
Agnico-Eagle Mines Ltd. LaRonde Division

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Preissac, Quebec
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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, northwestern Quebec. Accountable (net of smelter charge) production to December 31st 2004 has been 2.66 million ounces of gold, 19.2 million ounces of silver, 61.8 metric thousand tonnes of copper and 255.7 metric thousand tonnes of zinc from 17.0 metric million tonnes of ore.

In 2005, all the mineral reserves and most of the mineral resources at LaRonde are located near the Penna shaft. The reserves and resources occur 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 2005 mineral resources and mineral reserves estimate at LaRonde was based on the following economic parameters: 360 $US/ounce of gold, 5.42 $US/ounce of silver, 0.95 $US/pound of copper, 0.41 $US/pound of zinc, 1.418 $C/1 $US exchange rate and total mining and milling costs that varied between 37$C/tonne to 55$C/tonne depending on the zone for LaRonde I and 55$C/tonne to 60$C/tonne for LaRonde II. The mineral reserve and mineral resource estimate was made using either inverse-distance block modelling techniques or the polygonal method (for inferred resources).

Total proven reserves for LaRonde I and II is estimated to be 5.891 million metric tonnes grading 3.11 g/t gold for 589,738 ounces of gold, 90.8 g/t silver, 0.43% copper and 4.46% zinc whereas probable reserves are estimated at 31.044 million metric tonnes grading 4.52 g/t gold for 4,514,562 ounces of gold, 46.3 g/t silver, 0.32% copper and 2.22% zinc.

LaRonde I (from surface to 2.36 km depth) proven mineral reserves are estimated to be 5.891 million metric tonnes grading 3.11 g/t gold for 589,738 ounces of gold, 90.8 g/t silver, 0.43% copper and 4.46% zinc, probable mineral reserves are estimated at 13.524 million metric tonnes grading 2.89 g/t gold for 1,256,852 ounces of gold, 80.3 g/t silver, 0.31% copper and 4.02% zinc. Indicated mineral resources are estimated to be 1.807 million metric tonnes grading 2.39 g/t gold for 138,761 ounces of gold, 33.5 g/t silver, 0.17% copper and 2.35% zinc.

LaRonde II (from 2.36 to 3.3 km depth) probable mineral reserves are estimated at 17.520 million metric tonnes grading 5.78 g/t gold for 3,257,709 ounces of gold, 20.1 g/t silver, 0.33% copper and 0.83% zinc. Indicated mineral resources are estimated to be 1.796 million metric tonnes grading 2.73 g/t gold for 157,871 ounces of gold, 22.0 g/t silver, 0.30% copper and 1.0% zinc. Finally, the inferred mineral resources are estimated to be 9.825 million metric tonnes grading 6.43 g/t gold for 2,031,770 ounces of gold, 27.6 g/t silver, 0.32% copper and 2.11% zinc.
4. Introduction and Terms of Reference

This document presents the 2005 mineral reserves and mineral resources estimate for the LaRonde Mine release by February 23rd 2005 by Agnico-Eagle Mines Ltd and covering the LaRonde, Elcoco and Terrex properties. This estimate was completed in compliance with National Instrument 43-101 and include information concerning geology, assaying, quality control, estimation method, mining operation, recoverability, market, contracts, environmental considerations, taxes, capital and operating cost estimates, economic analysis, pay back and mine life.

This estimate present inventory information including the ongoing mining operation and diamond drilling results up to December 31st 2004 for the LaRonde I portion of the deposit down to 2.36 km below surface. New information was considered up to January 31st 2005 for the LaRonde II portion of the deposit from 2.36 km to 3.3 km below surface based on the information collected in diamond drill holes initiated from the level 215 western exploration drift.

The estimate is presented in a database report format (file no. RapRes05-01.mdb) and is 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. LONG2005.dwg).

Total proven & probable reserves at LaRonde (total LaRonde I & II) are estimated to be 36.9 million tonnes grading 4.30 g/t gold, 53.4 g/t silver, 0.34% copper and 2.57% zinc and contains 5.104 million ounces of gold (Table 1). Indicated mineral resources stands at 3.6 million tonnes grading 2.56 g/t gold, 27.8 g/t silver, 0.24% copper and 1.68% zinc and contains 0.297 million ounces of gold whereas inferred minerals resources stand at 9.8 million tonnes grading 6.43 g/t gold, 27.6 g/t silver, 0.32% copper and 2.1% zinc and contains 2.032 million ounces of gold (Table 2).

The detail of the inventory for reserves and resources for the LaRonde I (above 2.36 km depth) and LaRonde II (under 2.36 km depth) is also provide in Table 3 to 6.

The 2005 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 Engineer, Geologist, chief geologist at the LaRonde Division, who is fully qualified per the standards outlined in the National Instrument 43-101. The results of the 2005 LaRonde mineral reserves and mineral resources estimate were released to the public in a press release dated February 23th, 2005.
## LaRonde

### Summary of Estimation Reserve

<table>
<thead>
<tr>
<th>Block Category</th>
<th>DILUTED GRADE</th>
<th>TOTAL PRODUCTION (DILUTED)</th>
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<tr>
<td></td>
<td>Au (g/t) Ag (g/t) Cu (%) Zn (%) Pb (%)</td>
<td>Tons (MT) Au (g) Ag (g) Cu (Kg) Zn (Kg) Pb (Kg)</td>
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<tr>
<td>Proven</td>
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<td>Probable</td>
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<td>Block Category</td>
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<td>--------------------</td>
</tr>
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## LaRonde I

### Summary of Estimation Reserve

**RapRes05-01**

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<th>Block Category</th>
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<th>Pb (%)</th>
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<th>Au (g)</th>
<th>Ag (g)</th>
<th>Cu (Kg)</th>
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<tr>
<td>Proven Total</td>
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**Table 3**
# LaRonde I

## Summary of Estimation Indicated Resource

Table 4

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# LaRonde II

## Summary of Estimation Reserve

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<th>Zn (%)</th>
<th>Pb (%)</th>
<th>Tons (MT)</th>
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<th>Ag (g)</th>
<th>Cu (Kg)</th>
<th>Zn (Kg)</th>
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<td>20.056</td>
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<td>0.83</td>
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<td>17,519,780</td>
<td>101,326,156</td>
<td>351,382,673</td>
<td>58,633,433</td>
<td>144,813,840</td>
<td>5103,659</td>
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<tr>
<td>Probable</td>
<td>5.784</td>
<td>20.056</td>
<td>0.33</td>
<td>0.83</td>
<td>0.03</td>
<td>17,519,780</td>
<td>101,326,156</td>
<td>351,382,673</td>
<td>58,633,433</td>
<td>144,813,840</td>
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### LaRonde II

#### Summary of Estimation Indicated Inferred Resource

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<th>Block Category</th>
<th>Au (g/t)</th>
<th>Ag (g/t)</th>
<th>Cu (%)</th>
<th>Zn (%)</th>
<th>Pb (%)</th>
<th>Tons (MT)</th>
<th>Au (g)</th>
<th>Ag (g)</th>
<th>Cu (Kg)</th>
<th>Zn (Kg)</th>
<th>Pb (Kg)</th>
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</thead>
<tbody>
<tr>
<td>Indicated Resource</td>
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<td>0.08</td>
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<td>4,910,330</td>
<td>39,444,326</td>
<td>5,442,751</td>
<td>17,910,663</td>
<td>1,445,889</td>
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<tr>
<td>Inferred Resource</td>
<td>6.432</td>
<td>27.555</td>
<td>0.32</td>
<td>2.11</td>
<td>0.06</td>
<td>9,825,462</td>
<td>63,195,145</td>
<td>270,742,622</td>
<td>31,454,996</td>
<td>207,174,938</td>
<td>6,207,909</td>
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</tbody>
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**Table 6**
RapRes05-01
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.
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 78°15’W, Longitude 48°15’N). The mine is approximately 600 km northwest of Montreal, Quebec (Figure1). 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 leases 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).

In 2003 Agnico-Eagle Mines Ltd acquired a 100% interest in the Terrex property to the south of the LaRonde property. The Terrex property is subject to the following royalties: a 5% net profits royalty (NPI) to Delfer Gold Mines Inc., a 1% of the net smelter returns royalty (NSR) to Breakwater Resources Ltd and a 2% of the net smelter returns (NSR) to...
Barrick Gold Corporation (refer to Orr, Burt, Allan & Boyd Asset Agreement Deal 2003). By the end of 2003, exploration works started to extend outside of the LaRonde property on to the Terrex property where the down plunge extension of the 20 North gold zone was discovered.

Figure 1
Location Map
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 (Figure 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 (Figure 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 nearby properties. The sufficiency of surface rights for mining and other availability issues are addressed in Scherkus (1986), Roscoe Postle (1999, 2001, 2002).
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 (3,537 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 exploitable by open pit method (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
Figure 3
LaRonde Longitudinal, Penna Shaft Zones

2005 Mineral Resources & Mineral Reserves Estimate

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<th>CATEGORY</th>
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<th>SILVER</th>
<th>COPPER</th>
<th>ZINC</th>
<th>TONNES (000's)</th>
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<td>LaRonde I</td>
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<tr>
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<td>19 415</td>
<td>1 847</td>
</tr>
<tr>
<td>Indicated Resource</td>
<td>2.39</td>
<td>33.51</td>
<td>0.17</td>
<td>2.35</td>
<td>1 807</td>
<td>139</td>
</tr>
</tbody>
</table>

| LaRonde II     |      |        |        |      |                |                |
| Probable Reserve | 5.78 | 20.06  | 0.33   | 0.83 | 17 520         | 3 258          |
| Total Mineral Reserves | 5.78 | 20.06  | 0.33   | 0.83 | 17 520         | 3 258          |
| Indicated Resource | 2.73 | 21.96  | 0.30   | 1.00 | 1 796          | 158            |
| Inferred Resource | 6.43 | 27.56  | 0.32   | 2.11 | 9 825          | 2 032          |
Gold (formerly zone 19), 20 North Zinc, 20 South and zone 6 (at depth) were found in 1992 and 1993 (Figure 3).

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 2,250 metres, the shaft changeover was completed and the 4,500 tonnes per day production hoist and ore handling facilities were commissioned. The LaRonde mill capacity was increased to 4,500 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 ounces of gold accountable production in June. In October Hoisting and ore handling facilities were expended to reach 6,300 tonnes per day at the Penna Shaft. The LaRonde mill capacity was also increased to 6,300 tonnes per day at the beginning of October.

Since 2002, exploration works took place on level 215 (2,150m depth) from an exploration drift located in the mafic rock approximately 400m to the north of the favourable rock sequence that host the massive sulfides lenses on the LaRonde property. The LaRonde II exploration campaign is following the deep extension of the Penna shaft lenses down to a depth of 3,300m.

The following tables describe the cumulative payable metal production at LaRonde, the division’s production history by shaft and the 2004 underground production per zone.
### Table 7: LaRonde Milled Production Summary to December 31st 2004

<table>
<thead>
<tr>
<th>Year</th>
<th>ORE MILLED (SHORT TONS)</th>
<th>GOLD GRADE (OZ/TON)</th>
<th>PRODUCTION PAID NET FROM SMELTER</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td>GOLD (OZ)</td>
</tr>
<tr>
<td>1988*</td>
<td>309 429</td>
<td>0.10</td>
<td>25 792</td>
</tr>
<tr>
<td>1989</td>
<td>693 825</td>
<td>0.14</td>
<td>84 974</td>
</tr>
<tr>
<td>1990</td>
<td>749 377</td>
<td>0.14</td>
<td>98 326</td>
</tr>
<tr>
<td>1991</td>
<td>652 390</td>
<td>0.20</td>
<td>115 831</td>
</tr>
<tr>
<td>1992</td>
<td>601 055</td>
<td>0.24</td>
<td>134 474</td>
</tr>
<tr>
<td>1993</td>
<td>638 523</td>
<td>0.26</td>
<td>152 355</td>
</tr>
<tr>
<td>1994</td>
<td>620 217</td>
<td>0.25</td>
<td>144 584</td>
</tr>
<tr>
<td>1995</td>
<td>728 064</td>
<td>0.25</td>
<td>167 209</td>
</tr>
<tr>
<td>1996</td>
<td>729 362</td>
<td>0.24</td>
<td>159 558</td>
</tr>
<tr>
<td>1997</td>
<td>785 552</td>
<td>0.21</td>
<td>154 515</td>
</tr>
<tr>
<td>1998</td>
<td>776 726</td>
<td>0.21</td>
<td>150 443</td>
</tr>
<tr>
<td>1999</td>
<td>798 402</td>
<td>0.12</td>
<td>90 035</td>
</tr>
<tr>
<td>2000</td>
<td>1 415 898</td>
<td>0.14</td>
<td>173 852</td>
</tr>
<tr>
<td>2001</td>
<td>1 805 248</td>
<td>0.15</td>
<td>234 860</td>
</tr>
<tr>
<td>2002</td>
<td>1 963 129</td>
<td>0.14</td>
<td>260 183</td>
</tr>
<tr>
<td>2003</td>
<td>2 448 603</td>
<td>0.11</td>
<td>236 653</td>
</tr>
<tr>
<td>2004</td>
<td>2 976 956</td>
<td>0.10</td>
<td>271 567</td>
</tr>
<tr>
<td>Total</td>
<td>18 692 756</td>
<td>0.16</td>
<td>2 655 211</td>
</tr>
</tbody>
</table>

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

### Table 8: LaRonde Cumulative Production Extracted to December 31st, 2004

<table>
<thead>
<tr>
<th>Description</th>
<th>TONS</th>
<th>AU (g/t)</th>
<th>AG (g/t)</th>
<th>CU (%)</th>
<th>ZN (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Penna Shaft</td>
<td>9 222 822</td>
<td>4.21</td>
<td>79.49</td>
<td>0.42</td>
<td>4.06</td>
</tr>
<tr>
<td>Open Pit #2</td>
<td>173 256</td>
<td>2.41</td>
<td>4.91</td>
<td>0.24</td>
<td>0.33</td>
</tr>
<tr>
<td>Shaft #1</td>
<td>6 539 500</td>
<td>6.82</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Shaft #2</td>
<td>959 516</td>
<td>7.34</td>
<td>40.67</td>
<td>0.73</td>
<td>2.54</td>
</tr>
<tr>
<td>Grand Total LaRonde</td>
<td>16 895 094</td>
<td>5.38</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

### Table 9: 2004 LaRonde Production by Zone

<table>
<thead>
<tr>
<th>Description</th>
<th>TONS</th>
<th>AU (g/t)</th>
<th>AG (g/t)</th>
<th>CU (%)</th>
<th>ZN (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total 20 South Zone</td>
<td>55 254</td>
<td>5.20</td>
<td>51.14</td>
<td>0.33</td>
<td>2.62</td>
</tr>
<tr>
<td>Total 20 North Zone</td>
<td>2 592 706</td>
<td>3.29</td>
<td>90.45</td>
<td>0.55</td>
<td>4.26</td>
</tr>
<tr>
<td>Total Zone 7</td>
<td>46 984</td>
<td>6.11</td>
<td>32.43</td>
<td>0.45</td>
<td>2.13</td>
</tr>
<tr>
<td>Total 2004 Penna Shaft Reconciled Production</td>
<td>2 694 944</td>
<td>3.38</td>
<td>88.63</td>
<td>0.54</td>
<td>4.19</td>
</tr>
</tbody>
</table>
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 (geological Group). The units are, from north to south: 1) 400m of the Kewagama Group which is made up of thick band of interbedded wacke; 2) 1,600m of the Blake River Group, a volcanic assemblage which hosts all the known economic mineralisation on the property; and 3) 600m of the Cadillac Group, made up of thick band of wacke interbedded with pelitic schist and minor iron formation (Figure 4).

At LaRonde, the Blake River Group is composed of the Hébécourt 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 Hébécourt 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 Dubé et al., 2003).

Dubé et al. (2002) have identified on the LaRonde property several regionally correlatable units (Lafrance et al., 2002) that comprise the Northern Tholeitic Basalt, the Lower Transitional and the 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 utilized by Dubé et al. (2002).

The Northern Tholeiitic Basalt member (Hébécourt 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 glomeroporphyritic and can be traced laterally for several kilometres. Although this unit hosts portions of the Mouska and former Mic-Mac gold mines, no significant mineralisation is associated with it on the LaRonde property.
Figure 4

Regional Geology

From Lafrance et al. 2002
The Northern Tholeiitic Basalt is interbedded with quartz-porphryritic rhyolite sillon and dyke units (Bousquet formation lower member) close to the southern contact. This portion of the tholeiitic basalt member intruded by rhyolitic sillon and dyke is up to 150 metres thick on the LaRonde mine property. These dyke & sill units 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 shear associated sulphide-rich low-grade 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 (Dubé 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 either undeformed or sheared over several metres, with local fault gouge (Figure 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
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 (Figure 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

In the 2005-01 Mineral Resources and Mineral Reserves Estimate no tonnage occurs at Shaft no.1. For detailed description of mineralized zones at shaft no1 please refer to 2001 Ore Reserve Report by Marc H. Legault.

11.2 DESCRIPTION OF THE MINERALIZED ZONES AT SHAFT no. 2

In the 2005-01 Mineral Resources and Mineral Reserves Estimate no tonnage occurs at Shaft no.2. For detailed description of mineralized zones at shaft no2 please refer to 2001 Ore Reserve Report by Marc H. Legault.

11.3 DESCRIPTION OF THE MINERALIZED ZONES AT THE PENNA SHAFT

Mineralisation at LaRonde at vicinity of the Penna Shaft occurs along five distincts mineralized horizon known as Zone 7, 6, 20 North Gold (Zone 19), 20 North Zinc (Zone 20) and 20 South (Zone 21). Although, the economically mineralized portion of those horizon are restricted to local sulfide rich lenses, the five mineralized horizons could be traced all over LaRonde I (surface to 2.36 km depth) and LaRonde II (below 2.36 km depth) portion of the deposit.

In the LaRonde I portion of the deposit, economic mineralisation occurs along the five different horizons, whereas at depth in the LaRonde II portion of the deposit, potentially economic mineralisation is known in zone 7, 20 North Gold & 20 North Zinc. A narrow gold-rich lens parallele and located to the north of the 20 North Gold Zone is also recognized and known as the 20 North Gold “North” Zone (Zone 19 North).
Zone 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 (100,000 tonnes), partially exposed by the 20-32 drift at shaft no. 1 (level 86 at Penna Shaft) (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 sulphide and chlorite alteration (Mailloux, 1998).

The zone no. 7 lens intersected in several drill holes near level 170 (block 73 about 415,000 tonnes) also consists of massive to semi-massive sulphide (1-5 metres thick) containing up to 80% pyrite, 10-15% sphalerite and 1-3% chalcopyrite with rare millimetre grains of visible gold. The zone is up to 120 metres in east-west length, it has been traced vertically for almost 350 metres from its apex near level 158 down to level 191 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 197 (Figure 6). The lens (block 74), representing roughly 425,000 tons of ore, is up to 4 metres in horizontal thickness, 150 metres in east-west length and has been traced down-plunge to just below level 221. 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.

In the LaRonde II portion of the deposit, the down plunge portion of the zone no. 7 horizon has been intersected in the exploration drill holes originating from the Penna shaft and level 215 exploration drift. The only potentially economic intersection is known as block RF-74. At depth, 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 6 at Penna shaft (blocks 62 and 63)**

In LaRonde I, 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 110,000 tonnes) is a narrow (3 metres) lens of stringer and semi-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).

A smaller lens (68,000 tonnes) of massive pyrite-sphalerite and interpreted to be zone no. 6 occurs immediately East and below the level 146 Penna shaft crosscut (block PB62).

**Zone 20 North Gold (blocks 191 to 199)**

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 (Figure 7 & 8). The zone is known from 980m depth on the LaRonde property and plunge toward the south-south-west at depth and cross progressively over on the Terrex Property at 2.8 km depth and was intersected down to 3.2 km below surface in the LaRonde II portion of the deposit.

At the Penna Shaft above level 155, the zone occurs within a weakly to moderately sheared, sericitized and silicified felsic lapilli tuff horizon. Thickness of the zone that occurs immediately south of a garnet-bearing dacite tuff horizon range from 3 to 10 meters. The mineralisation consists of finely disseminated to locally semi-massive pyrite (5-30%) and millimetre to centimetre-thick chalcopyrite (1-10%) veinlets transposed to slightly discordant with the main fabric. Rare disseminated millimetre-sized clots of visible gold could also be observed associated with the chalcopyrite or quartz veins.
Figure 7
Section 7440E Zone 20 North and 20 South
Penna Shaft Upper Mine

ZONE 20 SOUTH
ZONE 20 NORTH GOLD
ZONE 20 NORTH ZINC

AGNICO-EAGLE MINES LIMITÉE
DIVISION LARONDE
7440 E
Influence (+/-20m)
Definition Zone #20N et #20S
ORE LIMIT
FAULT

LEGEND
MASSIVE SULPHIDE
ACTINOLITE
FELDSPAR
SULPHIDE BLOCK
ALTERATION
SULPHIDE ZONE
GARNET

ZONE 20 SOUTH
ZONE 20 NORTH ZINC
ZONE 20 NORTH GOLD

LEGEND
MARINE SULPHIDE
SULPHIDE ZONE
ALTERATION
SULPHIDE BLOCK
GARNET
FELDSPAR
Figure 8
Section 7080E Zone 20 North and 20 South
Penna Shaft Lower Mine

ZONE 20 SOUTH
ZONE 20 NORTH ZINC
ZONE 20 NORTH GOLD

LEGEND

PENNA SHAFT CROSS SECTION
7080 E
LOOKING WEST (+/- 20m)
The sulphide-rich lapilli tuff horizon closely follows the fractured north contact with the 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.

The gold grade is generally directly related to the chalcopyrite content within the 20 North Gold zone.

Below level 155, towards level 236 the sulphide-rich lapilli tuff horizon continue to enclose gold and copper mineralisation but also progressively the overlying massive sulfide mineralisation. Usually, the gold grade within the massive sulfide will also be closely related to the chalcopyrite content. Around level 215, the massive sulphide portion of the 20 North lens will mostly be copper and gold rich, only the extreme eastern portion of the lens will have significant zinc content. Between level 215 towards level 236, the zinc concentration is decreasing all the mineralised phased (disseminated, semi-massive and massive sulphide ore) are only gold rich with variable amount of silver and copper.

In the LaRonde II portion of the deposit, the 20 North Gold zone continue to occurs within the sulphide-rich lapilli tuff horizon and the overlying massive sulfide lens. Progressively towards depth, the massive sulfide mineralisation is replaced by a silicified pyrite (10-20%)-chalcopyrite (tr-5%) stringer zone with aluminosilicate porphyroblasts (mostly kyanite and andalousite) that became progressively more abundant with increasing depth and that represent locally up to 30% of the mineralized zone (Figure 9).

Finally, at the bottom of the LaRonde II portion of the deposit 2.8 km below surface within the western extension of the zone, massive sulfide mineralisation was reintercepted in the latest drill holes (Figure 10). This massive sulfide mineralisation is not only gold and copper rich but also contain significant amount of zinc and silver. This polymetallic Au-Ag-Cu-Zn rich massive sulphide type of ore was also notice in LaRonde I between Level 215 and 170 within the eastern extension of the deposit.

Zone 20 North Zinc (blocks 201, 202, 203 and 204)

The 20 North Zinc zone is a massive sulphide zinc-silver zone usually in contact with (to the south of) the 20 North Gold zone and is entirely located on the LaRonde Property. The zone has been traced, in the central portion of the property, 100 to 150 metres north of the Cadillac sedimentary Group contact (Figure 7 & 8). 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
Figure 9
Section 6840E Zone 20 North
LaRonde II

LaRonde II CROSS SECTION
6840 E
LOOKING WEST (+/- 60m)
Figure 10
Section 6360E Zone 20 North
LaRonde II
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 20 North Zinc massive sulphide lens. Below level 155, the 20 North Zinc zone migrates toward the south-east contact of the massive sulphide lens. Below level 215 the Zinc mineralisation is less continuous but continue to occurs as isolated lenses down to 2.5 km below surface.

Zone 20 South Gold (blocks 210, 211, 212, 213 & 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 (Figure 7 & 8). 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.

The lenses generally occur as 100 to 300 metre long (east-west) by 300 to 500 metre along plunge and 3 to 15 metre thick (north-south) pods of ore within the more extensively developed sulphide-rich sheared horizon. The sheared horizon, 5-15 metres in thickness, consists of disseminated pyrite, pyrrhotite and minor sphalerite within 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 in between altered andesite units of 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), centred 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 upper lens (block 211 LaRonde & block 210, 215 El Coco) was largely depleted by the end of 2004.

The lowermost lens (block 212 and 213), centred near level 194 at section 7200E, is irregularly shaped, generally less than 5 metres in thickness, and mostly disseminated in character. No mining occurs yet in that second lens by the end of 2004.

Below level 215, some isolated values were intercepted along the 20 south horizon, but to date no continuity was confirm surrounding these isolated values.
12. Exploration

In 2004, exploration work on the LaRonde and Terrex properties consisted solely of underground diamond drilling.

12.1 2004 Drilling Results

A summary of diamond drilling completed in 2004, broken down by project, is presented in Table 10. The list of holes completed in 2004, with all zones mid-point intercepts is presented in Appendix B. Diamond drilling results are plotted on the individual zone longitudinals attached to the report.

12.2 2005 Drilling Program

In 2005, budget calls for 9,715 metres of delineation diamond drilling, 18,740 metres of definition drilling and 24,475 metres of exploration drilling. Delineation drilling (9,715 metres) will mostly consist of flat diamond drill holes (15m x 30m) generally at the corners of every mining block. Definition drilling (18,740 metres, spacing 40m x 40m) is going to continue: 1) in the eastern extension of the 20 North zone in between level 215 and 160, (bloc PB193) 2) in the 20 South zone (bloc PB212 and 213) and 3) in zone 20 north between level 236 and 215. Deep exploration drilling (project 10; 24,475 metres) from the level 215-exploration drift will be focused on the deep resources to reserves conversion program and exploration toward the Bousquet property.

Deep exploration of the 20 North and 20 South zones, from the level 215 exploration drift, consists of continuing a pattern of north-south drill intercepts on an average 120 metre (horizontally east-west) by 100 metres (vertically) intervals, in the area of more widely spaced azimuth exploration drill holes completed in the past years either from the Penna shaft station or the lower mine decline.

12.3 Drilling Contractor

Forages Garant of Évain, Québec, does all diamond drilling on the property under a contract with Agnico-Eagle Mines Ltd up to June 2005.
Table 10: Summary of Diamond Drilling 2004

<table>
<thead>
<tr>
<th>Project Number</th>
<th>Drill Hole Category</th>
<th>Number of Holes</th>
<th>Metres drilled in 2004</th>
</tr>
</thead>
<tbody>
<tr>
<td>351</td>
<td>Delineation</td>
<td>180</td>
<td>10 936.4</td>
</tr>
<tr>
<td>350</td>
<td>Definition</td>
<td>54</td>
<td>7 575.1</td>
</tr>
<tr>
<td>10</td>
<td>LaRonde II Exploration</td>
<td>41</td>
<td>30 099.0</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td></td>
<td><strong>275</strong></td>
<td><strong>48 610.5</strong></td>
</tr>
</tbody>
</table>

(See detailed Appendix B)

12.4 Procedure for describing drill core

All the drillcore recovered at LaRonde are described by graduate geologists (or geological engineers) whom are either members of the OIQ (Quebec Order of Engineers) or OGQ (Quebec Order of Geologists) or of both of them. In the past, some drill core was logged by a non-graduate geologist who was supervised and reviewed by a LaRonde mine geologist. Each hole has a drill log associated with it. Current drill core descriptions are interred 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 (3.64 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 box 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 15 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. The EZ-shot system, which is a digital magnetic-type compass system, electronically measures the azimuth, plunge, magnetic field and temperature. This system has 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 to 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 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 8) 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;
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 2005) 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, that could be related to the non-continuous nature of 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 2003, 2004 and 2005).

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 composites is presented in Appendix B and on longitudinal sections. For the sake of brevity, there 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 laboratory 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 2004, 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 Edgdouglul, chief chemist). The principal assay laboratory for chip samples is the LaRonde assay laboratory (exceptionally they can be sent to an independent laboratory). A program supervised by CANMET (Proficiency Testing Program for Mineral Analysis Laboratories) allow those laboratories to have access to standard samples control procedures and certification. ALS Chemex and Bourlamaque and Technilab laboratories certificate for 2003 are available in Appendix B. The 2004 certificates will be delivered in April 2005.

15.3 LaRonde assay laboratory procedures

Sample preparation
1. The samples are dried at 180° 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 ¼ inch 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:

<table>
<thead>
<tr>
<th>Element</th>
<th>Extraction</th>
<th>Method</th>
</tr>
</thead>
<tbody>
<tr>
<td>Au</td>
<td>Fire-assay (up to 1 A.T.)</td>
<td>Fire-assay, gravimetric finish</td>
</tr>
<tr>
<td>Ag</td>
<td>HF-HCL-HN03</td>
<td>Atomic absorption</td>
</tr>
<tr>
<td>Cu</td>
<td>HF-HCL-HN03</td>
<td>Atomic absorption</td>
</tr>
<tr>
<td>Zn</td>
<td>HF-HCL-HN03</td>
<td>Atomic absorption</td>
</tr>
<tr>
<td>Pb</td>
<td>HF-HCL-HN03</td>
<td>Atomic absorption</td>
</tr>
</tbody>
</table>

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 received 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

**Delineation drill hole control samples**
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. 20North, 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 results from delineation drill holes are studied at the end of every quarter (3 months period) and the year-end report is presented in Appendix B.

In 2004, all the standard samples that came back with irregular values compared to the average results were duplicated 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 usually in the range of the average only a few sample were duplicated for base metal. Erratic distribution of the gold even in the homogenized standard samples seems to be the reason of occasional higher-grade values in standard samples.

**Chip sample control samples**
In 2004, a new procedure was put in place in order to make a quality control on chip samples. Commercial certified homogenized standards were acquired in order to be introduced with each chip sample batch (usually 15 to 20 chip samples per batch). The gold rich certified standards coming from sulphide ore (see details in Appendix B) is introduce at a rate of 1 standard for each 15 to 20 chip samples.

In 2004, no major discrepancy was observe compared to the certified standards average (see graph in appendix B). In 2005 a protocol to make a follow up on samples that differ from the average grade certified will be put in place.

**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 1 g/t Au, 7 g/t Ag, 0.30% Cu, 0.50% Zn and 0.5% Pb as 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 author’s 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 be 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 appendix B), excluding delineation drill holes intercepts, 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 2004 was completely reviewed by LaRonde geologist team (actual and former members) before the 2004 mineral reserve report. The supervising geologists including the author, prior to the estimate of the 2005 LaRonde mineral reserves and resources estimate verified the 2004 sample data.

The results of the 2005 mineral reserve and resource estimate was compared to those of the 2004 estimate. No anomalies were discovered by the author (refer to 2004-2005 comparison Appendix B).

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 and exploration activities at the LaRonde Mine are located on the LaRonde, El Coco and Terrex properties as reported in the section Property description and location section 6. Other properties owned 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 section 4.

18. Mineral Processing and Metallurgical Testing
The LaRonde Division’s milling complex has been in operation since 1988 and currently processes at a minimum 7,000 tons per day capacity. Results of mineral processing and metallurgical testing analyses are presented in section 25.1.2. Additional details are presented in Girard et. al. (1999, 2001) and reviewed by Roscoe Postle Associates (1999, 2001, 2002).

19.1 Introduction


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 parameters 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.

In 2003, the pre-feasibility study by Émond 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) east of section 6600E into probable reserves with the new diamond drilling information in 2002.

In the 2004 Mineral Resource and Mineral Reserves Estimate, resource were converted to reserves down to 3,000m below surface considering the progress of the ongoing LaRonde II exploration program and the LaRonde Deep Mine 2003 pre feasibility Study by Emond et al. (Appendix B).
In the 2005 Estimate, conversion of resources to reserves continue above 3,000m depth towards the west as the exploration-definition drilling continue as long as a detail feasibility study directed by Carol Plummer Eng. manager of the LaRonde II project that will be available by mid-2005.

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. According to the statistical analysis: zone 7 below level 220, the western portion of the 20 North zone and all the resources located below elevation 2000m 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 40$C to 55$C / tm depending on the level and the zone above level 236. For LaRonde II below level 236 60$C/tm was consider as the lower cut off NSR values. A small portion of the 20 North Gold zone located in the western extension of the probable reserves barely respect that economic criteria ranging between 50 to 60$ 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 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 would be of that nature but are not included in the 2005 Estimate since they are 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 6, 7, 20 North and 20 South) are sufficient to quantify an inventory of economic massive to disseminated gold-copper or zinc silver mineralisation as probable mineral reserve (see Table 11). A grid spacing in the order of 100 metres vertical by 120 metres horizontal is adequate for massive to disseminated gold-copper mineralisation (Zone 20 North Gold). A grid spacing in the order of 80 metres vertical by 50 metres horizontal is requested for Zone 20 North zinc whereas a grid of 80 metres vertical by 60 metres horizontal is adequate for Zone 20 South. Finally smaller lenses such as zone 6, 7 & 22 will need a grid spacing of 60 metres vertical by 50 metres horizontal.

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 11. Drill hole spacing for Indicated Resources and Probable Reserves

<table>
<thead>
<tr>
<th>Zone</th>
<th>Vertical spacing</th>
<th>Horizontal spacing</th>
</tr>
</thead>
<tbody>
<tr>
<td>20 North Gold</td>
<td>100m</td>
<td>120m</td>
</tr>
<tr>
<td>20 North Zinc</td>
<td>80m</td>
<td>50m</td>
</tr>
<tr>
<td>20 South</td>
<td>80m</td>
<td>60m</td>
</tr>
<tr>
<td>7, 6 and 22</td>
<td>60m</td>
<td>50m</td>
</tr>
</tbody>
</table>

19.4 Changes in the estimate since February 2004

Mining reserves and mineral resources at LaRonde since February 25th 2004 have been affected both by new diamond drilling results, reconciliation data and the ongoing LaRonde II feasibility study.

A) Steady production rate at above 8000 short tons per day in 2004 continue to keep production cost low at 53 C$/tonne. NSR cut off used for the estimate range from 50 to 55 C$/tonne for the gold and the gold-zinc rich portion of the deposit above the bottom of the Penna shaft.

In zinc only portion of the deposit NSR cut off was kept at 37$ C$/tonne according to the long term mine plan that forecast lower mining cost at the end of the mine life. Incremental gold portion (above 40 C$/metric ton) of the deposit located in the immediate footwall of an economic zinc zone (above 50 C$/metric ton) was also considered in the 2005 Estimate.

B) Economic evaluation of the NSR cut off for tonnage located below level 215 following economic study by Provencher 2002, Émond 2003 and Émond et al 2003 was considered as following: NSR cut off progressively increasing from 50C$/metric tons up to 55C$/metric ton from level 215 down to level 236 and 60C$ from level 239 to 3,000 metres depth. The latest report by Émond et al, 2003 allowed new indicated resources to be converted into a probable mineral reserve down to 3,000m metres.
C) Definition and exploration drilling continued during 2004 from exploration drifts on levels 170, 194 and 215. The 2004 exploration and definition drilling results at the Penna shaft are summarised below:

1. Definition drilling of the western edge of zone 20 North (and 20 South) from the level 194 exploration drift was completed. No major change in the zone outline was observed.

2. Exploration drilling of zone 20 North Gold from level 215 below the bottom of the Penna Shaft continue in 2004. The ongoing exploration program for the LaRonde II project with systematic drilling at about 100m vertical x 120 m horizontal allowed to transfer a portion of mineral resource located above 3,000 metres depth into probable reserves on LaRonde and Terrex properties.

Among the latest drill holes in the LaRonde II project, some of them returned very significant result as listed below:

<table>
<thead>
<tr>
<th>Drill Hole</th>
<th>True Thickness (m)</th>
<th>Au g/t</th>
<th>Ag G/t</th>
<th>Cu %</th>
<th>Zn %</th>
<th>Pb %</th>
<th>X</th>
<th>Y</th>
<th>Z</th>
</tr>
</thead>
<tbody>
<tr>
<td>3215-83</td>
<td>12.2</td>
<td>6.0</td>
<td>4.4</td>
<td>0.15</td>
<td>0.02</td>
<td>0.01</td>
<td>6 594</td>
<td>2 561</td>
<td>2 060</td>
</tr>
<tr>
<td>3215-84</td>
<td>16.0</td>
<td>6.0</td>
<td>15.0</td>
<td>0.25</td>
<td>0.01</td>
<td>0.01</td>
<td>6 761</td>
<td>2 660</td>
<td>2 339</td>
</tr>
<tr>
<td>3215-85D</td>
<td>3.7</td>
<td>8.8</td>
<td>68.0</td>
<td>0.37</td>
<td>9.98</td>
<td>0.01</td>
<td>5 992</td>
<td>2 522</td>
<td>2 017</td>
</tr>
<tr>
<td>3215-89</td>
<td>17.1</td>
<td>5.3</td>
<td>6.2</td>
<td>0.13</td>
<td>0.02</td>
<td>0.01</td>
<td>6 638</td>
<td>2 652</td>
<td>2 317</td>
</tr>
<tr>
<td>3215-95</td>
<td>12.4</td>
<td>7.4</td>
<td>54.8</td>
<td>0.52</td>
<td>6.26</td>
<td>0.06</td>
<td>6 329</td>
<td>2 568</td>
<td>2 159</td>
</tr>
<tr>
<td>3215-99</td>
<td>14.0</td>
<td>7.0</td>
<td>76.0</td>
<td>0.42</td>
<td>6.42</td>
<td>0.26</td>
<td>6 120</td>
<td>2 521</td>
<td>2 012</td>
</tr>
<tr>
<td>3215-99A</td>
<td>5.7</td>
<td>2.1</td>
<td>21.7</td>
<td>0.13</td>
<td>17.12</td>
<td>0.03</td>
<td>6 210</td>
<td>2 574</td>
<td>2 189</td>
</tr>
<tr>
<td>3215-102</td>
<td>2.9</td>
<td>8.3</td>
<td>83.2</td>
<td>0.29</td>
<td>5.72</td>
<td>0.30</td>
<td>6 121</td>
<td>2 553</td>
<td>2 148</td>
</tr>
<tr>
<td>3215-103A</td>
<td>15.1</td>
<td>7.0</td>
<td>6.2</td>
<td>0.12</td>
<td>0.04</td>
<td>0.01</td>
<td>6 578</td>
<td>2 545</td>
<td>1 991</td>
</tr>
<tr>
<td>3215-105A</td>
<td>7.4</td>
<td>9.3</td>
<td>13.6</td>
<td>0.54</td>
<td>0.06</td>
<td>0.00</td>
<td>6 200</td>
<td>2 496</td>
<td>1 855</td>
</tr>
</tbody>
</table>

D) The 2005 reconciliation study for the 2004 production (Gosselin 2005 in prep) continue to indicated that the model used in the estimation process is adequate with gold grade 2% higher at the mill compared to the model and zinc grade 3.2% higher at the mill whereas tonnage was slightly lower by 0.8%. Significant discrepancy was observed in the silver and copper estimate as detailed below.
Table 12. 2004 production reconciled highlights

<table>
<thead>
<tr>
<th>2004 Production LaRonde</th>
<th>Production reconciled</th>
<th>Calculated Production with the model Estimate</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>AU (g/t)</td>
<td>AG (g/t)</td>
</tr>
<tr>
<td>Stope &amp; development</td>
<td>3.39</td>
<td>89.3</td>
</tr>
<tr>
<td>Variance</td>
<td>2.0%</td>
<td>10.6%</td>
</tr>
</tbody>
</table>

Calibration factor was kept at 1.05 for the tonnage, gold calibration factors for remobilization along north-south fractures were also kept unchanged. Since silver and copper grades seems to be underestimate calibration factors of 1.05 and 1.10 respectively were introduced in the 2005 estimated. This is still conservative considering the 10.6% and 15.5% upgrade observed.
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 2005, the mineral reserves for only one lens: zone 6 (block 62) was estimated using the polygonal method (as in the 2003 and 2004 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 beginning of commercial operations (see Blackburn, 1991). It is a simple and quick method that was adequate because the ore zones 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 grade and the specific gravity of the ore 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 whereas in areas of high density of information, inverse power distance block modelling method is used for indicated resources and proven and probable reserves.

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).

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 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\textsuperscript{power} from a particular estimation point (the unit-sized block). For example:

\[
\text{Inverse distance squared grade estimate} = \text{weight each sample’s grade inversely to distance}^2 \\
\text{and} \\
\text{divide by sum of inverse distance}^2
\]

The use of different power levels according the search distance is a function of the information density, and of the zone.

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, 2000a, 2000b, 2001, 2003 & 2004) and also 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), cumulative probability plots, or total metal content. Gold and Silver values used are variably topcut depending on the zone (Table 13). The topcut grades used in polygonal estimation is the same than the one used in the block model.

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 at Table 14 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 Table 14 (the search and estimation parameter files and a DATAMINE reference list can be consulted in Appendix B).

Table 13: Top cut grades for drill holes and channel sample assays

<table>
<thead>
<tr>
<th>Zone</th>
<th>Au (g/t)</th>
<th>Ag (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>6</td>
<td>51.4286</td>
<td>857.143</td>
</tr>
<tr>
<td>7</td>
<td>68.5714</td>
<td>342.857</td>
</tr>
<tr>
<td>20 North Gold</td>
<td>51.4286</td>
<td>857.143</td>
</tr>
<tr>
<td>20 North Zinc</td>
<td>17.1429</td>
<td>857.143</td>
</tr>
<tr>
<td>20 South</td>
<td>68.5714</td>
<td></td>
</tr>
</tbody>
</table>
### Table 14: Zone Anisotropies

<table>
<thead>
<tr>
<th>Zone 20 North Gold</th>
<th>Block 191-192</th>
<th>sigma 1</th>
<th>Range</th>
<th>Azimuth</th>
<th>Plunge</th>
</tr>
</thead>
<tbody>
<tr>
<td>Au</td>
<td>112</td>
<td>267</td>
<td>-22</td>
<td>67</td>
<td>114</td>
</tr>
<tr>
<td>Ag</td>
<td>115</td>
<td>267</td>
<td>-22</td>
<td>100</td>
<td>114</td>
</tr>
<tr>
<td>Cu</td>
<td>125</td>
<td>267</td>
<td>-22</td>
<td>60</td>
<td>114</td>
</tr>
<tr>
<td>Zn</td>
<td>90</td>
<td>266</td>
<td>-42</td>
<td>80</td>
<td>100</td>
</tr>
<tr>
<td>Pb</td>
<td>75</td>
<td>266</td>
<td>-42</td>
<td>45</td>
<td>100</td>
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<tr>
<td>SG</td>
<td>105</td>
<td>131</td>
<td>-79</td>
<td>55</td>
<td>271</td>
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<tr>
<td>Au</td>
<td>112</td>
<td>267</td>
<td>-22</td>
<td>67</td>
<td>114</td>
</tr>
<tr>
<td>Ag</td>
<td>115</td>
<td>267</td>
<td>-22</td>
<td>100</td>
<td>114</td>
</tr>
<tr>
<td>Cu</td>
<td>125</td>
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<td>114</td>
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<tr>
<td>Zn</td>
<td>90</td>
<td>266</td>
<td>-42</td>
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<td>100</td>
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<tr>
<td>Pb</td>
<td>75</td>
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<td>SG</td>
<td>105</td>
<td>131</td>
<td>-79</td>
<td>55</td>
<td>271</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Zone 20 North Zinc</th>
<th>Block 201-202</th>
<th>sigma 1</th>
<th>Range</th>
<th>Azimuth</th>
<th>Plunge</th>
</tr>
</thead>
<tbody>
<tr>
<td>Au</td>
<td>90</td>
<td>89</td>
<td>-18</td>
<td>80</td>
<td>256</td>
</tr>
<tr>
<td>Ag</td>
<td>90</td>
<td>106</td>
<td>-86</td>
<td>80</td>
<td>263</td>
</tr>
<tr>
<td>Cu</td>
<td>140</td>
<td>246</td>
<td>-69</td>
<td>100</td>
<td>91</td>
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<tr>
<td>Zn</td>
<td>95</td>
<td>263</td>
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<td>60</td>
<td>98</td>
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<tr>
<td>Pb</td>
<td>75</td>
<td>202</td>
<td>-86</td>
<td>70</td>
<td>85</td>
</tr>
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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 15 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$^1$ 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 progressively replaced over the years.

The 20 South zone was revised to 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).

In 2004 all the other zones at the mine estimate by IPD interpolation method were also revised. Up to 2003 inverse distance$^3$ and inverse distance$^1$ was used in zone 20 North and zone 7 since local variability was high and regional variability was low. With the new approach proposed by D’Amours 2003 and 2004, variogram study of pairs of composite samples was restricted to the economic envelopes of the different zones. The amount of information (drill hole intercepts) available in every zones also increased over the years. New variography studies suggested that now a IPD interpolation method using distance$^2$ was also more appropriate in zone 7 and 20 North.

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 3300 metres (1700-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. In 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 SG values are composited along with the other metals. The SG of a model block is assigned to it using the IPD2 interpolation method as all other metals.

7. In 2002 and 2003, 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 reconciliation study). Following the 2004 Reconciliation study, the tonnage calibration factor was reduced to 1.05.

8. In 2005, a tonnage calibration factor of 1.05 has been applied to all reserves and resources.

9. A dilution study (CMS survey analysis) of Penna shaft stopes allows the determination of the equivalent meter factor that represents the average dilution (meters of overbreak) 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 reserve block 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 both sides 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 PB62 and indicated resource blocks RD63 and RD22. No dilution factor was applied to inferred mineral resource (see Appendix A).

12. Dilution grade for reserve block 62 was estimated using drill holes representative assay data in the wall rock (see Appendix A). Resource block RD63 and RD22 was estimated using same dilution grade than reserve block 62 in reason of apparent similar geological context and properties.

13. The economic viability of each intercept was tested by using a complex Excel logarithm (NSR2005.xls see Appendix B) developed by the LaRonde metal marketing department since January 1997 and modified throughout the years according to metal prices fluctuation and new smelting contract. For the 2005 estimate, according to the SEC (Securities and Exchange Commission) guideline using the following metal prices assumptions: 360$US/oz gold, 5.42$US/oz silver, 0.95$/lb copper, 0.41$/lb zinc and a 1 $US/ 1.418 $C exchange rate.

14. Using DATAMINE generated NSR values longitudinal sections, an NSR cut-off of 50$C/tonne to 55$C/tonne for gold rich material in LaRonde I to 60$C/tonne for LaRonde II 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$/tonne (see Appendix B). The bulk NSR must be greater than 50$/tonne.

16. The minimum cut-off for areas of 20 North Zinc which mining is planned later than 2010 has been set at 40$/tonne.

19.6 Results
# LaRonde I

## Summary by Category and Zone

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<thead>
<tr>
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<th>Zone</th>
<th>Au (g/t)</th>
<th>Ag (g/t)</th>
<th>Cu (%)</th>
<th>Zn (%)</th>
<th>Tons (Met.)</th>
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<th>Ag (g)</th>
<th>Cu (Kg)</th>
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## LaRonde I

### Summary by Category and Zone

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<td>2.891</td>
<td>80.335</td>
<td>0.31</td>
<td>4.02</td>
<td>0.46</td>
<td>13 523 841</td>
<td>39 092 508</td>
<td>1 086 431 510</td>
<td>41 596 772</td>
<td>543 414 186</td>
<td>62 669 117</td>
</tr>
</tbody>
</table>
# LaRonde I

## Summary by Category and Zone

<table>
<thead>
<tr>
<th>Category</th>
<th>Zone</th>
<th>DILUTED GRADE</th>
<th>TOTAL PRODUCTION (DILUTED)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Au (g/t)</td>
<td>Ag (g/t)</td>
</tr>
<tr>
<td>Indicated</td>
<td>20N AU</td>
<td>2.659</td>
<td>30.288</td>
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<td></td>
<td>20N ZN</td>
<td>0.779</td>
<td>48.887</td>
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<td></td>
<td>20S</td>
<td>3.410</td>
<td>28.291</td>
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<tr>
<td></td>
<td>El Coco</td>
<td>3.555</td>
<td>18.356</td>
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<td></td>
<td>6</td>
<td>3.338</td>
<td>25.586</td>
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<td>7</td>
<td>3.172</td>
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<tr>
<td>Indicated</td>
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<td>2.388</td>
<td>33.512</td>
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RapRes05-01
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<tr>
<th>Category</th>
<th>Zone</th>
<th>Au (g/t)</th>
<th>Ag (g/t)</th>
<th>Cu (%)</th>
<th>Zn (%)</th>
<th>Pb (%)</th>
<th>Tons (Met.)</th>
<th>Au (g)</th>
<th>Ag (g)</th>
<th>Cu (Kg)</th>
<th>Zn (Kg)</th>
<th>Pb (Kg)</th>
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</thead>
<tbody>
<tr>
<td>Probable</td>
<td>20N AU</td>
<td>5.784</td>
<td>20.056</td>
<td>0.33</td>
<td>0.83</td>
<td>0.03</td>
<td>17 519 780</td>
<td>101 326 156</td>
<td>351 382 673</td>
<td>58 633 433</td>
<td>144 813 840</td>
<td>5 103 659</td>
</tr>
<tr>
<td>Probable</td>
<td>Total</td>
<td>5.784</td>
<td>20.056</td>
<td>0.33</td>
<td>0.83</td>
<td>0.03</td>
<td>17 519 780</td>
<td>101 326 156</td>
<td>351 382 673</td>
<td>58 633 433</td>
<td>144 813 840</td>
<td>5 103 659</td>
</tr>
</tbody>
</table>
## LaRonde II

### Summary by Category | Zone

<table>
<thead>
<tr>
<th>Category</th>
<th>Zone</th>
<th>DILUTED GRADE</th>
<th>TOTAL PRODUCTION (DILUTED)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Au (g/t)</td>
<td>Ag (g/t)</td>
</tr>
<tr>
<td>Indicated Resource</td>
<td>20N AU</td>
<td>Somme</td>
<td>3.067</td>
</tr>
<tr>
<td>Indicated Resource</td>
<td>20N ZN</td>
<td>Somme</td>
<td>1.709</td>
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<tr>
<td>Indicated Resource</td>
<td>Total</td>
<td>Somme</td>
<td>2.734</td>
</tr>
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</table>
# LaRonde II

## Summary by Category Zone

<table>
<thead>
<tr>
<th>Category</th>
<th>Zone</th>
<th>Au (g/t)</th>
<th>Ag (g/t)</th>
<th>Cu (%)</th>
<th>Zn (%)</th>
<th>Tons (Met.)</th>
<th>Au (g)</th>
<th>Ag (g)</th>
<th>Cu (Kg)</th>
<th>Zn (Kg)</th>
<th>Pb (Kg)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Inferred Resource</td>
<td>20N AU</td>
<td>6.610</td>
<td>25.035</td>
<td>0.30</td>
<td>2.15</td>
<td>9 125 940</td>
<td>60 325 007</td>
<td>228 469 123</td>
<td>27 684 574</td>
<td>195 891 652</td>
<td>5 431 440</td>
</tr>
<tr>
<td>Inferred Resource</td>
<td>7</td>
<td>4.103</td>
<td>60.432</td>
<td>0.54</td>
<td>1.61</td>
<td>699 522</td>
<td>2 870 138</td>
<td>42 273 499</td>
<td>3 770 422</td>
<td>11 283 286</td>
<td>776 469</td>
</tr>
<tr>
<td>Inferred Resource</td>
<td>Total</td>
<td>6.432</td>
<td>27.555</td>
<td>0.32</td>
<td>2.11</td>
<td>9 825 462</td>
<td>63 195 145</td>
<td>270 742 622</td>
<td>31 454 996</td>
<td>207 174 938</td>
<td>6 207 909</td>
</tr>
</tbody>
</table>

**Note:** The table above provides a summary of the metal content and production for different categories and zones at the LaRonde II site. The values represent averages and totals for various metals such as gold (Au), silver (Ag), copper (Cu), zinc (Zn), and lead (Pb).
19.7 Exceptions for the year 2005 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 2004.

The grade of broken ore stored in the stopes, orepasses, surface stockpiles and mill silos on December 31st 2004 (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 2004 inventory estimated by the engineering department.

Probable reserves and indicated and inferred mineral resources were estimated using the most up to date diamond drill hole and chip sample information restricted to December 31st 2004 For LaRonde I. For LaRonde II, results up to February 2005 were included.

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

1. Some exploration holes below level 215 have incomplete collar location data (3215-99 and 99A) or down hole surveys. Final collar surveys are generally done when drilling planned on a particular set up is completed. For the 2005 estimate, a preliminary survey was completed to locate the drill holes on ongoing drilling section or planned collar location were used for delineation or definition drill holes. The likely changes will not be significant;

2. Check assays were pending at the time of the estimate for drill hole 3215-103A but a standard sample was used to insure a quality control and the result of the check assays receive latter confirm the first assays.

3. Reserves blocks PB62 was evaluated using estimated SG whereas all the other reserves were evaluated by wireframe block model method.

4. Dilution grade for reserve block PB62 and indicated resource blocks RD22 & 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.

5. Indicated resources blocks RD22 and RD63 were not re evaluated in 2005. They were kept like in 2003 using IPD 3 and IPD 1 interpolation method.

6. 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).

7. The octant method was not used in the estimation of the lower Penna shaft zones (193, 194, 195, 196, 203, 204, 205 and 212).
20. Other Relevant Data and Information

No additional information or explanation is necessary within the terms of reference of this technical report. Additional details are presented in Girard et. Al. (1999, 2001) and reviewed by Roscoe Postle Associates (1999, 2001, 2002) and section 25.
21. Interpretation and Conclusions

The following conclusions can be made:

1. In 2004, the Penna shaft operation attained production levels of 7,399 metric tons per day with operating cost of C$52.84 per metric ton that was higher than budgeted (C$49.85 per metric ton). 2005 Budget operating cost per metric ton is expected to be C$52.39.

2. Block model inverse power distance estimation appears to be a suitable method for mineral reserve estimations at the LaRonde mine.

3. Proven and probable reserve in the 20 South zone block 211/215 are almost depleted by the end of 2004.

4. Definition diamond drilling completed in 2004 in zone 20 North Gold toward the west in between levels 164 & 215 was completed without major changes compared to last year estimate.

5. The 2005 Reconciliation Study (Gosselin 2005) suggests that the 2004 model was appropriate for tonnage as long as gold and zinc grade estimate when compared to the mill production.

6. New calibration factors were introduced in 2005 for silver grade (1.05) and copper grade (1.10) in the 2005 Mineral Reserve and Mineral Resource estimate based on the 2004 and 2005 reconciliation studies.

7. The increasing amount of drill holes intercepts below the bottom of the Penna Shaft following a nearly north-south orientation and a grid spacing of 120 m horizontal and a vertical spacing of 100m down to 3,000m below surface allow new resources conversion towards the west.

8. Occurrence of a new high grade gold and base metal lens in the 20 North gold zone to the west at depth such as in drill hole 3215-99 that intersected 14.0 meters true thickness at 7.0 g/t of gold 76 g/t of silver 0.42% Cu and 6.42% Zn could have a significant impact on the LaRonde II study.
22. Recommendations
The following general recommendations can be made:
1. Conducted geostatistical studies for copper and silver grade that are underestimate with the model when compared to the mill.
2. Completed the wireframing, use specific dilution and density data and complete block modelling techniques estimation for reserve blocks 62 and indicated resource blocks 63 and 22. This would complete all the lenses in the proven-probable and indicated resource category;
3. Continue to review the current block models parameters periodically using grade and tonnage reconciliation data;
4. Complete the definition diamond drilling between levels 215 and 155 toward the eastern limit of the deposit;
5. Continue the definition drilling program in zone 20 North (block 194) between levels 218 and 236 from level 218 exploration drift;
6. Continue the exploration program for LaRondeII using on section North-South diamond drill holes from the 215 level exploration drift.

Respectfully submitted,
Guy Gosselin, Engineer, Geologist
Chief Geologist, Author
Agnico-Eagle Mines Ltd., LaRonde Division
March, 23th 2005
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24. Date
This report is effective by February 23\textsuperscript{th} 2004. It was completed by March 23\textsuperscript{th} 2005.
25. Additional Requirements for Technical Reports on Production Properties

This section addresses additional requirements for technical reports on development properties and production properties. It is based on a study prepared by Agnico-Eagle Mines Ltd. Technical staff (Girard et al., 2001) titled “7,000 TPD Expansion Study, Eight-year mine plan 2001-2008” and dated June 9th, 2001.

25.1 MINING OPERATIONS

25.1.1 Mining Methods

The LaRonde Mine was originally developed utilizing a 3,961 foot deep shaft (Shaft #1) and an underground ramp access system. The ramp access system is available to move materials and equipment by vehicle from surface down to the 25th Level of Shaft #1 where it connects to the Penna Shaft ramp system on Level 122. The mineral reserve accessible from Shaft #1 was depleted in September 2000. Shaft #1 is currently being used as a second escape way and provides services for the Penna Shaft (i.e. ventilation, compressed air, water). Although Shaft #1 was decommissioned in 2004, the ramp access system will continue to be used as a second escape way.

A second production shaft (Shaft #2), located 4,000 feet to the east of Shaft #1, was completed in 1994 down to a depth of 1,722 feet and was used to mine Zones #6 and #7. Both ore zones were depleted in March 2000 and the workings were allowed to flood up to the 6th Level. Shaft #2 in this way serves as a water cistern for production at the Penna Shaft.

The Penna Shaft (formerly called Shaft #3), located approximately 1,640 feet to the east of Shaft #1, was completed down to a depth of 7,380 feet in March 2000. The Penna Shaft is used to mine Zones 20 North, 20 South, 7, and 6 and is the only production area active at LaRonde (Figure 11).
The Penna Shaft is circular, 18.2 feet in diameter and cement lined; it is arranged in roughly five longitudinal compartments: 2 compartments for hoisting of waste and ore rock to the surface using 27 ton-capacity transporters (or skips) connected to two 4,000 horsepower (hp) hoisting motors located on the surface (for a hoisting capacity of 8,730 tons per day from 7,220 foot-depth); 1 compartment for transporting personnel and materials from surface to each level access point using a double-decked transporter (or cage) connected to a 4,000 hp hoisting motor located on the surface; 1 compartment that is equipped with ladders for use as an alternate escape way; and a compartment that serves to route conduits for water, fuel, cement, compressed air, ventilation and electricity from surface to underground.
From the Penna shaft, access to the ore zones is provided by a series of sublevels spaced at regular 100 foot and 130 foot intervals, depending on their depth, from either a service ramp or directly from the shaft stations (Figure 12). Access to throughout the LaRonde mine by all the mechanical equipment is assured by drifts that are generally 14 feet by 15 feet in cross-sectional size and driven at a grade of 4% from horizontal (service ramps are generally driven at a grade of 17% from horizontal).

Access to fresh air is generally provided to each sublevel via a network of raises up to 22 feet in diameter (in addition to the Penna Shaft) that connect to the surface. Electric motor-driven ventilators (up to 3,500 hp) with a total capacity of 1.25 million cubic feet per minute (cfm) force climate-controlled air underground using an 13 million watt-rating (mwr) air-refrigeration plant (currently under construction), a 120 million BTU heating unit and a noise suppression system. In the same way, a separate network of raises allow for the evacuation, generally from each sublevel to surface, of heat and nauseous gases (emanating mainly from underground diesel motor equipment and explosive combustion gases) also using electric motor-driven ventilators (including a 3,500 hp surface ventilator with a 1.35 million cfm capacity and also equipped with a noise suppression system).

Equipment maintenance and major repairs are done in either of three main garages located on surface and underground on levels 134 and 206 at the Penna Shaft. A
maintenance garage located on Level 10 at Shaft #1 has been abandoned. Surface road maintenance and rock haulage equipment are furnished, operated and maintained by contractors under LaRonde Division supervision.

Waste rock and ore extracted during mining are transported along each sublevel and dumped into a network of passes that generally lead by gravity directly (or by transfer using hauling equipment) to two separate shaft loading stations (located on levels 152 and 220). Waste rock also serves to fill empty stopes or is stored in abandoned drifts. The ore pass systems lead to crushing stations linked by conveyors to the shaft loading stations. There are two crushing stations at LaRonde, each with a capacity to crush 400 tons per hour to less than 4 inches size; one station is located below Level 149 and the other on Level 219. Waste rock and crushed ore are stored in dedicated silos located near the loading stations on Levels 149 and 220 until it can be loaded into skips for transport to surface. Total underground ore and waste storage capacity is roughly 10,000 tons and 5,000 tons, respectively. Waste rock can either be skipped to the surface or to a mid-shaft dumping station located near level 86 so that it can be recycled into the waste pass system. The crushing, conveying, loading and dumping operations are fully automated.

Water is used underground for dust suppression, lubrication and cooling equipment is transported by pipeline from surface (while fresh potable water is transported by container from surface). Used water is collected in large sumps and piped directly to surface for treatment using high capacity dirty water pumps (335 gallons per minute) capable of handling acid water and 5% solids in suspension.

Four mining methods have historically been used at the LaRonde Mine: open pit for the two surface deposits, sublevel retreat, longitudinal retreat with cemented backfill, and transverse open stoping with both cemented and unconsolidated backfill. The primary source of ore at the LaRonde Mine continues to be from underground mining methods.

Currently, two mining methods are used depending on the thickness of the zones:

- Where the thickness of the economic interval of the mineralized zone varies between a minimum mining width of 10 feet up to 75 feet or more in thickness, for example in zones 20 North and 20 South, the ore is mined using transverse open stoping with both cemented and unconsolidated backfill. Stope length is generally 50 feet along strike (Figure 13) while the stope height is fixed by the sublevel spacing (that varies between 100 and 120 feet depending on the depth of the sublevel).
- Where the thickness of the economic interval of the mineralized zone is consistently narrow (less than 12 feet), for example in Zone 7, in some portions of Zone 20 South and along the margins of Zone 20 North, a longitudinal retreat mining method is used with cemented backfill or paste fill (fig. 13). Stopes are also mined in 45-foot long sections along strike (the stope height is always fixed by sublevel spacing).
In the transverse open stoping method, the ore zone stoping block is almost completely undercut and overcut (from the zone’s footwall to hangingwall generally) by transverse drifts from vertically adjacent sublevels (figs. 12 and 13). This allows access of mechanical equipment to prepare production, to completely extract the ore and to finally backfill the stoping block. In a mining area selected for mining by transverse open stoping, 50% of the ore is mined in the first pass (primary stopes) and filled with cemented rock fill or paste fill from the paste backfill plant located on the surface at the milling facility. On the second pass, the remainder of the ore (secondary stopes located between two primary stopes) is mined and filled with unconsolidated waste rock fill or paste fill.

In longitudinal retreat mining (fig. 13), the ore zone is completely undercut and overcut by vertically adjacent sublevel drifts that follow the strike length of ore zone. The mining sequence begins with the stoping block located farthest from its most adjacent sublevel access point; once a stope has been mined and backfilled, the mining cycle is repeated in each next adjacent stope in a (retreating) direction back to the access point until 100% of the ore is mined in a particular sublevel mining area.

The surface paste backfill plant has been designed to produce 200 short tons of fill per hour. An additional feature of the paste backfill plant is its ability to mix paste fill with
crushed rock providing a higher (stiffer) strength fill (or reduced binder). The backfill is delivered, on batch basis from the plant attached to the LaRonde, underground by a network of pipes and boreholes up to 6 inches in diameter that lead to selected empty stopes. Alternately, cement slurry produced from either the LaRonde cement plant, located at the recently acquired Bousquet 2 property or from a contractor plant located near the Penna Shaft, can also be delivered on a batch basis from surface via a pipeline for backfilling or construction purposes.

The impact of stress transfer following a stope production blast has been minimized by adopting an ascending pyramidal mining sequence (1-3-5; see figure 11). This sequence has been adopted for both the upper and lower parts of the ore body. On the lower levels, the stopes are mined following a fixed sequence along two ascending retreat fronts located on levels 194 and 215.

The ore zones have been subdivided into five blocks on the basis of elevation.

<table>
<thead>
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<th>BLOCK NUMBER</th>
<th>LEVELS INCLUDED</th>
<th>ZONE</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>125 to 86</td>
<td>20N AND 20S</td>
</tr>
<tr>
<td>2</td>
<td>152 to 125</td>
<td>20N AND 20S</td>
</tr>
<tr>
<td>3</td>
<td>194 to 152</td>
<td>20N, 20S AND 7</td>
</tr>
<tr>
<td>4</td>
<td>215 to 194</td>
<td>20N, 20S AND 7</td>
</tr>
<tr>
<td>5</td>
<td>215 to 236</td>
<td>20N AND 20S</td>
</tr>
</tbody>
</table>

Sill pillars have been established at the base blocks 1 to 4 whereas a descending mining method is forecast below Level 215. In the lower part of the mine, two mining areas are active (ascending from Levels 215 and 194) and a third has been proposed descending from Level 215; in the upper part of the mine, two mining areas are active (ascending from levels 152 and 125).

The proposed descending mining front (below Level 215) will descend on a level-by-level basis under backfill (undercut and fill). Currently, the mining sequence is under review with respect to mining on three sublevels, thereby proceeding under fill on only one occasion.

The mining plan is based on zone accessibility (feet of development) and the gold grade independent of the mining rate. Zone 20 North (Zinc) in the upper area is used as a buffer zone and its output is limited to what is required to fulfill the mining rate requirement after the production from other sources have been maximized.
25.1.2 Metallurgical Processes

Surface facilities at the LaRonde Mine include a 7,000 tons of ore treated per day milling complex, which has been expanded three times from the original 2,000 tons of ore treated per day rate that was established in 1988. The first expansion to 3,600 tons per day was completed at the beginning of 2000 and the second expansion to 5,000 tons per day was completed at the end of the third quarter of 2001. The expansion to 7,000 tons per day was completed in October 2002. This expansion consisted of additions to the grinding and precious metals circuits and modifications to the copper and zinc flotation circuits. A new ore handling system was completed at the end of 1999. It included a truck dump linked by a new conveyor gallery to a 5,000-ton coarse ore bin. The coarse ore bin feeds a semi-autogenous (SAG) mill that was installed at the end of 1999. Ore from the Penna Shaft is transported over the 1.2 kilometre distance to the ore handling facility by 35-ton trucks.

During 2004, the mill processed approximately 2.7 million tons of ore, averaging 7,399 tons of ore treated per day (tpd) and operating over 94.0% of available time.

Figure 14: LaRonde Milling Flowsheet - 7000 tons per day

The milling complex consists of a grinding, gravity, copper flotation, zinc flotation, and a precious metals recovery circuit and refinery (fig. 14). A copper concentrate containing
approximately 75% of the gold and byproduct silver and copper is recovered. The zinc flotation circuit produces a zinc concentrate containing approximately 5% of the gold. The remaining 20% is recovered by a precious metals circuit including a refinery using the Merrill Crowe process and shipped as doré bars. Both the zinc and copper flotation circuits consist of a series of column and mechanical cells that sequentially increase the zinc concentrate and copper concentrate quality. The precious metals circuit, that processes the tails from the base metal flotation circuit, consists of: 1) A leaching circuit (using cyanide); 2) A counter-current decantation (CCD) circuit that progressively removes the solids (tailings) from the precious metal-rich leachate; and 3) A refinery that precipitates, filters, and finally, calcines the precious metal-rich zinc pulp in order to produce doré.

Currently, water is treated at various facilities at the LaRonde Division. Prior to the water entering the tailings pond system, cyanide is removed at a cyanide destruction facility (associated with the paste backfill plant) using a sulphur dioxide (Inco) process. A secondary treatment plant located between the #1 and #2 polishing ponds uses a peroxysilica process to complete the cyanide destruction process. In addition, water with higher than permissible acidity is treated by lime in the mill complex prior to being released to the environment. In the first quarter of 2004, in response to revised Federal mining effluent regulations, the Company completed and commissioned a new water treatment plant that will eliminate tailing effluent toxicity immediately prior to discharge. The plant uses a biological treatment process. Prior to the completion of the water treatment plant, the Company retained excess water in its tailing pond complex. At the end of March 2004 treated water released from the plant successfully passed a toxicity test. The flow rate is steadily being increased as the biomass used to effect treatment continues to build up.

Tailings not recycled underground in paste backfill, are stored in tailings ponds covering an area of approximately 293 acres and waste rock is stored in two waste rock piles with a combined volume of approximately 50.4 million cubic feet. The Company holds mining claims to the north-east, to the east and to the south-east of the tailings ponds that would allow expansion of the tailings ponds and the establishment of additional waste disposal areas.

25.1.3 Production Forecast

Revenues from mining operations are expected to be as 2004 in 2005 as current market conditions indicate that metal prices will be similar as those realized in 2004.

The table below summarizes our estimated production for 2005 (as of January 1st 2005):

<table>
<thead>
<tr>
<th>Payable metal production</th>
<th>2005 Estimate</th>
<th>2004 Actual</th>
<th>% Change from 2004</th>
</tr>
</thead>
<tbody>
<tr>
<td>Gold (ounces)</td>
<td>287,921</td>
<td>271,567</td>
<td>6%</td>
</tr>
<tr>
<td>Silver (000’s ounces)</td>
<td>5,475</td>
<td>5,699</td>
<td>-4%</td>
</tr>
<tr>
<td>Copper (000’s pounds)</td>
<td>18,553</td>
<td>22,816</td>
<td>-19%</td>
</tr>
<tr>
<td>Zinc (000’s pounds)</td>
<td>161,171</td>
<td>167,283</td>
<td>-4%</td>
</tr>
</tbody>
</table>

25.2 RECOVERABILITY

25.2.1 Introduction

Precious metals recovery models used in the 7,000 tpd feasibility study (Girard et. Al., 2001) were developed using actual operating data from January 1998 to April 2001 and from March 1998 to April 2001, for gold and silver respectively. For base metals, the evaluation considered daily operating data for the period of October 2000 to April 2001 (corresponding to the first 7 months of processing of Penna shaft ore at 5000 tpd).

In 2004, zinc recoveries averaged 83.5% with peak monthly recovery rates of 85.4% (on a monthly basis). Zinc concentrate quality has averaged over 54.23% zinc. Copper recoveries have averaged over 78.9%, with peak monthly recovery rates of 83.1% (on a monthly basis). Copper concentrate quality averaged 14.85% for the year.

Since commercial production began in 1988, gold recoveries have averaged over 92%. During 2004, gold recoveries averaged 91.5%.

25.2.2 Gold Recoveries

Figure 15 illustrates gold recovery versus feed grade. Monthly operating data from January 1998 to April 2001 has been used to develop the gold recovery model. The gold recovery model curve has been established using the correlated results between final tails to feed grade. Gold recovery, in the grade spectrum of 2 g/t to 10 g/t, is expected to increase from approximately 93% to 95%.

25.2.3 Silver Recoveries

Similarly, silver’s recovery versus feed grade relationship has also been derived using monthly averages from the period of March 1998 to April 2001. The silver model curve as well as daily and monthly operating data is illustrated in Figure 16. The established model correlates well for low as well as with higher silver feed grades. Silver recovery, in the grade range between roughly 30 g/t and 100 g/t, is expected to increase from 60% to 85%.
25.2.4 Copper Recoveries

The copper recovery versus feed grade model has been estimated using daily operating data from October 2000 to April 2001 (fig. 17). The copper recovery model has been established based on the correlation between feed and tails grade similarly to the precious metal models. Copper recovery, for grades that vary between 0.06% and 0.65%, is expected to increase from roughly 31.5% to 79.5%.

25.2.5 Zinc Recoveries

The zinc performance curve has been determined using the average performances of similar Canadian operations. This has proved to be realistic as operating performance fell in line with the model. However, the model was slightly adjusted to better represent the actual LaRonde operating data. Figure 18 outlines both the model and registered daily operating data covering the period of October 2000 to April 2001. Zinc recovery, in the grade range between 1.5% and 6.5%, is expected to increase from 74% to 88%. Adjustment to the circuit throughput increase (from 2000 tpd to 5000 tpd during this period) explains in part that some of the results obtained were lower than the model. In addition, the lower recoveries were registered at the higher feed grade range because circuit design was exceeded.
Figure 16  Silver Recovery vs Feed Grade Relationship
January 2000 - April 2001

Feed Grade - g/t Ag

Ag Recovery (%)
Figure 17  Copper Recovery vs Feed Grade Relationship  
October 2000 - April 2001

Copper recovery (\%) = Conc \times (Head - Tails) 
Head \times (Conc - Tails)

Actual Recovery - Daily basis  
Copper Recovery Model

Figure 18  Zinc Recovery vs Feed Grade Relationship  
October 2000 - April 2001

Zinc recovery (\%) = Conc \times (Head - Tails) 
Head \times (Conc - Tails)

Actual Recovery - Daily basis  
Zinc Recovery Model
25.3 MARKETS

25.3.1 Introduction

The LaRonde Division currently extracts marketable gold, silver, copper and zinc. The marketable metals produced are shipped to smelters and refineries produced in the form of doré, copper concentrate and zinc concentrate.

25.3.2 Doré

Both the Canadian Royal Mint in Ottawa, Ontario, Canada and Johnson Matthey Canada Ltd. in Ontario, Canada are refiners for LaRonde Division gold/silver doré bars produced from the leaching circuits. Transportation is undertaken by SECU from the LaRonde Division to either refinery. The refined gold and silver is allocated to the Agnico-Eagle Mines Ltd. account for future sale by the Company.

25.3.3 Copper concentrate

The LaRonde Division has produced and marketed copper concentrate since 1988. Prior to 2001, copper concentrate shipments were sent exclusively to Noranda Inc.’s (Noranda) Horne smelter complex in Rouyn-Noranda, Quebec, Canada. At the time of the 7,000 TPD Feasibility Study in 2001, the Company had entered into sales contracts for copper concentrate with Noranda and Dowa Mining Co. Limited (Dowa) in Japan. This sales diversification resulted in more competitive smelting terms.

Copper concentrates are trucked to the Noranda’s Horne Smelter located about 50 km West from LaRonde. Concentrates for Dowa are trucked from the mine to the Cadillac storage facilities and transferred by rail to the Port of Montreal and/or Quebec by the Canadian National Railway and then by ocean vessels for transportation to the Port of Aomori in Japan for delivery to the Kosaka Copper Smelter.

25.3.4 Zinc Concentrate

LaRonde has produced zinc concentrate since 1998 and has shipped the production mainly to the Akita Smelter of Dowa in Japan and to Teck-Cominco Ltd.’s (Cominco) Trail Smelter in Kimberley, British Columbia, Canada. With the high level of production experienced in 2001, sales have also been made Noranda’s CEZ smelter in Valleyfield, Quebec, Canada as well as to three other trading companies, with these companies selling on to smelters in Europe and Asia. Since 2004, new smelting agreement was also concluded for the zinc with Noranda’s smelter in Timmins Ontario, Canada.

Zinc concentrate was used to be trucked from the mine to the Cadillac storage facilities and then transferred by rail directly to the smelters in Canada or to the Ports of Montreal and/or Quebec by the Canadian National Railway for shipment by ocean vessel to the
port of final destination. The zinc concentrate are now mostly trucked to Timmins Ontario.

25.4 CONTRACTS

The terms for contracts for certain mining, diamond drilling, construction, consulting, transportation, smelting, refining and smelting activities negotiated by the LaRonde Division are within market parameters.

In the case of copper and zinc concentrate smelting contracts, metal accountability, penalties, quotation periods and payment timing are fixed as are the principal non-monetary terms of the contracts. Specific terms are unique to each contract and are negotiated in confidence. In accordance with international practice, the treatment charges, refining charges and price participation are negotiated annually in line with the Japanese and European benchmark.

In the case of copper concentrate smelting contracts, the applicable metal accountabilities are generally 95% to 98% for gold, and 93% to 97% for silver, in both cases varying with grade and 100% less one unit for copper. Copper concentrate quality is expected to be about 22% copper, about 3-9 ounces per dry short ton (dst) for gold and about 30-90 ounces per dst for silver. This will vary year to year and will gradually decrease as a function of head feed grade reduction. Penalty charges in total for deleterious elements (zinc, lead, bismuth, tellurium, arsenic and antimony) are approximately US$27 per dst ± US$5. Treatment and other charges are determined by the market, but in total the overall cost of smelting, refining, penalties, freight, storage and other offsite cost, including assaying, representation and marketing is approximately US$225 per dst. Of this cost, US$85 per dst ± US$18, is accounted for by refining charges only.

In the case of zinc concentrate smelting contracts, when applicable, total metal accountabilities are generally about 50% to 70% for gold, and 40% to 60% for silver, in both cases varying with grade, and 85% for zinc. Zinc concentrate quality is expected to be approximately 53% to 56% Zinc, approximately 0.1-0.3 ounces per dst for gold and 6-12 ounces per dst for silver. This will vary year to year as a function of head feed grade reduction. LaRonde zinc concentrates are relatively clean and penalty charges in total for deleterious elements (iron and cadmium) are approximately US$2 per dst. Treatment and other charges are determined by the market, but in total the overall cost of smelting, refining, freight, storage and other offsite costs, including assaying, representation and marketing is roughly US$240 per dst.

25.5 ENVIRONMENTAL CONSIDERATIONS

The LaRonde Division mining and mineral processing operations and exploration activities are subject to extensive Canadian federal and provincial laws and regulations, and local laws and regulations governing waste disposal, toxic substances, environmental protection, mine safety and other matters.
The Quebec Mining Act requires mining operations to submit a rehabilitation plan and a financial guarantee covering 70% of the cost of rehabilitation work on accumulation areas. The LaRonde mine accumulation areas comprise the tailings pond and the waste rock piles. With the help of Golder and Associates, the LaRonde rehabilitation plan was prepared and first submitted to the Quebec Ministry of Natural Resources (MNR) in May 1996 (the reclamation costs at that time were estimated to be $17 million).

In response to comments from MNR and from the Ministry of the Environment (ME), the plan was revised by Golder and re-submitted in June 2001. The total rehabilitation cost was estimated at C$21.7 M non-discounted and at C$7.4 M in discounted terms (using 3% inflation and 10% discounting rates). At December 31, 2004, the Company had a total reclamation provision of $5.9 (C$7.1) million allocated for the LaRonde.

Once MNR approves the rehabilitation plan, annual payments of the guarantee will be made over the next 15 years in the following proportions: 0.8%, 2.5%, 4.1%, 5.8%, 7.4%, 9.1%, 10.7%, 12.4%, 14.1%, 15.7% and 17.4%.

In January 2003, the Company received a notice of infraction from the Quebec Ministry of the Environment in connection with a controlled discharge of water of excess toxicity, which was carried out over a three-month period in the summer of 2002. The purpose of the discharge was to establish favourable construction conditions for the increase of tailings pond capacity in the autumn of 2002. No fine was payable in respect of the notice of infraction, "however, the notice required production of a report detailing the causes of algae proliferation at the LaRonde Mine, which was delivered in 2003."

### 25.6 TAXES

#### 25.6.1 Introduction

The LaRonde operation is currently subject to the following taxes:

- Quebec mining duties
- Federal and provincial income taxes
- Federal and provincial capital taxes

#### 25.6.2 Quebec Mining Duties

Mining duties are payable to the Quebec Ministry of Natural Resources, Wildlife and Parks, at a rate of 12% of income from mining operations. For 2005, under the metal price and foreign exchange rate assumptions used in this report, the Company does not expect to pay cash mining duties due to the availability of capital and exploration pools. In the past, through the utilization of these pools, the Company was generating a cash refund of 12% of its exploration and development expenditures. Given the Company’s
depleting capital and exploration pool balances, the Company does not expect to generate large refunds in 2005 and expects to begin paying cash mining duties in 2006.

25.6.3 Federal and Provincial Income Taxes

For 2004, federal income taxes are payable at a rate of 26%. This rate is effectively reduced by a resource allowance available to mining companies. The resource allowance is calculated at 25% of a company’s resource profits. The federal government has introduced legislation that will change the taxation of resource profits. By 2007, the federal income tax rate will be reduced to 21% and the federal resource allowance will be eliminated. The new legislation will also allow for the deductibility of provincial mining duties paid. The Company expects the new legislation to have a neutral impact on income taxes. Provincial income taxes are assessed on a similar basis as the current federal system. The Company currently is assessed at a blended provincial rate (Ontario & Quebec) of approximately 8% (after provincial resource allowances). Although many provinces have indicated their intention to mirror the federal changes in the resource taxation structure, neither Ontario nor Quebec have introduced such changes yet. In 2005, the Company does not expect to pay cash income taxes due to the availability of tax loss carry-forwards and the availability of capital, development and exploration pools. Given the Company’s current capital, development and exploration pool balances, the earliest the Company expects to be paying cash income taxes is 2006.

25.6.4 Federal and Provincial Capital Taxes

Capital taxes are assessed on the Company’s total capitalization. Federal capital taxes are currently being phased out and will be completely eliminated by 2008. For 2005, federal capital taxes will be payable at 0.175% of corporate capitalization, of which LaRonde capitalization is the largest component. Provincial capital taxes are assessed on essentially the same base as federal capital taxes. Currently the Company is assessed provincial capital taxes at a blended rate (Ontario and Quebec) of approximately 0.4%. Quebec capital taxes may be reduced by a tax credit of 20% of eligible exploration expenditures. Eligible expenditures include mainly surface exploration and mine construction and pre-production.

25.7 CAPITAL AND OPERATING COST ESTIMATES

25.7.1 CAPITAL COST ESTIMATE

25.7.1.1 Introduction

In 2003, the Company began construction of a water treatment plant that eliminates tailing effluent toxicity immediately prior to discharge. The plant was commissioned in 2004. Also in 2004, the Company commissioned a bulk air cooling plant in order to moderate the effect of increased summer month ambient temperatures. In 2004, the
Company also began the development of a ramp between levels 215 and 236 to access ore currently outside the Penna Shaft infrastructure.

The Agnico-Eagle regional development team will continue to evaluate a development project at LaRonde to access the mineral resource base located outside of the Penna Shaft infrastructure (the “LaRonde II” project). The phase planned for 2005 is to continue the deep drilling program from the Level 215 drift and the bottom of the Bousquet Shaft and to incorporate those results into a feasibility study which is expected to be completed in late 2005.

25.7.1.2 Capital Cost Estimate for LaRonde in 2005

In 2005, Agnico-Eagle expects to spend approximately $25.1 million for project and sustaining capital expenditures at LaRonde and $1.6 million on the LaRonde II drilling and feasibility study. These capital expenditures are expected to be completely funded out of operating cash flow from LaRonde. An expected increase in gold and by-product metal production and rising metal prices will be the key drivers to attaining strong operating cash flows.

The following table provides a summary of the 2005 estimated capital and exploration expenditures (thousands):

<table>
<thead>
<tr>
<th>Project</th>
<th>Capitalized</th>
<th>Expensed</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>LaRonde I projects</td>
<td>$11,100</td>
<td>nil</td>
<td>$11,100</td>
</tr>
<tr>
<td>LaRonde I sustaining</td>
<td>$14,100</td>
<td>nil</td>
<td>$14,100</td>
</tr>
<tr>
<td>LaRonde II drilling &amp; feasibility</td>
<td>$1,600</td>
<td>nil</td>
<td>$1,600</td>
</tr>
</tbody>
</table>

According to current economic parameters, future capital expenditures, including closure costs, are expected to be funded by operating income.

25.7.1.3 Capital Cost Estimate for Extracting Reserves outside the Penna Shaft Infrastructure

The pre-feasibility study for LaRonde II include the portion of probable reserve as evaluated on December 31st, 2003 for LaRonde that is located between Level 302 (former level 317 at the time of the study) upward to the base of Level 236 and was 2.928 million ounces of gold in 16.982 million tons (the base of Level 236 is considered to be the limit of the Penna Shaft infrastructure). The basis for determining that this portion of the LaRonde mineral resource is classified as a probable mineral reserve are prefeasibility studies (see section 19.1 and Appendix B). The studies so far show that a winze alternative (which could result in extracting the ore through the Penna Shaft), at a rate of 5,000 tpd, should it be chosen, would theoretically require a capital expenditure of $C282 million. The reserves below Level 236 have not been incorporated into a mine plan.
The “LaRonde II” project is being evaluated and is expected to be completed by mid 2005. The feasibility study will likely affect the current capital cost estimates for LaRonde.

25.7.2 OPERATING COST ESTIMATE

25.7.2.1 Introduction

In 2004, production costs decreased 6% to $98 million from $105 million. Operating costs per short ton were lower by 8% compared to 2003 at C$48 per short ton. The following table summarizes the components of operating cost per short ton:

<table>
<thead>
<tr>
<th>Operating Cost items</th>
<th>2004</th>
<th>2003</th>
</tr>
</thead>
<tbody>
<tr>
<td>Definition Drilling</td>
<td>0.32</td>
<td>0.29</td>
</tr>
<tr>
<td>Stope Development</td>
<td>4.69</td>
<td>6.75</td>
</tr>
<tr>
<td>Mining</td>
<td>9.09</td>
<td>9.84</td>
</tr>
<tr>
<td>Underground Services</td>
<td>14.24</td>
<td>14.73</td>
</tr>
<tr>
<td>Milling</td>
<td>15.02</td>
<td>15.67</td>
</tr>
<tr>
<td>Surface Services</td>
<td>1.08</td>
<td>1.28</td>
</tr>
<tr>
<td>Administration</td>
<td>3.49</td>
<td>3.69</td>
</tr>
<tr>
<td>Total Minesite Cost per ton:</td>
<td>C$47.93</td>
<td>C$52.25</td>
</tr>
</tbody>
</table>

Operational improvements and higher tonnage milled were the main factors contributing the decrease in minesite cost per ton in 2004. At the end of 2004, $7.4 million of minesite production costs were allocated to inventory which was significantly increased from the small inventory adjustments made in 2003. Due to the large increases in production volumes in 2004, zinc and copper concentrate inventories where much higher at the end of 2004 compared to 2003. This inventory adjustment reflects the production costs associated with unsold concentrates at the end of the year.

In the fourth quarter of 2004, 793,000 tons were milled at LaRonde, which represents a 26% increase over the same quarter of 2003. Fourth quarter gold production was 2% lower than 2003 due to higher than expected dilution on the lower level mining horizons. Total cash costs per ounce in the fourth quarter were $13, a 94% decrease over 2003 and a new quarterly record. Increased byproduct production and increasing byproduct metal prices were the principal factors driving total cash costs per ounce lower in the quarter. Minesite costs per ton in the fourth quarter were C$48 per short ton compared to C$54 per short ton in the prior year. The improvement in minesite costs per ton was due to the 26% increase in mill throughput compared to the fourth quarter of 2003.

For 2005, tons milled and production costs are expected to remain essentially unchanged and thus minesite costs per ton are also expected to remain essentially unchanged. In 2004, total cash costs to produce an ounce of gold decreased to $56 from $269 in 2003 and $182 in 2002. Total cash costs are comprised of minesite costs reduced by net silver, zinc and copper revenue. Total cash costs per ounce are affected by various factors such as the number of gold ounces produced, operating costs, US$/C$ exchange rates, production
royalties and byproduct metal prices. The table below illustrates the variance in total cash costs per ounce attributable to each of the variables which affected it. The most significant factors contributing to the decrease in total cash costs per ounce in 2004 were higher byproduct revenue as a result of increased production and higher byproduct metal prices, elimination of the El Coco royalty and increased gold production. Total cash costs per ounce is not a recognized measure under US GAAP and is described more fully below.

<table>
<thead>
<tr>
<th>2004</th>
<th>2003</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total cash costs per ounce (prior year)</td>
<td>$269</td>
</tr>
<tr>
<td>Lower (higher) gold production</td>
<td>(35)</td>
</tr>
<tr>
<td>Stronger Canadian dollar</td>
<td>29</td>
</tr>
<tr>
<td>Higher (lower) El Coco royalty</td>
<td>(47)</td>
</tr>
<tr>
<td>Cost associated with increased throughput</td>
<td>39</td>
</tr>
<tr>
<td>Foreign exchange hedge gains</td>
<td>(18)</td>
</tr>
<tr>
<td>(Higher) lower byproduct revenue</td>
<td>(181)</td>
</tr>
<tr>
<td>Total cash costs per ounce (current year)</td>
<td>$56</td>
</tr>
</tbody>
</table>

The tables below show the cash operating cost (on a per once basis) and the operating cost per ton at LaRonde since 2001.

<table>
<thead>
<tr>
<th></th>
<th>2004</th>
<th>2003</th>
<th>2002</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>(thousands, except ounces and per ounce amounts)</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Production costs (loss)</td>
<td>$98,168</td>
<td>$104,990</td>
<td>$75,969</td>
</tr>
<tr>
<td>Adjustments:</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Byproduct revenues, net of smelting refining and marketing charges</td>
<td>($82,521)</td>
<td>($41,254)</td>
<td>($27,850)</td>
</tr>
<tr>
<td>El Coco royalty</td>
<td>-</td>
<td>($12,888)</td>
<td>($10,764)</td>
</tr>
<tr>
<td>Accretion expense and other</td>
<td>($493)</td>
<td>($151)</td>
<td>($708)</td>
</tr>
<tr>
<td>Cash operating costs</td>
<td>$15,154</td>
<td>$50,697</td>
<td>$36,647</td>
</tr>
<tr>
<td>Gold production (ounces)</td>
<td>271,567</td>
<td>236,653</td>
<td>260,183</td>
</tr>
<tr>
<td>Cash operating costs (per ounce)</td>
<td>$56</td>
<td>$215</td>
<td>$141</td>
</tr>
<tr>
<td>El Coco royalty (per ounce)</td>
<td>$ -</td>
<td>$54</td>
<td>$41</td>
</tr>
<tr>
<td>Total cash operating costs (per ounce)</td>
<td>$56</td>
<td>$269</td>
<td>$182</td>
</tr>
<tr>
<td></td>
<td>2004</td>
<td>2003</td>
<td>2002</td>
</tr>
<tr>
<td>------------------------------</td>
<td>--------</td>
<td>--------</td>
<td>--------</td>
</tr>
<tr>
<td><strong>Production costs (loss)</strong></td>
<td>$98,168</td>
<td>$104,990</td>
<td>$75,969</td>
</tr>
<tr>
<td><strong>Adjustments:</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Inventory (i) and hedging (ii) adjustments</td>
<td>$12,107</td>
<td>$54</td>
<td>$609</td>
</tr>
<tr>
<td>El Coco royalty</td>
<td></td>
<td>($12,888)</td>
<td>($10,764)</td>
</tr>
<tr>
<td>Accretion expense and other</td>
<td>($314)</td>
<td>$519</td>
<td>($1,301)</td>
</tr>
<tr>
<td>Mine site operating costs (US$)</td>
<td>$109,961</td>
<td>$91,637</td>
<td>$64,513</td>
</tr>
<tr>
<td>Mine site operating costs (C$)</td>
<td>$142,702</td>
<td>$127,931</td>
<td>$101,289</td>
</tr>
<tr>
<td>Tons milled (000’s tons)</td>
<td>2,977</td>
<td>2,449</td>
<td>1,963</td>
</tr>
<tr>
<td>Operating costs per ton</td>
<td>$48</td>
<td>$52</td>
<td>$52</td>
</tr>
</tbody>
</table>

Notes:
(i) Under the Company’s revenue recognition policy, revenue is recognized on concentrates when legal title passes. Since total cash operating costs and operating costs per ton are calculated on a production basis, this adjustment reflects the portion of concentrate production for which revenue has not been recognized in the year.
(ii) Hedging adjustments reflect gains and losses on the Company’s derivative positions entered into to hedge the effects of foreign exchange fluctuations on production costs. These items are not reflective of operating performance and thus have been eliminated when calculating operating costs per ton.

### 25.7.2.2 Operating Cost Estimate LaRonde

Table 15 describes for LaRonde, the current and recent past operating costs and the estimated future operating costs based on the 7,000 tpd feasibility study (Girard et al. 2001) that was revised in 2002 (RPA, 2002). The cost estimates were based on mining the reserves (as they were estimated on December 31, 2000) down to Level 236. The NSR cut-off used to determine the reserves in this study use, in part, this estimate.

In addition more recent studies (Provencher, 2003; Emond, 2003) have refined the operating costs estimate for mining by ramp down to Level 236:

<table>
<thead>
<tr>
<th>Reserve Block below level 215</th>
<th>Operating cost per ton</th>
</tr>
</thead>
<tbody>
<tr>
<td>215-221</td>
<td>C$49</td>
</tr>
<tr>
<td>221-227</td>
<td>C$50</td>
</tr>
<tr>
<td>227-233</td>
<td>C$51</td>
</tr>
<tr>
<td>233-239</td>
<td>C$52</td>
</tr>
<tr>
<td>239-245</td>
<td>C$54</td>
</tr>
</tbody>
</table>
Table 15: LaRonde Operating Cost per ton (according to 7000 ton/day study by Girard et al. 2001, revised 2002 and adjusted with data up to 2004)

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
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<th></th>
<th></th>
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<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Ore processed (000's tons)</td>
<td>1,805</td>
<td>1,963</td>
<td>2,449</td>
<td>2,623</td>
<td>2,623</td>
<td>2,623</td>
<td>2,623</td>
<td>2,623</td>
<td>2,623</td>
<td>2,623</td>
<td>2,623</td>
<td>2,623</td>
<td>2,623</td>
<td>2,623</td>
<td>2,623</td>
<td>685</td>
</tr>
<tr>
<td>Tons/day</td>
<td>5,000</td>
<td>5,453</td>
<td>6,803</td>
<td>7,287</td>
<td>7,287</td>
<td>7,287</td>
<td>7,287</td>
<td>7,287</td>
<td>7,287</td>
<td>7,287</td>
<td>7,287</td>
<td>7,287</td>
<td>7,287</td>
<td>7,287</td>
<td>1,903</td>
<td></td>
</tr>
<tr>
<td>Definition Drilling</td>
<td>$0.21</td>
<td>$0.35</td>
<td>$0.29</td>
<td>$0.20</td>
<td>$0.20</td>
<td>$0.20</td>
<td>$0.20</td>
<td>$0.20</td>
<td>$0.20</td>
<td>$0.20</td>
<td>$0.20</td>
<td>$0.20</td>
<td>$0.20</td>
<td>$0.20</td>
<td>$0.20</td>
<td></td>
</tr>
<tr>
<td>Underground Services</td>
<td>$14.51</td>
<td>$13.27</td>
<td>$14.73</td>
<td>$13.17</td>
<td>$10.73</td>
<td>$10.13</td>
<td>$10.15</td>
<td>$10.15</td>
<td>$10.15</td>
<td>$10.15</td>
<td>$10.15</td>
<td>$10.15</td>
<td>$10.15</td>
<td>$10.15</td>
<td>$10.15</td>
<td>$7.56</td>
</tr>
<tr>
<td>Surface services</td>
<td>$1.22</td>
<td>$1.54</td>
<td>$1.28</td>
<td>$1.33</td>
<td>$1.00</td>
<td>$1.00</td>
<td>$1.00</td>
<td>$1.00</td>
<td>$1.00</td>
<td>$1.00</td>
<td>$1.04</td>
<td>$1.04</td>
<td>$1.04</td>
<td>$1.04</td>
<td>$1.04</td>
<td>$0.59</td>
</tr>
<tr>
<td>Administration</td>
<td>$3.77</td>
<td>$4.29</td>
<td>$3.69</td>
<td>$3.69</td>
<td>$2.85</td>
<td>$2.85</td>
<td>$2.85</td>
<td>$2.85</td>
<td>$2.85</td>
<td>$2.85</td>
<td>$2.85</td>
<td>$2.85</td>
<td>$2.85</td>
<td>$2.85</td>
<td>$2.85</td>
<td>$2.01</td>
</tr>
<tr>
<td>Total</td>
<td>$52.78</td>
<td>$51.60</td>
<td>$52.25</td>
<td>$46.57</td>
<td>$43.36</td>
<td>$43.39</td>
<td>$42.38</td>
<td>$42.39</td>
<td>$42.39</td>
<td>$33.52</td>
<td>$33.53</td>
<td>$33.57</td>
<td>$33.57</td>
<td>$33.57</td>
<td>$33.57</td>
<td>$30.09</td>
</tr>
</tbody>
</table>

Tonnage milled reflects 95% mill availability
25.7.2.3 Operating Cost Estimate for Extracting Reserves outside the Penna Shaft Infrastructure

As described in section 25.7.1.3, a portion of probable reserve at LaRonde is located between Level 302 upward to the base of Level 236 (the base of Level 236 is considered to be the limit of the Penna Shaft infrastructure). The basis for determining that this portion of the LaRonde mineral resource is classified as a probable mineral reserve are prefeasibility studies (see section 19.1 and Appendix B). The studies so far show that a winze alternative (which could result in extracting the ore through the Penna Shaft), at a rate of 5,000 tpd, should it be chosen, would have an average operating cost of C$51.39 per ton. The reserves below Level 236 have not been incorporated into a mine plan.

The “LaRonde II” project is being evaluated and is expected to be completed by mid-2005. The feasibility study will better fix the estimated operating cost estimate for extracting reserves outside the Penna Shaft infrastructure.

25.8 ECONOMIC ANALYSIS

25.8.1 LaRonde Forecast and Sensitivity Analysis

For 2005, the Company expects total cash costs to be in the range of $135 to $145 per ounce of gold compared to $56 achieved in 2004. Revenue from mining operations and total cash cost estimates are based on gold and byproduct metal price assumptions that are well below current prices and prices realized in 2004. If current metal prices and exchange rates were used, total cash operating costs would be well below $100 per ounce and revenues from mining operations would increase significantly. Net silver, zinc and copper revenue is treated as a reduction of production costs in arriving at estimates of total cash costs per ounce, and therefore production and price assumptions play an important role in these estimates. The table below summarizes the metal price assumptions, exchange rate and production assumptions used in deriving the estimated total cash costs per ounce for 2005.

<table>
<thead>
<tr>
<th>Assumptions</th>
<th>Production</th>
<th>Price</th>
</tr>
</thead>
<tbody>
<tr>
<td>Gold (ounces)</td>
<td>280,000</td>
<td>$375</td>
</tr>
<tr>
<td>Silver (000’s ounces)</td>
<td>5,500</td>
<td>$6.00</td>
</tr>
<tr>
<td>Zinc (000’s pounds)</td>
<td>160,000</td>
<td>$0.45</td>
</tr>
<tr>
<td>Copper (000’s pounds)</td>
<td>18,000</td>
<td>$1.15</td>
</tr>
<tr>
<td>US$/C$ exchange rate</td>
<td></td>
<td>$1.27</td>
</tr>
</tbody>
</table>

The estimated sensitivity of LaRonde’s 2005 estimated total cash operating costs to a 10% change in the metal prices and exchange rates assumptions above follows:
The LaRonde operation has shown positive cash flows since 2000 and currently estimates that, given current market and tax conditions and expected cost estimates for 2005 that are shown above, that positive cash flows will also be realized in 2005. Because the current proven and probable reserves for LaRonde I have, by definition, an estimated value much greater than the operating costs forecast (and that future capital expenditures and taxes are not expected to be significant) to extract those reserves, cash flows are expected to be positive until those reserves are exhausted.

25.8.2 LaRonde II Prefeasibility Level Forecast and Sensitivity Analysis

As described in section 25.7, a portion of LaRonde probable reserve, namely 2.928 million ounces of gold in 16.982 million tons grading 0.17 oz/ton gold, 0.54 oz/ton silver, 0.34% copper and 0.10% zinc (based on the December 31st 2003 estimate), occurs below level 236 (the base of Level 236 is considered to be the limit of the Penna Shaft infrastructure). The basis for determining that this portion of the LaRonde mineral resource is classified as a probable mineral reserve are prefeasibility studies (see section 19.1 and Appendix B) that describe a winze alternative (which could result in extracting the ore through the Penna Shaft) with the following key assumptions:

<table>
<thead>
<tr>
<th>Extraction Rate</th>
<th>5,000 tpd</th>
</tr>
</thead>
<tbody>
<tr>
<td>Life of project</td>
<td>23 years</td>
</tr>
<tr>
<td>Production time</td>
<td>18 years</td>
</tr>
<tr>
<td>Gold price</td>
<td>$325 per ounce</td>
</tr>
<tr>
<td>Silver price</td>
<td>$5.00 per ounce</td>
</tr>
<tr>
<td>Copper price</td>
<td>$0.80 per pound</td>
</tr>
<tr>
<td>Zinc price</td>
<td>$0.50 per pound</td>
</tr>
<tr>
<td>US$/C$</td>
<td>1.40</td>
</tr>
<tr>
<td>Net Smelter Return cut-off</td>
<td>C$49.90-54.43/ton</td>
</tr>
<tr>
<td>Mill recovery</td>
<td>Variable (estimated using LaRonde recovery models)</td>
</tr>
<tr>
<td>Operating cost</td>
<td>C$51.39 per ton</td>
</tr>
<tr>
<td>Capital cost (millions)</td>
<td>C$282</td>
</tr>
<tr>
<td>Operating cost</td>
<td>C$51.39 per ton</td>
</tr>
</tbody>
</table>

The following results were obtained in the simulation:

<table>
<thead>
<tr>
<th>Annual gold production</th>
<th>230,000 ounces over 12 years</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cumulative cash flow (millions)</td>
<td>C$107</td>
</tr>
</tbody>
</table>
Payback | In year 14
---|---
Net present value (before tax) @ 8% disc. | -C$46 million
IRR (before tax) | 4%

A sensitivity analysis of the internal rate of return (IRR), before tax, of the winze option simulation is presented in Figure 20.

**Figure 19: Sensitivity of IRR (before tax) of LaRonde II Prefeasibility, Winze Option 5000 tons per day**

The winze scenario for mining the current indicated mineral resource located below Level 236 at LaRonde is mainly sensitive to gold price and gold grade.

The financial analysis shows that although the LaRonde II project, winze option, as of December 31st 2003, is economically marginal, the prefeasibility study shows that the mineral resource below Level 236 can classified as a mineral reserve. The reserves below Level 236 have not yet been incorporated into a mine plan.

The economic analysis is preliminary and does not reflect the estimated future of the LaRonde II project; there is a deep drilling program from the Level 215 drift and the bottom of the Bousquet Shaft designed to discover and convert into indicated mineral resource, inferred mineral resource in the LaRonde II project. The results of this program (which will likely enhance the economics) and other studies will be incorporated into a feasibility study that is expected to be completed by mid 2005. The feasibility study will provide the basis for a final decision by a financial institution to finance the development of the LaRonde II deposit for mineral production.
25.9 PAYBACK

Currently the LaRonde Mine is an operating mine that carries no operational debt. Forecasted expenditures, including closure costs, are expected to be funded by operating income. Therefore, payback period is not applicable.

25.10 MINE LIFE

The portion of proven and probable reserves as of December 31\textsuperscript{th} 2004 for LaRonde that occur above Level 236 is approximately 1.85 million ounces of gold in 19.4 million tons. The projected mine life to extract those reserves from the Penna Shaft is about 8 1/2 years at a mining rate of 7,287 tons per day.

As described in section 25.7, a portion of LaRonde probable reserve, namely 16.982 million tons (based on the 2004 Estimate, but now revised at 17.520 million tons containing 3.258 million ounces of gold in the 2005 estimate), occurs below level 236 (the base of Level 236 is considered to be the limit of the Penna Shaft infrastructure). The basis for determining that this portion of the LaRonde mineral resource is classified as a probable mineral reserve are prefeasibility studies (see section 19.1 and Appendix B). The studies so far show that a winze option (which would entail extracting the ore through the Penna Shaft), at a rate of 5,000 tpd, should it be chosen, would result in a theoretical production period of almost 18 years to completely extract the reserves and take 5 years to put into production. The reserves below Level 236 have not been incorporated into a mine plan.

There is a deep drilling program from the Level 215 drift and the bottom of the Bousquet Shaft designed to discover and convert into indicated mineral resource, inferred mineral resource in the LaRonde II project). The results of this program and other studies will be incorporated into a feasibility study that is expected to be completed by mid-2005. The feasibility study will likely affect the mine life expected at LaRonde.
26. Illustration

Figure 1  Localisation map  
Figure 2  Surface plan LaRonde property  
Figure 3  LaRonde longitudinal Penna shaft zones  
Figure 4  Regional geology  
Figure 5  LaRonde property detailed geology  
Figure 6  Section 7320E Zone 7 Penna Shaft  
Figure 7  Section 7440E Zone 20N and 20S Penna Shaft upper mine  
Figure 8  Section 7080E Zone 20N and 20S Penna Shaft lower mine  
Figure 9  Section 6840E Zone 20N LaRonde II  
Figure 10  Section 6360E Zone 20N LaRonde II  
Figure 11  LaRonde 2002 Five year Mine Plan Longitudinal Section  
Figure 12  LaRonde Typical Level Arrangement  
Figure 13  Longitudinal and Transverse Mining Layout  
Figure 14  LaRonde Milling Flowsheet - 7000 tons per day  
Figure 15  Gold Recovery vs Feed Grade Relationship  
Figure 16  Silver Recovery vs Feed Grade Relationship  
Figure 17  Copper Recovery vs Feed Grade Relationship  
Figure 18  Zinc Recovery vs Feed Grade Relationship  
Figure 19  Sensitivity of IRR (before tax) of LaRonde II Prefeasibility, Winze Option 5000 tons per day
APPENDIX A

Dilution and S.G. tables

For complete version please contact:

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Fax: 416-367-4681
APPENDIX B

Claim map; mining leases LaRonde and El Coco property; Ore reserve calculation memos; Assay Laboratory procedures; Standard sample preparation procedure; List of 2004 drill hole zone intercepts; Proposed 2005 diamond drilling budget; Penna Shaft stope reconciliation table; Dilution study results; Datamine user guide on grade estimation; 2005 block model search parameter and estimation parameter files; List of holes moved in zone 20N Au, Zn 20S and 7 zones in the wireframe for reserve estimate; 2004 Quality Assurance Quality Control report; 2004 standard sample procedure results for chip samples; 2003b Émond et al LaRonde Deep Mine Feasibility study executive summary. Statement of qualifications;

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APPENDIX C

2005 Mineral Reserve and Resource Metric Summary

For complete version please contact:

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APPENDIX D

2005 Mineral Reserve and Resource Metric Calculation at the Penna Shaft by zone, block and stope

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Tel: (819) 759-3700 #230  
Fax: (819) 759-3641  
Email: guy.gosselin@agnico-laronde.com

Certificate of Qualified Person

A. My Name is Guy Gosselin and I am a geological engineer and chief geologist at Agnico-Eagle Mines Ltd, LaRonde Mine Division, 20 Route 395 Cadillac Québec JOY 1C0. I reside at 60 Chemin de Peupliers, Preissac, (Québec) JOY 2E0.

B. I am a Qualified Person for the purpose of National Instrument 43-101 of the Canadian Securities Administrators having the following education degrees:

- Graduated of L’Université du Québec A Chicoutimi, Québec in 1994 with a B. Sc. degree in geological engineering;
- Graduated of L’Université du Québec A Chicoutimi, Québec in 1998 with a M. Sc. A. degree in Earth sciences;

The 2005 Mineral Resource and Mineral Reserve Estimate is based on my direct supervision as the chief geologist at the LaRonde Mine Division.

I am a registered Engineer in the Province of Québec (OIQ member 121625) and a registered Geologist in the province of Québec (OGQ member 761). I am a member of the Prospectors and Developers Association of Canada.

My relevant experience for the purpose of this report is:

- Have been a graduated geologist since 1994
- I have completed a master degree on the Doyon Mine (Barrick-Cambior) deposit from 1994 to 1998 on the timing of the mineralisation in the lithologic and deformation events that occurs in the deposit.
- Have been working at the Doyon Mine on a steady basis from 1996 to 2000 as exploration, production and senior production geologist.
- Have been working with Agnico-Eagle Div LaRonde since 2000 as exploration, production geologist and promote as chief Geologist in August 2002.

C. I am a shareholder of Agnico-Eagle Mines Ltd. and I hold option to purchase a certain number shares of Agnico-Eagle Mines Ltd.

D. I am responsible for the entire report.
E. I am not aware of any material fact or material change with respect to the subject matter of the Report, which is not reflected in the Report, the omission to disclose which makes the Report misleading.

F. I have read National Instrument 43-101. I am applying the test set out in section 1.5 of National Instrument 43-101. This report is prepared in compliance with this Instrument and form 43-101F1.

Dated this twenty-third day of March, 2005.

Signature of Qualified Person

Guy Gosselin Engineer, Geologist

Print name of Qualified Person

Stamped by Guy Gosselin
Engineer 121625, Québec
Geologist 761, Québec
I, Guy Gosselin Engineer and Geologist, do hereby certify that:

1. I am the Chief Geologist of:
   Agnico Eagle Mines Ltd.  
   LaRonde Mine Division  
   20 Road 395  
   Cadillac, Quebec, Canada  
   JOY 1C0

2. I graduated with a degree in Engineering Geology B.Sc. from L’Université du Québec À Chicoutimi, Québec in 1994. In addition, I have obtained a master degree in Earth Sciences from L’Université du Québec À Chicoutimi, Québec 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  
   Prospectors and Developers Association of Canada #14298.

4. I have worked as geologist for a total of 10 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 certified that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.


7. I have not had prior involvement with the property that is the subject of the Technical Report.
8 I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

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

I am employee of Agnico-Eagle Mines Ltd since June 2000

I am a shareholder of Agnico-Eagle Mines Ltd. and I hold option to purchase a certain number 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 23th Day of March, 2005.

[Signature]

Signature of Qualified Person

Guy Gosselin

Print name of Qualified Person
CONSENT of AUTHOR

TO:

British Columbia Securities Commission  
Alberta Securities Commission  
Saskatchewan Securities Commission  
The Manitoba Securities Commission  
Ontario Securities Commission  
Commission des valeurs mobilières du Québec  
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 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 twenty third day of March, 2005.

______________________________  
Signature of Qualified Person

Guy Gosselin
Engineer, Geologist  
Print Name of Qualified Person