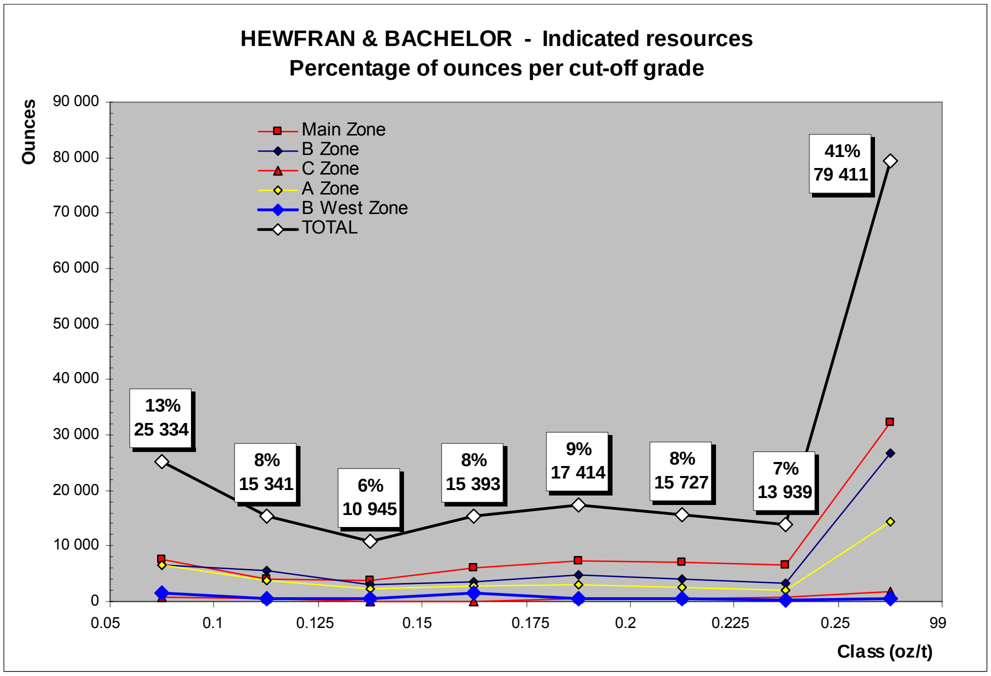


Figure IX-1: Contained gold (% and oz) versus cut-off grades for the Indicated Resources (Hewfran and Bachelor)

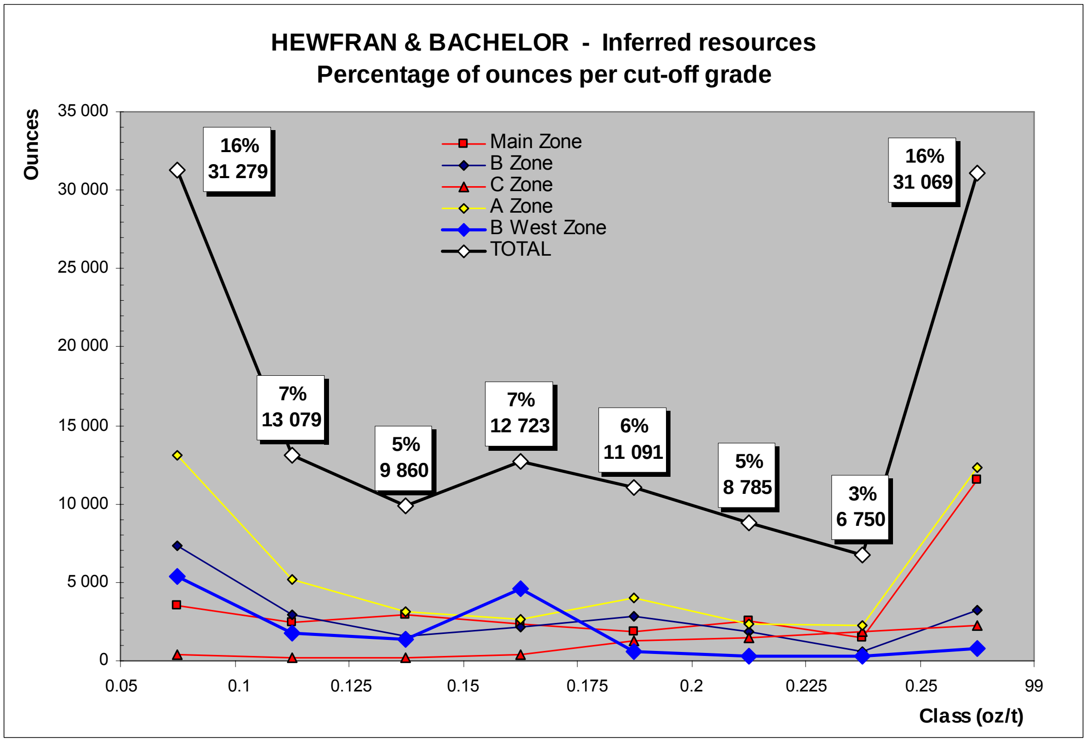


Figure IX-2: Contained gold (% and oz) versus cut-off grades for the Inferred Resources (Hewfran and Bachelor)

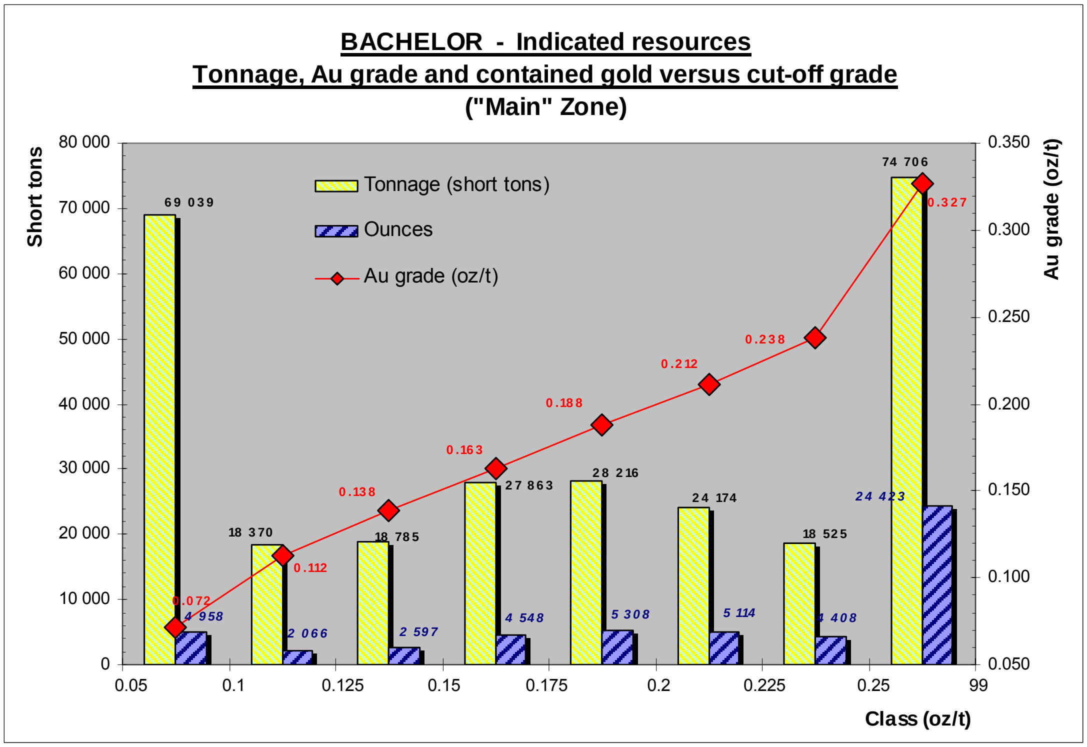


Figure IX-3: Tonnage, Au grade and contained gold versus cut-off grade for the Indicated Resources (Bachelor "Main" zone)

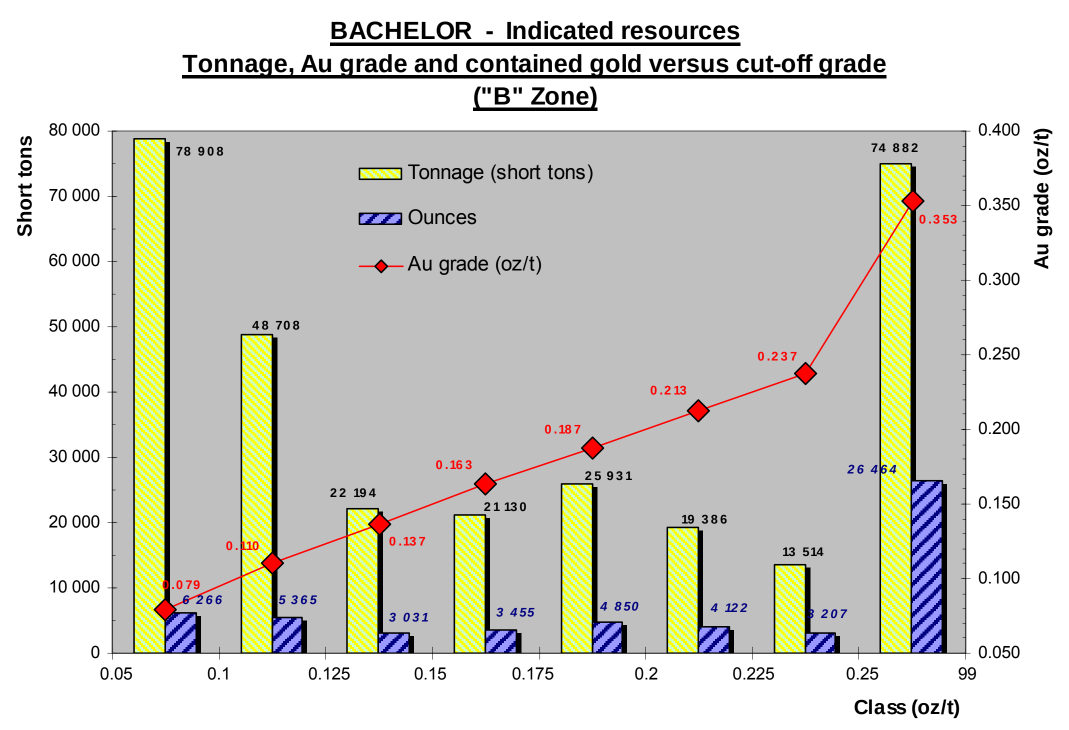


Figure IX-4: Tonnage, Au grade and contained gold versus cut-off grade for the Indicated Resources (Bachelor "B" zone)

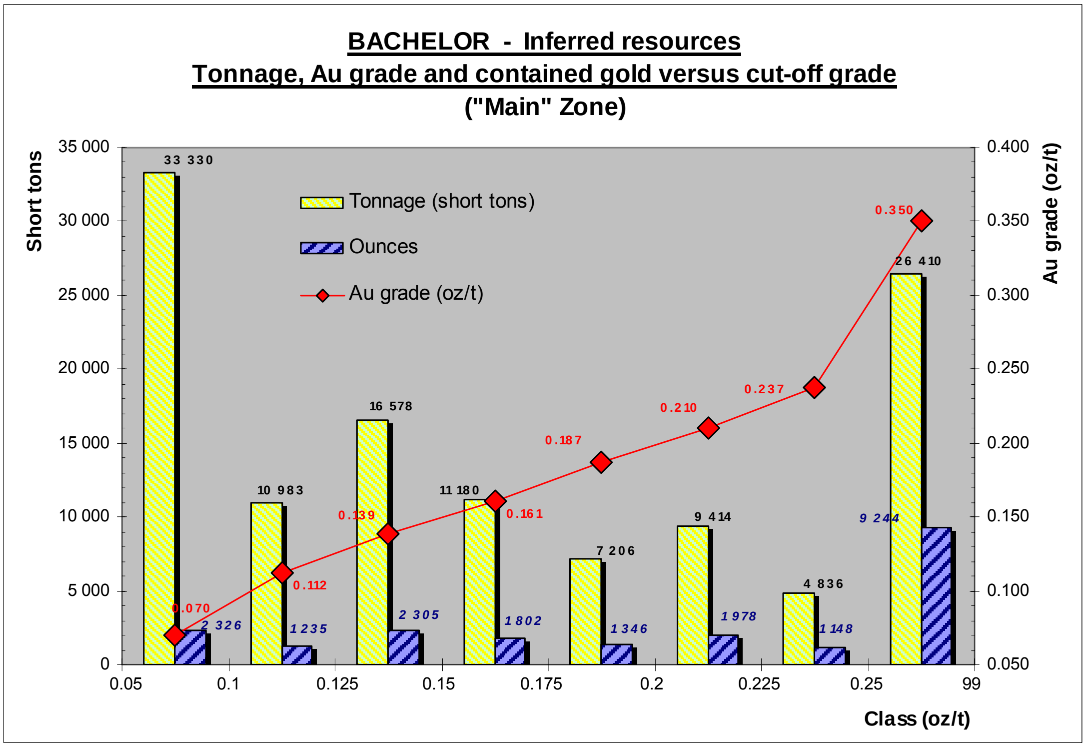


Figure IX-5: Tonnage, Au grade and contained gold versus cut-off grade for the Inferred Resources (Bachelor "Main" zone)

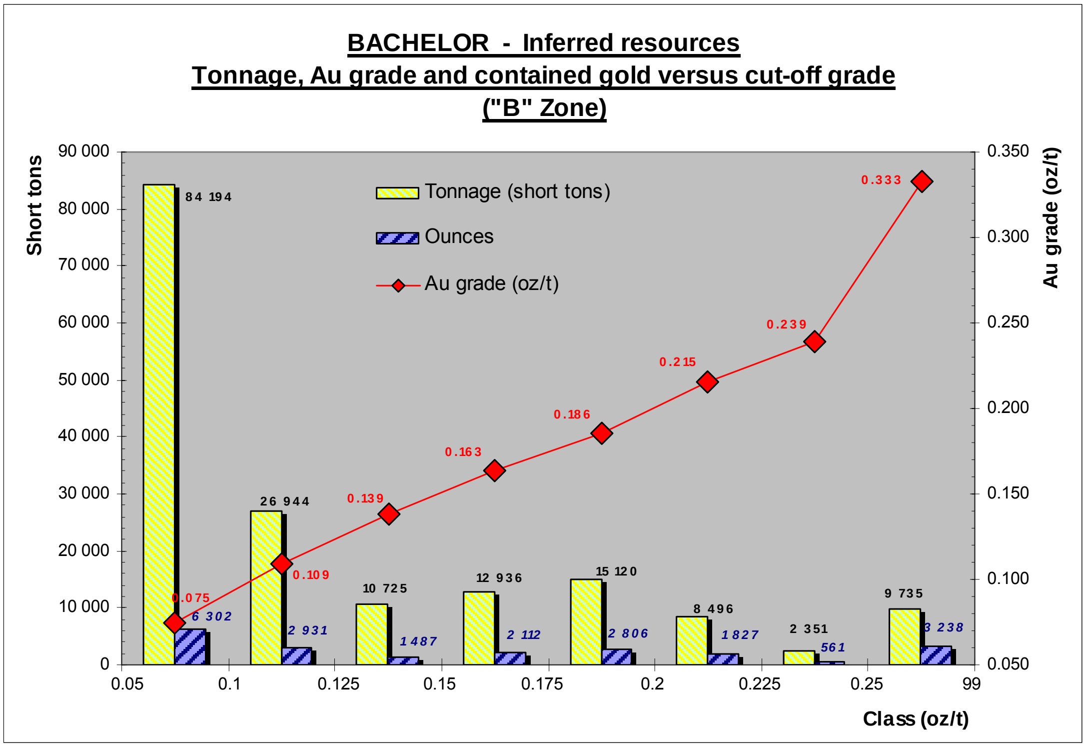
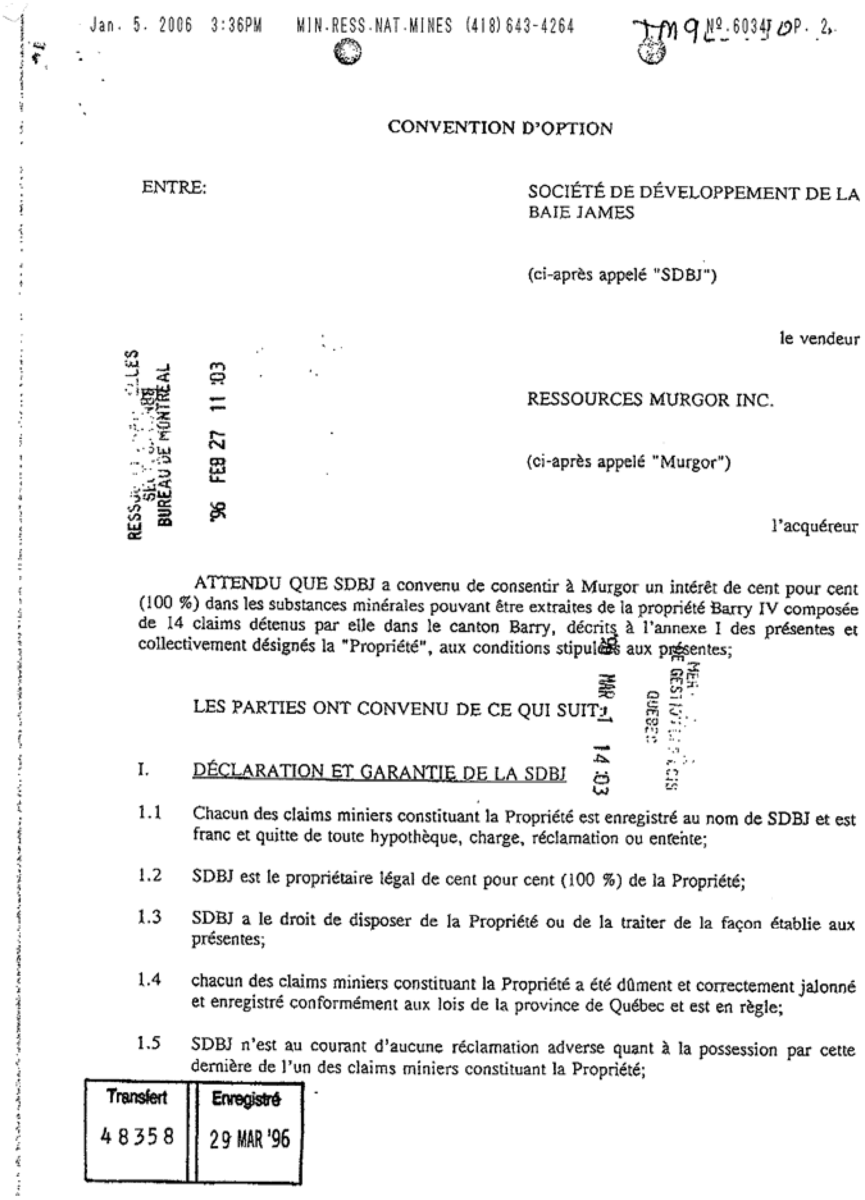


Figure IX-6: Tonnage, Au grade and contained gold versus cut-off grade for the Inferred Resources (Bachelor "B" zone)

BARRY-1 APPENDICES

Appendix 1: Agreements between the SDBJ, Murgor Resources Inc., Freewest Resources Inc. and Metanor Resources Inc.



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CONVENTION D'OPTION

ENTRE:

SOCIÉTÉ DE DÉVELOPPEMENT DE LA BAIE JAMES

(ci-après appelé "SDBJ")

le vendeur

RESSOURCES MURGOR INC.

(ci-après appelé "Murgor")

l'acquéreur

BUREAU DE MONTREAL
'96 FEB 27 11 :03

ATTENDU QUE SDBJ a convenu de consentir à Murgor un intérêt de cent pour cent (100 %) dans les substances minérales pouvant être extraites de la propriété Barry IV composée de 14 claims détenus par elle dans le canton Barry, décrits à l'annexe I des présentes et collectivement désignés la "Propriété", aux conditions stipulées aux présentes;

LES PARTIES ONT CONVENU DE CE QUI SUIT:

'96 MAR 21 14 :03
QUEBEC

I. DÉCLARATION ET GARANTIE DE LA SDBJ

1.1 Chacun des claims miniers constituant la Propriété est enregistré au nom de SDBJ et est franc et quitte de toute hypothèque, charge, réclamation ou entente;

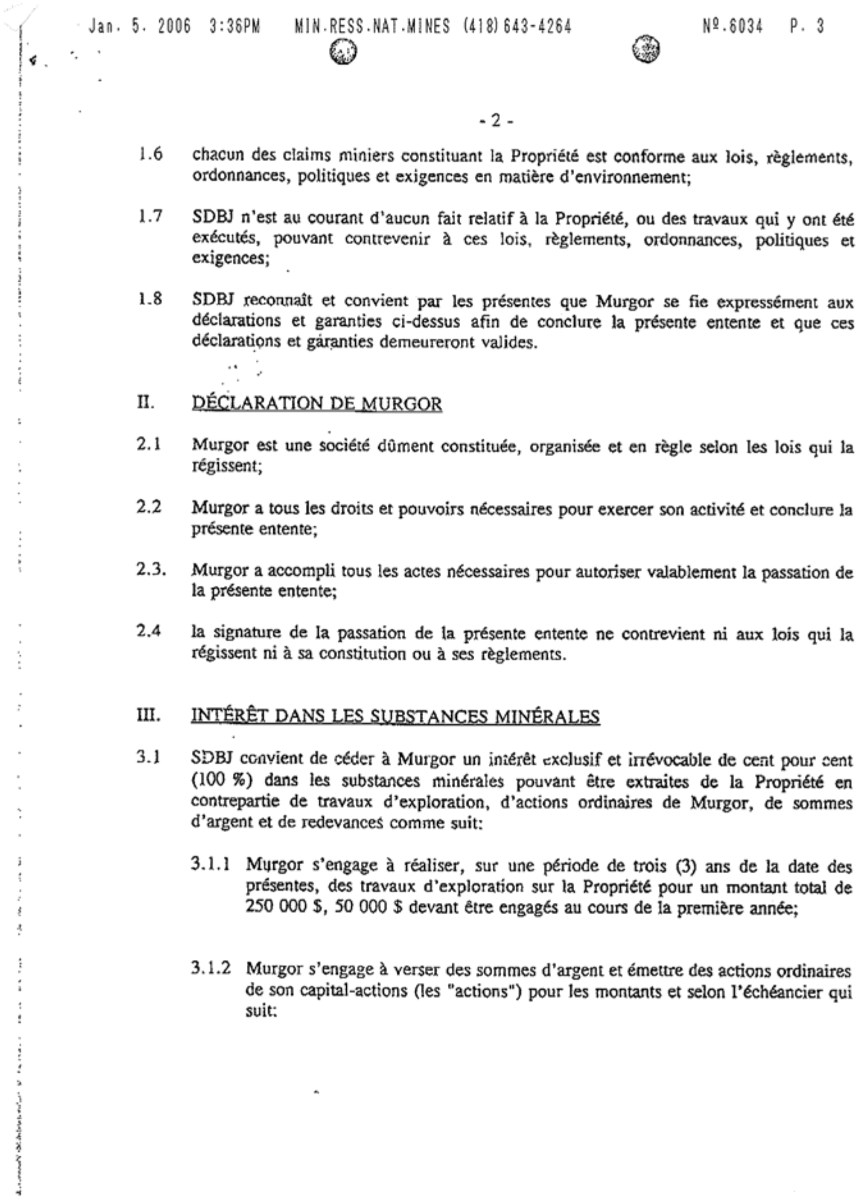
1.2 SDBJ est le propriétaire légal de cent pour cent (100 %) de la Propriété;

1.3 SDBJ a le droit de disposer de la Propriété ou de la traiter de la façon établie aux présentes;

1.4 chacun des claims miniers constituant la Propriété a été dûment et correctement jalonné et enregistré conformément aux lois de la province de Québec et est en règle;

1.5 SDBJ n'est au courant d'aucune réclamation adverse quant à la possession par cette dernière de l'un des claims miniers constituant la Propriété;

Transfert	Enregistré
48358	29 MAR '96



1.6 chacun des claims miniers constituant la Propriété est conforme aux lois, règlements, ordonnances, politiques et exigences en matière d'environnement;

1.7 SDBJ n'est au courant d'aucun fait relatif à la Propriété, ou des travaux qui y ont été exécutés, pouvant contrevenir à ces lois, règlements, ordonnances, politiques et exigences;

1.8 SDBJ reconnaît et convient par les présentes que Murgor se fie expressément aux déclarations et garanties ci-dessus afin de conclure la présente entente et que ces déclarations et garanties demeureront valides.

II. DÉCLARATION DE MURGOR

2.1 Murgor est une société dûment constituée, organisée et en règle selon les lois qui la régissent;

2.2 Murgor a tous les droits et pouvoirs nécessaires pour exercer son activité et conclure la présente entente;

2.3. Murgor a accompli tous les actes nécessaires pour autoriser valablement la passation de la présente entente;

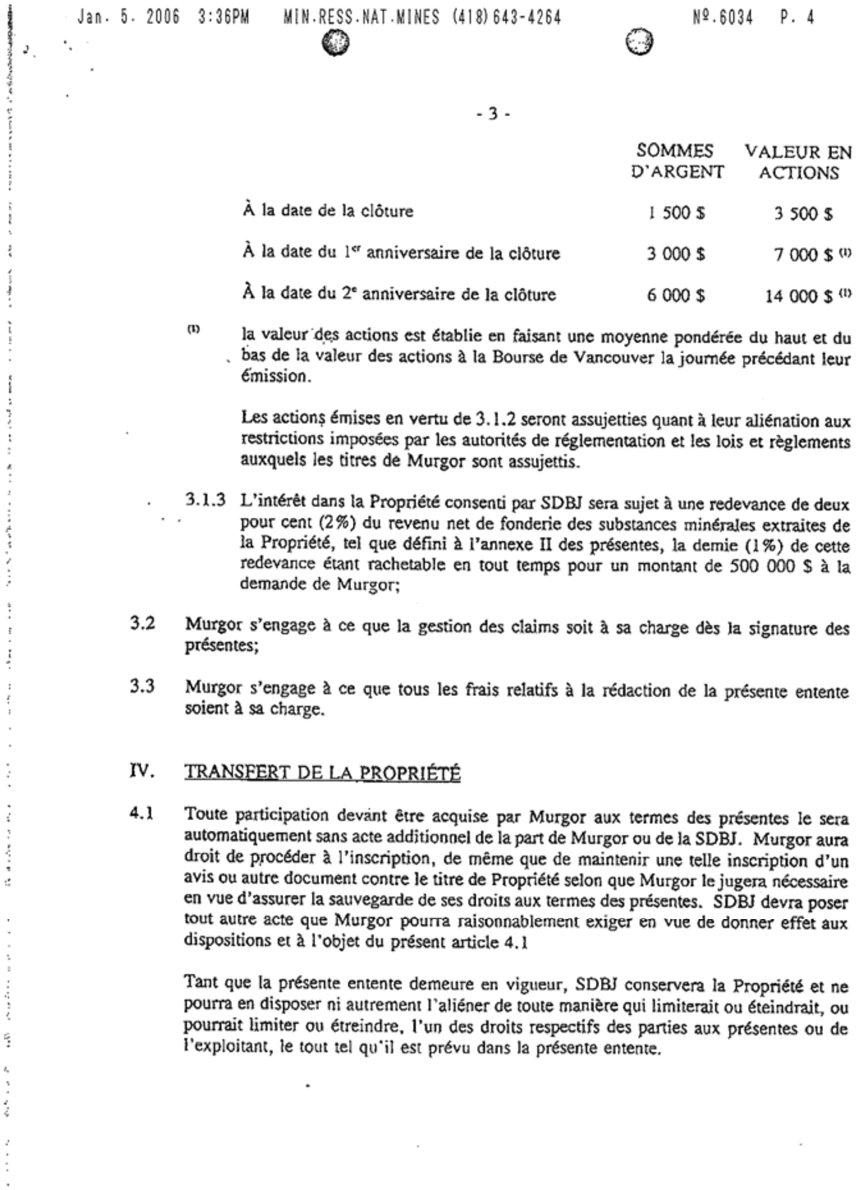
2.4 la signature de la passation de la présente entente ne contrevient ni aux lois qui la régissent ni à sa constitution ou à ses règlements.

III. INTÉRÊT DANS LES SUBSTANCES MINÉRALES

3.1 SDBJ convient de céder à Murgor un intérêt exclusif et irrévocable de cent pour cent (100 %) dans les substances minérales pouvant être extraites de la Propriété en contrepartie de travaux d'exploration, d'actions ordinaires de Murgor, de sommes d'argent et de redevances comme suit:

3.1.1 Murgor s'engage à réaliser, sur une période de trois (3) ans de la date des présentes, des travaux d'exploration sur la Propriété pour un montant total de 250 000 $, 50 000 $ devant être engagés au cours de la première année;

3.1.2 Murgor s'engage à verser des sommes d'argent et émettre des actions ordinaires de son capital-actions (les "actions") pour les montants et selon l'échéancier qui suit:



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	SOMMES D'ARGENT	VALEUR EN ACTIONS
À la date de la clôture	1 500 $	3 500 $
À la date du 1^{er} anniversaire de la clôture	3 000 $	7 000 $ (1)
À la date du 2^e anniversaire de la clôture	6 000 $	14 000 $ (1)

(1) la valeur des actions est établie en faisant une moyenne pondérée du haut et du bas de la valeur des actions à la Bourse de Vancouver la journée précédant leur émission.

Les actions émises en vertu de 3.1.2 seront assujetties quant à leur aliénation aux restrictions imposées par les autorités de réglementation et les lois et règlements auxquels les titres de Murgor sont assujettis.

3.1.3 L'intérêt dans la Propriété consenti par SDBJ sera sujet à une redevance de deux pour cent (2%) du revenu net de fonderie des substances minérales extraites de la Propriété, tel que défini à l'annexe II des présentes, la demie (1%) de cette redevance étant rachetable en tout temps pour un montant de 500 000 $ à la demande de Murgor;

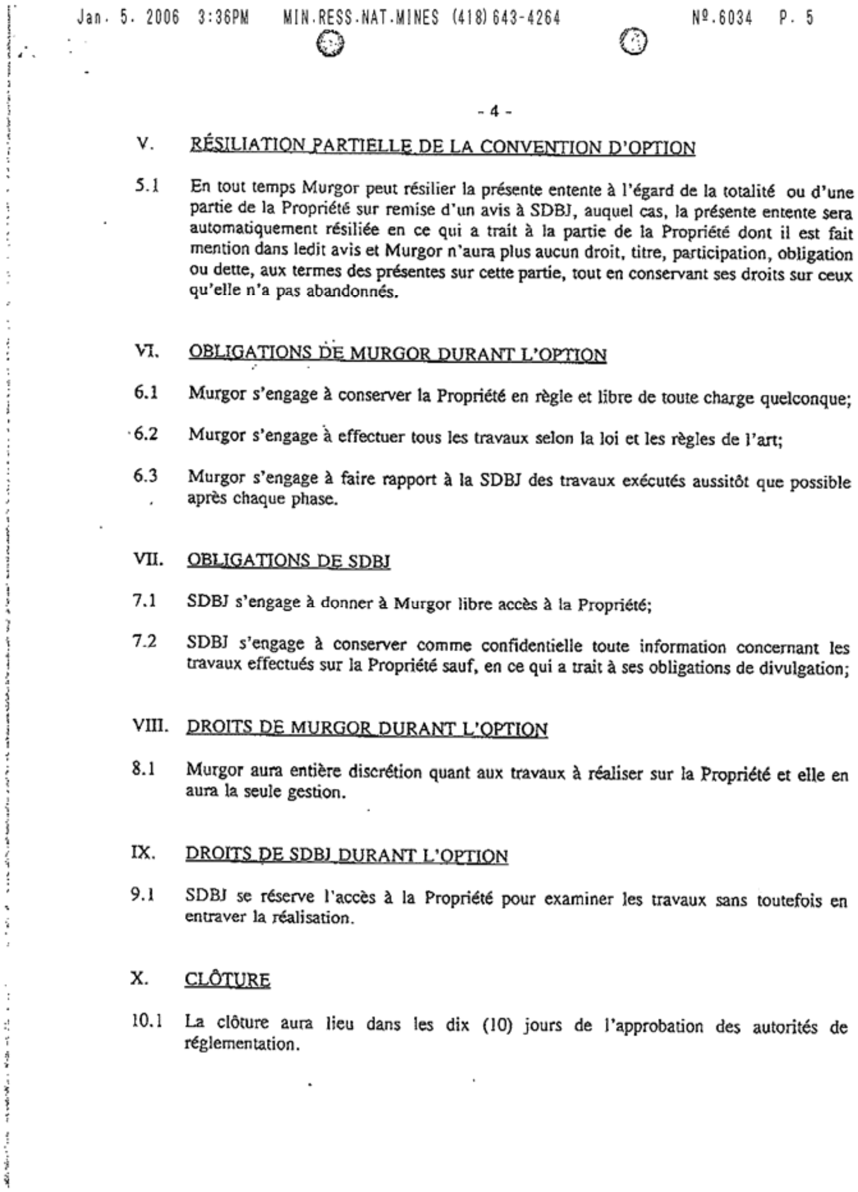
3.2 Murgor s'engage à ce que la gestion des claims soit à sa charge dès la signature des présentes;

3.3 Murgor s'engage à ce que tous les frais relatifs à la rédaction de la présente entente soient à sa charge.

IV. TRANSFERT DE LA PROPRIÉTÉ

4.1 Toute participation devant être acquise par Murgor aux termes des présentes le sera automatiquement sans acte additionnel de la part de Murgor ou de la SDBJ. Murgor aura droit de procéder à l'inscription, de même que de maintenir une telle inscription d'un avis ou autre document contre le titre de Propriété selon que Murgor le jugera nécessaire en vue d'assurer la sauvegarde de ses droits aux termes des présentes. SDBJ devra poser tout autre acte que Murgor pourra raisonnablement exiger en vue de donner effet aux dispositions et à l'objet du présent article 4.1

Tant que la présente entente demeure en vigueur, SDBJ conservera la Propriété et ne pourra en disposer ni autrement l'aliéner de toute manière qui limiterait ou éteindrait, ou pourrait limiter ou étreindre, l'un des droits respectifs des parties aux présentes ou de l'exploitant, le tout tel qu'il est prévu dans la présente entente.



V. RÉSILIATION PARTIELLE DE LA CONVENTION D'OPTION

5.1 En tout temps Murgor peut résilier la présente entente à l'égard de la totalité ou d'une partie de la Propriété sur remise d'un avis à SDBJ, auquel cas, la présente entente sera automatiquement résiliée en ce qui a trait à la partie de la Propriété dont il est fait mention dans ledit avis et Murgor n'aura plus aucun droit, titre, participation, obligation ou dette, aux termes des présentes sur cette partie, tout en conservant ses droits sur ceux qu'elle n'a pas abandonnés.

VI. OBLIGATIONS DE MURGOR DURANT L'OPTION

6.1 Murgor s'engage à conserver la Propriété en règle et libre de toute charge quelconque;

6.2 Murgor s'engage à effectuer tous les travaux selon la loi et les règles de l'art;

6.3 Murgor s'engage à faire rapport à la SDBJ des travaux exécutés aussitôt que possible après chaque phase.

VII. OBLIGATIONS DE SDBJ

7.1 SDBJ s'engage à donner à Murgor libre accès à la Propriété;

7.2 SDBJ s'engage à conserver comme confidentielle toute information concernant les travaux effectués sur la Propriété sauf, en ce qui a trait à ses obligations de divulgation;

VIII. DROITS DE MURGOR DURANT L'OPTION

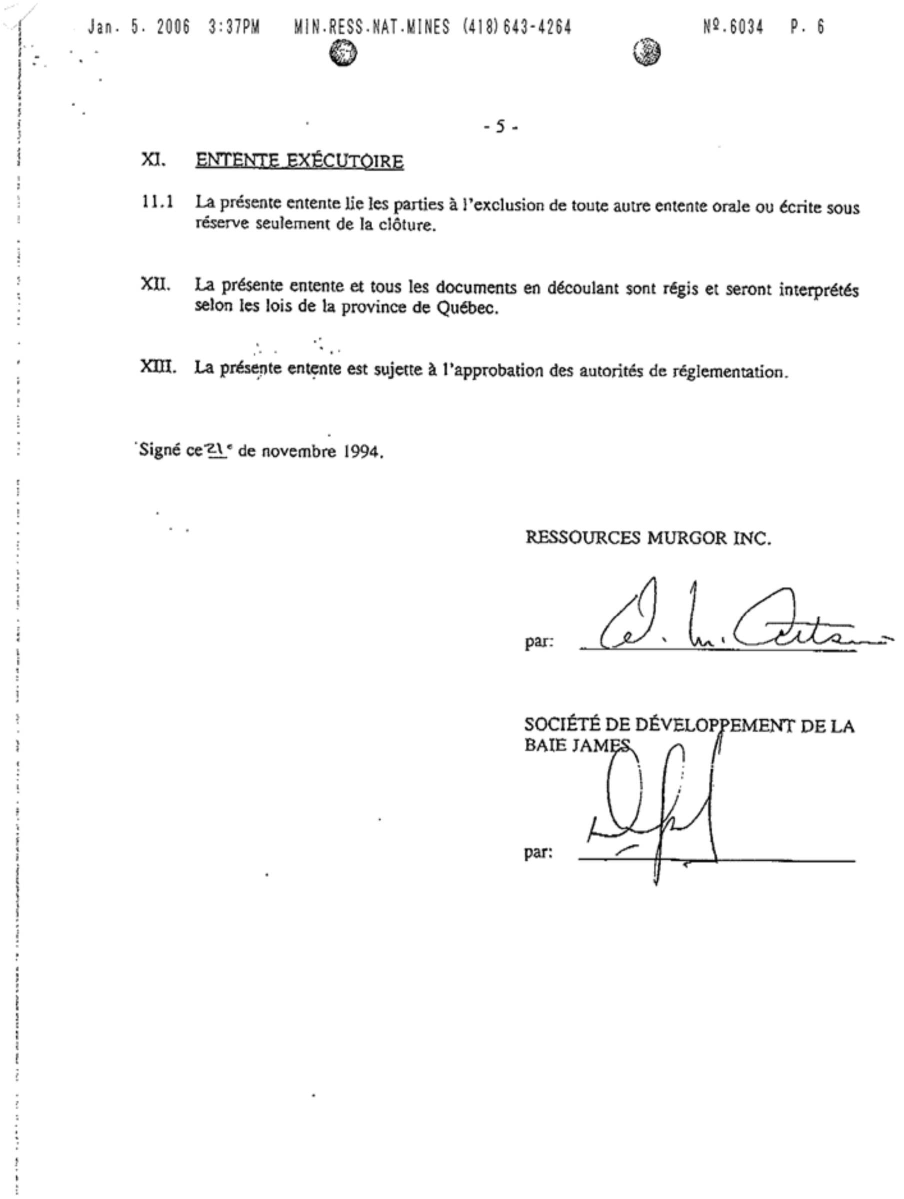
8.1 Murgor aura entière discrétion quant aux travaux à réaliser sur la Propriété et elle en aura la seule gestion.

IX. DROITS DE SDBJ DURANT L'OPTION

9.1 SDBJ se réserve l'accès à la Propriété pour examiner les travaux sans toutefois en entraver la réalisation.

X. CLÔTURE

10.1 La clôture aura lieu dans les dix (10) jours de l'approbation des autorités de réglementation.



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XI. ENTENTE EXÉCUTOIRE

11.1 La présente entente lie les parties à l'exclusion de toute autre entente orale ou écrite sous réserve seulement de la clôture.

XII. La présente entente et tous les documents en découlant sont régis et seront interprétés selon les lois de la province de Québec.

XIII. La présente entente est sujette à l'approbation des autorités de réglementation.

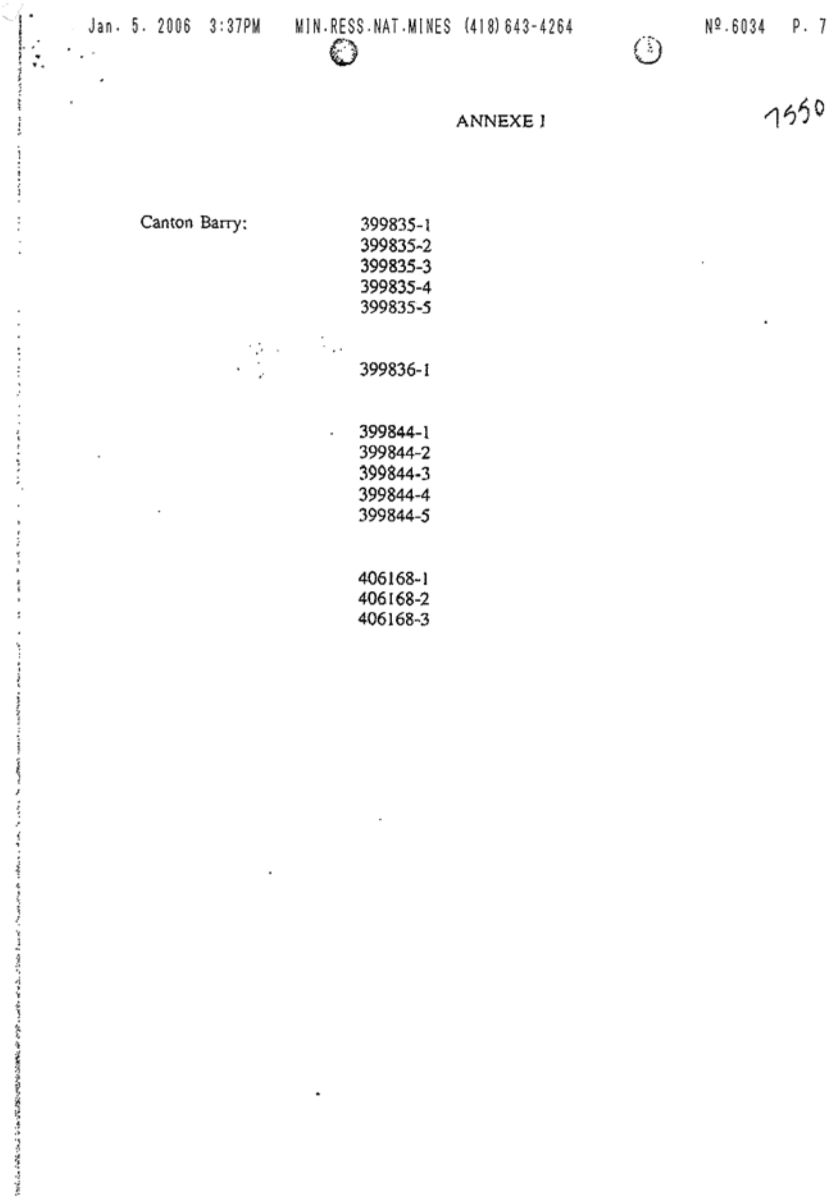
Signé ce 21e de novembre 1994.

RESSOURCES MURGOR INC.

par:

SOCIÉTÉ DE DÉVELOPPEMENT DE LA BAIE JAMES

par:



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1550

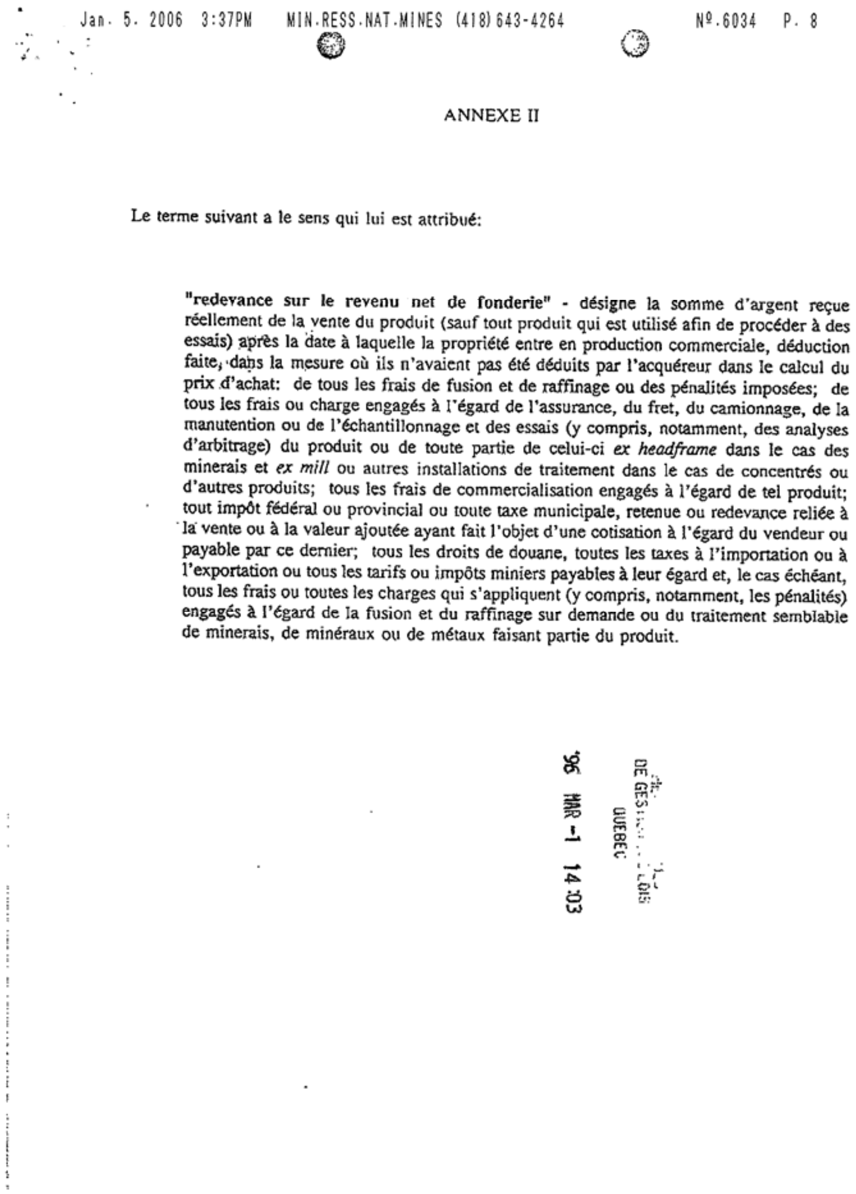
ANNEXE I

Canton Barry:
399835-1
399835-2
399835-3
399835-4
399835-5

399836-1

399844-1
399844-2
399844-3
399844-4
399844-5

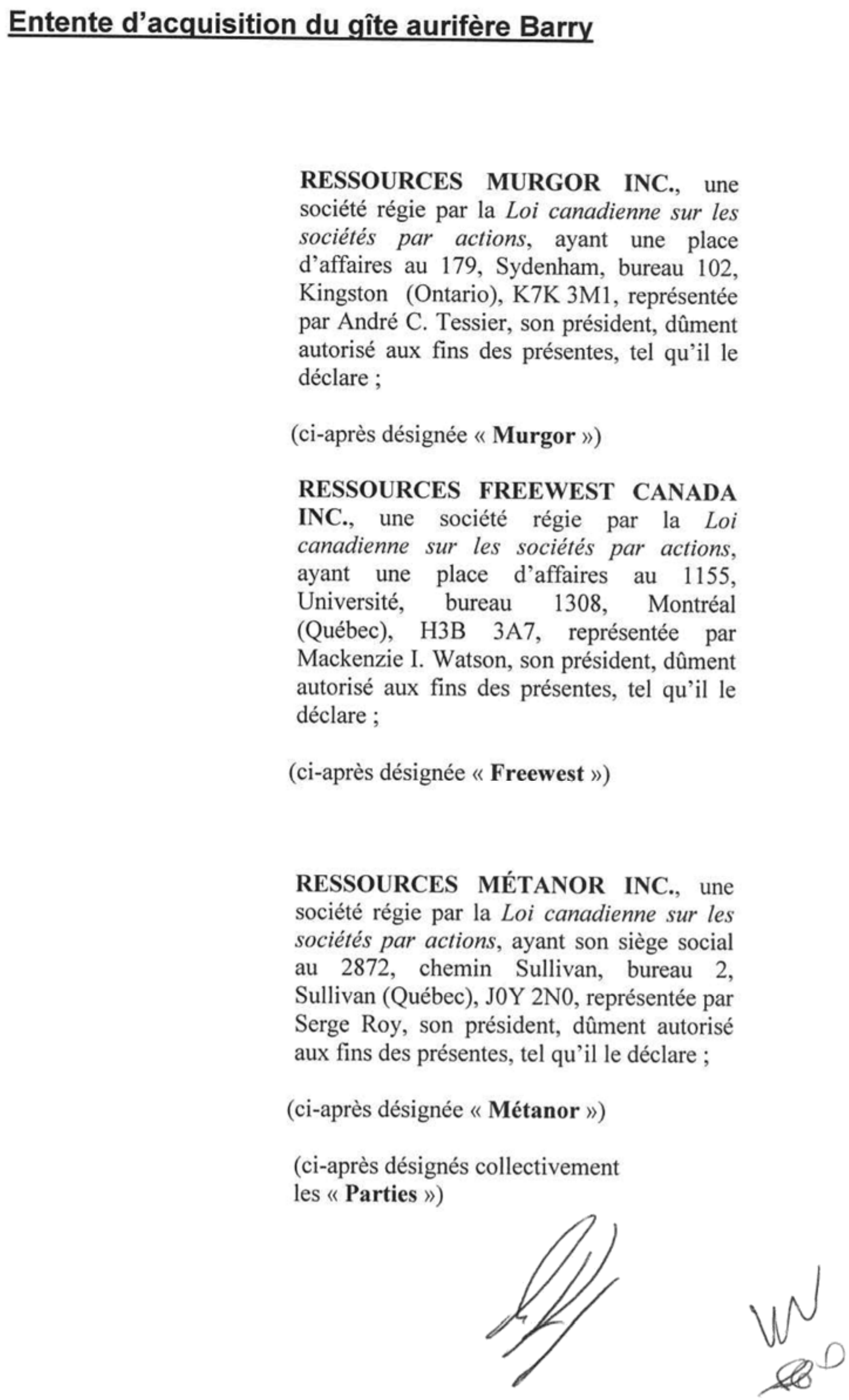
406168-1
406168-2
406168-3



ANNEXE II

Le terme suivant a le sens qui lui est attribué:

"redevance sur le revenu net de fonderie" - désigne la somme d'argent reçue réellement de la vente du produit (sauf tout produit qui est utilisé afin de procéder à des essais) après la date à laquelle la propriété entre en production commerciale, déduction faite, dans la mesure où ils n'avaient pas été déduits par l'acquéreur dans le calcul du prix d'achat: de tous les frais de fusion et de raffinage ou des pénalités imposées; de tous les frais ou charge engagés à l'égard de l'assurance, du fret, du camionnage, de la manutention ou de l'échantillonnage et des essais (y compris, notamment, des analyses d'arbitrage) du produit ou de toute partie de celui-ci *ex headframe* dans le cas des minerais et *ex mill* ou autres installations de traitement dans le cas de concentrés ou d'autres produits; tous les frais de commercialisation engagés à l'égard de tel produit; tout impôt fédéral ou provincial ou toute taxe municipale, retenue ou redevance reliée à la vente ou à la valeur ajoutée ayant fait l'objet d'une cotisation à l'égard du vendeur ou payable par ce dernier; tous les droits de douane, toutes les taxes à l'importation ou à l'exportation ou tous les tarifs ou impôts miniers payables à leur égard et, le cas échéant, tous les frais ou toutes les charges qui s'appliquent (y compris, notamment, les pénalités) engagés à l'égard de la fusion et du raffinage sur demande ou du traitement semblable de minerais, de minéraux ou de métaux faisant partie du produit.



Entente d'acquisition du gîte aurifère Barry

ENTRE :

RESSOURCES MURGOR INC., une société régie par la *Loi canadienne sur les sociétés par actions*, ayant une place d'affaires au 179, Sydenham, bureau 102, Kingston (Ontario), K7K 3M1, représentée par André C. Tessier, son président, dûment autorisé aux fins des présentes, tel qu'il le déclare ;

(ci-après désignée « **Murgor** »)

ET

RESSOURCES FREEWEST CANADA INC., une société régie par la *Loi canadienne sur les sociétés par actions*, ayant une place d'affaires au 1155, Université, bureau 1308, Montréal (Québec), H3B 3A7, représentée par Mackenzie I. Watson, son président, dûment autorisé aux fins des présentes, tel qu'il le déclare ;

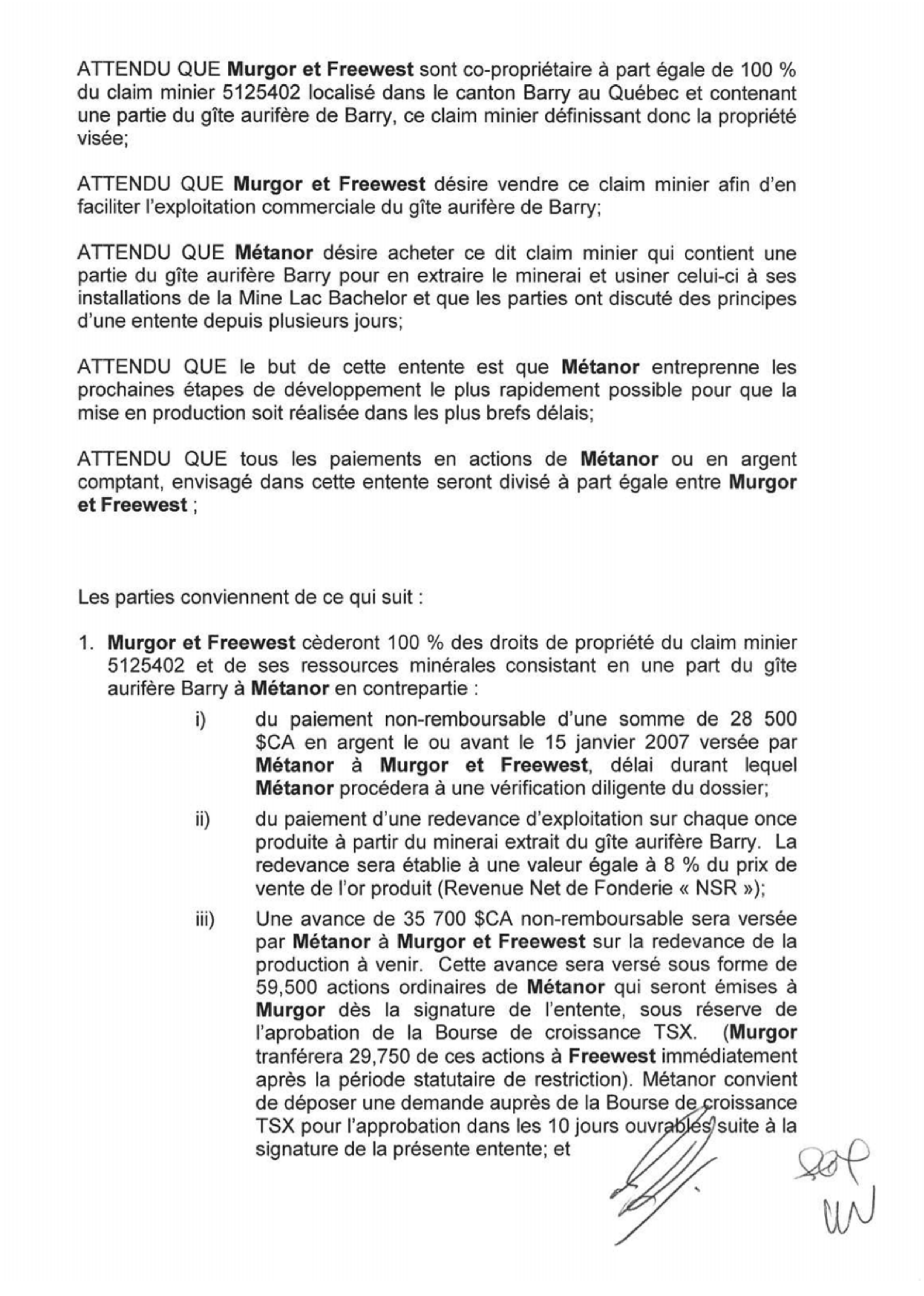
(ci-après désignée « **Freewest** »)

ET :

RESSOURCES MÉTANOR INC., une société régie par la *Loi canadienne sur les sociétés par actions*, ayant son siège social au 2872, chemin Sullivan, bureau 2, Sullivan (Québec), J0Y 2N0, représentée par Serge Roy, son président, dûment autorisé aux fins des présentes, tel qu'il le déclare ;

(ci-après désignée « **Métanor** »)

(ci-après désignés collectivement les « **Parties** »)



ATTENDU QUE **Murgor et Freewest** sont co-propriétaire à part égale de 100 % du claim minier 5125402 localisé dans le canton Barry au Québec et contenant une partie du gîte aurifère de Barry, ce claim minier définissant donc la propriété visée;

ATTENDU QUE **Murgor et Freewest** désire vendre ce claim minier afin d'en faciliter l'exploitation commerciale du gîte aurifère de Barry;

ATTENDU QUE **Métanor** désire acheter ce dit claim minier qui contient une partie du gîte aurifère Barry pour en extraire le minerai et usiner celui-ci à ses installations de la Mine Lac Bachelor et que les parties ont discuté des principes d'une entente depuis plusieurs jours;

ATTENDU QUE le but de cette entente est que **Métanor** entreprenne les prochaines étapes de développement le plus rapidement possible pour que la mise en production soit réalisée dans les plus brefs délais;

ATTENDU QUE tous les paiements en actions de **Métanor** ou en argent comptant, envisagé dans cette entente seront divisé à part égale entre **Murgor et Freewest** ;

Les parties conviennent de ce qui suit :

1. **Murgor et Freewest** cèderont 100 % des droits de propriété du claim minier 5125402 et de ses ressources minérales consistant en une part du gîte aurifère Barry à **Métanor** en contrepartie :
   - i) du paiement non-remboursable d'une somme de 28 500 $CA en argent le ou avant le 15 janvier 2007 versée par **Métanor** à **Murgor et Freewest**, délai durant lequel **Métanor** procédera à une vérification diligente du dossier;
   - ii) du paiement d'une redevance d'exploitation sur chaque once produite à partir du minerai extrait du gîte aurifère Barry. La redevance sera établie à une valeur égale à 8 % du prix de vente de l'or produit (Revenue Net de Fonderie « NSR »);
   - iii) Une avance de 35 700 $CA non-remboursable sera versée par **Métanor** à **Murgor et Freewest** sur la redevance de la production à venir. Cette avance sera versé sous forme de 59,500 actions ordinaires de **Métanor** qui seront émises à **Murgor** dès la signature de l'entente, sous réserve de l'aprobation de la Bourse de croissance TSX. (**Murgor** tranférera 29,750 de ces actions à **Freewest** immédiatement après la période statutaire de restriction). Métanor convient de déposer une demande auprès de la Bourse de croissance TSX pour l'approbation dans les 10 jours ouvrables suite à la signature de la présente entente; et

   iv) Une deuxième avance de 35 700 $CA en argent sera versée par **Métanor** à **Murgor et Freewest** sur la redevance de la production à venir. Cette deuxième avance sera versée à la première des deux situations suivantes : 1. 30 jours après l'obtention du permis d'exploitation ou 2. le premier janvier 2008.

Le remboursement des deux avances contemplé dans les paragraphes 1iii) et 1iv) se fera par une réduction de 50 % des montants de la redevance due à Murgor lors des coulées d'or qui suivront le début de l'exploitation commerciale du gîte aurifère Barry ;

2. Pour le calcul des redevances, le prix de l'or qui servira de référence à chaque coulée produite sera établi sur la base du prix du marché à chaque contrat de vente effectif;
3. Dans l'esprit de cette entente, les redevances seront payable à **Murgor et Freewest** par **Métanor** à la date de réception du produit de la vente de l'or;
4. **Murgor et Freewest** s'engage à transférer l'intérêt dans les substances minérales de la propriété des claims qui renferment le gîte aurifère Barry à **Métanor** dès que le paiement de 28 500 $CA aura été effectué;
5. **Murgor** devra transmettre aussitôt que possible toute l'information disponible sous quelque forme que ce soit (rapports, études, analyses ou toute autre donnée pertinente);
6. **Murgor** facilitera le transfert de tout permis, certificat d'autorisation ou autre autorisation spécifique de nature à favoriser le développement le plus rapide du projet ;
7. **Murgor** mettra à la disposition de **Métanor** les facilités d'hébergement du campement dont elle dispose à proximité de la propriété, et ce, pour la durée intégrale des activités de développement et d'exploitation du gîte aurifère Barry et **Métanor** s'engage à remettre ces installations d'hébergement dans le même état qu'elle les a trouvé au début de ses opérations;
8. **Métanor** assumera toute responsabilité résultant des opérations minières qui seront réalisées après la signature de l'entente, et elle s'engage à garder **Murgor** exempte de quelque recours que ce soit découlant des dites opérations minières;
9. **Métanor** procédera avec diligence à mettre en exploitation le gîte aurifère Barry au profit mutuel des deux entreprises. Les parties conviennent toutefois qu'aucune date spécifique de mise en production n'est convenue, non plus qu'aucun volume spécifique de production périodique puisque aucune date d'émission de permis ou de certificat d'autorisation n'est définie. Nonobstant ce qui précède, les parties conviennent que si l'exploitation du gite aurifère de Barry n'a pas débuté dans les 24 mois de la signature des présentes, les claims miniers seront retourné à Murgor libres de toute sureté ou intérêt ;
10. À compter de la date de signature de cette entente, **Métanor** s'engage a payer la redevance de deux pour cent (2%) du revenu net de fonderie (« **NSR** ») des substances minérales extraites des claims miniers mentionnés

à l'Entente d'acquisition (la « **Redevance** ») payable à **M. Jacques Duval** tel que le prévoit l'entente entre **Murgor et Freewest** et **M. Jacques Duval** et ci-jointe. **Métanor** convient et s'engage à indemniser et à tenir **Murgor et Freewest** à couvert de toute perte, réclamation, demande, poursuite, dommage, frais ou responsabilité, absolus ou conditionnels (incluant intérêts et frais raisonnables) que **Murgor et Freewest** pourrait, directement ou indirectement, subir ou encourir en raison de toute inexécution de l'obligation de la part de **Métanor** de payer la Redevance à **M. Jacques Duval.**

11. L'entente sera conditionnelle à l'obtention de l'autorisation des autorités des marchés financiers, et la date de cette autorisation sera celle à laquelle tous les versements dus à la signature devront être versés;
12. La présente entente est soumise aux lois de la province de Québec;

EN FOI DE QUOI, LES PARTIES ONT SIGNÉ LA PRÉSENTE CONVENTION CE ___^{e} JOUR DE DÉCEMBRE 2006.

M. Serge Roy
Président
Ressources Métanor Inc.
2872, chemin Sullivan, suite 2
Sullivan, Qc
J0Y 2N0

M. André C. Tessier
Président
Ressources Murgor Inc.
179, Sydenham, suite 102
Kingston, Ont,
K7K 3M1

M. Mackenzie I. Watson
Président
Ressources Freewest Canada Inc.
1155, University, suite 1308
Montréal, Qc
H3B 3A7

Entente d'acquisition du gîte aurifère Barry

ENTRE :

RESSOURCES MURGOR INC., une société régie par la *Loi canadienne sur les sociétés par actions*, ayant une place d'affaires au 179, Sydenham, bureau 102, Kingston (Ontario), K7K 3M1, représentée par André C. Tessier, son président, dûment autorisé aux fins des présentes, tel qu'il le déclare ;

(ci-après désignée « **Murgor** »)

ET :

RESSOURCES MÉTANOR INC., une société régie par la *Loi canadienne sur les sociétés par actions*, ayant son siège social au 2872, chemin Sullivan, bureau 2, Sullivan (Québec), J0Y 2N0, représentée par Serge Roy, son président, dûment autorisé aux fins des présentes, tel qu'il le déclare ;

(ci-après désignée « **Métanor** »)

(ci-après désignés collectivement
les « **Parties** »)

ATTENDU QUE **Murgor** est le propriétaire à 100 % des claims 399844-4, 399844-5, 406168-1, 406168-2, 406168-3 et 399836-1, localisé dans le canton Barry au Québec, et contenant la majorité du gîte aurifère de Barry ; ces claims définissant donc la propriété visée;

ATTENDU QUE **Murgor** désire vendre ces claims pour faciliter l'exploitation commerciale du gîte aurifère de Barry;

ATTENDU QUE **Métanor** désire acheter les claims mentionnés ci-haut pour en extraire le minerai du gîte aurifère Barry et usiner celui-ci à ses installations de la Mine Lac Bachelor;

ATTENDU QUE le but de cette entente est que **Métanor** entreprenne les prochaines étapes de développement le plus rapidement possible pour que la mise en production soit réalisée dans les plus brefs délais;

Les parties conviennent de ce qui suit :

1. **Murgor** cèdera 100 % des droits de propriété des claims suivants : 399844-4, 399844-5, 406168-1, 406168-2, 406168-3 et 399836-1 à **Métanor**, incluant la minéralisation du gîte aurifère Barry, en contrepartie :

   i) du paiement non-remboursable d'une somme de 171 500 $CA en argent le ou avant le 15 janvier 2007, délai durant lequel **Métanor** procédera à une vérification diligente du dossier;

   ii) du paiement d'une redevance d'exploitation sur chaque once produite à partir du minerai extrait du gîte aurifère Barry. La redevance sera établie à une valeur égale à 8 % du prix de vente de l'or produit (Revenue Net de Fonderie « NSR »);

   iii) Une avance de 214 300 $CA non-remboursable sera versée par **Métanor** à **Murgor** sur la redevance de la production à venir. Cette avance sera versé sous forme de 357,166 actions ordinaires de **Métanor** qui seront émises à **Murgor** dès la signature de l'entente, sous réserve de l'aprobation de la Bourse de croissance TSX. Métanor convient de déposer une demande auprès de la Bourse de croissance TSX pour l'approbation dans les 10 jours ouvrables suite à la signature de la présente entente; et

   iv) Une deuxième avance de 214 300 $CA en argent sera versée par **Métanor** à **Murgor** sur la redevance de la production à venir. Cette deuxième avance sera versée à la première des deux situations suivantes : 1. 30 jours après l'obtention du permis d'exploitation ou 2. le premier janvier 2008.

   Le remboursement des deux avances contemplé dans les paragraphes 1iii) et 1iv) se fera par une réduction de 50 % des montants de la redevance due à Murgor lors des coulées d'or qui suivront le début de l'exploitation commerciale du gîte aurifère Barry ;

2. Pour le calcul des redevances, le prix de l'or qui servira de référence à chaque coulée produite sera établi sur la base du prix du marché à chaque contrat de vente effectif;

3. Dans l'esprit de cette entente, les redevances seront payable à Murgor par **Métanor** à la date de réception du produit de la vente de l'or;

4. **Murgor** s'engage à transférer l'intérêt dans les substances minérales de la propriété des claims qui renferment le gîte aurifère Barry à **Métanor** dès que le paiement de 171 500 $CA aura été effectué;

5. **Murgor** devra transmettre aussitôt que possible toute l'information disponible sous quelque forme que ce soit (rapports, études, analyses ou toute autre donnée pertinente);

6. **Murgor** facilitera le transfert de tout permis, certificat d'autorisation ou autre autorisation spécifique de nature à favoriser le développement le plus rapide du projet ;

7. **Murgor** mettra à la disposition de **Métanor** les facilités d'hébergement du campement dont elle dispose à proximité de la propriété, et ce, pour la durée intégrale des activités de développement et d'exploitation du gîte aurifère Barry et **Métanor** s'engage à remettre ces installations d'hébergement dans le même état qu'elle les a trouvé au début de ses opérations;

8. **Métanor** assumera toute responsabilité résultant des opérations minières qui seront réalisées après la signature de l'entente, et elle s'engage à garder **Murgor** exempte de quelque recours que ce soit découlant des dites opérations minières;

9. **Métanor** procédera avec diligence à mettre en exploitation le gîte aurifère Barry au profit mutuel des deux entreprises. Les parties conviennent toutefois qu'aucune date spécifique de mise en production n'est convenue, non plus qu'aucun volume spécifique de production périodique puisque aucune date d'émission de permis ou de certificat d'autorisation n'est définie. Nonobstant ce qui précède, les parties conviennent que si l'exploitation du gite aurifère de Barry n'a pas débuté dans les 24 mois de la signature des présentes, les claims miniers seront retourné à Murgor libres de toute sureté ou intérêt ;

10. À compter de la date de signature de cette entente, **Métanor** s'engage a payer la redevance de deux pour cent (2%) du revenu net de fonderie (« **NSR** ») des substances minérales extraites des claims miniers mentionnés à l'Entente d'acquisition (la « **Redevance** ») payable à la Société de Développement de la Baie James (« **SDBJ** ») tel que le prévoit l'entente entre **Murgor** et la **SDBJ** et ci-jointe. **Métanor** convient et s'engage à indemniser et à tenir **Murgor** à couvert de toute perte, réclamation, demande, poursuite, dommage, frais ou responsabilité, absolus ou conditionnels (incluant intérêts et frais raisonnables) que **Murgor** pourrait, directement ou indirectement, subir ou encourir en raison de toute inexécution de l'obligation de la part de **Métanor** de payer la Redevance à **SDBJ**

10. L'entente sera conditionnelle à l'obtention de l'autorisation des autorités des marchés financiers, et la date de cette autorisation sera celle à laquelle tous les versements dus à la signature devront être versés;

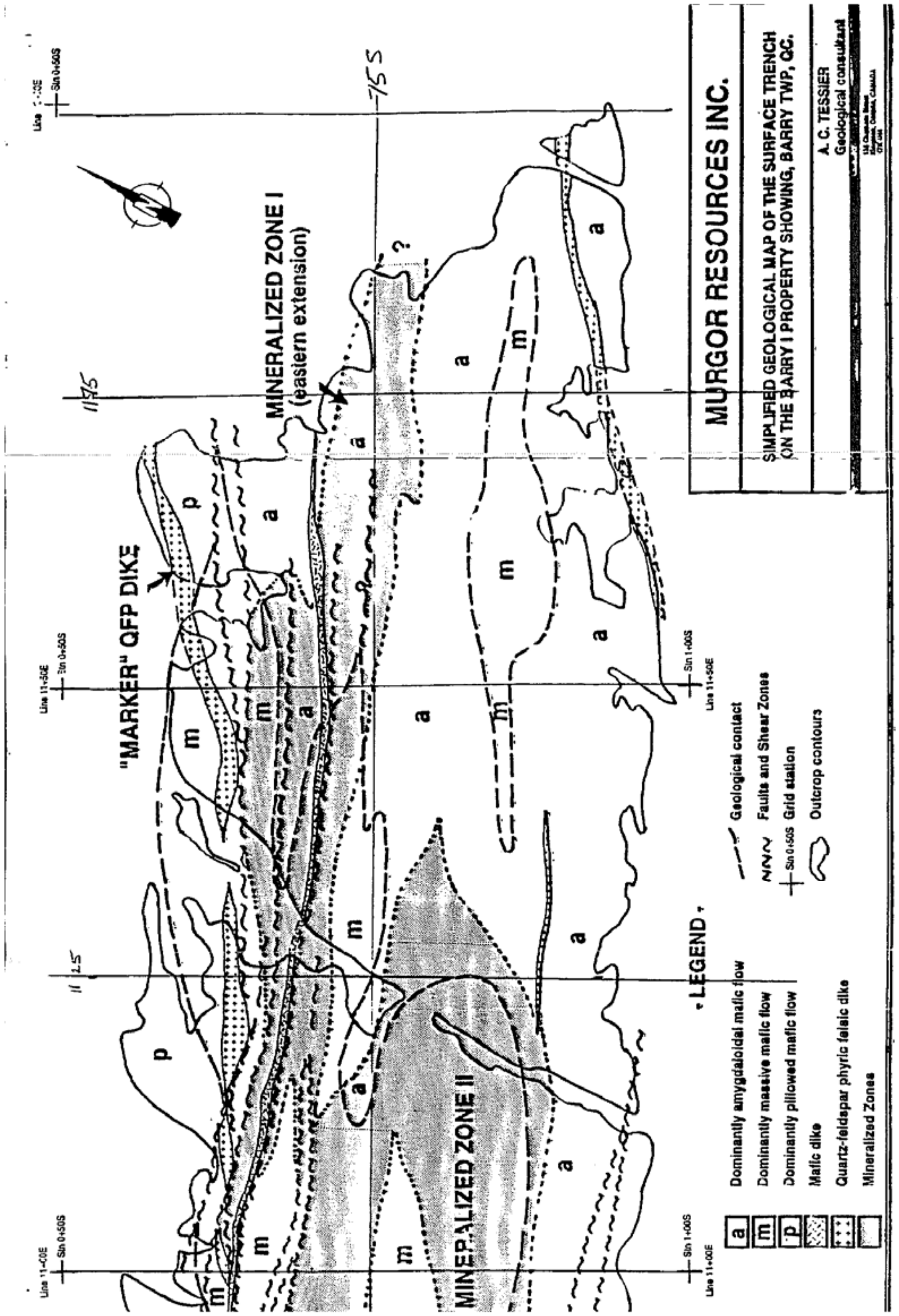
11. La présente entente est soumise aux lois de la province de Québec;

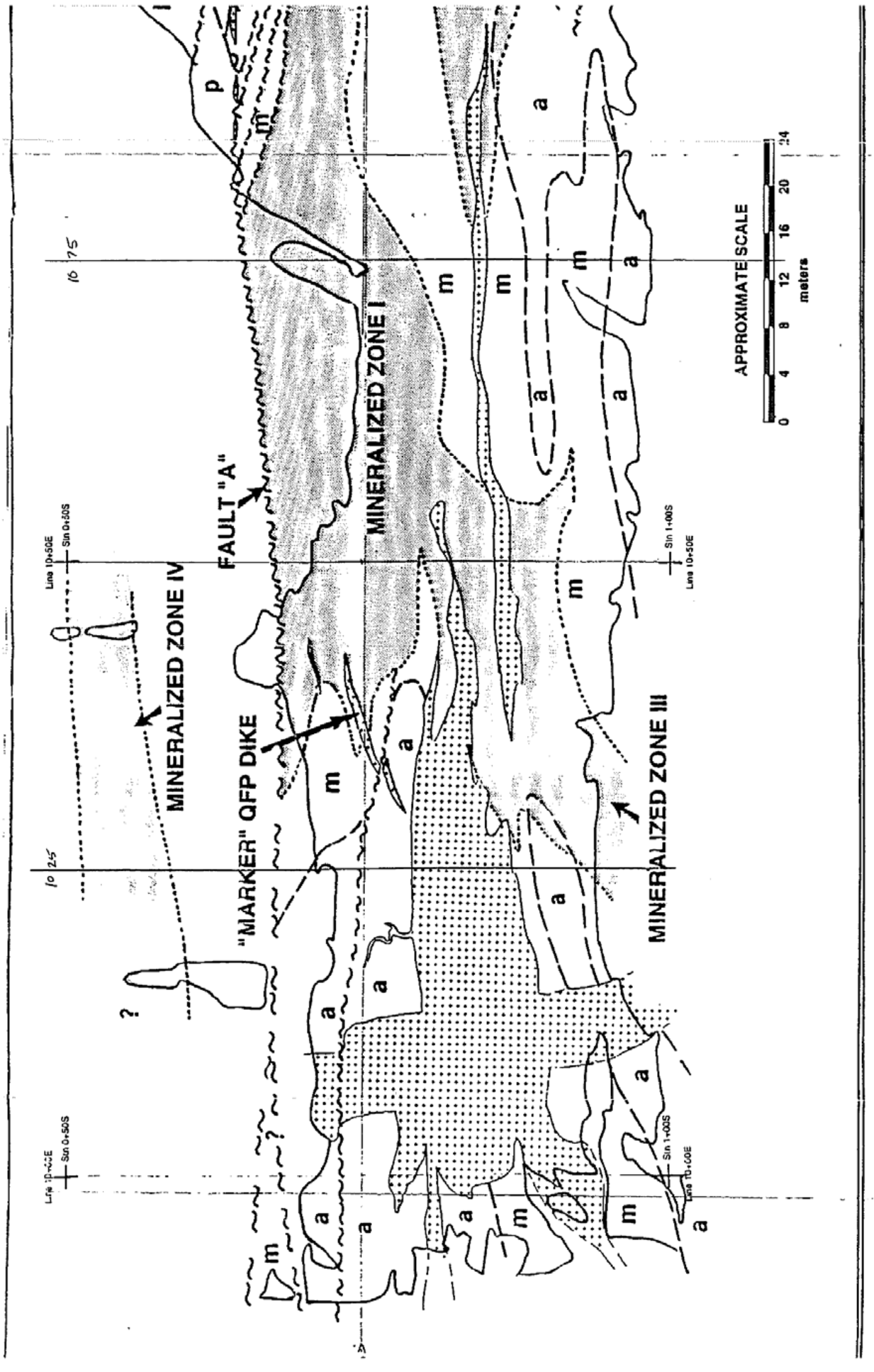
EN FOI DE QUOI, LES PARTIES ONT SIGNÉ LA PRÉSENTE CONVENTION CE ___e JOUR DE DÉCEMBRE 2006.

M. Serge Roy
Président
Ressources Métanor Inc.
2872, chemin Sullivan, suite 2
Sullivan, Qc
J0Y 2N0

M. André C. Tessier
Président
Ressources Murgor Inc.
179, Sydenham, suite 102
Kingston, Ont,
K7K 3M1

Appendix 2: Scan on the simplified version of the Barry I Main Zone Area mapped area from the 1995 mapping.




MINERALIZED ZONE IV
"MARKER" QFP DIKE
FAULT "A"
MINERALIZED ZONE I
MINERALIZED ZONE III
APPROXIMATE SCALE
0 4 8 12 16 20 24
meters
Line 10+50E
Stn 0+50S
Stn 1+00S
Line 10+50E
Line 10+00E
Stn 0+50S
Stn 1+00S
Line 10+00E
10 25
10 75
p
m
a
?

Appendix 3: Procedure of samples preparation and assay.

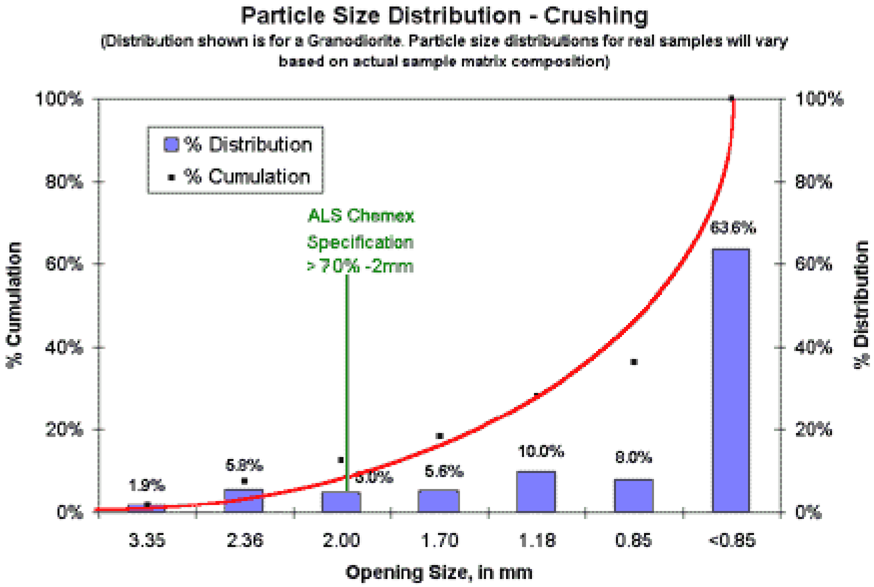
ALS Chemex

Sample Preparation Procedures

(From ALS Chemex web site)

Crushing

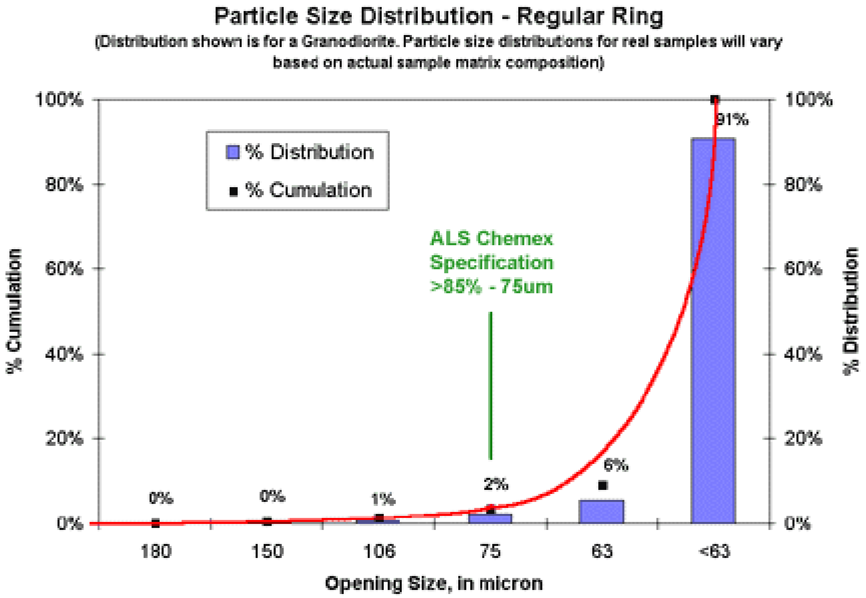
Samples that require crushing are dried at 110-120°C and then crushed with either an oscillating jaw crusher or a roll crusher. The ALS Chemex QC specification for crushed material is that >70% of the sample must pass a 2 mm (10 meshes) screen (as in graph below).



Note that if the whole sample need to be pulverized, then this condition becomes irrelevant. Crushing charges are based on the sample weight. The entire sample is crushed, but depending on the method, only a portion of the crushed material may be carried through to the pulverizing stage. That amount, typically 250 g to 1 kg, is subdivided from the main sample by use of a riffle splitter. If splitting is required, a substantial part of the sample (the "reject" or “spare”) remains. Ordinarily we retain a 1-2 kg split of this reject, but if a client wishes to pay a small additional charge, then we will retain the entire reject.

Pulverizing

A whole or split portion derived from the crushing process is pulverised using a ring mill. The size of the split is determined by the client based on the pulverising procedure that is selected. Split sizes for manganese or chrome steel rings are typically 250 g to 4 kg; however split sizes for zirconia rings are 100 g and those for tungsten carbide rings are only 75 g. Because of the relative lightness of these latter two materials, the size of the sample to be pulverized must necessarily be reduced to these weights in order to achieve the ALS Chemex QC specification for final pulverizing, namely that >85% of the sample be less than 75 microns (200 meshes) (see graph below).



For those samples which require enhanced homogeneity, such as samples which are known to exhibit coarse gold behaviour, intermediate pulverization of the entire sample (or a representative split) is also available

Gold assays

Gravimetric Methods

Gravimetric methods involve the use of balances to weigh the element of interest, either in its pure elemental form or as a chemical compound. One of the most common gravimetric determinations is that of gold and silver following a fire assay fusion and cupellation. The precious metal bead that remains following cupellation is an alloy of silver and gold. Weighing this bead will give the total weight of silver and gold. If the bead is then treated with dilute nitric acid, it is possible to remove the silver quantitatively. The residual mass consists of pure gold, which can then be weighed separately, thus allowing the silver to be determined by difference. The balances used for this purpose are microbalances capable of weighing to the nearest microgram (one millionth of a gram). Analysis of bullion for gold, silver and base metal content is another common procedure.

Lead Collection

The standard fire assay procedure has been used for millennia to dissolve and separate gold, silver and other precious metals. In the first part of the fire assay, precious metals are dissolved using an aggressive fusion mixture consisting of litharge (lead oxide) and a variety of other fluxes such as sodium carbonate, borax, silica, potassium nitrate and household flour. During the complex reactions that occur between sample and the flux mixture, the litharge is reduced to molten lead and the silica within the sample is oxidized to a borosilicate slag. The molten lead that is produced within the reaction mixture forms as tiny droplets throughout. Because of the high specific gravity of the lead droplets, they filter down through the reaction mixture, dissolving and collecting the precious metals as they do so. In an ideal fusion, the end result is a clean two-phase melt in which the barren borosilicate slag floats on top of the molten lead containing the precious metals. When this two-phase melt is poured into an iron mold to cool, the lead solidifies and can be recovered.

The subsequent separation of lead and precious metals occurs during the next step known as cupellation.

Cupellation

Cupellation most commonly refers to that part of the fire assay process. Following a successful fusion, the analyst is left with a lead "button" which contains all the precious metals from a particular sample. Cupellation is the process by which the lead is separated from the precious metals. Cupellation is considered "total" if the lead is removed in its entirety and "partial" if it is not. For the determination of gold, silver, platinum, palladium, a total cupellation is standard. In this case, the lead button is placed on a magnesia cupel in a furnace at 960-1000°C. At this temperature, the lead melts and is simultaneously oxidised. Part of the lead is volatilised and part is drawn into the cupel by capillary attraction. Eventually the lead is entirely removed and what remains behind is a small precious metal bead that represents the entire precious metal content of the original sample. This bead can then be analysed by a variety of methods.

Appendix 4: Sections of the Barry I Main Zone Area deposit.

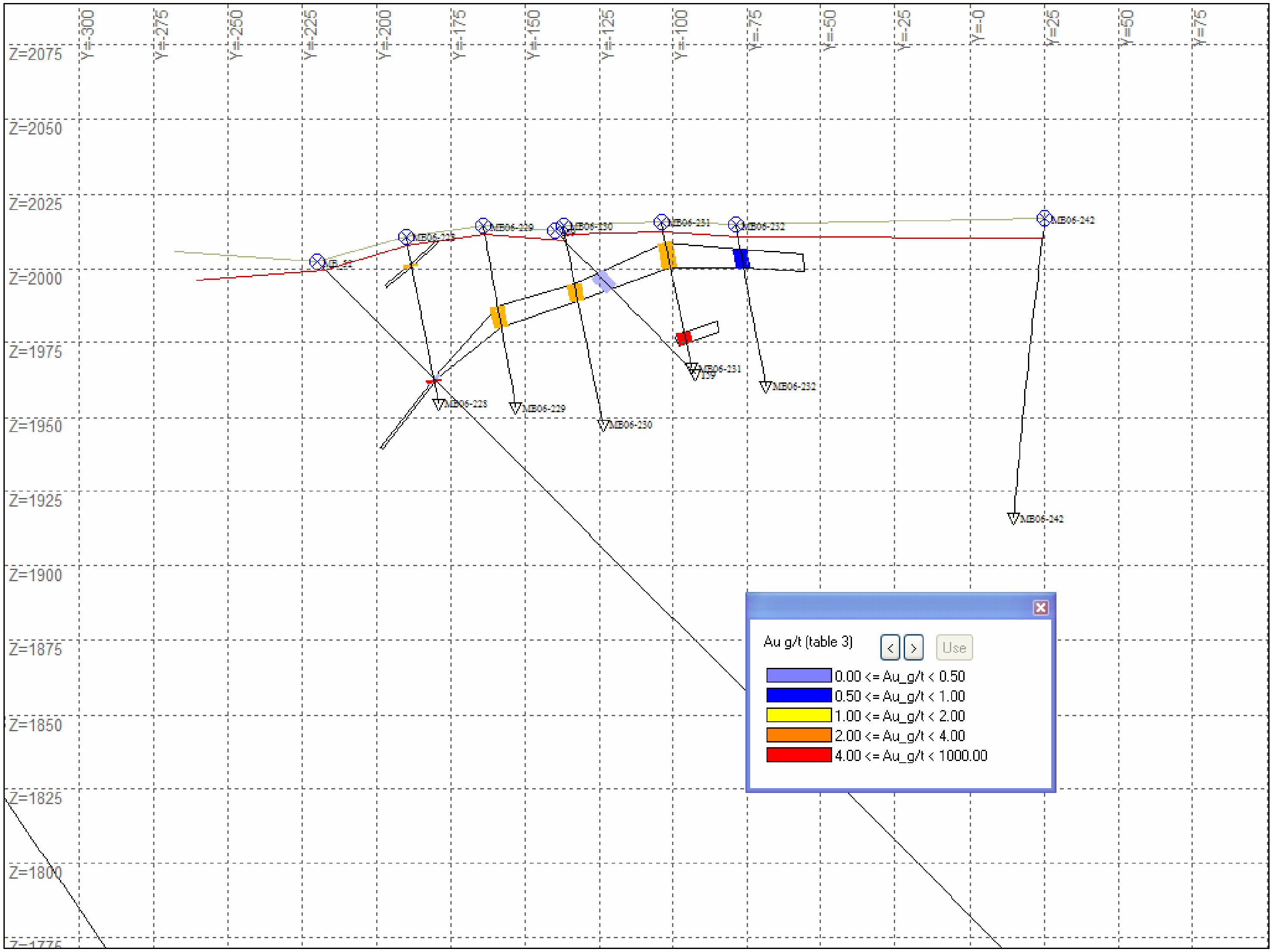


Figure 4: Section 650mE

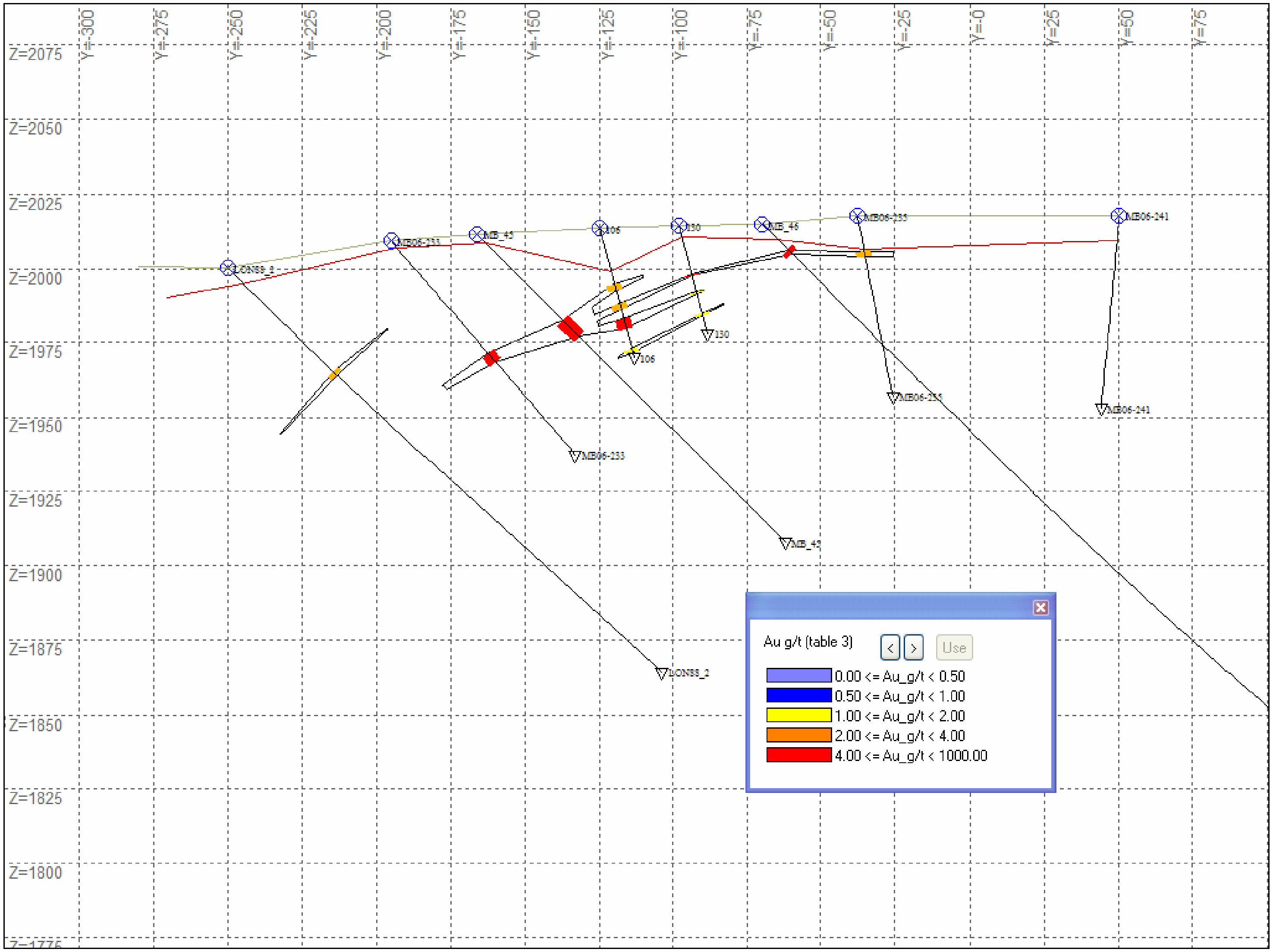


Figure 5: Section 700mE

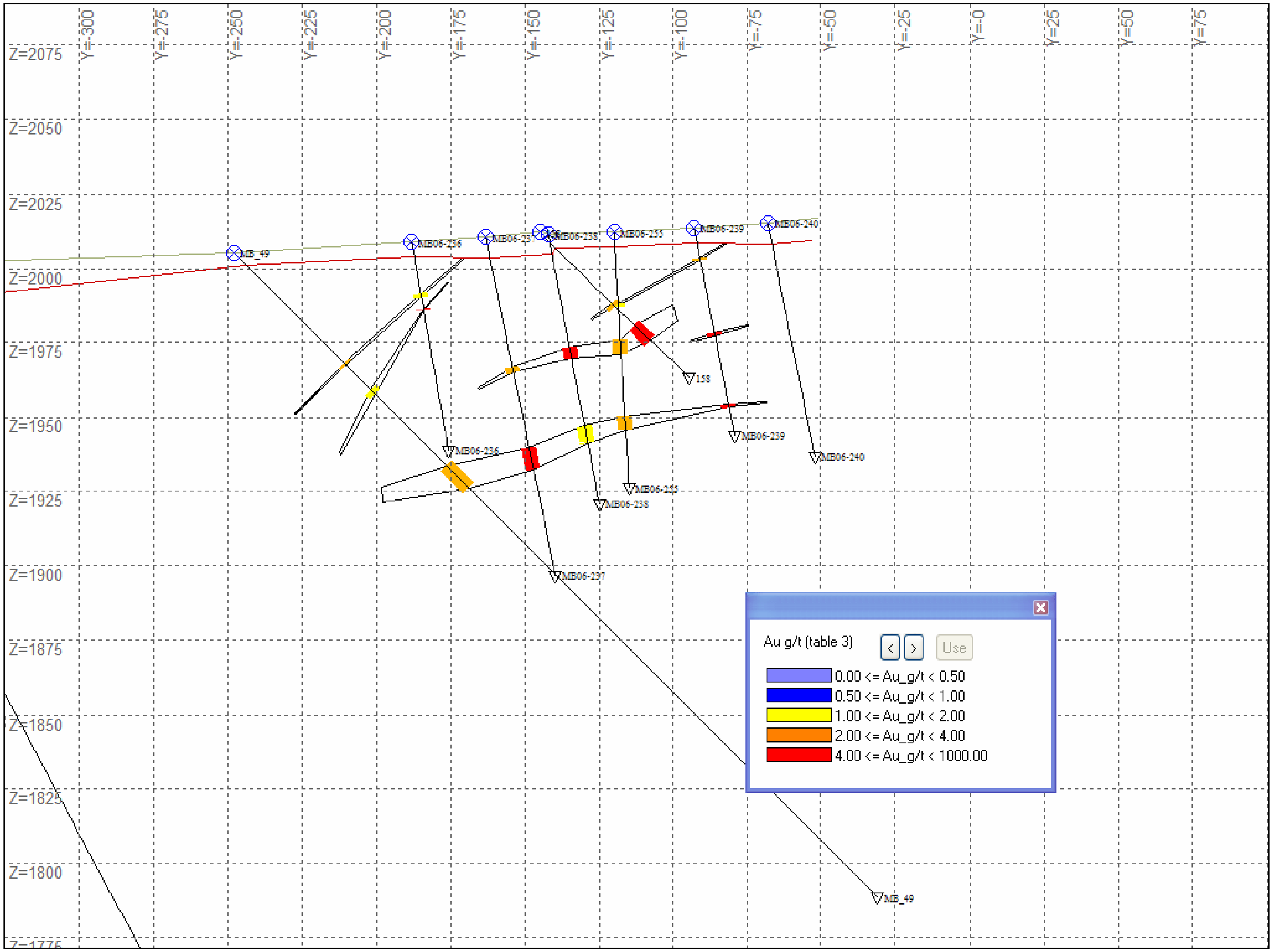


Figure 6: Section 750mE

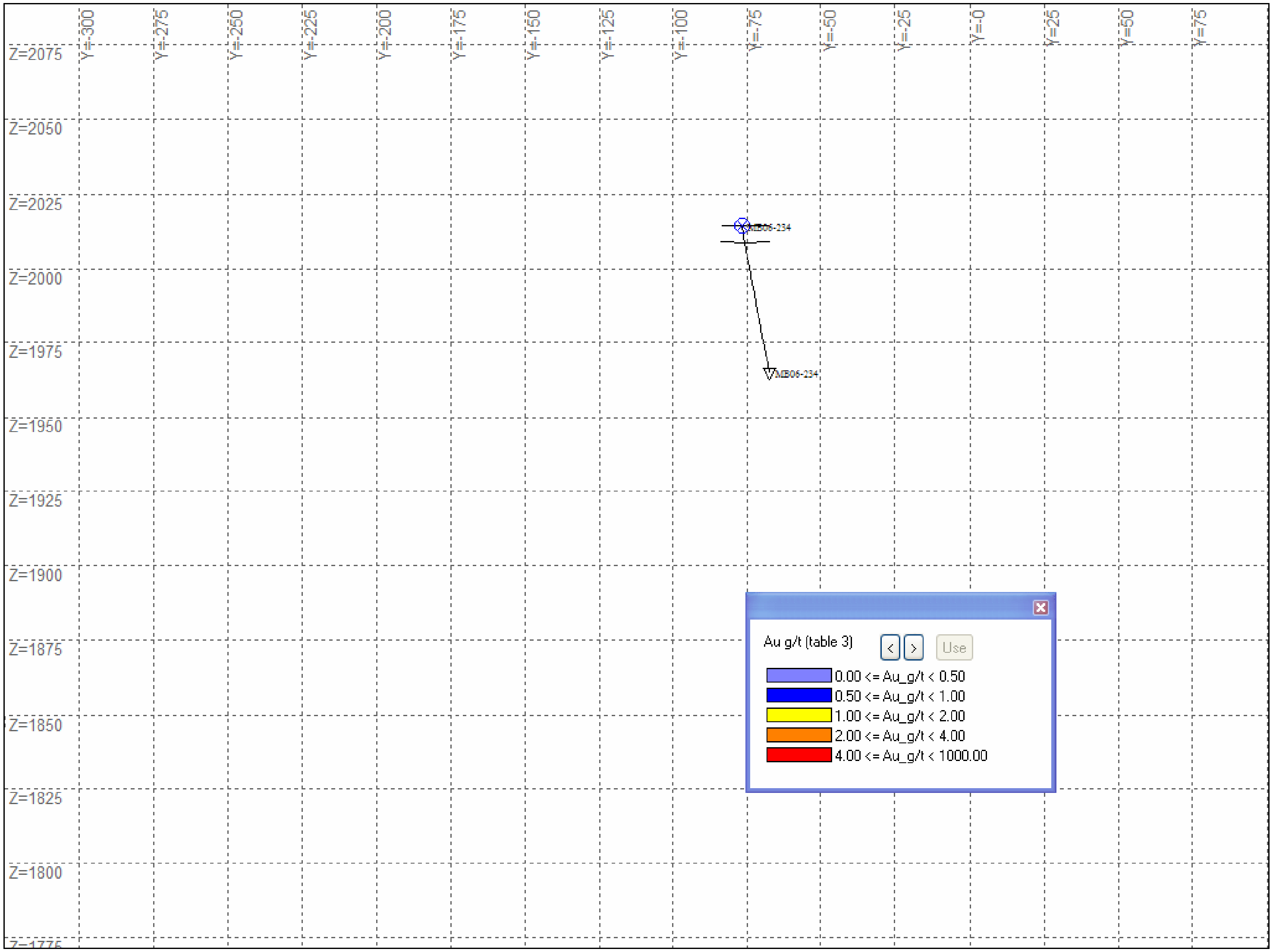


Figure 7: Section 775mE

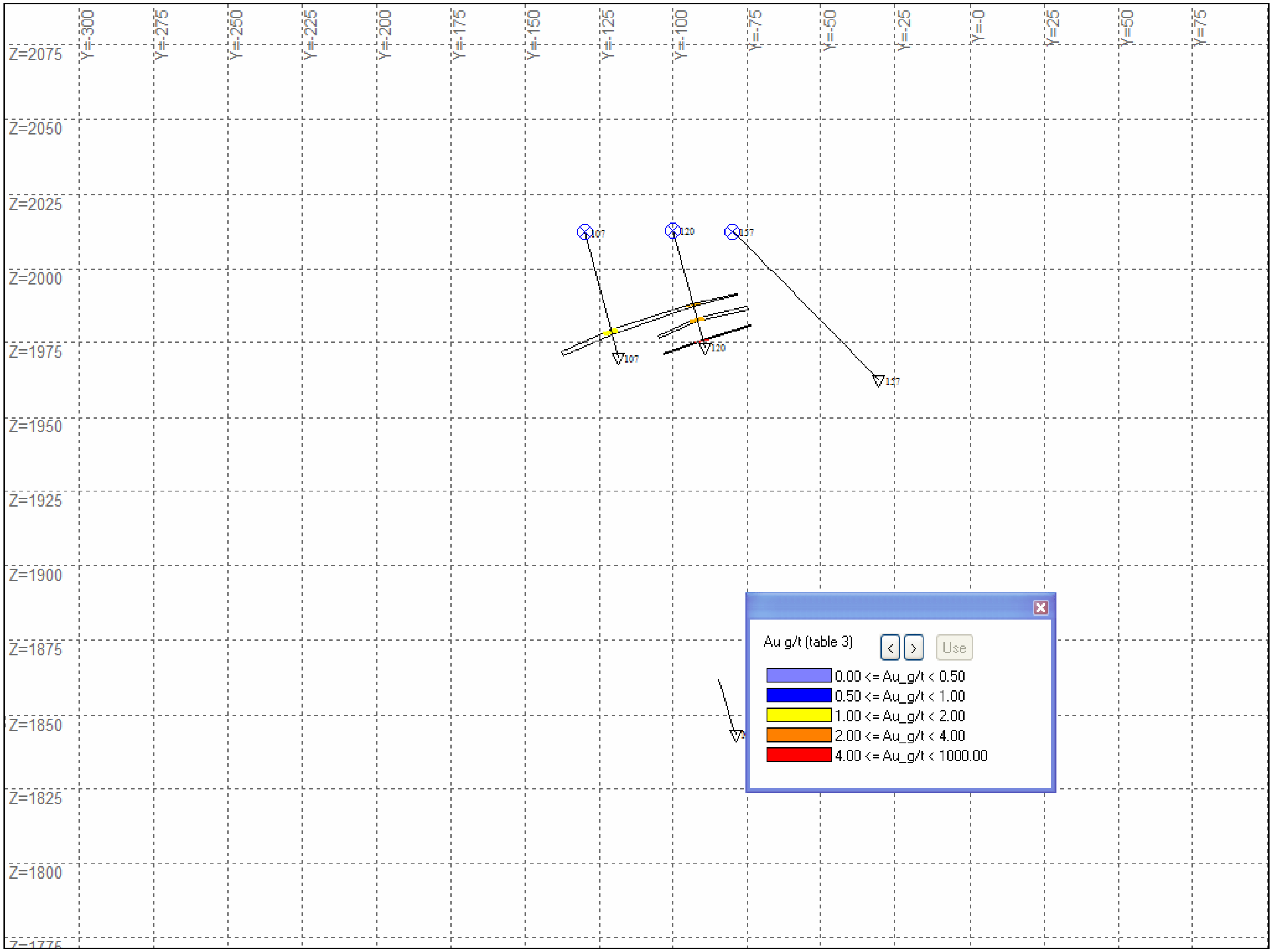


Figure 8: Section 791mE

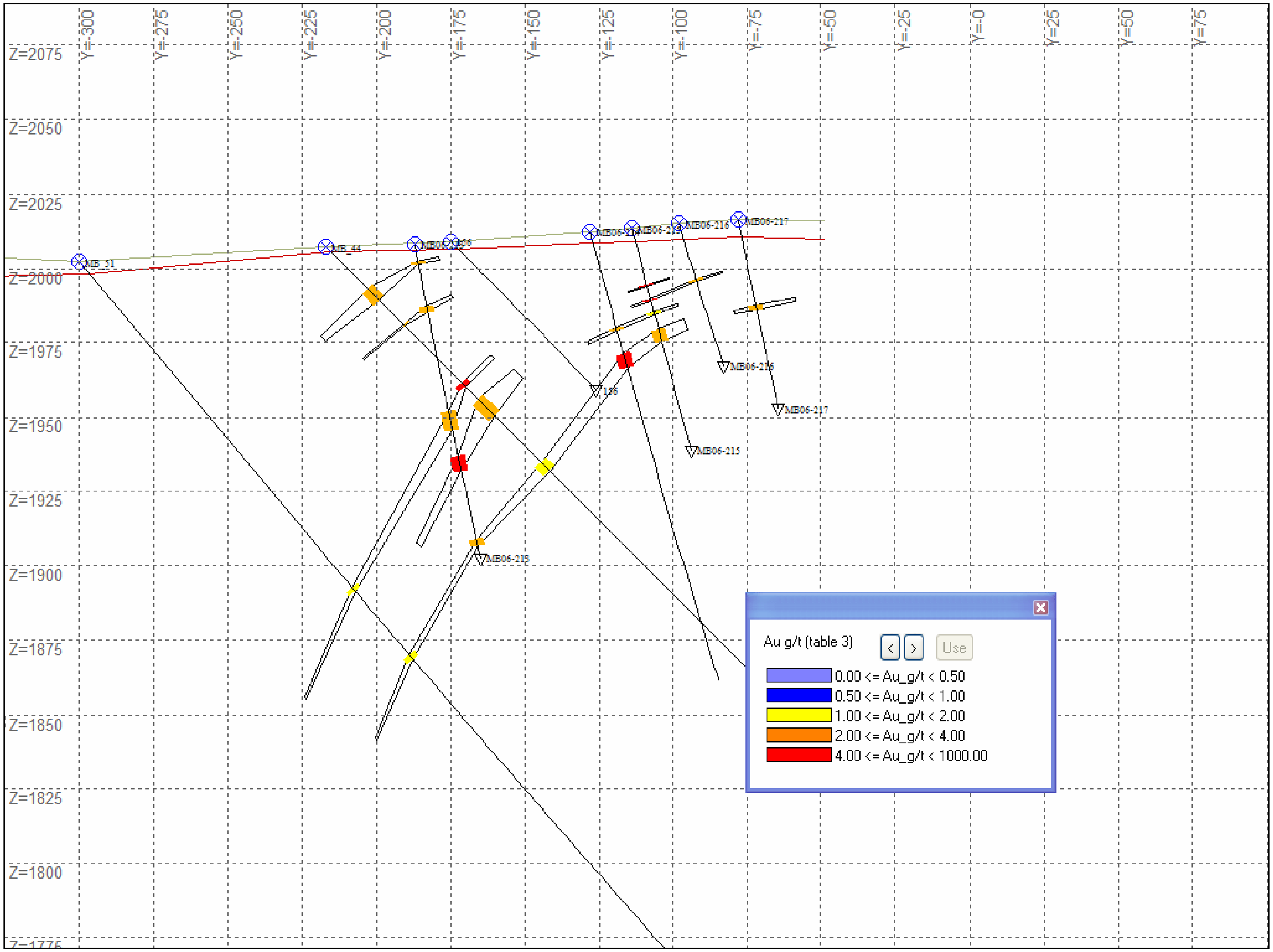


Figure 9: Section 800mE

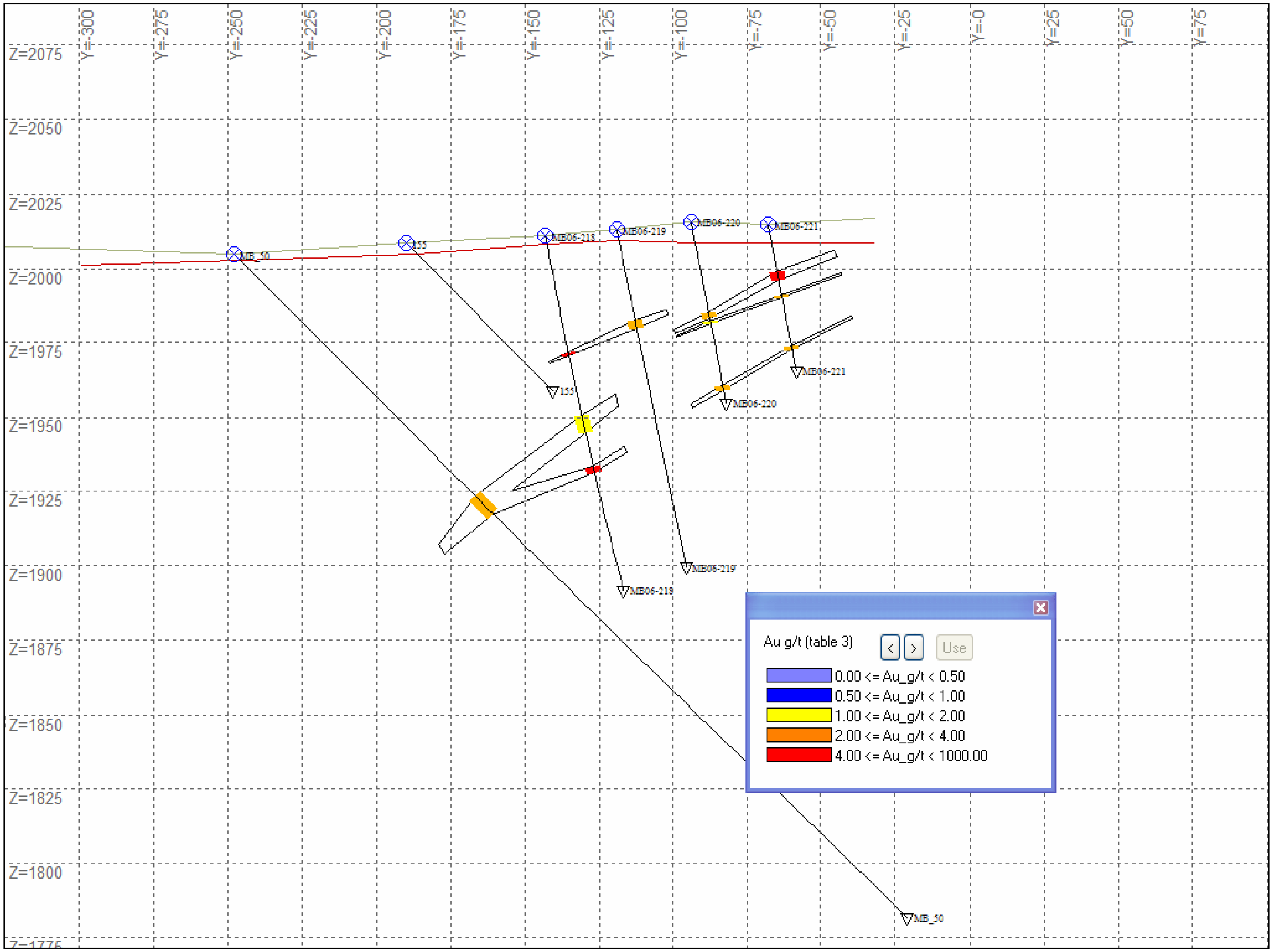


Figure 10: Section 850mE

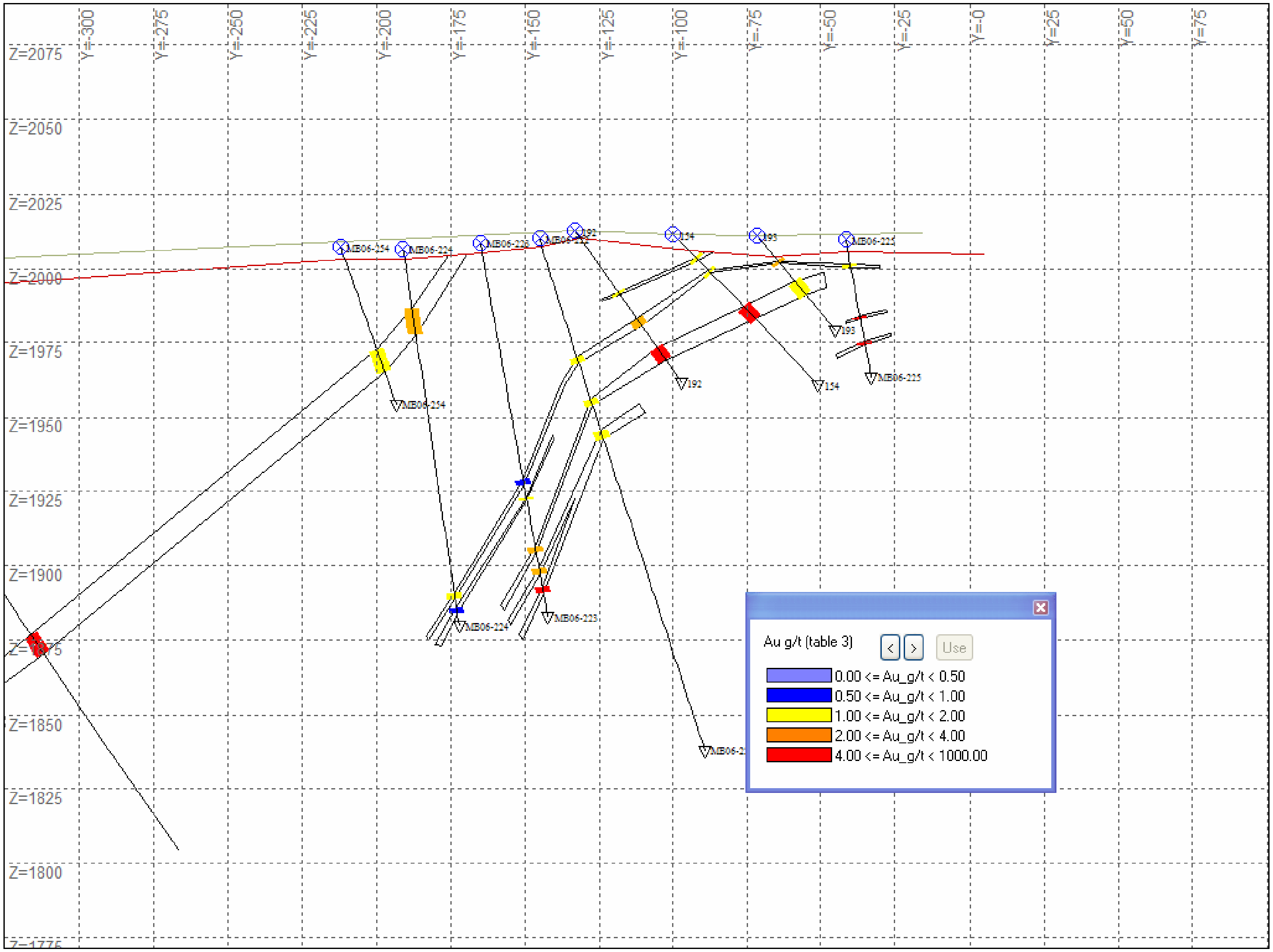


Figure 11: Section 900mE

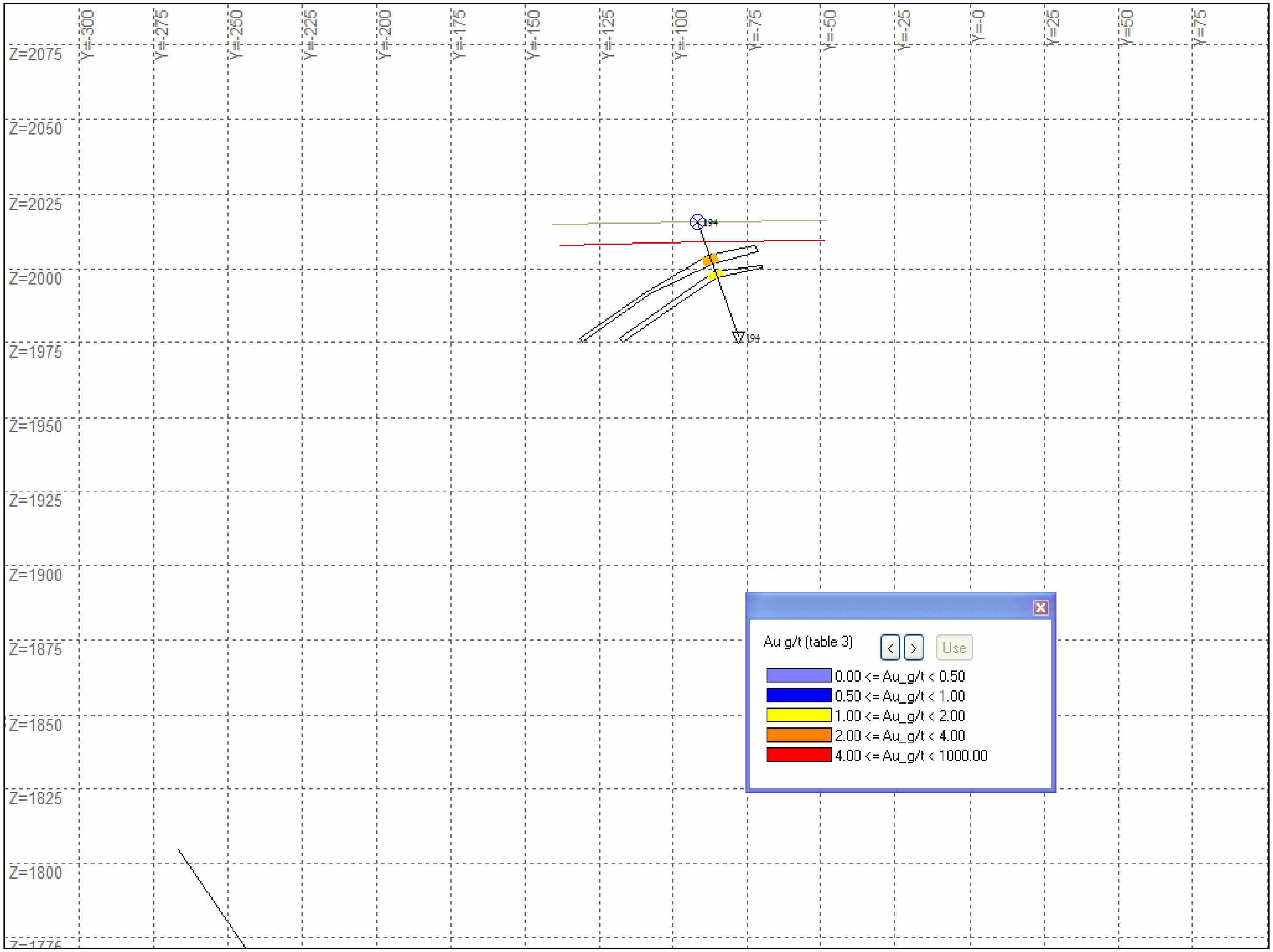


Figure 12: Section 933mE

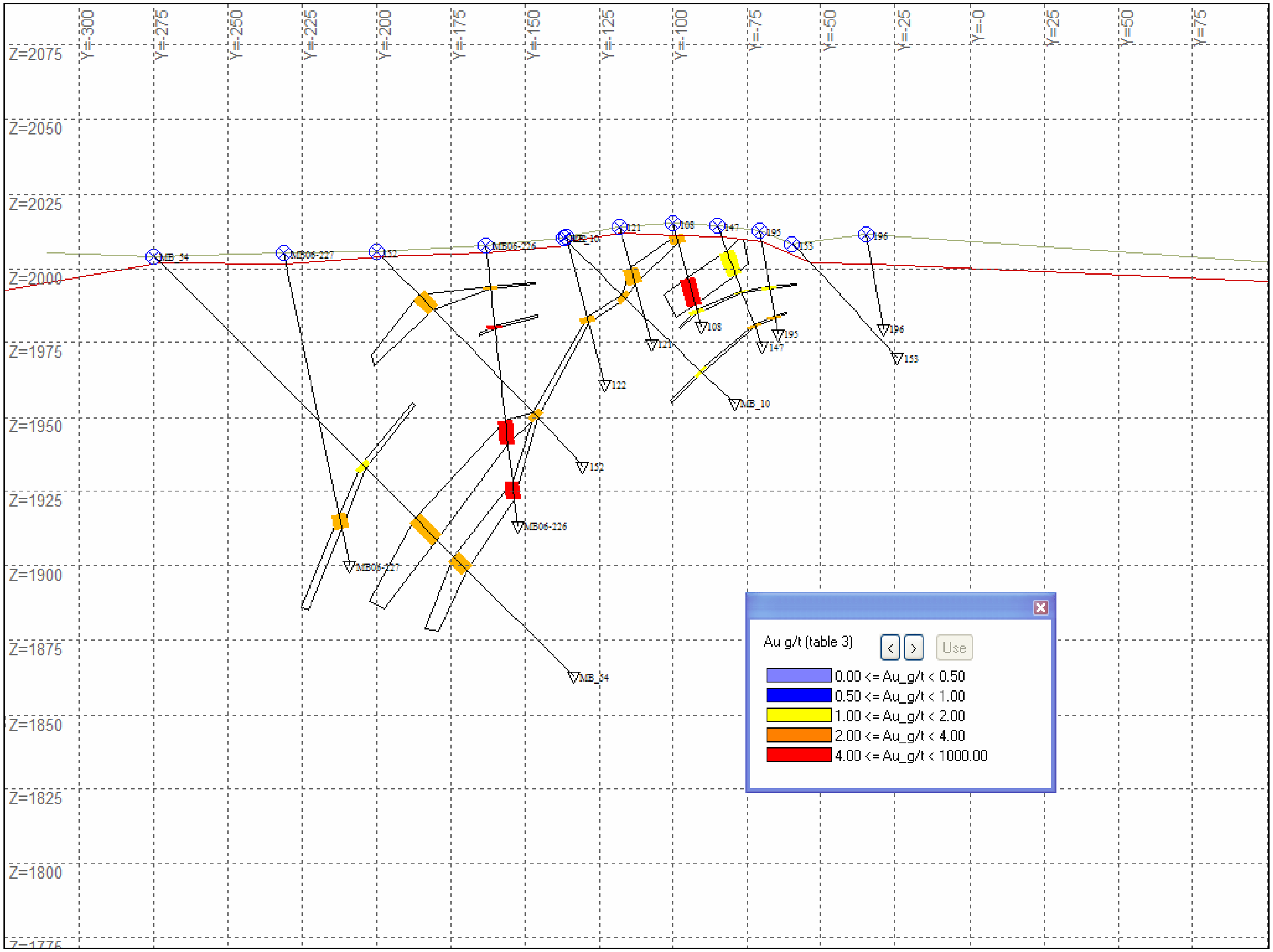


Figure 13: Section 950mE

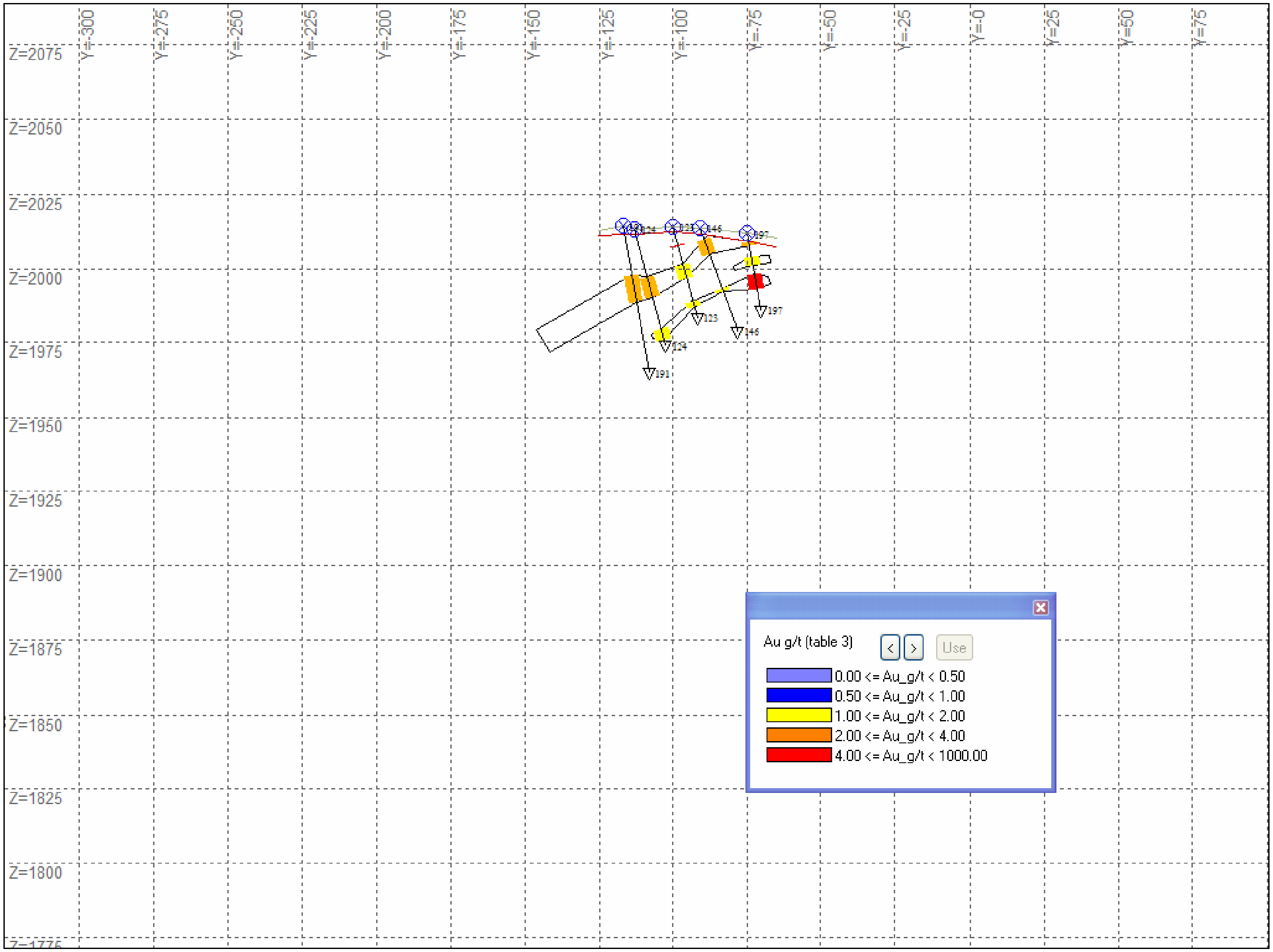


Figure 14: Section 975mE

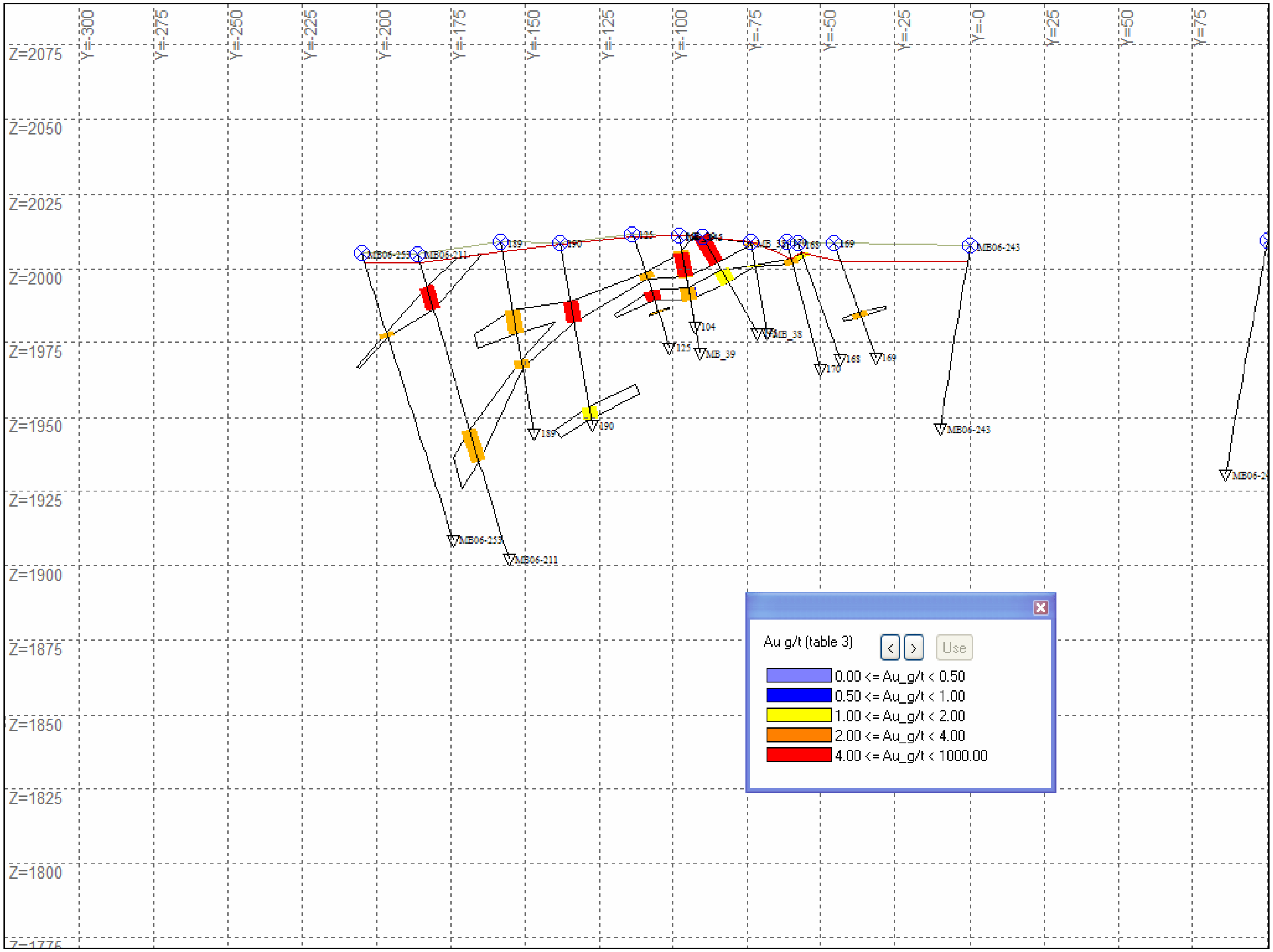


Figure 15: Section 995mE

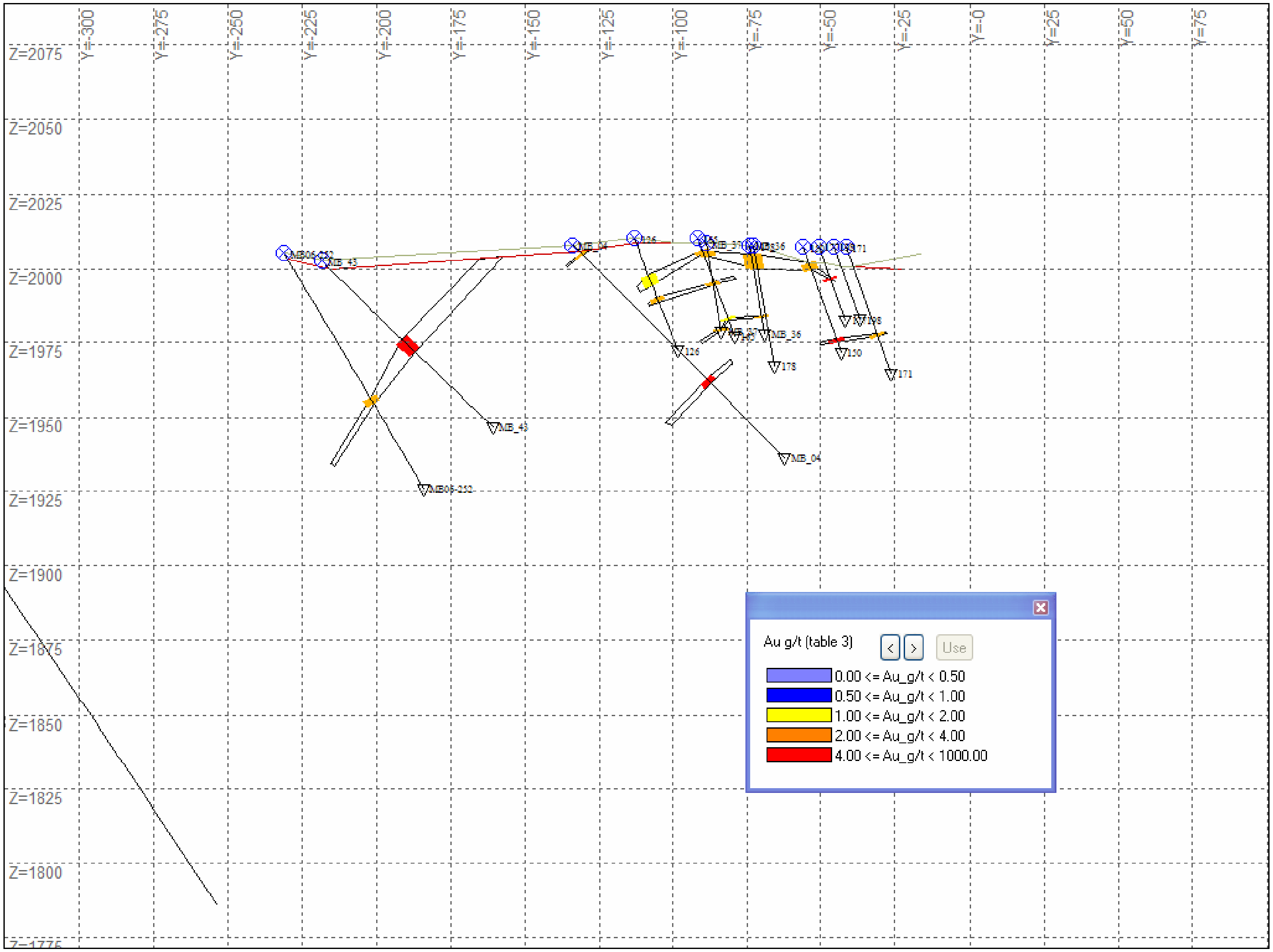


Figure 16: Section 1019mE

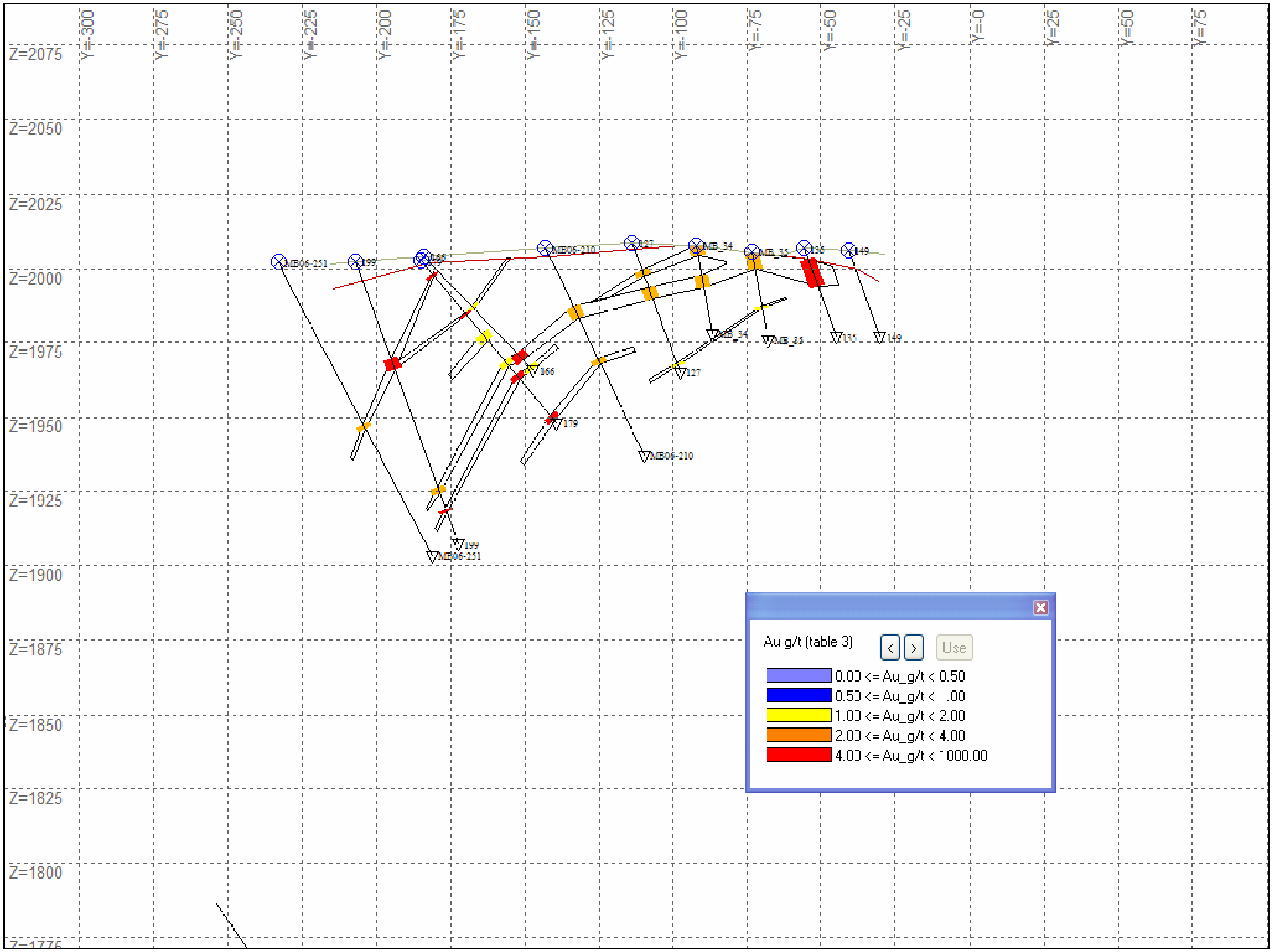
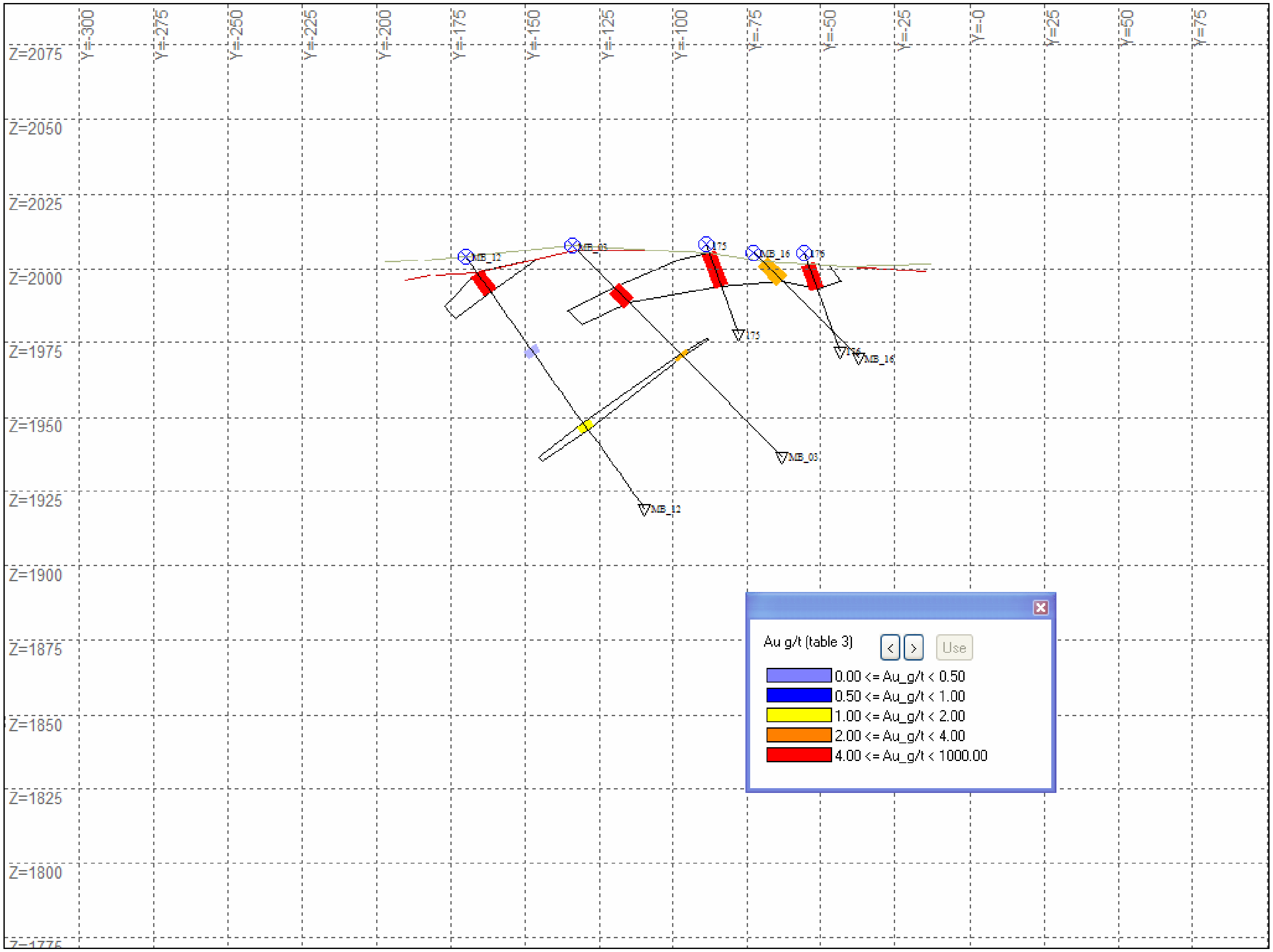


Figure 17: Section 1037mE



🟪 0.00 <= Au_g/t < 0.50
🟦 0.50 <= Au_g/t < 1.00
🟨 1.00 <= Au_g/t < 2.00
🟧 2.00 <= Au_g/t < 4.00
🟥 4.00 <= Au_g/t < 1000.00

Figure 18: Section 1048mE

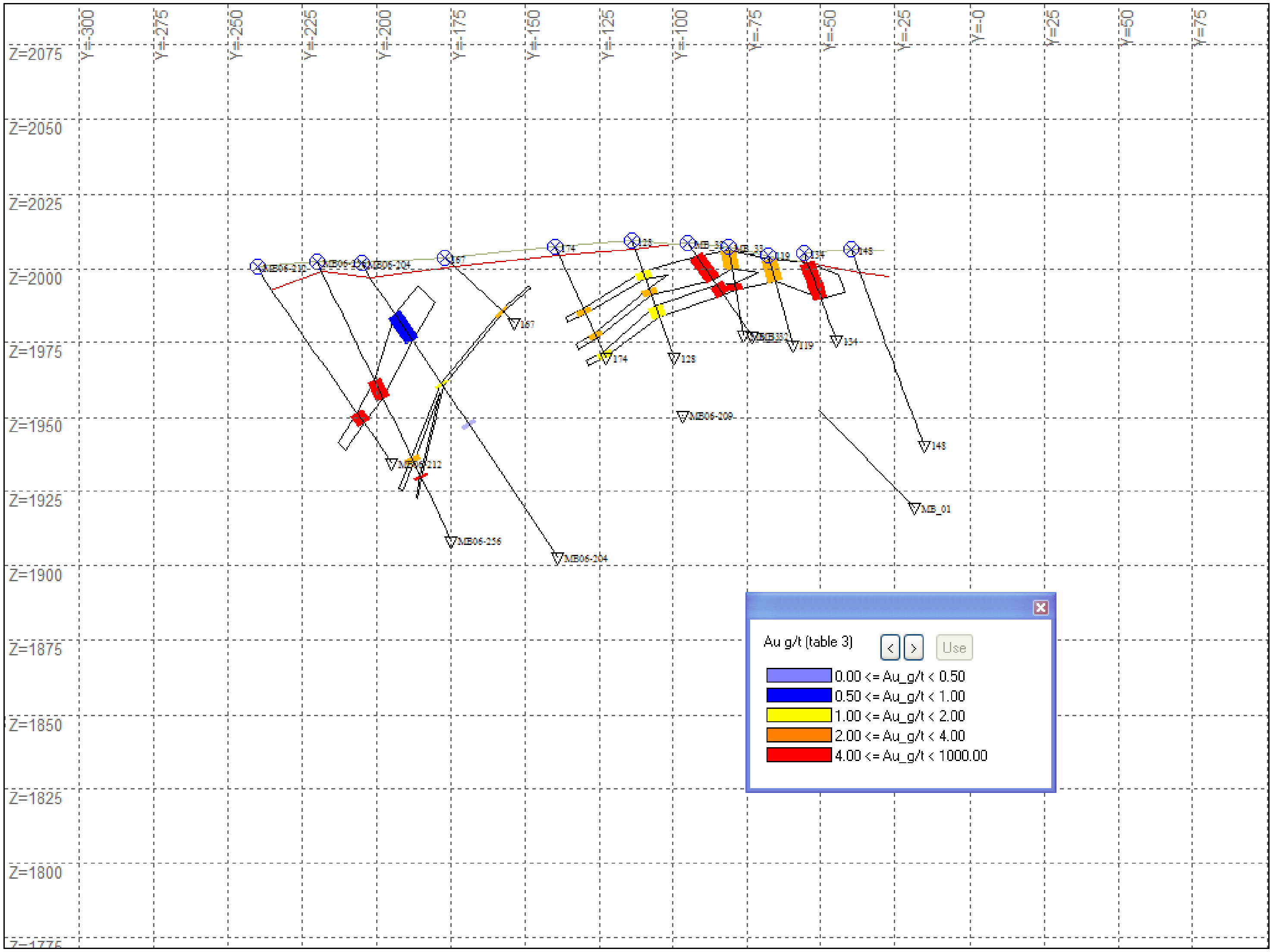


Figure 19: Section 1057mE

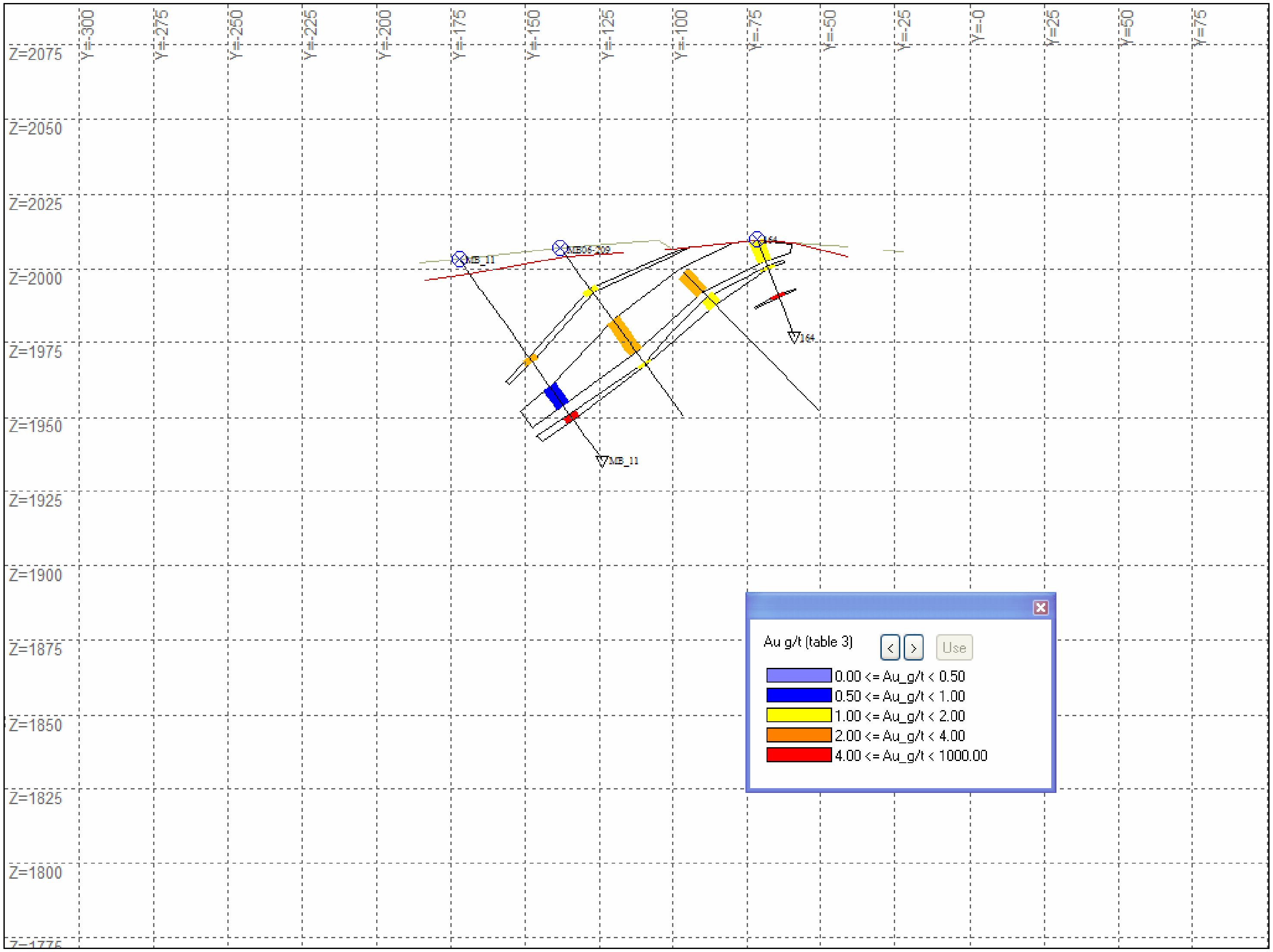


Figure 20: Section 1065mE

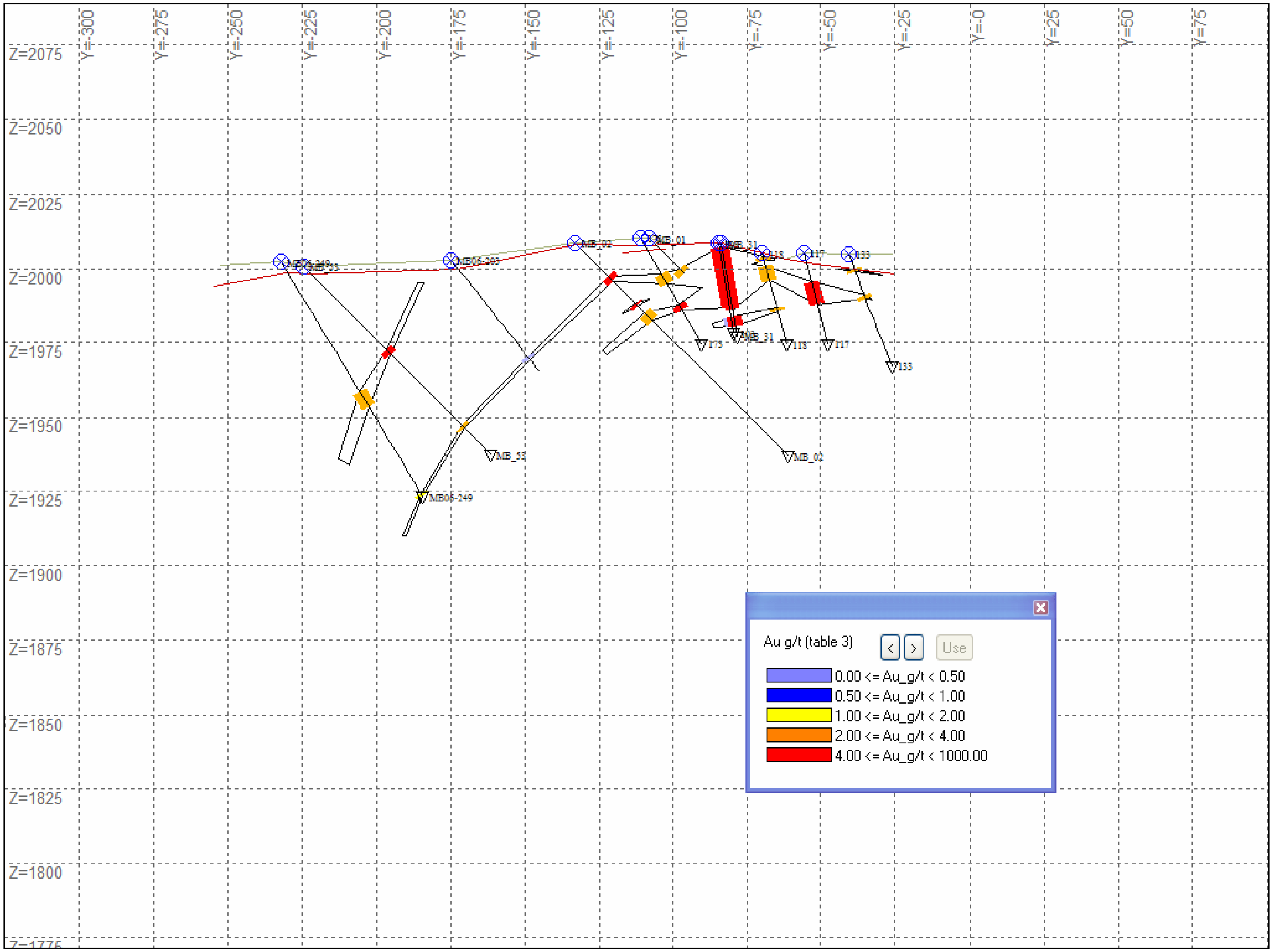


Figure 21: Section 1073mE

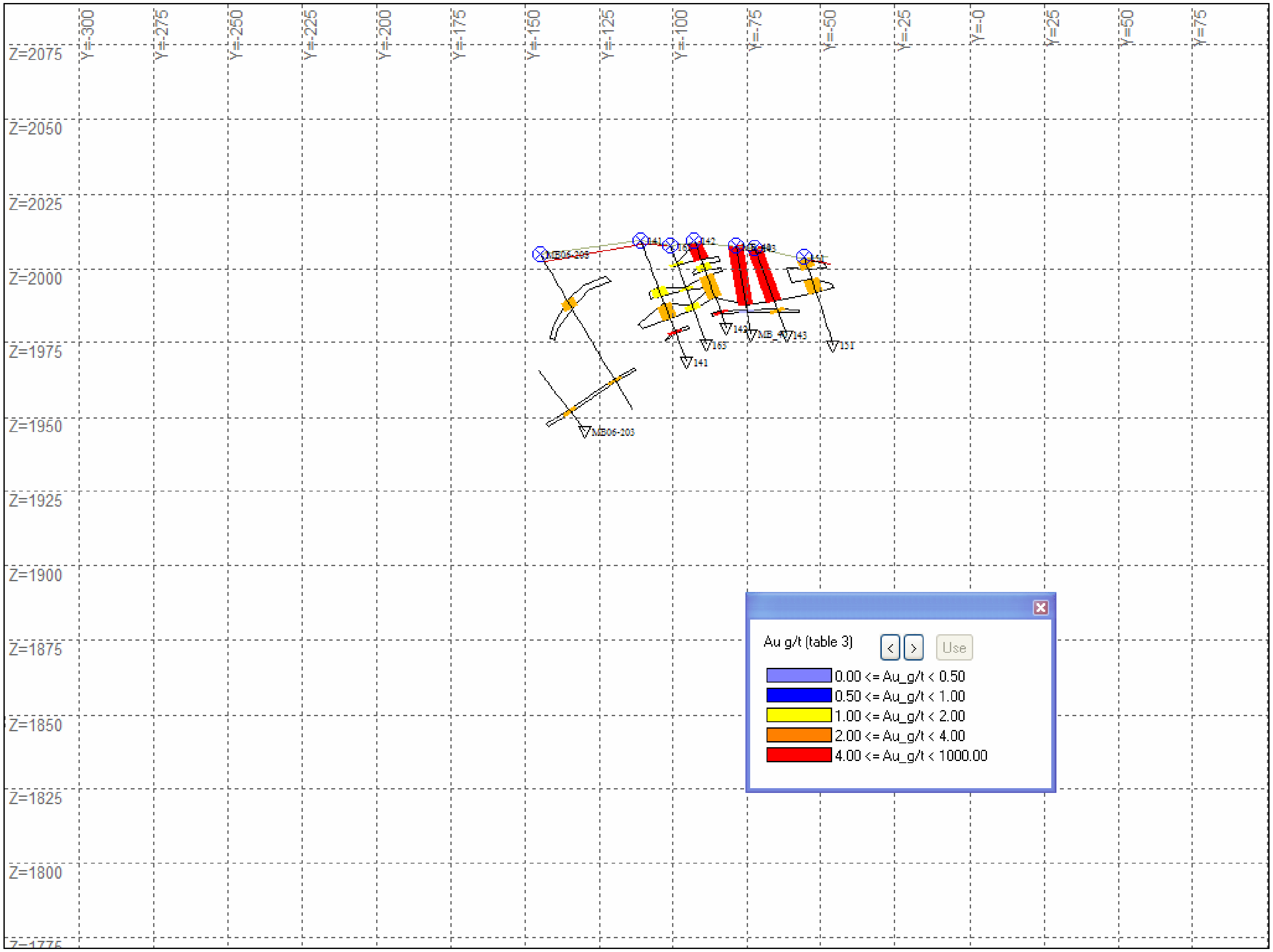


Figure 22: Section 1089mE

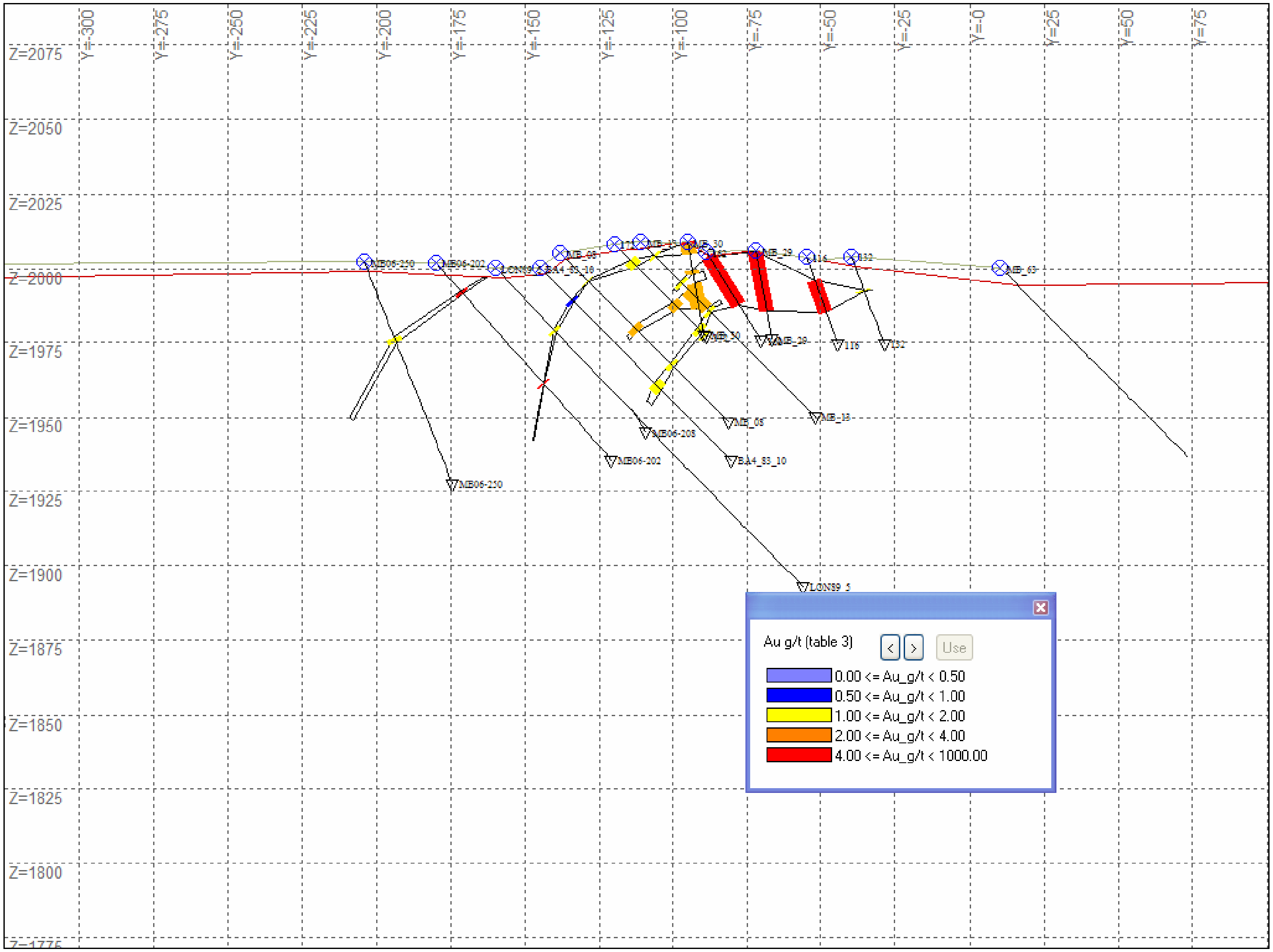


Figure 23: Section 1098mE

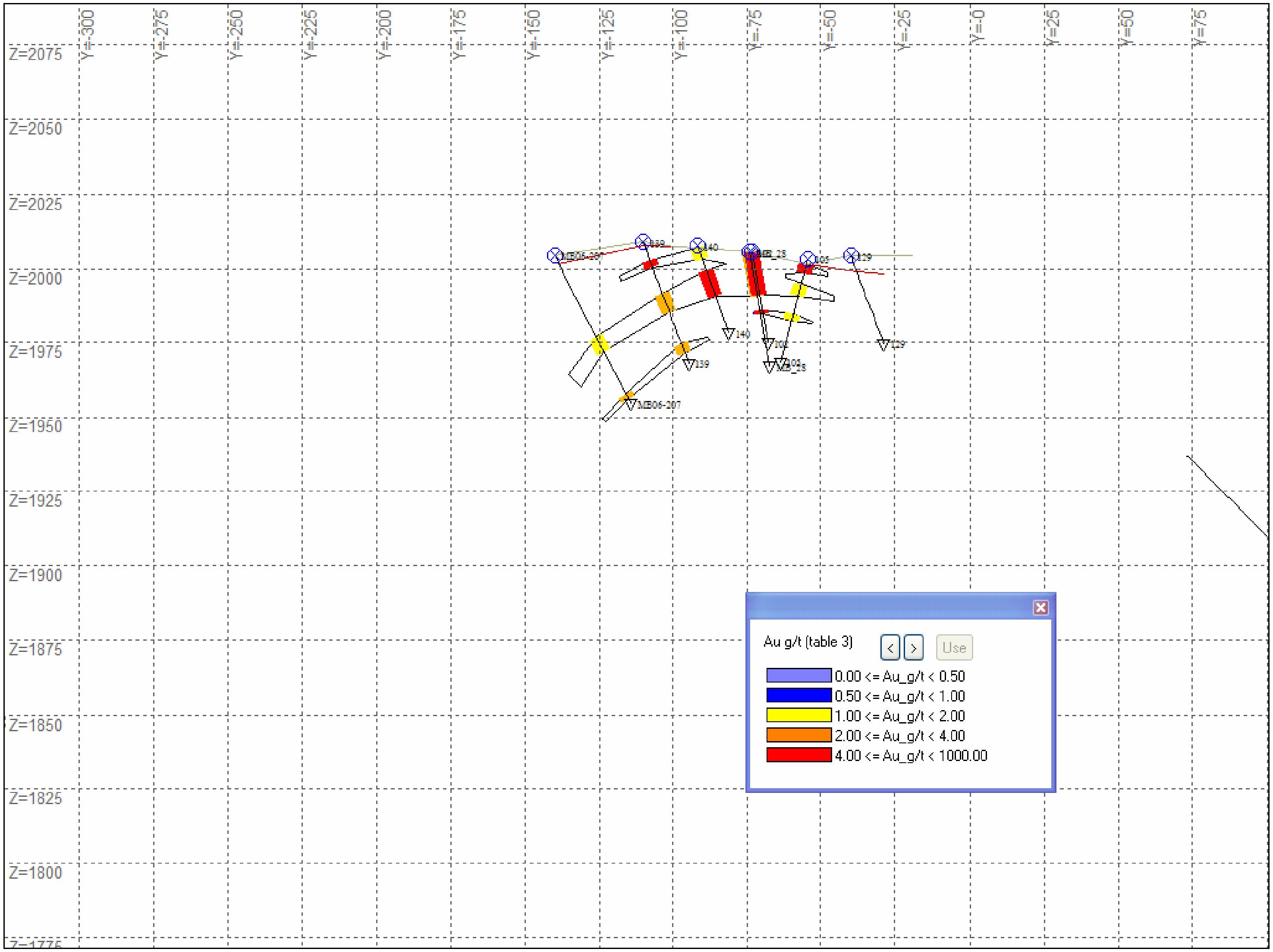


Figure 24: Section 1111mE

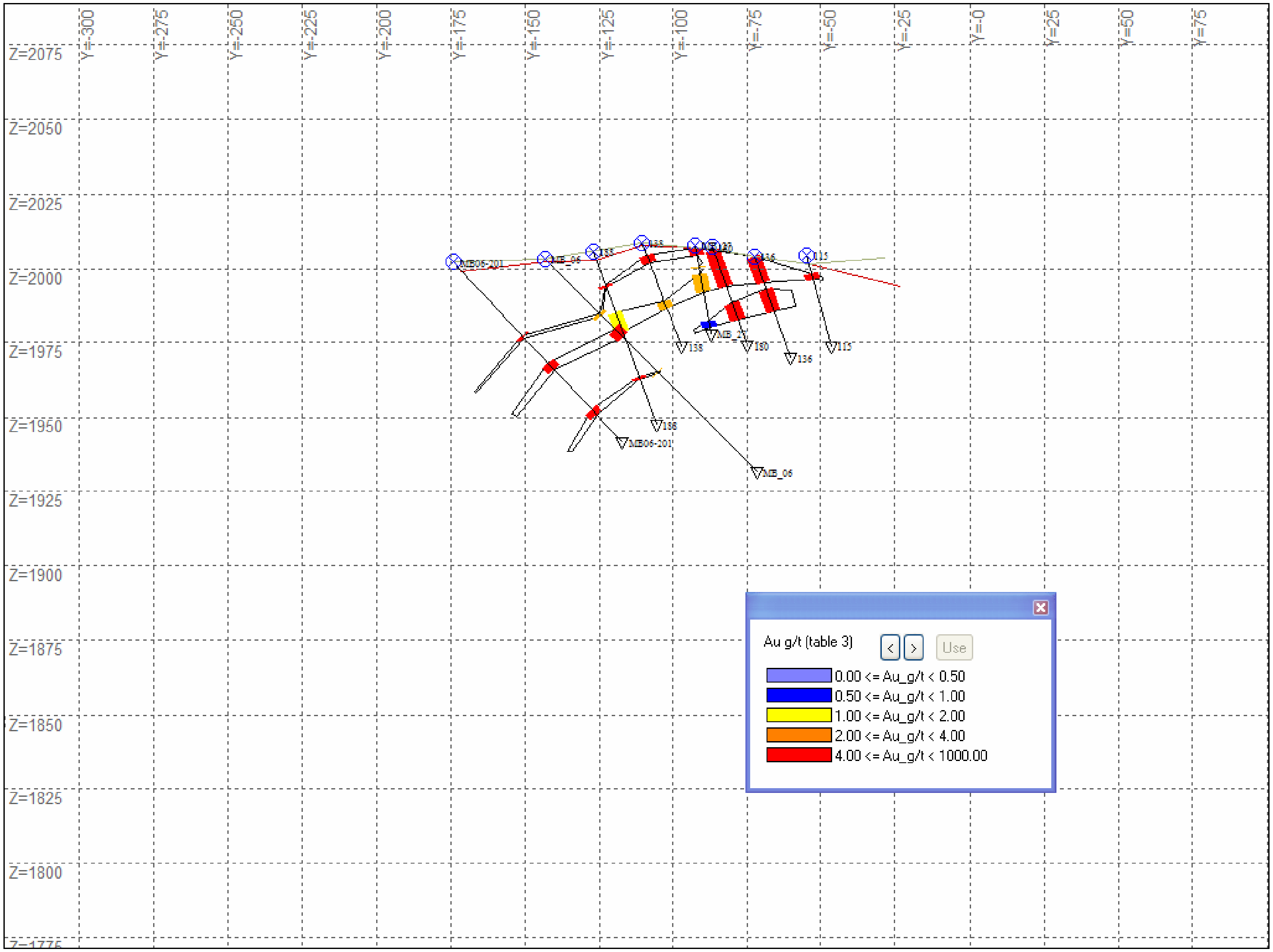


Figure 25: Section 1125mE

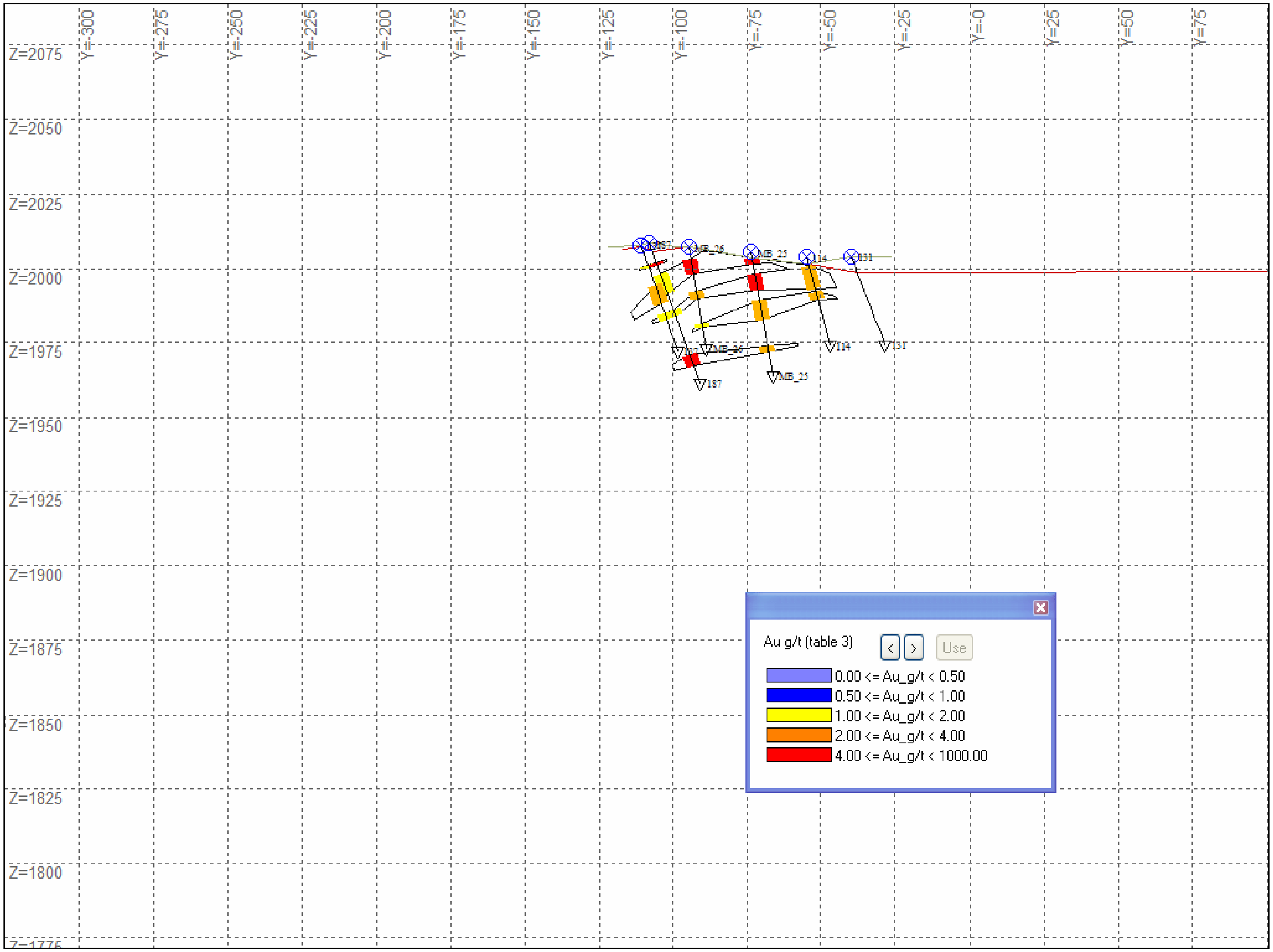


Figure 26: Section 1137mE

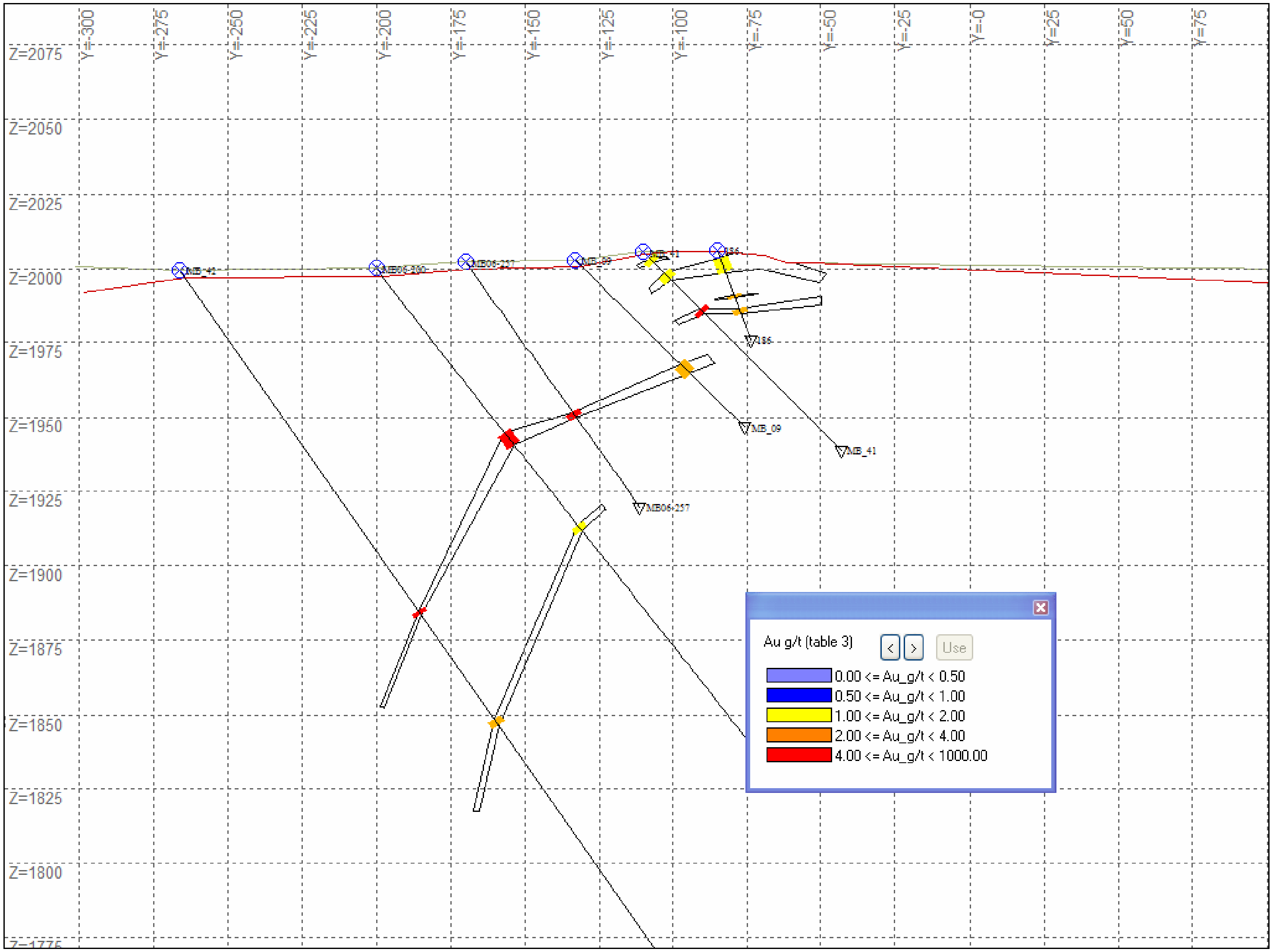


Figure 27: Section 1147mE

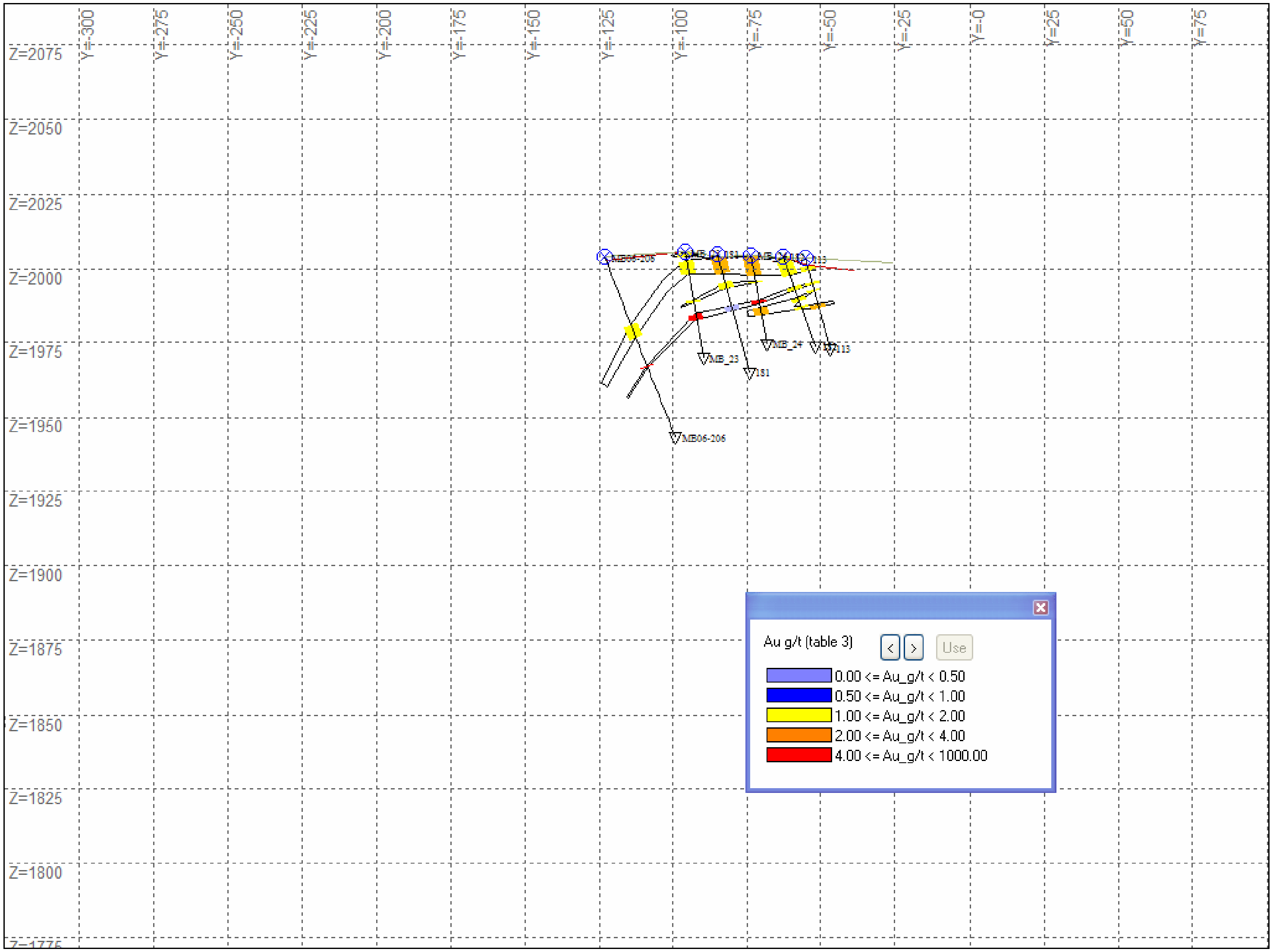


Figure 28: Section 1161mE

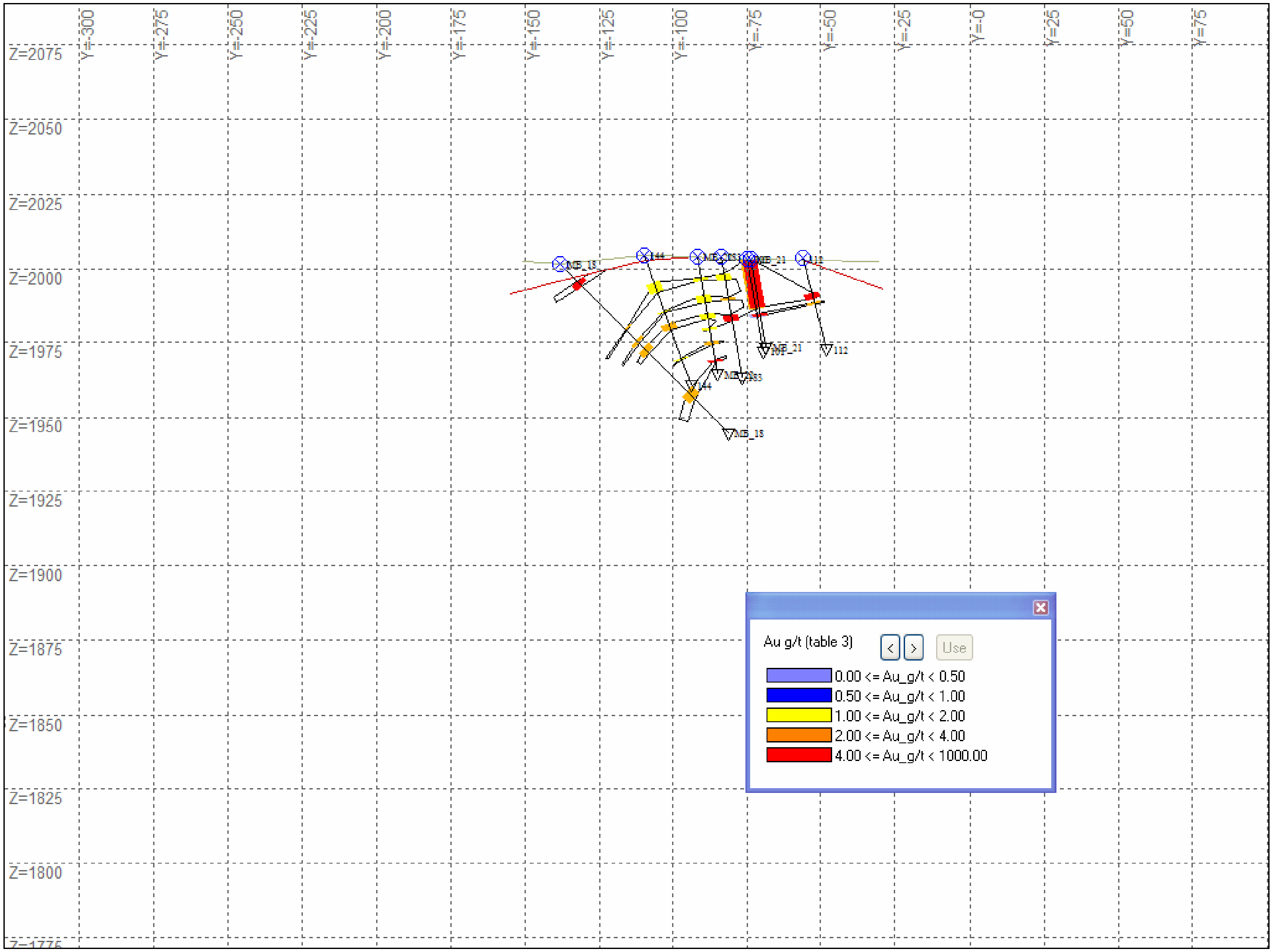


Figure 29: Section 1175mE

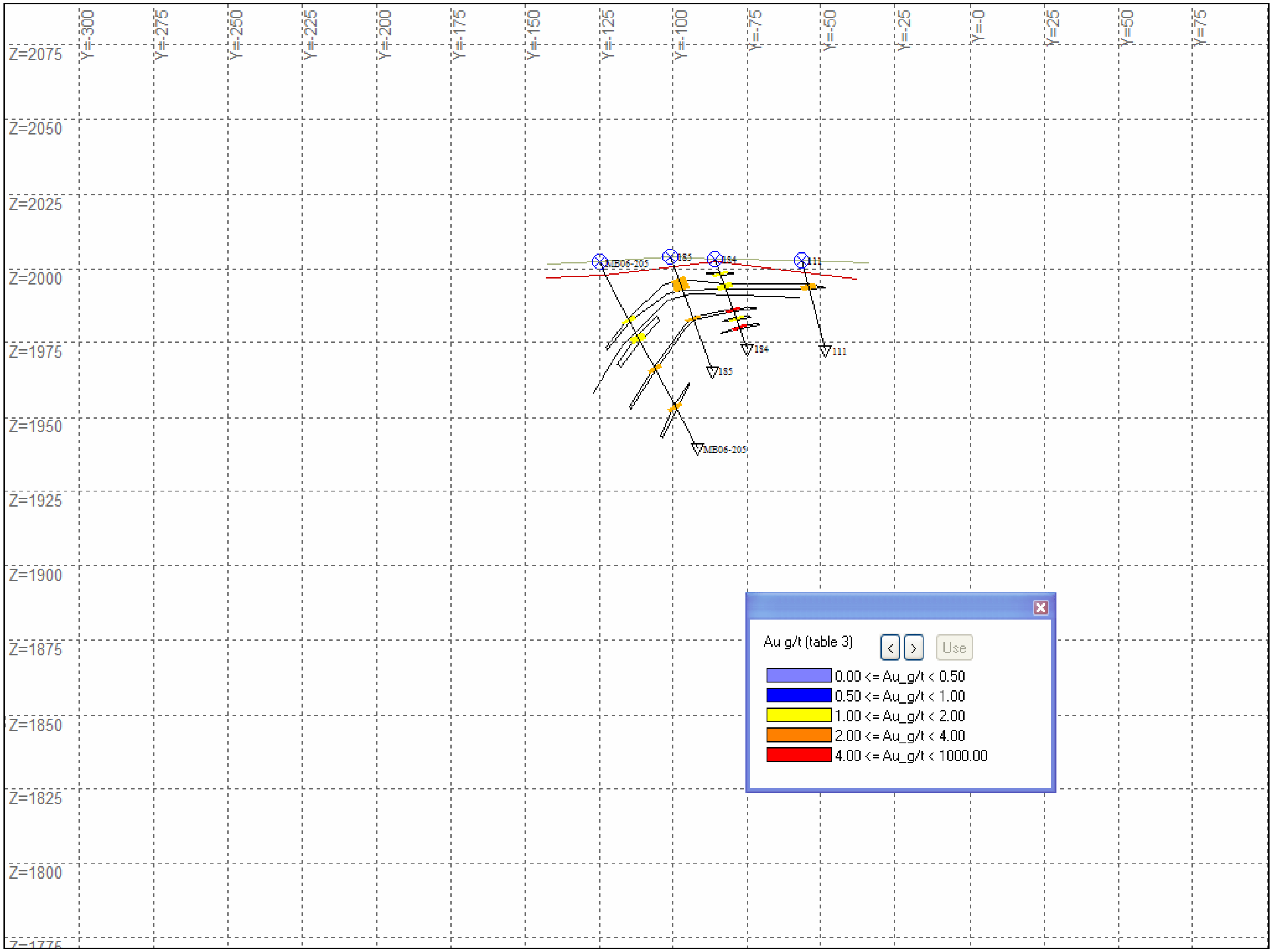


Figure 30: Section 1185mE

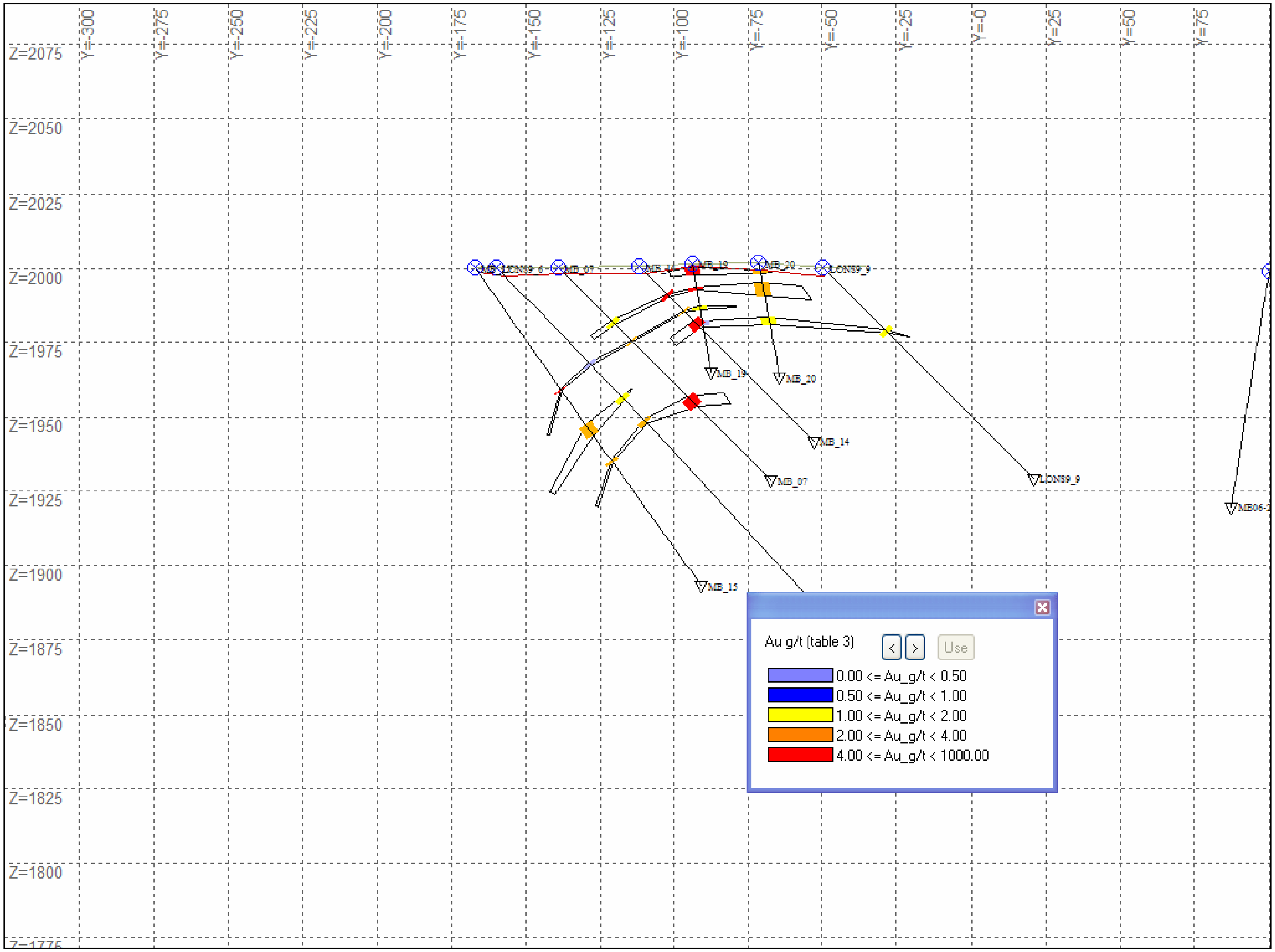


Figure 31: Section 1200mE

Appendix 5: Qualification certificate of Yann Camus, Eng.

Yann Camus, Eng.
6285 Chambord
Montréal (Québec) H2G 3B8
Email : ycamus@geostat.com

CERTIFICATE OF AUTHOR

1. I am working for:
   Systèmes Géostat International Inc.
   10, boul. de la Seigneurie Est, Suite 203,
   Blainville (Québec) J7C 3V5

2. I graduated with a geological engineer degree from the Polytechnique in 2000.

3. I am a member of the Ordre des ingénieurs du Québec.

4. I have worked as a geological engineer for 6 years with Geostat and did Mineral Resource estimation since then.

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 purpose of NI 43 -101.

6. I am responsible for the preparation of all the sections of the technical report titled "Technical Report - Resources Evaluation of May 2007 on the Barry I Project, Barry township, Metanor Resources Inc." (The "Technical Report"). I visited the Barry I property on May 14th, 2007 for the preparation of this report.

7. I am not aware of any material fact or material change with respect of 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 public 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 15th Day of May 2007

Appendix 6: Queen’s University Report on Barry Gold Recovery

Gold Recovery From Murgor Resources' Ore Using Flotation, Cyanidation and Gravity Separation

by

S. Kelebek,

Department of Mining Engineering
Queen's University, Kingston, ON, Canada

July 2006

Submitted to:

Murgor Resources

Executive Summary

Two composite samples of a gold-bearing ore deposit owned by Murgor Resources were submitted to Queen's mineral processing laboratories for investigations to characterize them for their hardness and find out their potential for gold recovery. One of the composites (designated as Murgor 1 in tabulated data) had a gold grade of about 5.18 g/tonne compared to the other (Murgor 2) at about 5.54 g/tonne.

The ore hardness determination using Bond's standard method indicated a ball mill grindability index, BW_i of 11 kWh/tonne for composite 2. The work index of composite 1 was determined to be 10.4 kWh/tonne using a comparative method. These work index values suggest that the Murgor ore as a whole can be classified as a "medium-hard" to "soft" ore.

Three methods of gold recovery/extraction process used were flotation, cyanidation and Knelson gravity separation. The particle size of feed samples in these tests ranged from a coarse P_{80} of 205 μm (micrometers) to 53 μm. A total of eight flotation tests were carried out with amyl and isopropyl xanthate as a collector combination and Dow Froth 250/MiBC as frother. The flotation of composite 1 at the P_{80} of 205 μm yielded a mass recovery of 3.66%, which corresponded to a gold recovery of 91.8% at an overall grade of 130.2 g/tonne. The gold recovery increased to 94.2% at a grade of 147.6 g/tonne and to 95.8% at 139.8 g/tonne, when the P_{80} values were reduced to 92 μm and further to 53 μm, respectively. Generally, the behaviour of composite 2 with slightly smaller recoveries was similar to that of composite 1. All flotation tests were carried out using soda ash (Na_2CO_3) as a pH regulator in order to promote bulk flotation of all sulphide minerals. In the last test, however, the soda ash was replaced by lime (more economical to use). Interestingly, this test was more selective since the mass recovery was somewhat reduced, but it did not indicate a significant adverse effect on gold recovery.

A total of six cyanidation tests were carried out with sodium cyanide solutions at 1 g/L and 2 g/L using composite 1. These were conventional leaching tests in bottles which were continued for 48 hours. The grind size of the samples was changed from 137 μm to 53 μm. These tests indicated gold extraction levels ranging from 94.2 to 97.5%. As the cyanide strength increased from 1 g/L to 2 g/L gold extraction showed an incremental increase, most strikingly at the beginning of leaching. However, the grind size had a bigger impact on gold leaching. Leaching kinetics data suggest that when the particle size is appropriate (e.g., P_{80} at about 53 μm) over 95% of gold can be leached into cyanide solution in less than 10 hours and in majority of cases, there is practically no difference between 24 h and 48 h data. The consumption of sodium cyanide depended on the solution strength used in leaching. In tests carried out at 1 g/L, it was in a range from 0.5 to 0.6 kg sodium cyanide per tonne of feed, while it increased to 1.34 to 1.49 kg sodium cyanide per tonne of ore when the leaching was conducted in a 2 g/L sodium cyanide solution. Lime consumption was much lower compared to the cyanide varying from 0.15 to 42 kg lime per tonne of ore.

A total of nine gravity separation tests were carried out using a laboratory Knelson concentrator device. Mass recoveries to concentrate products strongly depended on the fluidization water pressure used. However, the mass recoveries were significantly higher in Knelson separation tests than the flotation tests. These tests indicated that high gold recoveries can be obtained generally in low water pressures of 3 to 4 psi. For example, a relatively high gold recovery of 92% is possible only when the mass recovery is in 20-30% range, which translates into a concentrate gold grade of 15-20 g/tonne. At higher water pressures, the mass recovery is decreased significantly (to less than 10%), but this occurs at the expense of high losses of gold. For example, the gold recovery obtained at a water pressure of 9 psi was only about 75%. Higher recoveries of gold at lower water pressures, suggest simultaneous recovery of other minerals relatively high in specific gravity and their generally poor separation efficiency. In one test, magnetite which is known to exist in the samples, was removed using magnetic separation, but this resulted in relatively high losses of gold since significant amount of gold bearing particles were also recovered into magnetics concentrate, probably through a mechanical entrapment/carry over mechanism.

In conclusion, cyanidation as a chemical method provided the highest extraction of gold from Murgor samples tested. However, flotation as a physical separation method provides the second best option. Possible improvements of gravity separation would require a more detailed testing program using circuit simulation in a continuous mode. However, that the high grade and recoveries of gold can be obtained that are comparable to cyanidation or flotation cases, appears to be doubtful.

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1. Introduction

Two composites of a gold bearing ore from North Western Quebec (Murgor Resources) were submitted to the mineral processing laboratories of Queen's in spring of 2006. The composites were partially wet quarters of drill core samples supplied in plastic bags. One of the two composites was reported to have a higher gold grade (Murgor sample 2) compared to the other (Murgor sample 1).

According to an earlier study on samples of this ore (Lariviere, 1997), the mineralized veins consisted of quartz, ankerite and albite with a number of accessory minerals observed in the veins. These were namely biotite (+/- sericite), chlorite, pyrite, pyrrhotite, magnetite, ilmenite, chalcopyrite and gold. Gold was reported to be as free gold in gangue minerals within veins and altered wallrocks, as well as along microfractures in pyrite. Amenability studies have been undertaken to see the processibility of these samples. Methods tested involved cyanidation which is a well established method of gold leaching, flotation due to reported association of gold with sulphide minerals, primarily pyrite and gravity separation in view of the presence of "free gold" reported in the previous study. In addition to these recovery processes, grindability studies were also conducted to provide information about hardness of this ore.

These investigations have recently been completed. A detailed summary of the work carried out is presented in this report.

2. Objectives and scope

An important objective of this bench-scale testwork is to characterize the ore samples from Murgor Resources using representative charges prepared from two composites submitted. This characterization involved particle size distributions and P_{80} values under various grinding conditions and hardness, specifically ball mill Bond work index.

The ultimate objective of this investigation is to provide a comparative data on gold recovery processes using froth flotation, conventional cyanidation bottle tests, and gravity separation using Knelson gravity separation device.

3. Testing methods, reagents and procedures

3.1. Ore Samples & feed preparation:

The samples were supplied in nylon bags in separate pails in the form of two composites made out of blending of quarter drill cores. They were placed in a deep freezer for storage until charge preparation stage. The samples as received were somewhat wet. Therefore, they were spread on the laboratory bench for half a day.

Dry samples were crushed in a laboratory jaw crusher, followed by a gyratory crusher, and a roll crusher that was run in a closed circuit with a Sweco screen. For the Bond work index determinations, the screen size used was 6 mesh. For the flotation, it was 10 mesh. Screen undersize from each charge preparation stage was split into separate charges weighing about 1 kg using a rotary splitter with an adjustable speed vibratory feeder. All charges were kept in the deep freezers until they are required for testing.

3.2. Reagents:

The collector used in flotation tests was a mixture of two xanthates, namely PAX (i.e., potassium amyl xanthate) and IPX (sodium isopropyl xanthate). Frother was MIBC (methyl iso butyl carbinol). When needed, an appropriate amount of xanthate was dissolved in deionized water to prepare an aqueous solution at 0.1% wt strength. The frother was also used in its full strength in the form of droplets. Soda ash (Na_2CO_3) was used as a pH regulator. This is commonly used to promote bulk flotation of sulphide minerals. Since the current objective was to get the greatest gold recovery possible, and lime as the more common pH regulator has some known depressing effects for pyrite, soda ash was a good choice. As a comparison, soda ash was replaced by lime in the last test to provide data since lime is cheaper than soda ash. In another test, copper sulphate was used as an activator for flotation of sulphide minerals. Finally, a separate test involved use of sodium hydrosulphide (NaSH), which is known to promote flotation of oxidized sulphide minerals.

3.3. Testing equipment:

Batch tests were carried out using standard Denver laboratory rod mill and Denver flotation machine with a 2-L cell. Grinding media typically was of stainless and mild steel (at 50-50 mixture).

Cyanidation tests were carried out in 2 L glass bottles with their mouths open. The bottles were rotated with their slurry charges on rolls at about 60 rpm.

Gravity separation tests were carried out using a laboratory size (with a 3-inch cone) Knelson Concentrator.

Concentrate products from various tests were filtered using a vacuum filtration unit. The tailings were filtered using pressure filters. Filtered products were dried in standard laboratory drier(s) with automatic temperature control.

3.4. Procedures:

3.4.1. Bond's Standard Ball Mill Grindability

According to the Bond's standard ball mill test procedure (Bond, 1961), a sample of ore (finer than 6 mesh) is placed in a 1000 cc graduate cylinder and it is shaken for packing and the weight of 700 cc ore volume is taken as the feed for the grinding tests. Ground sample is sieved and mass split at mesh of grind (65 mesh) is recorded. In the current case, the feed sample (1245.45 g) is found to contain 14.27% wt -65 mesh. It is ground for a duration of 100 revolutions of the standard mill, after which the ground ore is sieved using 65 mesh screen to find out the mass split. The -65 mesh material obtained represents the mill product, with the rest being oversize (+65 mesh) representing the circulating load, which is to be combined with a portion of the new feed to equalize the feed weight for the next cycle (i.e., to bring it to 1245.5 g again). The amount of new feed to be added is the amount of -65 mesh displaced (503.27 g in the current case). Diagrammatic description of the ball mill work index determination according to the Bond method is shown in Figure 1.

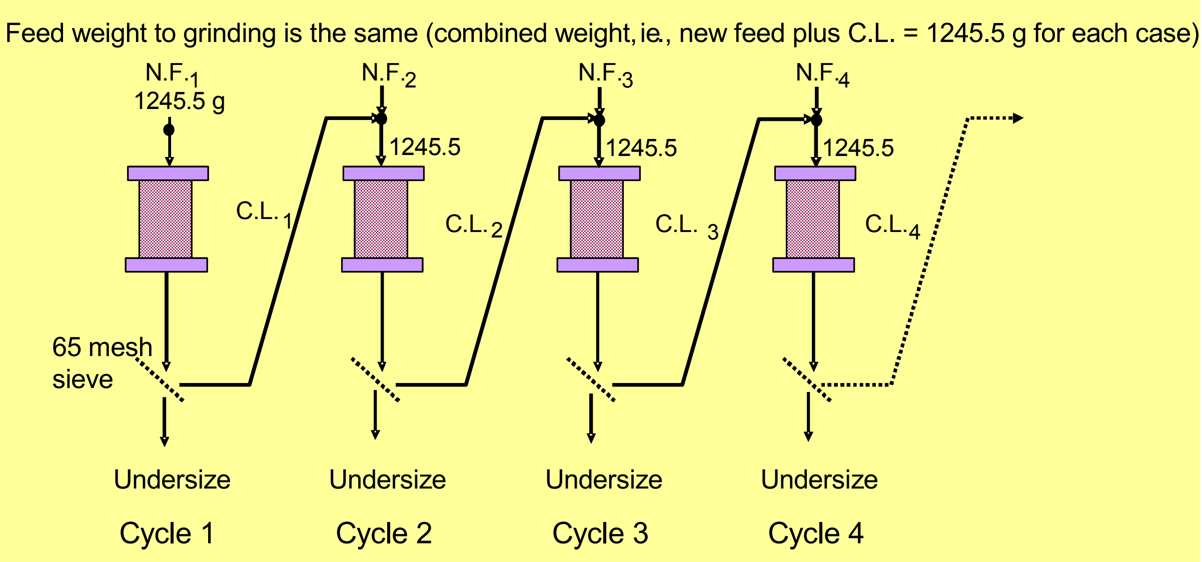


Figure 1. Schematic of sequential steps in Bond's Grindability Tests

The next cycle will start at a new rpm value based on the net g of -65 mesh material estimated. According to the Bond's standard procedure, circulating load is always set at 250% (which means that circulating ratio is 2.5). Since the total feed weight in each cycle is 1245.5 g (in this case), the product weight to maintain the target circulation ratio will be theoretically 1245.5/3.5 = 355.84 g. The division is by 3.5 instead of 2.5 because the total feed (combined feed) includes fresh feed. The flow diagram above is shown in Figure 2 in a closed circuit along with results of tabulated calculations given in section 4.1.1 that should be referred to for clarity.

3.4.2. Flotation

Charges to be used in flotation tests were subjected to grinding at various lengths of time to determine the "grinding curve". The pulp density in grinding was adjusted to

67% solids by adding appropriate amount of tap water. Soda ash addition to the grinding mill was generally about 300 g/tonne. Part of xanthates was also added in the grinding mill (see flotation reports for details). Pulp transfer time into flotation cell was about 5 minutes in each case.

Flotation feed slurry had an initial pH of 8.7 to 9.0. Pulp potential and pH levels were monitored during flotation tests. Pulp potential measurements were carried out using a portable unit (model 3000) from VWR Scientific, which was equipped with a bright gold electrode as part of a combination electrode with respect to a silver/silver chloride reference. Another unit from the model was used for pH measurements with a combination electrode and probe for temperature compensation. The performance of the redox probe was occasionally checked using a ferrous-ferric ammonium sulphate solution as a redox standard.

Flotation gas rate is based on the natural suction of air into pulp through the shaft of impeller set at 1200 rpm. Total flotation time was fixed at 7 minutes. For the majority of the flotation tests, a total of four concentrates were skimmed off at 0.5, 1.5, 2 and 3 minutes.

Solid products from all separation tests were dried and weighed for the construction of mass balance tables. Analysis of iron, copper, and lead was carried out using an Atomic Absorption Spectrophotometer (Perkin-Elmer, Model 2380). The amount of sulphur and total carbon was also determined using a sulphur analyzer from LECO (Carbon and Dual Range Sulphur Analyzer, Model SC-444DR). Both pieces of analytical equipment were interfaced with computers for direct output on the monitor. The methods involved the use of appropriate standards.

Analyses of gold in all products were carried out using the fire assay procedure developed for sulphur bearing ores (Yen, 2001). In cases where the amount of solids was insufficient for an assay ton (29.17 g), a concentrate sample has been prediluted with a gold-free granite sample, which was taken into account in mass balancing. Analyses of feed samples were done in triplicate.

3.4.3. Cyanidation

For extraction of gold, representative charges nominally at 1 kg were ground for required time, filtered in a pressure filter and split into two equal half charges. By means of a funnel, these moist samples were placed into cyanidation bottles, which were essentially 2.5 L old acid bottles made of glass and contacted with cyanide solutions of 1 g/L and 2 g/L at about 35% solids. Normally, the strength of the cyanide solution is lower in conventional cyanidation, e.g, 0.25 to 0.50 g/L. The concentration of cyanide solutions in the current work was kept deliberately high to see the full potential of the ore for cyanide leaching. Any additional work for optimization should include tests at lower cyanide concentrations and produce corresponding data on reagent consumption as well.

Bottle roll cyanidation tests were performed at three different particle sizes with nominal P_{80} values from 137 μm down to 53 μm. The bottles were placed on the rolls in tilted position with their mouth open during the entire cyanidation period. The rolling speed was about 60 rpm. Protective alkalinity of cyanidation slurries was ensured with addition of calcium hydroxide ($Ca(OH)_2$) introduced in powder form. The tests were continued for 48 hours with periodic sampling. Before sampling a bottle was removed from the rolls and placed on the bench for the settling of particles for about 5 minutes which was usually sufficient to withdraw a 10 ml supernatant from pregnant liquid phase via a 15 ml pipette. For each sampling period two solution samples were taken for analyses of cyanide and gold respectively. Measurements of pH and cyanide strength were carried out after each sampling. Additions of sodium cyanide as fresh make up to keep its initial level as constant and $Ca(OH)_2$ for pH control were based on standard titration work with para-dimethylaminobenzarhodamine and phenolphthalein as respective indicators. Further details that can provide information on procedures can be found in test reports in appendix.

Analysis of gold was carried out using Atomic Absorption Spectroscopy. In this case, the standard solutions used for calibration were in cyanide matrix having the same cyanide concentrations as those that were used for leaching. The concentrations of standard solution were selected so that all readings from pregnant solutions were within the linear range of calibration.

3.4.4. Knelson gravity concentration

The Knelson concentrator is a centrifugal bowl-type of device developed in B.C., Canada. It is essentially a high speed ribbed rotating cone with a drive unit. It utilizes the principles of hindered settling classification in a centrifugal force field. Heavy particles are forced out against the walls and trapped between the ribs while the lighter particles are carried away by the water flowing out. The cone is surrounded by a pressurized water jacket that forces water through holes in the cone to keep the bed of heavy particles fluidized.

Operating procedure for the Knelson concentrator recommend a water pressure of about 13 psi to be used for 3-inch model, which was taken into account in the current work. However, although the mass recovery was low at this pressure, the recovery of heavy sulphide particles appeared to be incomplete. Thus, lower water pressures were also tested to produce grade-recovery data. The device was initially operated with water until a stable flow pressure was obtained at a desirable level. The slurry of a sample was then fed into the unit under steady operating conditions. The material that was retained in the cone is the concentrate. The material flowing out is the tails. However, these tails were fed again into the device in order to increase the mass recovery simulating a scavenging operation. The scavenger tails were taken out as the final tails for filtration while the combined concentrate was fed into the unit again to obtain a cleaner concentrate. The tails from this step was considered as a middlings.

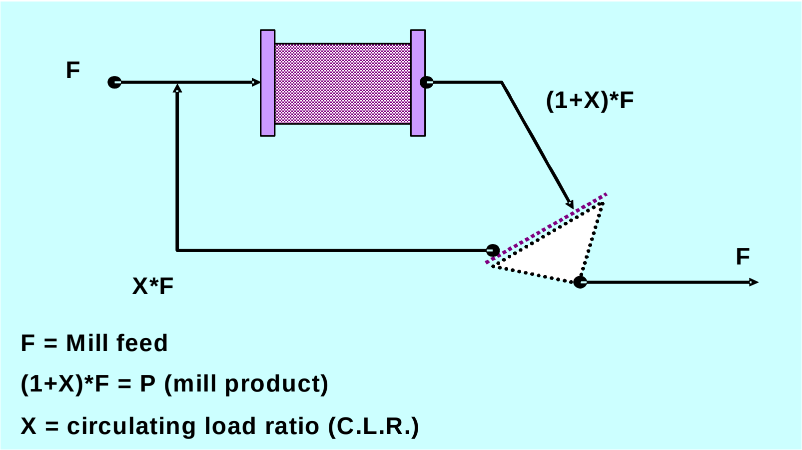
4. Results and Discussion

4.1. Grinding tests

*4.1.1. **Work Index***

As explained earlier, the method of work index determination starts with grinding the ore occupying 700 mL volume in a laboratory Bond ball mill and continues with a sequence of cycles which are linked through circulating loads. Size distributions of the feed and grinding product are needed (see appendix for tabulated data) to derive basic parameters in estimation of a work index, which, by definition, is the total energy required to reduce a particle from an infinite size to 100 μm. This is used as a measure of hardness and varies from low values such as 7 kWh/tonne for clay to high values such as 14 kWh/tonne for quartz. However, some uncommon materials such as emery is known to have a very high work index value at 64 kWh/tonne (Kelly Spottiswood, 1982). The ball mill Bond test simulates the closed circuits shown in Figure 2. The circulating load is targeted at 250% as the standard value adopted by

CONTINUOUS MODE AND MASS BALANCE:



X = 2.5 (standard C.L. ratio), Then
P/F = (1+X) = 2.5+1 = 3.5
or (F/3.5)= P which is the ideal mill product expected at 250% circulating load.

For the sample used = 1245.45 / 3.5 = 355.84 g

Figure 2. Bond's ball mill grindability test in closed circuit

Fred Bond. Results of the grindability tests obtained for composite #2 are shown in the following table. These tests were continued for five cycles to reach equilibrium. In tests with other ores, the achievement of equilibrium may take up to seven cycles. Particle size distributions of the feed and the product (i.e., the average of last two cycles) are shown in Figure 3. The 80% passing size for the feed (F_{80}, 1635 μm), the product (P_{80}, 180 μm) and the ball mill grindability value (Gbp) were used along with the mesh of grind used (65 mesh) in Bond's equation shown below to get a work index value of 11 kWh/tonne.

Table 1. Detailed results from Bond's ball mill tests for work index determination

Sample: Murgor #2 (-6 mesh) (14.52% -65 mesh in original feed by sieving = 180.84 g
Ideal potential product: 355.84

<u>Next number of revolutions at 250% C.L.:</u>

355.84 - 73.07 = 282.77 g (production of net amount of -65 mesh)
282.77/3.22 = 88

Cycle No.	New Feed (g)	Number of Rev's	Grams of minus -65 mesh				
			Mill Prdct	Mill Feed	Net Prdct	Net g/rev.	C.L.(g)
1	1245.45	100	503.27	180.84	322.43	3.22	742.18
2	503.27	88	344.10	73.07	271.03	3.09	901.35
3	344.10	99	335.84	49.96	285.88	2.89	909.61
4	335.84	106	355.27	48.76	306.51	2.88	890.18
5	355.27	106	355.84	51.59	304.25	2.88	889.61
6	355.84						

Mesh of grind (P_1): 65 mesh
Gbp = 2.88 g/rev
F_{80} = 1635 μm (mean)
P_{80} = 180 μm

$$Wi = 44.5 / (P_1)^{0.23} * (Gbp)^{0.82} * ((10/P_{80}^{0.5}) - (10/F_{80}^{0.5}))$$

Wi = 11.0 kWh/tonne

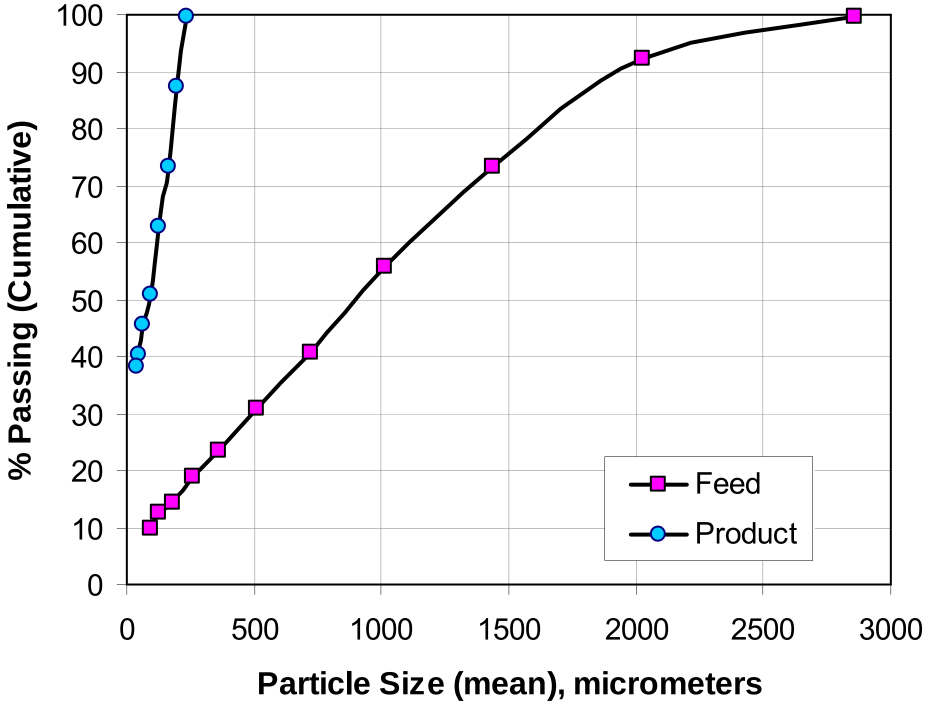


Figure 3. Particle size distributions for the feed and product in Bond's test

Work index determination for the other composite was done according to comparative method due chiefly to lack of samples for standard Bond method. The comparative method was proposed by Bruce and Campbell (1976) at CANMET. This method is applicable to the current ore samples from Murgor Resources. In general, if the work index of one sample is known by determination based on standard Bond method, it can be used as a reference to estimate the work index value of another ore sample similar in nature. This requires grinding of the reference ore and the test sample with unknown Wi value to be ground under identical conditions for exactly the same period so that they receive the same amount of energy for size reduction. The energy for size reduction is related to work index and characteristics of feed and product (i.e., K80; 80 % passing size values) in Bond's third law of communition by

$$W = Wi * \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}}\right)$$

The feed particle size distributions of the representative charges (-10 mesh) for two composites that were used in the current testwork are given in Figure 4. As can be seen, the two samples have almost identical size distributions with a common K80 of 1240 µm. Since these two samples originated from similar quarters of drill core samples and have experienced exactly the same size reduction steps (Jaw, Gyratory, Roll crushers closed with Sweco vibrating screens), one can assume that they received the same energy input for same product size distributions. Thus, their work index value for fine crushing would be expected to be about the same.

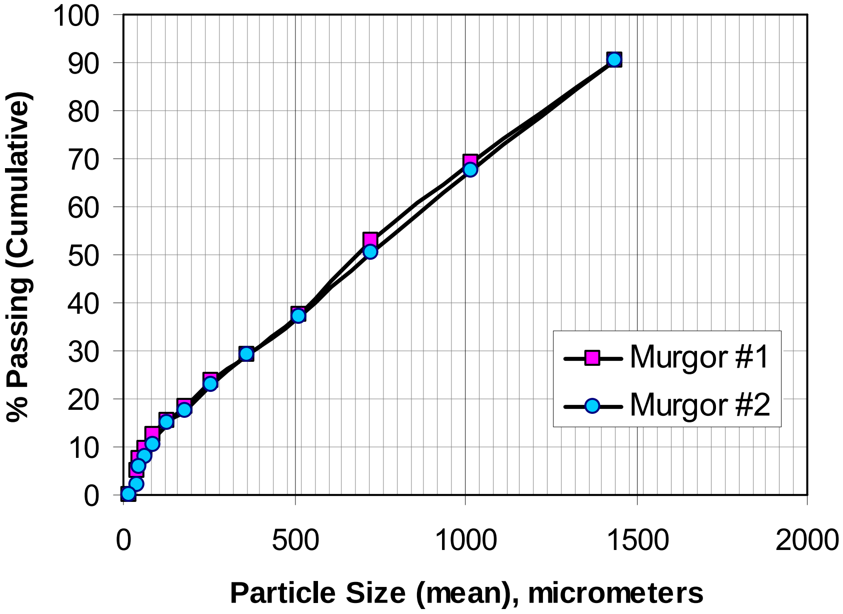


Figure 4. Particle size distribution of representative ore charges (-10 mesh).

The particle size distributions for the products from the representative charges are shown in Figure 5. For this purpose three different grinding periods were used for each composite. The grinding conditions were exactly the same in terms of pulp density, pH, type of grinding mill and grinding media, etc. As can be noted from the results, for the size reduction to finer grinding size, some consistent differences emerge. Now, K80 values for composite 1 (i.e., Murgor 1) are lower than composite 1 for all three grinding cases.

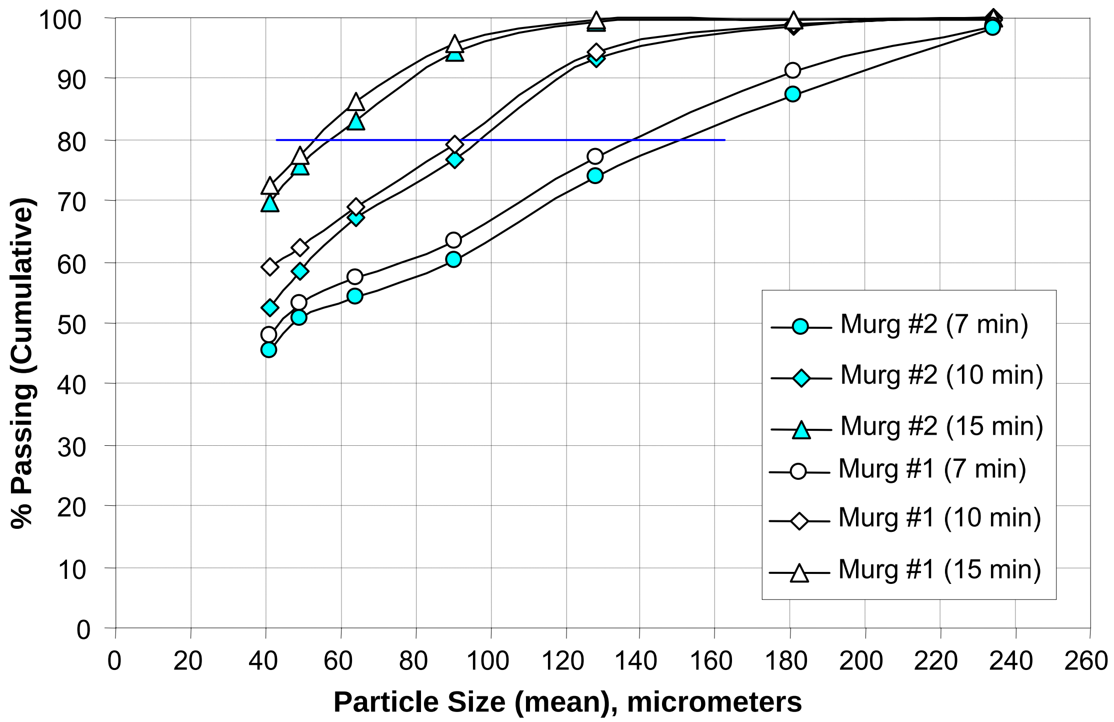


Figure 5. Particle size distribution of various products from grinding of the -10 mesh samples.

Since the grinding conditions were the same, energy input for the size reduction of these two composites (W) is the same. Thus, the following can be written for two samples:

$$W_{iM2} * \left(\frac{10}{\sqrt{P_{80M2}}} - \frac{10}{\sqrt{F_{80M2}}} \right) = W_{iM1} * \left(\frac{10}{\sqrt{P_{80M1}}} - \frac{10}{\sqrt{F_{80M1}}} \right)$$

And Wi value for composite 1 as the unknown value can be obtained from:

$$W_{iM1} = \left(\frac{\left(\frac{10}{\sqrt{P_{80M2}}} - \frac{10}{\sqrt{F_{80M2}}} \right)}{\left(\frac{10}{\sqrt{P_{80M1}}} - \frac{10}{\sqrt{F_{80M1}}} \right)} \right) * W_{iM2}$$

F_{80} and P_{80} values of composites 1 and 2 and work index for composite 2 and the calculated values of work index for composite 1 are shown in Table 2.

Table 2. Results of work index determination for composite 1.

Comparative data for samples	Murgor 2	Murgor 1	Murgor 2	Murgor 1	Murgor 2	Murgor 1
F_{80} (μm)	1240	1240	1240	1240	1240	1240
P_{80} (μm)	150	137	98	92	57	53
$10\,(F_{80}^{0.5}-P_{80}^{0.5})/(F_{80}^{0.5}{*}P_{80}^{0.5})$	0.53	0.57	0.73	0.76	1.04	1.09
W_{iM2} = 11 (kWh / tonne)	11.0		11.0		11.0	
W (kWh / tonne)	5.86		7.99		11.45	
W_{iM1} (kWh / tonne)		10.27		10.53		10.50
W_{iM1} (Average) (kWh / tonne)	**10.4**					

The work index values according to grinding periods for 7, 10 and 15 minutes and corresponding P_{80} values of 137, 92 and 53 μm, are 10.27, 10.53 and 10.50 kWh/tonne. Thus, the average of 10.4 kWh/tonne is adopted for the work index of composite 1 and its hardness is somewhat smaller than that of composite 2.

4.1.2. ***Feed preparation for separation test***

One of the main objectives of the current testwork was to carry out gold recovery/extraction tests with samples at a minimum of three different grind sizes. In order to produce these grind sizes, the laboratory grinding mill was calibrated. Figure 6 shows “grinding curve” for composite 1. The P_{80} values are plotted as a function of three grinding periods for the feed and tails that were obtained from the flotation tests. As can be noted, the data for the feed and tailing are superimposed. There is practically no difference in size characteristics. Main reason for this behaviour is the fact that mass recovery to the concentrates in flotation tests was generally very low, as it will be discussed in the next section. Thus, size distribution was not affected significantly. The same behaviour is apparent also from the tails of cyanidation tests. Results involving size distribution of cyanidation tails are shown in Figure 7. The same explanation is applicable here. In general, the soluble fraction of the ore in

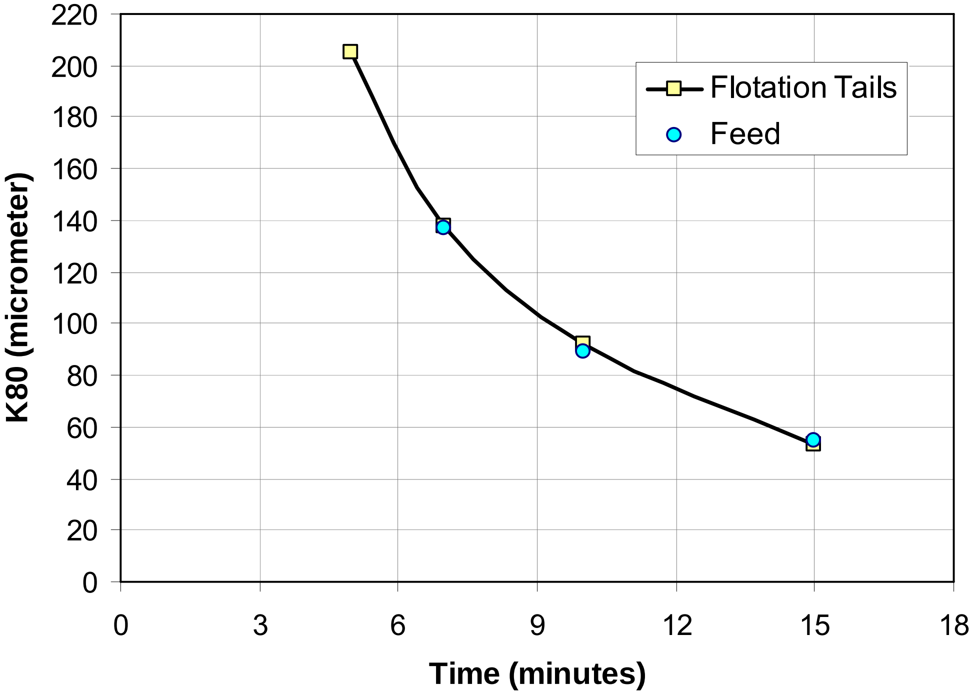


Figure 6. Grinding curve for composite 1

cyanidation is too low to have an influence on the size distribution. However, due to some attrition experienced by particles in agitated slurry, the weight fraction of the finest fraction is somewhat higher in tailings. This effect is visible in Figure 7. This figure also gives an indication on reproducibility of results.

4.2. Flotation tests

4.2.1. **Flotation Kinetics**

A total of eight flotation tests were carried out with amyl and isopropyl xanthates as a collector combination and Dow Froth 250 as a frother. The dosage of each collector was 60 g/tonne and the frother was 50 g/tonne. These dosages were kept constant for all tests. Test reports and detailed mass balance tables are appended for both composites. Flotation kinetics for composite 1 obtained at three grind sizes are

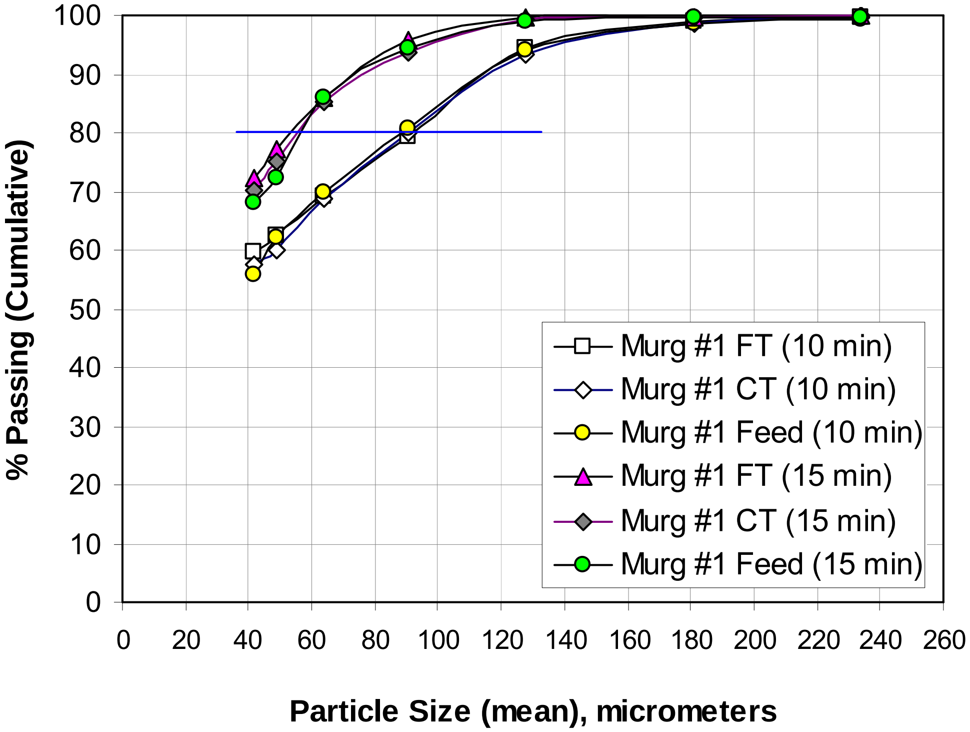


Figure 7. A comparison of size distribution of feed and tailings from separation tests

shown along with corresponding mass recoveries in Figure 8. The flotation at a P_{80} of 205 μm yielded a mass recovery of 3.66%, which corresponded to a gold recovery of 91.8%. The gold recoveries increased to 94.2% and 95.8%, when the P_{80} values were reduced to 92 μm and further to 53 μm, respectively. Generally, the flotation kinetics of composite 2 was similar to that of composite 1 (see balances).

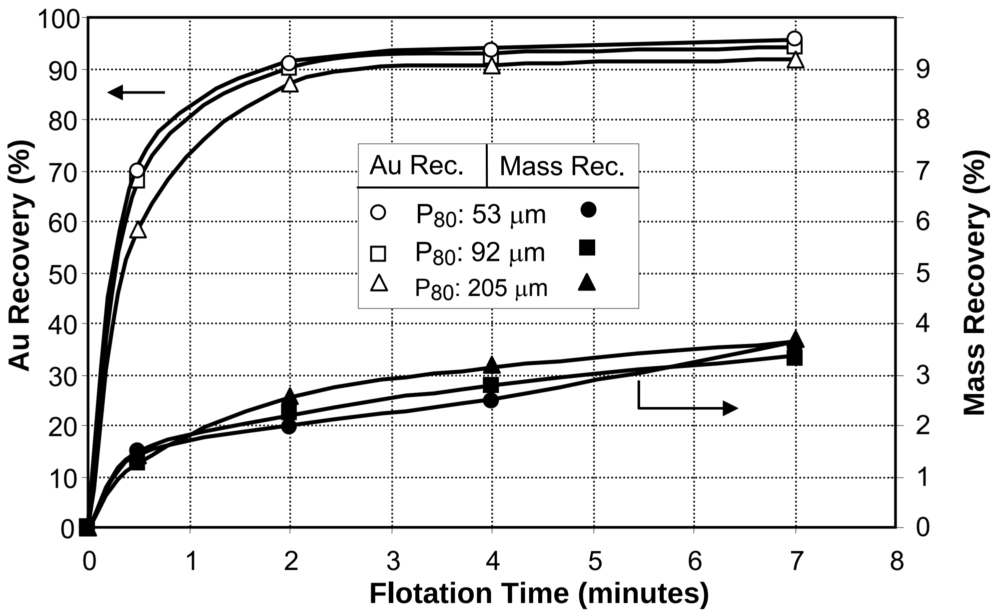


Figure 8. Flotation kinetics of composite 1 along with corresponding mass recoveries at various grind sizes

4.2.2. **Effect of Lime**

All flotation tests were carried out using soda ash (Na_2CO_3) as a pH regulator in order to promote recovery of all sulphide minerals. It was assumed that some incremental gold recoveries may be associated with recovery of all sulphides. Soda ash is used in flotation of platinum bearing low grade sulphides in South African operations although it is relatively more expensive compared to lime. In the last test of the current program, however, the soda ash was replaced by lime to provide data to see whether or not more favourable economics with lime can be taken advantage of. The results of these tests are compared in Figure 9 for composite 1 at the same grind size (P_{80} of 53 µm). The use of lime reduced the recovery of gold initially. However, with increased stage addition of reagents this adverse effect was eliminated and the recoveries at subsequent stages were practically the same. It should also be noted that the test with lime was more

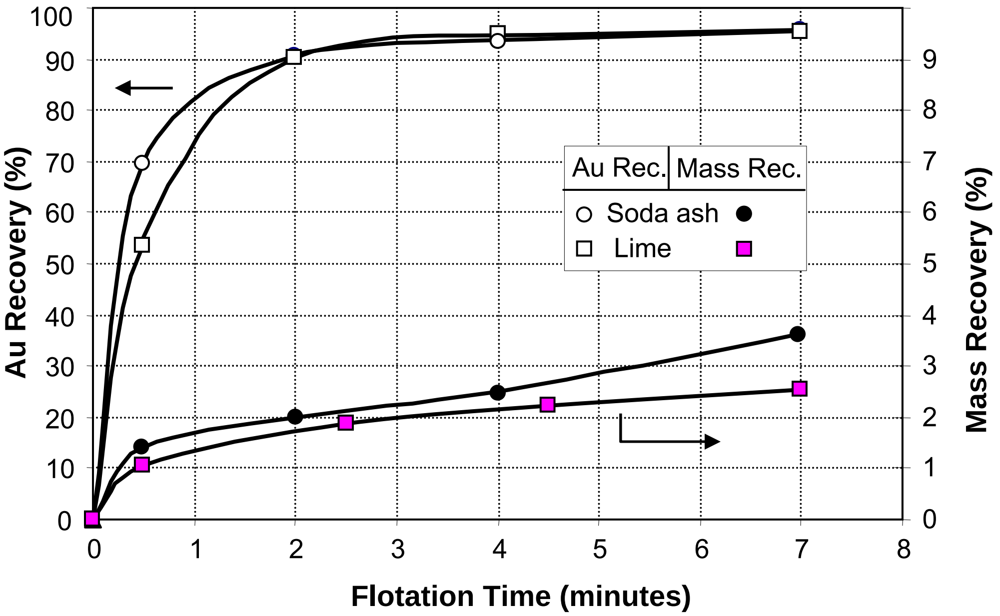


Figure 9. A comparison of flotation kinetics of composite 1 at a P_{80} of 53 μm with soda ash and lime.

selective since its mass recovery was reduced significantly. This somewhat surprising result suggests that not all the sulphides are associated with gold to the same degree. It might well be that part of pyrite and perhaps most of pyrrhotite that also exists in the samples were not significantly associated with gold. Therefore, their depression in the presence of lime did not result in significant gold losses. Recovery of gold in the lime system compared to soda ash system was lower only by 0.4%. This point is speculative at this time, but further investigations along with mineralogical analysis on flotation products can provide a more detailed understanding on this matter.

4.2.3. **Grade-recovery performance**

*4.2.3. 1. **The impact of grind size***

Figure 10 shows a comparison of grade and recoveries of gold obtained from composite 1 with respect to grind size. As in the case of base metals, the gold grade is significantly improved with particle size. The finer the particle size of the flotation feed, the greater the degree of liberation of sulphides and gold from the associated

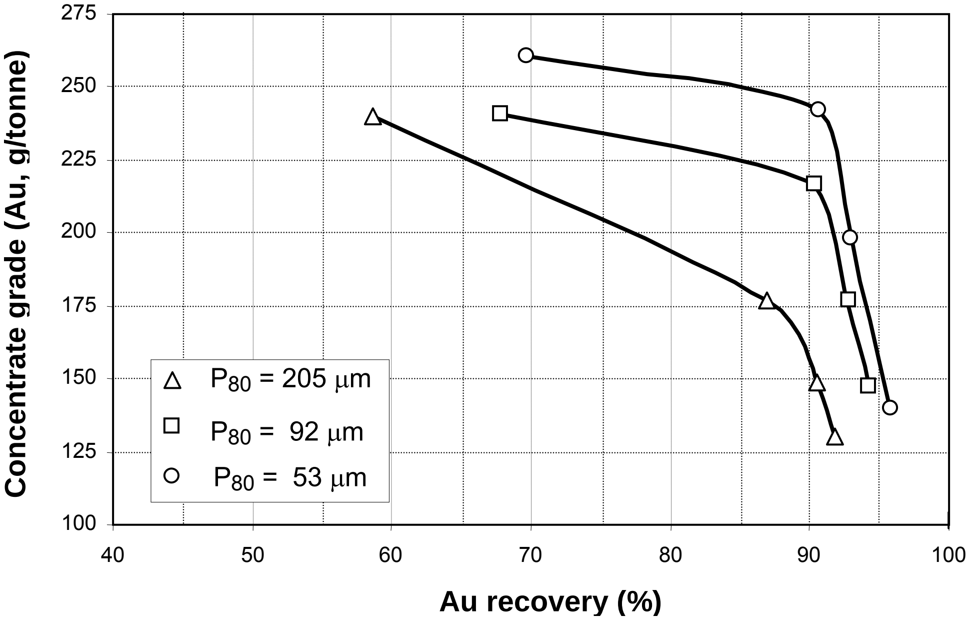


Figure 10. Grade-recovery behaviour of gold from composite 1 under various grind sizes.

mineralogical matrix, and the greater the gold recovery. It should be noted that incrementally there is a greater jump in the grade-recovery when the particle size is reduced from a P_{80} of 205 μm to 92 μm. With a further decrease in P_{80} value, the overall recovery shows less increase, possibly due to liberation problems. It seems that achievement of a 95% gold recovery requires the particle size to be between a P_{80} of 92 and 53 μm. Since the mass recovery in flotation was so low, in the particle size range investigated, the combined flotation concentrates have a gold grade of over 125 g/tonne, regardless of the particle size range used.

4.2.3. 2. ***Soda ash vs. lime***

It was noted earlier that replacement of soda ash by lime caused a significant reduction in the mass recovery. Although the same pH level was targeted for this particular comparison, the pH in the case of soda ash was 9.37 while it was 9.32 in the case of lime. As a consequence of a lower mass recovery, the gold grade of the concentrate also increased significantly. This can be noted from Figure 11, which shows cumulative gold grade as a function of cumulative recovery. The gold grade in the case of lime increased to about 193.5 g/tonne from a value of about 140 g/tonne, which was obtained with soda ash as the pH regulator. This increase in gold grade of the concentrate came at the expense of 0.4% loss in gold recovery.

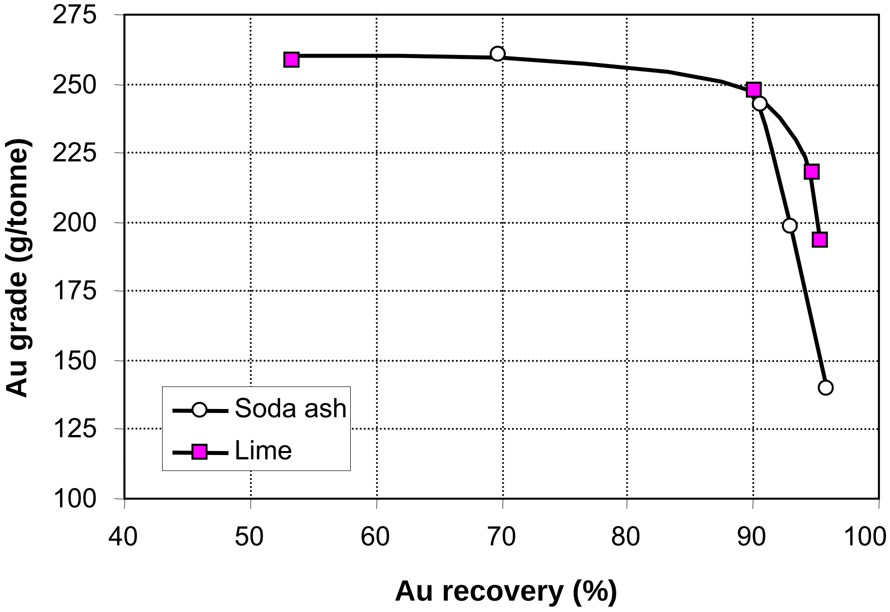


Figure 11. Grade-recovery behaviour of gold from composite 1 with lime and soda ash.

4.2.3.3. ***Effect of other relevant reagents***

In an effort to maximize gold recovery additional reagents were used in an exploratory manner. These reagents were sodium hydrosulphide (NaSH), copper sulphate (Cu_2SO_4) and fuel oil which were tested by addition after skimming off the concentrate 2. The feed sample for each case was composite 2. These reagents were tested at a P_{80} value of 57 μm. As it can be noted from Figure 12, among these reagents only copper sulphate seemed to have had some positive impact. Copper sulphate is well established as an activator for slow floating sulphide minerals. Its use was justified for this effect. NaSH is known to combat adverse effect of oxidation on sulphide minerals. Thus, its use is also justifiable on this basis.

Fuel oil is known to aid flotation of sulphides and hydrophobic particles. Despite the desirable objective and justification of their use, the latter two didn't seem to have caused any improvement in recovery of gold. In fact, the recovery was somewhat lower. However, these single tests, whether indicating an apparent improvement (e.g., copper sulphate) or an adverse effect (e.g., the other two) are not enough to make a concrete statement. There is always some scatter in flotation data. Therefore, additional tests can be carried out in efforts to maximize gold recovery.

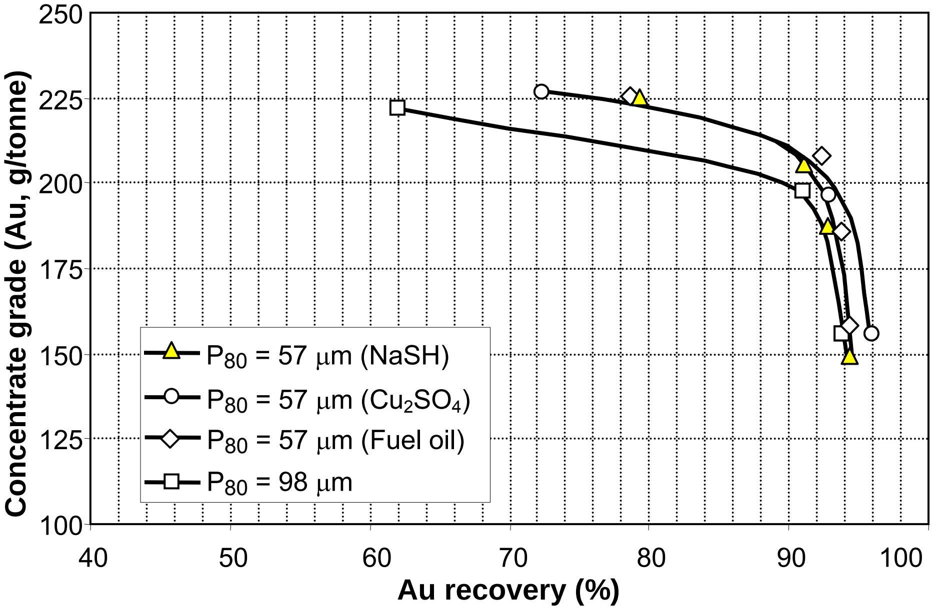


Figure 12. Grade-recovery behaviour of gold from composite 2 under various conditions.

4.2.4. **Recovery of other components in the ore**

Flotation products in some tests were analyzed for a number of elements in addition to gold. These involved sulphur, carbon, iron, copper and lead. The concentrations of the base metals are generally too low to be of any economical interest. Detailed behaviour of these elements can be examined from the mass balance tables appended. Here, a few points will be made regarding sulphur, iron and copper.

Figure 13 shows flotation recovery of gold as a function of sulphur recovery from flotation of composite 1. The recoveries of about 70% and lower are incremental stage recoveries. The total recoveries were also included in this comparison. As can be noted, there is a definite correlation between the recoveries of these two, despite some scatter. This is a good indication of association of gold with sulphides, the most predominant of which is pyrite, which is known as a common iron sulphide gangue mineral.

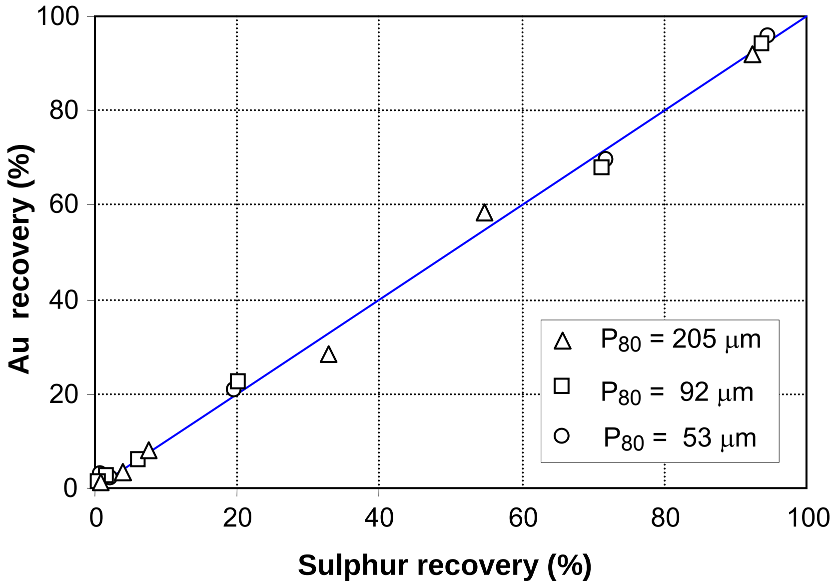


Figure 13. Gold recovery versus sulphur recovery in flotation concentrates of composite 1

The behaviour of iron and copper were also examined similarly. Figure 14 shows a plot of gold recoveries as a function of iron and copper recoveries. As can be seen, the trends are entirely different and there is no 1:1 correlation that was so apparent in the case of sulphur. Iron recoveries increase with gold recovery almost linearly. However, the level of iron recoveries is very low. This suggests presence of additional iron in these samples, i.e., iron from other sources than iron sulphides such as pyrite and pyrrhotite. The mineralogical source of this iron is such that it does not respond to flotation collectors used for gold and sulphide minerals. Two possible candidates are magnetite and ankerite, which were mentioned in the introduction section. Since these are essentially hydrophilic minerals they report to flotation tails when the collector is of xanthate type. Copper seemed to exhibit some correlation initially, but deviates from this behaviour at higher recovery levels. This suggests two possibilities. One is that part of copper bearing minerals is of non-sulphide origin or it is interlocked. Regardless of the nature of copper-bearing mineral(s), lack of mineralogical association with gold is apparent.

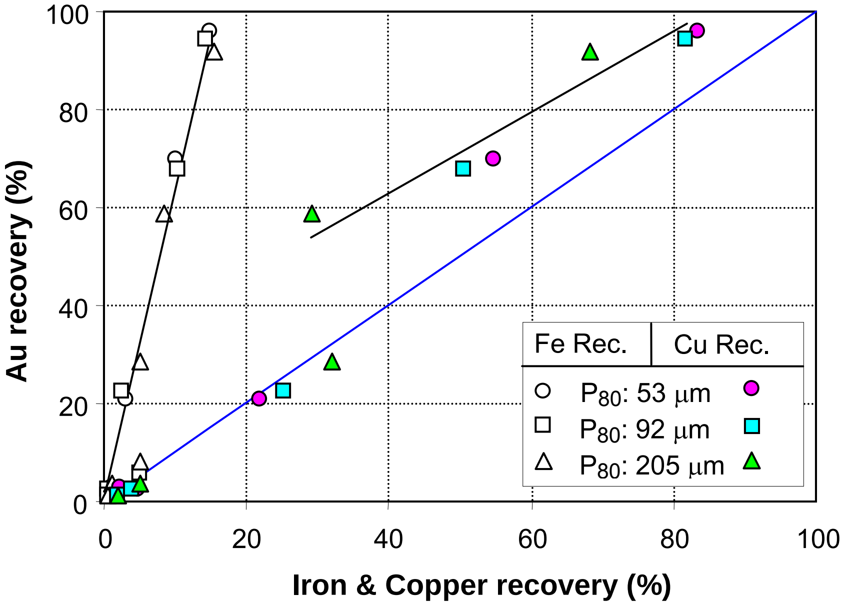


Figure 14. Gold recovery versus recovery of iron and copper in flotation concentrates of composite 1

4.3. Cyanidation tests

4.3.1. **Detailed results and estimation of gold extraction**

A total of six cyanidation tests were conducted. Due to sample availability, all of these tests were carried out only with composite 1. Sodium cyanide (technical grade) was used at two strengths, namely 1 g/L and 2 g/L. Particle size was varied from a P_{80} of 137 μm to 53 μm. Results from a typical cyanide leaching test are shown in Table 3, as an example. All other mass balances and cyanidation data are appended. Each table shows a summary of test conditions in the top section which identifies the sample, P_{80} value, amount of solids used in a test, slurry density and sodium cyanide strength used. The results show time of sampling, concentration of gold in pregnant sample solutions, their volumes and mass balance details along with steps on how the extraction of gold was estimated. In the example shown, gold extraction was calculated to be 94.2% at a P_{80} of 137 μm with a sodium cyanide concentration of 1 g/L. This recovery level is based on analysis of liquid and solid phases obtained during the test. The table also shows calculated heads which varied from 5.10 g/tonne to 5.29 g/tonne and they are in reasonable agreement with the actual head for composite 1 (i.e., 5.18 g/tonne).

Table 3. Results of cyanidation and data analysis (Test 1A)

Test 1A		
Sample Murgor 1		
Particle Size, P_{80}	137	μm
Initial solids (g)	503	g
Initial Vol of CN Solution	1010	ml
Density (% solids)	33	% wt.
Cyanide Concentration	1	g/L

		Au in sampling solution					
Time (h)	Au (ppm)	Sample Vol. (ml)	Sample Vol cum. (ml)	Au (mg)	Au (mg) Cum.	Au (mg) ext.	% Extraction
0	0	0	0	0	0	0	0
1	0.63	20	20	0.013	0.013	0.64	24.1
2	1.20	20	40	0.024	0.037	1.20	45.4
3	1.56	20	60	0.031	0.068	1.55	58.7
5	2.07	20	80	0.041	0.109	2.03	77.0
8	2.48	20	100	0.050	0.159	2.42	91.4
18	2.53	20	120	0.051	0.209	2.46	93.2
24	2.55	20	140	0.051	0.260	2.48	93.8
48	2.56	20	160	0.051	0.312	2.49	94.2

Total sample for anayses (ml)	160	0.31
Preg. Soln at the end	850	2.18
Tails at the end (g)	498	
Tails assay (g/tonne)	0.31	0.15
Au in reconstituted feed (mg)		2.64
Calculated head (g/tonne)		5.25
Actual head (g/tonne)		5.18

4.3.2. **Cyanide leaching kinetics**

Kinetics curves were developed for cyanidation in order to gather detailed information rate of gold extraction. The leaching tests were continued for 48 hours. The sampling of pregnant solution was frequent enough to have a good resolution on initial extraction kinetics of gold. The initial sampling time were 1, 2, 3, 5 and 8 hours. Figure 15 shows a comparison of gold extraction kinetics at two cyanide strengths. As the cyanide strength increased from 1 g/L to 2 g/L gold extraction showed an incremental increase from 94.2% to 94.8%. Another striking difference was observed during the initial period of leaching. Like the overall gold extraction, the kinetics of extraction at the higher sodium cyanide concentration was definitely better. These trends on gold extraction and kinetics continued for other samples tested at finer particle sizes.

Figure 16 shows leaching kinetics for the sample ground to a P_{80} of 92 μm. As can be noted from this figure and the related mass balance table (see appendix), the overall gold recovery is higher at 96% and 96.7% when the sodium cyanide concentration is at 1 g/L and 2 g/L, respectively. Again, the higher concentration of sodium cyanide resulted in a higher extraction of gold as well as a more favourable

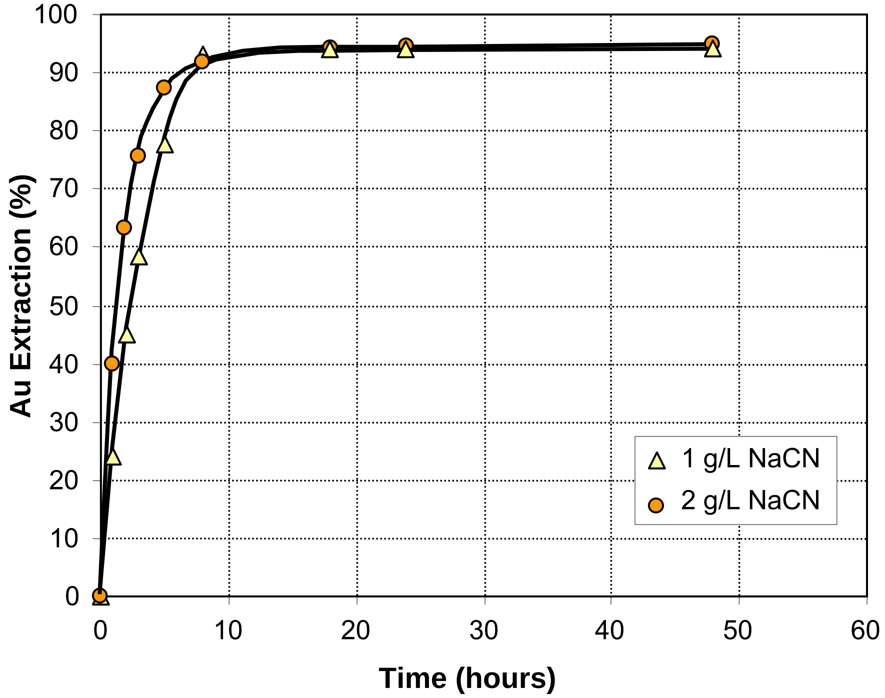


Figure 15. Gold extraction kinetics from composite 1 in cyanide solutions at two concentrations and at P_{80} of 137 micrometers

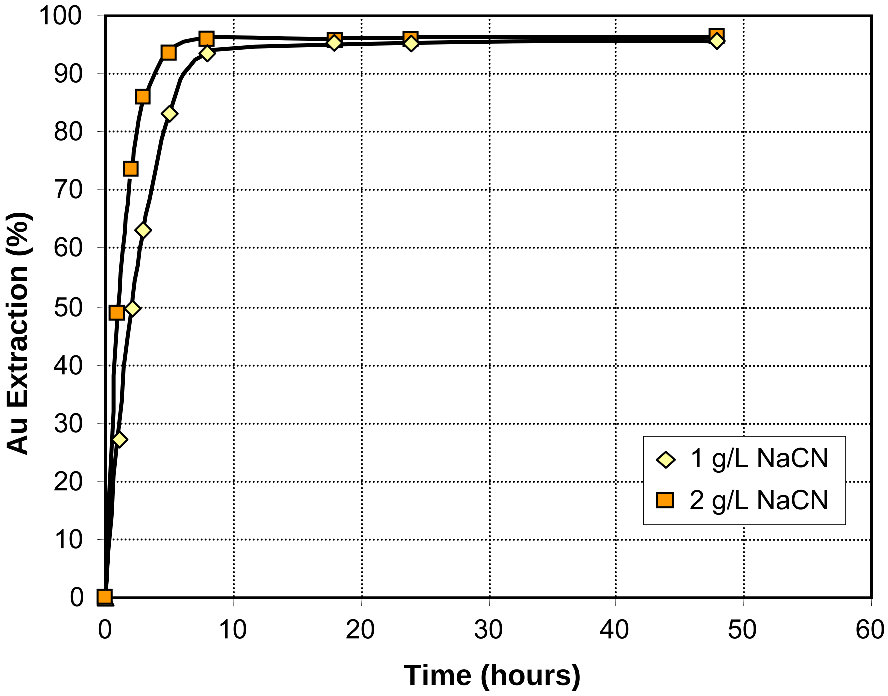


Figure 16. Gold extraction kinetics from composite 1 in cyanide solutions at two concentrations and at P_{80} of 92 micrometers

extraction kinetics. An incremental difference of 0.7% for gold leaching is consistent with 0.6 % in the previous case. However, the grind size had a bigger impact on gold leaching. For example, the gold recovery increased from 94.3% to 96% accounting for a difference of 1.7% that can be attributed to the decrease in P_{80} from 137 μm to 92 μm.

Figure 17 shows the leaching data for the finest particle size tested (i.e., P_{80} = 53 μm). Now, the overall gold extraction reaches 97.2% at the lower sodium cyanide concentration used and it increases to 97.6% at the higher sodium cyanide concentration at 2 g/L. Leaching kinetics data suggest that when the particle size is appropriate (e.g., P_{80} at about 53 μm) over 95% of gold can be leached into cyanide solution in less than 10 hours and there is practically no difference between 24 h and 48 h data.

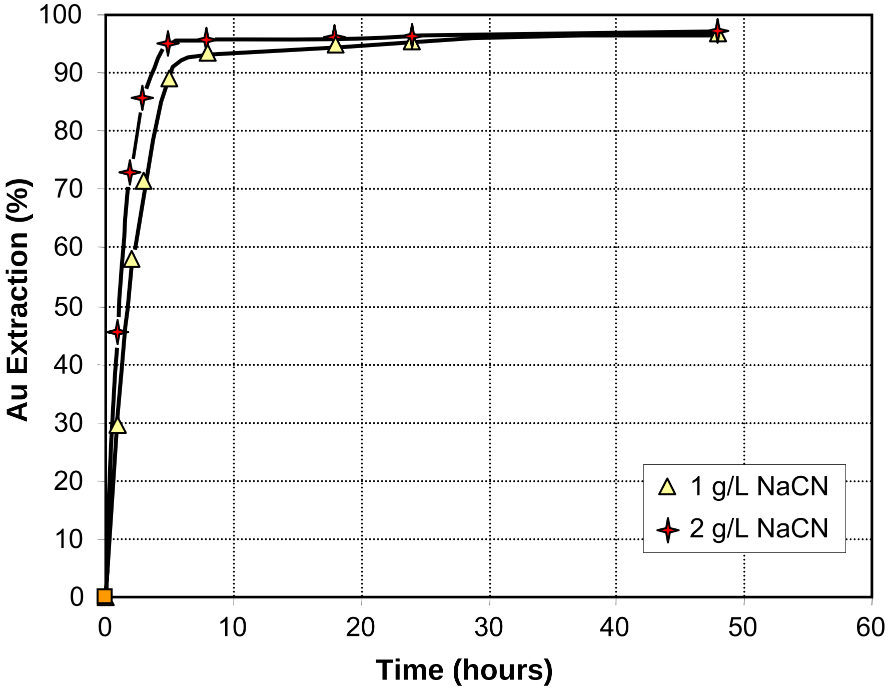


Figure 17. Gold extraction kinetics from composite 1 in cyanide solutions at two concentrations and at P_{80} of 53 micrometers.

Figure 18 shows a comparison of gold recoveries obtained by flotation and gold extraction by cyanide leaching as a function of particle size. Leach 1 and leach 2 refer to extraction in 1 g/L and 2 g/L sodium cyanide solutions respectively. Dependence of gold recovery/extraction on P80 appears to be almost linear and the superiority of chemical extraction of gold is clear.

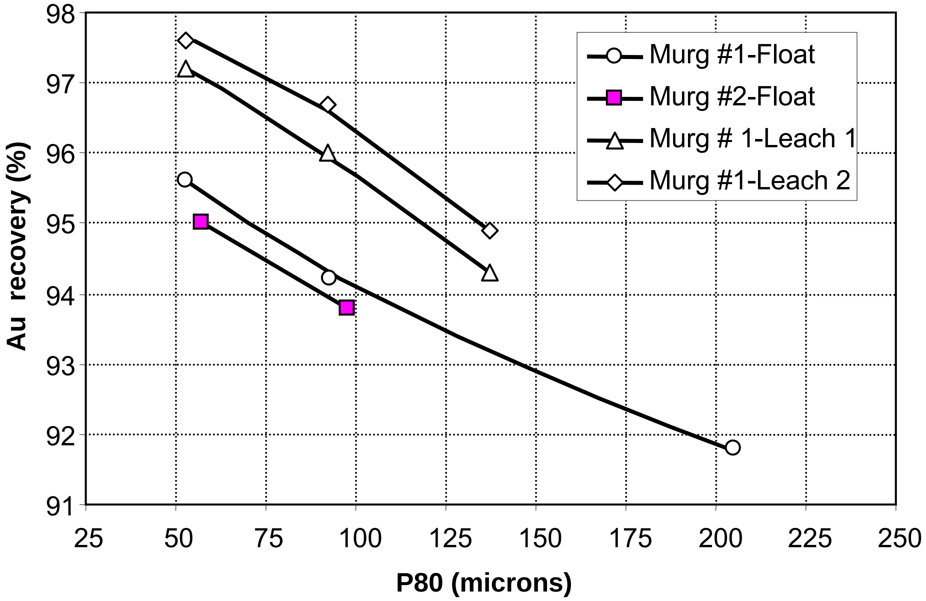


Figure 18. Dependence of gold recovery/extraction on particle size in flotation and cyanidation.

Based on analysis of pregnant solution, sodium cyanide and lime using titration, an estimate of their consumption during leaching was obtained. The consumption of sodium cyanide depended on the solution strength used in leaching. In the tests carried out at 1 g/L, it increased from 0.5 to 0.6 kg sodium cyanide per tonne of feed as the P_{80} value decreased from 137 μm to 53 μm. When its initial concentration was 2 g/L sodium cyanide solution, the sodium cyanide consumption increased from 1.34 to 1.49 kg per tonne of ore. Cyanide is prone to chemical alteration as it is not very stable in solutions. An important factor contributing to cyanide consumption is related to its decomposition in slurry during leaching. This can vary depending on the slurry chemistry from a value less than 1 kg/tonne to more than 2 kg/tonne and influence the economics of the process. In general, the higher the initial concentration of cyanide, the greater the rate of leaching, but also the greater the rate of cyanide decomposition during leaching. Thus, there is a trade off. The most suitable strength of cyanide for leaching can be determined by optimization tests. Unless judged necessary for extraction targets, high concentration of cyanide in leaching should be avoided as it also implies higher costs for its destruction for environmental reasons at the end. Lime consumption was much lower compared to the cyanide varying from 0.15 to 42 kg lime per tonne of ore. Further details can be seen from related results tabulated in the appendices section.

4.4. Gold recovery by Knelson gravity concentrator

A laboratory Knelson concentrator with 3-inch cone was used for two different grind sizes for gold recovery from composites 1 and 2. Figure 19 shows gold grade and recoveries obtained from composite 1 at a P_{80} of 92 μm (empty symbols) and 137 μm (filled symbols). Three water pressures from 3 to 9 psi were tested for fluidization of particles for the P_{80} of 92 μm and two from 4 to 7.5 psi for P_{80} of 137 μm. Compared to the case with flotation, the data points are highly scattered due to nature of separation using this device. Flotation separates mineral particles based on their hydrophobicity, a surface property that only sulphide particles develop in the collector system used. If the particles are hydrophilic (predominantly non-sulphide minerals) they are not recovered. In the case of gravity separation, there are a series of particles with similar specific gravities, magnetite, pyrite, ankerite etc. which can be simultaneously recovered. Their hindered settling behaviour in the same medium is more complicated. However, regardless of data scatter, it can be generally noted that high water pressures are associated with lower recoveries and high gold grades. The mass recoveries in these tests are much greater than those obtained in flotation tests. For example, the mass recovery for the results reported in Figure 19 ranged from 12% to 37%. The higher the mass recovery, the higher the gold recovery.

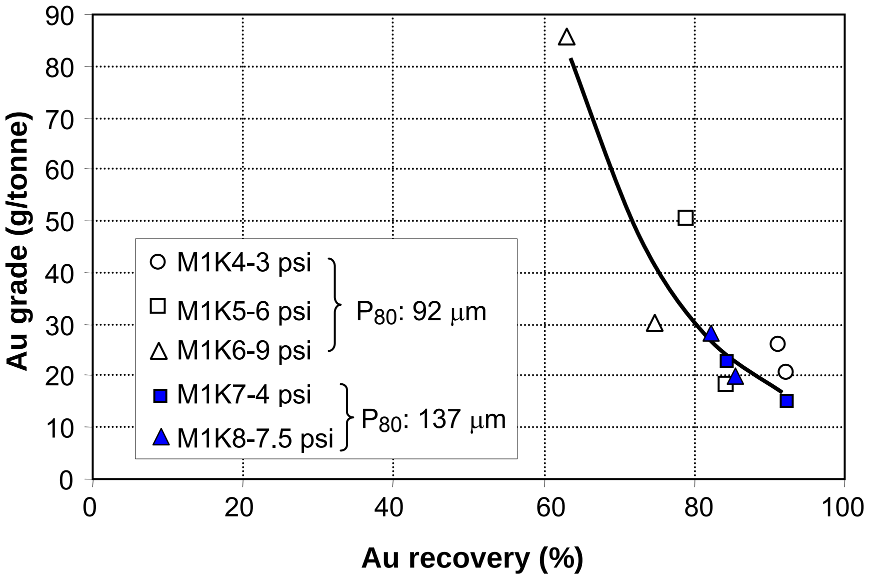


Figure 19. Gold recovery using Knelson concentrator from composite 1 at various operating water pressure for fluidization and two grind sizes.

Results obtained for composite 2 are shown in Figure 20. The particle size distribution in this particular case was 150 µm and fluidization water pressure was changed from 4 psi to 13 psi. Again, the data is highly scattered. However, similar trends may be noted. High gold recoveries above 90% require low operating pressures which promote higher mass recovery.

The concentrate products from Knelson separator are characterized by their relatively large weights. As noted previously, this is the basic reason for relatively poor grades. Magnetite was known to be present in this ore. It has a specific gravity of 5.15 and little heavier than pyrite itself (with a specific gravity of about 5) as the main gold carrier. Thus it is gravity-recovered along with gold. An additional test was carried out which involved removal magnetite prior to gravity separation to see to what extent gold grade can be improved as a result.

Mass balance of this test is appended along with others. In this test, it was found that the amount of magnetic product from magnetic separation was only about 5.8%. So its dilution effect on the concentrate was very limited. Furthermore, the magnetic

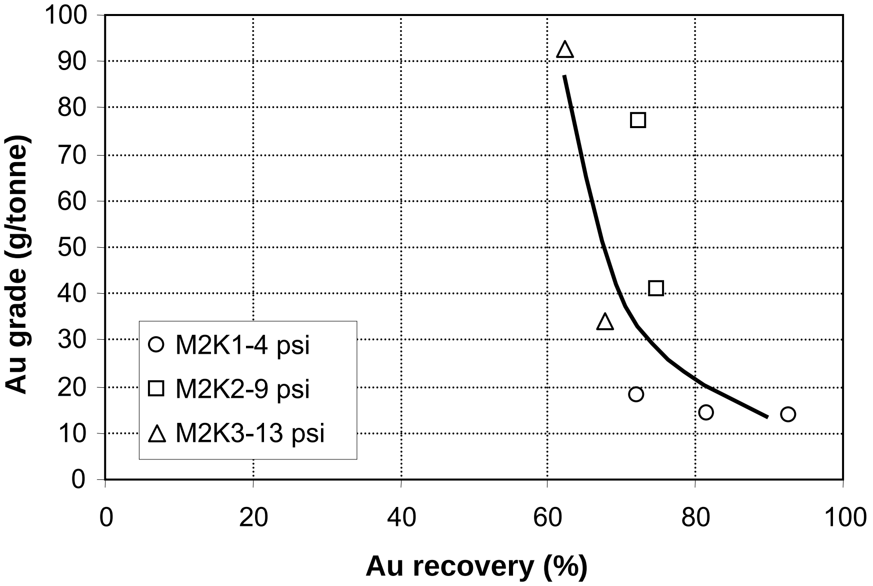


Figure 20. Gold recovery using Knelson concentrator from composite 2 at three operating water pressure for fluidization (P_{80} = 150 μm)

product separated had a high gold content at about 10 g/tonne. Although the origin of this is not believed to be mineralogical in nature, a magnetic separation stage in the process flowsheet will not serve any useful purpose. The dilution problem is not specific to the presence of magnetite in this ore.

The graph showing the dependence of gold recovery and/or extraction on particle size is represented as Figure 21 which now includes additional data from tests with Knelson gravity separation. As can be noted, the slope of line going through these data points is smaller. So, it can be said that within particle size range studied, the gold recovery by this method is relatively less dependent on the particle size. Further details on separation characteristics of mineralogical components can be found in the mass balance tables in the appendix. For example, the correlation between gold recovery and sulphur recovery can be noted from the Knelson data as well. The iron data, however, is different in that 2-3 times more iron bearing minerals are recovered

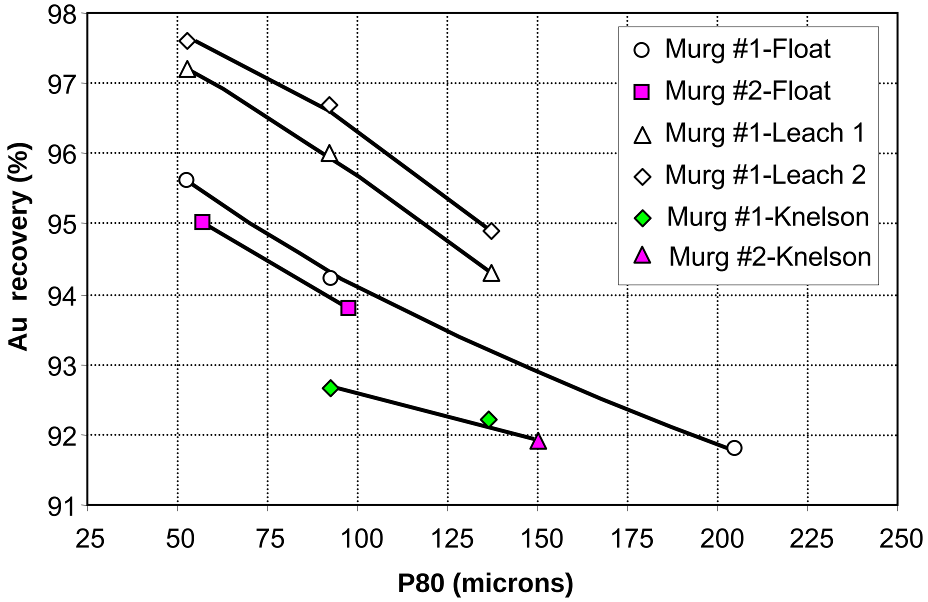


Figure 21. Dependence of gold recovery/extraction on particle size in flotation, cyanidation and gravity separation.

into the concentrates and middlings, compared to the case with flotation. It can be concluded that gravity separation by Knelson concentrator cannot compete with flotation as another physical method of separation. However, these results are likely to be improved in a continuous mode of operation on a larger scale.

5. Conclusions

From the tests carried out on two composites from Murgor Resources it can be concluded that:

1) Work index of composite 2 is 11 kWh/tonne as determined using Bond's standard ball mill grindability compared to 10.4 kWh/tonne for composite 1 as determined using comparative method. Based on these values on hardness, Murgor ore can be categorized as a medium to soft ore.

2) The mineralogy of Murgor ore allows high gold recovery using flotation. Recoveries from 91.9% to 95.8% are obtainable at a mean P_{80} level of 205

μm to 53 μm. Overall gold grades over 125 g/tonne are possible due to low mass recoveries of concentrates at 3.4-3.7 % of the ore.

3) Gold occurrence in Murgor ore is highly amenable to gold leaching by conventional cyanidation yielding 94.2% to 97.5% extraction at a mean P_{80} level of 137 μm to 53 μm. Sodium cyanide consumption varies from 0.5-0.6 kg/tonne to 1.34-1.49 kg/tonne at a sodium cyanide concentration of 1g/L and 2 g/L, respectively.

4) Gold in Murgor ore is also recoverable by gravity separation as demonstrated using a lab size Knelson concentrator. However, efficiency of separation is comparatively poor. High gold recoveries (max. 92-93%) are possible at the expense of low grades (e.g.,15-20 g/tonne) due to dilution of concentrates by simultaneous recovery of barren minerals that are of high specific gravity.

6. Recommendations

The testing program described in this report has been directed to reveal process characteristics of Murgor samples. With all the samples generated from quarters of drill cores, a number of tests were carried out, which involved work index, flotation, cyanidation and gravity separation. Considering all results, it can be conclusively stated that the Murgor ore is a good ore in terms of recoverability/extractability of gold. However, additional work can be carried out for confirmation and if desired this work can be done delegated to a different laboratory facility for independent assessment. In the current work, no cyanidation data could be provided on Murgor 2 since there was no sample left. Additional tests are recommended to get data in this area and these tests should be conducted at lower cyanide strengths such as 0.25 g/L and 0.5 g/L. Data on consumption of cyanide and lime should also be provided. It is also recommended that some products from selected separation tests be submitted for mineralogical analysis in order to provide a more detailed mineralogical characterization of this ore as well as to provide data on the nature of gold losses. Finally, it is recommended that a short pilot plant campaign be undertaken on the process of choice for the treatment of Murgor ores.

7. Acknowledgements

The testwork described in this report has been carried out by the writer of this report as the principal investigator with valuable assistance by E. Yalcin, N. Balakrishnan and M. Bailey. Cooperation of every team member is greatly acknowledged. Dr. W. Yen who has acted as an additional resource person in this work is also gratefully thanked.

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Appendices

Appendix A: Tabulated data on grinding: particle size data on all tests

Appendix B: Tabulated data on flotation

Appendix C: Tabulated data on cyanidation

Appendix D: Tabulated data on Knelson gravity separation

APPENDIX A: Tabulated data on grinding: particle size data on all tests

Table 4. Work Index testwork: feed size distribution

Feed Size for Bond Test (Murgor 2)

Sieves (mesh)	Sieve size (μm)	Mean Size (μm)	Wt. Retained (g)	Wt. Retained Cum. (g)	Wt. Retained Cum. (%)	Wt. Passing Cum. (%)
-6 + 8	3350	2855	0.93	0.93	0.51	99.49
-8 + 10	2360	2030	13.05	13.98	7.71	92.29
-10 + 14	1700	1440	34.17	48.15	26.55	73.45
-14 + 20	1180	1015	31.94	80.09	44.16	55.84
-20 + 28	850	725	27.22	107.31	59.17	40.83
-28 + 35	600	513	18.07	125.38	69.13	30.87
-35 + 48	425	363	13.11	138.49	76.36	23.64
-48 + 65	300	256	8.63	147.12	81.12	18.88
-65 + 100	212	181	7.9	155.02	85.48	14.52
-100 + 150	150	128	3.21	158.23	87.25	12.75
-150 + 200	106	91	5.55	163.78	90.31	9.69
-200	75		17.58	181.36	100.00	0.00

F_{80} 1635 microns (mean)

Table 5. Work Index testwork: product size distribution

Product Size (Average from the Last Two Cycles)

Sieves (mesh)	Sieve size (μm)	Mean Size (μm)	Wt. Retained (g)	Wt. Retained Cum. (g)	Wt. Retained Cum. (%)	Wt. Passing Cum. (%)
65		234	0.55	0.55	0.39	99.61
(-65+80)	212	196	17.47	18.02	12.62	87.38
(-80+100)	180	165	20.08	38.10	26.68	73.32
(-100+150)	150	128	14.92	53.02	37.13	62.87
(-150+200)	106	91	17.25	70.27	49.21	50.79
(-200+270)	75	64	7.38	77.65	54.37	45.63
(-270+325)	53	49	7.48	85.13	59.61	40.39
(-325+400)	45	42	2.84	87.97	61.60	38.40
-400	38		54.84	142.81	100.00	0.00

P_{80} 180 microns

Table 6. Particle size comparison of Murgor #1 and Murgor #2 (-10 mesh)

Feed Size (Grinding for separation tests)
Murgor #1

Sieves (mesh)	Sieve size (μm)	Mean Size (μm)	Wt. Retained (g)	Wt. Retained Cum. (g)	Wt. Retained Cum. (%)	Wt. Passing Cum. (%)
-10 + 14	1700	1440	11.96	11.96	9.56	90.44
-14 + 20	1180	1015	26.62	38.58	30.83	69.17
-20 + 28	850	725	20.55	59.13	47.26	52.74
-28 + 35	600	513	18.99	78.12	62.44	37.56
-35 + 48	425	363	10.31	88.43	70.68	29.32
-48 + 65	300	256	6.82	95.25	76.13	23.87
-65 + 100	212	181	6.69	101.94	81.47	18.53
-100 + 150	150	128	3.63	105.57	84.38	15.63
-150 + 200	106	91	4.12	109.69	87.67	12.33
-200 + 270	75	64	3.25	112.94	90.27	9.73
-270 + 325	53	49	2.98	115.92	92.65	7.35
-325 + 400	45	42	2.85	118.77	94.92	5.08
-400	38	19	6.35	125.12	100.00	0.00

Feed Size (Grinding for separation tests)
Murgor #2

Sieves (mesh)	Sieve size (μm)	Mean Size (μm)	Wt. Retained (g)	Wt. Retained Cum. (g)	Wt. Retained Cum. (%)	Wt. Passing Cum. (%)
-10 + 14	1700	1440	14.31	14.31	9.69	90.31
-14 + 20	1180	1015	33.84	48.15	32.59	67.41
-20 + 28	850	725	24.96	73.11	49.49	50.51
-28 + 35	600	513	19.82	92.93	62.91	37.09
-35 + 48	425	363	11.97	104.90	71.01	28.99
-48 + 65	300	256	8.87	113.77	77.01	22.99
-65 + 100	212	181	7.81	121.58	82.30	17.70
-100 + 150	150	128	4.12	125.70	85.09	14.91
-150 + 200	106	91	6.35	132.05	89.39	10.61
-200 + 270	75	64	4.26	136.31	92.27	7.73
-270 + 325	53	49	3.1	139.41	94.37	5.63
-325 + 400	45	42	5.22	144.63	97.90	2.10
-400	38	19	3.1	147.73	100.0	0.00

Table 7. Particle size distribution from Murgor #1 and Murgor #2 after 7 minutes of grinding

Murgor 1 (ground with Denver R.M. for 7 min.)

Sieves (mesh)	Sieve size (μm)	Mean Size (μm)	Wt. Retained (g)	Wt. Retained Cum. (g)	Wt. Retained Cum. (%)	Wt. Passing Cum. (%)
65		234	1.80	1.80	1.37	98.63
(-65+100)	212	181	9.9	11.70	8.89	91.11
(-100+150)	150	128	18.2	29.90	22.72	77.28
(-150+200)	106	91	18.12	48.02	36.49	63.51
(-200+270)	75	64	8.17	56.19	42.70	57.30
(-270+325)	53	49	5.2	61.39	46.65	53.35
(-325+400)	45	42	7.01	68.40	51.98	48.02
-400	38	15	63.19	131.59	100.00	0.00

Murgor 2 (ground with Denver R.M. for 7 min.)

Sieves (mesh)	Sieve size (μm)	Mean Size (μm)	Wt. Retained (g)	Wt. Retained Cum. (g)	Wt. Retained Cum. (%)	Wt. Passing Cum. (%)
65		234	2.38	2.38	1.65	98.35
(-65+100)	212	181	16.03	18.41	12.80	87.20
(-100+150)	150	128	18.92	37.33	25.95	74.05
(-150+200)	106	91	19.67	57.00	39.62	60.38
(-200+270)	75	64	8.75	65.75	45.70	54.30
(-270+325)	53	49	5.4	71.15	49.46	50.54
(-325+400)	45	42	7.3	78.45	54.53	45.47
-400	38	15	65.41	143.86	100.00	0.00

Table 8. Particle size distribution from Murgor #1 after 10 and 15 minutes of grinding

Murgor 1 (ground with Denver R.M. for 10 min.)

Sieves (mesh)	Sieve size (μm)	Mean Size (μm)	Wt. Retained (g)	Wt. Retained Cum. (g)	Wt. Retained Cum. (%)	Wt. Passing Cum. (%)
65		234	0.75	0.75	0.56	99.44
(-65+100)	212	181	1.01	1.76	1.31	98.69
(-100+150)	150	128	6.31	8.07	6.00	94.00
(-150+200)	106	91	17.98	26.05	19.37	80.63
(-200+270)	75	64	14.52	40.57	30.16	69.84
(-270+325)	53	49	10.42	50.99	37.91	62.09
(-325+400)	45	42	8.28	59.27	44.06	55.94
-400	38	15	75.24	134.51	100.00	0.00

Murgor 1 (ground with Denver R.M. for 15 min.)

Sieves (mesh)	Sieve size (μm)	Mean Size (μm)	Wt. Retained (g)	Wt. Retained Cum. (g)	Wt. Retained Cum. (%)	Wt. Passing Cum. (%)
65		234	0.40	0.40	0.32	99.68
(-65+100)	212	181	0.22	0.62	0.49	99.51
(-100+150)	150	128	0.54	1.16	0.91	99.09
(-150+200)	106	91	6.02	7.18	5.66	94.34
(-200+270)	75	64	10.81	17.99	14.19	85.81
(-270+325)	53	49	17.25	35.24	27.79	72.21
(-325+400)	45	42	5.39	40.63	32.04	67.96
-400	38	15	86.2	126.81	100.00	0.00

Table 9. Particle size distribution of flotation tails from Murgor #1 feed after 15, 10 and 5 minutes of grinding

Murgor 1 Float Tails #1 (feed ground with Denver R.M. for 15 min.)

Sieves (mesh)	Sieve size (μm)	Mean Size (μm)	Wt. Retained (g)	Wt. Retained Cum. (g)	Wt. Retained Cum. (%)	Wt. Passing Cum. (%)
65		234	0.11	0.11	0.10	99.90
(-65+100)	212	181	0.13	0.24	0.23	99.77
(-100+150)	150	128	0.31	0.55	0.52	99.48
(-150+200)	106	91	3.74	4.29	4.06	95.94
(-200+270)	75	64	10.37	14.66	13.87	86.13
(-270+325)	53	49	9.26	23.92	22.63	77.37
(-325+400)	45	42	5.22	29.14	27.57	72.43
-400	38	15	76.54	105.68	100.00	0.00

Murgor 1 Float Tails #2 (feed ground with Denver R.M. for 10 min.)

Sieves (mesh)	Sieve size (μm)	Mean Size (μm)	Wt. Retained (g)	Wt. Retained Cum. (g)	Wt. Retained Cum. (%)	Wt. Passing Cum. (%)
65		234	0.23	0.23	0.20	99.80
(-65+100)	212	181	1.17	1.40	1.21	98.79
(-100+150)	150	128	5.09	6.49	5.61	94.39
(-150+200)	106	91	17.44	23.93	20.67	79.33
(-200+270)	75	64	11.81	35.74	30.88	69.12
(-270+325)	53	49	7.69	43.43	37.52	62.48
(-325+400)	45	42	3.44	46.87	40.49	59.51
-400	38	15	68.88	115.75	100.00	0.00

Murgor 1 Float Tails #3 (feed ground with Denver R.M. for 5 min.)

Sieves (mesh)	Sieve size (μm)	Mean Size (μm)	Wt. Retained (g)	Wt. Retained Cum. (g)	Wt. Retained Cum. (%)	Wt. Passing Cum. (%)
65		234	9.6	9.60	7.81	92.19
(-65+100)	212	181	25.03	34.63	28.17	71.83
(-100+150)	150	128	11.47	46.10	37.50	62.50
(-150+200)	106	91	5.91	52.01	42.31	57.69
(-200+270)	75	64	14.8	66.81	54.35	45.65
(-270+325)	53	49	5.82	72.63	59.08	40.92
(-325+400)	45	42	3.01	75.64	61.53	38.47
-400	38	15	47.29	122.93	100.00	0.00

Table 10. Particle size distribution of flotation tails from Murgor #2 feed after 10 and 15 minutes of grinding

Murgor 2 Float Tails (feed ground with Denver R.M. for 10 min.)

Sieves (mesh)	Sieve size (μm)	Mean Size (μm)	Wt. Retained (g)	Wt. Retained Cum. (g)	Wt. Retained Cum. (%)	Wt. Passing Cum. (%)
65		234	0.1	0.10	0.07	99.93
(-65+100)	212	181	1.56	1.66	1.24	98.76
(-100+150)	150	128	7.52	9.18	6.85	93.15
(-150+200)	106	91	22.04	31.22	23.28	76.72
(-200+270)	75	64	12.59	43.81	32.67	67.33
(-270+325)	53	49	11.89	55.70	41.54	58.46
(-325+400)	45	42	7.81	63.51	47.36	52.64
-400	38	15	70.59	134.10	100.00	0.00

Murgor 2 Float Tails (feed ground with Denver R.M. for 15 min.)

Sieves (mesh)	Sieve size (μm)	Mean Size (μm)	Wt. Retained (g)	Wt. Retained Cum. (g)	Wt. Retained Cum. (%)	Wt. Passing Cum. (%)
65		234	0.2	0.20	0.15	99.85
(-65+100)	212	181	0.21	0.41	0.30	99.70
(-100+150)	150	128	0.54	0.95	0.70	99.30
(-150+200)	106	91	6.91	7.86	5.77	94.23
(-200+270)	75	64	15.2	23.06	16.93	83.07
(-270+325)	53	49	10.1	33.16	24.35	75.65
(-325+400)	45	42	7.98	41.14	30.21	69.79
-400	38	15	95.05	136.19	100.00	0.00

Table 11. Particle size distribution of cyanidation tails from Murgor #1 feed after 7 minutes of grinding

Sieve Mesh Size	A1 (g)	A2 (g)	Combined. (g)
65	1.39	1.52	2.91
(-65+100)	10.73	11.26	21.99
(-100+150)	13.97	13.55	27.52
(-150+200)	17.41	16.97	34.38
(-200+270)	5.38	8.69	14.07
(-270+325)	9.86	8.62	18.48
(-325+400)	3.51	3.19	6.7
-400	0.22	0.22	0.44
+400 mesh	62.65	64.76	127.41
Total wt.	117.1	118.2	235.30

Murgor 1 Cyanidation Tails, A1 & A2 (7 min)

Sieves (mesh)	Sieve size (μm)	Mean Size (μm)	Wt. Retained (g)	Wt. Retained Cum. (g)	Wt. Retained Cum. (%)	Wt. Passing Cum. (%)
65		234	2.91	2.91	1.24	98.76
(-65+100)	212	181	21.99	24.90	10.63	89.37
(-100+150)	150	128	27.52	52.42	22.38	77.62
(-150+200)	106	91	34.38	86.80	37.06	62.94
(-200+270)	75	64	14.07	100.87	43.07	56.93
(-270+325)	53	49	17.78	118.65	50.66	49.34
(-325+400)	45	42	6.3	124.95	53.35	46.65
-400	38	15	109.25	234.20	100.00	0.00

Table 12. Particle size distribution of cyanidation tails from Murgor #1 feed after 10 minutes of grinding

Sieve Mesh Size	A3 (g)	A4 (g)	Combined (g)
65	0.36	0.37	0.73
(-65+100)	1.42	1.56	2.98
(-100+150)	6.52	6.35	12.87
(-150+200)	17.23	16.09	33.32
(-200+270)	12.76	15.41	28.17
(-270+325)	11.85	9.87	21.72
(-325+400)	3.47	2.81	6.28
-400	3.02	4.03	7.05

+400 mesh	57.07	56.96	114.03
Total wt.	125.01	125.5	250.51

Murgor 1 Cyanidation Tails, A3 & A4 (10 min)

Sieves (mesh)	Sieve size (μm)	Mean Size (μm)	Wt. Retained (g)	Wt. Retained Cum. (g)	Wt. Retained Cum. (%)	Wt. Passing Cum. (%)
65		234	0.73	0.73	0.29	99.71
(-65+100)	212	181	2.98	3.71	1.49	98.51
(-100+150)	150	128	12.87	16.58	6.64	93.36
(-150+200)	106	91	33.32	49.90	19.98	80.02
(-200+270)	75	64	28.17	78.07	31.27	68.73
(-270+325)	53	49	21.72	99.79	39.96	60.04
(-325+400)	45	42	6.28	106.1	42.48	57.52
-400	38	15	143.63	249.7	100.0	0.00

Table 13. Particle size distribution of cyanidation tails from Murgor #1 feed after 15 minutes of grinding

Murgor 1 Cyanidation Tails, A5 & A6 (15 min)

Sieve Mesh Size	A5 (g)	A6 (g)	Combined (g)
65	0.09	0.11	0.20
(-65+100)	0.12	0.15	0.27
(-100+150)	0.49	0.54	1.03
(-150+200)	7.38	7.19	14.57
(-200+270)	10.74	9.84	20.58
(-270+325)	15.26	10.38	25.64
(-325+400)	3.94	8.49	12.43
-400	0.33	1.21	1.54

+400 mesh	39.27	37.13	76.4
Total wt.	125.38	125.07	250.45

Murgor 1 Cyanidation Tails, A5 & A6 (15 min)

Sieves (mesh)	Sieve size (μm)	Mean Size (μm)	Wt. Retained (g)	Wt. Retained Cum. (g)	Wt. Retained Cum. (%)	Wt. Passing Cum. (%)
65		234	0.20	0.20	0.08	99.92
(-65+100)	212	181	0.27	0.47	0.19	99.81
(-100+150)	150	128	1.03	1.50	0.60	99.40
(-150+200)	106	91	14.57	16.07	6.42	93.58
(-200+270)	75	64	20.58	36.65	14.64	85.36
(-270+325)	53	49	25.64	62.29	24.89	75.11
(-325+400)	45	42	12.43	74.7	29.85	70.15
-400	38	15	175.59	250.3	100.0	0.00

APPENDIX B: Flotation reports and mass balance sheets

Table 14. Flotation test report: F1M1

TEST : 1 DATE: May 5, 2006 FEED: Murgor Ore #1

OBJECTIVE: Rougher-scavenger float for gold recovery

GRINDING CONDITIONS (Denver Mill):

MILL : *MS -Denver* ROD TYPE: *50% SS*

CHARGE: *~1000 g* 66.7 % Solids *Tap*

WATER: *500 ml.*

GRIND: 0.5 g Na_2CO_3 GRIND TIME: 15 min.

20 ml PAX+ 20 iPrpX

FLOAT CONDITIONS: FLOATED BY: S.K

CELL TYPE & VOLUME: *2 L - Denver Cell*

IMPELLER RPM: *1200* REPULP WATER: *Tap*

Ro GAS RATE: *3* Or air valve fully open

\# of STROKES: *30/min*

REAGENTS	ml	g/Ton	STAGE	COND TIME	FLOAT TIME	pH Start	pH End	REDOX / pH: as is:	REDOX mV
Redox (with calomel reference)								After 1 min stirring	
						as is	9.37	Cond.: 418 µs/cm	351
pH (Soda ash)									
(PAX) @ 0.1%	10	10	Cond.	2					
(IPX) @ 0.1%	10	10							
Frother (0.1%)	25	25	Cond.	1			9.28		237
(Dow Froth 250)									
			Conc. 1		0.5		9.27		215
(PAX) @ 0.1%	10	10	Cond.	1			9.22		186
(IPX) @ 0.1%	10	10							
Frother (0.1%)	10	10	together	min					
			Conc. 2		1.5		9.19		183
(PAX) @ 0.1%	10	10							
(IPX) @ 0.1%	10	10	Cond.	1			9.16		160
Frother (0.1%)	10	10	together	min					
			Conc. 3		2		9.17		
(PAX) @ 0.1%	10	10							
(IPX) @ 0.1%	10	10	Cond.	1					
Frother (0.1%)	5 ml	5	together	min					
							9.22		139
(PAX) Total		60							
(IPX) Total		60							
Frother Total		50							

Table 15. Flotation mass balance: F1M1 (P_{80} = 53 μm)

M1Test 1	T I M E	M A S S		A S S A Y S						
Product	Min.	(g)	(%)	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)	Pb (%)	Zn (%)
Conc. 1	0.5	14.33	1.42	260.8	44.8	0.71	46.1	0.30	0.02	0.08
Conc. 2	1.5	5.74	0.57	195.8	30.4	1.29	34.4	0.30	0.06	0.25
Conc. 3	2	5.1	0.51	24.0	3.88	2.04	10.10	0.07	0.02	0.17
Conc. 4	3	11.61	1.15	13.3	0.71	2.19	6.69	0.01	0.004	0.08
Tails		973.03	96.36	0.23	0.05	2.40	5.78	0.001	0.002	0.01
Calc Head		1,009.8	100.0	5.31	0.88	2.37	6.54	0.008	0.003	0.012
Actual Head		1,011	100	5.18	0.80	2.40	6.67	0.007	0.004	0.012
Cumulative										

Min.	Wt. (g)	Mass Rec	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)	Pb (%)	Zn (%)
0		0							
0.5	14.33	1.42	260.8	44.8	0.71	46.1	0.30	0.02	0.08
2.0	20.07	1.99	242.2	40.7	0.88	42.8	0.30	0.03	0.13
4.0	25.17	2.49	198.0	33.2	1.11	36.2	0.25	0.03	0.14
7.0	36.78	3.64	139.7	23.0	1.45	26.9	0.18	0.02	0.12
	1009.8	100.0	5.31	0.88	2.37	6.54	0.008	0.003	0.012

			R E C O V E R I E S (%)						
Min.	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu	Pb	Zn
0	14.33	1.42	69.7	71.9	0.43	10.01	54.65	10.36	8.99
0.5	5.74	0.57	21.0	19.5	0.31	2.99	21.85	12.21	11.20
2.0	5.1	0.51	2.29	2.2	0.44	0.78	4.83	3.78	6.90
4.0	11.61	1.15	2.88	0.92	1.06	1.18	2.12	1.82	7.02
7.0	973.03	96.36	4.17	5.4	97.76	85.05	16.55	71.83	65.88

	Cumulative		C U M U L A T I V E R E C O V E R I E S (%)						
Min.	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu	Pb	Zn
0		0	0	0	0	0	0	0	0
0.5	14.33	1.42	69.7	71.9	0.43	10.01	54.65	10.36	8.99
2.0	20.07	1.99	90.7	91.4	0.74	12.99	76.50	22.57	20.19
4.0	25.17	2.49	92.9	93.6	1.17	13.77	81.33	26.35	27.09
7.0	36.78	3.64	95.8	94.6	2.24	14.95	83.45	28.17	34.12
	1009.8	100.0							

Table 16. Flotation test report: F2M1

TEST :	2				DATE:			FEED:	Murgor Ore # 1	
OBJECTIVE:		Rougher-scavenger float for gold recovery								
GRINDING CONDITIONS (Denver Mill):						FLOAT CONDITIONS:			FLOATED BY:	S.K
MILL :	*MS -Denver*	ROD TYPE:		*50% SS*		CELL TYPE & VOLUME:			*2 L - Denver Cell*	
CHARGE:	*~1000 g*	66.7	% Solids	*Tap*		IMPELLER RPM:		*1200*	REPULP WATER:	*Tap*
WATER:	*500 ml.*					Ro GAS RATE:		*3*	Or air valve fully open	
GRIND:	0.3 g Na_2CO_3	GRIND TIME:		10 min.		# of STROKES:		*30/min*		
20 ml PAX+ 20 iPrpX		ml	g/Ton	STAGE	COND	FLOAT	pH	pH	REDOX / pH: as is:	REDOX
REAGENTS					TIME	TIME	Start	End		mV
Redox									After 1 min stirring	
							as is	8.86	Cond.: 340 μs/cm	382
pH (Soda ash)										
(PAX) @ 0.1%		10	10	Cond.	2					
(IPX) @ 0.1%		10	10							
Frother (0.1%)		25	25	Cond.	1			8.79		199
(Dow Froth 250)										
				Conc. 1		0.5		8.78		184
(PAX) @ 0.1%		10	10	Cond.	1			8.76		171
(IPX) @ 0.1%		10	10							
Frother (0.1%)		10	10	together	min					
				Conc. 2		1.5		8.75		191
(PAX) @ 0.1%		10	10							
(IPX) @ 0.1%		10	10	Cond.	1			8.77		173
Frother (0.1%)		10	10	together	min					
				Conc. 3		2		8.82		181
(PAX) @ 0.1%		10	10							
(IPX) @ 0.1%		10	10	Cond.	1					
Frother (0.1%)		5 ml	5	together	min					
				Conc. 4		3		8.75		180
(PAX) Total			60							
(IPX) Total			60							
Frother Total			50							

Table 17. Flotation mass balance: F2M1 (P_{80} = 92 μm)

M1Test 2	TIME	MASS		ASSAYS						
Product	Min.	(g)	(%)	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)	Pb (%)	Zn (%)
Conc. 1	0.5	14.99	1.49	240.5	41.7	0.67	45.8	0.24	0.03	0.08
Conc. 2	1.5	7.17	0.71	167.4	24.9	1.57	24.5	0.25	0.05	0.25
Conc. 3	2	5.72	0.57	22.6	2.53	2.22	8.19	0.05	0.01	0.17
Conc. 4	3	6.05	0.60	12.3	0.99	2.23	7.41	0.02	0.003	0.08
Tails		972.89	96.63	0.32	0.06	2.35	5.90	0.001	0.002	0.01
Calc Head		1,006.8	100.0	5.28	0.87	2.32	6.65	0.007	0.003	0.013
Actual Head		1,009	100	5.18	0.80	2.22	6.67	0.007	0.004	0.012
Cumulative										
	Min.	Wt. (g)	Mass Rec	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)	Pb (%)	Zn (%)
	0		0							
	0.5	14.99	1.49	240.5	41.7	0.67	45.8	0.24	0.03	0.08
	2.0	22.16	2.20	216.8	36.3	0.96	38.9	0.25	0.03	0.13
	4.0	27.88	2.77	177.0	29.3	1.22	32.6	0.21	0.03	0.14
	7.0	33.93	3.37	147.6	24.3	1.40	28.1	0.17	0.02	0.13
		1006.8	100.0	5.28	0.87	2.32	6.65	0.007	0.003	0.013

			RECOVERIES (%)						
Min.	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu	Pb	Zn
0	14.99	1.49	67.8	71.2	0.43	10.27	50.64	13.27	9.34
0.5	7.17	0.71	22.6	20.3	0.48	2.62	25.31	10.55	13.90
2.0	5.72	0.57	2.43	1.6	0.54	0.70	4.02	1.30	7.69
4.0	6.05	0.60	1.40	0.68	0.58	0.67	1.92	0.63	3.64
7.0	972.89	96.63	5.78	6.1	97.97	85.74	18.12	74.26	65.43

	Cumulative		CUMULATIVE RECOVERIES (%)						
Min.	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu	Pb	Zn
0		0	0	0	0				
0.5	14.99	1.49	67.8	71.2	0.43	10.27	50.64	13.27	9.34
2.0	22.16	2.20	90.4	91.6	0.91	12.89	75.94	23.82	23.24
4.0	27.88	2.77	92.8	93.2	1.46	13.59	79.96	25.11	30.93
7.0	33.93	3.37	94.2	93.9	2.03	14.26	81.88	25.74	34.57
	1006.8	100.0							

Table 18. Flotation test report: F3M1

TEST :	3			DATE:	May 5, 2006			FEED:	Murgor Ore #1	
OBJECTIVE:			Rougher-scavenger float for gold recovery							
GRINDING CONDITIONS (Denver Mill):						FLOAT CONDITIONS:			FLOATED BY:	S.K
MILL :	*MS -Denver*	ROD TYPE:		*50% SS*		CELL TYPE & VOLUME:			*2 L - Denver Cell*	
CHARGE:	*~1000 g*	66.7	% Solids	*Tap*		IMPELLER RPM:		*1200*	REPULP WATER:	*Tap*
WATER:	*500 ml.*					Ro GAS RATE:		*3*	Or air valve fully open	
GRIND:	0.3 g Na_2CO_3	GRIND TIME:		5 min.		# of STROKES:		*30/min*		
20 ml PAX+ 20 iPrpX		ml	g/Ton	STAGE	COND	FLOAT	pH	pH	REDOX / pH: as is:	REDOX
REAGENTS					TIME	TIME	Start	End		mV
Redox									After 1 min stirring	
							as is	8.73	Cond.: 203 µs/cm	355
pH (Soda ash)										
(PAX) @ 0.1%		10	10	Cond.	2					
(IPX) @ 0.1%		10	10							
Frother (0.1%)		25	25	Cond.	1			8.68		203
(Dow Froth 250)										
				Conc. 1		0.5		8.67		173
(PAX) @ 0.1%		10	10	Cond.	1			8.64		164
(IPX) @ 0.1%		10	10							
Frother (0.1%)		10	10	together	min					
				Conc. 2		1.5		8.65		145
(PAX) @ 0.1%		10	10							
(IPX) @ 0.1%		10	10	Cond.	1			8.63		141
Frother (0.1%)		10	10	together	min					
				Conc. 3		2		8.62		112
(PAX) @ 0.1%		10	10							
(IPX) @ 0.1%		10	10	Cond.	1					
Frother (0.1%)		5 ml	5	together	min					
				Conc. 4		3		8.66		118
(PAX) Total			60							
(IPX) Total			60							
Frother Total			50							

Table 19. Flotation mass balance: F3M1 (P_{80} = 205 μm)

M1Test 3	TIME	MASS		ASSAYS						
Product	Min.	(g)	(%)	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)	Pb (%)	Zn (%)
Conc. 1	0.5	12.69	1.27	240.3	38.9	0.89	43.5	0.19	0.02	0.07
Conc. 2	1.5	12.88	1.28	114.6	23	1.39	26.0	0.20	0.02	0.13
Conc. 3	2	6.09	0.61	29.8	5.87	2.05	11.00	0.07	0.02	0.16
Conc. 4	3	5.05	0.50	14.2	1.27	2.27	7.07	0.03	0.004	0.08
Tails		966.3	96.34	0.44	0.07	2.36	5.65	0.003	0.003	0.01
Calc Head		1,003	100.0	5.19	0.90	2.33	6.43	0.008	0.003	0.012
Actual Head		1,009	100	5.18	0.80	2.22	6.67	0.007	0.004	0.012
Cumulative										
	Min.	Wt. (g)	Mass Rec	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)	Pb (%)	Zn (%)
	0		0							
	0.5	12.69	1.27	240.3	38.9	0.89	43.5	0.19	0.02	0.07
	2.0	25.57	2.55	177.0	30.9	1.14	34.7	0.20	0.02	0.10
	4.0	31.66	3.16	148.7	26.1	1.32	30.1	0.17	0.02	0.11
	7.0	36.71	3.66	130.2	22.7	1.45	26.9	0.15	0.02	0.11
		1003.0	100.0	5.19	0.90	2.33	6.43	0.008	0.003	0.012

			RECOVERIES (%)						
Min.	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu	Pb	Zn
0	12.69	1.27	58.6	54.9	0.48	8.55	29.16	8.38	7.66
0.5	12.88	1.28	28.4	32.9	0.77	5.19	32.13	8.20	13.93
2.0	6.09	0.61	3.49	4.0	0.53	1.04	5.10	3.56	8.00
4.0	5.05	0.50	1.37	0.71	0.49	0.55	1.90	0.72	3.54
7.0	966.3	96.34	8.17	7.5	97.72	84.67	31.71	79.14	66.87

	Cumulative		CUMULATIVE RECOVERIES (%)						
Min.	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu	Pb	Zn
0		0	0	0	0				
0.5	12.69	1.27	58.6	54.9	0.48	8.55	29.16	8.38	7.66
2.0	25.57	2.55	87.0	87.8	1.25	13.74	61.29	16.58	21.60
4.0	31.66	3.16	90.5	91.8	1.79	14.78	66.40	20.13	29.60
7.0	36.71	3.66	91.8	92.5	2.28	15.33	68.29	20.86	33.13
	1003	100							

Table 20. Flotation test report: F4M2

TEST : 4 DATE: May 9, 2006 FEED: Murgor Ore # 2

OBJECTIVE: Rougher-scavenger float for gold recovery

GRINDING CONDITIONS (Denver Mill):						FLOAT CONDITIONS:		FLOATED BY:	S.K
MILL :	*MS -Denver*	ROD TYPE:		*50% SS*		CELL TYPE & VOLUME:	*2 L - Denver Cell*		
CHARGE:	*~1000 g*	66.7	% Solids	*Tap*		IMPELLER RPM:	*1200*	REPULP WATER:	*Tap*
WATER:	*500 ml.*					Ro GAS RATE:	*3*	Or air valve fully open	
GRIND:	0.3 g Na_2CO_3	GRIND TIME:		10 min.		# of STROKES:	*30/min*		

20 ml PAX+ 20 iPrpX		ml	g/Ton	STAGE	COND	FLOAT	pH	pH	REDOX / pH: as is:	REDOX
REAGENTS					TIME	TIME	Start	End		mV
Redox									After 1 min stirring	
							as is	8.7		165
pH (Soda ash)										
(PAX) @ 0.1%		10	10	Cond.	2					
(IPX) @ 0.1%		10	10							
3 drops				Cond.	1			8.67		146
(MIBC)										
				Conc. 1		0.5		8.2		144
(PAX) @ 0.1%		10	10	Cond.	1			8.61		140
(IPX) @ 0.1%		10	10							
no frother		10	10	together	min					
				Conc. 2		2		8.63		133
(PAX) @ 0.1%		10	10							
(IPX) @ 0.1%		10	10	Cond.	1			8.6		117
no frother		10	10	together	min					
				Conc. 3		2		8.62		116
(PAX) @ 0.1%		10	10							
(IPX) @ 0.1%		10	10	Cond.	1	Comb.				
1 drop		5 ml	5	together	min					
(MIBC)				Conc. 4		2.5		8.58		118
(PAX) Total			60							
(IPX) Total			60							
Frother Total			25							

Table 21. Flotation mass balance: F4M2 (P_{80} = 98 μm)

M2Test 4	TIME	MASS		ASSAYS					
Product	Min.	(g)	(%)	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)	Pb (%)
Conc. 1	0.5	16.12	1.62	221.8	44.5	0.60	30.2	0.18	0.04
Conc. 2	2	10.46	1.05	160.4	33.7	1.13	30.7	0.21	0.04
Conc. 3	4.5	8.12	0.81	18.6	4.35	2.19	9.43	0.05	0.01
Conc. 4	3	0	0.00	0.0	0.00	0	0.00	0.00	0.000
Tails		962.72	96.52	0.37	0.07	2.15	5.10	0.002	0.001
Calc Head		997	100.0	5.78	1.18	2.11	5.81	0.007	0.003
Actual Head		1,000	100	5.59	1.16	2.26	6.67	0.007	0.004

Cumulative								
Min.	Wt. (g)	Mass Rec	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)	Pb (%)
0		0						
0.5	16.12	1.62	221.8	44.5	0.60	30.2	0.18	0.04
2.5	26.58	2.66	197.6	40.2	0.81	30.4	0.19	0.04
7.0	34.7	3.48	155.7	31.8	1.13	25.5	0.16	0.03
	34.7	3.48	155.7	31.8	1.13	25.5	0.16	0.03
	997.4	100.0	5.78	1.18	2.11	5.81	0.007	0.003

			RECOVERIES (%)					
Min.	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu	Pb
0	16.12	1.62	62.1	61.2	0.46	8.40	40.09	26.98
0.5	10.46	1.05	29.1	30.1	0.56	5.55	30.57	16.40
2.5	8.12	0.81	2.62	3.0	0.84	1.32	5.04	3.95
7.0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00
0.0	962.7	96.5	6.20	5.7	98.14	84.73	24.31	52.67

	Cumulative		CUMULATIVE RECOVERIES (%)					
Min.	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu	Pb
0		0	0	0	0			
0.5	16.12	1.62	62.1	61.2	0.46	8.40	40.09	26.98
2.0	26.58	2.66	91.2	91.2	1.02	13.95	70.66	43.38
4.0	34.7	3.48	93.8	94.3	1.86	15.27	75.69	47.33
7.0	34.7	3.48	93.8	94.3	1.86	15.27	75.69	47.33
	997.4	100						

Table 22. Flotation test report: F5M2

TEST :	5			DATE:	May 9, 2006			FEED:	Murgor Ore # 2	
OBJECTIVE:		Rougher-scavenger float for gold recovery								
GRINDING CONDITIONS (Denver Mill):						FLOAT CONDITIONS:			FLOATED BY:	S.K
MILL :	*MS -Denver*	ROD TYPE:		*50% SS*		CELL TYPE & VOLUME:			*2 L - Denver Cell*	
CHARGE:	*~1000 g*	66.7	% Solids	*Tap*		IMPELLER RPM:		*1200*	REPULP WATER:	*Tap*
WATER:	*500 ml.*					Ro GAS RATE:		*3*	Or air valve fully open	
GRIND:	0.3 g Na_2CO_3	GRIND TIME:		15 min.		# of STROKES:		*30/min*		
20 ml PAX+ 20 iPrpX		ml	g/Ton	STAGE	COND	FLOAT	pH	pH	REDOX / pH: as is:	REDOX
REAGENTS					TIME	TIME	Start	End		mV
Redox									After 1 min stirring	
							as is	8.72		140
pH (Soda ash)										
(PAX) @ 0.1%		10	10	Cond.	2					
(IPX) @ 0.1%		10	10							
3 drops				Cond.	1			8.7		141
(MIBC)										
				Conc. 1		0.5		8.63		154
(PAX) @ 0.1%		10	10	Cond.	1			8.63		140
(IPX) @ 0.1%		10	10	together	min					
no frother		10	10							
				Conc. 2		2		8.61		145
CuSO4 (dry powder)			100	Cond.	3					
(PAX) @ 0.1%		10	10							
(IPX) @ 0.1%		10	10	Cond.	1					150
no frother					min					
				Conc. 3		2		8.31		169
(PAX) @ 0.1%		10	10	together	1					
(IPX) @ 0.1%		10	10	Cond.	min	Comb.				
1 drop		5 ml	5							
(MIBC)				**Conc. 4**		2.5		8.34		152
(PAX) Total			60							
(IPX) Total			60							
Frother Total			4 drops							

Table 23. Flotation mass balance: F5M2 (P_{80} = 57 μm)

With CuSO4 after 2nd Conc.

M2Test 5	TIME	MASS		ASSAYS					
Product	Min.	(g)	(%)	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)	Pb (%)
Conc. 1	0.5	17.77	1.78	226.6	46.4	0.47	39.6	0.19	0.02
Conc. 2	2	8.57	0.86	133.1	28.8	1.37	26.9	0.24	0.02
Conc. 3	4.5	8.02	0.80	21.8	1.89	2.26	10.92	0.17	0.02
Conc. 4	3	0	0.00	0.0	0.00	0	0.00	0.00	0.000
Tails		966.2	96.6	0.23	0.05	2.17	5.01	0.002	0.003
Calc Head		1,001	100.0	5.56	1.13	2.13	5.86	0.008	0.003
Actual Head		1,000	100	5.59	1.16	2.26	6.67	0.007	0.004

Cumulative

Min.	Wt. (g)	Mass Rec	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)	Pb (%)
0		0						
0.5	17.77	1.78	226.6	46.4	0.47	39.6	0.19	0.02
2.5	26.34	2.63	196.2	40.7	0.76	35.4	0.20	0.02
7.0	34.36	3.43	155.5	31.6	1.11	29.7	0.20	0.02
	34.36	3.43	155.5	31.6	1.11	29.7	0.20	0.02
	1000.6	100.0	5.56	1.13	2.13	5.86	0.008	0.003

			RECOVERIES (%)					
Min.	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu	Pb
0	17.77	1.78	72.4	73.0	0.39	11.99	40.49	11.61
0.5	8.57	0.86	20.5	21.8	0.55	3.93	24.40	5.40
2.5	8.02	0.80	3.14	1.3	0.85	1.49	16.62	4.64
7.0	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00
0.0	966.2	96.6	3.99	3.8	98.21	82.59	18.49	78.35

	Cumulative		CUMULATIVE RECOVERIES (%)					
Min.	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu	Pb
0		0	0	0	0			
0.5	17.77	1.78	72.4	73.0	0.39	11.99	40.49	11.61
2.0	26.34	2.63	92.9	94.8	0.94	15.91	64.89	17.01
4.0	34.36	3.43	96.0	96.2	1.79	17.41	81.51	21.65
7.0	34.36	3.43	96.0	96.2	1.79	17.41	81.51	21.65
	1000.6	100						

Table 24. Flotation test report: F6M2

TEST : 6 DATE: FEED: Murgor Ore # 2

OBJECTIVE: Rougher-scavenger float for gold recovery

GRINDING CONDITIONS (Denver Mill):

MILL : *MS -Denver* ROD TYPE: *50% SS*

CHARGE *~1000 g* 66.7 % Solids *Tap*

WATER: *500 ml.*

GRIND: 0.3 g Na_2CO_3

20 ml PAX+ 20 iPrpX

GRIND TIME: 15 min.

FLOAT CONDITIONS: FLOATED BY: S.K

CELL TYPE & VOLUME: *2 L - Denver Cell*

IMPELLER RPM: *1200* REPULP WATER: *Tap*

Ro GAS RATE: *3* Or air valve fully open

# of STROKES: *30/min*

REAGENTS	ml	g/Ton	STAGE	COND TIME	FLOAT TIME	pH Start	pH End	REDOX / pH: as is:	REDOX mV
Redox						as is	8.54	After 1 min stirring	287
pH (Soda ash)									
(PAX) @ 0.1%	10	10	Cond.	2					
(IPX) @ 0.1%	10	10							
3 drops			Cond.	1			8.70		195
(MIBC)									
			Conc. 1		0.5		8.66		176
(PAX) @ 0.1%	10	10	Cond.	1			8.52		144
(IPX) @ 0.1%	10	10							
no frother	10	10	together	min					
			Conc. 2		2		8.57		188
NaSH	0.3	300	Cond.	3			8.92		111
(PAX) @ 0.1%	10	10							
(IPX) @ 0.1%	10	10	Cond.	1			8.97		106
no frother				min					
			Conc. 3		2		8.86		119
NaSH	0.3	300	Cond.	3			9.03		97
(PAX) @ 0.1%	10	10							
(IPX) @ 0.1%	10	10	Cond.	1			9.05		95
1 drop MIBC	5 ml	5	together	min					
			Conc. 4		2.5		8.91		104
(PAX) Total		350							
(IPX) Total		60							
Frother (MIBC) Total		4 drops							

Table 25. Flotation mass balance: F6M2 (P_{80} = 57 μm)

With NaSH after 2nd Conc.

M2Test 6	T I M E	M A S S		A S S A Y S		
Product	Min.	(g)	(%)	Au g/tonne	S %	C (%)
Conc. 1	0.5	19.40	1.94	225.1	48.0	0.41
Conc. 2	2.0	4.93	0.49	132.4	30.3	1.18
Conc. 3	2.0	2.82	0.28	27.1	4.31	1.97
Conc. 4	2.5	7.48	0.75	11.74	1.20	1.94
Tails		964.5	96.53	0.33	0.07	2.08
Calc Head		999.2	100.0	5.51	1.17	2.04
Actual Head		1,000	100.0	5.49	1.16	2.26
Cumulative						

Min.	Wt. (g)	Mass Rec	Au g/tonne	S %	C (%)
0		0			
0.5	19.40	1.94	225.1	48.0	0.41
2.5	24.33	2.43	206.3	44.4	0.57
4.5	27.15	2.72	187.7	40.2	0.71
7.0	34.63	3.47	149.7	31.8	0.98
	999.16	100.00	5.51	1.17	2.04

			R E C O V E R I E S (%)		
Min.	Wt. (g)	Mass Rec	Au	S	C
0	19.40	1.94	79.4	79.9	0.39
0.5	4.93	0.49	11.9	12.8	0.28
2.5	2.82	0.28	1.39	1.04	0.27
4.5	7.48	0.75	1.60	0.77	0.71
7.0	964.5	96.53	5.77	5.46	98.34

	Cumulative		CUMULATIVE RECOVERIES (%)		
Min.	Wt. (g)	Mass Rec	Au	S	C
0		0	0	0	0
0.5	19.40	1.94	79.4	79.9	0.39
2.0	24.33	2.43	91.2	92.7	0.67
4.0	27.15	2.72	92.6	93.8	0.95
7.0	34.63	3.47	94.2	94.5	1.66
	999.2	100.00			

Table 26. Flotation test report: F7M2

TEST :	7				DATE:			FEED:	Murgor Ore # 2	
OBJECTIVE:		Rougher-scavenger float for gold recovery								
GRINDING CONDITIONS (Denver Mill):						FLOAT CONDITIONS:			FLOATED BY:	S.K
MILL :	*MS -Denver*	ROD TYPE:		*50% SS*		CELL TYPE & VOLUME:			*2 L - Denver Cell*	
CHARGE:	*~1000 g*	66.7	% Solids	*Tap*		IMPELLER RPM:		*1200*	REPULP WATER:	*Tap*
WATER:	*500 ml.*					Ro GAS RATE:		*3*	Or air valve fully open	
GRIND:	0.3 g Na_2CO_3	GRIND TIME:		15 min.		# of STROKES:		*30/min*		
20 ml PAX+ 20 iPrpX		ml	g/Ton	STAGE	COND	FLOAT	pH	pH	REDOX / pH: as is:	REDOX
REAGENTS					TIME	TIME	Start	End		mV
Redox							as is	8.97	After 1 min stirring	155
pH (Soda ash)										
(PAX) @ 0.1%		10	10	Cond.	2			9.11		137
(IPX) @ 0.1%		10	10	Cond.	1					
3 drops										
(MIBC)										
				Conc. 1		0.5		9.20		141
(PAX) @ 0.1%		10	10	Cond.	1			9.14		125
(IPX) @ 0.1%		10	10	together	min					
no frother										
				Conc. 2		2		8.88		138
2 drops of Fuel Oil (Emul.)				Cond.	2			9.15		
no frother				together	min					
				Conc. 3		2		9.00		
(PAX) @ 0.1%		10	10	Cond.						
(IPX) @ 0.1%		10	10	together	1			9.11		
2 drops of Fuel Oil (Emul.)					min					
1 drop MIBC										
				Conc. 4		2.5		9.00		140
(PAX) Total			50							
(IPX) Total			50							
Frother Total			4 drops							

Table 27. Flotation mass balance: F7M2 (P_{80} = 57 μm)

With Fuel Oil after 2nd Conc.						
M2Test 7	T I M E	M A S S		A S S A Y S		
Product	Min.	(g)	(%)	Au g/tonne	S %	C (%)
Conc. 1	0.5	19.22	1.92	225.7	48.3	0.37
Conc. 2	2.0	5.02	0.50	146.1	31.3	1.35
Conc. 3	2.0	3.45	0.34	22.7	4.8	2.06
Conc. 4	2.5	5.20	0.52	8.25	1.76	2.14
Tails		966.0	96.71	0.32	0.04	1.91
Calc Head		998.9	100.0	5.50	1.15	1.88
Actual Head		1,000	100.0	5.49	1.16	2.26
Cumulative						
	Min.	Wt. (g)	Mass Rec	Au g/tonne	S %	C (%)
	0		0			
	0.5	19.22	1.92	225.7	48.3	0.37
	2.5	24.23	2.43	209.3	44.8	0.57
	4.5	27.68	2.77	186.0	39.8	0.76
	7.0	32.88	3.29	157.9	33.8	0.98
		998.9	100.0	5.50	1.15	1.88
				R E C O V E R I E S (%)		
	Min.	Wt. (g)	Mass Rec	Au	S	C
	0	19.22	1.92	78.9	80.7	0.38
	0.5	5.02	0.50	13.3	13.7	0.36
	2.5	3.45	0.34	1.42	1.44	0.38
	4.5	5.20	0.52	0.78	0.80	0.59
	7.0	966.0	96.71	5.54	3.36	98.29
		Cumulative		CUMULATIVE RECOVERIES (%)		
	Min.	Wt. (g)	Mass Rec	Au	S	C
	0		0	0	0	0
	0.5	19.22	1.92	78.9	80.7	0.38
	2.0	24.23	2.43	92.3	94.4	0.74
	4.0	27.68	2.77	93.7	95.8	1.12
	7.0	32.88	3.29	94.5	96.6	1.71
		998.9	100.00			

Table 28. Flotation test report: F8M1

TEST :	8				DATE:			FEED:	Murgor Ore # 1	
OBJECTIVE:		Rougher-scavenger float for gold recovery (with lime)								
GRINDING CONDITIONS (Denver Mill):						FLOAT CONDITIONS:			FLOATED BY:	S.K
MILL :	*MS -Denver*	ROD TYPE:		*50% SS*		CELL TYPE & VOLUME:			*2 L - Denver Cell*	
CHARGE	*~1000 g*	66.7	% Solids	*Tap*		IMPELLER RPM:		*1200*	REPULP WATER:	*Tap*
WATER:	*500 ml.*					Ro GAS RATE:		*3*	Or air valve fully open	
GRIND:	0.4 g Lime	GRIND TIME:		15 min.		# of STROKES:		*30/min*		
20 ml PAX+ 20 iPrpX		ml	g/Ton	STAGE	COND	FLOAT	pH	pH	REDOX / pH: as is:	REDOX
REAGENTS					TIME	TIME	Start	End		mV
Redox							as is	8.86	After 1 min stirring	160
pH (lime)										
(PAX) @ 0.1%		10	10	Cond.	2			9.32		130
(IPX) @ 0.1%		10	10							
3 drops				Cond.	1					
(MIBC)										
				Conc. 1		0.5		9.11		130
(PAX) @ 0.1%		10	10	Cond.	1			9.03		115
(IPX) @ 0.1%		10	10	together	min					
no frother										
				Conc. 2		2		8.88		113
								9.15		108
no frother										
				Conc. 3		2		9.05		114
(PAX) @ 0.1%		10	10	Cond.	1					
(IPX) @ 0.1%		10	10	together	min			9.16		
1 drop MIBC										
				Conc. 4		2.5		9.10		95
(PAX) Total			50							
(IPX) Total			50							
Frother Total			4 drops							

Table 29. Flotation mass balance: F8M1 (P_{80} = 53 μm)

With Lime instead of Soda Ash						
M1Test 8	T I M E	M A S S		A S S A Y S		
Product	Min.	(g)	(%)	Au g/tonne	S %	C (%)
Conc. 1	0.5	10.62	1.07	258.3	45.3	0.51
Conc. 2	2.0	8.09	0.81	233.1	38.5	0.84
Conc. 3	2.0	3.59	0.36	65.8	10.7	1.8
Conc. 4	2.5	3.00	0.30	11.43	1.98	2.21
Tails		968.0	97.45	0.24	0.04	2.09
Calc Head		993.3	100.0	5.17	0.88	2.06
Actual Head		1,000	100.0	5.18	0.82	2.26
Cumulative						
	Min.	Wt. (g)	Mass Rec	Au g/tonne	S %	C (%)
	0		0			
	0.5	10.62	1.07	258.3	45.3	0.51
	2.5	18.71	1.88	247.4	42.4	0.65
	4.5	22.30	2.25	218.1	37.3	0.84
	7.0	25.30	2.55	193.6	33.1	1.00
		993.3	100.0	5.17	0.88	2.06
				R E C O V E R I E S (%)		
	Min.	Wt. (g)	Mass Rec	Au	S	C
	0	10.62	1.07	53.4	54.9	0.26
	0.5	8.09	0.81	36.7	35.6	0.33
	2.5	3.59	0.36	4.61	4.39	0.32
	4.5	3.00	0.30	0.67	0.68	0.32
	7.0	968.0	97.5	4.58	4.42	98.76
		Cumulative		CUMULATIVE RECOVERIES (%)		
	Min.	Wt. (g)	Mass Rec	Au	S	C
	0		0	0	0	0
	0.5	10.62	1.07	53.4	54.9	0.26
	2.0	18.71	1.88	90.1	90.5	0.60
	4.0	22.30	2.25	94.7	94.9	0.91
	7.0	25.30	2.55	95.4	95.6	1.24
		993.3	100.0			

APPENDIX C: Cyanidation data

Table 30. Cyanidation C1&2(M1) (P_{80} = 137 µm)

Cyanidation Test Report

Test 1A	Murgor 1			
Feed	503	g		
Grind:	7	minutes	Ground @ 67% solids using a Denver Rod Mill as in Flotation tests	
Grind (P_{80}):	137	µm	& Repulped for cyanidation at about 33 % solids	
pH (initial)	8.13			
pH (after)	10-11			
Solution Vol.	1010	mL	NaCN purity ≈	95%
NaCN	1	g/L NaCN		

Time (hrs)	Added (g)				Residual (g)		Consumed (g)		pH	
	Actual		Equivalent (pure)							
	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	Start	End
0-1	1.06	0.20	1.01	0.151	1.00	0.1	0.01	0.051	10.75	10.33
1.-2	0	0.03	0.01	0.023	0.95		0.06	0.023	10.5	10.69
2.-3	0.06		0.06		0.93		0.085		10.69	10.65
3.-5	0	0.05	0	0.038	1.00	0	0	0.038	10.65	10.55
5.-8	0		0		1.00		0		10.69	10.62
8.-18	0		0		1.00		0		10.62	10.34
18-24	0.09	0.10	0.085	0.075	0.98	0.05	0.035	0.025	10.98	10.9
24-48	0.03	0.10	0.025	0.075	0.95	0	0.06	0.075	10.9	10.46
Total	1.24	0.48	1.19	0.36	0.95	0.15	0.25	0.21		

Reagent Consumption (kg/tonne of feed)	NaCN	**0.50**
	CaO	**0.42**

Cyanidation Test Report

Test 2A	Murgor 1			
Feed	503	g		
Grind:	7	minutes	Ground @ 67% solids using a Denver Rod Mill as in Flotation tests	
Grind (P_{80}):	137	µm	& Repulped for cyanidation at about 33 % solids	
pH (initial)	8.15			
pH (after)	10-11			
Solution Vol.	1001	mL	NaCN purity ≈	95%
NaCN	2	g/L NaCN		

Time (hrs)	Added (g)				Residual (g)		Consumed (g)		pH	
	Actual		Equivalent (pure)							
	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	Start	End
0-1	2.11	0.20	2.00	0.151	1.90	0.10	0.10	0.051	10.85	10.63
1.-2	0.11		0.10		1.85		0.15		10.63	10.60
2.-3	0.16		0.15		1.90		0.10		10.60	10.59
3.-5	0.11		0.10		2.00		0.00		10.59	10.54
5.-8	0				1.95		0.05		10.67	10.61
8.-18	0.05		0.05		1.95		0.05		10.61	10.54
18-24	0.05		0.05	0.000	1.85		0.15		10.61	10.50
24-48	0.16	0.03	0.15	0.023	1.85		0.15	0.023	10.63	10.44
Total	2.74	0.23	2.60	0.173	1.85	0.100	0.75	0.073		

Reagent Consumption (kg/tonne of feed)	NaCN	**1.49**
	CaO	**0.15**

Table 31. Cyanidation C3&4(M1) (P_{80} = 92 μm)

Cyanidation Test Report

Test 3A	Murgor 1		
Feed	502	g	
Grind:	10	minutes	Ground @ 67% solids using a Denver Rod Mill as in Flotation tests
Grind (P_{80}):	92	μm	& Repulped for cyanidation at about 33 % solids
pH (initial)	8.20		
pH (after)	10-11		
Solution Vol.	1000	mL	NaCN purity ≈ 95%
NaCN	1	g/L NaCN	

Time (hrs)	Added (g)				Residual (g)		Consumed (g)		pH	
	Actual		Equivalent (pure)							
	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	Start	End
0-1	1.05	0.20	1.00	0.151	0.98	0.10	0.03	0.051	10.68	10.47
1.-2	0.03		0.03		0.95		0.05		10.56	10.64
2.-3	0.05		0.05		1.00		0.00		10.64	10.47
3.-5	0.00		0.00		0.95		0.05		10.47	10.37
5.-8	0.05		0.05		1.00		0.00		10.65	10.62
8.-18	0.00		0.00		0.95		0.05		10.62	10.31
18-24	0.05	0.1	0.05	0.075	0.95	0.05	0.05	0.025	10.52	10.40
24-48	0.05	0.1	0.05	0.075	0.95	0.00	0.05	0.075	10.78	10.34
Total	1.29	0.40	1.23	0.302	0.95	0.15	0.28	0.152		

Reagent Consumption (kg/tonne of feed)	NaCN	**0.55**
	CaO	**0.30**

Cyanidation Test Report

Test 4A	Murgor 1		
Feed	502	g	
Grind:	10	minutes	Ground @ 67% solids using a Denver Rod Mill as in Flotation tests
Grind (P_{80}):	92	μm	& Repulped for cyanidation at about 33 % solids
pH (initial)	8.21		
pH (after)	10-11		
Solution Vol.	1000	mL	NaCN purity ≈ 95%
NaCN	2	g/L NaCN	

Time (hrs)	Added (g)				Residual (g)		Consumed (g)		pH	
	Actual		Equivalent (pure)							
	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	Start	End
0-1	2.11	0.20	2.00	0.151	1.95	0.10	0.05	0.051	10.68	10.50
1.-2	0.05		0.05		1.90		0.10		10.59	10.65
2.-3	0.11		0.10		1.95		0.05		10.65	10.63
3.-5	0.05		0.05		1.95		0.05		10.53	10.48
5.-8	0.05	0.10	0.05	0.075	1.93	0.05	0.08	0.025	10.60	10.52
8.-18	0.08		0.08		1.95		0.05		10.60	10.55
18-24	0.05		0.05		1.85		0.15		10.55	10.44
24-48	0.16	0.10	0.15	0.075	1.85	0.05	0.15	0.025	10.72	10.55
Total	2.66	0.40	2.53	0.302	1.85	0.200	0.68	0.102		

Reagent Consumption (kg/tonne of feed)	NaCN	**1.34**
	CaO	**0.20**

Table 32. Cyanidation C5&6(M1) (P_{80} = 53 µm)

Cyanidation Test Report

Test 5A Murgor 1
Feed 506 g
Grind: 15 minutes Ground @ 67% solids using a Denver Rod Mill as in Flotation tests
Grind (P_{80}): 53 µm & Repulped for cyanidation at about 33 % solids
pH (initial) 8.24
pH (after) 10-11
Solution Vol. 1000 mL NaCN purity ≈ 95%
NaCN 1 g/L NaCN

Time (hrs)	Added (g)				Residual (g)		Consumed (g)		pH	
	Actual		Equivalent (pure)							
	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	Start	End
0-1	1.05	0.20	1.00	0.151	0.95	0.10	0.05	0.051	10.60	10.43
1.-2	0.05		0.05		0.95		0.05		10.76	10.77
2.-3	0.05		0.05		0.98		0.03		10.77	10.65
3.-5	0.03		0.03		0.95		0.05		10.65	10.48
5.-8	0.05		0.05		1.00		0.00		10.48	10.38
8.-18	0.00		0.00	0.000	0.93		0.08		10.65	10.36
18-24	0.08	0.1	0.08	0.075	0.98	0.05	0.03	0.025	10.54	10.45
24-48	0.03	0.1	0.03	0.075	0.98	0.05	0.03	0.025	10.81	10.40
Total	1.34	0.40	1.28	0.302	0.98	0.20	0.30	0.102		

Reagent Consumption (kg/tonne of feed)	NaCN	**0.60**
	CaO	**0.20**

Cyanidation Test Report

Test 6A Murgor 1
Feed 506 g
Grind: 15 minutes Ground @ 67% solids using a Denver Rod Mill as in Flotation tests
Grind (P_{80}): 53 µm & Repulped for cyanidation at about 33 % solids
pH (initial) 8.21
pH (after) 10-11
Solution Vol. 1000 mL NaCN purity ≈ 95%
NaCN 2 g/L NaCN

Time (hrs)	Added (g)				Residual (g)		Consumed (g)		pH	
	Actual		Equivalent (pure)							
	NaCN	Ca(OH)2	NaCN	CaO	NaCN	CaO	NaCN	CaO	Start	End
0-1	2.11	0.20	2.00	0.151	1.95	0.10	0.05	0.051	10.66	10.52
1.-2	0.05	0.1	0.05	0.075	1.85	0.05	0.15	0.025	10.52	10.48
2.-3	0.16		0.15		1.95		0.05		10.48	10.47
3.-5	0.05		0.05		1.90		0.10		10.48	10.41
5.-8	0.11	0.10	0.10	0.075	2.00	0.05	0.00	0.025	10.52	10.48
8.-18	0.00	0.10	0.00	0.075	1.90	0.05	0.10	0.025	10.69	10.57
18-24	0.11		0.10		1.90		0.10		10.57	10.52
24-48	0.11		0.10		1.85		0.15		10.79	10.42
Total	2.68	0.50	2.55	0.377	1.85	0.250	0.70	0.127		

Reagent Consumption (kg/tonne of feed)	NaCN	**1.38**
	CaO	**0.25**

Table 33. Percent extraction of gold and sulphur based on head & tail assays

Murgor 1	Particle	Sodium	Tailing Assays		Extraction	
Test Designation	Size, P_{80} (microns)	Cyanide (g /L)	Au (g/Tonne)	S (%)	Au (%)	S (%)
T-A1	137	1	0.31	0.23	94.1	71.8
T-A2	137	2	0.27	0.21	94.9	73.8
T-A3	92	1	0.21	0.21	96.0	73.4
T-A4	92	2	0.18	0.15	96.5	81.3
T-A5	53	1	0.15	0.23	97.2	71.3
T-A6	53	2	0.13	0.22	97.7	72.5

Feed Assays:	Au	5.18 g/tonne
	S	0.80 %

Table 34. Cyanidation data and analysis (Test 2A)

Test 2A

Sample Murgor 1

Particle Size, P_{80}	137	µm
Initial solids	503	g
Initial Vol of CN Soln.	1001	ml
Pulp Density	33.4	% solids (wt.)
Cyanide Concentration	2.0	g/L

		Au in sampling solution					
Time (h)	Au (mg)	Sample Vol. (ml)	Sample Vol. (ml) Cum.	Au (mg)	Au (mg) Cum.	Au (mg) ext.	Extraction (%)
0	0	0	0	0	0	0	0
1	1.02	20	20	0.020	0.020	1.02	39.8
2	1.61	20	40	0.032	0.053	1.60	62.3
3	1.97	20	60	0.039	0.092	1.95	75.8
5	2.24	20	80	0.045	0.137	2.20	85.7
8	2.36	20	100	0.047	0.184	2.31	90.0
18	2.45	20	120	0.049	0.233	2.39	93.2
24	2.47	20	140	0.049	0.282	2.41	93.9
48	2.50	20	160	0.050	0.332	2.43	94.9

Total sample for anayses (ml)	160	0.33
Preg. Soln at the end	841	2.10
Tails at the end (g)	498	
Tails assay (g/tonne)	0.27	0.13
Au in reconstituted feed (mg)		2.57
Calculated head (g/tonne)		5.10
Actual head (g/tonne)		5.18

Table 35. Cyanidation data and analysis (Test 3A)

Test 3A		
Sample	Murgor 1	
Particle Size, P_{80}	92	μm
Initial solids	502	g
Initial Vol of CN Soln.	1000	ml
Pulp Density	33	% solids (wt.)
Cyanide Concentration	1.0	g/L

		Au in sampling solution					
Time (h)	Au (mg)	Sample Vol. (ml)	Sample Vol. (ml) Cum.	Au (mg)	Au (mg) Cum.	Au (mg) ext.	Extraction (%)
0	0	0	0	0	0	0	0
1	0.72	20	20	0.014	0.014	0.72	27.4
2	1.31	20	40	0.026	0.041	1.30	49.8
3	1.68	20	60	0.034	0.074	1.66	63.5
5	2.22	20	80	0.044	0.119	2.16	82.9
8	2.54	20	100	0.051	0.169	2.46	94.1
18	2.58	20	120	0.052	0.221	2.49	95.5
24	2.59	20	140	0.052	0.273	2.50	95.7
48	2.59	20	160	0.052	0.325	2.50	96.0

Total sample for anayses (ml)	160	0.32
Preg. Soln at the end	840	2.18
Tails at the end (g)	499	
Tails assay (g/tonne)	0.21	0.10
Au in reconstituted feed (mg)		2.61
Calculated head (g/tonne)		5.20
Actual head (g/tonne)		5.18

Table 36. Cyanidation data and analysis (Test 4A)

Test 4A		
Sample	Murgor 1	
Particle Size, P_{80}	92	µm
Initial solids	502	g
Initial Vol of CN Soln.	1000	ml
Pulp Density	33	% solids (wt.)
Cyanide Concentration	2.0	g/L

		Au in sampling solution					
Time (h)	Au (mg)	Sample Vol. (ml)	Sample Vol. (ml) Cum.	Au (mg)	Au (mg) Cum.	Au (mg) ext.	Extraction (%)
0	0	0	0	0	0	0	0
1	1.29	20	20	0.026	0.026	1.29	48.6
2	1.96	20	40	0.039	0.065	1.95	73.4
3	2.30	20	60	0.046	0.111	2.27	85.7
5	2.51	20	80	0.050	0.161	2.47	93.1
8	2.60	20	100	0.052	0.213	2.55	96.2
18	2.59	20	120	0.052	0.265	2.54	95.9
24	2.60	20	140	0.052	0.317	2.55	96.2
48	2.61	20	160	0.052	0.369	2.56	96.5

Total sample for anayses (ml)	160	0.37
Preg. Soln at the end	840	2.19
Tails at the end (g)	498	
Tails assay (g/tonne)	0.18	0.09
Au in reconstituted feed (mg)		2.65
Calculated head (g/tonne)		5.29
Actual head (g/tonne)		5.18

Table 37. Cyanidation data and analysis (Test 5A)

Test 5A		
Sample	Murgor 1	
Particle Size, P_{80}	53	µm
Initial solids	506	g
Initial Vol of CN Soln.	1000	ml
Pulp Density	34	% solids (wt.)
Cyanide Concentration	1.0	g/L

		Au in sampling solution					
Time (h)	Au (mg)	Sample Vol. (ml)	Sample Vol. (ml) Cum.	Au (mg)	Au (mg) Cum.	Au (mg) ext.	Extraction (%)
0	0	0	0	0	0	0	0
1	0.78	20	20	0.016	0.016	0.78	30.0
2	1.54	20	40	0.031	0.046	1.52	58.6
3	1.88	20	60	0.038	0.084	1.85	71.1
5	2.37	20	80	0.047	0.131	2.31	88.8
8	2.49	20	100	0.050	0.181	2.42	93.0
18	2.55	20	120	0.051	0.232	2.48	95.1
24	2.58	20	140	0.052	0.284	2.50	96.1
48	2.61	20	160	0.052	0.336	2.53	97.1

Total sample for anayses (ml)	160	0.34
Preg. Soln at the end	840	2.19
Tails at the end (g)	500	
Tails assay (g/tonne)	0.15	0.08
Au in reconstituted feed (mg)		2.60
Calculated head (g/tonne)		5.15
Actual head (g/tonne)		5.18

Table 38. Cyanidation data and analysis (Test 6A)

Test 6A		
Sample	Murgor 1	
Particle Size, P_{80}	53	µm
Initial solids	506	g
Initial Vol of CN Soln.	1000	ml
Pulp Density	34	% solids (wt.)
Cyanide Concentration	2.0	g/L

		Au in sampling solution					
Time (h)	Au (mg)	Sample Vol. (ml)	Sample Vol. (ml) Cum.	Au (mg)	Au (mg) Cum.	Au (mg) ext.	Extraction (%)
0	0	0	0	0	0	0	0
1	1.19	20	20	0.024	0.024	1.19	45.1
2	1.91	20	40	0.038	0.062	1.90	71.8
3	2.26	20	60	0.045	0.107	2.23	84.5
5	2.55	20	80	0.051	0.158	2.50	94.8
8	2.57	20	100	0.051	0.210	2.52	95.5
18	2.60	20	120	0.052	0.262	2.55	96.5
24	2.62	20	140	0.052	0.314	2.57	97.2
48	2.63	20	160	0.053	0.367	2.58	97.5

Total sample for anayses (ml)	160	0.37
Preg. Soln at the end	840	2.21
Tails at the end (g)	502	
Tails assay (g/tonne)	0.13	0.07
Au in reconstituted feed (mg)		2.64
Calculated head (g/tonne)		5.22
Actual head (g/tonne)		5.18

APPENDIX D: Tabulated data on gravity separation

Table 39. Knelson gravity separation mass balance (K1M2-P$_{80}$ 150 @ 4psi)

M2Test 1	MASS		ASSAYS				
Product	(g)	(%)	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M2 K1 C	93.62	28.28	18.22	3.52	1.69	11.1	0.011
M2 K1 M	28.53	8.62	1.33	0.27	2.01	4.15	0.005
M2 K1 T	208.84	63.10	0.73	0.11	2.29	5.09	0.007
Calc Head	330.99	100.0	5.73	1.09	2.10	6.70	0.008
Actual Head	336.0			1.10	2.30	6.67	0.007

Cumulative	Wt. (g)	Mass Rec	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M2 K1 C	93.62	28.28	18.2	3.5	1.69	11.1	0.011
M2 K1 M	122.15	36.90	14.3	2.8	1.76	9.5	0.009
M2 K1 T	331.0	100.0	5.73	1.09	2.10	6.70	0.008

			RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
M2 K1 C	93.62	28.28	89.9	91.5	22.80	46.72	39.01
M2 K1 M	28.53	8.62	2.00	2.1	8.27	5.33	5.68
M2 K1 T	208.84	63.10	8.08	6.4	68.93	47.94	55.30
	331.0	100.0	100.0	100.0	100.0	100.0	100.0

	Cumulative		CUMULATIVE RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
M2 K1 C	93.62	28.28	89.9	91.5	22.80	46.72	39.01
M2 K1 M	122.15	36.90	91.9	93.6	31.07	52.06	44.70
M2 K1 T	331.0	100.0	100.0	100.0	100.0	100.0	100.0

Table 40. Knelson gravity separation mass balance (K2M2-P_{80} 150 @ 9psi)

M2Test 2	MASS		ASSAYS				
Product	(g)	(%)	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M2 K2 C	19.91	5.95	70.00	11.8	1.69	26.4	0.013
M2 K2 M	18.84	5.63	2.49	0.41	1.76	4.80	0.005
M2 K2 T	295.91	88.42	1.64	0.20	2.19	5.01	0.007
Calc Head	334.66	100.0	5.76	0.90	2.14	6.27	0.007
Actual Head	336.0			1.10	2.27	6.67	0.007
Cumulative							
	Wt. (g)	Mass Rec	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M2 K2 C	19.91	5.95	70.0	11.8	1.69	26.4	0.013
M2 K2 M	38.75	11.58	37.2	6.3	1.72	15.9	0.009
M2 K2 T	334.7	100.0	5.76	0.90	2.14	6.27	0.007

			RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
M2 K2 C	19.91	5.95	72.3	77.8	4.71	25.06	10.74
M2 K2 M	18.84	5.63	2.44	2.6	4.64	4.31	3.83
M2 K2 T	295.91	88.42	25.24	19.6	90.65	70.63	85.43
	334.7	100.0	100.0	100.0	100.0	100.0	100.0

	Cumulative		CUMULATIVE RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
		0	0	0	0		
M2 K2 C	19.91	5.95	72.3	77.8	4.71	25.06	10.74
M2 K2 M	38.75	11.58	74.8	80.4	9.35	29.37	14.57
M2 K2 T	334.7	100.0	100.0	100.0	100.0	100.0	100.0

Table 41. Knelson gravity separation mass balance (K2M2-P_{80} 150 @ 13psi)

M2Test 3	MASS		ASSAYS				
Product	(g)	(%)	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M2 K3 C	13.89	4.16	81.95	12.2	1.33	26.2	0.014
M2 K3 M	27.081	8.11	3.81	1.57	2.01	7.38	0.007
M2 K3 T	292.92	87.73	2.01	0.20	2.18	5.36	0.007
Calc Head	333.89	100.0	5.48	0.81	2.13	6.40	0.007
Actual Head	336.0			0.80	2.30	6.67	0.007

Cumulative	Wt. (g)	Mass Rec	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M2 K3 C	13.89	4.16	81.9	12.2	1.33	26.2	0.014
M2 K3 M	40.971	12.27	30.3	5.2	1.78	13.8	0.009
M2 K3 T	333.9	100.0	5.48	0.81	2.13	6.40	0.007

			RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
M2 K3 C	13.89	4.16	62.2	62.6	2.60	17.07	8.19
M2 K3 M	27.081	8.11	5.63	15.7	7.65	9.36	7.51
M2 K3 T	292.92	87.73	32.16	21.7	89.75	73.57	84.31
	333.9	100.0	100.0	100.0	100.0	100.0	100.0

	Cumulative		CUMULATIVE RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
M2 K3 C	13.89	4.16	62.2	62.6	2.60	17.07	8.19
M2 K3 M	40.971	12.27	67.8	78.3	10.25	26.43	15.69
M2 K3 T	333.9	100.0	100.0	100.0	100.0	100.0	100.0

Table 42. Knelson gravity separation mass balance (K4M1-P_{80} 92 @ 3 psi)

M1Test 4	MASS		ASSAYS				
Product	(g)	(%)	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M1 K1 C	62.18	18.54	25.98	3.48	1.55	13.9	0.012
M1 K1 M	16.89	5.04	0.98	0.18	1.79	3.38	0.005
M1 K1 T	256.34	76.43	0.53	0.20	2.21	5.22	0.006
Calc Head	335.41	100.0	5.27	0.81	2.07	6.73	0.007
Actual Head	336.0			0.76	2.30	6.64	0.007

Cumulative	Wt. (g)	Mass Rec	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M1 K1 C	62.18	18.54	26.0	3.5	1.55	13.9	0.012
M1 K1 M	79.07	23.57	20.6	2.8	1.60	11.6	0.010
M1 K1 T	335.4	100.0	5.27	0.81	2.07	6.73	0.007

			RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
M1 K1 C	62.18	18.54	91.3	79.9	13.91	38.19	29.60
M1 K1 M	16.89	5.04	0.94	1.1	4.36	2.53	3.25
M1 K1 T	256.34	76.43	7.74	18.9	81.73	59.28	67.15
	335.4	100.0	100.0	100.0	100.0	100.0	100.0

	Cumulative		CUMULATIVE RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
M1 K1 C	62.18	18.54	91.3	79.9	13.91	38.19	29.60
M1 K1 M	79.07	23.57	92.3	81.1	18.27	40.72	32.85
M1 K1 T	335.4	100.0	100.0	100.0	100.0	100.0	100.0

Table 43. Knelson gravity separation mass balance (K5M1-P_{80} 92 @ 6 psi)

M1Test 5	MASS		ASSAYS				
Product	(g)	(%)	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M1 K2 C	26.83	8.12	50.43	7.85	1.26	22.3	0.017
M1 K2 M	52.92	16.01	1.71	0.26	1.56	4.59	0.005
M1 K2 T	250.75	75.87	1.07	0.11	2.32	5.50	0.007
Calc Head	330.50	100.0	5.18	0.76	2.11	6.71	0.007
Actual Head	336.0			0.76	2.30	6.67	0.007

Cumulative	Wt. (g)	Mass Rec	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M1 K2 C	26.83	8.12	50.4	7.9	1.26	22.3	0.017
M1 K2 M	79.75	24.13	18.1	2.8	1.46	10.5	0.009
M1 K2 T	330.5	100.0	5.18	0.76	2.11	6.71	0.007

			RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
M1 K2 C	26.83	8.12	79.0	83.6	4.84	26.95	18.46
M1 K2 M	52.92	16.01	5.29	5.5	11.83	10.96	10.95
M1 K2 T	250.75	75.87	15.70	10.9	83.33	62.09	70.60
	330.5	100.0	100.0	100.0	100.0	100.0	100.0

	Cumulative		CUMULATIVE RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
M1 K2 C	26.83	8.12	79.0	83.6	4.84	26.95	18.46
M1 K2 M	79.75	24.13	84.3	89.1	16.67	37.91	29.40
M1 K2 T	330.5	100.0	100.0	100.0	100.0	100.0	100.0

Table 44. Knelson gravity separation mass balance (K6M1-P_{80} 92 @ 9 psi)

M1Test 6	MASS		ASSAYS				
Product	(g)	(%)	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M1 K3 C	13.16	3.93	85.71	12.8	0.85	31.2	0.022
M1 K3 M	31.22	9.31	6.74	0.96	1.64	5.85	0.007
M1 K3 T	290.9	86.76	1.55	0.23	2.33	5.70	0.007
Calc Head	335.28	100.0	5.34	0.79	2.21	6.71	0.007
Actual Head	336.0			0.76	2.30	6.67	0.007

Cumulative	Wt. (g)	Mass Rec	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M1 K3 C	13.16	3.93	85.7	12.8	0.85	31.2	0.022
M1 K3 M	44.38	13.24	30.2	4.5	1.41	13.4	0.012
M1 K3 T	335.3	100.0	5.34	0.79	2.21	6.71	0.007

			RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
M1 K3 C	13.16	3.93	63.0	63.5	1.51	18.26	11.80
M1 K3 M	31.22	9.31	11.76	11.3	6.92	8.11	9.31
M1 K3 T	290.9	86.76	25.21	25.2	91.57	73.64	78.89
	335.3	100.0	100.0	100.0	100.0	100.0	100.0

	Cumulative		CUMULATIVE RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
M1 K3 C	13.16	3.93	63.0	63.5	1.51	18.26	11.80
M1 K3 M	44.38	13.24	74.8	74.8	8.43	26.36	21.11
M1 K3 T	335.3	100.0	100.0	100.0	100.0	100.0	100.0

Table 45. Knelson gravity separation mass balance (K7M1-P_{80} 137 @ 4 psi)

M1Test 7	MASS		ASSAYS				
Product	(g)	(%)	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M1 K7 C	61.12	19.21	22.92	2.84	1.53	11.7	0.012
M1 K7 M	38.89	12.22	3.36	0.185	1.78	4.78	0.007
M1 K7 T	218.12	68.56	0.60	0.34	2.57	5.53	0.007
Calc Head	318.13	100.0	5.22	0.80	2.27	6.64	0.008
Actual Head	322.5			0.76	2.35	6.64	0.007

Cumulative	Wt. (g)	Mass Rec	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M1 K7 C	61.12	19.21	22.9	2.8	1.53	11.7	0.012
M1 K7 M	100.01	31.44	15.3	1.8	1.63	9.0	0.010
M1 K7 T	318.1	100.0	5.22	0.80	2.27	6.64	0.008

			RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
M1 K7 C	61.12	19.21	84.3	68.0	12.93	34.02	29.67
M1 K7 M	38.89	12.22	7.86	2.8	9.57	8.82	10.35
M1 K7 T	218.12	68.56	7.82	29.2	77.50	57.17	59.99
	318.1	100.0	100.0	100.0	100.0	100.0	100.0

	Cumulative		CUMULATIVE RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
M1 K7 C	61.12	19.21	84.3	68.0	12.93	34.02	29.67
M1 K7 M	100.0	31.44	92.2	70.8	22.50	42.83	40.01
M1 K7 T	318.1	100.0	100.0	100.0	100.0	100.0	100.0

Table 46. Knelson gravity separation mass balance (K8M1-P_{80} 137 @ 7.5 psi)

M1Test 8	MASS		ASSAYS				
Product	(g)	(%)	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M1 K8 C	47.91	15.18	27.79	3.52	1.51	13.6	0.012
M1 K8 M	20.36	6.45	2.60	0.18	1.71	4.40	0.006
M1 K8 T	247.37	78.37	0.96	0.10	2.41	5.46	0.007
Calc Head	315.64	100.0	5.14	0.62	2.23	6.63	0.008
Actual Head	322.5			0.76	2.35	6.64	0.007

Cumulative	Wt. (g)	Mass Rec	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M1 K8 C	47.91	15.18	27.8	3.5	1.51	13.6	0.012
M1 K8 M	68.27	21.63	20.3	2.5	1.57	10.8	0.011
M1 K8 T	315.6	100.0	5.14	0.62	2.23	6.63	0.008

			RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
M1 K8 C	47.91	15.18	82.1	85.6	10.29	31.12	24.81
M1 K8 M	20.36	6.45	3.26	1.9	4.95	4.29	5.15
M1 K8 T	247.37	78.37	14.67	12.6	84.76	64.60	70.04
	315.6	100.0	100.0	100.0	100.0	100.0	100.0

	Cumulative		CUMULATIVE RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
M1 K8 C	47.91	15.18	82.1	85.6	10.29	31.12	24.81
M1 K8 M	68.3	21.63	85.3	87.4	15.24	35.40	29.96
M1 K8 T	315.6	100.0	100.0	100.0	100.0	100.0	100.0

Table 47. Knelson gravity separation mass balance (K9M1-P_{80} 92 @ 4 psi)

M1Test 9	MASS		ASSAYS				
Product	(g)	(%)	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M1 K9 C	71.16	22.28	16.89	2.71	1.75	4.79	0.010
M1 K9 M	31.41	9.84	4.91	0.69	2.10	3.25	0.009
M1 Mags	18.6	5.83	10.04	0.96	1.19	48.4	0.010
M1 K9 T	198.16	62.05	0.62	0.09	2.59	4.48	0.007
Calc Head	319.4	100.0	5.21	0.78	2.27	6.99	0.008
Actual Head	322.50	100.0		0.76	2.35	6.67	0.007

Cumulative	Wt. (g)	Mass Rec	Au g/tonne	S %	C (%)	Fe (%)	Cu (%)
M1 K9 C	71.16	22.28	16.9	2.71	1.75	4.79	0.010
M1 K9 M	102.57	32.12	13.2	2.09	1.86	4.32	0.010
M1 Mags	319.35	37.95	12.7	1.92	1.75	11.10	0.010
M1 K9 T	300.73	100.0	5.21	0.78	2.27	6.99	0.008

			RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
M1 K9 C	71.16	22.28	72.2	77.1	17.16	15.3	26.7
M1 K9 M	31.41	9.84	9.27	8.66	9.09	4.57	10.8
M1 Mags	18.6	5.83	11.23	7.14	3.05	40.4	7.24
M1 K9 T	198.16	62.05	7.32	7.13	70.70	39.8	55.2
	319.35	100.0	100.0	100.0	100.0	100.0	100.0

	Cumulative		CUMULATIVE RECOVERIES (%)				
	Wt. (g)	Mass Rec	Au	S	C	Fe	Cu
M1 K9 C	71.16	22.28	72.2	77.1	17.16	15.28	26.7
M1 K9 M	102.6	32.12	81.5	85.7	26.24	19.85	37.5
M1 Mags	121.2	37.95	92.7	92.9	29.30	60.23	44.8
M1 K9 T	319.4	105.83	100.0	100.0	100.0	100.0	100.0