AGNICO EAGLE MINES LTD
Form 6-K
May 20, 2003

Form 6-K

SECURITIES AND EXCHANGE COMMISSION

Washington D.C. 20549

Report of Foreign Issuer

Pursuant to Rule 13a-16 or 15d-16 of

The Securities Exchange Act of 1934

For the Month of May 2003

Agnico-Eagle Mines Limited

(Translation of registrant's name into English)

145 King Street East, Suite 500, Toronto, Ontario M5C 2Y7

[Indicate by check mark whether the registrant files or will file annual reports under cover Form 20F or Form 40-F.]

Form 20-F X Form 40-F _______

[Indicate by check mark whether the registrant by furnishing the information contained in this Form is also thereby furnishing the information to the Commission pursuant to Rule 12g3-2(b) under the Securities Exchange Act of 1934.

Yes _______ No X

[If "YES" is marked, indicate below the file number assigned to the registrant in connection with Rule 12g3-2(b):82-____________

-2-

SIGNATURE

Pursuant to the requirements of the Securities Exchange Act of 1934, the registrant has duly caused this report to be signed on its behalf by the undersigned, thereunto duly authorized.

AGNICO-EAGLE MINES LIMITED

Date: May 20, 2003 By: (signed) David Garofalo

Vice-President, Finance and
Chief Financial Officer

2003 LARONDE MINERAL RESOURCE & MINERAL RESERVE ESTIMATE

AGNICO-EAGLE MINES LTD. LARONDE DIVISION

Guy Gosselin, P. Eng., P. Geo.
Chief Geologist
Agnico-Eagle Mines Ltd., Laronde Division
Preissac, Quebec
Effective February 19th, 2003
Dated May 12th, 2003

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3. SUMMARY

Since 1988, Agnico-Eagle Mines Ltd., through its Laronde division, has operated a mine-mill complex near the village of Preissac, north western Quebec. Accountable (net of smelter charge) production to December 31st 2002 has been 2.15 million ounces of gold, 9.5 million ounces of silver, 42.3 kilo tonnes of copper and 134.3 kilo tonnes of zinc from 12.0 million tonnes of ore.

In 2003, all the mineral reserves and most of the mineral resources at Laronde are located near the Penna shaft. The reserves and resources occurs as several sulphide-rich lenses which are found along five different stratigraphic horizons: 6, 7, 20 North Gold, 20 North Zinc and 20 South.

The 2003 mineral resources and mineral reserves estimate at Laronde was based on the same parameters that were used in 2001 and 2002 estimate: 300$/ounce gold, 5$/ounce silver, 0.80$/pound copper, 0.50$/pound zinc, 1.47 $US/$C exchange rate and total mining and milling costs that varied between 39$C/tonne to 59$C/tonne depending on the zone. The mineral reserve and mineral resource estimate was estimated using either inverse-distance block modelling techniques or the polygonal method.

Total proven mineral reserves are estimated to be 7.232 million tonnes grading 2.68 g/t gold, 97.59 g/t silver, 0.39% copper and 4.95% zinc. Total probable mineral reserves are estimated at 30.590 million tonnes grading 3.45 g/t gold, 63.19 g/t silver, 0.37% copper and 2.93% zinc. The total indicated mineral resources are estimated to be 0.588 million tonnes at a grade of 3.94 g/t gold, 14.85 g/t silver, 0.17% copper and 0.55% zinc. Finally, the total inferred mineral resources at the Penna shaft are estimated to be 20.892 million tonnes grading 5.92 g/t gold, 13.02 g/t silver, 0.33% copper and 0.08% zinc.

4. INTRODUCTION AND TERMS OF REFERENCE

This document present the year 2003 mineral reserves and mineral resources estimate, for Agnico-Eagle Mines Ltd.'s Laronde Division. The estimate is presented in a database report format (file no. RapRes03-01.mdb) and are summarised by category in the final tabulation. The mining reserves and mineral resources are calculated in metric S.I. units and are converted to the Imperial System in the final tabulation. The outlines of the reserve and resource blocks are displayed by zone on separate AutoCAD format longitudinal sections and are also summarised on a composite longitudinal plan (drawing no. LONG2003.dwg).

The mineral reserves and mineral resources estimate for the Laronde Division, present inventory information (with the exception of the reserves and resources located below the bottom of the Penna Shaft) which is current as of December 31st 2002. Blocks of mineral resources outlined at shafts no. 1 and no. 2 are described in the report but are excluded in the final tabulation of mining reserves and resources. The mining mineral reserves and mineral resources calculated for the Penna shaft however includes results from the current exploration diamond-drilling program (as recent as February 1st 2003) taking place on level 215 in the western exploration drift.

Total proven & probable reserves and indicated resources at Laronde are estimated to be 38.4 million tonnes grading 3.32 g/t gold, 68.92 g/t silver, 0.37% copper and 3.27% zinc and contains 4.097 million ounces of gold (Table 1). Inferred minerals resources stand at 20.9 million tonnes grading 5.92 g/t gold, 13.02 g/t silver, 0.33% copper and 0.08% zinc and contains 3.978 million ounces of gold (Table 2).

The 2003 mineral reserves and mineral resources estimate and all the information presented in this report are the responsibility of the geology department at the Laronde Division of Agnico-Eagle Mines Ltd. The report was prepared under the direction of Guy Gosselin P. Eng., P. Geol., the chief geologist at the Laronde Division, who is fully qualified per the standards outlined in the National Instrument 43-101. The estimate and report have been reviewed and verified as being compliant with National Instrument 43-101 Standards of Disclosure for Mineral Projects and form 43-101F1 by Marc H. Legault P. Eng., Manager of project evaluation for Agnico-Eagle mines Ltd. and Qualified Person.

The results of the 2003 Laronde mineral reserves and mineral resources estimate were released to the public in a press release dated February 19th, 2003.

5. DISCLAIMER

This document contains certain statements that involve a number of risks and uncertainties. There can be no assurance that such statements will prove to be accurate; actual results and future events could differ materially from those anticipated in such statements.

	DILUTED GRADE				
BLOCK CATEGORY	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)

Probable	SOMME	3.455	63.187	0.37	2.93	0.32
Proven	SOMME	2.684	97.588	0.39	4.95	0.62
Indicated Resource	SOMME	3.936	14.851	0.17	0.55	0.04
	TOTAL GENERAL	3.317	68.924	0.37	3.27	0.38

BLOCK CATEGORY		TOTAL PRODUCTION (DILUTED) Au (g)	Ag (g)	Cu (Kg)
Probable	SOMME	105,685,002	1,932,909,320	112,686,721
Proven	SOMME	19,414,605	705,788,178	28,546,331
Indicated Resource	SOMME	2,314,779	8,733,129	989,252
	TOTAL GENERAL	127,414,387	2,647,430,627	142,222,304

SUMMARY OF ESTIMATION INFERRED RESOURCE RapRes03-01

BLOCK CATEGORY		DILUTED GRADE Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)
Inferred Resource	SOMME	5.923	13.024	0.33	0.08	0.02
	TOTAL GENERAL	5.923	13.024	0.33	0.08	0.02

SUMMARY OF ESTIMATION INFERRED RESOURCE

BLOCK CATEGORY		TOTAL PRODUCTION (DILUTED) Au (g)	Ag (g)	Cu (kg)	Zn (
Inferred Resource	SOMME	123,739,856	272,102,202	69,917,201	16,80
	TOTAL GENERAL	123,739,856	272,102,202	69,917,201	16,80

6. PROPERTY DESCRIPTION AND LOCATION

Most of the mining and milling activities at Laronde are centred on mining lease BM-796 which covers 491.886 hectares in Cadillac and Bousquet townships, Rouyn-Noranda mining district, north-western Quebec, Canada (NTS 32D/01: Latitude 78DEG.15'W, Longitude 48DEG.15'N). The mine is approximately 600 km northwest of Montreal, Quebec. The mining lease, which was issued in October 1988 and is valid for a 20-year period, is registered under the name of Dumagami Mines Ltd., which was amalgamated with Agnico-Eagle Mines Ltd. in 1989. In the Province of Quebec, the holder of a mining lease is generally also granted the surface rights. At Laronde, the Quebec Ministry of Transport retains surface rights over a small portion of the BM-796 mining lease that underlies Regional Highway 395 portion (8.16 hectares; block 33, Bousquet Township). The Laronde tailing facilities currently cover 167.61 hectares of BM-796 (Figure 2).

In 2000, mining activities began to extend onto a portion of the neighbouring El Coco property (owned 100% by Agnico-Eagle Mines Ltd.). A mining lease BM-854, contiguous with BM-796 and covering 59.58 hectares, was granted to Agnico-Eagle Mines Ltd in June 2001 for a 20-year period covering lot 62 in Cadillac Township (Val d'Or Mining District) where the new underground mining infrastructures are located. This mining lease replaces three mining claims (417570-1, 417570-2 and 417570-4) and a portion of two others (417570-3 and 417570-5) of the El Coco property, all located in Cadillac Township (see claim map in Appendix B).

Surface rights lease no. 816693, registered at the Ministry of Natural Resources lands registry office in Amos, Quebec under the name of Agnico-Eagle Mines Ltd. and renewable annually, covers a portion (122.333 hectares) of the Laronde tailing facilities, which extends outside the BM-796 mining lease in Cadillac Township (on the El Coco property). A second, annually renewable surface rights lease (no. 807400) covers the pipeline that supplies water to the mine site from Lake Preissac.

The mining lease are legally surveyed. The surface rights leases and area of the tailing ponds on BM-796 are approximate. A Table describing the annual fees and expiry date for each of the leases is presented in Appendix B.

Production from the Laronde mining lease BM-796 is not subject to any royalty. Production of minerals and mineral substances from the portion of the El Coco property west of Section 8780E (Laronde mining grid reference) is subject to a 50% net profits royalty. A royalty equal to 4% of the net smelter returns will also be derived from future production of minerals and mineral substances from the portion of the El Coco property east of Section 8780E (refer to Dionne and Boyd, 1999). Environmental liability and permit issues are addressed in the report by Roscoe Postle (2002, 2001) and Girard et al (2001).

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7. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Laronde property is located in the municipalities of Preissac and Cadillac, roughly midway (60 kilometres) between the cities of Rouyn-Noranda and Val d'Or, Quebec. The property can be accessed from either Val d'Or and Rouyn-Noranda by Highway 117 then northward for approximately 2 kilometres along Regional Highway 395 (Fig. 1).

The property is relatively flat; the maximum relief is about 40 meters and the topography slopes relatively gently down from north to south. All the surface water drains southeast into Dormenan Creek, which follows the southern property boundary and is a tributary to Noir Creek, located 2 kilometres to the east. The latter flows northward into Lake Preissac, about 4 kilometres to the north of the Laronde property. Climate allows for year-round mining. Surface mining, milling and mine tailing infrastructures cover roughly 60% of the Laronde mining lease (Fig. 2). A boreal-type forest consisting mainly of black spruce, poplar and minor birch, tamarack and balsam fir covers the remaining portion of the Laronde (and El Coco) property. The sufficiency of surface rights for mining and other availability issues are addressed in Scherkus (1986), Roscoe Postle (1999, 2001, 2002).

FIGURE 1

LOCATION MAP

[GRAPHIC]

FIGURE 2

SURFACE PLAN LARONDE MINE

[GRAPHIC]

8. HISTORY

Marquis et al. (1992) presented a comprehensive description of the exploration and development work completed on the Laronde property prior to 1989 (the work was also summarised by Trudel et al., 1992).

In 1937, Scott Chibougamau Mines Ltd. completed 70 square metres of outcrop stripping and trenching. This work uncovered a number of quartz-tourmaline-pyrite-pyrrhotite veins with traces of chalcopyrite and sericite and also revealed the presence of massive sulphides (pyrite) on the property. This showing was found in the vicinity of the original Laronde no.1 open pit (or East) zone.

In 1961, Rio Tinto Canadian Exploration Ltd. and O'Brien Gold Mines Ltd. conducted a reconnaissance survey in the area. Seven conductors were investigated.

In 1963, Dumagami Mines Ltd. staked 46 claims (696.1 hectares) covering the Scott Chibougamau Mines' Au-Ag-Cu showing.

In 1963 and 1964, Dumagami Mines Ltd. completed geological, magnetic and electromagnetic surveys and 51 diamond drill holes (10,274 metres). Most of the holes tested the area around the main showing (East zone) and while the rest were scattered along the axis of the mineralized zone.

In 1965, Dumagami Mines Ltd. published a resource (calculated to a depth of 243 metres) of 1,120,000 tonnes grading 6.5 g/t Au, 19.9 g/t Ag and 0.29% Cu. Judging that the grades were too low to justify an economic operation, work was suspended on the property.

In 1974, Mentor Exploration and Development Company Ltd. (part of the Agnico-Eagle Group of companies) joined Noranda Mines and Iso Mines who had been, since 1961, the principal shareholders of Dumagami Mines Ltd. A revised resource of 2,353,000 tonnes grading 3.3 g/t Au, 9.3 g/t Ag and 0.14% Cu was calculated to a depth of 268 metres.

In 1975, Dumagami Mines Ltd. completed 19 diamond drill holes (1,364 metres) to evaluate the open pit potential of the reserves indicated to a depth of 61 metres. Some overburden stripping over the main zone of mineralisation (East zone) and metallurgical tests were also completed. A decline in the price of gold cancelled plans to bring the main zone into production in 1976.

In 1979, Agnico-Eagle Mines Ltd. became a shareholder of Dumagami Mines Ltd. (joining Noranda Mines Ltd. and Mentor Exploration and Development Company Ltd.) and operator of the exploration program on the property.

In 1980, detailed geological mapping and lithogeochemistry and additional overburden stripping were completed over the main zone of mineralisation and on the rest of the property. 20 diamond drill holes were also completed (3537 metres).

In 1981, Dumagami Mines Ltd. published a resource (to a depth of 221 metres) consisting of 2.455 million tonnes grading 2.91 g/t Au of which 576,200 tons grading 2.67 g/t Au were judged to be mineable by open pit methods (Adamcik and Bailly, 1981). Surface diamond drilling over the entire property continued in 1982 and 1983

Between 1983 and 1985, Dumagami Mines Ltd. carried out an underground and surface exploration program on the main zone of mineralisation consisting of: 1) a three compartment shaft to a depth of 435 metres; 2) underground development consisting of 3,347 metres on five levels; 3) underground definition and exploration drilling totalling 18,985 metres; and 4) a surface diamond drilling program totalling 7,349 metres.

In January 1986, Dumagami Mines Ltd. published a revised resource (to a depth of 221 metres) for the no.3 and no.5 lenses (East zone) that totalled 1,971,669 tonnes grading 3.19 g/t Au. Although the deposit was judged to be uneconomic, approval was given to pursue a limited surface drill program to the west of the main zones of mineralisation and a single underground drill hole (Scherkus, 1986).

In early 1986, a new and relatively gold-rich zone of mineralisation (West zone) was discovered at depth and to the west of the previous mineralisation (the discovery hole 86-3 intersected 7.76 g/t Au over 9.1 metres at a vertical depth of 854 metres). A further 820 metres of underground development on 2 levels and 3,894 metres of diamond drilling were completed.

In 1987, Dumagami Mines Ltd. completed a positive feasibility study (Anderson, 1987) that recommended building a 1,360 tonne per day concentrator. Shaft no. 1 was deepened to 975 metres. The combined reserve and resource of the East and West zones was estimated at 4,969,596 tonnes grading 4.59 g/t Au and 0.42% Cu.

Commercial production at Laronde began in October 1988. In 1989, the production rate was increased to 1,810 tonnes per day. In December 1989, Dumagami Mines was amalgamated into Agnico-Eagle Mines Ltd.

In 1990, a surface and underground exploration program was initiated over the eastern portion of the Laronde property. The surface diamond-drilling program led to the discovery in 1991 of the no.4 zone (open pit no.2) and of the no.6 and no.7 zones in 1992.

The underground exploration program, initiated in 1990, consisted of exploration drilling of the favourable horizon from a main exploration drift (860 metres below surface) that extended to the eastern boundary of the Laronde property. A small lens corresponding to zone no.7 (block 72) was discovered in 1991 while zones 20 North Gold (formerly zone 19), 20 North Zinc, 20 South and no. 6 (at depth) were found in 1992 and 1993 (Fig.3).

FIGURE 3

LARONDE LONGITUDINAL PENNA SHAFT ZONES

[GRAPHIC]

In 1993, zone no.4 was test-mined with a small open pit. Open pit no.2 reserves for zone no.4 consisted of 112,000 tonnes grading 3 g/t Au, 7 g/t Ag, 0.1% Cu and 0.5% Zn. Open pit no.2 was mined-out by 1999 and milling of the stockpiled ore was completed in 2000.

In 1994, shaft no. 1 was deepened to 1205 metres and shaft no. 2 was completed to a depth of 525 metres. When mining began at shaft no.2 in 1995, reserves for the no.6 zone were estimated at 739,251 tonnes grading 9.42 g/t Au, 36.57 g/t Ag, 1.14% Cu and 2.44% Zn. Reserves for zone no.7 were then estimated to be 207,984 tonnes grading 4.64 g/t Au, 61.06 g/t Ag, 0.08% Cu and 4.50% Zn.

In 1994, the Penna shaft underground exploration and development program and mill expansion program was initiated.

In 2000 a transition took place in production from shaft 1 & 2 toward the new commissioned Penna Shaft. Underground production at shaft no.2 ceased in April, whereas underground production from shaft no.1 zones stopped in October 2000. The Penna shaft was completed to a depth of 2250 metres, the shaft changeover was completed and the 4500 tonnes per day production hoist and ore handling facilities were commissioned. The Laronde mill capacity was increased to 4500 tonnes per day.

In 2001, surface exploration on the El Coco Property led to the discovery of zone 22, 1.5 km east of the Penna Shaft 300m below surface. Level 86 (860m depth) exploration drift was extended toward the east across the El Coco Property.

In 2002, the Laronde Division reached the benchmark of 2 million accountable ounces of gold production in June. In October Hoisting and ore handling facilities were expended to reach 7000 tonnes per day at the Penna Shaft. The

Laronde mill capacity was also increased to 7000 tonnes per day at the beginning of October.

The following tables describe the cumulative payable metal production at Laronde, the division's production history by shaft and the 2002 underground production per zone.

TABLE 3: PRODUCTION SUMMARY TO DATE

YEAR	ORE MILLED (SHORT TONS)	GOLD GRADE (OZ/TON)	GOLD (OZ)	PRODUCTION PAID SILVER (OZ)	NET FROM SMELTE COPPER (LBS)
1988*	309,429	0.10	25,792	39,868	412,270
1989	693,825	0.14	84,974	127,339	1,394,567
1990	749,377	0.14	98,326	167,886	2,023,417
1991	652,390	0.20	115,831	164,572	3,869,050
1992	601,055	0.24	134,474	266,412	7,267,126
1993	638,523	0.26	152,355	270,671	9,207,872
1994	620,217	0.25	144,584	268,004	10,267,443
1995	728,064	0.25	167,209	330,532	12,183,871
1996	729,362	0.24	159,558	295,674	10,489,087
1997	785,552	0.21	154,515	279,938	8,844,441
1998	776,726	0.21	150,443	269,985	6,151,063
1999	798,402	0.12	90,035	277,327	3,282,471
2000	1,415,898	0.14	173,852	1,128,234	4,943,421
2001	1,805,248	0.15	234,860	2,524,146	4,096,247
2002	1,963,129	0.14	260,183	3,093,543	8,927,100
TOTAL	13,267,197	0.18	2,146,992	9,504,132	93,359,446

*Includes tune-up period. Production started on October 1st 1988

TABLE 4: LARONDE CUMULATIVE PRODUCTION TO DECEMBER 31, 2002

DESCRIPTION	TONNES	AU (g/t)	AG (g/t)	CU (%)	Z
Penna Shaft	4,341,915	5.01	76.00	0.27	
Open Pit #2	173,256	2.41	4.91	0.24	
Shaft #1	6,539,500	6.82			
Shaft #2	959,516	7.34	40.67	0.73	

GRAND TOTAL LARONDE	12,014,187	6.14

TABLE 5: 2002 DETAILED LARONDE PRODUCTION

DESCRIPTION	TONNES	AU (g/t)	AG (g/t)	CU (%)
Total 20 South Zone	741,858	7.85	76.26	0.27
Total 20 North Zone	965,053	2.97	83.70	0.41
TOTAL 2002 PENNA SHAFT RECONCILED PRODUCTION	1,706,911	5.09	80.47	0.35

9. GEOLOGICAL SETTING

The Laronde mining division forms part of the Doyon-Bousquet-Laronde Mining Camp. Geologically, the Laronde property is located near the southern boundary of the Archean-age (2.7 Ga) Abitibi Sub-Province with the Pontiac Sub-Province within the Superior Province of the Canadian Shield. The most important regional structure is the Cadillac-Larder Lake fault zone making the contact between the Abitibi and the Pontiac sub-provinces, located approximately 2 km to the south of the Laronde property.

The geology that underlies the Laronde mining property consists of three East-West trending, steeply south dipping and generally southward facing regional lithological units. The units are, from north to south: 1) the Kewagama Group which is made up of a 400 metre thick band of interbedded wacke; 2) the Blake River Group, a 1,600 metres thick volcanic assemblage which hosts all the known economic mineralisation on the property; and 3) the Cadillac Group, made up of 600 metre thick band of wacke interbedded with pelitic schist and minor iron formation (Fig. 4).

At Laronde, the Blake River Group is composed of the Hebecourt and Bousquet formations (Lafrance ET AL., 2002). The regional sequence shows a basalt flows basement overlain by andesitic to rhyolitic flows and fragmental rocks associated with local volcanic centres. Three members present on the property could be identified regionally, one within the Hebecourt formation and two others within the Bousquet formation. These are, from north to south: 1) the Northern Tholeiitic Basalt member; 2) the Lower Transitional member; and 3) the Upper Felsic member (Moorhead ET AL., 2000, Lafrance ET AL., 2002 and Dube ET AL., 2003).

Dube ET AL. (2002) have identified on the Laronde property several regionally correlatible units (Lafrance ET AL., 2002) that comprise the Northern Tholeitic Basalt, Lower Transitional and Upper Felsic members. Although several smaller lithological units are described in the following sections dealing with mineralisation, these units are not identified by the name identified by Dube ET AL.(2002).

The Northern Tholeiitic Basalt member (Hebecourt formation) consists locally of a 750 metre thick homoclinal sequence of relatively undeformed, southward facing, tholeiitic, massive to pillow basalt flows. Some of the flows are glomeroporhyritic and can be traced laterally for several kilometres. Although this unit hosts the Mouska and former Mic-Mac gold mines, no significant mineralisation is associated with it on the Laronde property.

FIGURE 4

REGIONAL GEOLOGY

[GRAPHIC]

The Northern Tholeiitic Basalt is interbedded with Quartz-porphyritic Rhyolite sill and dyke units (Bousquet formation lower member) close to the southern contact. This portion of the Tholeiitic Basalt member intruded by rhyolitic sill and dyke is up to 150 metres thick on the Laronde mine property. These dyke & sill unit consists of several metre-thick fine-grained felsic layer of tholeiitic to transitional affinity, which is characterized by 3-15% mm-sized blue-grey coloured quartz phenocrysts and equally abundant albite phenocrysts. Narrow mafic volcanic intervals are also present locally. Although no significant mineralisation or alteration is known to occur within this unit at Laronde, a low-grade, sulphide-rich, shear zone-hosted gold zone (Bousquet zone no. 6) is partly enclosed within it, 300 metres west of Laronde mining lease's western limit.

The Lower Transitional Volcanic member (Bousquet formation) is 200 to 350 metres in thickness on the Laronde mine property. Although all of the mineralized zones at the Doyon and Bousquet no. 1 mines are enclosed within this unit, no significant mineralisation has been discovered so far within it on the Laronde property. At Laronde, the lower member is characterized by transitional tholeiitic to calc-alkaline basalt-andesite flows and coarse andesitic lapilli block tuff (consisting of scoriaceous feldspar-rich fragments within an intermediate to mafic matrix) with minor horizons of intermediate to felsic tuff. The contact between the Lower Transitional Volcanic member and the Upper Felsic member is generally strongly sheared and faulted over several metres.

The Upper (or Southern) Felsic member, which hosts all the significant gold and base metal mineralisation on the Laronde property, varies in thickness from 150 metres in the vicinity of shaft no. 1 to over 550 metres thick at Penna shaft (Dube ET AL 2003). This stratigraphic interval is characterized by the dominance of quartz and feldspar porphyritic rhyodacite to rhyolitic flows, breccia and lapilli block tuff over fine-grained felsic tuff. Andesitic to dacitic flows and tuff are common in the northern part of the unit while blue- and grey-quartz porphyritic rhyolite tuff and lapilli block tuff horizons occur in the southern portion of the unit. Minor andesite flow horizons and possible sills have also been observed. The contact between the Mixed Volcanic member to the north and the Upper Felsic member is generally strongly sheared and faulted over several metres. The contact between the Upper Felsic member of the Bousquet formation and the Cadillac Sedimentary Group is undeformed to sheared over several metres, with local fault gouge (Fig. 5).

Zones of strong sericite and chlorite alterations, which enclose massive to disseminated sulphide mineralisation follow steeply dipping, east-west trending, anastomosing shear zone structures from east to west across the property. These shear zones comprise a larger structure, the Doyon-Dumagami Structural Zone, which hosts several important gold occurrences (including the Doyon and Bousquet deposits) and has been traced for over 10 kilometres within the Blake River Group from the Laronde property westward to the Mouska gold deposit.

FIGURE 5

LARONDE GEOLOGY

[GRAPHIC]

10. DEPOSIT TYPES

More than a dozen economic massive to disseminate polymetallic sulphide zones that vary in size from 50,000 to 40,000,000 tonnes are known on the Laronde mine property (Fig. 3). The mineralized zones are generally oriented east west and dip steeply to the south (parallel to the geological fabric). Each zone is identified by a number, which is based on its relative stratigraphic or structural position within the Upper Felsic member of the Blake River Group-Bousquet formation. Because more than one orebody may occur within a particular mineralized horizon, they have been assigned a block number (for example, block 71 represents zone no. 7 at shaft no. 2 while block 72 is zone no. 7 type at the Penna shaft). The zones are briefly described below in the order that they occur at each shaft, from north to south.

11. MINERALISATION

11.1 DESCRIPTION OF THE MINERALIZED ZONES AT SHAFT NO. 1
Zone no. 5 (blocks 51, 52, 53 and 54)

Zone no. 5 was the main production area at shaft no. 1 that has produced over 6.5 million tonnes of gold-copper ore (depleted April 2000). It occurs within the pyrite-rich core of a major east west trending, steeply dipping sericite-andalusite-kyanite schist band, several tens of metres thick, located roughly 100 metres north of the Cadillac sedimentary contact. This andalusite-bearing schist lens has been traced for approximately 500 metres east west from the western property boundary.

The zone no. 5 orebody and other minor discontinuous pyritic andalusite-bearing schist lenses (including the small South zone which was mined out near surface 120 metres south-west of zone no. 5) are enclosed within a 100-metre thick sericite alteration envelope, which can be traced as far east as the Penna shaft. Zone no. 5 is interpreted to be a high-sulphidation volcanogenic massive sulphide deposit (see Sillitoe, Hannington and Thompson, 1996; Marquis et al., 1994; Marquis, 1990).

Pyrite-rich massive sulphide lenses and strongly silicified sulphide-bearing zones occur within the pyritic andalusite schist zone. The zone no 5 orebodies are generally made up of en-echelon lenticular massive pyrite lenses, generally a few metres in thickness. The ore zones in massive pyrite lenses pass laterally into adjacent siliceous andalusite-bearing rock with 10-50% disseminated and foliation-parallel stringer sulphide and finally into the pyritic andalusite schist. Compositional banding is common and the intense deformation within the pyrite lenses is witnessed by boudinage structures and by the presence within the sulphide of foliated to massive silicified wallrock fragments. Some massive pyrite lenses may only contain minor gold values; gold is associated with copper sulphide mineralisation. Gold most commonly occurs microscopically along pyrite grains or in fractures and inclusions within pyrite grains either as inclusions within copper-bearing minerals (generally chalcopyrite and lesser bornite) or as calaverite.

The `East' portion of zone no 5, which is the original discovery zone, was mined along with another discrete lens (no. 3, now completely mined out) via open pit from surface to a depth of 70 metres and by underground methods to level 4 at shaft no. 1. The remaining reserves (block 52) extend below level 4 to almost level 10. A low-grade band of mineralisation separates the eastern zone no. 5 lens from the western portion.

The `West' portion of zone no. 5 plunges steeply to the west onto the Bousquet no. 2 mine. On the Laronde side, the zone extends from surface to 1,100-metre depth (just below shaft no. 1, level 26). Sulphide mineralisation is more massive, thicker, continuous and has a higher copper content than zone no. 5 East. The ore zone commonly follows the faulted and sheared southern contact of the andalusite-bearing sericite schist zone where disseminated to sheared veinlet sphalerite are also concentrated (Fig. 6).

Zone no. 4 (blocks 41, 42 and 43)

Zone 4 is a disseminated sulphide gold zone that has been traced to the east and a few metres south of zone no. 5. Originally observed in the eastern no. 1 open pit wall between zones no. 3 and no. 5, mineralisation consists of a narrow siliceous sulphide zone (less than 10 metres in thickness) within a broader sericite schist (east-west trending and steeply south dipping). The host rock is a deformed felsic lapilli-block tuff near the contact with a blue quartz-eye felsic tuff to the south. Mineralisation consists of 5-7% disseminated pyrite, sphalerite and minor chalcopyrite-bornite within a strong sericite schist envelope.

Although gold grades are low, in 1999, a small near-surface 173,000 tonne lens (block 41) was mined in Open Pit #2 east of Highway 395 and completely milled in 2000. Production from level 4 was completed in 2000. Low-grade indicated resource category blocks (blocks 41 and 42) surround the pit to below level 4 at shaft no. 1 (not included in the 2003 Reserves and Resources Estimate).

FIGURE 6

SECTION 6300E ZONE 5 SHAFT #1

[GRAPHIC]

11.2 DESCRIPTION OF THE MINERALIZED ZONES AT SHAFT NO. 2

Zone no. 7 at shaft no. 2 (block 71)

The zone no. 7 mineralized horizon has been traced across the entire Laronde property. It is characterized by the presence of several 1 to 5 metre-thick massive sulphide occurrences (sphalerite-rich pyrite lenses with very minor chalcopyrite). These massive sulphide zones occur along a more widespread horizon of disseminated pyrite-sphalerite mineralisation located less than 20 metres south of the faulted and sheared contact between the Mixed Volcanic unit and Southern Felsic unit. So far only lenses with a significant gold content are of economic interest.

Zone no. 7 at shaft no. 2 (approximately 230,000 tonnes) was discovered (along with zone no. 6) by surface diamond drilling in 1991. It consists of 1-3 metre thick lens of massive sulphide (pyrite with 15-20% sphalerite and minor chalcopyrite), which occurs at the contact between an altered rhyodacite lapilli tuff and a relatively unaltered dark andesite tuff to the south (Fig 7). Economic gold values are generally restricted to the massive sulphides. A sheared chlorite alteration zone (with 10-50% disseminated to sheared stringer pyrite-pyrrhotite with minor chalcopyrite and sphalerite and low gold values) often occurs immediately north and laterally to the west of the massive sulphides. A sericite alteration envelope occurs in the lapilli tuff north of the chlorite alteration zone and has 5-20% disseminated pyrite and minor disseminated sphalerite. Lapilli and block sized fragments of massive sulphide occur, although rarely, in the andesite tuff to the south. Graded bedding in the andesite tuff confirms the south facing of zone no. 7 at shaft no. 2. Mining exhausted this lens early in 2000.

Zone no. 6 at shaft no. 2 (block 61)

Zone no. 6 type mineralisation is only recognised in the eastern portion of the Laronde mining lease. It is associated with a narrow mineralized shear zone, less than 10 metres in thickness, which follows a horizon of felsic lapilli-block tuff approximately 100 metres south of the mixed volcanic unit contact. Two zone no. 6 orebodies have been discovered so far are (each roughly 800,000 tonnes in size) and contain, in addition to gold, significant copper, zinc and silver grades. As will be shown below, at shaft no. 2, zone no. 6 occurs roughly at a similar stratigraphic position as zone no. 7 (Gervais, 1996). Zone no. 6 at shaft no. 2 is a volcanogenic massive sulphide lens with an associated zone of disseminated and stringer mineralisation and alteration (transposed parallel to foliation) which occurs at the isoclinally folded contact between an underlying altered andesite flow and overlying less-altered rhyodacite polygenic lapilli-block tuff (Fig. 8).

FIGURE 7

SECTION 7600E ZONE 7 SHAFT #2

[GRAPHIC]

FIGURE 8

SECTION 7780E ZONE 6 SHAFT #2

[GRAPHIC]

The east west trending and steeply south dipping axial plane of the fold (a steeply east-plunging and east-facing anticline) forms the laterally extensive shear zone and fault. The massive sulphide lens migrates gradually from its position on the overturned north limb of the fold below level 3 to the south limb closer to the surface. The associated mineralized and zoned chlorite-sericite alteration zone occurs only on the north limb where it is transposed parallel to and to the west of the massive sulphide lens.

The underlying andesite flow unit forms the core of the isoclinal anticline. The andesite is generally massive (with rare 1-5 mm sized amygdales) but narrow zones of flow breccia have been observed along its southern contact (the northern contact is the mineralized shear zone). The andesite is strongly silicified and bleached (sericite and local green-mica). A thin horizon of sulphide rich and variably altered felsic volcaniclastic (lapilli tuff with andesite and porphyritic rhyolite fragments) overlies the andesite. The massive sulphide lens covers the lapilli tuff; the lens is generally less than 10 metres thick and consists of pyrite with 5-15% sphalerite and 2-3% chalcopyrite. Rare visible gold is associated with chalcopyrite and chalcopyrite occurs commonly along millimetre-thick north-south fracture planes within the massive sulphides. Metal zoning is present in the massive sulphides with sphalerite being more common on the stratigraphic top and margins of the lens while chalcopyrite is concentrated in the central basal portion of the lens. Gold grades however are relatively uniform throughout the massive sulphides. As with zone no. 7, a structurally transposed sulphide-rich zone of intense black and green coloured chlorite alteration (with 30-50% disseminated and transposed centimetre-sized stringers of pyrite, pyrrhotite and minor chalcopyrite with associated gold values) underlies the massive sulphide lens and extends westward into the underlying sulphide-rich felsic lapilli tuff. A sericite alteration halo, with 10-30% disseminated to veinlet pyrite and 5-10% sphalerite surrounds the chlorite zone within the underlying lapilli tuff. A polygenic rhyodacite lapilli-block tuff unit overlies the massive sulphide lens; this unit contains metre-thick felsic tuff bands that display graded bedding. Lapilli and block-sized fragments of massive sulphide (some gold bearing) are irregularly distributed in the layers of overlying the rhyodacite volcaniclastic rock.

A relatively unaltered andesitic tuff unit overlies the rhyodacite lapilli-block tuff. This tuff and underlying felsic volcaniclastic rock are the same units which host zone no. 7 at shaft no. 2 (a narrow barren pyrite-rich tuff unit locally marks the basal andesite tuff contact near zone no. 6 at shaft no. 2). Finally a quartz-feldspar porphyritic rhyodacite tuff, which occurs in the structural footwall and hangingwall of zone no. 6, forms the top of the stratigraphy in the shaft no. 2 area.

The zone no. 6 lens at shaft no. 2 was completely mined out in 1999.

Zone no. 5C at shaft no. 2 (block 55)

Zone 5C at shaft no. 2 is a small (50,000 tonne) east west trending and steeply south dipping disseminated sulphide gold zone that occurs less than 90 metres to the southwest of zone no. 7. Originally intersected in drill core in 1991, this zone was explored by underground development on level 6 at shaft no. 2. The mineralisation consists of 5-7% disseminated pyrite, 5-10% veinlet sphalerite, minor chalcopyrite and traces of visible gold associated with a 3-5 metre thick zone of fracture-controlled silica-sericite flooding. The siliceous sulphide zone follows a narrow shear that crosses a sericitized host quartz-porphyritic rhyolite tuff. The mineralisation is often

transposed sub-parallel to foliation. Raising in the 5C lens from level 6 indicated that a weakly inclined fault, located midway to level 5, displaces the zone several metres. Measured resource blocks are still indicated at shaft no. 2 but are not included in the present reserves and resources estimate.

11.3 DESCRIPTION OF THE MINERALIZED ZONES AT THE PENNA SHAFT

Zone no. 7 at Penna shaft (blocks 72, 73 and 74)

Three separate lenses comprise the gold-rich zone no. 7 probable reserve blocks that are interpreted to be down-dip equivalents of zone no. 7 at shaft no. 2.

The small zone no. 7 lens (120,000 tonnes), partially exposed by the 20-32 drift at shaft no. 1 (block 72) is very similar to zone no. 7 at shaft no. 2. It consists of a metre-wide massive pyrite lens (with 5-10% sphalerite and very minor chalcopyrite), which occurs at a sheared and altered contact between a rhyodacite lapilli-block tuff and a dacite tuff to the south. A metre-wide zone of strong chlorite alteration (with 10-30% pyrite) follows the northern contact of the siliceous massive sulphide lens and passes, 10 metres to the west laterally to a sheared metre-wide band of kyanite-sericite alteration (not present at shaft no. 2). Rocks north and south of the massive sulphide are variably silicified. Gold is erratically distributed within the massive sulphide and chlorite alteration (see Mailloux, 1998).

The zone no. 7 lens intersected in several drill holes near level 170 (block 73) also consists of massive sulphide (1-5 metres thick) containing 80% pyrite, 10-15% sphalerite and 1-3% chalcopyrite with rare millimetre grains of visible gold. Over 150 metres in east-west length, it has been traced vertically for almost 350 metres from its apex near level 152 down to level 194 where it pinches out. This lens occurs at the identical structural position as the other zone no. 7 lenses identified elsewhere on the property (i.e. 20 to 30 metres south of the Mixed Volcanic member/Upper Felsic member contact). However, the local volcanic stratigraphy, which hosts this second zone no. 7 lens, differs from the other lenses.

Near level 170, a 10 to 20 metre thick band of andesite volcanic occurs within the rhyodacitic lapilli-block tuff horizon that normally form the structural footwall to the massive sulphide lenses found elsewhere. This andesite horizon is identical texturally to the silicified andesite that forms the stratigraphic footwall to the zone no. 6 lens mined at shaft no. 2. In this area, the andesite is only slightly silicified and is more commonly weakly hematized and weakly bleached. It also is not in direct contact with the sulphide zone. The sulphides occur immediately south of a locally garnet-bearing, generally sulphide rich lapilli tuff horizon. A weakly pyrite-mineralized, feldspar-quartz bearing, rhyodacite lapilli tuff horizon occurs immediately south of the massive sulphides.

The third lens of probable category, zone no. 7 type, disseminated to massive sulphides reappears just below level 194. The lens (block 74), representing roughly 180,000 tons of ore, is up to 3 metres in horizontal thickness, 120 metres in east-west length and has been traced down-plunge to just below level 206. Sharing the same structural position as the other zone no. 7 type

lenses, it consistently occurs within 5 to 10 metres south of an andesite volcanic band 20 to 30 metres in thickness that completely replaces the usual rhyodacite lapilli horizon commonly found at this structural position. Again the massive sulphides occur south of a lapilli tuff unit (sometimes garnet-bearing but more often chloritized and sheared). Immediately south of the sulphides is a feldspar-quartz bearing rhyodacite tuff or lapilli tuff horizon

The down plunge portion of the zone no. 7 horizon has been intersected in only a few exploration drill holes originating from the Penna shaft (block RF-74). At depth, potentially economic intersections along the zone no. 7 horizon vary in texture from stringer type mineralisation (with 2-3% pyrite) to over 70% pyrite (massive sulphide mineralisation) with low percentages of sphalerite, chalcopyrite and rare of millimetre-sized visible gold grains.

Zone no. 6 at Penna shaft (blocks 62 and 63)

Two massive and disseminated sulphide lenses are interpreted to be the down-dip extensions of zone no. 6 gold-copper-zinc mineralisation that were mined at shaft no. 2.

The larger of the two (block RD63) is a narrow (3 metre) lens of predominantly massive pyrite with 1-2% sphalerite (up to 20% locally) and trace amounts of chalcopyrite. Resource estimate was updated in 2002 with a drilling campaign that reduced the spacing in between drill holes to about 80m vertically and 120 m horizontally in the immediate area of the discovery hole (20-123). One of the new drill holes intersected the lenses while all the others intersected disseminated mineralisation along the horizon.

A smaller lens of massive pyrite-sphalerite and interpreted to be zone no. 6 occurs immediately East and below the level 146 shaft crosscut (block PB62).

Zone no. 20 North Gold (blocks 191, 192, 193 194 and 195)

The 20 North Gold zone is a disseminated to massive sulphide gold-copper zone which has been traced in the central portion of the property, 120 to 150 metres north of the Cadillac sedimentary Group contact. The zone occurs within a weakly sheared and locally fractured, weakly sericite and silicified altered, felsic lapilli tuff horizon, several metres in thickness that occurs immediately south of a garnet-bearing dacite tuff horizon. The mineralisation consists of 30-50% very finely disseminated to massive pyrite, 1-10% transposed millimetre to centimetre-thick chalcopyrite veinlets (including some 1 to 5 millimetre-thick up to 1 metre-long, discontinuous, foliation-oblique chalcopyrite veinlets), trace millimetre-sized clots of bornite, very rare disseminated millimetre-sized clots of visible gold (always associated with the chalcopyrite) and minor disseminated sphalerite. The sulphide-rich lapilli tuff horizon closely follows the fractured north contact with the no. 20 North Zinc massive sulphide lens. When the massive sulphide horizon is absent, the sulphide rich tuff band is in fault contact to the south with either altered andesite or rhyodacitic, blue quartz-eye bearing lapilli-block tuff.

Gold is generally directly related to the chalcopyrite content. Below level 149, the massive sulphide lens becomes more copper rich and with this transition, the gold content increases significantly. The 20 North Gold zone at depth occurs within booth massive and disseminated sulphides with aluminosilicate porphyroblasts (mostly kyanite and andalousite) that became progressively more abundant with increasing depth (Fig 9 & 10).

Zone no. 20 North Zinc (blocks 201, 202, 203, 204 and 205)

The 20 North Zinc zone is a massive sulphide zinc-silver zone, which has been traced, in the central portion of the property, 100 to 150 metres north of the Cadillac sedimentary Group contact (Fig. 9 & 10). It generally consists of several massive sulphide lenses made up of 50 to 90%, 1 to 3 millimetre-sized pyrite, 10-50% light brown coloured disseminated sphalerite with minor chalcopyrite and galena. Narrow (usually less than 1 metre in thickness) and laterally discontinuous bands of variably graphitic argillite occur more commonly near or at the southern contact zone of the massive sulphide lenses. The argillites bands are generally strongly sheared or even faulted and are not traceable for more than 10 metres in an east-west direction. They are also more commonly found at the edges of the massive sulphide lenses.

The massive sulphide lenses vary between 1 to 30 metres in thickness and the largest lens has been traced for up to 600 metres horizontally and over 1500 metres vertically. It occurs in sheared faulted contact to the south with either a blue quartz-porphyritic rhyolite tuff (to lapilli tuff) horizon or a strongly altered and mineralized fine-grained andesite volcanic horizon. The sulphide-rich lapilli tuff horizon making up the 20 North Gold zone follows the sheared and fractured north contact of the no. 20 North Zinc massive sulphide lens. Below level 149, 20 North Zinc migrates toward the southern contact of the massive sulphide lens.

Zone no. 20 South Gold (blocks 211, 212 & 215)

Several 1 to 2 million tonne-size gold-copper-zinc-silver-bearing sulphide lenses comprise the 20 South Gold zone. The lenses are arranged along a sheared sulphide-rich horizon, which has been traced in the central portion of the Laronde mining lease (and eastward across onto the El Coco property) within 20 metres north of the Cadillac sedimentary Group contact. The 20 South horizon is oriented east-southeast with an 80-85 degree South dip. The economic lenses discovered so far are arranged en-echelon about a steeply westward plunging rake.

FIGURE 9

SECTION 7440E ZONE 20 NORTH AND 20 SOUTH PENNA SHAFT UPPER MINE

[GRAPHIC]

FIGURE 10

SECTION 7080E ZONE 20 NORTH AND 20 SOUTH PENNA SHAFT LOWER MINE

[GRAPHIC]

The lenses generally occur as 100 to 300 metre long (east-west) by 300 to 500 metre tall (in elevation) and 3 to 15 metre thick (north-south) pods of ore within the more extensively developed sulphide-rich sheared horizon (Fig 9 & 10). The sheared horizon, 1-5 metres in thickness, consists of disseminated pyrite, pyrrhotite and minor sphalerite in a sericitic siliceous (and locally chloritic) moderately to strongly sheared matrix cut by irregular zones of foliation parallel faulting.

The 20 South sulphide horizon occurs along the southern margin of an altered andesite unit, 5-20 metres in thickness. The alteration commonly affecting the andesite consists of strong matrix silicification with titanite micro-stringers (the rock is pink coloured, looking like an hematisation alteration, in reason of the titanite micro-phenocrist that occurs concentrated along fractures due to the alteration process). Foliation-parallel cm-thick bands of sericite-green mica alteration overprint this pink coloured titanite alteration. Minor disseminated coarse sulphide grains (pyrrhotite-pyrite) are observed. Sulphide content in the andesitic rock increases (to 10-30% pyrite-pyrrhotite) and sericite-green mica alteration strengthens 5-10 metres north of the sheared zone (at the expense of silica-titanite alteration).

Immediately south of the zone is often found a narrow band (up to 10 metres in thickness) of altered and sulphide-rich (30% pyrite-sphalerite) andesite. Generally wedged between the Cadillac Group sediments to the south and the andesite is a 1 to 5 metre thick band of blue quartz-eye bearing rhyolite tuff or lapilli tuff followed by a thicker horizon of rhyolitic lapilli tuff unit 1-10 metres in thickness.

Gold values in the 20 South zone, whether in the disseminated facies or in the massive sulphides, are always associated with chalcopyrite or with millimetre-size clots of native gold within 1-5 millimetre clots or veinlets of chalcopyrite. The chalcopyrite is generally disseminated in the sulphide matrix or remobilized along north south trending and steeply dipping crosscutting millimetre-thick discontinuous veinlets located throughout the sulphide zone. The presence however of significant amounts (5 to 10%) of pyrrhotite with chalcopyrite is associated with lower than average gold values.

Two main lenses of economic mineralisation have been defined so far (blocks 211 and 212) . Block 211 crosses onto the El Coco property (block 215).

The uppermost lens (block 211), centered near level 122 at section 7500E, is made up of two types of mineralisation: 1) Disseminated sulphide type mineralisation (up to 10 metres in thickness) is common on the Laronde property between levels 152 and 118; whereas 2) Massive sulphide type mineralisation (up to 10 metres in thickness) becomes the dominant type of ore as the no. 211 lens crosses eastward onto the El Coco property and above level 125.

The lowermost lens (block 212), centred near level 194 at section 7200E, is irregularly shaped, generally less than 5 metres in thickness, and mostly disseminated in character.

12. EXPLORATION

In 2002, exploration work on the Laronde and El Coco property consisted solely of diamond drilling.

12.1 2002 DRILLING RESULTS

A summary of diamond drilling completed in 2002, broken down by project, is presented in Table 6 and a detailed 2002 drilling schedule is presented in Appendix B. The lists of holes completed in 2002, with all zones mid-point intercepts are also presented in Appendix B. Diamond drilling results are plotted on the individual zone longitudinals attached to the report.

12.2 2003 DRILLING PROGRAM

In 2003, the definition and exploration budget calls for 54,110 metres of diamond drilling (Appendix B). Delineation drilling on the Laronde property will total 11,810 metres. Definition drilling (project 8, spacing 40m x 40m) with a total of 15,800m is going to continue: 1) in the western extension of the 20 North zone, from section 7000E to the western limit of the zone (bloc PB193) 2) in the 20 South zone (bloc PB212) and 3) in zone 7 lenses (blocks PB 73 & PB 74). Deep exploration drilling (project 10; 26,500 metres) from the level 215-exploration drift will be focused on the deep resources to reserves conversion program.

Deep exploration of the 20 North and 20 South zones, from the level 215 exploration drift, consists of completing a pattern of north-south drill intercepts on an average 120 metre (horizontally east-west) by 120 metres (vertically) intervals, in the are of more widely spaced azimuth exploration drill holes completed in the past years from the Penna shaft station.

Exploration along the level 86-exploration drift will be postponed since all the energy will be focussed on the deep mineral resources conversion program on level 215.

12.3 DRILLING CONTRACTOR

Forages Garant of Evain, Quebec, does all diamond drilling on the property under a contract with Agnico-Eagle mines ltd.

TABLE 6: SUMMARY OF DIAMOND DRILLING 2002

PROJECT NUMBER	HOLE CATEGORY	NUMBER OF HOLES	METRES DR
351	Delineation Laronde et El Coco	238	12
8	Definition	62	17

10	Deep Exploration	42	16
11	Level 86 Eastern Exploration Drift	24	13
11	Surface El Coco Exploration	7	6
TOTAL		373	65

(see detailed Appendix B)

12.4 PROCEDURE FOR DESCRIBING DRILL CORE

All the drillcore recovered at Laronde are described by graduated geologist (or geological engineers). In the past, some drill core was logged by a non-graduated geologist and was supervised and reviewed by a Laronde mine geologist. Each hole has a drill log associated with it. Current drillcore description are inputted directly onto a portable computer in the core logging facilities using DH Logger software. Descriptions of lithology (using mine specific unit lithological terms), texture, structure, alteration, mineralisation and sampling are noted. Once the sample results are returned from the laboratories, the results are plotted on sections and plans at the appropriate scale. The geologist logging the core is responsible for interpreting the geological results and for compiling the mineralized zone composite in the log. The geologist is also responsible for verification of the log. A paper copy of the log is then signed and placed in a core log registry stored in the Laronde geology department vault. All the historical drill logs, previously logged or compiled in Log II software format have been converted in 2001 into DH Logger format

12.5 RELIABILITY OF RESULTS

Less than 100% core recovery in ore is rare; in all cases where mineralized core was lost, the routine procedure is to install a wedge and to redrill the missing mineralized interval to obtain 100% recovery. Intervals of lost core in waste rock are noted in the drill logs but are not redrilled. Interpreted and verified drill results are highly reliable. Uncertain historical results are excluded but noted.

13. DRILLING

13.1 CORE SIZE

All of the current exploration and definition diamond drilling at Laronde recovers BQ size (3.35 centimetre diameter) core using industry standard wire line methods.

The core is stored in consecutive order (standard left to right and top to bottom sequence) in 1.5 metre long wooden trays (core boxes) that have a 6-metre capacity. The drill contractor identifies the location of the core samples along the hole by placing properly identified wooden markers at 3-metre spacing. The drill contractor also specifically identifies intervals of incomplete core recovery (due to grinding or washout).

Once the box is full, it is closed tightly shut with a matching lid using common fencing wire. The drill hole's identification and box number is then identified on the lid using an indelible ink marker.

13.2 DRILL HOLE IDENTIFICATION

Each drill hole at Laronde has a unique identification that is related to the location of the collar. Exploration and definition drill holes are numbered differently than delineation drill holes.

Exploration and definition drill hole at Laronde are identified using this procedure: the drill hole identification number firstly identifies from which shaft and level the drill hole originated, and then the order in which the drill hole was completed. For example, definition drill hole no.3122-05, was drilled from the Penna shaft (shafts 1, 2 and Penna are identified by the numbers 1, 2 and 3) and was the 5th hole that was drilled from level 122.

A delineation drill hole is used in the underground development stage of each production stope at the Penna shaft. Each delineation hole is numbered using a simple 8 integer series that identifies from which sublevel, zone, stope draw point, and order that it was completed. For example, delineation drill hole no.14320681 was drilled on level 143, it tested zone 20 North Zinc (zones 6, 7, 20 North Gold, 20 North Zinc and 20 South are identified by the numbers 06, 07, 19, 20 and 21 respectively), it was drilled in draw point no.68, and it was the first hole to be drilled from this collar location.

13.3 CORE STORAGE

With only few exceptions, drill cores from all the exploration holes are stored in their entirety in the Laronde core library. Generally, for drill holes from definition drilling programs (that systematically test the target horizons on evenly spaced intervals along a given section line) only core from the flat holes are kept. Because whole samples of delineation drill core are taken, wall rock core samples are discarded.

Each stored core boxes is identified with an aluminium tag that has the appropriate drill hole information embossed on it (including hole number, box number, the core interval stored in the box). Boxes belonging to individual drill holes are stored consecutively in a core rack located outside on the Laronde site. An inventory is kept for each core rack and is copied into an electronic data bank by the geology department.

13.4 PROCEDURES

DRILL HOLE LAYOUT

The following procedure is followed when laying out an approved drill hole:

1. The geology department prepares the drill hole layout on a copy of the most current 1:250 scale underground development map (provided by the engineering department) that covers the proposed collar location. Information such as collar coordinates (which reference Laronde mine grid), drill hole azimuth and plunge, hole length and special comments are noted directly on the layout.

2. A copy is forwarded to the diamond drilling contractor foreman, the

surveying department and to the mine department (for their information).

3. The drill hole collar is identified by the surveyors who also layout out the drill hole starting azimuth by setting front and back sights into the drift's walls (using drilled metal spads which are identified with fluorescent tape or paint).

4. The contractor sets the diamond drill onto the collar and aligns the drill along a string tied thought between the front and back sight spads. The plunge of the drill is fixed using a spirit level.

5. Once the hole is completed, the geology department re-issues the layout to the surveying department that who returns to the collar location of the hole and directly measures the final coordinates, azimuth and plunge.

6. The coordinates, azimuths and plunge are entered into both a handwritten drill hole registry and an electronic data bank. Each entry is dated and initialled and then checked by a second member of the surveying department (who also initials the entry). The registry is stored in the geology department vault.

CEMENTING OF COMPLETED DRILL HOLES

In accordance to the Quebec mining regulations, after the drill holes are completed and surveyed, they are cemented either at the collar (over a 5 metres length) or, in the case of delineation drill holes in ore, completely filled using a grout cement mixture. A contractor completes plugging of the borehole. The list of cemented holes is also kept into a handwritten registry stored in the geology vault. Since 2001, hole cementation is also register in the data base of the DH logger system and identified on the front page of every drill hole log.

ORIENTATION TESTS

The position of surface and underground drill holes at the Laronde Division is determined using a combination of numerous in-the-hole orientation test methods. The methods that are or were employed are:

1. Gyroscopic surveys performed by specialized companies (Sperry Sun Drilling Services or CBC Welnav);

2. Deflection type Maxibor surveys by specialized companies (Reflex Instrument Canada or Boreinfo) and Lightlog surveys completed by Agnico-Eagle or by a contractor (Techdel International);

3. EZ-Shot single-shot magnetometer-accelerometer-temperature tests (Reflex Instrument Canada) and single or simultaneous-double single-shot compass tests (Pajari Instruments or Sperry Sun Drilling Services) completed by Agnico-Eagle staff or by the drilling company;

4. Rotodip inclinometer tests (Techdel International) read by the drilling company; and

5. Acid-dip tests interpreted by Agnico-Eagle.

The collar azimuth, plunge and co-ordinates of all drill holes, which all the methods use as a starting reference, are determined by a mine survey. In the event that the collar azimuth and plunge cannot be measured, then the planned azimuth and plunge are used. Similarly, the planned co-ordinates of the drill holes are used when the collar position cannot be surveyed.

At Laronde, there is a procedure that is employed when several orientation test methods are available for positioning a particular bore hole. The results from orientation tests considered to be more precise take precedent over and replace orientation data collected from methods considered to be less precise. This procedure is described in the following paragraphs:

1. The gyroscopic survey method is considered to be the most precise. The collar azimuth used for the conventional gyroscopic survey is taken from the directly measured mine survey while the initial plunge of the borehole is read directly from the gyroscope's inclinometer. The more expensive North-Seeker gyroscopic survey does not require a surveyed collar azimuth. Gyroscopic surveys are not designed for holes that plunge at angles shallower than 30 degrees from horizontal.

2. The Maxibor survey method is a deflection-type method that is more advanced than the Lightlog and, although not considered to be as precise as a gyroscope, it is adequate for holes that have a shallow dip. The collar azimuth is taken from the mine survey.

3. Should the gyroscopic (and Maxibor) survey be incomplete or absent, Lightlog deflection-type survey data is used. The collar azimuth used for the Lightlog method is taken from the mine survey while the initial plunge of the borehole is read directly from the instrument's inclinometer. In the case that the gyroscopic survey is incomplete, the departing azimuth of the Lightlog survey is corrected to that of the last gyroscopic reading while the plunge is read directly from the Lightlog instrument's inclinometer.

4. In the event that Lightlog survey data is incomplete or absent, then single-shot compass-based orientation tests are used to determine the borehole position. When simultaneous- double compass test results are available, the average azimuth (corrected for magnetic north) and plunge are calculated for the survey point and used in calculating the drill hole position. In the case that the azimuth data is determined to be erroneous because of magnetism, the plunge data takes precedent over acid-dip or Rotodip measurements.

5. A new EZ-shot system, which is a digital magnetic-type compass system, electronically measures the azimuth, plunge, magnetic field and temperature. This system has recently been used at Laronde since year 2000 as a quicker more effective alternative to the older compass-based Pajari systems.

6. When compass-based orientation tests are absent, Rotodip plunge tests take precedent over acid-dip tests.

The geology department enters orientation data gathered for each hole daily into a registry. Wedge data is also noted. Each entry is dated and initialled and the registry is stored in the department vault. Should a data point error be interpreted, the point is biffed, noted and also initialled. Once the data is transferred into the data bank, the geologist completing the drill hole log also checks it for errors.

13.5 RELATIONSHIP BETWEEN CORE LENGTH AND THICKNESS

At Laronde, the relationship between the core length and the true thickness of the mineralisation is as follows:

1. The true north-south horizontal thickness of the mineralisation intersected in a drill hole is always reported on longitudinals and plans.

2. The north-south horizontal thickness is measured directly from an interpreted drill hole plan or section. The thickness is measured from the intercept midpoint along the drill hole.

3. In an area of sparse information where it is not possible to properly interpret the trend of the mineralisation, the thickness of the intercept is measured from the drill hole zone that is locally interpreted to be east west trending and dipping 70-90 degrees south.

14 SAMPLING METHOD AND APPROACH

At Laronde, the estimate of mineral reserves and resources in mineralized zones are based on systematic sampling using diamond drill core or chip sampling collected in underground development headings (or both methods)

14.1 CHIP SAMPLING METHOD

The mining methods currently in use (transversal and longitudinal blasthole stoping), requires for each of the mining blocks (15 metre East-West length), the excavation of a 5 metre (or more) height drift through the entire horizontal width of the mineralized lens (from the north wall to south wall ore\waste limits of the mineralisation) the excavation are driven in ore at vertical intervals of 30 metres (below level 122) to 40 metres (above level 122).

During the mining block excavation process, successive vertical north-south oriented exposures across of the entire mineralized zone (and wall rock) are chip (on panel) sampled by the geology department staff. In this manner, each stoping block which is in the proven reserve category by definition) will have 2 to 4 (or more) complete lines of chip samples assay results (lines spaced 3 to 5 metres in an East-West direction) often both at the top and bottom of the 30-40 metre high mining block.

The following method is taken for chip samples:

1. The location and orientation of the chip sample line always reference a mine survey plug located in the ceiling nearby;

2. The wall is carefully washed with a fresh water using a hose and, if needed, scaled for loose rock;

3. The sampler marks the samples off continuously at regular intervals (between 0.3 and 1.5 metres) at a height of 1.5 metres above the floor (or exceptionally, on the ceiling);

4. Samples are measured to the nearest 10 centimetres;

5. Sample intervals must coincide with lithological boundaries (the sampler describes the location and the geology of each sample in a sample note book);

6. The sampler takes a continuous representative rock sample using a hand-held geologist hammer and places the rock fragments, of uniform volume in a sturdy plastic bag;

7. A sample tag, specially made of waterproof paper and indelible ink, is also placed in the bag (each sample number is unique);

8. The samples collected should represent the same volume collected as a BQ size drill core sample (roughly 2 kilograms per metre of chip sample);

9. Samples of ore are always bordered by wall rock samples. If the waste rock/ore contact is not properly exposed, either an additional slash will be requested to properly expose the contact or a short core drill hole will be completed. Exceptionally a percussion sample hole will be taken and the drill sludge will be sampled over an appropriate interval (following geology);

10. The sample bags are tied shut using blast hole lead wire and brought immediately to the shaft station for transfer to the assay lab.

The location and geological data is transferred to a electronic chip sample data bank using Century System Ltd. Software. Assays results are combined with the geological data for use in the estimate process.

14.2 CORE SAMPLING METHOD.

Diamond drilling is the initial method of collecting a continuous series of samples through zones of mineralisation on a regular pattern. At the exploration stage, drill holes are designed to cut the mineralisation at a reasonably high angle (generally greater than 45 degrees). At the ore body definition stage (in order to transfer inferred mineral resource to the indicated resource category), drill holes are designed to cut the mineralisation, at an appropriately spaced pattern, on either a north-south vertical section basis (see figures 6 to 10) or on a horizontal section. The appropriate spacing of intercepts depends on the geological and geostatistical characteristics of each zone (see section 19.2).

Once the drill hole samples are extracted, the method for taking core samples is as follows:

1. The core is washed with fresh water using a hose;

2. Once the geology and location of the samples is described (see section 12.4), the geologist carefully 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;

3. A sample tag, specially made of waterproof paper and indelible ink, is placed at the end of the sample interval. Each sample number is unique;

4. The core is generally sampled over regular intervals that vary between 20 centimetres and 1.5 metres (1.0 metre samples are most common at Laronde);

5. Samples are measured to the nearest 10 centimetres;

6. Samples intervals have to coincide with lithological boundaries;

7. Samples of ore must always be properly bordered by samples of waste.

Should an anomalous value return from an isolated sample, the geologist is required to return to the core interval and take additional bordering samples;

8. Generally 0.5 metre long samples are purposefully taken on the borders of obvious ore zones in order to minimize the effect of sample contamination of wall rock by high grade ore;

14.3 FACTORS THAT CAN MATERIALLY IMPACT THE SAMPLING RESULTS

1. Errors in properly locating chip samples (erroneous coordinates) can impact results;

2. Although the sampler attempts to collect uniform representative sample of adequate size, in some cases this is not possible (e.g. rock too hard to fragment properly); the accuracy and reliability of the local results may therefore be impacted (it is interpreted to be minimal).

3. Sample identification errors (missing or disfigured sample tickets) can impact the results.

In all cases described above for chip sampling errors, data verification procedure consist of the sampler manually plotting assay results daily on sketches and comparing the results against the geological observations; suspected erroneous results are always discarded and the samples are retaken.

4. Intervals of missing core in mineralisation (due to grinding) can impact the results;

5. Errors in properly locating drill holes in mineralisation (due to either erroneous orientation test, hole identification, sample depth and sample interval) can also impact the results

In all cases described above for drill hole sampling, data verification procedure consist of the geologist reviewing the drill holes either on plotted sections and plan or viewing 3 dimensional computer images and checking for any inconsistencies. Suspected erroneous results are corrected or the sample intervals (if half-core of the sample exist) is resampled.

The effect of undetected chip sampling and drill hole sampling errors in areas of active mining, considering the good reconciliation results (Gosselin in prep.) is considered to be negligible. Undetected errors in areas of future mining or exploration are minimized through additional sampling

14.4 SAMPLE QUALITY AND REPRESENTATIVITY

At Laronde, sample recovered through diamond drilling are of high quality (the mineralisation in core is intact with no possibility of loss due to wash out). The quality of chip samples is assured with proper sampling techniques by the geology department staff, but as shown by D'Amours (2002), there may have a minor discrepancy between drill holes and chip samples due to a lower quality in chip samples. Core and chip samples are considered to be representative of the bulk of the mineralisation as is witnessed by good reconciliation between the forecasted and recovered grades of mining blocks (Legault, 2001a, 2002a, Gosselin in prep.).

In zone 20 North Gold, The presence of erratic crosscutting veinlets of remobilized gold-chalcopyrite oriented sub vertically, North-South, is thought to be the cause of a minor sampling bias in both North-South oriented drill holes and chip samples. The sampling bias is discussed in D'Amours (2002) and in section 19.2.

14.5 OTHER SAMPLE DESCRIPTIONS

A description of the rock type, geological controls, widths of all the mineralized zones is presented in section 9. A list of individual drill hole zone composite is presented in Appendix B and on longitudinal sections. For the sake of brevity, the is no list of zone composites for chip sample lines.

15. SAMPLE PREPARATION, ANALYSIS AND SECURITY

15.1 CHIP SAMPLE COLLECTION PROCEDURE

The Laronde geology department sampler either deposits the samples into a storage box located in the shaft house at surface or leaves them at the shaft stations for the collection by the mine cage tenders who transfer them to the surface storage box. The storage box is collected daily and transported to the Laronde Lab by mine transport worker. All chip samples are sent to the Laronde Assays laboratory.

15.2 CORE COLLECTION PROCEDURE

The Laronde geology department sampler either takes a half core split sample of the core using a core saw or mechanical core-splitting device or samples the whole core depending on the situation (see section 13.3).

The following procedure is used at Laronde when sawing core samples:

1. The core shack area must be kept as clean as possible at all times;

2. The core saw used for sampling must always have a fresh clean source of water to cool and lubricate the circular saw blade and to reduce the risk of contamination;

3. The water and rock cuttings must drain unobstructed away during the cutting process;

4. Care must be taken not to introduce a sampling bias during cutting (for example, for core samples in irregular mineralisation, representative samples may have to be chosen; a cutting line drawn directly onto the core may be necessary);

5. As the core sample is sawed in half, the samples chosen for assay are collected in an individual clear 25-cm by 40-cm 6 mil gauge plastic sample bag. The other identical half core witness sample is replaced carefully in the box according to its original orientation (the correct end of the core up hole, for example). One of the two sample tags is placed in the plastic bag that is then securely stapled shut;

6. The other identical sample tag is stapled into the core box at the end of the marked sample interval.

The procedure for taking half core-split samples differs slightly from the one for sawing core samples in half:

1. The mechanical splitter has to be properly cleaned (with a brush or a jet of compressed air) prior to cutting every sample;

2. Metal cake pans are used to collect the sample fragments. Once the individual sample is completed, or if either of the cake pans is full, the pans are emptied into a plastic sample bag. The successive sampling routine is the same as above.

In the case of whole core samples, the entire sample is collected consecutively down the interval (along with the sample tag) and placed in the sample bag. The bag is then securely stapled shut.

Several samples are stored together in a sealed burlap bag along with a sample request form. The sealed samples bag lots are transported to the various labs by a commercial courier service.

A sample request form has to be completed prior to dispatch of the samples. The request specifies the name of the laboratory, the person making the request, the date, the sample series, the metals to be assayed, the units that the results should be reported, the analytical method and any special instructions.

In 2002, the principal assay laboratories for drill hole core samples were: Laboratoires d'Analyse Bourlamaque of Val d' Or (Linda Melinbardis, chief chemist), ALS Chemex Chimitec of Val d'Or (Richard Deschambault, manager) and Technilab of Ste-Germaine, Quebec (Ahmed Edgdougul, chief chemist). The principal assay laboratory for chip samples is the Laronde assay laboratory (exceptionally they can be sent to an independent laboratory). None of the laboratory are certified by a standards association.

15.3 LARONDE ASSAY LABORATORY PROCEDURES

SAMPLE PREPARATION

1. The samples are dried at 180DEG. C for about 30 minutes;

2. The samples are layed out in order in metal pans and registered on the assay report sheet;

3. Prior to crushing and pulverizing, the equipment is cleaned using compressed air;

4. The entire samples is passed through a 1/4inch jaw-crusher;

5. The crushed sample is passed through a Riffle-type separator 4 times (mucks), 6 times (chips) or 8 times (core) in order to obtain a mass of roughly 250 grams. Coarse reject material is kept or thrown out depending on the sample type;

6. The split sample is pulverized to 80% -200 mesh using a disk pulverisor;

7. Silica is pulverized through the equipment between each sample;

8. The pulp samples are homogenized using an orbital mixer or by using the 4-corner carpet method.

ANALYSIS

The Laronde division laboratory Fire-assays core samples and chip samples in the following way:

ELEMENT	EXTRACTION	METHOD
Au	Fire-assay (up to 1 A.T.)	Fire-assay, gravimetric finish
Ag	HF-HCL-HN03	Atomic absorption
Cu	HF-HCL-HN03	Atomic absorption
Zn	HF-HCL-HN03	Atomic absorption
Pb	HF-HCL-HN03	Atomic absorption

15.4 INDEPENDENT ASSAY LABORATORY PROCEDURES

The core sample preparation and assaying procedures for the various certified independent laboratories used are similar to those presented above (see Appendix B). An independent laboratory performs also specific gravity measurements on selected core samples.

15.5 QUALITY CONTROL MEASURES AND CHECK ASSAY PROCEDURES

SUMMARY

At Laronde the quality control measures, check assays and duplicate assay procedures are different for samples from chip sampling, exploration and definition drill holes and from stope delineation drill holes.

CHECK ASSAY PROCEDURE

All exploration and definition drill core samples are sent to independent assay laboratories for analysis (prior to 1999, exceptionally they could be assayed at the Laronde laboratory). For samples that occur within, or adjacent to, potentially economic ore zones, pulp and reject witness samples are recovered from the primary laboratory and systematically sent to a second independent assay laboratory for check reanalysis.

Delineation drill core samples are sent to an independent laboratory for analysis but the sample results are not checked at a second laboratory.

Chips samples assay results that are receive from the Laronde assay lab are not checked at a second laboratory. The geology department sometimes requests check assays from the Laronde laboratory on the pulps and/or rejects of certain anomalous chip samples. Dubious or missing results calls for an immediate resampling of the panel sample interval.

The results from check assaying program are averaged together (original assay, pulp check and reject assay) and it is only the average value that is reported in the drill log.

CONTROL SAMPLE PROCEDURE

All the samples coming from a delineation drill hole are sent to the laboratories with a control sample (one control sample for every group of samples that compose a drill hole intercept). The control samples are prepared at the mine site laboratory from representative mineralised material from the Penna shaft lenses. Batch of 15 to 25 kg are crushed, homogenized and split in 50 to 250g samples (average). Each control sample is put in an individual clear 25-cm by 40-cm 6 mil gauge plastic sample bag with a unique paper stamped sample tags that is placed in the plastic bag and then securely stapled shut. One standard sample is sent for every delineation drill hole zone intercepts samples group (i.e. 20 North, 20 South, 6 & 7) and reported with the core sample assays results. Irregular results that came back from those standard samples are checked in another independent laboratory to see if any analytical problems occur within those reports.

The control sample procedure in delineation drill holes is study every quarter (3 months period) and the year-end report is presented in Appendix B.

In 2002, all the standard samples that came back with irregular values compare to the average were duplicate with similar results in the check assays procedure with another independent laboratory, suggesting that the high values were not related to analytical problem. Base metal in those samples came back with values in the range of the average. Erratic distribution of the gold even in the homogenized standard samples seems to be the reason of occasional higher-grade values in standard samples.

DUPLICATE AND STANDARD

The quality control procedure for check sample results for exploration and definition drill holes is slightly different. All values (above a certain predetermined minimum grade) with a variance greater than 20% with the current average are identified. These samples must be reviewed by the supervising geologist and either they are averaged or excluded. The minimum significant grades are presented in Appendix B.

ASSAY CONTROL MEASURES

For exploration and definition drill hole samples, each assay report from an independent laboratory is accompanied by a separate report on duplicate and standard assay results. Data checking of the independent lab results for accuracy and precision by Laronde is qualitative.

DISCUSSION OF ADEQUACY OF SAMPLING, SAMPLE PREPARATION, SECURITY AND ANALYTICAL PROCEDURES

It is the authors opinion That the use of independent laboratories, the quality control measures and the automated check assay variance monitoring have been adequate control measures employed in the past at Laronde. Specific monitoring for accuracy and precision, which might include control charts, precision plots and other scatter plots might also useful as a double check for laboratory-based and field based sampling errors.

16. DATA VERIFICATION

To the author's knowledge, all the intercepts reported in this document (see section 19.5 and li appendix B) including those reported in previous years have had check assays.

The inclusion or exclusion of specific assay data completed during the sample data verification process prior to 2002 was completely reviewed by Laronde geologist team (actual and former members) before the 2002 mineral reserve report. The supervising geologists including the author, prior to the estimate of the 2003 Laronde mineral reserves and resources estimate verified the 2002 sample data.

The results of the 2003 mineral reserve and resource estimate was compared to those of the 2002 estimate. No anomalies were discovered by the author.

Roscoe-Postle and Associates reviewed the 2002 Laronde mineral reserve and mineral resources estimate (RPA 2002) and did not find any errors due to sampling or analytical procedures.

17. ADJACENT PROPERTIES

Mining activities at the Laronde Mine are located on the Laronde and El Coco properties as reported in the section Property description and location section 4. Other properties own by the company along the Laronde-Bousquet-Doyon Belt will not be discussed in this document because it is outside of the terms of reference.

18. MINERAL PROCESSING AND METALLURGICAL TESTING

This item will not be directly discussed in this document. Please refer to the reports by Girard et al. (1999, 2001) and Roscoe Postle Associates (1999, 2001, 2002).

19. 2003 LARONDE MINERAL RESOURCE AND MINERAL RESERVE ESTIMATE

19.1 INTRODUCTION

The 2003 Mineral Resource and Mineral Reserve Estimates combines MINERAL RESOURCE grade and tonnage information and MINERAL RESERVE data and has served as a base for decisions on major expenditures at the Laronde division. The definitions follow those adopted by the CIM and set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects.

The CIM Standards describe completion of a Preliminary Feasibility Study as the minimum prerequisite for the conversion of Mineral Resources to Mineral Reserves. A Preliminary Feasibility Study is a comprehensive study of the viability of a mineral project that 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, and where an effective method of mineral processing has been determined. This Study must include a financial analysis based on reasonable assumptions of technical, engineering, operating and economic factors and evaluation of other relevant factors that 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.

The report by Girard et al. (1999) and Roscoe Postle Associates (1999) are the feasibility reports that formed the basis for upgrading the mineral resource occurring above level 220 at the Penna shaft into probable reserves once the shaft station drilling programs were completed in these areas in 1999. The report by Girard et al. (2001) provided the technical and economic factors necessary to transform the mineral resource between levels 220 and 236 into probable reserves when the level of diamond drilling information was increased in 2000. The report by Provencher (2002) (appendix B) provided the technical and economic factors necessary to transform the mineral resource between levels 236 and 245 into probable reserves with the diamond drilling information collected in 2001. Finally, the pre-feasibility study by Emond (2003) (appendix B) provided the technical and economic factors necessary to transform the major part of the mineral resource between levels 245 and elevation 2200m (2800m depth) west of section 6600E into probable reserves with the new diamond drilling information in 2002.

19.2 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.

INFERRED 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 reasonable 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, pits, workings and drill holes.

The inferred mineral resource for 20 North Gold and zone 7 below level 220 and located in the western extension of the deposit (west of 6600E) in the 2003 mineral resource estimate has not been sufficiently drilled. According to the statistical analysis, the western portion of the 20 North zone (below level 220 and elevation 2200m) and all the resources located below elevation 2200m are not sufficiently sampled to be able to classify it higher than inferred mineral resource. Waste-rock dilution and the percent extraction are not considered in the inferred mineral resource estimate.

INDICATED 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.

The actual economic criteria for mineral reserves based on the different economic studies is 55$ C/tm. Small portions of the 20 North Gold zone located below level 220 in the western extension of the probable reserves did not respect that criteria ranging between 50 to 55$ C/tm, potential increase in the gold price could increase the values of this part of the lens but for now, those tons are considered as indicated resources. Zone 6 and zone 22 located to the east of the Penna shaft lenses complex are similar in grade, size and shape compared to the lenses that were mined out at shaft #2 and in reserve blocks 73 & 74. Drill hole spacing is also similar to allow a good estimate of those lenses. But no mining plan is available on those lenses yet and prevents upgrading higher than indicated mineral resource.

MEASURE RESOURCE

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, 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.

No measured mineral resources occur in this reserves and resources estimate. The mineral resources at shafts no.1 and no.2 are not included and considered as uneconomic with the actual knowledge of those resources and current metals prices.

19.3 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.

PROBABLE 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.

At Laronde, regular adequately spaced grid-shaped patterns of diamond drill hole intercepts through a known horizon (such as zone horizons 4, 5, 6, 7, 20 North and 20 South) are sufficient to quantify a inventory of economic massive to disseminated gold-copper or zinc silver mineralisation as probable mineral reserve (see Table 6). A grid spacing in the order of 100 metres by 120 metres is adequate for massive to disseminated gold-copper mineralisation (5, 6, 7, 20 North Gold and 20 South). Because zinc-silver (20 North Zinc-type) mineralisation has demonstrated to have poorer continuity, an interval of 80 metres by 50 metres is necessary to classify probable reserves. Disseminated gold zones (like zone 4 and 5C) have a demonstrated poor continuity, 20 metres by 40 metres drill spacing are necessary to estimate a probable reserve.

PROVEN 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.

A Laronde, proven mineral reserve blocks are very well established by underground openings and a regular pattern of drill holes. Generally, the definition drill hole pattern is in the order of at least 40 metres by 40 metres for massive sulphide mineralisation (in practice proven reserves are drilled at 15 metre by 15 metre to 15 metre by 30 metre spacing). Mineral reserves in area extending at least one sublevel above or below openings and 15 metres east and west of a sampled opening are considered as proven mineral reserves.

TABLE 7. DRILL HOLE SPACING FOR INDICATED RESOURCES AND PROBABLE RESERVES

ZONE	VERTICAL SPACING	HORIZONTAL SPACING
20 North Gold	140m	140m
20 North Zinc	80m	50m
20 South	80m	60m
7, 6 and 22	60m	50m

19.4 CHANGES IN THE ESTIMATE SINCE JULY 2002

Mining reserves and mineral resources at Laronde, and at the Penna shaft in particular, since July 1st 2002 have been affected both by new diamond drilling results, new geostatistical results and by the upgrading of the milling and mining facilities corresponding to the 2001 feasibility report to 7000 short tons per day.

A) Mining and milling facilities were upgraded to 7000 short tons per day at the beginning of October 2002. This increased capacity will allow decreasing mining cost above 45.00C$/short ton (49.60C$/metric ton) during the five years plan 2002-2006 (Girard et al 2002). In addition, it is anticipated that, with the exhaustion of gold reserves around the Penna shaft, from the year 2010, the cyanidation of the ore will no longer be necessary. This would result in a decrease in total mining-milling costs in 2011 to 37.00C$/metric ton (33.50C$/short ton).

Despite the lowering of the production cost related to the increasing production capacity, cut off and economic parameters for reserves and resources were kept like in 2001, 2002.

B) Economic evaluation of the NSR cut off for tonnage located below level 215 was completed in 2002 as recommended by Roscoe Postle (2002). The economic analysis report by Emond (2003) indicated that mining of indicated resource was economic down to elevation 2200m (2800m depth) at an operating cost from 50C$ up to 59C$ per ton in increasing increments according to depth below the bottom of the Penna shaft. This report allowed new indicated resources to be converted into a probable mineral reserve.

C) Definition and exploration drilling continued during 2002 from levels 86, 170, 194 and 215 and from diamond drill based in the lower mine ramp access level 194 and 215. The 2002 exploration and definition drilling results at the Penna shaft are summarised below:

1. The level 86-exploration drift (EX-86-33E) was extended toward the east on the El Coco property to reach 9373E. Exploration drilling was completed in 2002 covering the immediate down dip (steeply dipping to the west) possible extension of the shallow zone 22 lens up to section 9000E.

2. Definition drilling of the western edge of zone 20 North (and 20 South) from the level 194 exploration drift allowed reduction of the spacing between drill hole to 40m x 40m (in the 20 North zone) and 80m vertical x 40m horizontal in the 20 South zone up to section 7000E.

3. Exploration drilling of zone 20 North Gold from level 194 and 215 below level 245 in 2002 transferred a portion of mineral resource located east of section 6600E into probable reserves with a spacing between drill hole intercepts of about 100m vertical x 120 m horizontal.

4. Finally, three new drilling intercepts were returned from zone 20 North Gold in the western extension of the lens at depth: 3194-56A, 3215-27B and 3215-22F. Those holes allow to extend the inferred mineral resources to the west and confirm the potential for a second parallel lens (a split of the 20North Gold zone) at depth and toward the west.

5. Resources estimate has been extended down to elevation 1800m (3200m depth).

D) New geostatistical analyses of zones 20 North were conducted in 2001 and 2002. Most of the parameters were established in the past following studies made by Wheeler (1997, 1999), Dagbert (2000) and D'Amours (2001, 2000a and 2000b) concerning: cutting factors, specific gravity adjustment factor, revised inverse power values, metal distribution. In 2002, new studies were conducted concerning variation of the assays relating to the orientation of the samples (chip and drill hole sampling bias) D'Amours (2002).

The north-south sample calibration factor was reevaluated with new reconciliation study information available (as recommended by Roscoe Postle 2002). With the increasing amount of information coming from north south drill holes in 2002, a variation in gold grade was observed compare to azimuth drill hole that were completed from the Penna Shaft station in the past. Geostatistical analyses in the 20North Gold zone was conducted by D'Amours (2002) to include in the new reserve estimate in July 2002 (Legault 2002c).

This study was conducted in order to compare results obtained in north-south samples from chip and drill hole with azimuth drill hole. This study made with simple q-q plot study identifies a difference related to the orientation of the samples. This study quantified the underestimation of the gold grade in north-south sample because of the under representation of gold remobilized along north-south fractures. In reserves estimate 2002-02 (Legault 2002c), chip and drill hole samples oriented between azimuth 170 and 190 degrees in the mine grid, in the 20 north gold zone, had the following grade calibrations applied: chip samples between 1 and 10 g/t were calibrated using a 1.2 factor; whereas drill hole samples between 3 and 10 g/t were calibrated using a 1.1 factor; and drill hole samples above 10 g/t were calibrated using a 1.2 factor.

A 2003 reconciliation study (Gosselin 2003 in prep.) evaluated the impact of these new geostatistical parameters. It appears that these calibration factors resulted in an overestimation of the gold grade compared to the mill. Changes in order to reduce these calibration factors were applied in the present reserves estimate. Chip and drill hole samples oriented between azimuth 170 and 190 degrees in the mine grid, in the 20 north gold zone, have now the following calibration factors applied: chip samples between 1 and 10 g/t were calibrated using a 1.1 factor, whereas drill hole samples between 3 and 10 g/t were calibrated using a 1.1 factor and drill hole samples above 10 g/t were not calibrated.

E) The specific gravity calibration factor of 1.1 was left unchanged (despite the recommendation made by Roscoe Postle 2002). The results observed in the 2003 reconciliation study (Gosselin in prep.) summarized in appendix B shows an excellent correlation between calculated tons send to the mill (with the 1.1 SG calibration factor) with the reconciled production.

G) As recommended by Roscoe Postle (2002) tonnage located in the western flank of the zone in the block PB194 was reclassified in indicated resources in reason of their grade, thickness, and location outside of the main trend of the zone. Zone 6 (PB63) was drilled on a smaller spacing (definition) and reevaluated with a significant decrease in tonnage.

19.5 ESTIMATION METHODS

At Laronde, mineral reserves are estimated either using the polygonal method (by longitudinal section) but mostly using inverse power distance (IPD) block modelling methods. Indicated resources are all estimated using inverse power distance (IPD) block modelling methods whereas inferred mineral resources are always estimated using the polygonal method. In 2003, the mineral reserves for only two lenses: zone 6 (block 62) and zone 7 (block 72) were estimated using the polygonal method (as in the 2002 estimate). All the other lenses (or portion of lenses) included in the mineral reserves were estimated using the IPD block modelling method.

The polygonal method (either transverse or longitudinal section) has been used at Laronde since the start of commercial operations (see Blackburn, 1991). It is a simple and quick method that was adequate because the orebodies at shaft no.1 were: 1) narrow and planar; and 2) the gold values, which was the only payable metal that was tracked, were relatively uniform and well distributed (within a narrow range 0.1 to 1.5 opt). The disadvantage of the polygonal method is due to its tendency to bias results (because the method underestimates the grade in low-grade areas and, conversely, overestimates the grade in high-grade areas; see Glacken, 1999)

Inverse power distance block modelling methods have been used at Laronde since 1998 for estimating reserves at the Penna and no.2 shafts where the zones: 1) are larger, irregularly shaped and thicker; 2) contain four payable metals (Au, Ag, Cu and Zn) which have a wide range of values; 3) have variable densities; and 4) the metals and densities are distributed irregularly in three dimensions. The method uses is also quick, relatively simple, statistically based and, to an extent, accounts for grade variability according to distance and direction (based on variograms).

Because of the widely spaced results, inferred mineral resource estimates at Laronde are determined by polygonal methods.

POLYGONAL BY LONGITUDINAL

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

A polygonal shaped perimeter surrounding each intercept point which links the mid-distance marks to surrounding intercepts is then drawn on the longitudinal section. The surface area of each polygon is measured off from the longitudinal section. The tonnage is the product of the polygon surface area multiplied by the intercept thickness and the specific gravity determined for each individual drill hole intercept (or exceptionally for the specific gravity determined for a zone in PB62 and 72).

At Laronde, drill hole data and chip samples are treated separately. The economic limits and horizontal thickness of a particular drill intersection is entered into an ACCESS type database

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file. The length-weighted average grade for gold, silver, copper, zinc, lead and specific gravity as well as the co-ordinate of the mid-point position of each intercept is then calculated. Chip sample data collected from development headings crosscutting ore zones are treated directly by DATAMINE ore reserve estimation software from which grade and 3-D positional data is then extracted.

The grade and positional data are then registered in a CSV (comma-delimited format) type file. DATAMINE software derives a report that describes longitudinal area for each polygon that is imported into ACCESS format for calculation and final ore reserve report.

INVERSE DISTANCE BLOCK MODEL

In an IPD estimate, samples are weighted inversely to their distance from the estimation point. This method's principle is that the grade of a particular unit block is more like samples that are closer to it; closer samples should therefore be weighted more highly.

The method consists firstly of building three-dimensional envelope (wireframe model) of a particular orebody using diamond drilling, chip sampling and underground mapping results. The orebody envelope is then filled with several unit-sized blocks to which a grade is then interpolated. The grade of an individual block within the orebody envelope is assigned to it using the inverse-distance-power estimation method. In practice, surrounding samples (those comprising various drill-hole intercepts for example) are weighted inversely to their linear distance(power) from a particular estimation point (the unit-sized block). For example:

Inverse distance squared grade estimate = weight each sample's grade inversely to distance(2)

and

divide by sum of inverse distance(2)

Different power levels for searching distance are use at Laronde in function of the information density, and of the zone. Power levels used at Laronde for IPD (Inverse Power Distance) estimation are 1, 2 and 3 (distance(1), distance(2) distance(3)).

At Laronde, the block model method involves using DATAMINE ore reserve modelling software. Once the envelope is created, DATAMINE captures all the sampling data inside the envelope (wireframe). The intersections are then composited so that the grade values for each drill-hole or chip sample can be represented over standard unit lengths.

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

Another problem with the inverse distance estimation method is that it can over-predict grades if the assay data has a positively skewed distribution. This can be observed by studying the statistics of the assay data. The over-prediction can be overcome by applying a statistically based topcut to the assays (see Glacken, 1999).

The discussion below of the various parameters used in the 3D inverse distance estimation is based on the analysis by Christian D'Amours (D'Amours, 2001, 2000a and 2000b) and a review by Dagbert (2000).

SAMPLE SELECTION

At Laronde, the selection of samples by zone along drill holes and chip sample runs for use in the estimation is based on the ACCES database table of intersections (M-zone), which are compiled by the geology department. In the case of chip sample intervals, the wireframe model physically makes the sample selection.

PERIMETER FILE

The perimeters are used to estimate the reserves of a particular stope. They consist of a pair of lines that define the floor and ceiling of each stoping block.

LENGTH OF COMPOSITE SAMPLES

The block modelling method requires that all the samples have an equal length. An analysis of samples from all four zones showed that 45-66% of the sample lengths had a length of 1.5 metres and that the gold and zinc grades were uniformly distributed throughout the sample intervals.

TOPCUT GRADES

The sample results for all the metals were examined using both frequency histograms (plotted either on a decimal or logarithmic scale) and cumulative probability plots. Gold and Silver values used in the block models are variably topcut depending on the zone (Table 8). The topcut grades used in polygonal estimation and presented on the zone longitudinals are slightly different (Appendix B).

VARIOGRAPHY TO ESTABLISH SEARCH PARAMETER FILES

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

In order to look for directions of variability, the variography was measured and compared along different directions until the best results for range were found. Orthogonal intermediate axes were then established. The modelled variogram parameters shown below are used in the search parameter files for each zone. Note that directional parameters are also used to weight distance proportional to variography in any particular direction. The whole idea behind the concept is: two nearby samples have values more similar than two samples that are far apart and the distance where samples could be considered similar is variant with directions. It is called anisotropy of variation.

The zone anisotropies derived from the variography are presented in Table9 (the search and estimation parameter files and a DATAMINE reference list can be consulted in Appendix B).

TABLE 8: TOP CUT GRADES FOR BLOCK MODELLING

ZONE	AU (G/T)	AG (G/T)
6	51.4290	857.1430
7	34.0000	200.0000
20 North Gold	51.4286	857.1425
20 North Zinc	67.0000	
20 South	17.1429	

TABLE 9: ZONE ANISOTROPIES

ZONE 20 NORTH GOLD Block 191-192-193	SIGMA 1 AZIMUTH/PLUNGE--RANGE (M)	SIGMA 2 AZIMUTH/PLUNGE--RANGE (M)	SI AZIMUTH/PLU (M)
Au	N094/-63--140	N263/-28--140	N356/-
Ag	N094/-63--110	N263/-28--60	N356/-
Cu	N094/-63--140	N263/-28--120	N356/-
Zn	N064/-32--45	N221/-58--35	N328/-
Pb	N064/-32--35	N221/-58--35	N328/-

ZONE 20 NORTH GOLD Block 194-195	SIGMA 1 AZIMUTH/PLUNGE--RANGE (M)	SIGMA 2 AZIMUTH/PLUNGE--RANGE (M)	SI AZIMUTH/PLU (M
Au	N094/-63--140	N263/-28--140	N356/-
Ag	N094/-63--110	N263/-28--60	N356/-
Cu	N094/-63--140	N263/-28--120	N356/-
Zn	N064/-32--45	N221/-58--35	N328/-
Pb	N064/-32--35	N221/-58--35	N328/-

ZONE 20 NORTH ZINC Block 201-202-203	SIGMA 1 AZIMUTH/PLUNGE--RANGE (M)	SIGMA 2 AZIMUTH/PLUNGE--RANGE (M)	SI AZIMUTH/PLU (M
Au	N277/-65--60	N087/-24--60	N178/-
Ag	N277/-65--85	N087/-24--60	N178/-
Cu	N277/-65--100	N087/-24--60	N178/-
Zn	N277/-65--85	N087/-24--50	N178/-
Pb	N277/-65--90	N087/-24--40	N178/-

ZONE 20 NORTH ZINC Block 204-205	SIGMA 1 AZIMUTH/PLUNGE--RANGE (M)	SIGMA 2 AZIMUTH/PLUNGE--RANGE (M)	SI AZIMUTH/PLU (M
Au	N277/-65--60	N087/-24--60	N178/-
Ag	N277/-65--85	N087/-24--60	N178/-
Cu	N277/-65--100	N087/-24--60	N178/-
Zn	N277/-65--85	N087/-24--50	N178/-
Pb	N277/-65--90	N087/-24--40	N178/-

ZONE 20 SOUTH	SIGMA 1 AZIMUTH/PLUNGE--RANGE (M)	SIGMA 2 AZIMUTH/PLUNGE--RANGE (M)	SI AZIMUTH/PLU (M
Au	N260/-68--80	N110/-20--60	N016/-
Ag			
Cu			
Zn			
Pb			

ZONE 6	SIGMA 1 AZIMUTH/PLUNGE--RANGE (M)	SIGMA 2 AZIMUTH/PLUNGE--RANGE (M)	SI AZIMUTH/PLU (M
Au	N239/-76--60	N094/-15--50	N001/-
Ag	N269/-10--80	N134/-76--60	N001/-
Cu	N092/-06--35	N214/-78--35	N001/-
Zn	N269/-12--20	N001/-10--20	N130/-
Pb	N094/-19--40	N244/-69--32	N001/-

ZONE 7	SIGMA 1 AZIMUTH/PLUNGE--RANGE (M)	SIGMA 2 AZIMUTH/PLUNGE--RANGE (M)	SI AZIMUTH/PLU (M

Au	N239/-76--60	N094/-15--50	N001/-
Ag	N269/-10--80	N134/-76--60	N001/-
Cu	N092/-06--35	N214/-78--35	N001/-
Zn	N269/-12--20	N001/-10--20	N130/-
Pb	N094/-19--40	N244/-69--32	N001/-

ZONE 22	SIGMA 1 AZIMUTH/PLUNGE--RANGE (M)	SIGMA 2 AZIMUTH/PLUNGE--RANGE (M)	SI AZIMUTH/PLU (M
Au	N239/-76--60	N094/-15--50	N001/-
Ag	N269/-10--80	N134/-76--60	N001/-
Cu	N092/-06--35	N214/-78--35	N001/-
Zn	N269/-12--20	N001/-10--20	N130/-
Pb	N094/-19--40	N244/-69--32	N001/-

BLOCK DIMENSION

The size of the blocks (or cells) that will fill the wireframe model is a function of sample grid size and the orebody geometry. The rule of thumb is that the blocks should never be smaller than 1/3 of the sample grid spacing. Because the delineation drill intercept spacing at Laronde is roughly 30 metres by 10 metres, the block size for all the zones is 12 metres by 12 metres by 3 metres. In order to match block volumes within the wireframe, two levels of sub-splitting allow for a minimum 3 metres by 3 metres by 1 metre sub-block (roughly 30.6 tonnes). The rest of the wireframe in thickness is completed by sub-block 3 metres by 3 metres by 0 (minimum infinite) to 1 metre.

INTERPOLATION METHOD

The choice of power used in the IPD interpolation method (either distance(2) or distance(3)) is considered to be subjective and subject to refinement (D'Amours, 2001). However the inverse distance(3) method previously used at Laronde (Wheeler, 1997) was replaced in the 20 South zone by the inverse distance(2) in 2001 reserve/resource estimates for two reasons:

1. The combination of the narrow pancaked-shaped geometry of the search ellipsoid (in the plane of mineralisation) and the imprecision of the drill hole data result in too many samples being ignored in the estimate because their position slightly off the wireframe.

2. The inverse distance cubed method is closer to a polygonal estimation.

This decision was supported by Dagbert (2000).

The inverse distance(3) method was kept in certain zones (20 north gold and zinc in high density information areas and in Zone 7 following validation of the parameters by Dagbert (2001).

NOTES

The factors used in the mining reserve and mineral resource estimate are summarised below:

1. All sections were drawn using Borehole manager database and Autocad software. All polygonal area and block model grade and tonnage estimate calculations were made on a PC using DATAMINE software and the results compiled in a Microsoft Access format databank.

2. Each potentially economic intercept was calculated using a minimum horizontal thickness of 9.2 feet (2.8 metres).

3. The maximum polygon radius is set at 275 metres. The inferred resource envelope was calculated to a depth of 3200 metres (1800-metre elevation) over an average strike width of 600 metres.

4. In the past, the specific gravity (SG) was assigned to each zone at shafts no.1 and no.2 and varied between 3.4 and 4.2 (see Appendix A). Several sources of information were consulted to fix the specific gravity for each zone: The report dated February 3, 1981 by Anton Adamcik and the 1983 M.Sc. thesis by Demetrios George Eliopoulos have been used for zone no. 5 at shaft no. 1. The specific gravity of ore samples of zone no. 6 at shaft no. 2 is compiled in the report by Gervais (1997).

5. From 1998 to 2001, core samples from the stope delineation flat holes, definition and deep exploration holes have had specific gravity determinations on the pulps (calculated by an independent laboratory). The results were compared statistically to the sample rock types by zone. For each zone, a SG value is assigned to samples that do not have a SG determination. The SG values are then composited along with the other metals. The SG of a model block is assigned to it using the nearest neighbour's method and the average SG of the stope is then determined.

6. Since 2001, resampling for SG determination in the ore zone of all available drill holes (definition and exploration drill hole kept at the core shack) was completed. Most of the new zones intercepts since 2001 (all the flat delineation, definition and exploration drill hole) were analyzed for SG at the same time than metal content (only exception are up and down delineation drill holes). SG is now estimate using block-modelling technique for all wireframed lenses. The S.G. values are composited along with the other metals. The S.G. of a model block is assigned to it using the nearest neighbour's method and the average S.G. of the stope is then determined.

7. A tonnage calibration factor of 1.1 has been applied to all blocks of reserves and indicated resources blocks RD22, RD63 and RD194 in order to reflect the observed variance between forecasted and mined tonnage which has been observed at the Penna Shaft (2001, 2002 and 2003 (in progress) reconciliation study) (summary in Appendix B).

8. A tonnage calibration factor of 1.05 has been applied to indicated resources block RD62 and RD72 and all inferred resources RF74, 194, 195,

196 and 197 that were calculated using polygonal method (summary in Appendix B).

9. A dilution study (CMS survey analysis) of Penna shaft stopes allows the determination of the equivalent meter factor that represent the average dilution (metre of over brake) in the footwall and hangingwall of the ore zone for each stope. Portion of lenses are then characterized with representative equivalent meter factors in booth walls, this allow the calculation of a dilution (%) variable in function of the thickness of the ore zone. (see the equivalent meter factor applied for each bloc in the listing Appendix D).

10. Dilution grade for the reserves (except reserves block 72 and 62) and indicated resources (except block 22 and 63) were calculated for all specific stope using block model method. Since 2001, the footwall and the hangingwall of the different ore zone are modelized and evaluated the same way than the ore zone using specific parameters determined by variography study: anisotropy, searching ellipsoids. Sampling of the wall material is now systematic over at least 3.5m (core length) to allow the wireframing of a 2.8m true thickness skin on booth side of the ore zone (to calculated dilution grade and density) in which the equivalent meter factor is applied for the dilution estimate (tonnage calculation).

11. A specific dilution factor, which varies between 10% and 20% by weight, was applied to probable reserves blocks 62 and 72 and indicated resource blocks 63. Indicated resource block 22 was undiluted by involuntary omission in regard of our internal standard practice. No dilution factor was applied to inferred mineral resource (see Appendix A).

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12. Dilution grade for reserve bloc 62 and 72 was estimated using drill holes representative assay data in the wall rock (see Appendix A). Resource block 63 was estimated using same dilution grade than reserve block 62 in reason of apparent similar geological context and properties, this procedure should be revised in 2003 using specific information for that lens.

13. The economic viability ofeach intercept was tested by using a complex Excel logarithm (Geo5a2001.xls) developed by the Laronde mill department in January 1997 and modified using the following metal prices: 300$US/oz gold, 5$US/oz silver, 0.80$/lb copper, 0.50$/lb zinc and a $US/$C exchange rate of 1.50 (as in 2002). Smelting charges for zinc and copper remain unchanged.

14. In order to fix the economic limits of the various ore-zone, each modelled stoping block or polygon sample point had an net smelter return (NSR) value calculated for it using the Geo5a2001.xls file modify with metal prices: 300$US/oz gold, 5$US/oz silver, 0.80$/lb copper, 0.50$/lb zinc and a $US/$C exchange rate of 1.47 (see appendix D). Smelting charges for zinc and copper remain unchanged. Average metal grade for each mining bloc in the reserve is use to evaluated the NSR value of each unit block within the wireframe (see detailed listing appendix D). Using DATAMINE generated NSR longitudinal sections, an NSR cut-off of 55$C (above level 220) to 59$C/tonne (increasing with depth between level 220 and elevation 2200m) was used to determine the economic limits.

15. The minimum cut-off for zone 20 North Gold where it is immediately in contact with mineral reserves of 20 North Zinc has been established as

39$/tonne (see Appendix B). The bulk NSR must be greater than 55$/tonne.

16. The minimum cut-off for areas of 20 North Zinc which mining is planned later than 2010 has been set at 45$/tonne.

19.6 RESULTS

METRIC RESOURCE AND RESERVE ESTIMATE

					DILUTED GRADE		
CATEGORY	ZONE		Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)
Probable	20N AU	SOMME	4.613	48.352	0.53	1.11	0.10
Probable	20N ZN	SOMME	1.037	99.325	0.10	6.61	0.78
Probable	20S	SOMME	4.099	24.455	0.21	1.22	0.14
Probable	6	SOMME	3.902	56.011	0.18	4.21	0.01
Probable	7	SOMME	5.624	33.895	0.35	1.66	0.07
Probable	TOTAL	SOMME	3.455	63.187	0.37	2.93	0.32

				TOTAL PRODUCTION (DILUTED)			
CATEGORY	ZONE		TONS (Met.)	Au (g)	Ag (g)	Cu (Kg)	Zn (Kg)
Probable	20N AU	SOMME	17,923,324	82,673,453	866,621,140	95,547,317	199,
Probable	20N ZN	SOMME	9,949,057	10,317,708	988,186,977	10,031,865	657,
Probable	20S	SOMME	1,630,963	6,685,886	39,885,745	3,454,800	19,
Probable	6	SOMME	61,823	241,236	3,462,753	111,445	2,
Probable	7	SOMME	1,025,299	5,766,720	34,752,706	3,541,294	16,
Probable	TOTAL	SOMME	30,590,466	105,685,002	1,932,909,320	112,686,721	896,

					DILUTED GRADE		
CATEGORY	ZONE		Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)
Proven	20N AU	SOMME	4.561	75.838	0.79	1.76	0.13

Proven	20N ZN	SOMME	0.951	119.055	0.10	7.67	1.02
Proven	20S	SOMME	4.935	59.320	0.27	3.05	0.44
Proven	TOTAL	SOMME	2.679	97.671	0.39	4.95	0.62

TOTAL PRODUCTION (DILUTED)

CATEGORY	ZONE		TONS (Met.)	Au (g)	Ag (g)	Cu (Kg)	Zn (Kg)
Proven	20N AU	SOMME	2,992,794	13,650,443	226,967,755	23,630,347	52,
Proven	20N ZN	SOMME	3,802,633	3,616,387	452,724,178	3,703,858	291,
Proven	20S	SOMME	416,488	2,055,387	24,706,187	1,143,943	12,
Proven	TOTAL	SOMME	7,211,915	19,322,217	704,398,121	28,478,148	357,

DILUTED GRADE

CATEGORY	ZONE		Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)
Broken Ore	Broken Ore	SOMME	4.533	68.200	0.33	4.05	0.46
Broken Ore	TOTAL	SOMME	4.533	68.200	0.33	4.05	0.46

TOTAL PRODUCTION (DILUTED)

CATEGORY	ZONE		TONS (Met.)	Au (g)	Ag (g)	Cu (Kg)	Zn (Kg)
Broken Ore	Broken Ore	SOMME	20,382	92,389	1,390,057	68,183	824,
Broken Ore	TOTAL	SOMME	20,382	92,389	1,390,057	68,183	824,

CATEGORY	ZONE		DILUTED GRADE Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)
Indicated Resource	20N AU	SOMME	4.034	6.577	0.12	0.03	0.00
Indicated Resource	El Coco	SOMME	4.104	19.024	0.23	0.52	0.08
Indicated Resource	6	SOMME	3.338	27.306	0.17	1.96	0.06
Indicated Resource	TOTAL	SOMME	3.936	14.851	0.17	0.55	0.04

CATEGORY	ZONE		TOTAL PRODUCTION (DILUTED) TONS (Met.)	Au (g)	Ag (g)	Cu (Kg)
Indicated Resource	20N AU	SOMME	266,572	1,075,300	1,753,310	316,875
Indicated Resource	El Coco	SOMME	217,197	891,366	4,131,867	497,282
Indicated Resource	6	SOMME	104,297	348,114	2,847,951	175,095
Indicated Resource	TOTAL	SOMME	588,066	2,314,779	8,733,129	989,252

METRIC RESOURCE AND RESERVE ESTIMATE

(PROVEN, PROBABLE, BROKEN ORE AND INDICATED RESOURCE)

		DILUTED GRADE Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)
TOTAL	SOMME	3.306	69.456	0.37	3.30	0.38

		TOTAL PRODUCTION (DILUTED) TONS (Met.)	Au (g)	Ag (g)	Cu (Kg)	Zn (Kg
TOTAL	SOMME	37,902,267	125,299,372	2,632,533,946	141,050,900	1,249,

METRIC RESOURCE AND RESERVE ESTIATE

			DILUTED GRADE				
CATEGORY	ZONE		Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)
Inferred Resource	20N AU	SOMME	5.986	11.482	0.33	0.03	0.02
Inferred Resource	7	SOMME	4.103	57.554	0.49	1.61	0.11
Inferred Resource	TOTAL	SOMME	5.923	13.024	0.33	0.08	0.02

			TOTAL PRODUCTION (DILUTED)			
CATEGORY	ZONE		TONS (Met.)	Au (g)	Ag (g)	Cu (Kg)
Inferred Resource	20N AU	SOMME	20,192,510	120,869,718	231,841,927	66,489,544
Inferred Resource	7	SOMME	699,522	2,870,138	40,260,275	3,427,657
Inferred Resource	TOTAL	SOMME	20,892,032	123,739,856	272,102,202	69,917,201

METRIC RESOURCE AND RESERVE ESTIMATE

(INFERRED RESOURCE)

		DILUTED GRADE				
		Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)
TOTAL	SOMME	5.923	13.024	0.33	0.08	0.02

TOTAL PRODUCTION (DILUTED)				
TONS (Met.)	Au (g)	Ag (g)	Cu (Kg)	Zn (Kg

TOTAL	SOMME	20,892,032	123,739,856	272,102,202	69,917,201	16,800

19.7 EXCEPTIONS FOR THE YEAR 2003 RESERVE-RESOURCE CALCULATIONS

At the Penna shaft, the proven mineral reserves were estimated using a combination of chip sample results, diamond drill hole information and broken ore inventories that were current to December 31st 2002.

The grade of broken ore stored in the stopes, orepasses, surface stockpiles and mill silos on December 31st 2002 (of the Penna shaft ore material) are estimated using non-reconciled muck sample results. The tonnage estimate of the broken ore is based on the December 31st 2002 inventory estimated by the engineering department.

At the Penna shaft, probable reserves and indicated and inferred mineral resources were estimated using the most up to date diamond drill hole and chip sample information and are not restricted to a December 31st 2002 deadline.

Exceptions to the procedures described in this estimate are listed below:

1. Some exploration holes below level 215 have incomplete collar location data or down hole surveys. Final collar surveys are generally done when drilling planned on a particular set up is completed. For the 2003 estimate, a preliminary survey was completed to locate the drill on planned collar location to insure proper drill hole spacing. The likely changes will not be significant;

2. Exploration drill holes 3215-17, 18, 20, 33 and 34 do not have final collar surveys;

3. Exploration drill holes 3215-32, 33,34 do not have down hole gyroscopic surveys;

4. Check assays were pending for drill hole 3215-39A for the 20 North gold and zinc zone intercepts.

5. An involuntary omission occurs during the resources estimate of indicated resources block RD22 where dilution was set at 0%.

6. Reserves blocks PB62 and PB72 were evaluated using estimated SG whereas all the other reserves were evaluated by wireframe block model method.

7. Dilution grade for reserves block PB62, PB72 and indicated resource block RD63 were estimated with the available surrounding information whereas all the other lenses in reserve and indicated resource are evaluated by wireframe block model method (except RD22 as mentioned above).

8. Analysis of interpretation by wireframe showed that in some circumstances (namely, local deviations in the wireframe and certain intersections that were slightly outside the wireframe), samples were not being interpolated. In order to counter this effect, all the holes intercepts that were located outside of the wireframe were translated into the wireframe along a horizontal north-south vector (see appendix B).

9. The octant method was not used in the estimation of the lower Penna shaft zones (193, 194, 195, 203, 204, 205 and 212).

10. The search ellipsoid in reserve blocks 194, 195, 204 and 205 was flattened from an 86 degree dip to a 76 degree dip in order to reflect the observed changed in the dip in the wireframe at depth.

Except for those issues discussed in the reports by Girard et al (1999, 2001) and Roscoe Postle and Associates (1999, 2001 and 2002), there are no other known issues, which may affect the 2001 Mineral Resource and Mineral Reserve Estimate at Laronde

20. OTHER RELEVANT DATA AND INFORMATION

This item will not be discussed because it is outside of the terms of reference. Refer to Girard et al (1999, 2001) and Roscoe Postle and Associates (1999, 2001 and 2002).

21. INTERPRETATION AND CONCLUSIONS

The following conclusions can be made:

1. The Penna shaft orebodies attain production levels of 7,000 tonnes per day with unit cost per tons 51.60$C/st in 2002 slightly lower than budgeted (53.01$C/st).

2. Block model inverse power distance estimation appears to be a suitable method for mineral reserve estimations at the Laronde Division Penna Shaft.

3. New drilling information in reserves blocks 202 and 203 allow completing the interpretation of the western extension of the zinc zone where the zone is duplicated (in reason of fold/fault or primary features of the lens) with the creation of reserve block 200 (since 2001 and completed in 2002).

4. The decrease in mineral reserves for 20 South zone block 212 is the results of new diamond drilling information (see Appendix B) for major changes in 2002 estimate.

5. Definition diamond drilling in the 20 North Gold zone between levels 215 and 170 help to refine geological interpretation and ore wireframing and results in a slight increase in estimate gold ounces contained.

6. The completion of the surface and underground diamond drilling campaign allow the interpretation and block modelling of Zone 22 on the El coco property.

7. Variography studies of gold grade in function of the sample orientation results in the application of calibration factors to north-south drill hole samples and chip samples that underestimate the gold grade compare to

azimuth drill hole samples.

8. The 2003 Reconciliation Study (Gosselin in prep.) confirmed the underestimation of the tonnage with the actual SG analysis process that was discussed in 2001 and 2002. The 1.1 SG calibration factor that has been introduce since 2001 in the estimate allows a better reconciliation with the mill-processed tonnage.

9. The increasing level of information available in the lower Penna Shaft area in 2002 (between levels 245 and elevation 2200m) allowed the conversion to probable reserves of some 2001 inferred resources with individual stope grades applying dilution parameters and SG correction factor.

10. Increasing gold-grade and thickness below the bottom of the Penna shaft was confirmed by 2001 and 2002 deep exploration campaign. Economic study and increase amount of drill hole intercepts allow the conversion of some part of it. Close to 4.0 million ounces are still estimated in inferred resources at depth and to the west. Level 215-exploration drift will allow increasing the level of information in the 20 North Gold zone at depth in 2003-04.

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22. RECOMMENDATIONS

The following general recommendations can be made:

1. Refresh geostatistical studies for zone 6 (blocks 62 and 63), 7 (blocks 72, 73 and 74) and 22 in order to verify parameters for block modelling calculation of these zones in time for the 2004 mineral resource and mineral reserve estimate;

2. Reserve blocks 62 and 72 should be wireframed and estimated using block modelling methods.

3. Footwall and hanging wall of reserves blocks 62 and 72 and indicated resource blocks 63 and 22 should be modelled by wireframe and evaluated by block modelling techniques. This would complete all the lenses in the proven-probable and indicated resource category.

4. With the increasing information in reserve blocks 193 & 203, a new geostatistical study will be needed to validate or adjust block modelling parameters;

5. Conduct a new geostatistical study on the searching ellipsoid and anisotropy parameters with the observed changes of dip (slightly flatter) at depth in blocks 194, 195, 204 & 205;

6. Continue to review the current inverse distance block models periodically using grade and tonnage reconciliation data;

7. Continue to review the tonnage calibration factor of 1.1 applied to 2003 reserves with year-end reconciliation data at a steady treatment rate of 7000 tpd with the newly commission processing plan at the mill;

8. Review the gold grade calibration factor applied to north-south samples in the 20 N Gold zone in regards of the 2003 production reconciliation study.

2003 will be the first complete year of significant production coming from the 20 North Gold zone in the lower mine;

9. Complete the definition diamond drilling between levels 215 and 170 up to the western limit of the deposit. The budgeted definition drilling program from level 194 that was approved for 2003 will allow to complete this program and continue the drilling toward the east and in the zone 20 South (block 212) and zone 7 (blocks 73 & 74);

10. Pursue the deep exploration program using standard sectional drilling from the 215 level deep exploration drift. The budgeted deep exploration program that was approved in 2003 will improve the level of information toward the west and at depth;

11. Complete the deep exploration program to allow evaluation for the next phase of work (engineering & economic studies).

Respectfully submitted,

Guy Gosselin, P.Eng., P.Geo. [SEAL]
Chief Geologist, Author
Agnico-Eagle Mines Ltd., Laronde Division
May 12th, 2003

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24. DATE

This report is effective by February 19th 2003. It was completed, reviewed and revised by May 12th 2003.

25. ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

This item will not be addressed in this report; Girard et al. (2001) discuss mining operations, recoverability, markets, contracts, environmental considerations, taxes, capital and operating cost estimates, economic analysis, payback and mine life at the Laronde Division.

26. ILLUSTRATION

APPENDIX A

Dilution and S.G. tables

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APPENDIX B

Claim map; mining leases Laronde and El Coco propertie; Ore reserve calculation memos; Assay Laboratory procedures; Standard sample preparation procedure; Detailed schedule of 2002 diamond drilling; List of 2002 drill hole zone intercepts; Proposed 2003 diamond drilling budget; Penna Shaft stope reconciliation table; Dilution study results; Datamine user guide on grade estimation; 2003 block model search parameter and estimation parameter files; List of holes moved in zone 20N Au, Zn and 20S zone in the wireframe for reserve estimate; Duquette 2003 Quality Assurance Quality Control report; Provencher 2002 Laronde level 233-245 economic study; Emond 2003 Laronde deep mine 2003 economic study. Statement of qualifications;

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CONSENT OF AUTHOR

TO:

British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Securities Commission
The Manitoba Securities Commission
Ontario Securities Commission
Commission des valeurs mobilieres du Quebec
Administrator of the Securities Act, New Brunswick
Nova Scotia Securities Commission
Prince Edward Island, Department of Community Affairs and Attorney General
Newfoundland and Labrador Securities Division, Department of Government Services and Lands

I, Guy Gosselin, P.Eng., P.Geol., do hereby consent to the filing of the written disclosure of the technical report titled, 2003 Laronde Mineral Resource and Mineral Reserve Estimate dated May 12, 2003 (the "Technical Report") and any extracts from or a summary of the Technical Report in Agnico Eagle Mines Limited's ("Agnico-Eagle") Annual Information Form dated May 1, 2003 consisting of Agnico-Eagle's Report on Form 20-F under the United States Securities Exchange Act of 1934 for the fiscal year ended December 31, 2002 (the "AIF"), and to the filing of the Technical Report with the securities regulatory authorities referred to above.

I also certify that I have read the disclosure being filed and I do not have any reason to believe that there are any misrepresentations in the information derived from the Technical Report or that the AIF contains any misrepresentations of the information contained in the Technical Report.

This letter is solely for your information in connection with the disclosure of contained in the AIF and the filing of the Technical Report, and is not to be referred to in whole or in part for any other purpose.

Dated this twelfth day of May, 2003.

(signed) Guy Gosselin	(Stamped by Guy Gosselin

Signature of Qualified Person	Ingenieur 121625, Quebec.)

Guy Gosselin P.Eng., P.Geo.

Name of Qualified Person

Guy Gosselin
Agnico-Eagle Mines Ltd., Laronde Mine Division
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Certificate of Author

I, Guy Gosselin P.Eng. P.Geol., do hereby certify that:

1. I am Chief Geologist of: Agnico-Eagle Mines Ltd. Laronde Mine Division 20 Road 395 Cadillac, Quebec, Canada J0Y 1C0

2. I graduated with a degree in Engineering Geology B.Sc. from L'Universite du Quebec A Chicoutimi, Quebec in 1994. In addition, I have obtained a Master degree in Earth Sciences from L'Universite du Quebec A Chicoutimi, Quebec in 1998.

3. I am a member of the following associations:
   Order of Engineer of Quebec (OIQ) registered #121625
   Order of Geologist of Quebec (OGQ) registered #761

4. I have worked as a geologist for a total of 9 years since my graduation from university.

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 am a "qualified person" for the purpose of NI 43-101.

6. I have prepared all sections of the technical report titled "2003 Laronde Mineral Resource & Mineral Reserve Estimate Agnico-Eagle Mines Ltd. Laronde Division" dated May 12th, 2003 relating to the Laronde and El Coco properties. I have worked for Agnico-Eagle Mines Limited ("Agnico-Eagle") at the Laronde and El Coco properties from June 2000 to the date hereof.

7. Other than as set out above, 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 to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

- 2 -

9. I am not independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

   I am a shareholder of Agnico-Eagle Mines Ltd. and I hold options to purchase common shares of Agnico-Eagle Mines Ltd.

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

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

Dated this 12th day of May, 2003

(signed) Guy Gosselin

Signature of Qualified Person

(Stamped by Guy Gosselin
Ingenieur 121625, Quebec.)

Guy Gosselin

Print name of Qualified Person

APPENDIX C

2003 Mineral Reserve and Resource Metric Summary

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APPENDIX D

2003 Mineral Reserve and Resource Metric Calculation at the Penna Shaft by zone, block and stope

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