

NI 43-101 Technical Report Preliminary Economic Assessment for the Strieborná Project, Eastern Slovakia

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1 SUMMARY

1.1 Introduction

Global Minerals Limited (Global Minerals) commissioned Mine Development Associates (MDA) and AGP Mining Consultants Inc. (AGP) to provide an Independent Mineral Resource Estimate and a Preliminary Economic Assessment (PEA), respectively, for their wholly-owned Strieborná property, near the town of Rožňava in the Gemer region of the Slovak Republic.

The Strieborná Project is located at latitude 48°40'29"N and longitude 20°32'31"E (Figure 1-1), 2 km to the north of the centre of the town of Rožňava in the southern foothills of the Spiš-Gemer Ore Mountains (*Spišsko-Gemerské Rudohorie*, also locally called the *Volovské Vrchy*), which form the southern part of the Slovak Ore Mountains (*Slovenské Rudohorie*) of the Western Carpathians. The property is easily accessible from several European centres and/or international airports (Vienna, Bratislava, and Budapest) by paved highways. The nearest airport is Košice International Airport situated 65 km east of Rožňava.

Figure 1-1: Project Location



The Slovak Republic lies in the mild zone of sub-continental climate, with distinctive seasonal changes and microclimate depending on elevation. The Strieborná Project, being located adjacent to the low ground of the Slaná River valley and its tributaries, has a mean annual temperature of about 8°C, with monthly average temperatures ranging

from approximately 0°C in January to 20°C in July, and annual rainfall of approximately 700 mm. Climate is not considered to be a factor restricting the duration of the operating season.

The principal assets of the Strieborná property encompass two mining licences known as Rožňava I and Rožňava III, located within the Rožňava mining cadastre area with ID No. 853011, Rožňava District (808), Kosice Region (008). The licenses allow the holder to mine polymetallic deposits including silver, copper, and antimony. There is no time limit on the licenses provided the holder complies with Slovakian mining law and annual filing/payment requirements.

This PEA has been prepared under the assumption that the Strieborná vein will be accessed through the upper workings of the existing Mária Mine. The mine will produce 110 kt/a of mill feed until early in Year 3, when annual production will increase to 180 kt/a. Seven years of production are planned. Longhole retreat, open-stope mining with rib and sill pillars is proposed, using mechanized equipment and truck haulage to surface. The stopes will not require backfill. Mining operations will be undertaken by owner-employed crews.

Mineral processing facilities will consist of a 500 t/d concentrator and associated infrastructure items, to be located approximately 2 km south of the mine haulage tunnel access. The concentrator plant will include simple crushing, grinding, flotation and dewatering. The flotation tailings will be filtered and deposited by truck on the refurbished existing tailings storage facility (TSF).

Metallurgical testwork has shown that a simple tetrahedrite flotation process is remarkably efficient. Very high copper and silver recoveries are achievable while rejecting almost all gangue minerals (primarily siderite). However, flotation concentrates also contain the penalty elements of mercury, antimony, and arsenic. To mitigate the penalties resulting from these elements, further processing options have been tested, including pyrometallurgical work to remove/reduce mercury, and hydrometallurgical work to develop an antimony removal circuit, similar to that practised historically at the Sunshine Mine in Idaho.

The pyrometallurgical option to reduce mercury is supported by testwork, and is included in this PEA's base case models. The hydrometallurgical options are still under development and require more work to allow inclusion in the PEA models. The opportunity to turn the current antimony penalty into a potential revenue stream has significant positive implications for the project, and will remain a focal point for further development of the project.

The industrial heritage of the Rožňava area means that project infrastructure is quite well developed: the plant is in an old industrial area, and will only require a process water pump station, some minor road work, and power and sewer tie-ins to municipal

services. The mine is already partially developed, and has existing surface and underground infrastructure in place.

The project as envisioned provides positive results at the base case metal prices of \$20/oz Ag and \$3.00/lb Cu. The pre-tax net present value (NPV) with a 5% discount rate is \$11 million using the base case metal prices. Post-tax, with the same 5% discount rate and metal prices, the NPV is \$5 million. Internal rates of return (IRR) are 12.42% pre-tax, and 8.3% post-tax. Payback on the project capital is expected to be 4.7 years pre-tax, and 5.1 years post-tax. Initial capital is estimated at \$25.3 million.

The results also show the Project's leverage to the higher metal prices, as indicated in the high case, with silver at \$22/oz and copper at \$3.30/lb. These higher metal prices increase the post-tax NPV at a 5% discount rate to \$19 million, with a 17.9% after-tax IRR, more than double the base case.

This assessment is preliminary in nature, as it includes Inferred Mineral Resources that are presently considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the assumptions used in this PEA will be realized.

1.2 Exploration and Mining History

Central European mining started in Celtic times, and at Rožňava the earliest record is from 1291. Mining at the Mária vein, 500 m southwest of the Strieborná vein, began at about this time, and continued on and off until the mid-1990s. Production was influenced by the availability of the latest mining and metallurgical technologies.

The Strieborná vein was discovered in 1981 during drifting from the Mária vein, with subsequent exploration largely by the Slovak Geological Survey (the "SGS"). The Strieborná vein was developed on four main levels, and exploration included underground channel sampling and drilling from both surface and underground. Historically, the commodities of interest at both the Mária vein and the Strieborná vein included iron, silver, copper, gold, antimony, mercury, and bismuth. Recovery of minerals has been by flotation, hand-sorting, and magnetic separation. Because of the complex metallurgy of the mineralization, not all metals were recoverable via any one metallurgical process.

Numerous non-National Instrument 43-101 (NI 43-101) compliant resource estimates (iron, copper, silver) on the Strieborná vein have been completed, including those in the 1990s by the SGS and in 1998 by Porcher Island Gold Corporation. In 1994, CMX Resource Ltd. (CMX), who then controlled the property, conducted underground panel sampling to verify the work done by the SGS, mapped 60% of the underground, and completed structural and mineralogical studies. They presented the results of their work to Sunshine Mining and Refining Company (Sunshine Mining) of Coeur d'Alene, Idaho, USA, who conducted metallurgical studies using its own proprietary hydrometallurgical

process, independent resource/reserve estimation, and a mining study. Sunshine Mining decided not to pursue the opportunity.

In the mid-1990s the Mária Mine, including the levels that accessed the Strieborná vein, was shut down and allowed to flood.

In 2007, the mining licences were transferred to Global Minerals Slovakia, and in 2011, Global Minerals began a dewatering and rehabilitation program. The mine is currently dewatered to just below Level 8. Between April 2011 and May 2013, two programmes of verification and expansion drilling were completed by Global Minerals, including 6,144 m in 40 drill holes from surface and underground platforms.

In 2008, Global Minerals published a NI 43-101 report that estimated resources of 1,950,000 tonnes containing 232 g/t Ag and 1.1% Cu in the Measured and Indicated categories, and 1,500,000 tonnes containing 280 g/t and 0.9% Cu in the Inferred category. An updated resource estimate is presented in this report.

1.3 Geology and Mineralization

The Strieborná Project is located in the Inner Western Carpathians, which form the eastward continuation of the Alps. Metasedimentary and metavolcanic rocks of the Paleozoic Gelnica Group, and epicontinental and marine sedimentary rocks of the Mesozoic Gočaltovo Group, underlie the Rožňava area.

The Drnava Formation, part of the Gelnica group, hosts the Strieborná vein and the adjacent Mária, Mayer, Pallag, and Podložná veins. This formation consists of coarse-grained metasedimentary rocks, laminated metapsammities and phyllites, and metamorphosed rocks resembling sheared porphyries (called porphyroids), but derived from psammitic sediments and/or pyroclastic rocks of acid composition.

The Strieborná vein and the adjacent Mária, Pallag, and Podložná veins are all tetrahedrite-bearing siderite plus quartz veins. Tetrahedrite is the principal sulphide mineral, with pyrite, arsenopyrite, and chalcopyrite occurring in subordinate quantities. Tetrahedrite carries the silver, copper, and antimony, and, reportedly, bismuth and mercury values. The chalcopyrite content seems to be higher in the southwestern portion of the vein and at depth, and the pyrite content increases with depth. The principal non-sulphide vein minerals are medium-to-coarse grained siderite and variable amounts of white quartz.

The Strieborná vein, which hosts the majority of the resources defined in the report, has a strike length of 1,300 m defined by drifts along the vein on four levels, and a vertical extent of 550 m defined by drifting and drilling. The average horizontal thickness of the Strieborná vein with grades greater than 100 g/t Ag is 2.23 m. The Strieborná vein reaches a maximum thickness of 13.6 m. The vein strikes northeast and has a propeller shape with subvertical dips in the centre, changing to southeastern dips in the

southwestern part of the vein, and to northwestern dips in the northeastern part of the vein.

The Podložná vein, which carries a small portion of the resource, is interpreted to have a strike length of about 980 m and a vertical extent of about 480 m, based on drifting, historical drilling, and recent drilling. The vein strikes 30° to 40°, and dips 80° to 85° southeast. The average horizontal thickness is 2.5 m, and the maximum width is 6.2 m.

1.4 Mineral Resource Estimate

MDA completed Global Minerals' third resource estimate in compliance with the disclosure and reporting requirements set forth in the Canadian Securities Administrators' NI 43-101, Companion Policy 43-101CP, and Form 43-101F1, as well as with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM)'s "CIM Definition Standards for Mineral Resources and Reserves, Definitions, and Guidelines" (CIM Standards) adopted by the CIM Council on May 10, 2014.

Silver, copper, antimony, mercury, and iron were estimated. This estimate includes the results from new underground channel and panel samples on mine Level 6 completed by Global Minerals since the previous estimate. This estimate also modelled and estimated the Podložná vein and a west split of the Strieborná vein, although these are small, representing less than four percent of the total resource.

The samples were taken from both core and underground. The core was drilled by the SGS and more recently by Global Minerals. The underground samples are almost entirely from the SGS, but since the last estimate, Global Minerals has also entered CMX underground samples. It is important to note that sampling at the Strieborná vein is unevenly distributed, with samples from underground levels and from the drilled area to the northeast. MDA chose 3.55 g/cm³ for the density to assign to the Strieborná vein, based primarily on density measurements made by Global Minerals. MDA assigned density values of 3.48 g/cm³ and 3.60 g/cm³ for the Strieborná west split and Podložná, respectively, based on a regression analysis comparing iron and density.

MDA composited the data to three-metre lengths; correlograms of the capped grades composited to three metres were made for each metal in multiple directions. MDA ran estimates using nearest neighbour, ordinary kriging, and inverse distance (ID) algorithms. The reported resource is the inverse distance to the fourth power (ID⁴) estimate. Restrictions on high-grade projections were done.

MDA classified the resources based on confidence and/or support for underlying information (i.e., QA/QC, original certificates), number of samples and/or drill holes, and distances to the nearest sample. Different criteria were used for each vein. In the Strieborná vein, Measured, Indicated, and Inferred categories were permissible. Podložná was assigned only Indicated and Inferred, as that vein has not yet been drilled

by Global Minerals. The west split only has Inferred, because its continuity is not well established, with only two drill hole intersections.

MDA reports resources at cut-off grades that are reasonable for deposits of this nature given anticipated mining methods and plant processing costs, while also considering economic conditions because of the regulatory requirement that a resource exist *“in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction.”*

A summary of the estimated resources at the reporting cut-off of 100 g/t Ag is presented in Table 1-1.

Overall, Global Minerals’ work completed to-date is very good. While their work only defines a small part of the deposit by drilling in the upper northeast portion of the vein, it confirms historical work, perceptions, and interpretations of the veins. MDA cannot state with first-hand experience that the historical work is of similar caliber, but during interpretation and evaluation of the data from both the SGS and CMX, MDA has been impressed with their work, documentation, and understanding.

Table 1-1: Strieborná Mineral Resource

Cut-off Ag (g/t)	Tonnes	Ag (g/t)	Cu (%)	Sb (%)	Fe (%)	Ag (oz)	Cu (t)	Sb (t)
Measured								
100.0	147,000	281	1.38	1.01	31.0	1,324,000	2,030	1,480
Indicated								
100.0	2,199,000	266	1.2	0.84	34.3	18,831,000	26,380	18,450
Measured and Indicated								
100.0	2,346,000	267	1.21	0.85	34.1	20,155,000	28,410	19,930
Inferred								
100.0	986,000	224	1.02	0.72	33.91	7,090,000	10,080	7,070

The restricted sample distribution defining the Strieborná mineralization made this a very complicated estimate, moderated only by the apparently well-behaved nature of the deposit. Tens of different estimates were completed attempting to impart grade distributions that were reasonable, as Global Minerals interprets them to be, while limiting estimation artifacts of the restricted sample distribution. Furthermore, as more work is done at Strieborná, more splays and structural complications will become evident, and high-grade shoots will be better defined.

There is potential mineralization beyond Inferred classification within the interpreted veins that is not reported. There is also very good potential to increase the resources by additional drilling on Strieborná and on Pallag, but particularly Podložná.

This assessment is preliminary in nature, as it includes Inferred Mineral Resources that are presently considered too speculative geologically to have economic considerations

applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the assumptions used in this PEA will be realized.

1.5 Geotechnical

AGP retained BGC Engineering Inc. (BGC) on behalf of Global Minerals to provide preliminary geotechnical evaluations to support the development of mining designs and cost estimates for the Strieborná Project.

Review of rock mechanics data, together with inspection of the accessible areas of the existing mine, allowed the development of initial design criteria for the sizing of stopes and pillars, together with recommended support assumptions. Ground conditions are expected to vary from fair to good.

The design criteria were provided for general guidance to the PEA-level mine design and for cost estimating purposes. As additional geological and geotechnical work is completed, the current designs will be reviewed and updated as required.

1.6 Underground Mining

The Strieborná vein will be accessed from the upper levels of the existing Mária Mine. A mechanized longhole retreat, open-stoping method, with truck haulage to surface, has been adopted in order to avoid the complicated, low productivity materials handling Mária system used historically. Planned rib and sill pillars comply with the geotechnical guidelines. Backfill will not be required.

Rehabilitation of existing workings has already been completed to Level 6. Further rehabilitation of the Mária subvertical shaft and mining levels below Level 6 will be required. The mine is currently dewatered to below Level 8, but additional pumping is necessary to lower the water level to Level 11. On Level 13 there is a connection between the Mária and Sadlovsky mines, consisting of a series of three watertight doors/seals, which was installed in the mid 1990s. These seals will remain submerged for the life of mine (LOM), but must remain intact to prevent inflow from the Sadlovsky Mine.

Although the Strieborná vein pinches and swells, the generally regular, steeply-dipping geometry of the vein, and average 4.9 m width, indicate that longhole stoping methods are appropriate. To minimize potential stope dilution resulting from longhole drilling, the vertical sublevel interval was limited to 17 m, and small 64 mm diameter longholes planned. AGP assumed a minimum planned stoping width of 1.5 m, adjusting the planned development and stope grades to include the non-vein dilution material.

The mine plans describe the potential to produce 983 kt of mill feed grading 307 g/t Ag and 1.34% Cu to the process plant over an operating period of six to seven years. Dilution in mill feed, defined as waste tonnes/mill feed tonnes, is estimated to be 13.5%.

99.6% of mill feed will be drawn from material classified as Measured and Indicated Resources.

The mine will produce 110 kt/a of mill feed operating on a system of three 8 h/d shifts, five days per week, until Q1 Year 3, when mine operations will be conducted on a seven day per week basis, and annual production will increase to 180 kt/a.

A new truck haulage ramp will be developed from the existing Meyer haulage tunnel to the Strieborná vein, and continued downdip to bisect the Strieborná vein, thereby providing multiple mining production faces and minimizing stope mucking distances. Fifteen tonne diesel trucks will be used in conjunction with single boom jumbos, 6.7-tonne load-haul-dump (LHD) trucks, simplified longhole drilling jumbos, and auxiliary service equipment. Smaller sized mining equipment will be used when mining narrower stopes. The mine mobile equipment fleet will be leased.

Mining operations will mainly be undertaken by owner-employed crews. The local workforce has little experience using modern mechanized mining equipment, and an effective training system will be required to increase skill levels.

Mine cost estimates are developed from first principles. The mining costs are estimated to include initial mine project capital of \$14.1 million during a 15-month period prior to the commissioning of the process plant. Thereafter, mining costs were estimated to comprise operating costs of \$59/t and sustaining capital of \$15.5 million.

This assessment is preliminary in nature, as it includes Inferred Mineral Resources that are presently considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the assumptions used in this PEA will be realized.

1.7 Metallurgy and Process

1.7.1 Metallurgical Testwork

There are two methods for metallurgical processing of mineralized rock from the Strieborná Project. In both cases, the material must first be crushed, ground to a fine pulp, and then subjected to a flotation process to remove the main gangue component (siderite). The flotation separation is very efficient, and produces a concentrate with a much smaller mass (approximately 5% of the feed) and higher copper, antimony, and silver grades. This flotation concentrate can then be subjected to either pyrometallurgical or hydrometallurgical processing for treatment of antimony and mercury prior to metal sales. For this PEA, the pyrometallurgical route has been selected. Development work is continuing on the hydrometallurgical route, as this has the potential to dramatically improve the Strieborná Project economics.

Most of the metallurgical testwork completed to date has been conducted on material from samples of coarse assay rejects that was sent to Blue Coast Research (BCR) in July

2011. The samples were crushed and blended to make a single master composite. The composites were designed to be representative of the average deposit grade at the time of blending. The blend contained 1.10% Cu, 1.10% S, 0.8 % Sb, and 270 g/t Ag.

Mineralogy showed that the composite contained approximately 73% siderite, 21% quartz, 3% muscovite, 1% pyrite, and 2% tetrahedrite.

A representative sample of this material was used for a rock hardness test (Bond Work index (BWi)). The sample showed the rock to be considerably softer than average ore bodies, with a BWi of 8.7 kWh/t.

This sample was then used for a series of rougher, cleaner, and locked-cycle flotation tests. Flotation results were very good; the locked-cycle test produced a concentrate grade of 5,900 g/t Ag and 23.2% Cu, with recoveries of 98.5% Ag, and 98.0% Cu. Work indicates that all silver and approximately 98% of the copper in the master composite is contained in tetrahedrite ((Cu, Ag)₁₂Sb₄S₁₃), therefore the antimony recovery was also 98.5%. In addition, mercury was also upgraded in the flotation concentrate to a grade of approximately 0.55%.

More recently, a second series of confirmatory flotation tests was run at BCR using material recovered from the Phase 1 mining area in June 2014. The composite sample used for this work graded 1.67% Cu, 400 g/t Ag, and 1.3% Sb.

Locked-cycle testing of this material was again very successful, with a slightly better result than previous work. A concentrate grade of 27% Cu, 6,500 g/t Ag, and 20.6% Sb was achieved, with recoveries of 98.2% Cu, 98.6% Ag, and 96.7% Sb. Mercury in this instance was almost 1%.

In all tests, the presence of significant antimony and mercury in the flotation concentrate mean that it cannot be processed by a smelter without incurring significant penalties. To achieve a more saleable concentrate, two downstream processes (pyrometallurgical and/or hydrometallurgical) are currently being investigated in order to remove the problematic elements.

1.7.2 Pyrometallurgy

Roasting tests have shown that mercury can be effectively vapourized from the flotation concentrate. A flotation concentrate similar to that mentioned above was roasted under various conditions. By roasting at 650°C for 170 minutes in a nitrogen gas environment, removal of 99.6% of the mercury was achievable, giving a grade in the calcine product of 30 ppm. Although antimony removal was not the key target during this testwork removal of up to 37.4% was achieved at the far higher temperature of 950°C.

It is anticipated that the calcine product could be sent to a smelter, where any remaining deleterious elements would be dealt with as penalties.

1.7.3 Hydrometallurgy

As an alternative to the above pyrometallurgical process, an alkali leach was tested. This alkali leach was used primarily by the Sunshine Mine in northern Idaho to treat tetrahydrite concentrates. In this process, the antimony was electrowon and sold as a metal byproduct.

In preliminary hydrometallurgical testing of a Strieborná flotation concentrate sample, both antimony and mercury were leached using a solution of sodium hydroxide and sodium sulphide. The best results were achieved at 95°C with 200 g/L sodium sulphide and 18 g/L sodium hydroxide; the solution was stirred for 24 hours at 500 rpm. The results from these tests gave 97% extraction of antimony and 92% extraction of mercury, with some gold and no silver losses. It is anticipated that the leach residue could be shipped to European smelters with greatly reduced penalties. The antimony in solution can be electrowon to produce saleable metal products. The mercury can be removed from solution using various methods to produce a stable product for disposal.

1.7.4 Recovery Methods

A preliminary process plant design has been completed consisting of a conventional 500 t/d crush, grind, flotation plant to produce a tetrahydrite concentrate. During the first two years of production the plant would operate at a reduced tonnage of 300 t/d, ramping up to the design throughput after that period. The flotation concentrate would be shipped off site for mercury removal by calcining, before being sold to a smelter.

1.8 Infrastructure and Site Layout

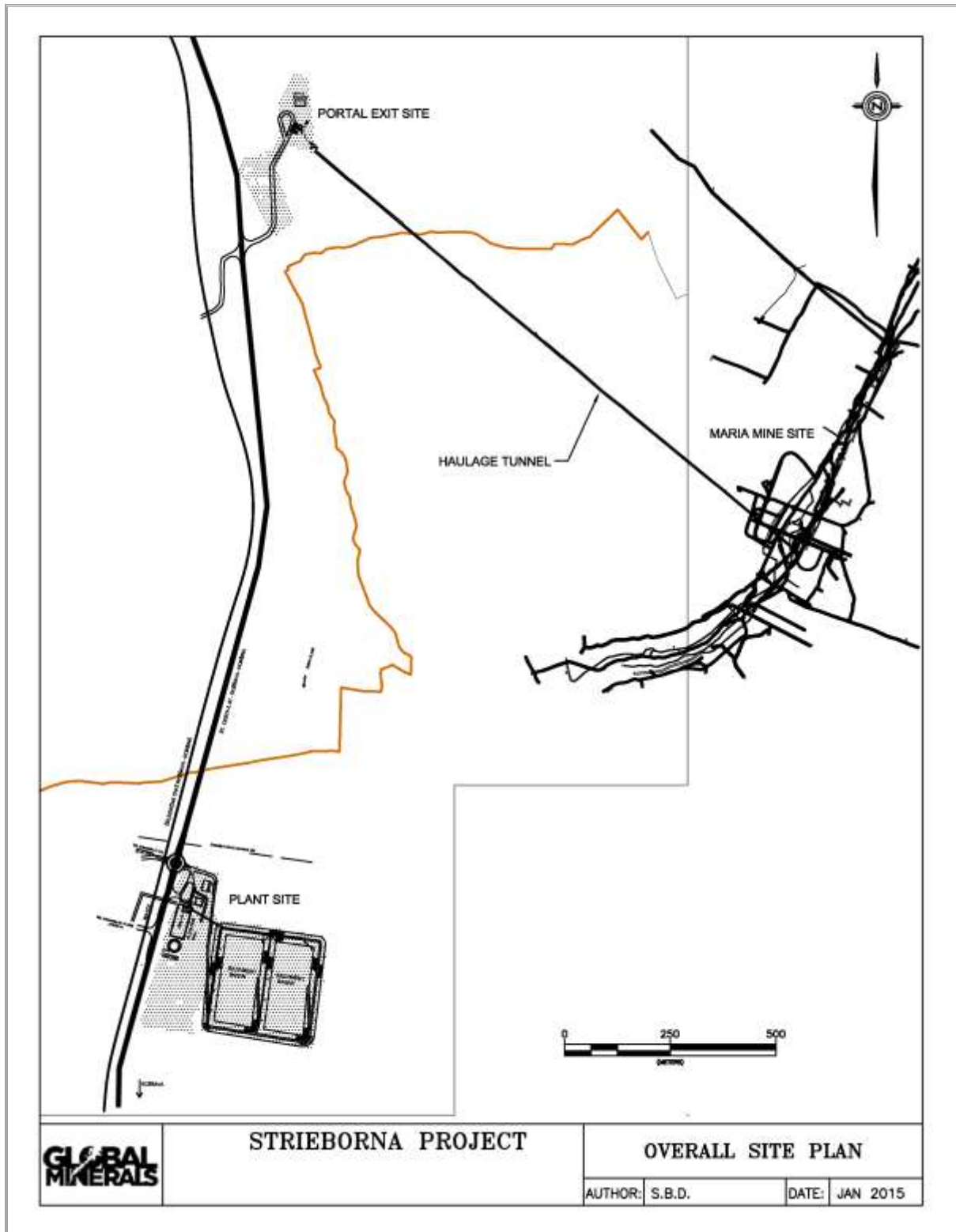
The project site comprises two distinct areas; the Mária Mine, site and the process plant facility and TSF. The two sites are interconnected via a public highway.

The existing Mária Mine site will be utilized as the main access to the Strieborná vein, with the haulage tunnel used to withdraw mineralized rock. The Mária site consists of the portal entrance, shaft, and mine facilities. Figure 1-2 shows the overall site plan.

Run-of-mine (ROM) material will be tipped into trucks and transported by road to the process plant, which will be located 2 km south of the haulage tunnel exit portal.

The primary crusher would be located at the process plant, together with administrative offices, maintenance shops, warehouse, and dry facilities. The TSF is adjacent to the process plant and is an existing facility. Plant tailings cake will be transported by truck across the highway to the TSF.

Figure 1-2: Overall Site Plan



Process water will be recycled as much as possible within the process plant, and fresh make-up water will be withdrawn from the Slaná River, approximately 400 m to the east.

Potable water and sewage will tie in to existing municipal services.

1.9 Capital and Operating Costs

1.9.1 Capital Costs

The capital costs for the Strieborná Project are summarized in Table 1-2. The costs are based on a seven-year project life, with throughput starting at 300 t/d, and expanding to 500 t/d in Year 3.

Table 1-2: Strieborná Project Capital Cost Summary

Capital Category	Total Capital (\$M)	Preproduction Year -3 to Year -1 (\$M)	Production Year 1 (\$M)	Production Year 2+ (\$M)
Underground Mining	29.5	10.6	10.7	8.2
Processing	10.2	5.8	3.9	0.5
Infrastructure	3.7	3.1	0.5	0.1
Environmental	0.2	0.0	0.0	0.2
Indirect Costs	4.3	3.2	0.9	0.2
Contingency	5.8	2.6	2.0	1.2
Total	53.6	25.3	18.0	10.3

Initial direct and indirect capital requirements (preproduction) are estimated at \$25.3 million. Production starts in Year 1, with additional capital requirements of \$18.0 million, with the majority being a result of continued underground development. The capital requirement in Year 1 is offset by revenue generated in the same year. From Year 2 onward, a further \$10.3 million is required, again offset by production revenues.

Contingency allowances are calculated from percentages shown in Table 1-3.

Table 1-3: Contingency Factors by Category

Capital Category	Contingency Allowance (%)
Underground Mining	12.0
Processing	16.1
Infrastructure	15.0
Environmental	10.0

Life of Mine (LOM) capital requirements are estimated to be \$53.6 million.

1.9.2 Operating Costs

Operating costs have been estimated for a 500 t/d mining and milling operation, with financial models also taking into account the fact that throughputs of 300 t/d are planned for the initial two years.

Total operating cost is \$92/t processed over a seven-year mine life.

Project operating cost estimates are summarized in Table 1-4 on a cost per tonne mined/milled basis during Years 1–2 (300 t/d processing rate) and Years 3–7 (500 t/d processing rate).

G&A costs are based on an average of 41 people, with 13 salaried staff, and 28 hourly employees.

Table 1-4: Operating Cost Summary for the Strieborná Project

Cost Area	Cost (\$/t)		
	300 t/d	500 t/d	LOM
Underground Mining	52.07	61.32	59.37
Processing	19.73	16.80	17.58
ROM and Tailings Haulage	1.25	1.25	1.25
G&A	18.64	11.18	13.82
Total	100.94	81.30	92.02

Breaking this down further, the underground operating costs (LOM average) are listed in Table 1-5.

Table 1-5: LOM Underground Operating Costs (\$/t)

Activity	Cost (\$/t)
Development and Mucking	4.54
Production Drilling and Blasting	4.80
Stope Mucking	2.87
Trucking	17.32
Mine Services	11.79
Owners Supervision	7.67
Power	10.39
Contingency	-
Total	59.37

Process plant operating costs are summarized in Table 1-6, both for the initial 300 t/d processing rate, and the subsequent 500 t/d rate.

Table 1-6: Operating Cost Estimate for the Strieborná Processing Plant

Cost Area	Cost (\$/t)	
	300 t/d	500 t/d
Labour – Mill	10.19	7.69
Power	4.17	4.17
Plant Maintenance Spares	1.69	1.28
Reagents	1.28	1.28
Mill Balls	0.77	0.77
Liners (Crusher + Mills)	0.52	0.52
Assay Laboratory	0.46	0.55
Safety Equipment	0.31	0.24
E&I Maintenance	0.20	0.15
Piping	0.08	0.08
Lubricants	0.06	0.06
Water	0.01	0.01
Subtotal	19.73	16.80

1.10 Economic Analysis

The results of the analysis of the Strieborná Project are encouraging. The project as envisioned provides positive results at the base case metal prices of \$20/oz Ag and \$3.00/lb Cu. The pre-tax NPV with a 5% discount rate is \$11 million using the base case metal prices. Post-tax, with the same 5% discount rate and metal prices, the NPV is \$5 million. IRR are 12.42% pre-tax, and 8.3% post-tax. Payback on the project capital is expected to be 4.7 years pre-tax and 5.1 years post-tax.

Detailed results of the analysis of the Strieborná Project are shown in Table 1-7.

Table 1-7: Discounted Cash Flow Results

Cost Category	Unit	Low Case	Base Case	High Case
Silver Price	\$/oz	18	20	22
Copper Price	\$/lb	2.70	3.00	3.30
Operating Costs				
Underground Mining	\$M	58.4	58.4	58.4
Processing, Tailings, and ROM Trucking	\$M	18.5	18.5	18.5
G&A	\$M	13.6	13.6	13.6
Subtotal On-site Costs	\$M	90.5	90.5	90.5
Transportation, Port Costs, Shipping	\$M	5.0	5.0	5.0
Total Operating Costs	\$M	95.5	95.5	95.5
Capital Costs				
Underground Mining	\$M	29.5	29.5	29.5
Processing	\$M	10.2	10.2	10.2
Infrastructure	\$M	3.7	3.7	3.7
Environmental Costs	\$M	0.2	0.2	0.2

Cost Category	Unit	Low Case	Base Case	High Case
Owners and Indirect Costs	\$M	4.3	4.3	4.3
Contingency	\$M	5.7	5.7	5.7
Total Capital Costs	\$M	53.6	53.6	53.6
Copper Revenue	\$M	68.7	74.9	81.0
- Less fuming, smelting, refining, penalties	\$M	76.6	76.6	76.6
Net Copper Revenue	\$M	-7.8	-1.7	4.4
Silver Revenue	\$M	161.6	179.6	197.5
- Less refining	\$M	5.4	5.4	5.4
Net Silver Revenue	\$M	156.2	174.2	192.1
Net Copper and Silver Revenue	\$M	148.4	172.5	196.5
Pre-Tax Cash Flow	\$M	-0.7	23.3	47.4
Total Tax	\$M	3.2	8.4	13.6
Post-Tax Cash Flow	\$M	-3.9	14.9	33.8
Pre-Tax NPV @ 5%	\$M	-7	11	29
Pre-Tax IRR	%	-0.4	12.4	23.6
Pre-Tax Payback Period	years	-	4.7	3.7
Post-Tax NPV @ 5%	\$M	-10	5	19
Post-Tax IRR	%	-2.3	8.3	17.9
Post-Tax Payback Period	years	-	5.1	4.0

The results also show the Project's leverage to the higher metal prices, as indicated in the high case, with silver at \$22/oz and copper at \$3.30/lb. These higher metal prices increase the post-tax NPV at a 5% discount rate to \$19 million, with a 17.9% after-tax IRR, more than double the base case.

Post-tax NPV sensitivities are shown in Table 1-8.

Table 1-8: NPV Sensitivity (\$ millions) – Post-Tax

Sensitivity Item	-20 %	-10 %	Base	10 %	20 %
Silver Grade	-18	-6	5	15	25
Silver Price	-19	-6	5	15	24
Total Capital Cost	12	8	5	1	-2
Total Operating Cost	16	10	5	-1	-7
Mercury Treatment Cost	7	6	5	3	2
Copper Treatment Cost	6	5	5	4	4
Antimony Concentrate Grade	9	7	5	2	0

This assessment is preliminary in nature, as it includes Inferred Mineral Resources that are presently considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the assumptions used in this PEA will be realized.

Copper and silver recoveries of 98% were applied to all tonnes milled.

Costs for the Strieborná Project are shown in terms of dollars per tonne (\$/t) mill feed, and dollars per dry metric tonne (\$/dmt) of silver-copper concentrate (Ag-Cu Conc.) in Table 1-9.

Table 1-9: Base Case Operating Cost Estimates

Operating Cost Category	Mill Feed (\$/t)	Ag-Cu Conc. (\$/dmt)
Mining (mill feed and waste)	59.37	1,436.67
Processing, Tailings, and ROM Trucking	18.83	455.73
G&A	13.82	334.49
Subtotal On-site Costs	92.02	2,226.89
Transportation, Port Costs, Shipping	5.08	122.95
Total Operating Costs	97.10	2,349.84

1.11 Environmental Studies, Permitting, and Social Impact

The renewal of underground mining at the Strieborná Project using modern technology is expected to bring much needed employment, income, taxation, environmental protection, and social benefits to the local community and to Slovakia in general. Implementation of modern water treatment technology to dewater the flooded underground workings, treat, and discharge the water has been accomplished within the permitting framework of the Slovakian government, and is compliant with EU standards. Global Minerals operates as a valued and participatory corporate citizen in its operations. The physical, chemical, biological, social, and regulatory aspects of the project are being considered under the various applicable EU environmental policies and related Framework documents.

A detailed environmental impact assessment (EIA) process is required by the Slovakian government, and is under preparation at time of writing this report. Potential negative social and environmental impacts will be identified, and appropriate steps taken to eliminate, minimize, mitigate, or ameliorate these impacts. Appropriate controls for treatment of mine dewatering have already been put in place, and are permitted, monitored, and inspected. Detailed operational planning and engineering for air quality protection, erosion and sediment control, surface water and groundwater quality, waste management, spill prevention control and countermeasures, tailings management, emergency response, community relations, worker health and safety, security, traffic, noise, dust, vibration, and other aspects will be incorporated into the overall social and environmental management and monitoring program (SEMMP) to be specifically applied to the Strieborná Project. A comprehensive bankable feasibility study to international standards is anticipated. It is expected that the project will be operated in a safe and environmentally and socially responsible manner within a modern sustainability framework using advanced approaches, technology, protocols, and management.

The project has demonstrated broad community support with a continuing stakeholder engagement program with wide reach, including governmental, non-governmental, social, and economic organizations throughout the local communities.

1.12 Recommendations and Proposed Budget

1.12.1 Geology

Establishment of two new drill stations at 100 m to 200 m to the southwest of the existing drill stations on Level 6 would be advantageous.

Also, MDA recommends that Global Minerals develop and explore along the Strieborná vein from Level 6.

The cost estimate for an additional four holes of 500 m to intersect down-dip or strike extensions is \$300,000. Continuing the infill drilling will require two new drill stations to be located 100 m to 200 m southwest of the existing drill stations on Level 6. The estimated cost for the drifting (\$250,000) and drilling (\$450,000) is \$700,000.

1.12.2 Geotechnical

The interpretations presented within this report require additional validation and testing with higher quality data before they can be applied to more intensive design studies. Recommended data collection and interpretation tasks are outlined below:

- The completion of core orientation and down hole geophysics would significantly improve the structural interpretations for the project area. A complete geological model and structural model should be considered necessary components at the next level of study.
- Geotechnical core holes should be drilled to target hanging wall, footwall, and mineralized rock masses. Core orientation techniques should be employed to determine geologic discontinuity orientations. Uniaxial compressive strength testing of samples should be undertaken. Borehole televiwer surveys can supplement the limited information available on the in situ stresses. A detailed review of the alteration types should be completed prior to initiation of the geotechnical core logging program.
- The potential for further division of the geotechnical units according to alteration may be evaluated through a point load-testing program.
- Hydrogeological testing and instrumentation should be installed in selected drill holes to provide data for groundwater modelling and excavation dewatering simulations.
- The stope dimensions in the Mária Mine appear to be greater than the recommendations derived for the Strieborná vein. Additional study is recommended to investigate the potential reasons for this discrepancy.

1.12.3 Mining

AGP recommends the following items for future mining studies:

- assess the potential to adapt stope designs based on updated geotechnical data with the aim of increasing resource recovery
- undertake studies that will allow definition of the mine ventilation
- better identify the dewatering requirements and methodology for the currently flooded areas of the existing Mária Mine workings
- seek new quotations for rehabilitation of the existing workings and shafts in line with the requirements and schedule outlined in this report
- attempt to quantify the water contained in the adjacent Sadlovsky Mine and mitigate the potential effects of water incursion from this property
- as the design of mine systems and infrastructure requirements are more clearly identified, quotations should be sought to confirm the adequacy of allowances included in this study
- confirm the suitability of equipment selection and costs used in this study, in order that quoted cost rates can be used in ongoing cost estimation
- investigate and identify adequate training systems to provide the necessary skill levels required for mechanized mining
- assess the potential to increase shift length on a two shifts per day basis.

1.12.4 Metallurgy

Additional metallurgical characterization of the Strieborná mineralization is required. Development in the following areas is recommended:

- pilot plant to create sufficient concentrates for pyrometallurgy and hydrometallurgy programs
- the hydrometallurgical and pyrometallurgical process options have the potential for significant improvement in project economics; these should continue to be developed at laboratory scale
- further large scale testing of mercury roasting processes should be scheduled
- optimization of flotation conditions (regrind).

Further metallurgical programs will allow for a more thorough understanding of product specifications. In particular, the removal of mercury and antimony from flotation concentrates using a combination of roasting and/or alkali leaching is deemed necessary to extract maximum value from the contained metals. AGP recommends further pyrometallurgical and hydrometallurgical testing.

1.12.5 Infrastructure

Additional testing and data collection is required to define the infrastructure and site layout requirements and associated costs in these areas:

- geotechnical testing and data to further develop construction requirements for the mill, water supply, and services sites
- hydrogeology and water quality testing on water sources surrounding the mine site to confirm available volumes and quality of water required to support the mill and services infrastructure
- materials testing and quarry location scoping for design and construction cost estimates
- foundation testing and material suitability studies need to be completed for the plant site and water retention pond locations
- archeological studies
- verification that existing municipal sewer and water lines have sufficient capacity for the project.

1.12.6 Water Management

As the mine dewatering is currently ongoing, a hydrogeological review of water in the mining area should be considered. This will entail determining if supply is sufficient in both quantity and quality for operation of the process plant and the mine. In addition, the precipitation and drainage areas need to be determined for proper estimating of diversion dam/ditches to minimize contact of freshwater with mining areas.

The TSF and waste rock management facility areas need to have complete hydrological evaluations completed for surface runoff, groundwater, and seepage.

1.12.7 Environmental

Global Minerals should continue along the path it has already developed with regards to collecting and analyzing information required for the submission of an ESIA.

Basic data collection needs to continue covering a wide range of diverse subjects, including weather, water flows, vegetation, wildlife, and socioeconomic information.

A detailed feasibility study will also be required to build upon the other information collected. The EIA and the feasibility study will require advancing forward in parallel with the technical experts providing input into the ESIA submission and subsequent permit application process. Detailed environmental and engineering information must be collected in at least the following areas:

- air quality
- water (surface and ground)
- cultural resources
- social and economic values

- meteorology
- geology and geochemistry
- vegetation
- soils
- visual resources
- wastes (solid and hazardous)

1.12.8 Markets and Smelters

Given the financial penalties for antimony in the concentrate are very high, any improvement in the quotes from smelters who have expressed an interest in Streiborna concentrate could have a positive impact on the project's economics.

Global has approximately 600 tonnes of mineralized material stockpiled at the Maria mine site remaining from the 2014 bulk sample exercise. A portion of this should be crushed and processed by pilot plant to make a flotation concentrate. This concentrate sample should be used to conduct large scale mercury and antimony roasting tests and the subsequent calcines should be sent to smelters for internal laboratory testing to establish commercial-level treatment terms.

The budgets given in Section 26.10 for testwork to support project development include the cost of pilot processing. A \$200,000 pilot run can generate 1,000 kg flotation concentrate for downstream testing

Of note, if the hydrometallurgical testwork program is able to develop further inroads into the antimony recovery process, then this metal could switch from being a penalty element to become a revenue stream. Development of this process option is recommended.

1.12.9 Recommendations Estimated Budget

Table 1-10 shows a summary of the total cost estimates for recommended work by discipline.

Table 1-10: Summary of Recommendations and Total Costs

	Recommended Budget (\$)
Resources – Exploration and Development	1,000,000
Geotechnical	700,000
Mining	600,000
Metallurgy	500,000
Infrastructure	200,000
Water Management	100,000
Environmental	200,000
Total	3,300,000

2 INTRODUCTION AND TERMS OF REFERENCE

2.1 General

This report was prepared at the request of Mr. Tim McCutcheon, Chief Executive Officer for Global Minerals Ltd. (“Global” or the “Company”). Global is an exploration and development company with its key asset, the 100%-owned Strieborná silver-copper-antimony project, located near the mining town of Rožňava in the Slovak Republic (Slovakia).

Global retained AGP to lead a consortium of consulting firms in the preparation of an updated technical report and preliminary economic assessment on their Strieborná property located near Rožňava, Slovakia. This technical report includes the results of the updated NI 43-101-compliant mineral resource estimate for the Strieborná property and complies with standards set out in NI 43-101, Standards and Disclosure for Mineral Projects, and pursuant to Form 43-101F.

2.2 Terms of Reference

Much of the text in this report was sourced from the last NI 43-101 compliant technical report titled “Resource Update and NI 43-101 Technical Report for the Strieborná Project, Eastern Slovakia” dated November 4, 2013.

Information, conclusions, and recommendations contained herein, are based on:

- two field examinations by MDA, including a study of relevant and available technical data
- discussions with Mr. William Bond and Mr. Martin Zahorec, both employees of Global Minerals
- the results of metallurgical testwork completed recently at Blue Coast Research, Kingston Process Metallurgy, and the University of British Columbia
- all other referenced documents are listed in Section 27.

All units used in this report are metric unless otherwise stated.

All monetary amounts are provided in US Dollars unless otherwise noted.

All Illustrations are embedded within the body of the report.

2.3 Qualified Persons

The Qualified Persons (QPs) responsible for the preparation of this Report, as defined in NI 43-101, are the following:

Andrew Holloway, CEng., P.Eng., Principal Process Engineer (AGP). Mr. Holloway is the Qualified Person under NI 43-101 responsible for this report. Mr. Holloway is experienced with metallurgical operations, process engineering, and project management.

Gordon Zurowski, P.Eng., Principal Mining Engineer (AGP) Mr. Zurowski is a professional Mining Engineer and is responsible for the financial evaluations in this report.

Steven Ristorcelli, C.P.G., Principal Resource Geologist (MDA) Mr. Ristorcelli is a Certified Professional Geologist. Mr. Ristorcelli made two site visits to the Strieborná Project, one from November 1 through November 3, 2012, and the other from May 6 through May 10, 2013. During the visits, he reviewed the exploration program, exploration data and drilling/logging/sampling procedures and took independent samples to verify mineralization. Mr. Ristorcelli also toured the accessible underground workings, viewed the underground drilling, examined existing core, and checked the surface locations of six drill holes on three drill sites. Project information, data, and electronic files were collected and reviewed jointly with Global Minerals. Quality Assurance/Quality Control (QA/QC) data and procedures were reviewed.

Geoffrey Challiner, CEng., Chief Underground Mining Engineer (AGP) Mr. Challiner is a professional Mining Engineer. Mr. Challiner visited the Strieborná project between 11 and 14 March 2013. During the visit, he toured the accessible underground workings and consulted with senior Global technical employees familiar with the project. Project information, data, and electronic files were collected and reviewed.

Chris Martin, M.Eng, C.Eng., Principal Metallurgist with Blue Coast Metallurgy Ltd., and Phillip Mackey, Ph.D., P.Eng., President, P.J. Mackey Technology Inc., are the Qualified Persons responsible for the metallurgical sections of this report.

Mr. Martin has extensive knowledge of metallurgical testwork program and process flow sheet development, including grinding, froth flotation, cyanidation, and mineralogical characterization.

Mr. Mackey has extensive knowledge of the pyrometallurgical testwork program and flow sheet development, including high temperature roasting, smelting, and refining of non-ferrous ores and concentrates. He has worked extensively and published a number of technical papers on the behaviour of elements such as lead, arsenic, antimony, bismuth, and mercury in roasting and smelting systems, as well as knowledge of the behaviour of gold and silver in these systems.

Lyn Jones, P.Eng., Senior Process Engineer (AGP) is the Qualified Person under NI 43-101 responsible for the Recovery Methods section of this report. Mr. Jones is experienced with metallurgical testwork program and process flow sheet development, including grinding, froth flotation, cyanidation, pressure oxidation, electrowinning, and mineralogical characterization.

Mario Colantonio, P.Eng., Engineering Manager (PES) is the Qualified Person under NI 43-101 responsible for the Infrastructure section of this report. Mr. Colantonio is an experienced civil engineer with experience in the development of new sites and expansion of existing facilities.

John G. Aronson, President (AATA). The environmental section of this report was written by John G. Aronson, President, AATA. Mr. Aronson has familiarized himself with all of the existing project documentation related to the social and environmental aspects of the Strieborná Project and its environs, but has not visited the site.

2.4 Site Visits and Responsibilities

AGP, MDA, and PES have conducted site visits to the project as shown in Table 2-1.

Table 2-1: Date of Site Visits and Areas of Responsibility

QP Name	Site Visit Dates	Area (Section) of Responsibility
Steven Ristorcelli	Nov 1 to 3 2012 May 6 to 10 2013	1.3, 1.4, 1.12.1, 7 to 12, 14, 25.1, and 26.2
Andrew Holloway	No Site Visit	1.1, 1.2, 1.9.1, 1.9.2, 1.12.6, 1.12.8, 2, 3, 4, 5, 6, 15, 19, 21.1.1, 21.1.5-7, 21.2.1, 21.2.4, 23, 24, 25.7, 25.9.25.10, 26.1, 26.7, 26.9, 26.10, and 27
Geoff Challiner	Mar 11 to 14, 2013	1.5, 1.6, 1.12.2, 1.12.3, 16, 21.1.2, 21.2.2, 25.2, 25.3, 26.3, and 26.4
Gordon Zurowski	No Site Visit	1.10, 22, and 25.8
Mario Colantonio	Aug 2 to 3, 2014	1.8, 1.12.5, 18, 21.1.4, 25.5, and 26.6
Lyn Jones	Apr 12 to 13, 2014	1.7.4, 17, 21.1.3, 21.2.3, 25.4.4, and 26.5.2
Chris Martin	No Site Visit	1.7.1, 1.7.3, 1.12.4, 13 (except 13.3.5), 25.4 (except 25.4.4), and 26.5.1
Philip Mackey	No Site Visit	1.7.2 and 13.3.5
John Aronson	No Site Visit	1.11, 1.12.7, 20, 25.6, and 26.8

Ms. Catherine Schmid, Senior Geotechnical Engineer of BGC, visited the property from March 11 to 13, 2013. Ms. Schmid contributed to the geotechnical components of the study. Mr. Roland Tosney, Senior Geotechnical Engineer of BGC also contributed to the analysis of the geotechnical portion of the report providing guidance to the underground design team.

In addition, **William Bond, C.P.G., M.Sc., Geol. and VP Exploration for Global Minerals** contributed much of the information given in Section 4 (Property Description and Location), Section 5 (Accessibility, Climate, Local Resources, Infrastructure, and

Physiography) and Section 6. Mr. Bond has been responsible for the Strieborná exploration program since 2010. Much of the fieldwork discussed in this report was conducted under his supervision. As an Officer of Global Minerals, Mr. Bond is NOT a Qualified Person under NI 43-101.

During the various site visits, items such as logistics/transportation, mining site, infrastructure, geology, and geotechnical assessment of the drill core, and mine staffing issues were assessed. This satisfies the condition of a site visit performed by independent qualified personnel for NI 43-101 regulations.

2.5 Effective Dates

The effective date of the PEA for the Strieborná Project is December 12, 2014

2.6 Previous Technical Reports

Two previous NI 43-101 compliant technical reports have been completed for the Strieborná Project:

1. Strieborná Vein Silver Project, Rožňava Mining District, Slovak Republic Latitude 48°40'29", Longitude 48°40'29" authored by Eur Ing Zygmunt Jakubiak of AMC Consultants Pty Ltd., dated April 22, 2008.
2. Resource Update and NI 43-101 Technical Report For the Strieborná Project, Eastern Slovakia authored by AGP Mining Consultants Inc., November 1, 2102.

3 RELIANCE ON OTHER EXPERTS

AGP has followed standard professional procedures in preparing the content of this Technical Report. Data used in the report has been verified wherever possible, and the report is based upon information believed to be accurate at the time of completion.

AGP has not verified the legal status or legal title to any permit, or to the legality of any underlying agreements for the subject properties regarding mineral rights, surface rights, permitting, and environmental issues in sections of this technical report.

In preparing this report, AGP has relied on several sources of information on the property, those based on maps, sections, and literature that were provided by Global Minerals. In particular, AGP has relied on the extensive knowledge and information provided by Global Minerals (W. Bond, W. Pincus, and M. Zahorec) in order to prepare Sections 4, 5, and 6 of this technical report.

Taxation, as outlined in Section 22, was completed based on guidance from Global Minerals and its in-country team. AGP has relied on their information.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Description

The principal assets of the Strieborná property encompass two mining licences known as Rožňava I and Rožňava III, located within the Rožňava mining cadastre area with ID No. 853011, Rožňava District (808), Kosice Region (008). The location of these licences is shown in Figure 4-1. Vertical planes projected down from the surface boundaries define the spatial boundaries.

The licences allow the holder to mine polymetallic ores including copper and silver. There is no time limit on the licences provided the holder complies with Slovakia mining law and annual filing/payment requirements. Slovak mining law requires mining to begin within five years from the date a mining licence is granted and in the case of Global Minerals, the licence was granted October 5, 2010.

The mining licence Rožňava I covers' extensive underground workings on the Mária vein, and exploration drifts on a series of smaller subsidiary veins known as Mária I, Mária II and Podložná. Mining licence Rožňava III covers the Strieborná Pallag veins.

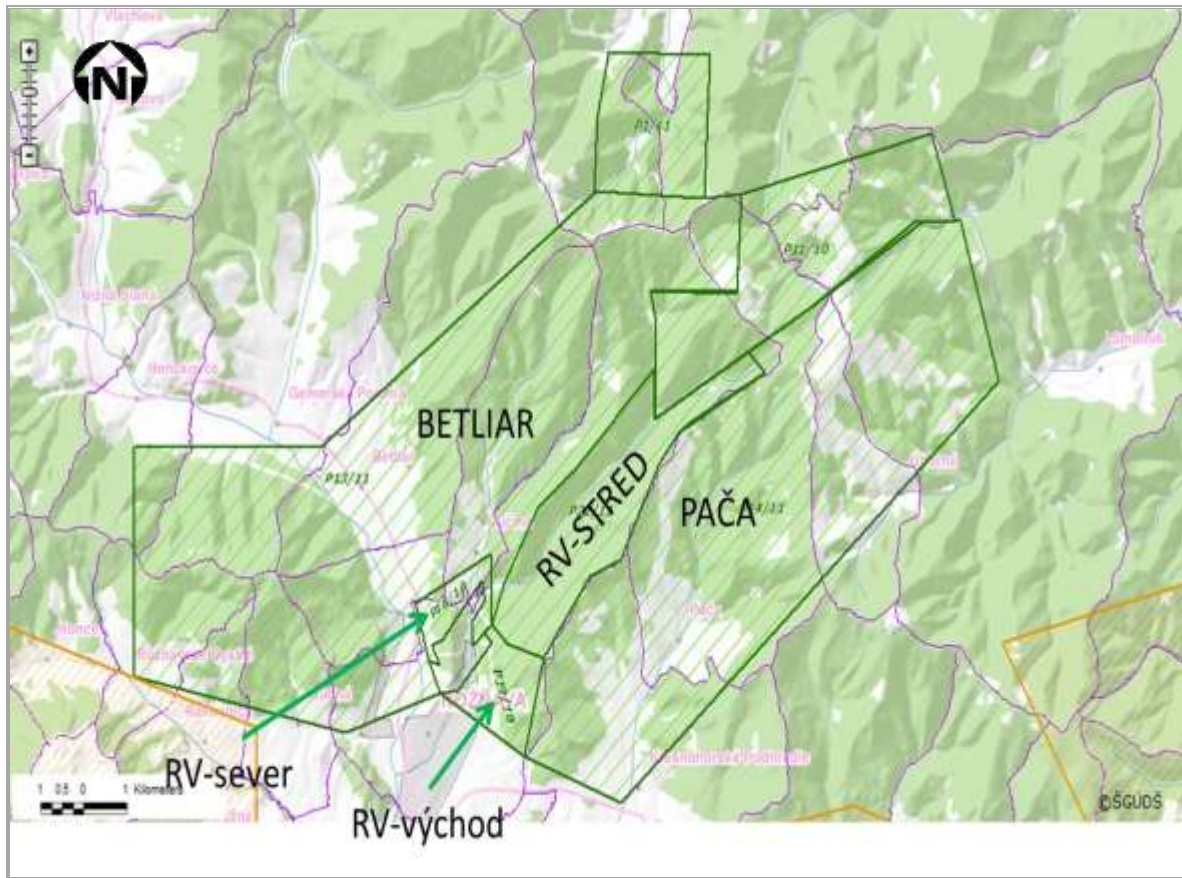
In addition to the mining licences and through the Company's Slovakian subsidiary, Global Minerals has rights to five exploration concessions (Table 4-1) that cover an area surrounding the mining concessions as shown in Figure 4-2. Exploration concessions are valid for four years and have certain fees and spending requirements associated with them. Global Minerals is current with all fees and has met the spending requirements. Exploration concessions can be renewed for two additional four-year periods.

Table 4-1: Exploration Concessions

Name	Date Granted	Expiry Date	Spending Commitment (€)
Betliar	7/26/2011	7/26/2015	169,944.70
Pača	7/26/2011	7/26/2015	280,000.00
Rožňava – východ	9/6/2010	10/5/2018	112,000.00
Rožňava – sever	9/6/2010	10/5/2018	86,800.00
Rožňava – stred	8/25/2011	8/25/2015	106,330.00

[illegible]

Figure 4-2: Exploration Concessions



4.2 Project Ownership

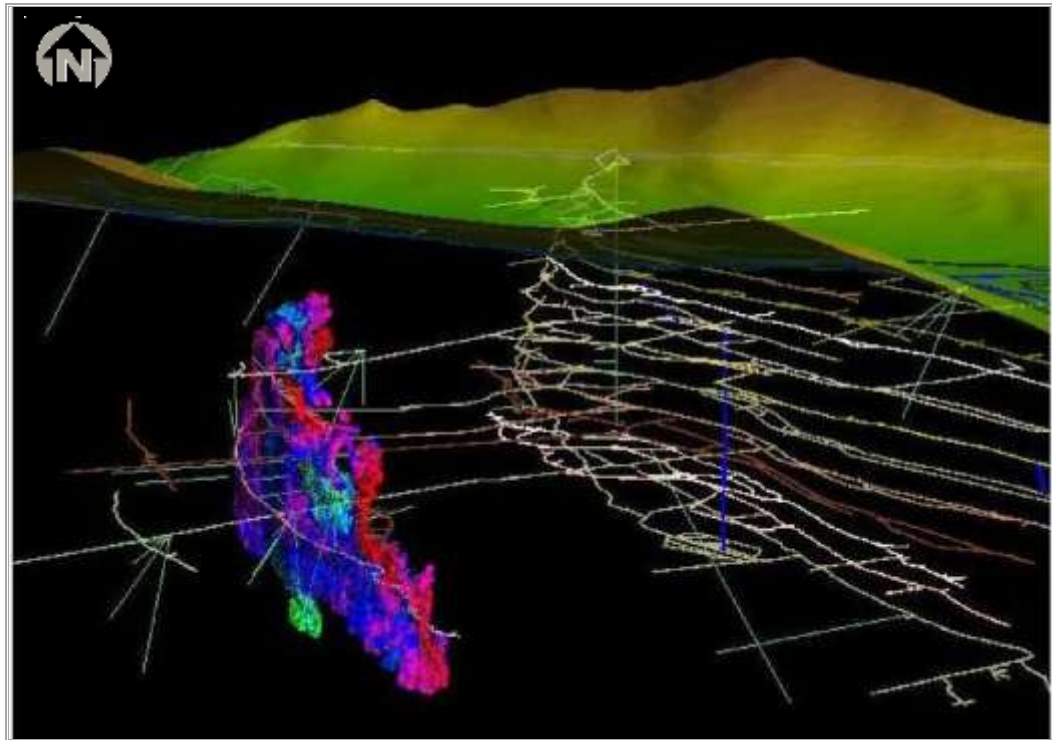
Mining licences Rožňava I and Rožňava III are held by Gemer-Can, s.r.o. and were acquired through a transfer from Global Minerals Slovakia, s.r.o. in October 2010. The Company owns 100% of Gemer-Can s.r.o. and the exploration concessions are owned by the Company's 100% subsidiary, Gemma s.r.o.

4.3 Description of Existing Underground Workings

A 3D diagram in Figure 4-3 shows the location of the Strieborná vein (purple-red on the left hand side of the diagram) in relation to the underground workings of the Mária Mine. The Strieborná vein is accessed underground through the Mária Mine workings. These were developed on thirteen levels to exploit the Mária vein and extend from the main surface portal at +360 m above Adriatic Datum down Level 13 at -170 m below Adriatic Datum.

The two parallel veins, Mária and Strieborná, are approximately 500 m apart and are connected by southeast trending crosscuts at Level 13 (-170 m), Level 10 (-20 m), Level 9 (+30 m), Level 8 (+80 m) and Level 6 (+180 m).

Figure 4-3: Underground Workings at Strieborná



In the Strieborná vein, there are drives on Levels 13, 10, 9, and 8 trending northeast and southwest from the main entry crosscuts. The longest drives are those on Level 13 and Level 10, which extend for approximately 1,300 m and 1,350 m, respectively. The drives on Levels 9 and 8 are relatively short, several metres on Level 9 and approximately 350 m on Level 8.

On Level 13, the southwest drive is in the vein so there are no sampling crosscuts, but there are 40 short sampling crosscuts off the northeast drive. On Level 10, there are 37 short sampling crosscuts off the southwest drive and 21 crosscuts off the northeast drive. On Level 8, there are eight sampling crosscuts off the southwest drive and five crosscuts off the northeast drive.

On Levels 10, 9, and 8 there are also sublevel drives located 8 m above the levels and connected to the level by short raises (ore passes). The sublevel drive above Level 10 is approximately 110 m long with three ore passes, the sublevel above Level 9 is approximately 90 m long with two ore passes and the sublevel above Level 8 is approximately 130 m long with three ore passes.

The main portal of Mária Mine, at +360 m is located at the surface mine facility area on the east slope of Kalvária-Tri Vrchy ridge, just above the road from Rožňava to Čučma. Near the portal, there is an inclined shaft down to Level 6, and from Level 6 there is a blind shaft down to Level 13 providing access to each of the five haulage crosscuts leading to the Strieborná vein.

The inclined shaft was rehabilitated and is now fully functional. Access is provided to Level 6 where a crosscut connects to two underground drilling platforms used to further explore Strieborná during the Company's 2012-2013 drilling campaign.

A vertical shaft called Nová Mária is located about 170 m south of the main portal and goes down approximately 70 m.

In 2012, Global Minerals erected a water treatment plant that treats pumped mine waters prior to discharge.

There is a haulage tunnel that starts underground from about 20 m down the inclined shaft and heads northwest for approximately 1,600 m to surface at +340 m on the west slope of Kalvária-Tri Vrchy ridge, above road 67. From here, a two-kilometre drive leads to the area of the proposed processing and tailings facilities.

4.4 Other Agreements, Encumbrances, and Liabilities

The proposed area for plant tailings storage is not presently owned/controlled by Global Minerals. On January 28, 2013, Global Minerals' local subsidiary signed an agreement to acquire the former tailings facility of the Mária Mine for storing tailings from future Strieborná mining operations. The purchase price was EUR 2,500,000 with payment spread out over a 12-month period. During the summer of 2014, the Company entered into negotiations to modify the payment schedule to better suit the overall Project development progress. Global Minerals has advised AGP that they expect that negotiations will be successful due to the mutual interests of the seller and Global Minerals to see a sale completed. AGP has been advised that Global Minerals expect to have a new agreement in place by 2nd Quarter, 2015.

At present, AGP is unaware of any royalties, back-in rights, payments, or other agreements and encumbrances, and any environmental liabilities to which the property is subject.

4.5 Required Permits

The Company will require a number of permits, such as land use, environmental, building and mine operating permits in order to reopen the Strieborná section of the Mária Mine. Currently, Global Minerals is discharging treated mine water under a discharge permit. Global Minerals has advised AGP that the appropriate steps are being taken to obtain all necessary permits in a diligent manner.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Location, Physiography, and Land Use

The Strieborná Project is located at latitude 48° 40' 29" N and longitude 20° 32' 31" E, 2 km to the north of the centre of the town of Rožňava in the southern foothills of the Spiš-Gemer Ore Mountains (*Spišsko-Gemerské Rudohorie*, also locally called the *Volovské Vrchy*), which form the southern part of the Slovak Ore Mountains (*Slovenské Rudohorie*) of the Western Carpathians.

Figure 5-1 illustrates physiography of the Rožňava area. The town of Rožňava occupies a low spur ridge between the Slaná River (the broad valley on the left centre of the image) and its tributary called Rožňavský Potok (seen above the name Rožňava), and is flanked from the west and northeast by forested ridges, Tvadova-Turecká (954 m) and Rákoš (800 m), respectively. Two high points mark the low north trending ridge that separates the valley of the Slaná River from Rožňavský Potok, one at 478 m called Kalvária and another at 581 m called Tri Vrchy.

The Strieborná property extends from the west flank of the Rákoš hill across Rožňavský Potok to the southern flank of the Kalvária-Tri Vrchy ridge.

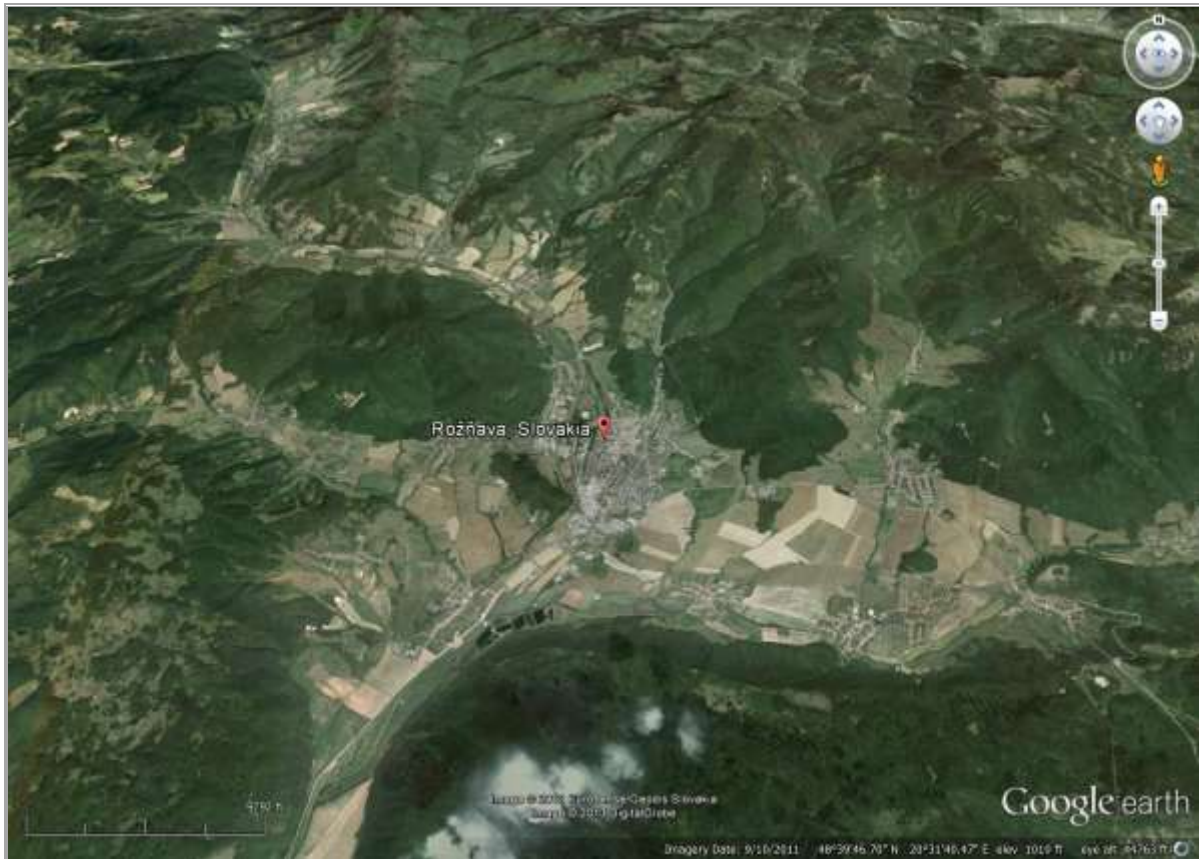
Secondary beech and oak forests cover the Rákoš hill and the Kalvária-Tri-Vrchy ridge, while the low ground along Rožňavský Potok is a loosely built-up residential area.

5.2 Accessibility

The property is readily accessible by the tarmac road linking Rožňava with the village of Ouňma. Rožňava is a county town with approximately 20,000 inhabitants linked by good roads with Košice and Poprad. The road distance to Košice along Road E571 is 70 km. The town of Rožňava is situated on a railway line linking Bratislava (the capitol) and Košice.

The nearest airport is Košice International Airport situated 14 km west of Košice. Czech Airlines, Tyrolean Airlines, and Slovak Airlines service the airport among others. Other international airports within easy reach are in Vienna, Bratislava, and Budapest.

Figure 5-1: Google Image of Rožňava Area Looking North



5.3 Climate

The Slovak Republic lies in the mild zone of sub-continental climate, with distinctive seasonal changes and microclimate depending on elevation. The low ground of the Slaná river valley and its tributaries has a mean annual temperature of about 8°C, with monthly average temperatures ranging from approximately 0°C in January to 20°C in July and annual rainfall of approximately 700 mm. The climate is not a factor restricting the length of the operating season.

5.4 Local Resources and Infrastructure

The town of Rožňava is home to a large mining community and has sufficient skilled labour and infrastructure (power, roads and railway) necessary for mining and processing ore from the Strieborná deposit. Rožňava has been a mining town since at least the 13th century, and has a mining museum and an old mint founded in the 17th century.

The former Mária Mine site offers ample space for the infrastructure and facilities necessary for mining operations. Additional land parcels have been acquired to accommodate all future needs of the mine and processing and tailing operations.

The Košice region as a whole has its own independent power infrastructure. The most significant power source of the region is the combined coal and gas power station SE – Elektrárne Vojany (EVO I and EVO II) with installed output 12 x 110 MW. The main distributor of the electric power in the Košice Region is the East Slovak power distributing company, Východoslovenská energetika (VSE) Košice.

The local telecom market has approximately 20 operators and 2 bodies of mobile telephone stations. In the field of data transmission, the region has its data network with the data junctions interconnected with the optical cable network. The city of Košice is the regional centre for the network infrastructure covering the whole region.

6 HISTORY

6.1 Mining History

Mining activities in Central Europe started in Celtic times (Hallstatt culture, pre-500 BC). The oldest written mention of Rožňava is dated 1291, in the donation letter of André III, King of the Hungarian Empire, who ceded this region to the Bishop of Estergon. High silver production from the Rožňava mines was supposed to recover the bishop's losses associated with his services protecting the interests of the king.

The Mária vein is among the oldest known and mined vein occurrences in the region. Mining law reforms in 1327 sparked a mining boom in the area in the second half of the 14th century, with the main efforts devoted to the mining of silver, copper, and gold.

In the 15th century there was a crisis in mining, due to the exhaustion of surface and near-surface deposits and the lack of efficient water-pumping devices, which lasted until the end of the 15th century, when efficient pumping devices were invented and employed in Rožňava mines. Several new processing plants were built, and a major mining boom began, the all-time golden age of Rožňava mining, which lasted into the beginning of the 16th century Rožňava; the prosperity of that period has never been repeated, largely due to exhaustion of the shallow oxidized parts of siderite veins containing copper, silver, and gold. A partial revival of mining in the Rožňava area began in the second half of the 16th century, with iron ore being the main commodity mined.

During the first half of the 19th century, the importance of mining activities decreased, but the Gemer area remained the leading iron production region in the entire Hungarian Empire. The century of steam brought new technologies to the mines, mainly drilling rods and rail transport. New mining companies were founded, with the leading role in those activities played by the nobility, notably the Andrassy family. Count Andrassy built iron works in Betliar and Rdnava in the middle of the 19th century, where siderite and limonite previously considered waste were profitably reprocessed.

Between 1900 and 1909, the Rožňavský baňa iron ore processing plant was constructed by a Hungarian railway company at Nadabula, just west of Rožňava and across the Slaná River. The impulse for the founding of the plant was a find of rich deposits of siderite ores in Turecká Hill. At the time, it was one of the most modern mining plants in Europe. The Mária Mine operated, with short-lasting closures, until the end of the Second World War.

In 1946 the Mária Mine and the Rožňavský baňa processing plant at Nadabula became part of the newly established state enterprise Železoruďné Bane š.p. (ŽELBA), headquartered at Spišská Nová Ves. Production figures for the Mária vein from 1951 to

1992 are 2.3 Mt (Grecola, 1995). The deposit provided complex siderite ore with a sulphide component containing silver, antimony, mercury, and bismuth.

In the mid-1990s, the mine was shut down and allowed to flood. Global Minerals began a dewatering and rehabilitation program in 2011. The mine is currently dewatered to Level 8 or slightly higher. The inclined shaft from surface to Level 6 has been rehabilitated and provides access for exploration activities. Surface facilities have either been demolished if not needed, or rehabilitated for ongoing exploration and development activities.

6.2 Processing History

MacLeod (1995) describes the history of ore processing from the Mária vein. Before 1965, Mária ore was transported to Čučma where it was processed in an old plant. In 1965, ore processing by froth flotation was introduced at the Nadabula plant, where sulphides and siderite were floated separately. This replaced the older siderite sorting plants (previously separated by hand and sorted on sorting machines). In the 1960s, flotation sulphide concentrates from the Mária Mine were sold to Belgian and Japanese smelters. In 1978, siderite flotation at Nadabula was shut down due to high diesel fuel consumption. Crushed ore from the Mária Mine was transported 40 km north to Rudňany for sulphide flotation, and the siderite by product was discarded as waste at the tailings site. Sulphide flotation concentrate was stockpiled at Rudňany. In 1981, the processing plant at Nadabula was reopened after installation of a magnetic separator, which was used to process siderite ore from Sadlovský and other mines, and to reprocess old flotation waste from the Mária Mine stored at Rudňany. Starting in June 1990, all Mária Mine ore was processed at Nadabula until the shutdown of mining in 1993.

6.3 Exploration History

The Strieborná vein was discovered in 1981 in Crosscut P-1, which had been driven eastwards from Level 13 of the Mária Mine, reaching the Strieborná vein at the elevation of -163 m below Adriatic Datum, and subsequently explored by the Slovakian Geological Survey ("the SGS"). By 1991, additional crosscuts had been driven between the Mária and Strieborná veins at Level 10 and Level 6, with driving and channel sampling in the Strieborná vein at Level 13 and Level 10. Mesarčík et al. (1991) reported on exploration work to date and presented estimates of the Strieborná iron, silver, and copper resources, which were independently reviewed and approved by the Slovak Geological Bureau in 1992. In 1994, based on the same data but using new resource classification categories, Mesarčík presented revised estimates of the Strieborná iron, silver, and copper resources; these estimates were independently reviewed and approved by the Slovak Ministry of Environment in 1995.

In 1992, Mesarčík supervised channel sampling along Level 8. At the same time, government subsidised mining of the Mária vein stopped. The Mária Mine was put on government funded care and maintenance to allow continued access to the Strieborná

vein. In 1993, mining licenses were issued to ŽELBA for the Strieborná and Mária veins. Government subsidised exploration work ceased in mid-1994.

In 1992, ŽELBA entered into a joint venture with London-based SAMAX Resources Ltd. (SAMAX) to assess whether sulphide concentrate stockpiles at Rudňany could be profitably processed using a recovery process developed by Ausmelt Ltd.

In mid-1994, SAMAX spun off its European assets to CMX Resource Limited (CMX), and the joint venture turned its attention to the Strieborná vein. The vein was accessed on Level 9, and sublevel drives were developed above Level 8 and Level 9. In the autumn of 1994, CMX mapped about 60% of all underground workings on the Strieborná vein and resampled the previous channel sample locations to verify results reported by the geological survey. In addition, the program included detailed structural and mineralogical studies (Jakubiak, 1994). CMX presented the results of their work to Sunshine Mining and Refining Company of Coeur d'Alene, Idaho, USA, which conducted metallurgical studies using its own proprietary hydrometallurgical process, independent resource/reserve estimation (Bond and Young, 1994), and a mining study (MacLeod). Sunshine Mining decided not to pursue the opportunity. In 1995, the joint venture between ŽELBA and CMX was terminated.

Approximately 30,000 tonnes of stockpiled development ore from the underground workings in the Strieborná vein were milled and processed at Rožňavský Baňa, resulting in approximately 3,000 tonnes of tetraedrite concentrate. Of that, 50 tonnes was set aside for, and largely consumed in, metallurgical tests, while the remaining concentrate was sold to a Peruvian copper smelter.

Between 1995 and 1997, several academic studies of the Strieborná vein deposit were undertaken by researchers from the mining faculty of the Technical University of Košice (geostatistical by Blišťan (1995), geochemical by Kondela (1997), mining by Hatala (1996)), and by the Institute of Geotechnics of the Slovak Academy of Sciences (hydro-metallurgical study by Florek (1996)).

Mesarčík et al. (1996) completed and presented his report on Level 8 exploration of the Strieborná vein, including updated resource estimates. Ivan Mesarčík was a member of the verification team mobilized by CMX, but for confidentiality reasons could not include any CMX data in his report.

In 1995, Feichtner Consulting Engineers GmbH of Austria (FCE, 1995) prepared a cost analysis for ŽELBA. There was an updated in-house cost analysis by Karoli (1997), a member of ŽELBA's staff. Also in 1997, new mining licences were issued to ŽELBA for Strieborná, and in 1998 new exploration licenses were issued.

ŽELBA was declared bankrupt in 2000. Their Rožňava assets (land, buildings, etc.) were purchased in 2004 as part of a bankruptcy sale by Baňa Siderit, subsequently renamed Rudohorská Investičná Spoločnosť. The former ŽELBA mining licences were put out to

tender by the Mining Bureau in January 2006. With the support of Global Minerals, PIDEKO CGF s.r.o. won the tender and was granted the mining licences RožňavaRožňava I and RožňavaRožňava III. The Company originally formed a joint venture with PIDEKO, which set up a local subsidiary company in March 2007 (Global Minerals Slovakia s.r.o.). Authorization for the transfer of the mining licences was granted on May 30, 2007, in File #427-1110/2007 of the Mining Bureau at Spišská Nová Ves, and the licenses were transferred to Global Minerals Slovakia on May 31, 2007. On October 4, 2010, the mining licences were transferred to Gemer-Can, s.r.o., which became a successor of Global Minerals Slovakia. The Mining Bureau Office at Spišská Nova Ves granted transfer. File #763-2194/2010 authorized transfer of Mining Licence Rožňava I. File #763-2194/2010 authorized transfer of mining licence Rožňava III. The Issuer has since completed acquisition of 100% ownership of Gemer-Can, s.r.o from its former partner.

The verification program conducted by CMX was the last exploration activity on the Strieborná vein prior to the Company's acquisition of the project. Following the dissolution of ŽELBA, the mine was allowed to flood.

At present, the Mária Mine has been dewatered and is working fully rehabilitated to Level 6 (170 masl elevation). Dewatering efforts are ongoing, and currently the water level at the mine is at approximately 70 masl elevation (8 m below Level 8 of the Mária Mine). Between April 2011 and May 2013, two programs of verification and expansion drilling were completed by GML, including 6,144 m in 40 drill holes from surface and underground platforms. During April–June 2014, GML developed 120 m of drift on Level 6, and exposed 20 m of the Strieborná vein via the drift and cross-cuts. Approximately 500 tonnes of material were removed as a bulk sample. Rožňavský Baňa mill has been dismantled and removed.

6.4 Historical Resource Estimate for the Strieborná Vein

Multiple historical estimates have been done for Strieborná by Ivan Mesarčík an employee of the SGS, and later by CMX in the early 1990s; several more are discussed below. The estimates reported in this History section are not NI 43-101-compliant, and are presented here only for completeness and historical perspective, except for the AMC estimate (Jakubiak, 2008), which has since been superseded by the resource estimate reported herein in Section 14. The reader cannot rely on those other estimates, but can get a sense of differences between historical estimates. The author has not done sufficient work to classify these historical estimates as current Mineral Resources or Mineral Reserves, and Global Minerals is not treating these historical estimates as current estimates.

Ivan Mesarčík completed the initial resource for the SGS in July 1991. This estimate is presented in Table 6-1 (as reported in Jakubiak, 1994, and Madeisky and Chernavska, 1998), and is based on underground channel sampling and drilling programs on Level 13 and Level 10. This work estimated an average vein thickness of 2.51 m for the material

reported at a 0.2% Cu cut-off. The SGS classified their estimate as class C₂, following the classification scheme in effect in eastern Europe at the time (Red Book), a resource classification system divided into classes designated A, B, Ct, and C₂ to describe resources ranging, respectively, from those with a high assurance of existence, to those with a low assurance of existence. According to Madeisky and Chernavska (1998), the C₂ classification corresponds to a “combined Indicated–Inferred resource.”

Table 6-1: Slovak Geologic Survey Estimated Global Resources Class C₂

Cut-off	Tonnes (x '000s)	Ag g/t	Cu %	Sb %	Hg %	Fe %
0.2% Cu	3,302	173.8	0.82	0.53	0.010	33.0
0.3% Cu	2,048	237.5	1.17	0.77	0.016	32.9
0.6% Cu	1,410	312.4	1.54	1.03	0.023	33.0
0.9% Cu	896	404.4	1.98	1.34	0.031	32.2
1.2% Cu	560	460.5	2.26	1.54	0.035	32.6

In May 1994, the Slovak Geological Office changed their classification categories, and the Strieborná estimate was reclassified to categories equivalent to Indicated and Inferred. The Ministry of the Environment of the Slovak Republic independently reviewed and approved this result in 1995 (Madeisky and Chernavska, 1998).

Jakubiak (1994) reported a “global resource figure stands at 2,938,800 tonnes at 220 g/t Ag, 1.05% Cu, 0.64% Sb, and 0.017% Hg, while resource at a cut-off grade of 150 g/t Ag stands at 854,200 tonnes at 433 g/t Ag, 1.92% Cu, 1.20% Sb, and 0.038% Hg.”

As reported by Madeisky and Chernavska (1998), “in December 1994, ‘Proven and Probable Geologic Resources’ for the Strieborná vein above and below Level 8: 1,570,000 tonnes at an average grade of 394.2 g/t Ag, 1.699% Cu, 1.120% Sb, and 0.036% Hg, at an average vein width of 4.8 m.”

According to Madeisky and Chernavska (1998), Ivan Mesarčík re-estimated the resource of the Strieborná vein in July 1996, using the data from Level 13 and Level 10, but also including data from Level 8 and from surface drilling: “‘Z-2’ Indicated Resource Estimate for the Strieborná vein above and below Level 8: 1,245,683 tonnes at an average grade of 400.6 g/t Ag, 1.761% Cu, and 33.65% Fe, with a 210 g/t Ag cut-off.”

Jakubiak (2008), on AMC Consultants letterhead, stated that a “mineral resource is reported without a cut-off grade applied as the form of the deposit and the mean silver, copper, antimony and iron grades are such that the Strieborná vein has reasonable prospects for economic extraction in its entirety. The terms Mineral Resource, Measured Mineral Resource, Indicated Mineral Resource and Inferred Mineral Resource have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy, and Petroleum as the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by CIM Council on August 20, 2000. The Measured and Indicated

Mineral Resource categories are defined with sufficient confidence to support a Preliminary Feasibility Study.” AMC’s resource estimate, presented in Table 6-2, is superseded by the resource reported in this document (Section 14).

Table 6-2: 2008 AMC Reported Resource

Mineral Resource Category	Quantity (Mt)	Horizontal Width (m)	Average Grades			
			Ag (g/t)	Cu (%)	Sb (%)	Fe (%)
Measured	215	5.8	468	2.2	1.4	20.6
Indicated	1,710	3.4	202	1.0	0.6	33.2
Inferred	1,500	3.2	280	0.9	0.6	33.6

Note: The above Mineral Resource includes an estimated 35,000 tonnes of material depleted during underground development.

In December 2012, MDA completed a non-NI 43-101-compliant in-house resource estimate for Global Minerals. In 2013, MDA completed a NI 43-101-compliant resource estimate, which is summarized Table 6-3. A summary of the estimated resources at the reporting cut-off of 100 g Ag/t is presented in Table 1-1. In 2014, MDA updated the estimate, superseding the 2013 estimate.

Table 6-3: Strieborná Mineral Resource

Cut-off (Ag g/t)	Tonnes	Ag (g/t)	Cu (%)	Sb (%)	Fe (%)	Ag (oz)	Cu (t)	Sb (t)
Measured								
100.0	132,000	267	1.32	0.98	31.0	1,134,000	1,740	1,290
Indicated								
100.0	2,158,000	266	1.20	0.84	34.4	18,455,000	25,820	18,090
M&I								
100.0	2,290,000	266	1.20	0.85	34.2	19,589,000	27,560	19,380
Inferred								
100.0	872,000	211	0.98	0.69	34.4	5,915,000	8,510	6,050

7 GEOLOGICAL SETTING

7.1 Geologic Setting

The reader is referred to Appendix A throughout this section.

7.2 Regional Geology

The Strieborná Project is located in the Inner Western Carpathians; the Carpathians form the eastward continuation of the Alps.

During the Early Paleozoic, there was a sedimentary trough in the region of the present Carpathians. During the Late Paleozoic Variscan orogeny, convergence of the basin resulted in deformation of the sediments and emplacement of granitic intrusions. After a short period of epicontinental sedimentation, the Carpathian area was again submerged, and marine (mainly calcareous) sediments were deposited from the Early Triassic through the Early Cretaceous, as the Tethys Sea occupied the general position of the Alps-Carpathians. Once again, convergence of the Tethys basin during the main mid-Cretaceous pulse of the Alpine orogeny caused strong deformation, nappe overthrusting, igneous intrusions, and volcanism. In the Miocene, there was subsequent magmatism and additional orogenic pulses of the Alpine orogeny in the Carpathians (Grecula, 1994, as cited by Jakubiak, 2008).

Within this part of the Inner Western Carpathians, which includes the Strieborná Project, lies a polydeformed and polymetamorphosed fold belt. Early Paleozoic volcanogenic flysch (Gelnica Group) was folded and metamorphosed to chlorite and cummingtonite-hornblende-biotite zones of the greenschist facies during the late Variscan orogeny. Metamorphism gave rise to early metamorphic foliation S0 and eventually produced schistosity S1, which remained parallel to bedding. The most characteristic feature of the Variscan orogeny was extensive granitization and diapiric rise of granitoids, exemplified by the northward-overtaken Volovec anticline, with granite emplaced in its core.

These Variscan structures are overlain with an angular unconformity by a Permian continental sequence (Gočaltovo Group), prograding to a lagoonal or shallow marine environment during the Lower Triassic. Early in the Alpine orogeny, the rocks of this region were thrust as a large and complex nappe over the adjacent tectonic-metallogenic unit of Veporicum.

There were four periods of Alpine deformation. The oldest deformation (D2), which has been dated as post-Triassic based on regional-scale stratigraphic evidence, was transversal to the earlier Variscan structures, and involved isoclinal or tight folding, with transposition of bedding and schistosity S1, to form pervasive slaty cleavage S2. On a regional scale, this deformation produced nappes with a northward vergence. The S2

cleavage predominates in the rocks hosting the Strieborná vein system. Implications of transposition are far reaching. First, it has produced lenticular layers of various lithologies, all with a similar orientation, and all parallel to cleavage S2. Second, any sedimentary siderite beds, if present, have been boudinaged and rotated towards parallelism with the cleavage. Third, most if not all fold closures have been eliminated. Finally, the stratigraphic sequence is oblique to cleavage S2 and parallel to enveloping surfaces S1, i.e., to tangents of relics of isoclinal fold hinges.

The three subsequent Alpine deformations (D3–D5) modified the pre-existing structures by further folding and faulting. The youngest deformation was synchronous with the development of epigenetic siderite, quartz, and sulphide mineralization within the region.

7.3 Local Geology

Rocks of the Gelnica and Gočaltovo Groups underlie the Rožňava area.

The Gelnica Group (Upper Cambrian to Lower Devonian) is the volcanogenic flysch deposited in the Early Paleozoic trough or rift basin of this region, and appears to have been deposited as an island arc. The group is subdivided into three formations, of which the upper two—Bystrý Potok Formation (Upper Silurian) and the Drnava Formation (Early Devonian)—are present in the project area. The Bystrý Potok Formation consists of coarse metasedimentary rocks, phyllites, metarhyolite tuffs, keratophyre and dacite porphyries, and phyllites with local intercalations of acid volcanic material and carbonate and jasper lenses. Carbonates include stratiform siderite deposits (e.g., siderite deposits at Nižná Slaná). The overlying Drnava Formation consists of coarse-grained metasedimentary rocks, passing upwards into laminated metapsammites and phyllites, and metamorphosed rocks resembling sheared porphyries (called “porphyroids”) but derived from psammitic sediments and/or pyroclastic rocks of acid composition.

The Gočaltovo Group comprises the Permian epicontinental rocks and overlying marine sedimentary rocks of the region. The group is subdivided into two formations, with the Lower Rožňava Formation (Early Permian) occurring in the Rožňava area.

7.4 Property Geology

The Strieborná vein, and the adjacent Mária, Mayer, Pallag, and Podložná veins, are all hosted in the Drnava Formation, described in Section 7.3. These northeast-striking, tetrahedrite-bearing siderite+quartz veins are spaced across Global Minerals’ two concessions in the following order, from northwest to southeast: Mária vein, with subsidiary Mária I and Mária II veins; Mayer vein (mentioned in historical reports, but details are unclear); Podložná vein; Strieborná vein; and Pallag vein. Additionally, recent drilling has intersected several apparently discontinuous veins on the northwest side of the Strieborná vein, one of which has been modelled as the West Split of the Strieborná

vein. The Strieborná, Podložná, and Pallag veins, the subjects of this report, will be described in Section 7.6, while the others will be described here.

Figure 7-1 is a generalized level plan showing the workings that developed or explored these veins. Figure 7-2 is a generalized cross-section showing the relationships of the veins; the Mayer vein is not shown. Neither the cross-section nor the level plan show the results of recent exploration.

The Mária vein has been mined out down to Level 9. The vein is 1,800 m long along strike, and extends 750 m down dip. It strikes northeast, with a variable dip of 50° to 70° to the northwest or southeast. The thickness of the vein is variable, with an average of 2.2 m and a maximum of 22 m, as reported by Jakubiak (2008). There are two sub-parallel veins, Mária I and Mária II, alongside the Mária vein near its northeastern end.

The Mayer vein was worked during the medieval period, and was opened by the Mayer adit and by headings at Level 6 of the Mária Mine. The vein was followed for 350 m along strike, and has a narrow width (0.4 m on average) and low grades. The Mayer vein is sub-parallel to the Mária vein. This vein has not been encountered during recent drilling, is not shown on historical geologic maps, and will not be discussed further.

Other veins in the Rožňava mining district typically have a northeasterly strike, are approximately 1.5 km long, and extend more than 500 m vertically. They exhibit an echelon distribution and have siderite as the principal ore mineral, often with ladder textures of quartz-siderite infilling. The near-surface parts of some veins are characterized by deep oxidation. The presence of tetrahedrite in the Mária and Strieborná area veins is distinctive.

Figure 7-1: Plan Map of Mária, Podložná, Strieborná, and Pallag Veins

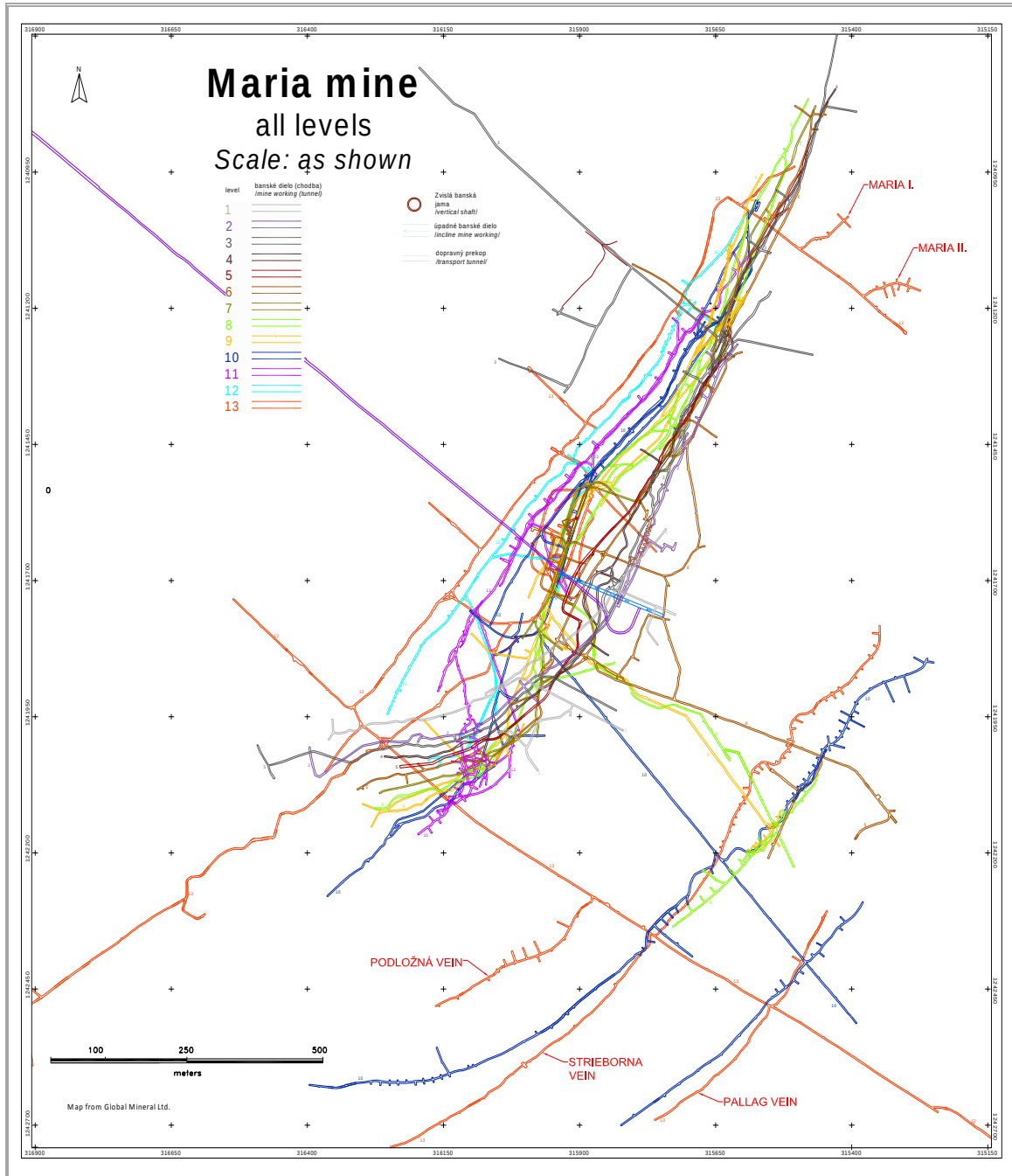
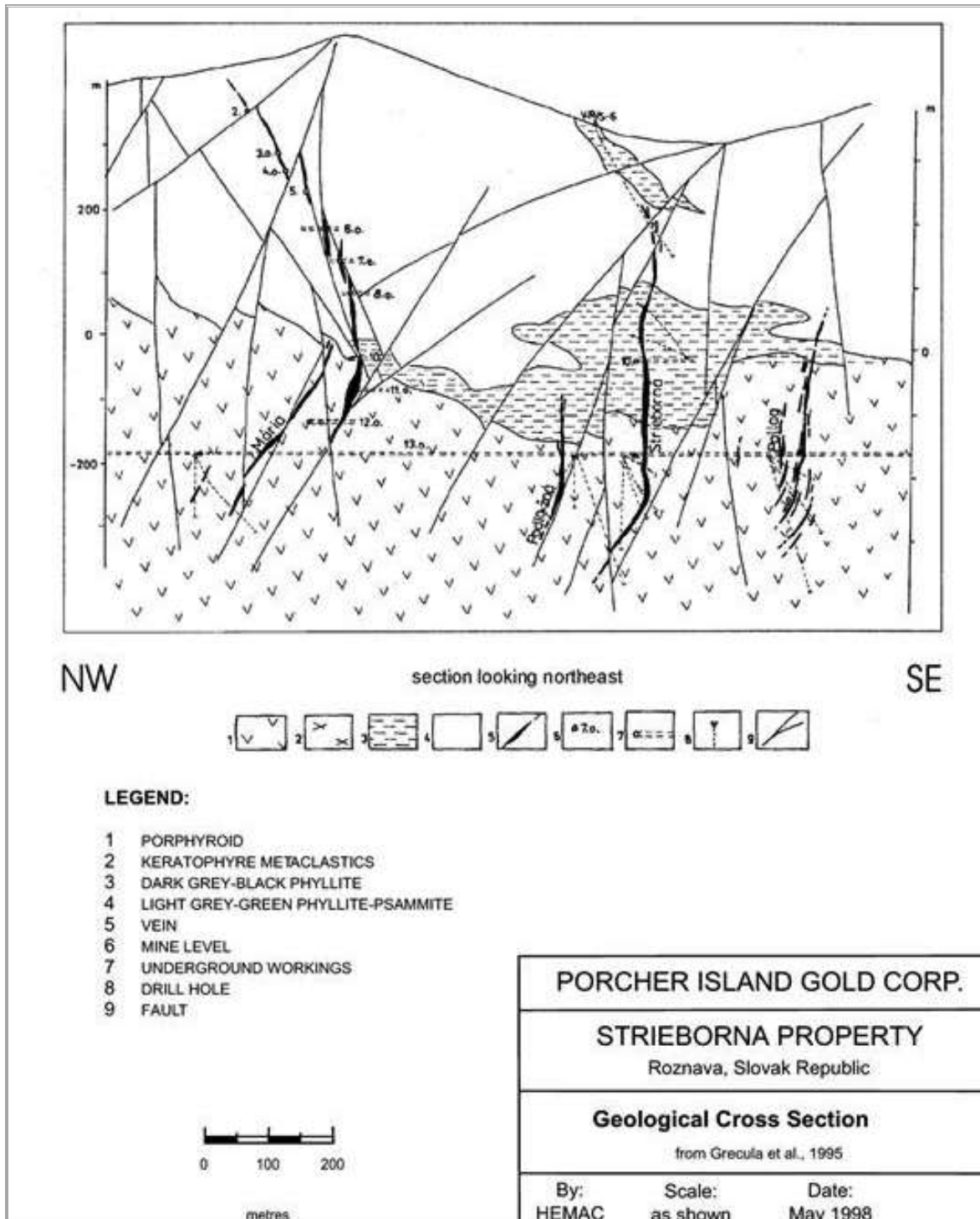


Figure 7-2: Generalized Cross-Section showing Geology and Mineralized Veins



Source: Madeisky and Chernavska, 1998

7.5 Mineralization

Mineralization on the Strieborná Project occurs in a series of tetrahedrite-bearing (silver, copper, and antimony) siderite-quartz veins. Three veins, in decreasing order of definition, are the Strieborná, Podložná, and Pallag veins. Each vein is described separately in Section 7.6, but many of the characteristics of the Strieborná vein, which follow, also apply to the others. Resources have only been estimated in the Strieborná and Podložná veins.

Host-rock types and brittle deformation have primarily influenced the intensity, morphology, and geometry of the Strieborná vein. Complex forms and variable thicknesses are characteristic on Level 8 and Level 9 where the vein passes through quartz-sericite-chlorite phyllite, siliceous psammitic slate, and folded bodies of dark pyritiferous slate. These upper areas of the Strieborná vein tend to have higher concentrations of tetrahedrite. On Level 10 and Level 13, where porphyroids predominate, the vein pattern is more consistent, and the mineralization is lower grade.

According to Mato (1994), where it is hosted by the porphyroid, the vein shows more evidence of cataclasis. Where it is hosted by the sericite-chlorite phyllite, the vein shows folding, lensoid shapes, and ladder structures. Mato also notes that black metapelite (phyllite) is a less favourable host for vein formation.

High-grade silver-copper-antimony mineralization occurs in a single siderite vein with abundant internal quartz ladder veins on Level 8. The vein is similar in morphology on Level 9 and Sublevel 9, but the strike and dip are variable. On Level 10 and Level 13, the principal vein extends for a distance of 1,300 m, with a consistent northeast strike and a steep but variable dip. Quartz ladder veins are less abundant here than on Level 8 and Level 9, and locally the vein consists of almost pure, coarsely crystalline siderite, with disseminated tetrahedrite.

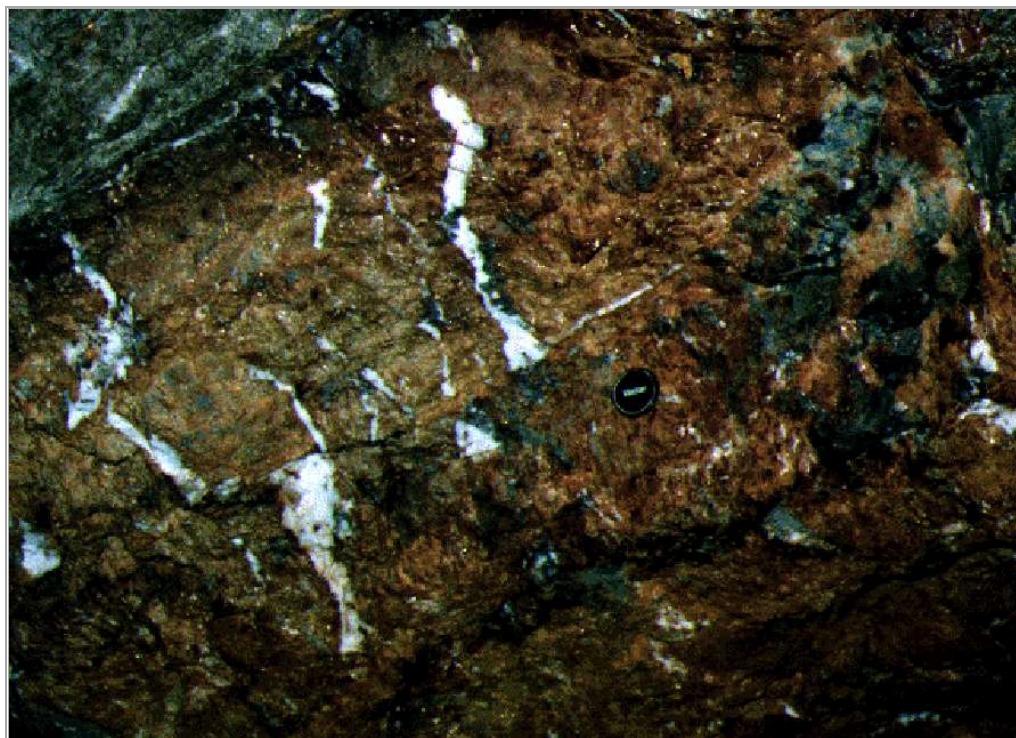
Ladder quartz veining (Figure 7-3) within the siderite veins of the Strieborná structure plays a significant role in the emplacement of sulphide mineralization. Sub-vertical faults or fractures opened along the contact between ladder quartz and siderite, and acted as conduits for the ascent of mineralizing fluids. The faults or fractures were gradually filled by sulphide mineralization. Subsequent deformation reactivated and reopened the subvertical faults or fractures (Figure 7-4). High concentrations of tetrahedrite up to 1 m thick were developed locally (Madeisky and Chernavska, 1998).

The principal non-sulphide vein minerals are medium- to coarse-grained siderite and variable amounts of white quartz, which generally forms prominent ladder veins. Coarse-grained calcite, ankerite, and muscovite occur in subordinate amounts, along with chlorite, albite, illite, kaolinite, and sericite.

Jakubiak (1994) describes the styles of mineralization as follows:

- pods (up to 30 cm in diameter) in quartz-siderite veins
- irregular chains (up to 20 cm thick) along contacts of quartz with siderite
- disseminated in coarse-grained siderite
- films along contacts with siderite boudins in central parts of the vein
- fracture-controlled veins in siderite (Figure 7-3)
- hydrothermal breccias (up to 1 m thick) with siderite and quartz floating in a tetrahedrite matrix.

Figure 7-3: Quartz Ladder Veins Emplaced in Siderite



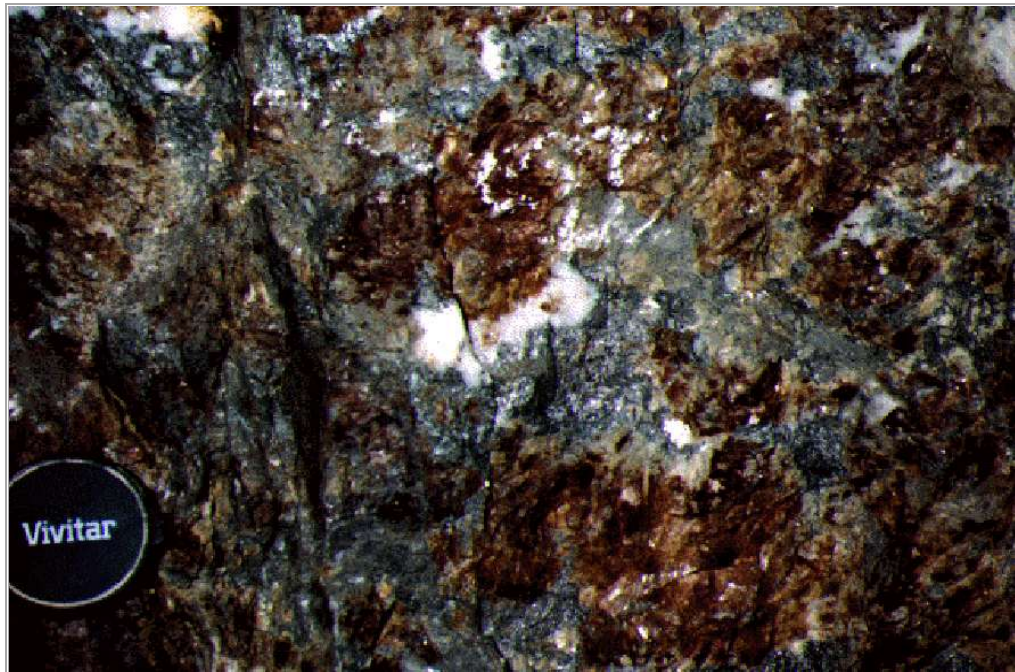
Note: Brown is siderite, white is quartz, and black in the vein is tetrahedrite.

As described by Sasvári and Mato (1996, as cited by Jakubiak, 2008), the Strieborná vein contains shoots with fine- to medium-grained, semi-massive to massive sulphide accumulations. Tetrahedrite is the principal sulphide mineral, with pyrite, arsenopyrite, and chalcopyrite occurring in subordinate quantities. Tetrahedrite carries the silver, copper, and antimony, and, reportedly, bismuth and mercury values. The chalcopyrite content seems to be higher in the southwestern portion of the vein and at depth, and the pyrite content increases with depth. Minor amounts of galena, sphalerite, covellite, magnetite, and pyrrhotite are disseminated, or occur in thin veinlets and micro-aggregates. Ullmannite, bismuthinite, jamesonite, boulangerite, gersdorffite (NiAsS), native bismuth, gold, stibnite, and lead-antimony-bismuth-copper sulphosalts occur in trace quantities. Locally, in some parts of the vein structure, there are also higher

amounts of marcasite, bornite, covellite, and hematite. All of these minerals form subhedral to euhedral grains on a micron to centimetre scale, and are interstitial to, or fill fractures within carbonates and quartz.

The mineral paragenesis and internal morphology of the Strieborná vein are described in great detail by Sasvári and Mato (1996, as cited by Jakubiak, 2008). Mesoscopic (outcrop scale) analysis of the internal morphology of the vein indicates that the sulphide mineralization and enclosing rocks have been modified by folding and successive stages of penetrative deformation, shearing, flattening, and extension, which resulted in tectonic thickening, attenuation, and boudinage (Figure 7-4). Sasvári and Mato distinguished 11 phases of penetrative deformation with mineralizing events.

Figure 7-4: Tetrahedrite Associated with Subvertical Fractures in Siderite Vein



Note: Brown is siderite, white is quartz, and black in the vein is tetrahedrite.

7.6 Vein Descriptions

The reader should refer to Figure 7-5 and Figure 7-6 for a visual representation of the following descriptions.

7.6.1 *Strieborná*

The Strieborná vein is roughly parallel to the Mária vein. It has a strike length of 1,300 m that is defined by drifts along the vein on Levels 8, 9, 10, and 13, and a 380 m vertical extent defined by drifting and drilling, but the actual resource model extends vertically for 550 m. The Strieborná vein has no surface exposure, although traces of old workings on the southwestern flank of Rákoš Hill may follow a sub-outcrop of the uppermost

fringe of the Strieborná vein or, more likely, one of its low-grade branches. The vein strikes northeast and has a propeller shape, with subvertical dips in the centre changing to southeastern dips in the southwestern part of the vein, and changing to northwestern dips in the northeastern part of the vein (Figure 7-6).

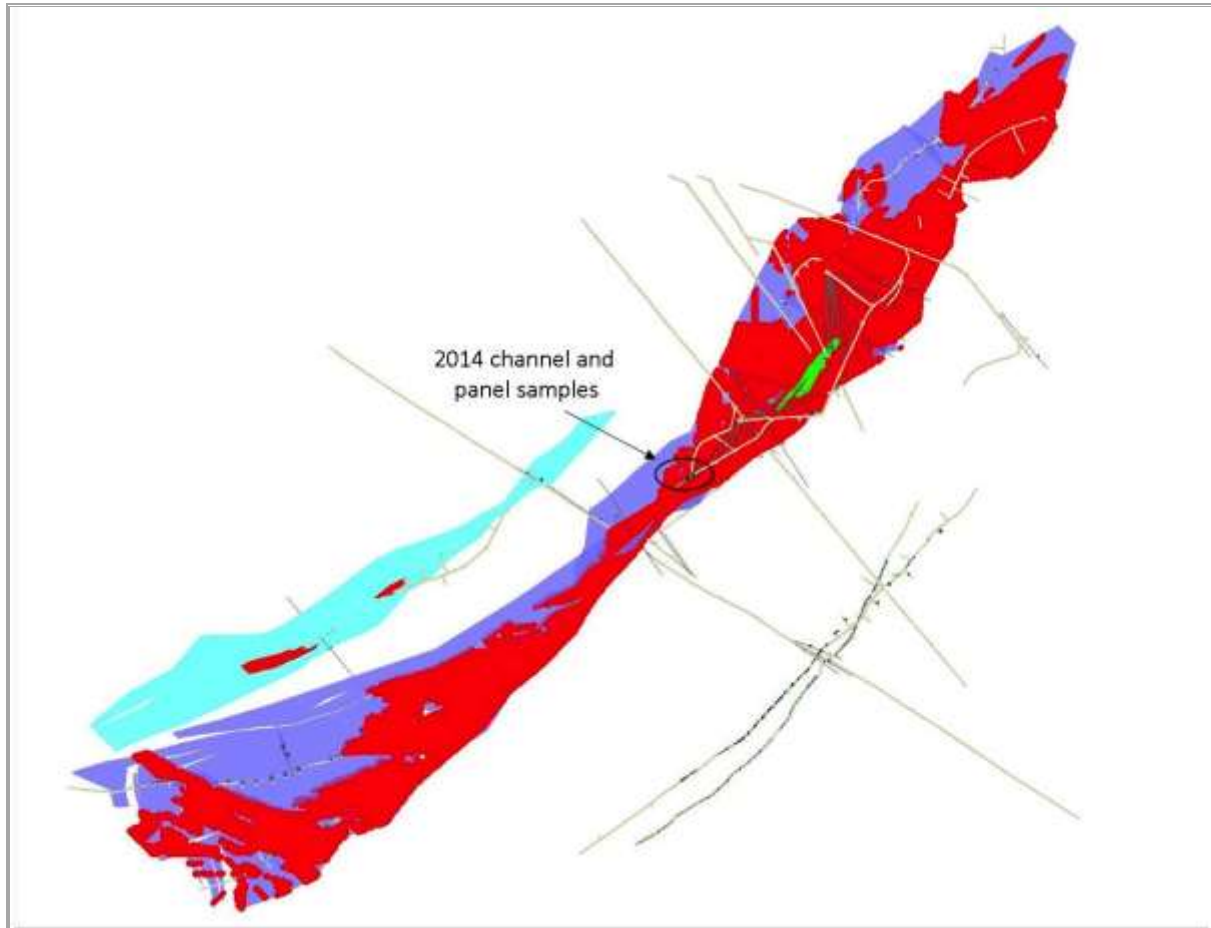
Because the Strieborná vein dips to both the southeast and northwest, it is difficult to define the hanging wall and footwall. Rather than use these terms, secondary veins are described as either being “on the northwest side” (closer to the Mária vein, for example the Podložná vein), or “on the southeast side” (Pallag vein).

The average horizontal thickness of the Strieborná vein (which is little different from its true thickness, as the vein is near vertical to vertical) is 2.1 m for everything modelled. The average horizontal thickness of those portions of the Strieborná vein with grades greater than 100 g/t Ag is 2.23 m, and it reaches a maximum horizontal thickness of 13.6 m.

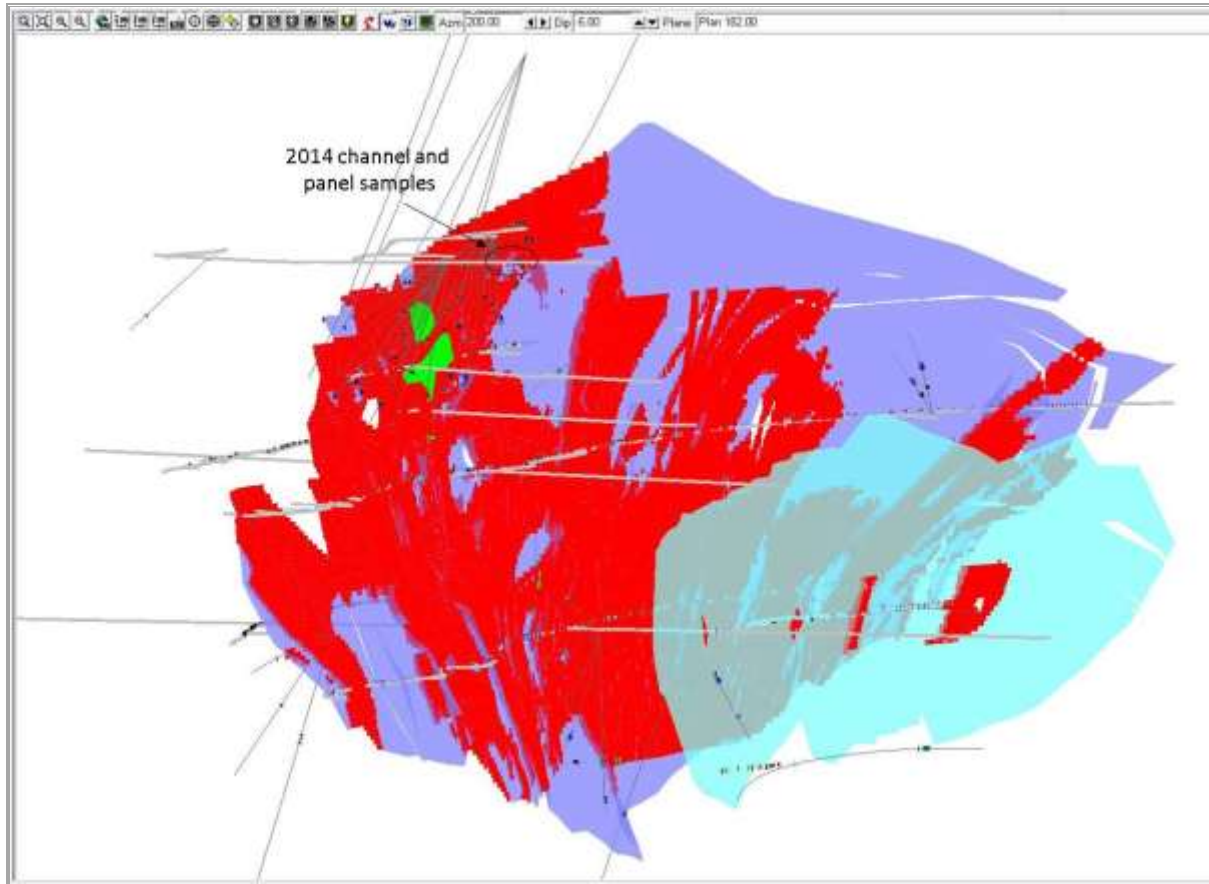
The Strieborná vein is not a single vein but is irregular in shape, with several splits. One of those splits, the West Split of the Strieborná vein, has been intersected by two drill holes. This split is on the northwest side of, and sub-parallel to, the main Strieborná vein. As drilled, it has a maximum strike length of ~100 m, with a maximum vertical extension of ~100 m, and a maximum width of ~5 m. Continued exploration of the West Split will better define its full potential.

Geologic mapping performed by Global Minerals in 2014 on mine Level 6 has been incorporated into the updated geological model. This mapping has allowed for refinement of the vein geometry interpretation in the upper part of the resource.

Figure 7-5: Three-Dimensional Pictorial, showing Propeller Shape of the Strieborná Vein



Note: Looking down, north is up; yellow represents the development workings; gray lines represent drill holes; purple represents the Strieborná vein, cyan represents the Podložná vein, and green represents the West Split of the Strieborná vein; the workings to the right are developed on the un-modelled Pallag vein; red represents estimated material grading ≥ 100 g/t Ag and classified as Measured, Indicated, or Inferred; note the way the Strieborná vein turns over as it strikes south toward the viewer.

Figure 7-6: Three-Dimensional Pictorial of the Veins

Note: Looking Azimuth 210° and down at 5°; yellow represents the development workings; gray lines represent drill holes; purple represents the Strieborná vein, cyan represents the Podložná vein, and green represents the West Split of the Strieborná vein; the workings to the right were developed on the un-modelled Pallag vein; red represents estimated material grading ≥ 100 g/t Ag and classified as Measured, Indicated, or Inferred; note the way the Strieborná turns over as it strikes south toward the viewer.

7.6.2 *Podložná*

The Podložná vein was discovered in a cross-cut driven from Level 13 of the Mária Mine, and lies 150 m northwest of the Strieborná vein. Where intersected on the cross-cut, the vein consists of two parallel vein structures, each 0.7 m to 0.8 m in width, mainly filled with siderite and scarce sulphides. As interpreted, the vein has a strike length of about 980 m, and a vertical extent of about 480 m, based on drifting and historical and recent drilling. The vein strikes 30° to 40°, and dips 80° to 85° southeast. The average width is 2.5 m, the maximum width is 6.2 m, and both grade and width tend to improve towards the southwest, where exploration was discontinued. The Podložná vein extends to a depth of 45 m below Level 13, based on its intersection in DH V-RS-13/32-86.

7.6.3 *Pallag*

The Pallag vein was first intersected at Level 13 after drifting through the Strieborná vein. It is located 250 m east-southeast of the Strieborná vein, and was followed by 430 m of drifting. Its maximum thickness reached up to 3 m, and averages 1.1 m. The vein dips 80° to 85° southeast in the southwestern part, steepens in the central part, and dips 75° to 80° northwest in the northwestern part. Deep sections of the vein were tested by drill holes V-RS-13/21, V-RS-13/22, and V-RS-13/23 to V-RS-13/85 (shown on Figure 7-3, but not labelled), which detected only broken stockwork sections up to 100 m below Level 13. In the upward direction, the vein was followed by drifting at Level 10 for 500 m in total. These workings intersected a non-compact parallel system of two or three veinlets 0.2 m to 0.7 m thick, with siderite fill and localized irregular pods of pyrite and chalcopyrite. Strike and dip are similar to those detected at Level 13, with a dip of 80° to 85° southeast, a strike of 30° to 40°, and an average thickness of 0.7 m. In mineralization, chalcopyrite prevails over tetrahedrite, and pyrite, pyrrhotite, tourmaline, and albite are locally present. Host rocks are porphyroid, gradually changing to sericite phyllites to quartz phyllites. No resources have been estimated for the Pallag vein, as the part of the vein defined to date is low grade; however, it is a very good exploration target.

8 DEPOSIT TYPES

The following information is taken with modifications from Jakubiak (2008).

Strieborná is a hydrothermal-metasomatic, silver- and copper-bearing, siderite plus quartz vein hosted in Early Paleozoic metamorphic rocks. The vein is believed to have formed due to the action of metamorphic fluid phases that attacked older stratiform siderite and transported the released metals as hydrocarbonates and complexes with hydrogen sulphide into open structures. Silver, copper, antimony, mercury, and bismuth are believed to have been derived from black shales, jasper horizons, and metal-enriched precipitates in volcanic complexes. The main minerals of economic interest are the silver–copper–antimony-bearing tetrahedrite and the siderite.

9 EXPLORATION

Because the resource completed for this report depends heavily on the results of historical channel sampling, this sampling, by both the SGS and on behalf of CMX, is described in this section.

9.1 Channel Sampling by Slovak Geological Survey and CMX Resource Ltd.

In 1991 and 1992, the SGS conducted extensive channel sampling of the underground workings, with a sample length generally of 1 m, cut perpendicular to the length of the vein. In places where the vein was wider than the width of the drift, cross-cuts were developed to expose the whole width of the vein. Most of the sampling was taken from Level 8, Level 10, and Level 13.

In 1994, CMX conducted sampling to verify the results of the SGS sampling, collecting two sets of samples: channel and panel. The CMX channel samples were generally 1 m wide, cut perpendicular to the SGS channel samples. The CMX panel samples were cut using a template grid to guide the taking of a series of spot samples at 20 cm intervals. Both the channel and panel samples were centred over the SGS sample intervals.

9.2 Exploration by Global Minerals Ltd.

During the 2011 field season, Global Minerals began drilling on the Strieborná Project, completing six holes from the surface into the upper, northeastern portion of the Strieborná vein, where previous drilling had been limited (Global Minerals news release, 13 June 2011). Three of the holes did not encounter any mineralization. Three holes intersected the Strieborná vein at elevations above Level 6. Global Minerals believes this is the first drilling at Strieborná for which gold was analyzed, although CMX did complete gold assaying on their underground sampling.

Between 2011 and 2013, Global Minerals dewatered and rehabilitated portions of the mine, commissioned compilation of existing regional data, and conducted metallurgical testwork. Metallurgical testing is discussed in Section 13. As part of the dewatering effort, a water treatment plant was installed to treat the water from the mine. Global Minerals plans to continue to dewater the workings, at least to Level 10. As of May 2014, the mine was dewatered to Level 8 or just above it, but rehabilitation of the shaft had not been started and a permit for access had not been issued. In 2012 and 2013, Global Minerals conducted underground exploration core drilling from Level 6 (the uppermost level). This drilling is discussed in Section 10.3.

During 2014, Global Minerals obtained five sets of underground channel samples and five sets of underground panel samples from the Strieborná vein exposed in new drifts

and cross-cuts on Level 6. Sampling, analytical, and QA/QC procedures are given in Section 11 and Section 12.

Figure 9-1: Level 6 after Rehabilitation and Pumping



In addition to the work specifically focused on the two mining licences and the Strieborná vein, Global Minerals has conducted regional exploration on the exploration concessions adjacent to the mining licences that it controls. During the 2011 field season, geochemical and geophysical surveying was conducted in an orientation survey over Strieborná, including mercury-vapor spectrometry of surface soils, resistivity (controlled source audio-frequency magnetotellurics; CSMAT), and induced polarization (IP) geophysical surveying. This work proved to be effective in identifying known veins (Global Minerals news release, 13 September 2011). Subsequent follow-up work was conducted in the Pača area, 3 km southeast of the Strieborná vein, and in the Čučma gold/antimony district, about 4 km to 5 km northeast of Strieborná, which had limited historical gold production dating back to the medieval period, and some antimony production after World War II (Global Minerals news releases, November 27, 2011, May 23, 2012, and May 6, 2013). CSMAT/IP surveys were completed over mercury anomalies. Mercury anomalies were found at both Čučma and Pača. Selected rock samples taken from old dumps in the Čučma district yielded assays of 5.2 ppm to 19.0 ppm Au, and 3.7% to 47.0% Sb.

10 DRILLING

10.1 Summary

Drilling data was a very small part of the exploration data that defined the previous resources at Strieborná. Drill data has gained significance after Global Minerals' 2011 drilling campaign, but still represents a small portion of the total number of samples. A map of the samples and drilling is shown in Figure 10-1, which presents a description of the exploration database.

10.2 Drilling by the Slovak Geological Survey

The following information has been taken from Jakubiak (2008). There are some discrepancies in historical reporting of the number of drill holes; however, all drill holes found, documented, and loaded into the database used for this report are given in Table 10-1. Although the metres of drilling exceed the metres of underground sampling, much of the drill core was not sampled due to being in the unmineralized host rocks.

Table 10-1: Summary Description of the Exploration Database

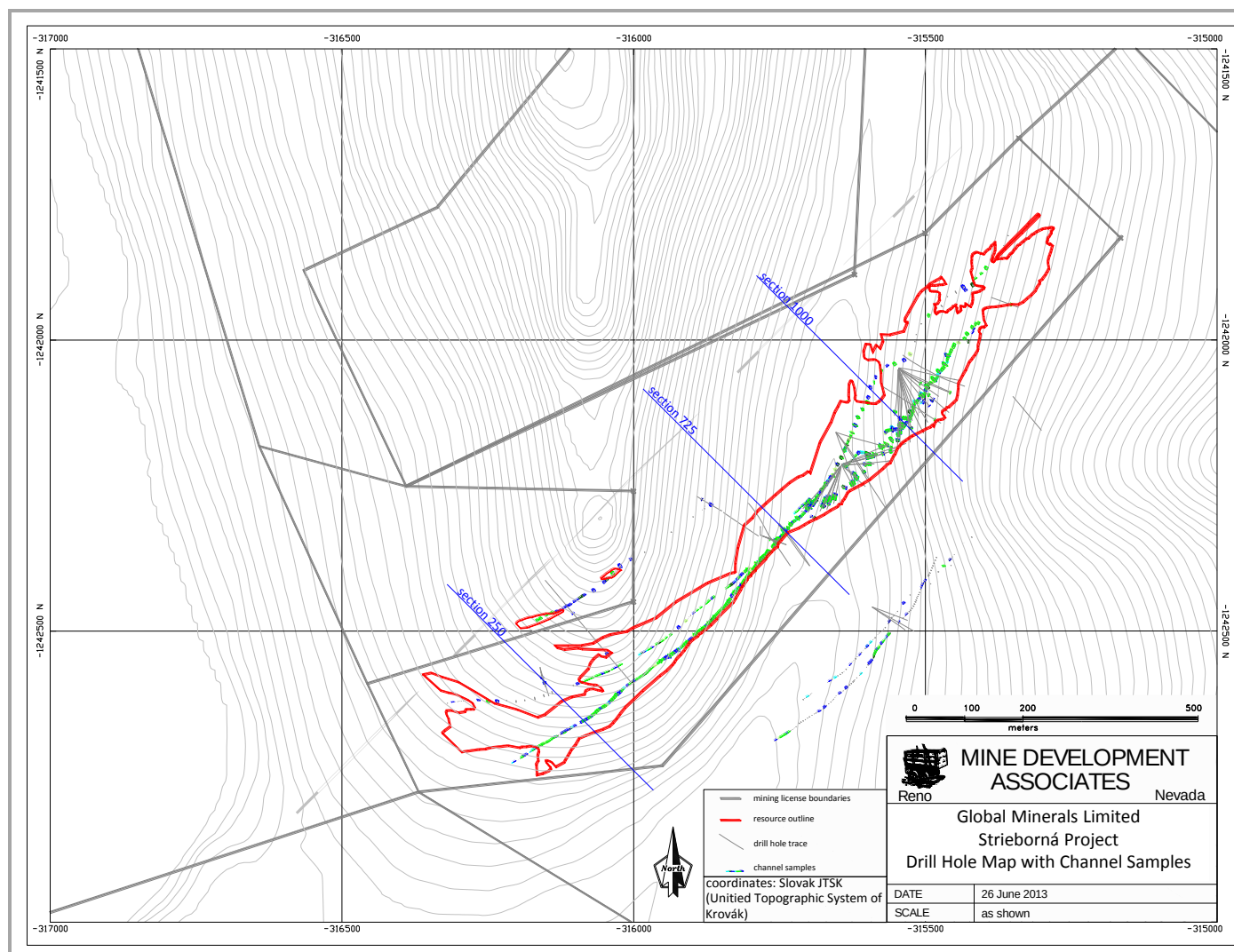
	No. of Holes*	Metres
Underground Sampling	923	1,648.80
Global Minerals' Core Drilling	40	6,043.15
Historical Core Drilling	24	3,089.30
Total	987	10,781.25

Samples	Number	Metres
Historical Core Samples	109	111.9
Global Minerals' Core Samples	559	454.7
Historical Underground Channel Samples	1,094	1,583.3
Historical Underground Panel Samples	25	25.0
Global Minerals' Underground Channel Samples	32	26.4
Global Minerals' Underground Panel Samples	24	22.5
Total	1,843	2,223.7

Note: *Consecutive sequences of underground samples were entered as if they were a drill hole.

The SGS treated drilling as a secondary exploration tool, used only to confirm the predicted locations of the Strieborná vein between, above, and below the underground workings. The SGS drilled 49 holes, including one surface drill hole and 13 underground holes that intersected the Strieborná vein and its subsidiary structures. Of the 49 holes, some were not assayed and/or did not have collar coordinates, and some were not drilled on the Strieborná vein. All holes were diamond drill (core) holes.

Figure 10-1: Strieborná Drill Hole Map



Drilling was done using two core rigs: a Toram 2 x 20, and a Diamec 250. Surface drill hole V-RS-6-83 started with 112 mm core, which was reduced in four steps to 76 mm. All underground drill holes were drilled using 76 mm diameter double core barrels. The core diameter from a 76 mm hole is 48 mm.

Locations of collars were surveyed with a theodolite, and down-hole surveys were recorded with a single-shot instrument at regular intervals ranging from 20 m to 50 m.

Core recovery was generally within a range of 90% to 100%, except for some holes drilled from Level 13 that had recoveries at some levels below 50%. Four holes drilled as a fan on Level 13 had core recoveries ranging from 42% to 68.7%. Drill logs were reported by Jakubiak (2008) to be of high quality and showed core recoveries against drill runs, hole diameter, graphic log, lithological descriptions, structural measurements in relation to core axis, and sample intervals, as well as surveyed collar coordinates and down-hole surveys.

10.3 Drilling by Global Minerals Ltd.

10.3.1 2011 Drilling

Global Minerals began drilling to explore the Strieborná vein in March 2011, completing six holes from the surface for a total of 1,703.4 m. Geo-Technic Consulting spol. s.r.o. (Geo-Technic) from Bratislava, Slovakia, was the drill contractor. A Power 7000 DS drill rig was used (Figure 10-2). The holes were drilled with HQ-size core and reduced to NQ-size only in a few instances.

Figure 10-2: Geo-Technic Drilling Strieborná Vein, 2011 Drill Program



10.3.2 *2012/2013 Drilling*

Underground core drilling began in the fall of 2012, from Level 6 (Figure 10-3). The contractor, Geo-Technic, used a Hagby 1000 core drill. From this level all drilling was angled down; the most shallow-dipping hole was -9°. Geo-Technic drilled approximately one-metre core runs using standard wire-line recovery. The holes were drilled with NQ-size core; a reduction to BQ size was necessary in a few instances.

The following is an outline of the drilling and sampling procedures used underground by Global Minerals at Strieborná:

1. A contracted surveyor sets the drill hole location, foresight, and backsight. The drillers set the dip of the hole using an inclinometer.
2. The core is hauled to the surface, where it is stored and logged within the fenced mine area inside a secured building near the portal of the Mária Mine.
3. Samples are photographed.
4. Core recovery, rock quality designation (RQD), and geotechnical data are logged. The RQD is the sum of all samples whose length is greater than or equal to two times the diameter of the core.
5. Core logging emphasizes lithology, structure, and alteration, but angles to core axes are also logged.
6. Core samples' starting and ending points are determined by geology and mineralization, and are a maximum of one metre long and a minimum of 20 cm long.
7. Samples for density are taken and cut perpendicular to the core.
8. Samples are sawn in half (except for duplicates).

10.3.3 *Drill Hole Collar Surveys*

For Global Minerals' 2012/2013 drilling, Alexander Palcso, a registered independent surveyor, used a Topcon GPC3005 to survey the drill hole collars. In addition to surveying the hole before and after drilling, Mr. Palcso also surveyed a backsight and foresight.

Figure 10-3: November 2012 Underground Drilling (Level 6)



10.3.4 Down-Hole Surveys

For Global Minerals' 2012/2013 underground drill program, a Tropari magnetic compass type survey was made on the first hole. The measurements were so illogical and improbable that the tests were redone with a Russian-made down-hole magnetic compass type instrument and a gyroscopic probe manufactured by Geosignal. As the latter two surveys confirmed each other, the gyroscope was selected for the remaining down-hole survey measurements. The company performing the surveys was Koral s.r.o., of Spišská Nová Ves. Measurements were taken every 10 m to 20 m, with one measurement at the collar to ensure, verify, or correct the collar set up.

Magnetic north is 4.3° east of true north.

10.4 Underground Sampling by Global Minerals Ltd.

Global Minerals performed underground sampling on mine Level 6 in 2014 (Figure 14-1). Five sets of channel samples were taken with azimuths of 146°, 155°, and 159°, approximately perpendicular to the vein. A total of 32 individual channel samples with an aggregate length of 26.4 m were collected. Sixty-six percent of the channel samples were about 1 m in length, and 34% were about 0.5 m in length. Five sets of panel samples were taken from a separate crosscut on mine Level 6 (Figure 10-4), for a total of 24 individual samples representing an aggregate sample length of 22.5 m. Panel samples were taken approximately perpendicular to the vein, and with azimuths of 146°, 155°, and 159°. Eighty-seven and a half percent of the panel samples represent sample lengths

of about 1 m, with the balance representing lengths of about 0.5 m. Sampling procedures are described in Section 11.1.

10.5 Summary

MDA has no direct experience with the SGS or CMX underground sampling, and cannot explicitly verify its accuracy. However, the descriptions of the work and the understanding of the deposit by those who did the work gives significant confidence in the results. Ivan Mesarčík, an independent contractor to Global Minerals, independently supervised the historical sampling.

MDA has observed and reviewed the drilling done by Global Minerals, and has independently sampled the core from their drilling. MDA is satisfied that the Global Minerals work was done in such a way as to produce accurate and reliable results, on which a resource can be defined with classification up to and including Measured.

In areas of sampling, such as along the drifts, and in the area where Global Minerals has recently drilled, the resource is adequately sampled. The remaining parts of the vein will require infill drilling prior to prefeasibility work.

One important point to be made regarding the sampling is that most core drilling intersected the vein in the upper parts of the deposit, and most underground vein sampling was located deeper than the drilling (Figure 14-1). Most core drilling has variable length samples, while most underground samples in the database are one metre long or more. For example, the SGS samples average 1.3 m (after removing excessively long samples), and Global Minerals drill samples average 0.8 m. The consequence of this is that the perspective of the deposit seems different based on each set of samples; it appears that the grades are more regular at depth, and more erratic higher in the system.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sample Procedures

The underground levels in the Strieborná vein have been geologically mapped and sampled in detail by both the SGS and by CMX. The volumetric majority of the data on which the resource estimate has been made is based on this underground sampling. This sampling has been done on levels that are not accessible at the present time, so no confirmation samples by Global Minerals have yet been taken.

11.1.1 *Sampling by the Slovak Geological Survey*

The following information on sampling by the SGS was taken from Jakubiak (2008).

Underground Channel Sampling

Channel sampling spacing along drifts averages about 5 m. The channels generally have a sample length of one metre; the database has several samples that are greater than 2 m long. Channel samples were 20 cm wide and 5 cm deep, and were cut along walls or backs of cross cuts and drifts using a jackhammer. Where the vein width was less than 1 m, both the footwall and the hanging wall were also sampled. In places where the vein was wider than the width of the drift, cross-cuts were developed at around 25 m or less to expose the whole width of the vein. In addition, grab samples were taken for bulk density tests and other special tests, as described in Section 11.2.1. Locations of the SGS samples, often with still-legible painted numbers, were visible during the CMX verification program, and were noted on the CMX geological maps. Jakubiak (2008) considered that the sample spacing was sufficient to determine short-range grade variability.

Field duplicates for the underground channel samples from Strieborná were submitted to the SGS, as mandated by the regulations then applicable.

Drill Core Sampling

All drill intercepts of siderite, siderite-quartz, or quartz veins, with or without sulphides, were marked up for sampling. Adjacent intervals in the hanging wall and footwall of each intersection were also selected for sampling. Drill core was sawn in half along the core axis. One-half core was sent for sample preparation and analysis, and the other was sent to the SGS sample archive warehouse in Spišská Nová Ves.

11.1.2 *Sampling by CMX Resource Ltd.*

In 1994, CMX conducted sampling to verify the results of the SGS. CMX collected two sets of samples: channel and panel. The first method involved cutting 15 cm x 5 cm channels of one-metre length at one-metre intervals, perpendicular to the channel

samples cut by the SGS. These samples are referred to as CMX channel samples. In some instances, several CMX channels were cut side-by-side, one metre apart, on any one channel cut by the SGS. Analytical results for samples representing each vein intersection were averaged to obtain the mean intersection grade, which was compared to the grade of the SGS channel.

The second method was a variation of the panel sampling method. Each sample represented a panel of one metre by one metre, centred on a channel sample cut by the SGS. A template grid consisting of diagonal lines at 20 cm intervals was placed on each sampled panel, and 5 cm-deep cone-shaped spot samples were taken with a moil (pointed chisel-like tool) and hammer from the intersections of the grid lines, and collected in a single bag. This sampling method resulted in approximately 50 small samples being composited into one single sample.

Jakubiak (1994) compared the results of the SGS channel sampling and CMX's channel sampling. He noted that the CMX channel samples returned silver and copper grades higher by a factor of 1.05 to 1.17 than the assays of the corresponding samples from the SGS work. Jakubiak also compared the CMX channel and panel samples, and noted that, excepting one sample, the panel samples returned consistently higher grades for silver, copper, antimony, and mercury. MDA performed its own analysis (Section 12.3).

11.1.3 *Sampling by Global Minerals Ltd.*

Drilling Samples

For the 2011 and 2012–2013 underground drilling, core samples range from a minimum of 20 cm long to a maximum of one metre. The length was determined by geology and mineralization. Samples were sawn in half, except for duplicates, which were quarter-core samples.

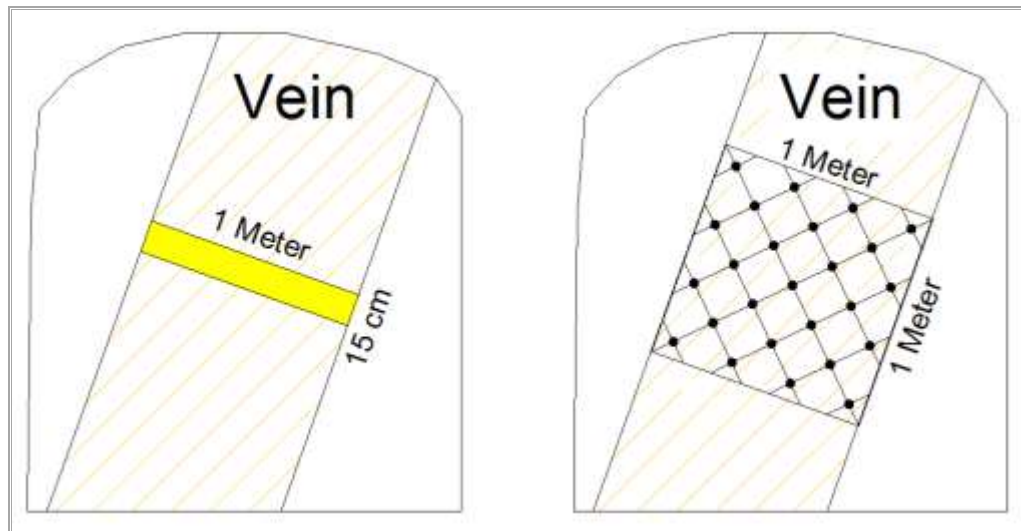
Underground Channel Samples

Channel samples were taken by cutting 15 cm x 5 cm channels of approximately one metre or less in length. The samples were taken perpendicular to the vein contacts, indicating actual vein width when possible (Figure 11-1). When the vein was less than one metre wide, a one-half-metre long channel was collected from the footwall and hanging wall when possible. A tarp was placed on the floor to catch the sample material and to ensure that only material from the channel went into the sample bag. The total sample was then split in half using the cone and quartering sample procedure, yielding two subsamples, each of approximately 2 kg/m sampled. The two subsamples were placed in poly sample bags, one of which was sent to the laboratory for analysis, and the other retained for future reference.

Underground Panel Samples

Panel samples were limited to one metre lengths, defined by a one-metre by one-metre grid consisting of diagonal lines at 20 cm intervals (Figure 11-1), with 10 cm-deep drill holes (approximately 3 cm in diameter) taken from the intersections of the grid lines and collected in a single bag. Only sample intersections within the vein were sampled. A tarp was placed on the floor to catch the sample material to ensure that only material from the panel went into the sample bag. Sample weights were approximately 0.8 kg to 1 kg.

Figure 11-1: Channel Sample (left) and Panel Sample (right) Methods of Global Resources



Source: Global Resources, 2014

Density Measurement Samples

Samples were taken from inside and outside of the veins for density measurements. Samples of drill core approximately 10 cm long were sawn at right angles to the core axis on both ends, making a cylinder. After cutting, the samples were dried and weighed repeatedly until the decrease in weight was negligible. Then five-micrometer measurements were taken of each of the samples' diameters and lengths, and volumes calculated from these measurements. The weights were then divided by that calculated volume to calculate bulk densities.

11.2 Sample Preparation and Analysis

11.2.1 Sampling by the Slovak Geological Survey

The following information on sample preparation and analysis by the SGS was taken from Jakubiak (2008).

Channel and drill core samples were crushed, homogenized, split, and pulverized to 0.09 mm at the Geologický Prieskum laboratory in Spišská Nová Ves (Geologický Prieskum). One split was retained for storage, and the second split was used for

assaying. Crushing was done in stages but without sample volume reduction until the crushed particle size was less than 1 mm.

Samples were analyzed for iron, copper, antimony, bismuth, mercury, silver, manganese, and silica at Geologický Prieskum. Samples were digested using combined and alkaline digestion. The following analytical methods were used: flame atomic absorption (F-AAS), AAS with electrothermal atomization (AAS-ETA), AAS-hybrid generation technique (AAS-TGH), mercury analyzer (AAS-AMA), inductively coupled plasma atomic emission spectrometry (ICP-AES), and x-ray fluorescence (XRF) spectrometry. The detection limits for the key elements are given in Table 11-1.

Table 11-1: Detection Limits for Analyses by the Slovak Geological Survey

Element	Detection Limit (ppm)	Analytical Method
Ag	0.04	AAS-ETA
Cu	0.1	ICP-AES
Sb	0.05	AAS-TGH
Fe	1	F-AAS
Hg	0.01	AAS-AMA

A spectral analysis assay was later added, using composite samples created by mixing channel samples over specific intervals of the underground development workings on either side of the central access cross cuts. Samples from DH V-RVS-7-93 were also composited for the same purpose. These samples were assayed for the above-mentioned elements, and also for arsenic, sulphur, potassium, calcium oxide, magnesium oxide, lead, zinc, and gold.

The SGS determined bulk density on waxed samples, of which 60 samples were from Level 10 and Level 13, and 24 samples were from Level 8. According to Mesarčík et al. (1996, cited by Jakubiak, 2008), bulk density correlates with the iron content; the correlation coefficient is 0.83. The SGS developed the formula based on a linear regression, and used that formula in its tonnage estimates. The formula is:

$$y = 2.5977 + 0.027071 \times \text{Fe (\%)}$$

Using the same samples, the SGS also conducted average moisture content tests. The average moisture content was found to be less than 0.1%.

Samples for special tests were used for thin and polished sections. Samples selected for petrographic studies were also analyzed for lithium, lead, boron, strontium, bismuth, zircon, yttrium, rubidium, silver, and carbon dioxide. Mineralogical samples were analyzed for arsenic, zinc, copper, iron, antimony, silver, bismuth, mercury, lead, gold, nickel, cobalt, and sulphur. Quantitative tests of ore minerals were conducted using a JEOL 733 Superprobe.

11.2.2 *Sampling by CMX Resources Ltd.*

MDA found no information on the sample preparation or analysis methodology used for CMX channel and panel samples.

11.2.3 *Sampling by Global Minerals Ltd.*

Drill samples were shipped to ALS Minerals (ALS) in Roşia Montană, Romania, for sample preparation and analysis. Sample preparation used ALS's Prep-31b procedure: "Log sample into tracking system; weigh; dry; fine crush entire sample to >70% -2 mm; riffle-split off up to 1 kg; pulverize split to >85% -75 µm." Gold was analyzed by fire assay with an atomic absorption (AA) finish on a 30 g nominal sample weight (ALS procedure Au-AA23; limits are 0.005 g Au/t (lower) and 10 g Au/t (upper)). Samples were also analyzed by 34-element ICP (ALS procedure ME-ICP41a) with aqua regia digestion followed by ICP-AES analysis. Silver, copper, and antimony were also analyzed by four-acid digestion with an ICP-AES or AA finish (ALS procedure OG62 Ag, Cu, Sb).

The 2014 channel and panel samples were also shipped to ALS in Rosia Montana, Romania, for sample preparation. Samples were fine-crushed in their entirety to >70% ≤2 mm, followed by riffle splitting and pulverizing of 1 kg to >85% ≤75 µm. Sample pulps were analyzed by ALS in their Loughrea, Ireland, laboratory by ICP-AES using aqua-regia and four-acid digestions (ALS procedures ME-ICP41a, ME-OG46, Cu-OG46, Ag-OG46, and ME-OG62). Pulps were also analyzed for silver by fire-assay with gravimetric finish at ALS Vancouver.

11.3 **Sample Security**

11.3.1 *Sampling by the Slovak Geological Survey*

In its programs, the SGS followed security protocols that restricted access to drill sites and underground workings to authorized personnel only, and required that all sample duplicates be delivered for storage at the SGS's sample storage warehouse at Spišská Nová Ves. Jakubiak (2008) reported that the duplicate samples have since been transferred to a storage facility at the technical University in Košice.

11.3.2 *Sampling by CMX Resource Ltd.*

MDA has no information on sample security measures used by CMX.

11.3.3 *Sampling by Global Minerals Ltd.*

For samples from their 2011 and 2012–2013 drilling campaigns, Global Minerals stored intervals with veining (plus several metres on either side) in a secure building near the portal of the Mária Mine. For their 2012–2013 drilling, the underground core was hauled to the surface where it was stored and logged within the fenced mine area, inside a secure building near the portal of the Mária Mine. Channel and panel samples taken

from mine Level 6 in 2014 were stored in the locked core shack prior to shipment to the ALS in Rosia Mantana, Alba, Romania.

11.4 Quality Assurance/Quality Control

11.4.1 Sampling by the Slovak Geological Survey

Jakubiak (2008) reported that internal quality assurance/quality control (QA/QC) analyses were done for approximately 11% of the samples analyzed by the SGS. External analyses were done on approximately 6% of the samples, in Turčianske Teplice, and at the Geoenvironmental Laboratories at Spišská Nová Ves. The SGS used its own system to analyze results of control analyses for various elements. Allowed tolerances varied up to 10% of the original analysis, depending on the analyzed element and its content. About 20% of iron analyses exceeded the allowed tolerances. Silver, copper, and antimony analyses exceeded the allowed tolerance limits only at very high concentration levels (>1,500 g/t Ag, >5% Cu, >4% Sb). For instance, sample SZJ-8/2C from Level 8 originally yielded 1,565 g/t Ag. Internal and external control analyses gave 1,430 g/t and 1,454 g/t, respectively. The allowed tolerance was 100 g/t. Mercury analyses were all within the allowed tolerance limits.

11.4.2 Sampling by Porcher Island Gold

Porcher Island Gold took 32 check samples on sample splits from Level 8 channel samples, which were submitted to ALS Chemex Labs Ltd. (now ALS Minerals), Vancouver, British Columbia. MDA did not have these results.

11.4.3 Sampling by CMX Resource Ltd.

MDA has no information on QA/QC measures used by CMX for their channel and panel sampling. The AMC Technical Report (Jakubiak, 2008) stated the following:

Madeisky and Chernavska, who visited the underground workings in the Strieborná Vein in 1998, submitted splits of original channel sample rejects for re-analysis. Except for sample SZ-8B, results of those analyses (Table 11-2) corroborated the original analytical results reported by the Geological Survey (Madeisky and Chernavska, 1998). Omitting this sample, the Chemex analyses average 872 g/t Ag and 3.42% Cu, whilst the Geological Survey's analyses for the same samples average 851 g/t Ag and 3.76% Cu, respectively.

Table 11-2: Comparison of the SGS (1996) and ALS Chemex (1998) Analyses

Sample Number	1996 SGS Ag, ppm	1998 Chemex Ag, ppm	1996 SGS Cu, %	1998 Chemex Cu, %
SZ-8A	234	274	0.910	0.991
SZ-8B	319	148	1.253	0.532
SZJ-8/2A	321	294	1.365	1.142
SZJ-8/2C	565	1,750	7.350	7.040
SZJ-8/5E	1,099	1,030	4.596	3.980

SZJ-8/5F	1,037	1,010	4.579	3.930
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Note: Analyses for Channel Samples from Level 8 above 210 g/t Ag cut-off grade

The Madeisky and Chernavska (1998) verification sampling does not give a measure of the total variability, this being the sum of sampling, sub-sampling, and analytical variability. The ALS results on crushed sample splits give a measure of the combined sub-sampling and analytical variability. A set of six samples is not sufficiently large to fully assess that variability, but the results shown above do give a measure of confidence in the SGS's sampling.

11.4.4 Sampling by Global Minerals Ltd.

Standards, blanks, and duplicates were inserted into the sample stream for the 2011 and 2012–2013 drilling samples. Standards and duplicates were also inserted into the sample sequence for the 2014 channel and panel samples.

Global Minerals used pulp standards made by WCM Sales Ltd. of Burnaby, BC. Four WCM standards were inserted for the 2011 and 2012–2013 drilling:

BL 112	Au ppb	Ag ppm	Cu ppm	Mo ppm	Pb ppm	Zn ppm	Ni ppm	Co ppm	W ppm
Lab's Recommended Average	<5	<0.3	4	<1	4	20	1	2	<10

CU 112	Cu %	Mo %	Ag g/t
Lab's Recommended Average	0.91	0.017	359

CU 163	Cu %	Mo %	Ag g/t	Au g/t
Lab's Recommended Average	1.06	0.156	99	4.35

CU 164	Cu %	Mo %	Ag g/t	Au g/t
Lab's Recommended Average	0.31	0.081	29	1.14

Standard BL 112 was also inserted with the 2014 channel and panel samples, along with WCM standards PM 1123, PM 1145, and PM 1147. The standard insertion rate was 6% for the 2014 channel samples, and 17% for the 2014 panel samples.

PM 1123	Cu %	Ag g/t	Au g/t
Lab's Recommended Average	0.31	31	1.42

PM 1145	Cu %	Ag g/t
Lab's Recommended Average	2.49	811

PM 1147	Cu %	Ag g/t	Au g/t
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Lab's Recommended Average	0.31	226	1.12
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Blank samples were either fine-grained quartz sand, or about 1 cm-sized limestone gravel. Both products were bought from suppliers. Sample sizes were greater than 1 kg and less than 2 kg. Duplicate drilling samples consisted of quarter-core sawed splits. Channel and panel sample duplicates were made by splitting each sample into two duplicate halves.

11.5 Summary Statement

MDA speculates panel sampling may be the best type of sampling for this deposit, since the tetrahedrite distribution has significant structural control, including occurrences as ladder veining, but panel sampling is the least common at Strieborná. Otherwise, MDA believes the sample preparation, security, and analytical procedures are adequate to support a mineral resource estimate.

12 DATA VERIFICATION

MDA could only view and verify Global Minerals' sampling; MDA is relying on the independent sampling performed by others, with whom MDA has no personal relationship. Jakubiak (1994) did an extensive mapping and sampling program on behalf of CMX. Madeisky (accompanied by others) examined the Strieborná vein at Level 8 in March 1998, and reported that "The workings are in very good condition, there is no muck on the floors, and the backs have held up very well" (Madeisky and Chernavska, 1998).

12.1 Database Audit

12.1.1 Drill-Collar Audit

During their 2012 and 2013 site visits, MDA took three GPS readings of surface drill sites to check for reasonableness of location. The original readings were in latitudes and longitudes presumed to be using WGS84 datum. The locations were transformed into South-Oriented Transverse Mercator using the S-JTSK datum (S-JTSK is a plane geodetic grid coordinate system called in Czech "System jednotné trigonometrické síť katastrální—translates as The System of the Unified Czech/Slovak Trigonometric Cadastral Net"). The results show reasonableness, although with a slight possible shift to the east. The comparison of these GPS-surveyed points to the database is acceptable (Table 12-1), considering the lack of precision of a hand-held GPS unit.

Table 12-1: Comparison of MDA Check of Drill Hole Locations to the Database

GPS Coordinates		Converted		Difference		Global Minerals Database	
Longitude	East	East	North	East	North	East	North
48.67286	20.54248	-315658.31	-1242161.29	-4.76	-2.74	-315653.55	-1242158.55
48.67366	20.54359	-315571.72	-1242077.05	1.87	2.57	-315573.59	-1242079.62
48.67416	20.54394	-315542.87	-1242022.99	-4.07	2.82	-315538.80	-1242025.81

12.1.2 Down-Hole Survey Audit

MDA audited the data from 28 of Global Minerals' down-hole surveys and found one trivial 100th of a degree error.

12.1.3 Geological Data Audit

The geological data entered into the database are only a code for whether a sample lies within a vein and which vein, and which vein it is, if it was taken from within a vein. While no "audit" per se was done, working with the data while modelling found them to be reasonable.

12.1.4 Assay Database Audit

The database was audited in three parts: historical data, surface drilling data, and underground drilling data.

Manual Audit of Historical Assays and Sample Locations

A manual audit was done on a large percentage of the SGS data. The errors found were minor and non-material, and were corrected.

Manual Audit of Global Minerals' Surface Drilling Assays

MDA compared analytical data from laboratory certificates to the Strieborná database and found no significant errors.

Automated Audit of Global Minerals' Underground Drilling Assays

MDA compiled laboratory assay sheets and compared results with the Strieborná database. No errors were found. No audit was made on the recently entered CMX check-sample data.

12.2 Specific Gravity Data Verification

MDA is only using the Global Minerals density measurements. No checks were done on these, but the procedures are sufficiently simple and the internal checks (multiple micrometer readings, reasonableness, drying) validate the results.

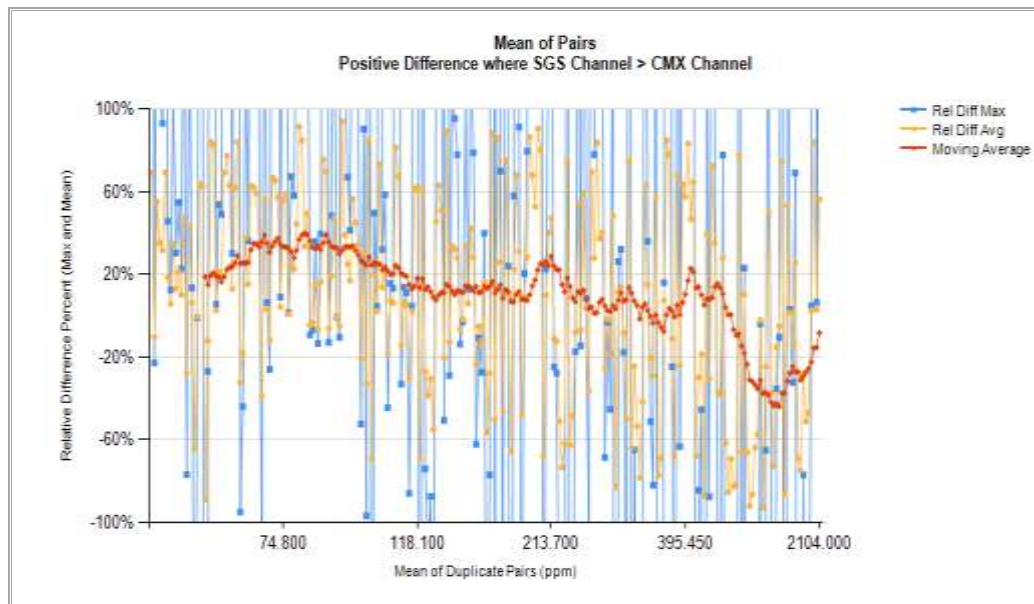
12.3 Channel Sample Data Verification

Jakubiak (1994) compares the the SGS channel sampling with the CMX channel and panel sampling. Jakubiak concludes, "These (panel) samples returned grades up to several times higher than the corresponding channel samples indicating that the entire assay database is suspect." Jakubiak (2008) later states, that "Panel sampling results were broadly comparable to the CMX channel sampling results and suggested that panel sampling was better at defining the grade on Level 13 and in parts of Level 10 where tetrahedrite occurs as scattered blebs and pockets in siderite. On the whole, the results of the CMX sampling indicated that the SGS sampling results tend to understate silver, copper, antimony, and mercury grades by at least 10%."

In part due to those conflicting statements, MDA has done its own comparison of the SGS and CMX sampling. MDA agrees with Jakubiak that panel samples, if done properly, should be more representative than channel samples. MDA compared CMX channel samples to panel samples, and found that at low grades panel samples were only slightly higher grade than the corresponding channel samples. However, at higher grades, panel samples are substantially higher grade than channel samples, in the order of around 50%. As a consequence, MDA concentrated on the review of the SGS channel sample grades compared to CMX channel samples.

The comparisons between the SGS channel sample grades and CMX channel sample grades showed that, at the low grades, approximately below 200 g/t Ag, the SGS samples were higher grade, but at the higher grades, over around 400 g/t Ag, CMX samples were much higher grade (Figure 12-1). Variability in grades is high, and that high variability would be expected for duplicate channel sampling.

Figure 12-1: Channel Samples by the SGS Compared to Channel Samples by CMX – Silver Grade



The biases between CMX and the SGS data are unexplained, and may have been caused by sampling or sub-sampling procedures or analytical bias. The CMX work substantially supports the use of the SGS data in a resource estimate, but also imparts sufficient unknowns and demonstrated bias such that no Measured material is based on the SGS data alone.

12.4 MDA Independent Verification of Mineralization

MDA took four independent core samples. Three were from pre-existing samples, and one was from STR-06-13-30, which had not yet been cut. Six pulp samples were taken. All samples were relabelled to numbers unknown to Global Minerals, but the samples were left with Global Minerals to send to ALS. The results of that sampling (Table 12-2) verify the general tenor of the mineralization, which is represented in the Strieborná database.

Table 12-2: Results of MDA Independent Samples

Global Number	Hole Number	From (m)	To (m)	Global (g/t Ag)	diff. %	MDA (g/t Ag)	Name/Type
NEW	STR-06-13-22	95.32	96.00	75.00	16	87.00	MDA-1/core
42287	STR-06-13-08	138.20	138.95	48.00	-38	30.00	MDA-2/core

NEW	STR-06-13-30	68.00	69.00	582.00	-35	380.50	MDA-3/core
42259	STR-06-13-07	142.53	143.36	66.00	-14	57.00	MDA-4/core
J042272	STR-06-13-07	148.77	149.65	127.00	6	135.00	MDA-5/pulp
J042274	STR-06-13-07	151.65	152.65	4.00	25	5.00	MDA-6/pulp
J042282	STR-06-13-08	134.20	135.20	53.00	0	53.00	MDA-7/pulp
J042283	STR-06-13-08	135.20	136.15	3.00	0	3.00	MDA-8/pulp
J042284	STR-06-13-08	136.15	136.45	61.00	0	61.00	MDA-9/pulp
J042285	STR-06-13-08	136.45	137.20	11.00	-95	0.50	MDA-10/pulp
				% Cu		% Cu	
NEW	STR-06-13-22	95.32	96.00	0.31	12	0.350	MDA-1/core
42287	STR-06-13-08	138.20	138.95	0.19	-33	0.127	MDA-2/core
NEW	STR-06-13-30	68.00	69.00	2.36	-35	1.535	MDA-3/core
42259	STR-06-13-07	142.53	143.36	0.28	-14	0.236	MDA-4/core
J042272	STR-06-13-07	148.77	149.65	0.52	5	0.542	MDA-5/pulp
J042274	STR-06-13-07	151.65	152.65	0.09	11	0.104	MDA-6/pulp
J042282	STR-06-13-08	134.20	135.20	0.20	4	0.208	MDA-7/pulp
J042283	STR-06-13-08	135.20	136.15	0.01	8	0.013	MDA-8/pulp
J042284	STR-06-13-08	136.15	136.45	0.24	4	0.253	MDA-9/pulp
J042285	STR-06-13-08	136.45	137.20	0.05	-92	0.004	MDA-10/pulp
				% Sb		% Sb	
NEW	STR-06-13-22	95.32	96.00	0.23	11	0.250	MDA-1/core
42287	STR-06-13-08	138.20	138.95	0.14	-35	0.091	MDA-2/core
NEW	STR-06-13-30	68.00	69.00	1.76	-36	1.125	MDA-3/core
42259	STR-06-13-07	142.53	143.36	0.20	-15	0.170	MDA-4/core
J042272	STR-06-13-07	148.77	149.65	0.39	3	0.403	MDA-5/pulp
J042274	STR-06-13-07	151.65	152.65	0.01	-17	0.010	MDA-6/pulp
J042282	STR-06-13-08	134.20	135.20	0.15	6	0.155	MDA-7/pulp
J042283	STR-06-13-08	135.20	136.15	0.01	-11	0.008	MDA-8/pulp
J042284	STR-06-13-08	136.15	136.45	0.18	3	0.184	MDA-9/pulp
J042285	STR-06-13-08	136.45	137.20	0.04	-93	0.003	MDA-10/pulp

12.5 2011 and 2012/2013 Drilling QA/QC

12.5.1 Blanks

Global Minerals inserted 22 blank samples into their sample stream. The blanks consisted of either sand or crushed limestone. Neither would have been blind to the lab. Graphs of these blanks in relation to the previous sample were made and reviewed. The results showed contamination in the sample preparation was insignificant.

12.5.2 Standards

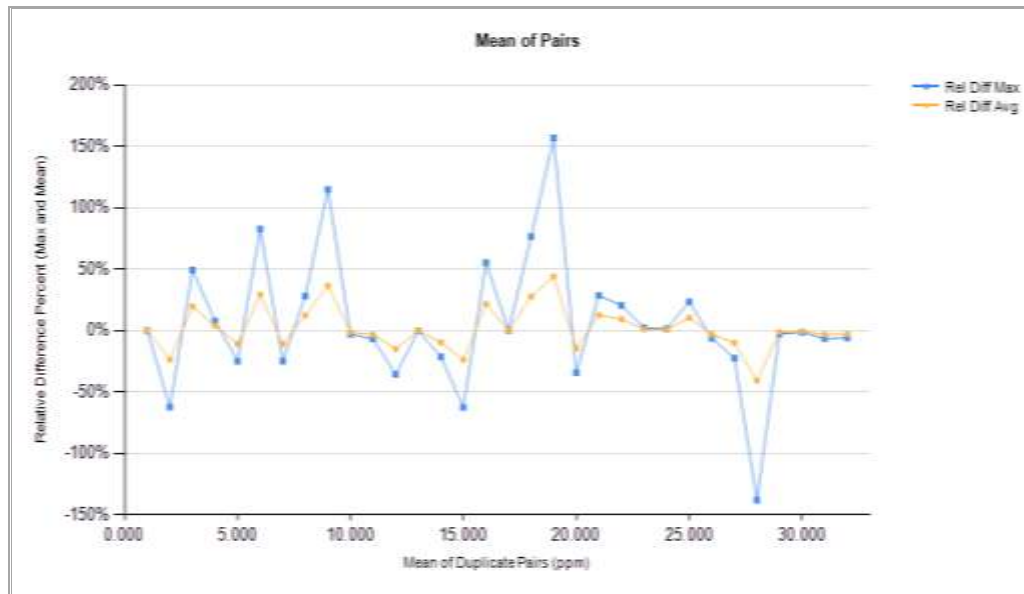
Global Minerals sent 33 standard pulp samples to the lab. The standards' grades are reported in Section 22.4.4. Graphs of each metal's standard grades in relation to the standard grade means, and two standard deviations, were made and reviewed. There

were no failures for copper and silver. There were no standards for mercury or antimony.

12.5.3 Duplicates

Global Minerals sent 32 core duplicate pulp samples to the laboratory. Analyses were completed for silver, copper, and antimony. Although there were expected high differences, averaging around 40%, there was no bias. Figure 12-2 presents the results of the duplicate sample analysis for silver; copper, and antimony have similar results.

Figure 12-2: Global Minerals Duplicate Sample Silver Grades on the Strieborná Vein



12.6 QA/QC for 2014 Channel and Panel Samples

Six pulps of standards and two field duplicates were inserted with the 2014 channel and panel samples, for a QA/QC sample insertion rate of 14%. There were no failures of standards in the 2014 sampling. The original and duplicate panel samples agreed well, with a difference of approximately 5% more silver, copper, and antimony in the original compared to the duplicate. However, the duplicate channel samples contained approximately 125% more silver, copper, and antimony compared to the original.

12.7 Summary Statement on Data Verification

MDA finds the Strieborná sampling and database suitable for use in this resource estimation. They are reliable, and, where verified, accurate. Because MDA has no way to verify the results of historical sampling, no Measured resources can be based on historical results alone. However, documentation and reported work procedures indicate that historical work was done to high quality.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Metallurgical testwork was undertaken by various laboratories on a sample of coarse assay rejects described by Global Minerals as being representative of the resource's average grade. These testwork programs can be divided into three distinct groups: mineral processing, pyrometallurgy, and hydrometallurgy.

Blue Coast Research (BCR) primarily conducted the mineral processing program, with complementary work conducted by SGS United Kingdom Ltd. (SGS-UK). These programs consisted of sample preparation, conventional bench-top flotation tests, and preliminary comminution studies. The objective of this testwork program was to develop and validate a conventional flotation flowsheet that produces a good grade silver concentrate at acceptable silver recovery.

Xstrata Process Support (XPS), Sudbury, ON, and Kingston Process Metallurgy Inc. (KPM), Kingston, ON conducted pyrometallurgical roasting tests. The work investigated the removal of mercury by volatilization from Strieborná concentrate by roasting at medium temperatures. The test program was carried out in two phases: Phase I, Initial tests on the small samples then available; Phase II, Investigation of the influence of time and temperature on mercury volatilization with slightly larger samples. This work showed that mercury could be successfully removed by volatilization.

Hydrometallurgical work was completed at the University of British Columbia (UBC), and at SGS Minerals Services (SGSM) in Lakefield, ON. For the hydrometallurgical option, a series of caustic sulphide leach tests were conducted with the objective of achieving high extraction of antimony and mercury from the flotation concentrate, therefore producing a high-grade concentrate with reduced penalty elements.

To complement the mineral processing work, a mineralogical analysis was undertaken by Process Mineralogical Consulting Ltd. (PMC), Maple Ridge, British Columbia. Two mineralogical studies were conducted: a modal balance of the metallurgical feed sample focusing on tetrahedrite deportment, and a deportment of mercury minerals found in the flotation concentrate.

Chris Martin, CEng., has overseen the mineral processing, mineralogical, and hydrometallurgical aspects of the program.

Philip Mackey, P.Eng., supervised and managed the pyrometallurgical aspects of this program.

13.2 Historical Testwork

No historical test data has been made available to the authors specific to the Strieborná Deposit. However, a series of hydrometallurgical research papers (Anderson, C.G., et al., 1993; Anderson, 2003; Anderson, C.G., et al., 2008), for the most part written by Dr. Corby Anderson, discuss the theory and testwork results for an alkali leach process to remove and recover antimony and mercury from a tetrahedrite flotation concentrate similar to that produced recently by BCR. The papers discuss the methods developed for the Sunshine Mine in Idaho, and its possible development for the use in the Rožňava (Strieborná) deposit.

The alkali leach process conducted at the Sunshine Mine (the “Sunshine Process”) was in operation from 1942 until 1999. In the Sunshine Process, a tetrahedrite concentrate is subjected to a caustic sulphide leach, which produces an antimony-rich solution and a copper/silver sulphide residue. The solution and residue are separated, the residue is sent to the silver smelter, and the solution undergoes electrowinning where antimony metal is produced (Anderson, C.G., et al., 1993).

Unlike the Sunshine Mine, the Strieborná deposit also contains high mercury grades. Dr. Anderson conducted tests using conditions similar to that used in the Sunshine process. In Dr. Anderson’s tests, the flotation concentrate was leached for 12 hours at 105°C, using 20 g/L S and 35 g/L NaOH in a 100 g/L mixture of flotation concentrate. Table 13-1 summarizes the results; removal of 95% Sb and 95% Hg was achieved from a concentrate containing 17.5% Sb and 0.55% Hg. No silver was lost to the in the leach process and only 20% of the gold was lost to the leach solution (Anderson, C.G., 2003).

Table 13-1: Feed and Residue of Alkali Leach Test

	Grade						
	Sb %	As %	Hg %	Cu %	Fe %	Ag g/t	Au g/t
Feed	17.3	1.85	0.55	26.1	12.4	6,400	2
Residue	0.87	0.74	0.03	26.1	12.4	6,400	1.6

Dr. Anderson also discusses some methods for recovery of mercury, antimony, and gold. To electrowin the antimony, the mercury must first be removed. Dr. Anderson suggests that mercury could be removed by cementation to produce mercury metal, or precipitated via pressure oxidation to produce a stable mercury sulphide. The excess gold in solution can then be recovered by an array of means, including electrowinning, gaseous precipitation, chemical precipitation, cementation, solvent extraction, or ion exchange (Anderson, C.G., 2008).

After removal of gold and mercury, the antimony can then be electrowon or precipitated.

Dr. Anderson describes that mercury precipitate—a stable mercury sulphide or metal—can be blended with stabilization cements and can be disposed of safely.

As with past operations at the Sunshine Mine in Idaho, antimony cathodes could be sold as a metallic product.

13.3 Recent Testwork

13.3.1 Sample Preparation

Approximately 123 kg of Strieborná coarse assay rejects were received by BCR from ALS Chemex, North Vancouver, BC, in late July 2011. These samples were inventoried and weighed. The “as received” samples produced a weighted, combined average of 1.60% Cu, 1.40% S, 1.15% Sb, 0.15 g/t Au, 381 g/t Ag, and 0.08 g/t As.

It was decided to produce a metallurgical master composite of approximately 85 kg in mass that matched the team’s best forecast of the average mineable grade of the Strieborná resource. Table 13-2 describes composite proportions from the “as received” samples.

Samples were selected, crushed, and blended, and a representative sub-sample of this master composite was taken and submitted for chemical assay at SGS Minerals in Vancouver. Triplicate samples were taken in order to produce a robust estimate of the mean head grade (silver, sulphur, antimony, and copper) for the master composite.

The chemical assay results show the blended metallurgical sample (STB master composite) has an average head grade of ~270 g/t Ag, 1.1% S, 1.1% Cu, and 0.8% Sb. It should be noted that the ratios of silver:copper and silver:antimony in the metallurgical master composite are slightly different than the ratios observed in the resource estimate forecast. This suggests that the tetrahedrite in the testwork sample is of slightly higher silver grade than that observed in the overall resource.

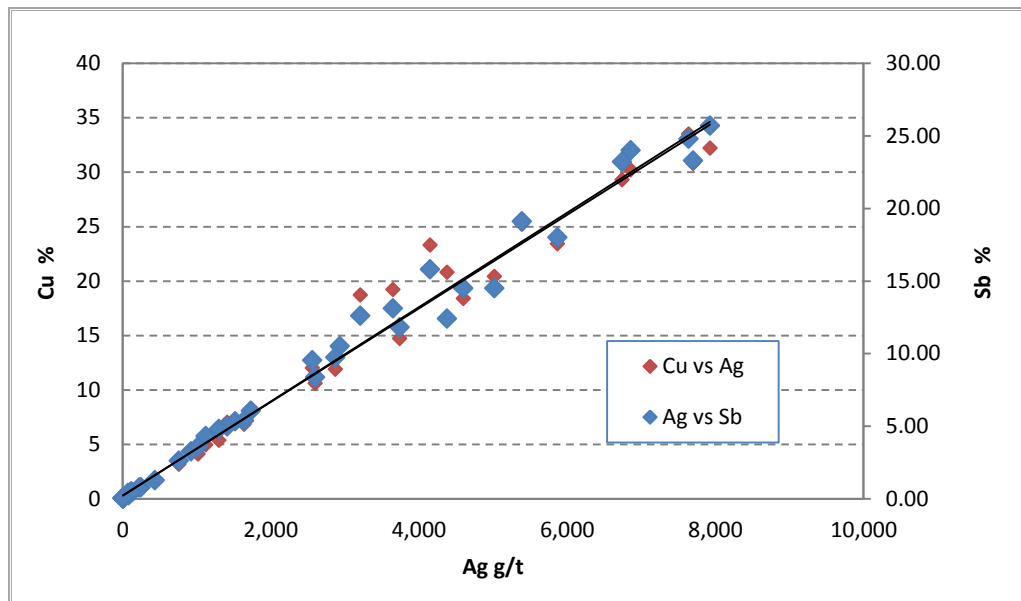
An investigation into possible correlations between elements was performed, and the scatter plots in Figure 13-1 were created to show the relationships between copper:silver and silver:antimony grades in the coarse assay rejects received from ALS Chemex.

The strongly proportional relationship between copper and silver is not unexpected, as the Strieborná samples are believed to contain complex tetrahedrite minerals, with silver fractionally substituted for copper. Thus, the majority of the copper, silver, and antimony within this deposit are found within the same single mineral.

Table 13-2: Metallurgy Composite Recipe “as received” Sample Numbers and Assays

Sample ID	Weight (g)	Au g/t	Ag g/t	As %	Cu %	S %	Sb %
JO42052	2,000	0.01	19	0.031	0.141	1	0.082
JO42053	3,157	0.09	192	0.031	0.804	0.71	0.586
JO42054	2,500	0.28	740	0.141	2.98	2.15	2.21
JO42055	3,359	0.05	111	0.007	0.469	0.49	0.35
JO42056	3,091	0.02	53	0.002	0.229	0.18	0.167
JO42057	4,000	0.09	259	0.02	1.15	0.88	0.86
JO42058	250	1.18	3,530	1.02	16.75	10	12.5
JO42059	2,000	0.1	271	0.044	1.105	1.1	0.811
JO42060	2,500	0.14	375	0.041	1.505	1.33	1.11
JO42062	2,000	0.03	69	0.008	0.285	0.66	0.206
JO42068	2,000	0.01	2	0.027	0.011	0.23	0.008
JO42071	3,000	0.15	362	0.146	1.715	1.94	1.225
JO42072	3,500	0.04	94	0.013	0.428	0.34	0.317
JO42073	1,768	0.09	223	0.04	0.978	0.72	0.724
JO42075	4,000	0.1	248	0.053	1.075	0.79	0.805
JO42076	4,000	0.11	252	0.053	1.11	1.25	0.826
JO42077	2,000	0.01	12	0.01	0.114	0.96	0.02
JO42078	2,000	0.01	9	0.012	0.067	0.47	0.021
JO42088	3,000	0.01	29	0.004	0.112	0.1	0.084
JO42090	2,000	0.15	442	0.058	1.845	1.71	1.36
JO42091	285	0.45	1,230	0.322	5	4.21	3.5
JO42092	268	1.8	4,520	0.669	17.9	10	11.75
JO42094	3,894	0.09	167	0.073	0.673	1.61	0.488
JO42095	3,500	0.03	55	0.006	0.222	0.16	0.169
JO42096	3,500	0.04	102	0.008	0.413	0.32	0.312
JO42097	2,500	0.23	533	0.199	2.29	1.79	1.685
JO42099	4,010	0.03	75	0.007	0.32	0.34	0.235
JO42100	1,025	0.26	700	0.138	2.85	3.08	2.1
JO42101	261	1.28	3,380	0.713	13.8	10	9.09
JO42102	1,012	0.31	721	0.2	3.17	2.5	2.38
JO42103	4,035	0.1	228	0.062	0.979	1.02	0.719
JO42105	3,028	0.03	47	0.003	0.193	0.13	0.147
JO42106	498.5	0.37	1,010	0.088	4.02	3.2	3
JO42107	3,004	0.11	278	0.025	1.065	0.83	0.792
JO42108	3,000	0.01	4	0.005	0.044	0.44	0.024
	85,945.9	0.10	238.9	0.051	1.013	0.978	0.737

Figure 13-1: Relationship between Silver:Copper and Silver:Antimony Head Grades



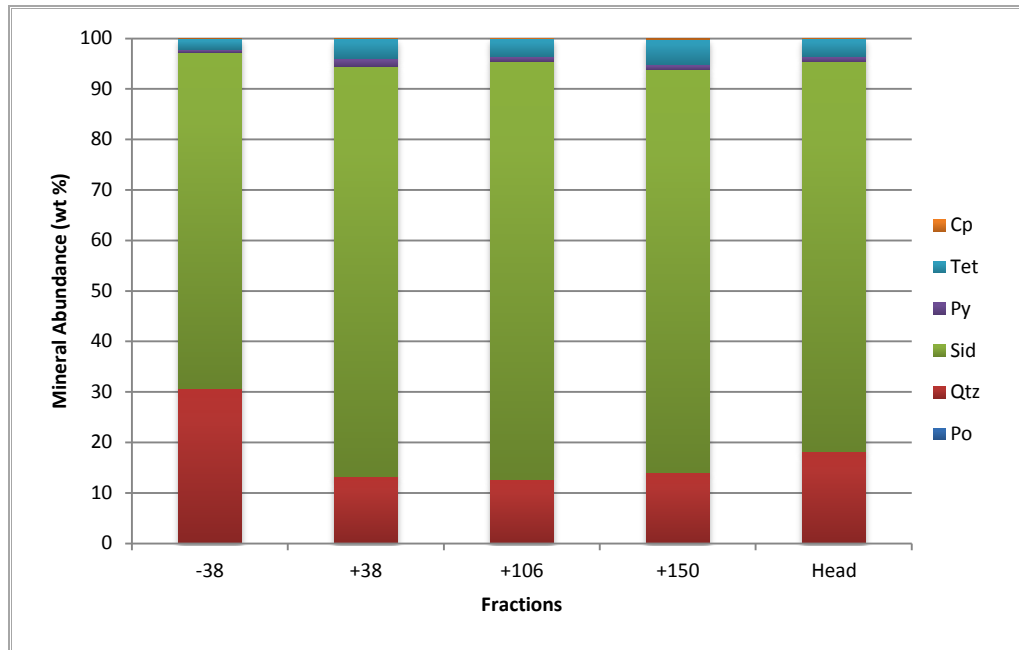
13.3.2 Mineralogy

Two mineralogical samples were studied at PMC. The master composite (STB MC) was sent for copper department, which also included a full modal analysis. The second sample was a flotation concentrate, which was used for mercury department. The flotation concentrate was used rather than the master composite for two reasons; statistically there are more grains of mercury in flotation concentrate, and mercury is only of concern in the leach or roaster feed (flotation concentrate).

Modal Mineralogy

A representative sample of STB MC was screened at 150 μm , 106 μm , and 38 μm to produce four size fractions. These four size fractions were mounted into polished sections. Size-by-size modal mineralogy was then conducted on these size fractions using image analysis in conjunction with a scanning electron microscopy (SEM). The results of this analysis are summarized in Figure 13-2.

Figure 13-2: Master Composite Modal Analysis



Mineralogical analysis showed that the STB MC sample is dominated by siderite (Sid), representing 77.2% of the sample by mass, with 18.2% quartz (Qtz) and 3.5% tetrahedrite (Tet). Chalcopyrite (Cp) and pyrite (Py) were also observed in lesser quantities. In addition to this information, semi-quantitative XRD Rietveld analysis (SQ-XRD) was undertaken on an un-sized sub-sample of the STB MC to provide a further estimate of modal mineralogy. Table 13-3 summarizes the results of the SQ-XRD analysis.

Table 13-3: SQ-XRD Analysis of STB MC

Mineral	Ideal Formula	Distribution (%)
Siderite	Fe_2CO_3	72.6
Tetrahedrite	$(\text{Cu, Fe, Ag, Zn})_{12}\text{Sb}_4\text{S}_{13}$	2.2
Muscovite	$\text{KAl}_2(\text{AlSi}_3\text{O}_{10})(\text{OH})_2$	3.3
Quartz	SiO_2	21.3
Pyrite	FeS	0.6
Total		100

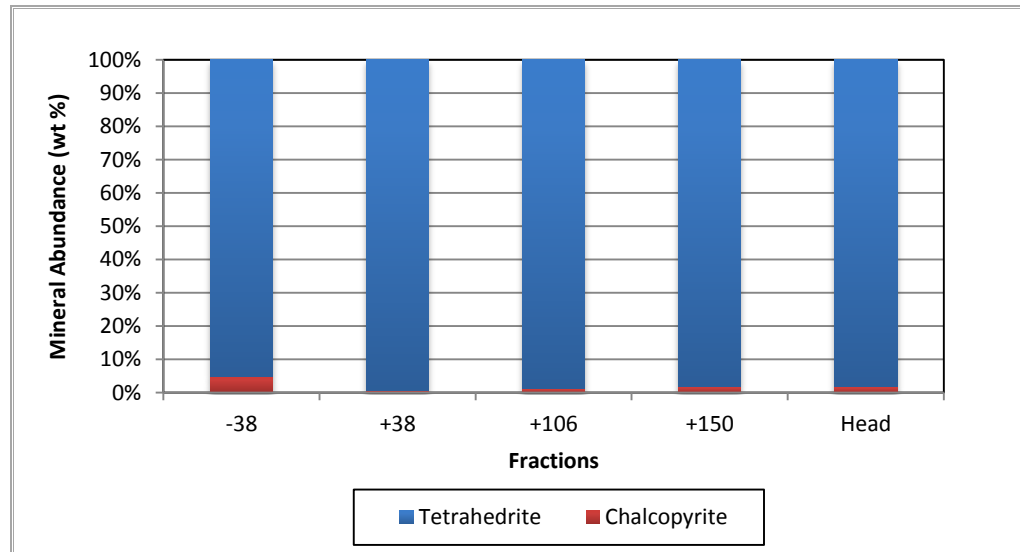
Siderite was also found to be the dominant mineral present (72.6%), followed by quartz (21.3%), muscovite (3.3%), tetrahedrite (2.2%), and pyrite (0.6%).

Mineral Composition Analysis and Elemental Department

Twenty-two tetrahedrite grains were selected for semi-quantitative energy dispersive spectrometry (SQ-EDS) analysis to determine the mineral composition of each grain. On

average, the tetrahedrite grains identified are composed of 27.2% S, 6.8% Fe, 34% Cu, 1.9% Ag, and 30% Sb. Using these values along with the composition of chalcopryite it is possible to calculate the copper deportment in the STB MC (Figure 13-3).

Figure 13-3: Normalized Copper Elemental Deportment in the STB MC



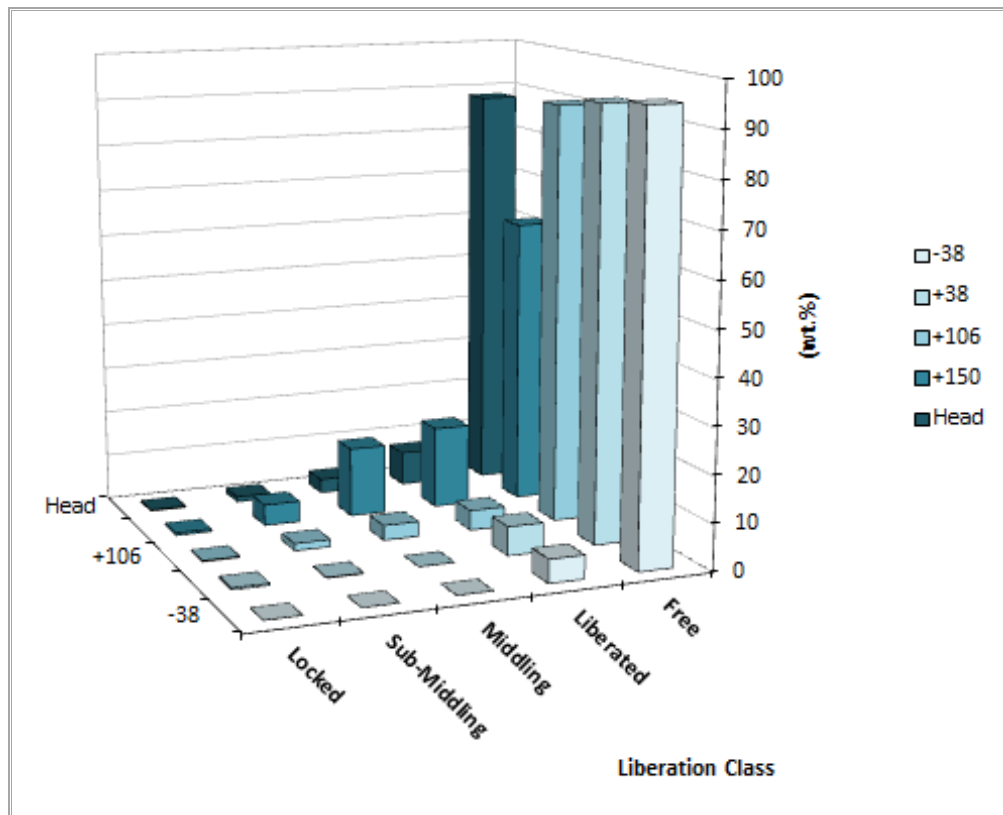
Overall, 98.1% of the copper is deported as tetrahedrite and 1.9% occurs as chalcopryite. In addition, it was determined that 100% of the antimony is present as tetrahedrite.

Liberation

Tetrahedrite liberation is summarized in Figure 13-4. Overall, tetrahedrite is extremely well liberated with over 88% of the particles being considered “free” at a grind size of ~100 µm. The degree of liberation decreases in the coarser size fractions with 62% of the +150 µm tetrahedrite particles being free compared to 95% of the -38 µm particles being free.

One hundred percent of the silver is found in well-liberated tetrahedrite minerals. This would suggest that high recoveries to high-grade concentrate should be achievable at a relatively coarse grind size.

Figure 13-4: Tetrahedrite Liberation by Size Fraction



Mercury Mineralogy

PMC ran a mineralogical study on a flotation concentrate produced by BCR. This study was conducted on a single polished section containing a representative sample of the flotation product. Therefore, although the sample is high in mercury, the statistics on particles in the higher end of the size distribution are not as good as those for the particles in the lower size fractions. Because of this, the results summarized in Table 13-4 should be viewed as indicative, but not statistically sound.

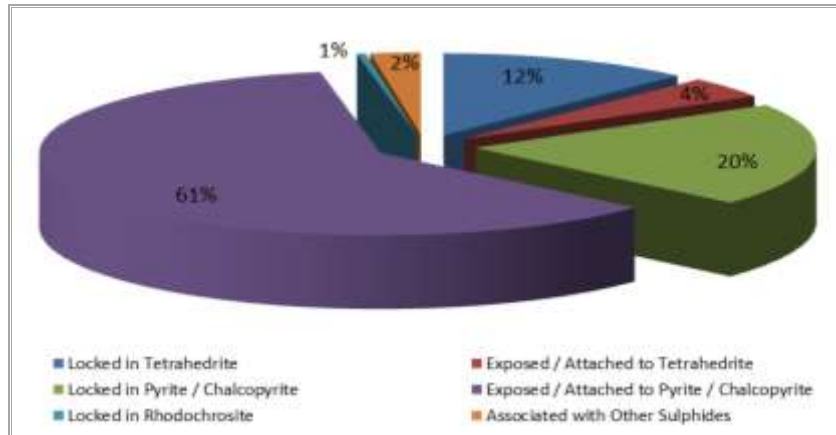
Table 13-4: Mercury Mineralogical Composition

Mineral	Ideal Formula	Distribution (%)
Cinnabar	HgS	90.0
Galkaite	(CS,Ti) (Hg,Cu,Zn) ₆ (As,Sb) ₄ S ₁₂	8.0
Gravellite	(Hg, BiS)	2.0
Total		100.0

Ninety percent of the mercury is contained within the mineral cinnabar. However, it should be noted that the remaining 10% of the mercury minerals are fine-grained, making it difficult for SEM to disassociate the mercury minerals from those surrounding them. This could lead to false interpretation of the mercury mineral complex.

Figure 13-5 is somewhat biased due to the unsized nature of the analysis. Several large cinnabar particles were found associated with pyrite/chalcopyrite. However, there is a clear trend of associations, as it appears that most of the mercury minerals are associated with pyrite/chalcopyrite and tetrahedrite. With the exception of a few large chalcopyrite associated particles, most were found to be fine-grained and locked within the silver-bearing minerals.

Figure 13-5: Mercury Liberation and Association



13.3.3 Mineral Processing

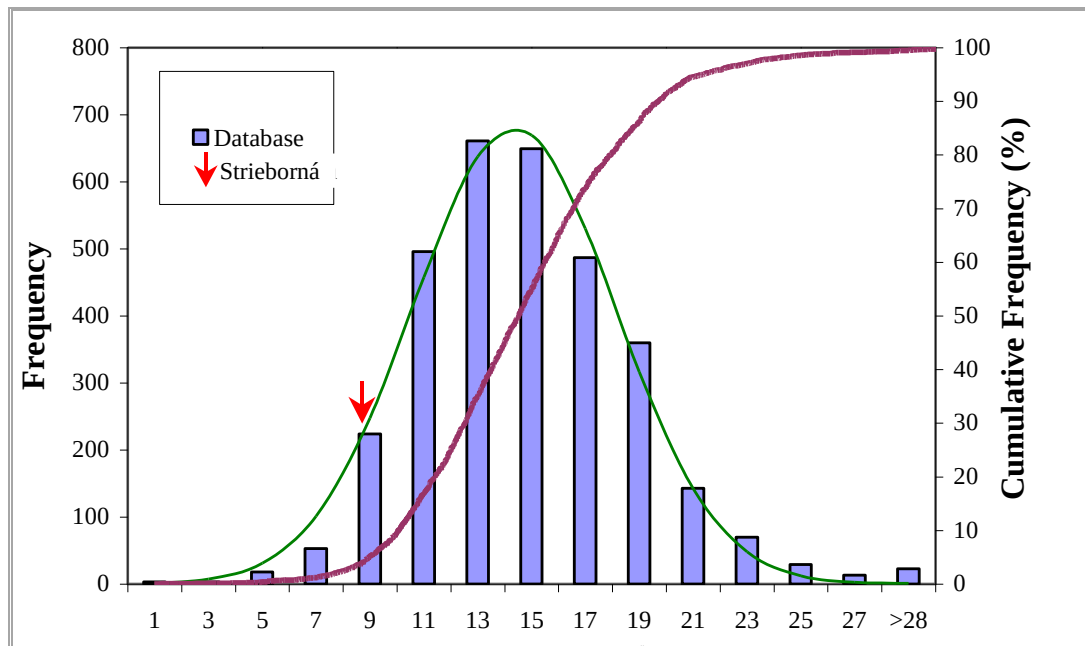
The mineral processing testwork program was undertaken at BCR. This program consisted of simple comminution testing, followed by a battery of scoping-level flotation flowsheet development tests. All work was completed on replicate 2.0 kg charges of the Strieborná master composite (STB MC). BCR conducted a Bond Ball Mill work index (BWi) test, rougher and cleaner flotation tests, and a locked-cycle flotation test.

An additional bulk flotation concentrate production program was completed at SGS-UK's metallurgical facilities in Cornwall, England, to produce a bulk flotation concentrate for pyro-metallurgical testing. These flotation tests were conducted on a different composite from assay reject samples delivered to the SGS-UK facility directly from the project site. The SGS-UK tests were completed using the flowsheet developed by BCR.

Bond Ball Work Index

A single BWi test was undertaken on drill core from the Strieborná deposit with a closing screen size of 100 mesh (150 µm). The metric BWi was determined to be 8.7 kWh/t, and the sample measured was deemed to fall into the "soft" category, as illustrated in Figure 13-6.

Figure 13-6: Bond Ball Mill Work Index Compared with the SGS-UK Database



Note: Purple line denotes cumulative distribution of data

Rougher Flotation

A series of rougher flotation tests were undertaken in order to assess the response of the STB MC to conventional upgrading via froth flotation. In total, four rougher flotation tests were undertaken using various flotation conditions. The main variable tested was primary grind size (P_{80}); other variables remained fixed, such as collector dosage, frother dosage, and flotation pH. Table 13-5 shows a summary of the conditions tested.

Table 13-5: Summary of SGS-UK Rougher Flotation Test Conditions

Test ID	Grind (P_{80} μ m)	Rougher Reagent Addition (g/t)				Flotation Time (min.)
		Lime	3477	SIPX	MIBC	
STB MC-T1	128	0	20	25	5	11
STB MC-T2	130	0	20	25	5	11
STB MC-T3	77	0	20	25	5	11
STB MC-T4	168	0	20	25	5	13

Figure 13-7 and Figure 13-8 show the results for these tests.

Figure 13-7: Rougher Test Silver Grade Recovery Curves

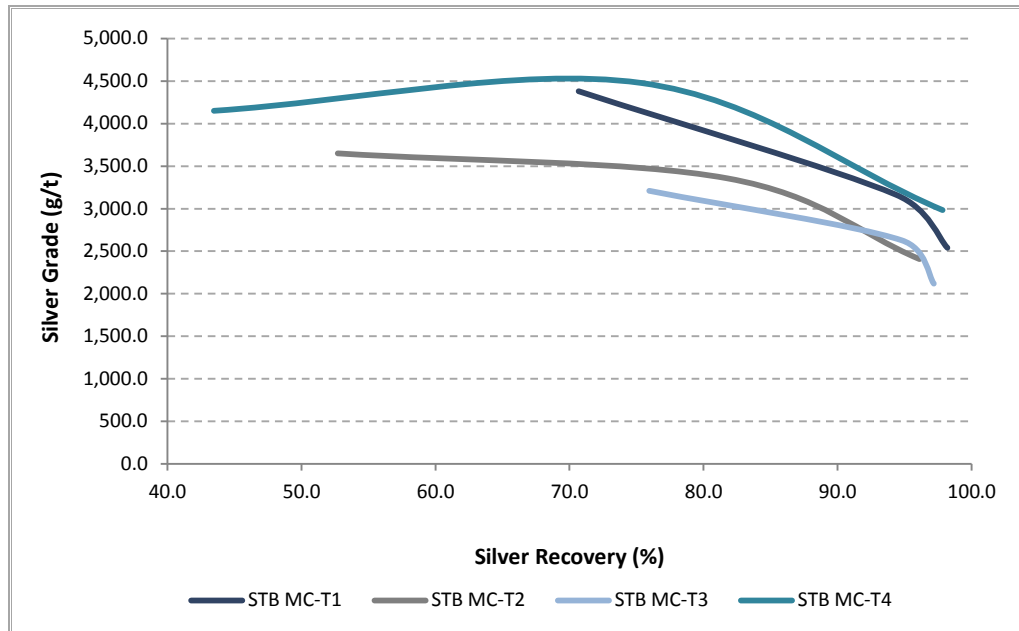


Figure 13-8: Rougher Test Mass Pull vs. Silver Recovery Curves

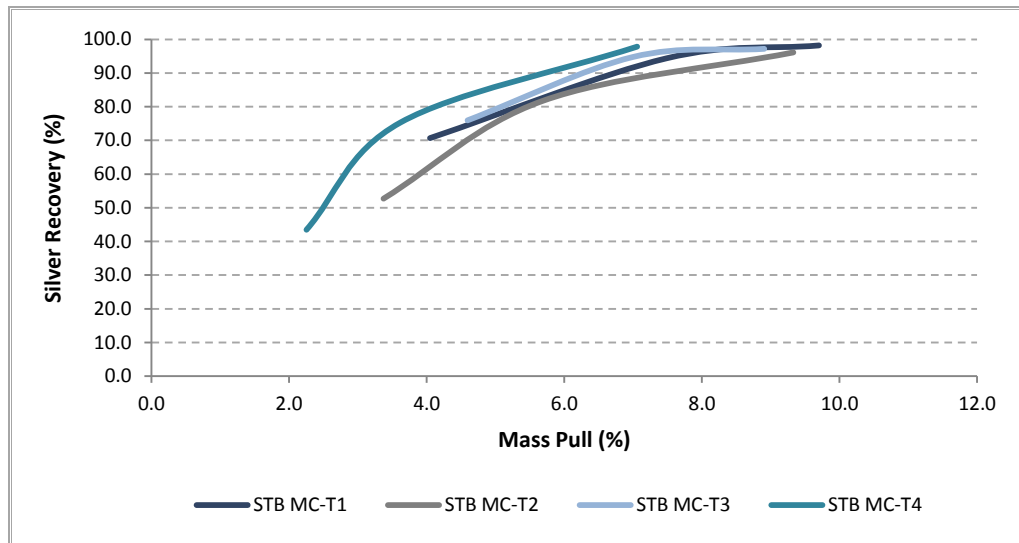


Table 13-6 show the results of all four tests, STB MC1 through STB MC4.

Table 13-6: Rougher Flotation Test Results, MC1 – 4

Test ID	Ro Mass Pull %	Ro Conc. Grade				Ro Conc. Recovery			
		Cu %	Ag g/t	S %	Sb %	Cu %	Ag %	S %	Sb %
STB MC-T1	9.71	12.0	2540	10.74	7.61	99.2	98.2	97.5	98.8
STB MC-T2	9.33	12.1	2407	10.66	8.62	97.7	96.1	93.2	96.7
STB MC-T3	8.91	11.7	2119	10.61	8.10	99.1	97.2	92.8	98.8
STB MC-T4	7.06	14.8	2983	14.81	10.27	98.3	97.8	95.7	97.5

All four of the initial rougher tests undertaken produced excellent silver recoveries to rougher concentrate at over 96.1% Ag recovery, with silver grades over 2,119 g/t Ag. Copper and antimony grades and recoveries were similarly high, suggesting silver and antimony recoveries are intimately linked to copper recoveries.

There does not appear to be any disadvantage in terms of pay-metal recovery to grinding coarser, as Test 4, at a primary grind P_{80} of $\sim 170 \mu\text{m}$, produced a silver recovery of over 98%, to the highest rougher concentrate grade of 2,983 g/t Ag. The highest overall recoveries were attributed to Test STB MC-T1 at a primary grind P_{80} of $\sim 130 \mu\text{m}$; therefore, it was decided to proceed with cleaner flotation using these conditions.

Cleaner Flotation

A series of cleaner flotation tests were undertaken with the goal of producing high-grade silver concentrates at acceptable silver recoveries. Six cleaner flotation tests were undertaken in order to assess the following variables:

- potential benefit of a regrind on the copper rougher concentrate prior to first cleaner flotation
- collector dosage (Cyttec 3477)
- high pH with lime.

Table 13-7 summarizes the conditions for the six tests.

Table 13-7: Summary of Cleaner Flotation Test Conditions

Test ID	Grind ($P_{80} \mu\text{m}$)	Rougher Reagents (g/t)		Time (min.)		Cleaner 1 Reagents (g/t)			Float Time (min.)
		3477	MIBC	Flotation	Regrind	Lime	3477	MIBC	
STB MC-T5	128	25	10	8	0	395	10	5	6
STB MC-T6	125	25	10	8	0	to pH 10	25	15	8
STB MC-T7	125	25	10	8	10	to pH 10	20	15	6
STB MC-T8	180	25	10	8	0	to pH 10	60	15	8
STB MC-T9	150	25	10	8	0	to pH 10	25	15	8
STB MC-T10	123	15	10	8	0	to pH 10	15	15	8

The overall first cleaner concentrate grades and recoveries for the six tests are summarized in Table 13-8.

Overall concentrate grades ranged from 4,730 g/t Ag to 5,423 g/t Ag, and from 20.0% Cu to 21.8% Cu, with silver recoveries ranging from 95.8% to 96.9%. Cleaner 1 concentrate mass pulls were very consistent, at between 4.8% and 5.2%. The grade recovery curves for these six tests are summarized in Figure 13-9 and Figure 13-10 and show that copper concentrates grading at least 25% Cu were produced, albeit at intermediate, but still perfectly acceptable copper recoveries.

Table 13-8: Summary of Cleaner Test Results

Test ID	Mass Pull (%)	Cleaner 1 Conc. Grade				Cleaner 1 Conc. Recovery			
		Cu %	Ag g/t	S %	Sb %	Cu %	Ag %	S %	Sb %
STB MC-T5	5.15	20.08	4,851	19.33	15.42	97.85	95.85	92.91	96.33
STB MC-T6	5.15	20.70	4,783	19.61	15.53	98.55	96.67	96.41	97.04
STB MC-T7	4.88	21.76	5,423	19.26	16.59	97.00	96.27	90.39	95.54
STB MC-T8	5.01	20.21	4,730	20.62	16.11	95.31	96.36	94.57	94.57
STB MC-T9	5.16	20.02	4,818	19.75	16.10	95.36	97.30	95.19	94.43
STB MC-T10	4.81	21.09	4,825	19.61	16.90	93.67	96.89	92.26	93.40

Figure 13-9: Summary of Cleaner Test Silver Grade Recovery Curves

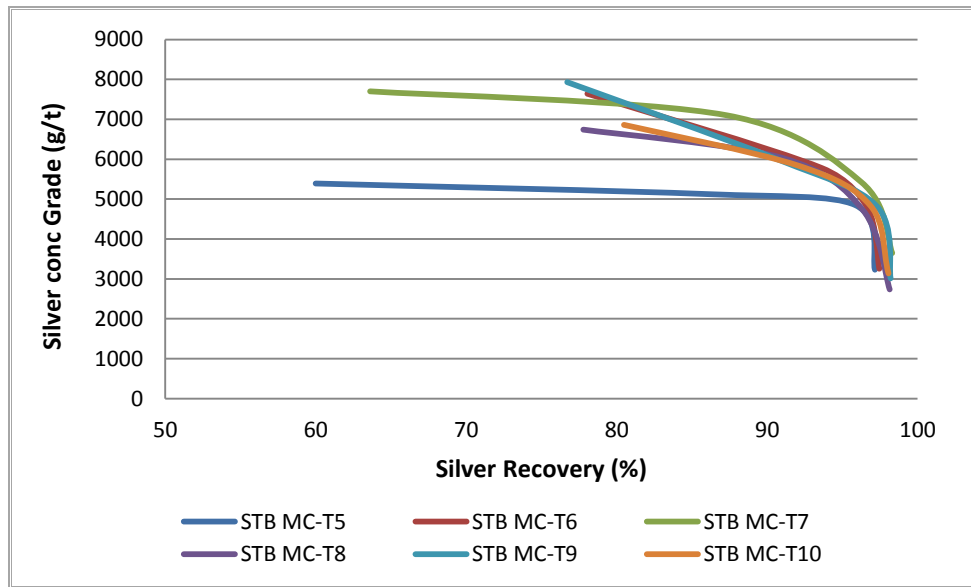
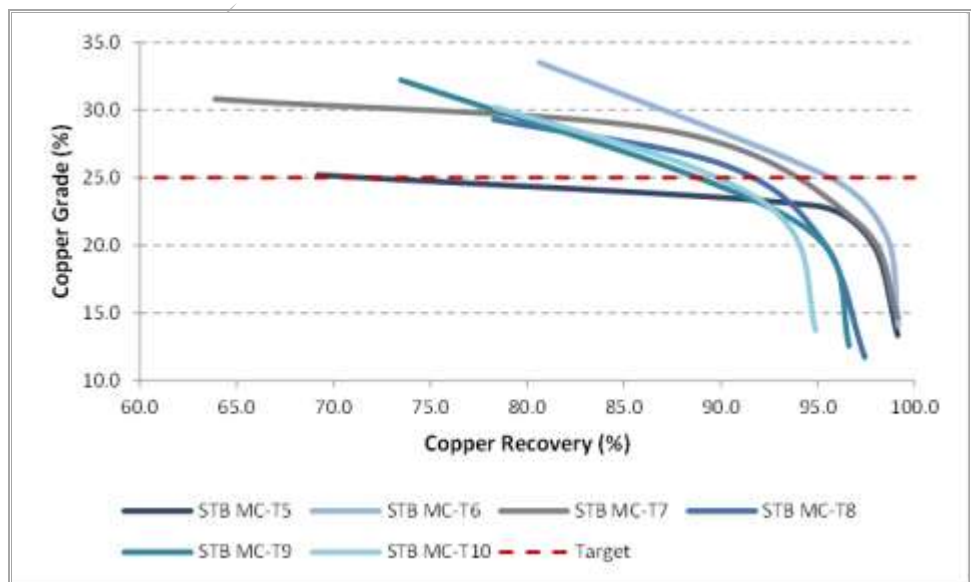


Figure 13-10: Summary of Cleaner Test Copper Grade Recovery Curves

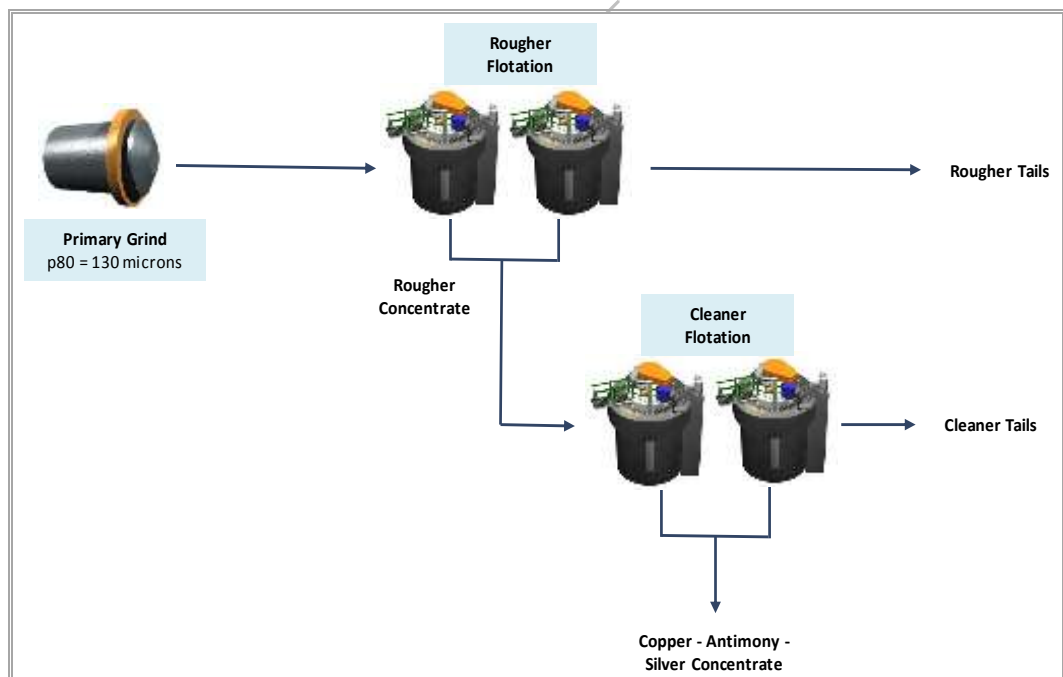


At a nominal 25% Cu concentrate grade, the copper recoveries ranged from ~73% to ~96%, with test STB MC-T6 producing the best overall metallurgical performance. At this grade recovery point, STB MC-T6 first cleaner concentrate graded ~5,800 g/t Ag at a 94% Ag recovery, and 18.7% Sb at a 94% Sb recovery. The mass pull to concentrate was 4.1%.

The cleaner tests showed that a regrind of the rougher concentrate prior to cleaning (STB MC-T7) does not increase metallurgical performance with respect to copper, and actually results in an inferior grade recovery curve compared to test STB MC-T6. However, the opposite is observed with respect to silver, where the highest grade silver concentrate is produced after a regrind, accordingly data from this program on the effect of regrinding on concentrate upgrading was inconclusive. As expected, lower collector dosage (STB MC-T10) produced a lower copper recovery to concentrate (~90% Cu recovery at a 25% Cu concentrate grade).

Figure 13-11 shows a conceptual flowsheet representing the above flotation conditions, which are discussed in more detail in Section 17.

Figure 13-11: Conceptual Flotation Process Flowsheet



Locked-Cycle Testing

A seven-cycle locked-cycle test was undertaken using the open cycle testwork flowsheet, with the first cleaner tailings being circulated back to midway down the rougher circuit. Table 13-9 summarizes the flotation conditions.

Table 13-9: Locked-Cycle Test Conditions

Stage	Reagents (g/t)				Time (min.)			pH
	Lime	3477	SIPX	MIBC*	Grind	Cond.	Froth	
Primary Grind	-	-	-		16	-	-	Nat.
Rougher 1	-	10	-	5	-	2	3	Nat.
Rougher 2	-	15	-	5	-	2	5	Nat.
Cleaner 1 Conc. 1	Y	-	-	5	-	0.5	1	10
Cleaner 1 Conc. 2	Y	10	-	5	-	2	2	10
Cleaner 1 Conc. 3	Y	15	-	5	-	2	5	10
Total	0	50	-	25	16	8.5	16	
Stage	Rougher		Cleaner					
Flotation Cell	4 L cell		2 L cell					
Speed: rpm	1,200		1,000					
Observations:	* Aerofroth 70 used							

Table 13-10 shows a summary of the full test results.

Table 13-10: Locked-cycle Test Recovery

Product	Weight (%)	Assays			% Distribution		
		Cu %	S %	Ag g/t	Cu %	S %	Ag %
Cleaner 1 Conc.	4.5	23.22	22.6	5,864	98.0	96.6	98.5
Cleaner 1 Tailings	0.4	0.58	1.07	129	0.2	0.4	0.2
Rougher Tailings	95.0	0.02	0.03	4	1.8	3.0	1.3
Feed	100	1.08	1.06	271	100	100	100

Silver recovery was 98.5% to a final concentrate grading ~5,900 g/t Ag. The copper recovery to this concentrate was also high, at 98.0% grading 23% Cu, while the circuit reached stability as early as Cycle 2, exhibiting excellent stability throughout.

Bulk Concentrate Production (SGS-UK)

SGS-UK was commissioned to produce a bulk flotation concentrate for roasting amenability tests. The 120 kg composite used for testing was produced from assay rejects, and assayed 1.18% Cu, 251 g/t Ag, 1.14% S, and 0.79 % Sb.

Table 13-11: Assay Reject Sample Numbers as Received by SGS-UK

J042261	J042201	J042294	J042370	J042460	J042435	J042414
J042262	J042203	J042300	J042371	J042459	J042436	J042415
J042271	J042242	J042301	J042372	J042443	J042442	J042417
J042264	J042245	J042302	J042374	J042447	J042441	J042409
J042263	J042260	J042303	J042375	J042448	J042439	J042396
J042288	J042252	J042305	J042376	J042449	J042440	J042418
J042289	J042248	J042306	J042350	J042450	J042419	J042397
J042265	J042249	J042307	J042352	J042451	J042420	J042398

J042266	J042251	J042308	J042351	J042453	J042421	J042399
J042272	J042175	J042313	J042359	J042452	J042422	J042400
J042267	J042253	J042314	J042361	J042454	J042423	J042402
J042268	J042254	J042316	J042360	J042455	J042424	J042403
J042269	J042176	J042317	J042362	J042446	J042425	J042404
J042282	J042178	J042320	J042369	J042456	J042393	J042405
J042278	J042180	J042335	J042464	J042458	J042412	J042406
J042280	J042195	J042338	J042463	J042426	J042395	
J042281	J042196	J042341	J042466	J042427	J042407	
J042284	J042198	J042346	J042465	J042428	J042392	
J042287	J042199	J042348	J042462	J042429	J042391	
J042286	J042200	J042349	J042461	J042438	J042408	

The SGS-UK team ran one rougher, one cleaner, and ten 10-kg bulk flotation tests. The cleaner flotation test was conducted using the best conditions produced by BCR, and produced results similar to those produced at BCR. The concentrate from this test yielded a copper recovery of 95.7% to a grade of 27.7%; silver recovery was 95.8% to a grade of 7,265 g/t.

Using these conditions, SGS-UK performed ten 10-kg bulk flotation tests. The concentrates were combined, filtered, assayed, and sent to a third party for roasting tests. The overall concentrate was balanced to give copper and silver recoveries comparable to the earlier testwork. The Inductively Coupled Plasma (ICP) scan of this bulk concentrate is shown in Table 13-12.

Table 13-12: Multi Element ICP Scan of Flotation Final Concentrate

Element	Assay (ppm)	Element	Assay (ppm)	Element	Assay (ppm)
Ag	0.77	In	<1	Sc	<1
Al	112	K	204	Se	<1
As	6,608	La	<1	Sn	88
B	12	Lu	<1	Sr	3
Be	<1	Mg	2,289	Ta	<1
Bi	2,474	Mn	1,789	Tb	<1
Ca	1,189	Mo	12	Th	<1
Cd	32	Na	493	Ti	2
Ce	<1	Nb	<1	Tl	<1
Co	97	Ni	331	U	<1
Cr	106	P	436	V	2
Cu	246,000	Pb	1,999	W	85
Fe	86,190	Rb	<1	Yb	1
Ga	<1	S	265,000	Zn	4,896
Hf	<1	Sb	192,000	Zr	133
Hg	7,200				

Flotation of Phase 1 Material

A small program of flotation testwork was conducted on the high-grade Phase 1 Zone material, which is expected to be mined early in the life of the mine. This was sampled in June 2014.

The conditions used represented a small departure from the earlier work, with a move to lower doses of collector with the objective of raising concentrate grades. Table 13-13 show these conditions.

Table 13-13: Conditions used in Batch Flotation Testing on Phase 1 Zone Composites

Test Stage	Reagents, g/t			Time, min		pH	Ep vs. Ag/AgCl
	Lime	3477	MIBC	Cond.	Froth		
<i>Master Comp.</i>							
F1 – 101 µm	65	16	65	4	12	6.5-7.5	-30 to +10
F2 – 123 µm	50	16	65	4	11	6.5-7.5	-30 to +10
F3 – 182 µm	65	16	81	4	11	6.5-7.5	-30 to +10
F4 – 130 µm	55	16	97.2	4	8	6.5-7.5	-30 to +10
<i>High Grade Comp.</i>							
F5 – 130 µm	60	16	81	4	11	6.5-7.5	-30 to +10

The resulting metallurgy was improved over previous work, with the rougher concentrates now assaying 24% to 27% Cu and up to 6,200 g/t Ag. Recoveries remained high at around 98% and as before, there was little effect of primary grind on flotation performance. The single test on the high grade composite yielded essentially the same metallurgy as the tests on the master composite, as shown in Table 13-14.

Table 13-14: Results from Batch Flotation Testing on Phase 1 Zone Composites

Test	Percentage (%)	Assays			% Distribution		
		Cu %	Ag %	Sb %	Cu %	Ag %	Sb %
<i>Master Comp.</i>							
F1 – 101 µm	6.5	26.8	6153	19.5	97.5	97.4	95.6
F2 – 123 µm	6.8	25.7	5962	18.9	97.5	97.8	94.8
F3 – 182 µm	6.8	24.6	5582	17.9	97.5	97.7	96.4
F4 – 130 µm	4.5	25.4	5814	18.2	98.9	99.0	97.7
<i>High Grade Comp.</i>							
F5 – 130 µm	9.1	24.6	5617	17.6	98.7	98.8	97.8

The cleaner tests explored the use of regrinding in multi-stage cleaning. In each case, the third cleaner float yielded the metallurgy as shown in Table 13-15.

Table 13-15: Results from Cleaner Testing on Phase 1 Zone Master Composite

Product	Weight (%)	Assays			% Distribution		
		Cu %	Ag %	Sb %	Cu %	Ag %	Sb %
F6 – No Re grind	5.6	28.6	6765	21.0	96.1	96.4	94.9
F7 – Fine Re grind	5.1	31.6	7573	23.1	97.6	98.0	96.1
F8 – Coarse Re grind	5.2	30.9	7226	22.3	97.3	98.1	96.2
F9 – Average Grade Comp.	3.3	33.6	7807	24.9	97.7	98.1	95.9

Earlier testwork had led to somewhat inconclusive results on the effect of re grind, but these results suggest value could indeed be gained from re grinding of the rougher concentrate, with silver grades reaching 7 kg/t to 8 kg/t in several of the tests, while maintaining copper and silver recoveries in the 98% range. Copper grades were over 30% for most tests.

However, the confirmatory locked-cycle test was run without a re grind to keep the flowsheet aligned with the design being costed at the time for this PEA by AGP. The resulting metallurgy is summarized Table 13-16. A concentrate assaying 27% Cu and 6.5 kg/t Ag was floated at between 98% to 99% metal recovery. The locked-cycle test was highly stable after the third cycle, validating the workability of the overall flowsheet. It seems likely that should a re grind be included, the concentrate grade would have been 1% to 2% higher in copper, at equivalent recoveries.

Table 13-16: Summarized Locked-cycle Metallurgy from Phase 1 Zone

Product	Weight (%)	Assays			% Distribution		
		Cu %	Ag g/t	Sb %	Cu	Ag	Sb
Cu Cleaner 3 Conc.	6.1	27.0	6530	20.6	98.2	98.6	96.7
Cu Cleaner 1 Tailings	0.9	1.19	251	1.1	0.6	0.6	0.8
Rougher Tailings	93.0	0.021	3.5	0.04	1.2	0.8	2.6
Feed	100.0	1.67	401.5	1.29	100.0	100.0	100.0

The mercury content was high at 0.96%, the assay ratio of mercury to silver being consistent with the earlier composites. A full elemental analysis is shown in Table 13-17.

Table 13-17: Full Element Scan of Final Concentrate from Phase 1 Zone Locked-cycle Test

Element	Unit	Method	Assay
Ag	ppm	4A-AA	6,488
Al	%	Fus-XRF	<0.01
As	ppm	Fus-XRF	16,300
B	ppm	ICP	41
Ba	ppm	ICP	<10
Be	ppm	ICP	<0.5
Bi	ppm	ICP	3,098

Element	Unit	Method	Assay
Mg	%	ICP	0.23
Mn	ppm	ICP	2,500
Mo	ppm	ICP	2,504
Ni	ppm	ICP	860
P	ppm	ICP	20
Pb	ppm	ICP	220
S	%	Leco	24.38

Element	Unit	Method	Assay
Ca	%	Fus-XRF	0.05
Cd	ppm	ICP	26.1
Co	ppm	ICP	250
Cr	ppm	ICP	<1
Cu	%	4A-AA	26.47
Fe	%	Fus-XRF	15.17
Ga	ppm	ICP	16
Hg	%	Cold fusion	0.96
K	%	ICP	<0.01

Element	Unit	Method	Assay
Sb	%	4A-AA	20.4
Sr	ppm	ICP	<1
Si	%	Fus-XRF	0.3
Ti	%	ICP	<0.01
Tl	ppm	ICP	<10
V	ppm	ICP	5
W	ppm	ICP	<10
Zn	ppm	ICP	6,300
Zr	ppm	ICP	<5

Environmental Testing

A composite of combined rougher and cleaner tailings was submitted to SGS in Vancouver for environmental testing. SGS performed an acid base accounting (ABA) test. Results showed the acid producing potential (AP) is 0.9; the neutralization potential is 10.6, and the overall net neutralization potential (Net NP) is 9.7.

13.3.4 Hydrometallurgical Testing

A series of experiments was performed to evaluate the effects of alkaline sulphide leaching of the Strieborná flotation concentrate. Antimony and mercury were selectively leached in sodium sulphide and sodium hydroxide solutions. Six tests were conducted to evaluate the leach extraction, using different concentrations of the hydroxide/sulphide solutions at different temperatures. The leach tests were all conducted for 24 hours; test conditions are shown in Table 13-18.

Table 13-18: First Round Sulphide Leach Conditions

Test No.	Solution Volume (mL)	Concentrate Mass (g)	NaOH (g/L)	Na ₂ S (g/L)	Na ₂ S Conc. Ratio (g/g)	Temp. (°C)	Time (h)
1	200	20	18	200	2	95	24
2	200	20	36	100	1	95	24
3	200	20	36	150	1.5	95	24
4	200	20	18	200	2	85	24
5	200	20	54	100	1	95	24
6	440	80	38	381	2.1	105	24

The best result gave 97% Sb and 92% Hg extraction. These results were achieved at 95°C with 200 g/L of Na₂S and 18 g/L of NaOH, with stirring at 500 rpm, over 24 h, and 20 g of concentrate in 200 mL of solution. 92% Hg extraction corresponded to 0.06% Hg (600 ppm) in the residue. Arsenic and copper were not leached under the conditions tested here. Silver and gold were only assayed for in one experiment and they did not appear to leach. The effect of temperature between 85°C and 105°C is inconclusive, but it appears as though 95°C is sufficient for optimal extraction.

Although 92% Hg extraction was achieved, it was deemed that 600 ppm of mercury in the residue was too high compared with the results obtained from roasting. In an attempt to reduce the mercury grade in the residue further, three more tests were conducted:

- Test 7: 24 h leach, followed by filtering and re-leach for 24 h
- Test 8: 48 h leach
- Test 9: 24 h leach followed by filtering, pulverizing, and a further 24 h leach.

Table 13-19 presents the results of these tests.

Table 13-19: Final Sulphide Leach Residue Assays

Test No.	As %	Sb %	Hg %	Cu %	Ag g/t	Au g/t
Head Test 7, 8, and 9	1.1	15.4	0.68	21.7	6,570	2.16
Test 7 (re-leach)	0.83	0.31	0.021	20	5,150	0.26
Test 8 (48 h)	1.24	0.86	0.035	21	6,330	0.37
Test 9 (regrinding)	0.77	0.13	0.01	22.1	5,920	0.44

Test 8 showed some improvement on the leach results achieved from the first round of testing, and Test 7 more so. The best results were found in Test 9, with mercury extraction reaching 98%, and antimony extraction reaching over 99%.

Concentrate Leach Tests at SGSM (Lakefield)

A three-stage batch-leaching experiment on Strieborná concentrate was run at SGS Minerals Services (SGSM) in Lakefield, Ontario. Specified conditions for each stage were temperature, 95°C; retention time, 24 hours; sodium sulphide concentration, 200 g/L; and NaOH concentration, 18 g/L.

Cumulative antimony extractions for each leaching stage (calculated from solid assays) were 57%, 88%, and 100%, while those for mercury were 23%, 95%, and 99%. The final mercury assay in the residue from the third and final stage was 120 ppm.

With each stage, leaching tended to stop after five to eight hours. This may be due to insufficient reagent addition, to a passivation phenomenon (e.g., a reaction product film), or to inhibition by soluble products (e.g., sulphide complexes). This is consistent with past practice at the Sunshine Mine, where further reagent was often needed after five to ten hours, so it seems likely that reagent depletion stopped the leach. In reality, a closely controlled leach might take significantly less time to complete than this test did.

Mercury Precipitation

Cementation testing was conducted on 500 mL samples of the first stage leach solutions. The concentration of mercury declined rapidly at the outset, i.e., in the first 30 to 60 minutes of precipitation, and in the better of the two tests the starter solution of 23 ppm mercury dropped down to 1.9 ppm mercury. It is possible that the cementation

reaction is favoured by high sulphide concentrations. However, further work would be required to prove this hypothesis and to determine the mechanism. Solid residue mercury concentrations were quite high, and future tests should look at reducing the consumption of antimony powder and examining the mineralogy of the residue. The latter should also be done to show that elemental mercury is indeed formed. The formation of a dense liquid mercury phase may allow for antimony powder/mercury separation using gravimetric methods.

13.3.5 Pyrometallurgical Testing

The first samples of Strieborná flotation concentrate produced from tests on Mária and Strieborná ore zones contained elevated levels of mercury. The presence of mercury in these ores was also known from prior work in this mining area, and was further confirmed in subsequent flotation testing. While a number of options for on-site processing were possible, shipment of the concentrate to a custom copper smelter was under consideration as a preferred, lower capital cost option.

However, in this case, the material would need to be pre-treated for mercury removal prior to shipment, and therefore pre-treatment methods were investigated. Table 13-20 lists the analysis of a sample of Strieborná concentrate produced early in the present phase of testing, which indicated a mercury content that was quite high, at 0.58% Hg.

Table 13-20: Analysis of Concentrate from Initial Milling Tests (2012)

Element/Compound	Analysis (wt.%)
Cu	17.5
Ag	0.4515
Fe	20.3
S	30.0
Pb	0.248
Zn	0.689
Ni	0.0434
Bi	0.277
As	1.0
Sb	13.5
Hg	0.584
SiO ₂	12.9
Al ₂ O ₃	0.822
CaO	0.453
MgO	0.978
K ₂ O	0.259
Na ₂ O	0.007

It is noted that the antimony (and copper) is mainly present in the concentrate as tetrahedrite (Cu₁₂Sb₄S₁₃), with mercury typically substituting for copper in the mineral

lattice. The iron is largely present as siderite (FeCO_3), but some also exists in the tetrahedrite lattice. A roasting/heating process at moderate temperatures and under nominally inert gas atmosphere conditions was envisioned for this pre-treatment step, based on successful operations reported elsewhere treating copper concentrate containing elevated levels of mercury (Stuart, M.; Down, R.F.). In addition, the roasting of mercury-containing ores in a rotary kiln or a multi-hearth roaster in an air atmosphere was a fairly common operation in the early part of the last century for the purposes of obtaining metallic mercury from mercury-containing ores for the gold producing industry in the USA (Duschak, L.H., Schuette, C.N.). Hence, with this background, supported by thermodynamic studies, the roasting option was investigated.

A two-phase laboratory program was carried out, as shown in Table 13-21, timed in part by the availability of small quantities of test concentrate, to investigate mercury volatilization from Strieborná concentrate by roasting. The overall objective of the work was to determine general conditions under which more than 99% mercury elimination could be obtained from concentrate having a nominal mercury level of 7,000 ppm mercury.

Table 13-21: Two Phase Test Program

Phase	Test Program	Period (2012)
Phase I	Limited number of small scale ceramic boat tests (static) (10 g to 15 g/test)	March-April
Phase II	Extended small-scale tray tests (static) (15 g to 20 g/test) covering a range of variables	August-October

A summary of the results of these tests is given in the following sections.

Phase I Small-Scale Crucible Tests (March to April 2012)

A review of published data on the operations at Gortdrum Mines in Ireland in the 1970s (Stuart, M., R.F. Down. 1974), showed that mercury present in association with tennantite-tetrahedrite minerals in a copper concentrate could largely be removed by volatilization using a multi-hearth roaster in a low oxygen atmosphere (0.5% O_2) at 600°C. In Gortdrum's case, an inert atmosphere (produced by sub-stoichiometric fuel oil firing in the roaster) was chosen largely to prevent sulphur oxidation, so that, as required by the custom copper smelter at the time, the calcine retained all or most of the sulphur present in the green concentrate. A similar situation was considered applicable for Strieborná, thus obviating the need to build an acid plant at site. As noted above, at the beginning of the last century, mercury was volatilized from mercury-containing ores by roasting the material in a rotary kiln or in a multi-hearth roaster with an air atmosphere. The mercury was collected by condensation for use in gold operations in the USA.

Further, thermo-chemical calculations showed that over a wide range of elevated temperatures (500°C to over 750°C); mercury vapour is in the stable phase at log P_{SO_2}

values from 0 to -2.8; and for oxygen partial pressures from $\log P_{O_2}$ from -5 to -15. It is noted that, towards the lower temperature in the above range and towards the higher levels of P_{O_2} , some mercury sulphate may form. Therefore, volatilization of mercury from Strieborná concentrate was expected as long as the temperature was sufficient to decompose tetrahedrite, and from a practical standpoint, not so high so as to cause a sintering action on the material.

Further testwork will be required to determine optimal conditions for mercury removal from Strieborná concentrate, including roasting temperature and time.

Phase I Roasting Tests

The Phase I tests were carried out at XPS in Sudbury, ON, with the chemical analyses performed by SGSM (Lakefield). Using approximately 80 g of Strieborná concentrate, four small-scale laboratory tests were carried out to determine the degree of mercury elimination during inert gas roasting. At this stage, any sintering phenomena were not investigated.

These initial roasting tests were carried out under a flow of nitrogen gas in a horizontal laboratory tube furnace having a refractory tube approximately 3 cm in diameter. The idea was in part to mimic the Gortdrum process using a nitrogen atmosphere, but with a static sample, unlike in the multi-hearth roaster where the action of the rabble arms causes a certain degree of mixing of the material during roasting. The tests used a small amount of concentrate placed in an alumina boat positioned within the refractory tube. Conditions for this limited suite of roasting tests are summarized in Table 13-22. A fifth test was considered, examining mercury volatilization in air (with green concentrate as the starting material), but there was insufficient sample material left after the previous tests had been completed. A thermogravimetric test was also carried out.

Table 13-22: XPS Phase I Laboratory Hg Volatilization Test Conditions

Test No.	Conditions	Starting Material and Initial Weight of Sample
1	600°C, N ₂ atmosphere	Green conc., 15 g
2	750°C, N ₂ atmosphere	Green conc., 15 g
3	950°C, N ₂ atmosphere	Green conc., 30 g
4	950°C, Air atmosphere	Calcine from Test 3, 15 g

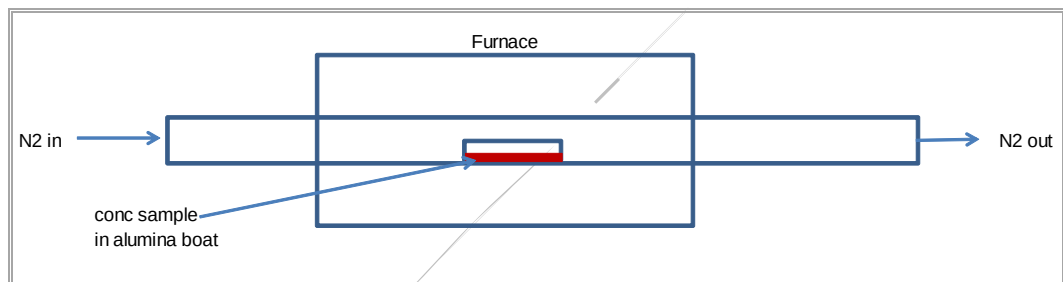
Note: The calcine produced in Test No. 3 was split in two, and one-half was used as the feed for Test No. 4 to determine the degree of antimony elimination when roasting was continued in an air atmosphere. This test was only undertaken to provide some idea of possible antimony behaviour under these conditions, recognizing that further testwork would be required.

The following procedure was used for the tests: the “as received” Strieborná concentrate was well-blended by mixing the material in a small container and a head sample was taken for chemical analysis. For each test, a weighed portion of concentrate was placed in the alumina boat, and the boat was placed inside the furnace tube. The tube was then placed inside the furnace, the nitrogen piping was connected, and a low flow of N₂ was started. The furnace was turned on and heated to bring the sample to the desired temperature for the particular test. Once at temperature, the furnace was held at the test temperature for 60 minutes. At the end of this time, the furnace was turned off and allowed to cool under a nitrogen flow.

Once at room temperature, the furnace tube was opened, the alumina boat removed, and a sample of the calcine taken for chemical analysis. A number of photographs of the head and calcine samples in the alumina boat are included in the test report.

The experimental arrangement used for the test is illustrated in Figure 13-12.

Figure 13-12: Sketch of Experimental Set-up for Phase I Hg Volatilization Tests at XPS



The results of the chemical analysis for each test are given in Table 13-23, while the calculated elimination of mercury, antimony, and sulphur in the tests is given in Table 13-24.

Some weight loss was observed in the tests, due in part to CO₂ evolution, likely from decomposition of iron carbonate (contained in the concentrate), and possibly a small amount of SO₂. It was noted that the indicated weight change between Test 3 and Test 4 does not appear to exactly correspond to the corresponding sulphur loss; however, this difference is not considered critical to the overall results.

The mercury and sulphur elimination values in Table 13-24 were based on element mass balance calculations.

Table 13-23: Sample Weights and Chemical Analysis of Phase I Test Samples

Test No.	Condition	Weight at End of Test (g)	Cu %	Fe %	S %	Sb %	Hg ppm
	Head sample-concentrate	10.0	22.6	15.2	22.4	17.7	7,210
1	600°C, N ₂ atmosphere	14.11	25.5	16.5	23.2	18.9	2,170
2	750°C, N ₂ atmosphere	12.91	27.3	18.8	24.0	18.3	0.5
3	950°C, N ₂ atmosphere	23.5	29.3	20.5	23.6	15.1	10.9

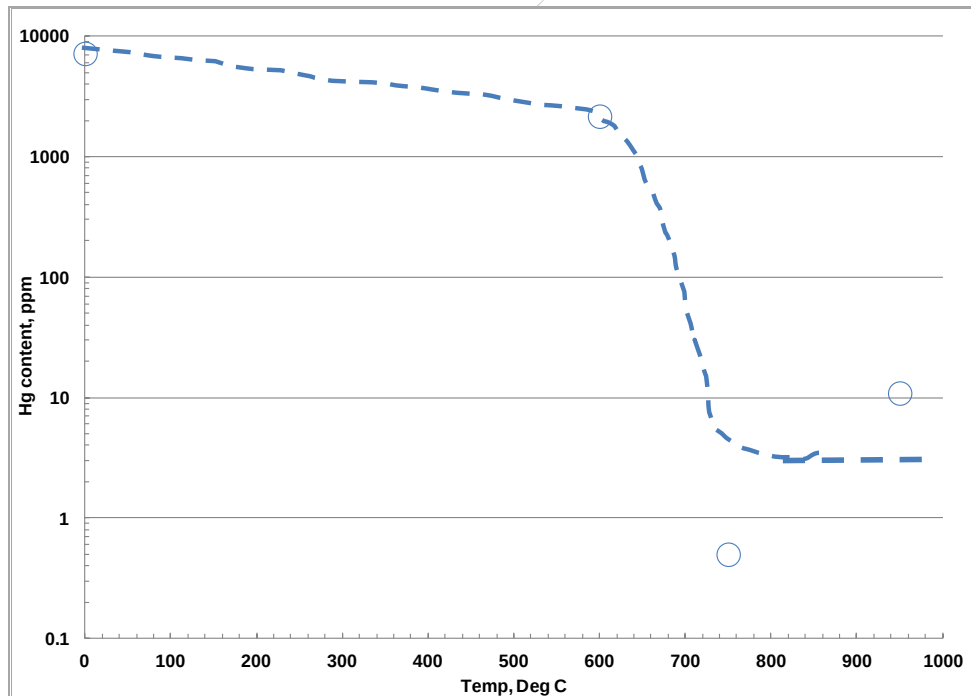
4	950°C, Air atmosphere	14.5	30.5	21.2	1.18	15.2	<0.3
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Table 13-24: Roasting Test Results – Phase I (XPS) Roasting Tests

Test No.	Condition	Weight Loss (%)	Elimination in Each Test		
			Hg %	Sb %	S %
1	600°C, N ₂	5.9	71.69	3.5	2.6
2	750°C, N ₂	13.9	99.99	15.4	7.8
3	950°C, N ₂	21.7	99.88	35.5	17.5
4	950°C, Air	24.0 (total)	~99.99+	37.4	95.2

The mercury contents in the calcine given as ppm as a function of temperature (under N₂ atmosphere) in the Phase I tests at XPS are graphically illustrated in Figure 13-13. The dotted line is an approximation representing the observed trend, in this case tending to flatten out at 3 ppm to 4 ppm mercury content (corresponding to an Hg elimination of about 99.95%) for roasting temperatures of over 750°C. However, the target mercury (>99.6%) may be obtained at a somewhat lower temperature than 750°C (likely around 650°C). Test No. 1 was run at 600°C and Test No. 2 at 750°C.

Figure 13-13: Mercury Content (ppm) as a Function of Temperature under N₂ Atmosphere



The conclusions based on the Phase I tests are as follows:

- Complete mercury elimination from Strieborná concentrate can be achieved by roasting up to 750°C, with the required elimination of 99.6% (~300 ppm mercury, for a nominal level of 7,000 ppm in concentrate) likely achieved at 650°C. While

this result was achieved by roasting in a nitrogen atmosphere, thermodynamic calculations and experience from Gortdrum Mines suggest the same result would have been achieved by roasting with a small amount of oxygen present (about 0.5% O₂). At this stage, any possible sintering effect was not determined.

- While the focus of the tests was on mercury elimination, the elimination of antimony was also briefly studied in Test 4. This test showed that the antimony elimination under these conditions (950°C and air atmosphere) was 37.4%. It is noted that this was just an initial test, and antimony elimination was not optimized.
- The preliminary results therefore indicated that roasting of Strieborná concentrates prior to shipment can lead to a concentrate sufficiently free from mercury to ensure mercury would not impede its marketability. Antimony elimination was more problematic, although marketing studies indicate that custom smelters would accept these concentrates (with an appropriate penalty).
- The above results suggested a one-step roasting process would be sufficient in a commercial operation at site. The specific roasting times and temperatures were to be determined in the Phase II tests (ideally using a stirred device to simulate the rabble arms in a hearth roaster, or mixing action in a rotary kiln).

Phase II – Extended Small Scale Crucible Test Program (July to September 2012)

With some small, additional quantities of Strieborná concentrate available by August 2012, additional tests were planned, typically using a slightly larger test mass than in Phase I tests, but now also covering an extended range of test variables, particularly roasting temperature and time. The Phase II tests were carried out at KPM in Kingston, ON, with the chemical analyses performed by SGSM (Lakefield).

The Phase II test program was undertaken with the objectives of covering a wider range of test conditions, observing the volatilization of mercury under a nitrogen atmosphere (including the effect of time and temperature), plus some testing in air, both to examine the sintering phenomenon, and to obtain initial data to develop a preliminary processing concept based on the use of a multi-hearth roaster for roasting. An alternative would be a rotary kiln. For the Phase II tests, a static mass of concentrate was also used.

The test conditions selected for these tests are presented in Table 13-25.

Table 13-25: Experimental Test Conditions for the Phase II Tests at KPM

Run	Temperature (°C)		Time (min.)		Gas
1	550		40, 120, 360		N ₂
2	650		30, 90, 270		N ₂
3	750		20, 60, 180		N ₂
4	500	850	40, 120	60	Air

Experimental Arrangement

The experiments were conducted using a sealed quartz furnace tube, 102 mm in diameter by 1.83 m long (4" x 6 ft), with the midsection of the tube positioned inside a 0.33 m square furnace (Figure 13-14). A ventilation system was also in place (visible in Figure 13-14).

Three small circular quartz trays, each 55 mm in diameter, were used to contain the sample inside the furnace (Figure 13-15, Figure 13-16). The trays were attached to quartz rods 1.22 m (4 ft) long, which allowed the trays to be moved independently in and out of the furnace hot zone.

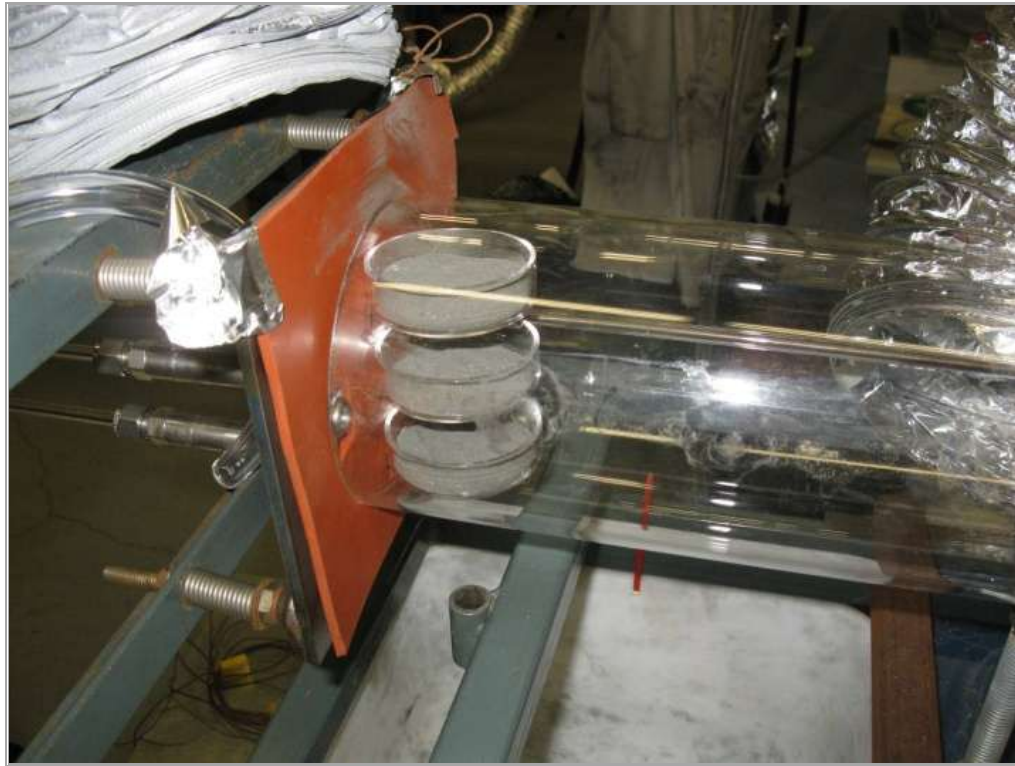
Figure 13-14: Furnace Used in Phase II Testing



Figure 13-15: Test Tray Used to Contain Concentrate Sample for Roasting Testwork

The gas delivered to the furnace could be varied and the flow controlled as needed. Provision was made to exhaust the off-gas and scrub this before venting. The furnace arrangement allowed for either all trays to be positioned inside the furnace at the beginning of a test run, or trays could be sequentially introduced to the furnace at different times. In this way, the first tray to be introduced could be roasted for a longer time. In this manner, the tests were efficiently carried out over a range of temperatures and roasting times.

Figure 13-16: Three Loaded Trays – Positioned inside the Furnace Reactor Tube (Phase II tests)



Test Results – Mercury Analysis and Elimination

At the end of each test, the material in the tray was observed and photographed. Upon removal of the roasted material, a sub-sample was taken for chemical analysis. A mineralogical examination was also performed. The results, only in terms of mercury content and elimination percentage, are summarized in Table 13-26. This data is plotted in Figure 13-17 and Figure 13-18.

A roasting time of approximately 160 to 170 minutes at 650°C was found adequate to bring the mercury level down to approximately 30 ppm mercury or lower (~99.6% Hg elimination). The calcine after the test at 650°C is illustrated in Figure 13-19 and Figure 13-19. The material in each of the three trays was observed to have lightly sintered together; however, each sample could be easily broken when light pressure was applied by hand.

Table 13-26: Results of Phase II Tests at KPM

Sample	Mass		Hg Assay (ppm)	Hg Elimination
	(g)	Relative		
120913 (650-30) Feed	19.8	100.0	6,462	100.0
120913 (650-30) Calcine	17.7	89.6	54	99.3
120913 (650-90) Feed	19.8	100.0	6,462	100.0
120913 (650-90) Calcine	17.5	88.1	131	98.2
120913 (650-270) Feed	19.9	100.0	6,462	100.0
120913 (650-270) Calcine	17.0	85.5	4	100.0
120917 (550-40) Feed	19.7	100.0	6,462	100.0
120917 (550-40) Calcine	18.1	92.0	3,430	51.2
120917 (550-120) Feed	21.0	100.0	6,462	100.0
120917 (550 -120) Calcine	18.9	90.0	1,990	72.3
120917 (550-360) Feed	19.8	100.0	6,462	100.0
120917 (550-360) Calcine	17.6	88.9	730	90.0
120919 (750-20) Feed	18.4	100.0	6,462	100.0
120919 (750-20) Calcine	16.2	87.9	4	100.0
120919 (750-60) Feed	19.8	100.0	6,462	100.0
120919 (750-60) Calcine	17.2	86.9	2	100.0
120919 (750-180) Feed	20.5	100.0	6,462	100.0
120919 (750-180) Calcine	17.2	83.7	2	100.0
120924 (500-40) Feed	20.6	100.0	6,462	100.0
120924 (500-40) Calcine	19.3	93.6	1,820	73.7
120924 (500-120) Feed	19.1	100.0	6,462	100.0
120924 (500-120) Calcine	17.7	92.8	186	97.3
120924 (500-850) Feed	20.2	100.0	6,462	100.0
120924(500-850) Calcine	18.1	89.5	9	100.0

Figure 13-17: Mercury Analysis in Calcine vs. Roasting Time (Phase II)

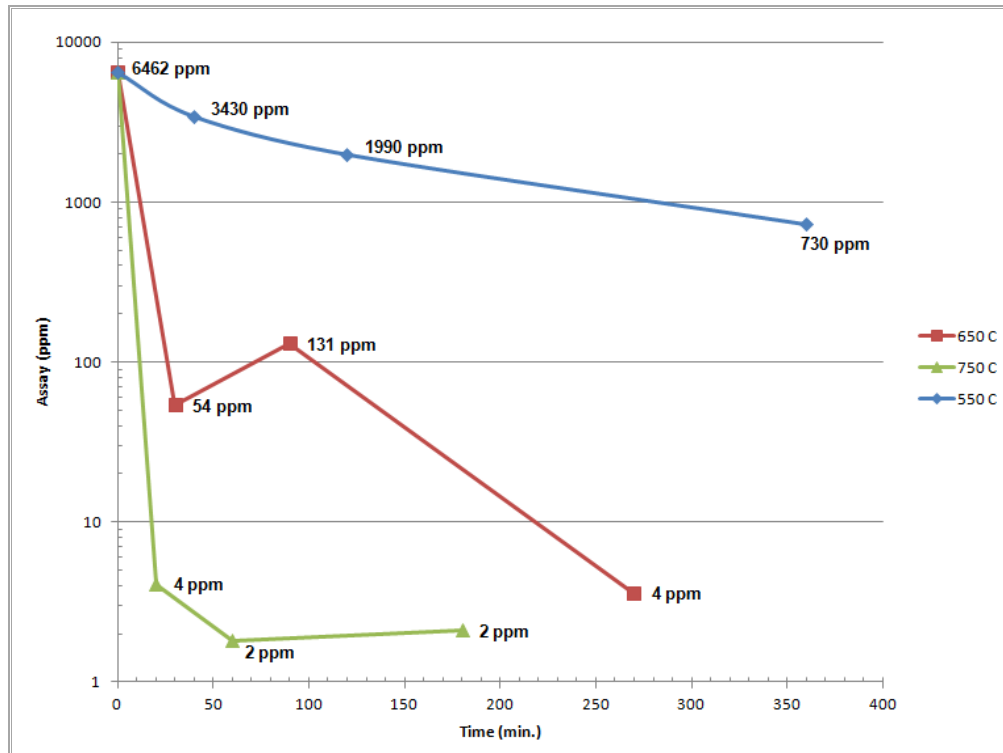


Figure 13-18: Mercury Elimination vs. Time (Phase II)

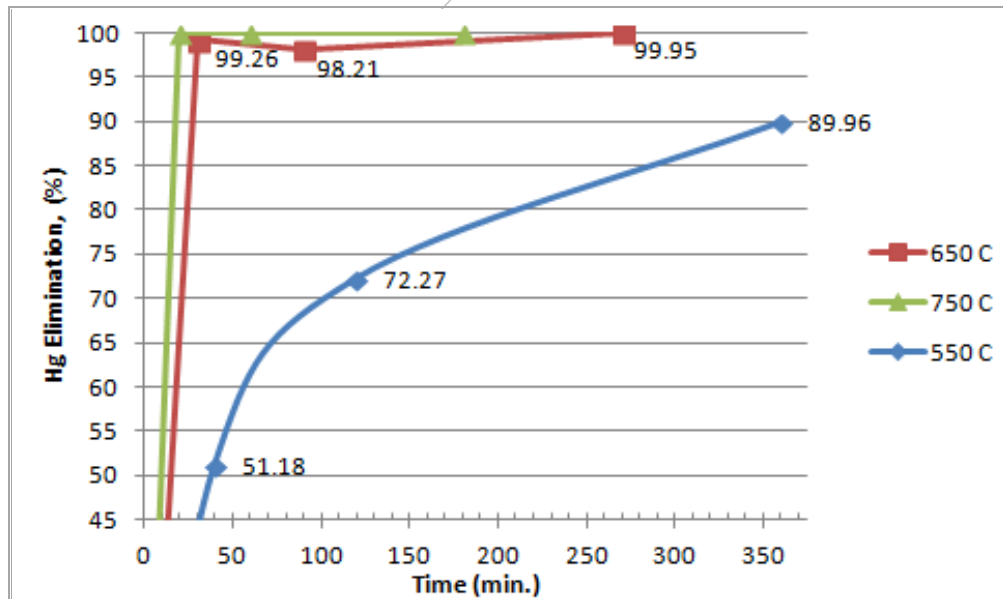


Figure 13-19: Roasted Calcine in Three Trays from the Phase II Test at 650°C



A cross-section of the sample from the 30-minute test at 650°C showed three layers (Figure 13-20), with the top and bottom appearing similar (light grey), and a thin, darker middle layer.

Figure 13-20: Roasted Calcine after the Phase II Test for 30 minutes at 650°C



DELA Recycling Solutions – 2013

Follow up calcining testwork on Strieborná concentrate was carried out at DELA Recycling Solutions in Dorsten, Germany. DELA, now operating under the name of NOR, operates a commercial rotary kiln for thermal treatment of custom feed materials. The facility has the potential to treat flotation concentrate from Strieborná and recover the contained mercury as a synthetic cinnabar product for disposal.

A 4 kg sample of concentrate was generated by the bulk flotation testwork at SGS-UK as described earlier in this section. The final concentrate graded 25.5% Cu, 19.1% Sb, 6,821 g/t Ag, 0.70% Hg, and 22.6% S. The sample was shipped to DELA for pyrometallurgical testwork.

The initial tests were conducted using 200 g of concentrate loaded into a small, rotating vessel in a lab furnace. The rotating vessel was intended to simulate the tumbling motion of an industrial kiln, but preliminary testwork indicated poor mercury removal results, attributed to low air flow through the kiln and re-precipitation of mercury within the vessel upon cooling.

Because of the difficulty in obtaining representative results with the lab set-up, all subsequent testwork was carried out on 10 g samples in an open crucible without mixing. The results indicated successful elimination of mercury at temperatures ranging from 550°C to 700°C, and at retention times from 20 to 120 minutes.

A third sequence of tests at DELA, intended to optimize conditions in the target temperature range of 630°C to 650°C, was conducted on 120 g of sample. The results indicated better than 99.9% Hg removal after 60 minutes of calcining at 650°C. Partial sintering of the material was observed for all tests in this series. It should be noted that all of the testwork at DELA was conducted in air, in contrast to the KPM results, where most of the work was done in a nitrogen atmosphere.

Concluding their test program, DELA reported that *“after the first test runs DELA GmbH is convinced that treatment of the material in industrial scale can be conducted successfully”* (Anon, 2013).

DELA Calcine Leach Testing at SGSM (Lakefield)

A composite sample of calcine test residues from the DELA runs, weighing roughly 120 g, was sent to SGSM (Lakefield) in September 2013 for leaching testwork. The objectives of this test program were to confirm mercury extraction to low residue concentrations, collect information and samples necessary for experimental work aimed at mercury separation from the alkaline sulphide solutions, and investigate the potential for leaching antimony from a DELA calcine sample.

Two calcine leach tests were conducted at SGSM in order to evaluate the potential for selective leach extraction of antimony for sale as a byproduct. This process would have

the added benefit of reducing the portion of the final concentrate smelter penalty associated with antimony. The feed used for this testwork consisted of a composite sample of calcine reject from the roasting tests conducted at DELA. XRD analysis indicated that the antimony contained in the feed sample was present predominantly as tetrahedrite, but that there was some iron-antimony-oxide (tripuhyite) as well. A caustic sulphide leach, under conditions that had proven effective for the Strieborná concentrate, was selected for the first leaching test.

Results indicated relatively poor antimony extraction. In the initial test, after eight hours of leaching, only 22% of the antimony had been leached, even with the use of relatively high reagent additions and a low pulp density. In the follow-up test, the sample was hand pulverized to break up any fused particles that may have been generated in the calcining step, and the sodium sulphide concentration was increased. Despite these changes, the leaching of antimony increased only slightly, to 30% after eight hours of leaching.

Despite this poor result, the option to treat the concentrate by roasting to remove the mercury first, followed by leaching of the antimony, still appears to be the preferred approach to producing a silver/copper concentrate with the lowest smelter penalties. Further testing is needed, as the formation of the insoluble tripuhyite might be averted if the roast is conducted in nitrogen, thus allowing for subsequent leaching of the antimony.

The principal benefits to this approach could be as follows:

- Roasting provides an effective and robust means of mercury removal, and produces a stable mercury end product.
- Roasting is available by toll-treatment at a third-party facility, eliminating the need for capital investment, or process permitting.
- The “Sunshine Process” has been commercially applied to recovering antimony from tetrahedrite concentrates using caustic sulphide leaching.
- An antimony removal step can be added to the flowsheet to reduce the smelter penalties and generate a saleable by-product, but does not have to be part of the process from day 1. This provides an opportunity to fully develop the process and plant design, as well as defer capital costs to later in the project.

The fundamental advantage of the roast/leach option is that it disconnects the essential step of mercury removal from the value-adding step of antimony removal, while also providing the most robust technology for that essential step.

The potential remains for recovery of both mercury and antimony in a single hydrometallurgical flowsheet, but the results to date clearly indicate that much more testwork is needed to develop confidence in this approach. Specifically, the conditions required to consistently achieving a residue mercury grade of less than 50 ppm need to

be better defined and proven, and the method for precipitating mercury from the leachate into a stable product for disposal will require a significant amount of development work to achieve a viable flowsheet.

14 MINERAL RESOURCE ESTIMATES

MDA has completed an updated mineral resource estimate that includes new underground channel and panel sampling completed by Global Minerals since the previous estimate by MDA (Holloway et al., 2013). In addition, MDA expects that further exploration of the Podložná vein is likely to increase the estimated resource. The Pallag vein was not estimated because the data available indicate that it is low grade.

14.1 Introduction

MDA classifies resources in order of increasing geological and quantitative confidence into Inferred, Indicated, and Measured categories to be in compliance with the “CIM Definition Standards – For Mineral Resources and Mineral Reserves” (2014) and therefore Canadian National Instrument 43-101. CIM mineral resource definitions are given below, with CIM’s explanatory material shown in italics.

14.1.1 Mineral Resource

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated, and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth’s crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated, or interpreted from specific geological evidence and knowledge, including sampling.

Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase ‘reasonable prospects for eventual economic extraction’ implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic

extraction. Assumptions should include estimates of cut-off grade and geological continuity at the selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing.

Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

14.1.2 Inferred Mineral Resource

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.

14.1.3 Indicated Mineral Resource

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with

sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Pre-Feasibility Study, which can serve as the basis for major development decisions.

14.1.4 Measured Mineral Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

Modifying Factors

Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social, and governmental factors.

MDA reports resources at cut-offs that are reasonable for deposits of this nature given anticipated mining methods and plant processing costs, while also considering economic conditions, because of the regulatory requirements that a resource exists “in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction.”

14.2 Database

The exploration database used for the resource estimate is described in Table 14-1. The samples are made up of core and underground samples. The core was drilled by the SGS, and more recently by Global Minerals. The underground samples are almost entirely from the SGS and CMX, but Global Minerals has undertaken a small quantity of underground channel and panel sampling since the last estimate.

Table 14-1: Descriptive Statistics of the Strieborná Database – Entire Database

	Unit	Valid N	Median	Mean	Std. Dev.	CV	Minimum	Maximum
Length	m	2,615	-	12.66	-	-	0.10	79.69
Ag	g/t	1,833	50.80	190.33	354.62	1.86	0.01	4,660.00
Au	g/t	586	0.015	0.081	0.194	2.39	0.000	1.885
Fe	%	1,545	31.180	28.588	11.738	0.41	1.100	42.830
SiO ₂	%	791	20.680	24.294	20.729	0.85	0.250	83.450
Cu	%	1,834	0.298	0.914	1.565	1.71	0.001	18.450
Sb	%	1,833	0.155	0.589	1.066	1.81	0.000	12.500
Hg	%	1,751	0.004	0.017	0.039	2.29	0.000	0.566
Bi	%	667	0.003	0.017	0.043	2.57	0.000	0.728
As	ppm	895	190.0	634.7	1,471.6	2.3	5.0	13,350.0
Pb	ppm	640	30.0	183.8	583.2	3.2	0.0	11,200.0
Zn	ppm	640	110.0	315.0	558.1	1.8	5.0	4,750.0
Mn	%	1,216	1.465	1.295	0.723	0.56	0.010	9.740
Mg	%	611	1.280	1.121	0.705	0.63	0.050	2.940
Ni	g/t	611	0.000	0.002	0.005	2.59	0.000	0.050
P	ppm	560	25.0	131.3	168.4	1.3	0.0	1,310.0
Core Recovery	%	489	100.0	99.3	5.0	0.1	26.6	100.0
RQD	%	489	70.5	66.2	27.0	0.4	0.0	195.9
Density	g/cm ³	108	3.30	3.25	0.45	0.14	2.63	4.09

One important point on the sampling is that most core drilling has intersections in the upper parts of the deposit, and most underground sampling is deeper (Figure 14-1). Most core drilling has variable length samples, while most underground samples in the database are one metre in length or longer. The consequence of this is that the perspective of the deposit is quite different based on those samples, in that it appears that the grades are more regular at depth and more erratic higher in the system.

Figure 14-1: Sample Distribution on the Strieborná Vein

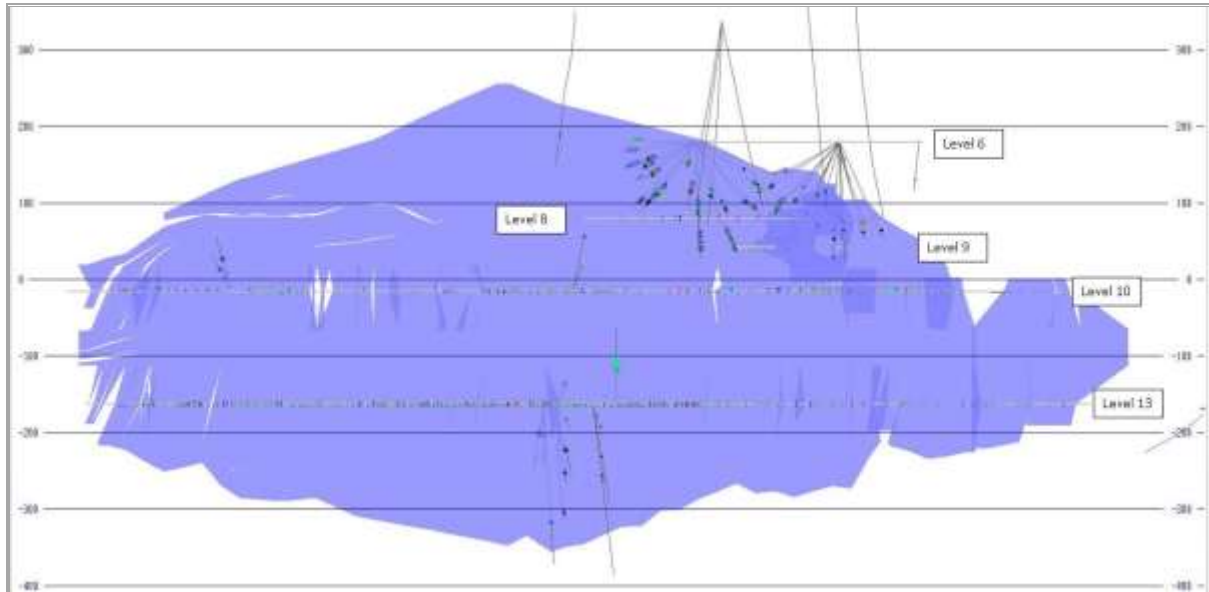


Table 14-2: Descriptive Sample Statistics of the Strieborná Vein

	Unit	Valid N	Median	Mean	Std. Dev.	CV	Minimum	Maximum
Strieborná Main Vein (55)								
Ag	g/t	1152	111.15	249.93	397.41	1.59	0.10	4,660.00
Ag Capped	g/t	1152	108.5	232.30	318.60	1.37	0.10	2,000.00
Au	g/t	354	0.037	0.124	0.239	1.920	0.000	1.885
Au Capped	g/t	354	0.037	0.124	0.239	1.920	0.000	1.885
Fe	%	863	33.800	33.279	6.701	0.201	1.680	42.830
SiO ₂	%	367	15.460	17.086	13.238	0.775	1.380	79.590
Cu	%	1152	0.512	1.133	1.731	1.528	0.001	18.450
Cu Capped	%	1152	0.512	1.099	1.510	1.374	0.001	9.000
Sb	%	1152	0.331	0.767	1.182	1.541	0.000	12.500
Sb Capped	%	1152	0.331	0.752	1.077	1.432	0.000	7.000
Hg	%	1103	0.008	0.022	0.045	1.989	0.000	0.566
Hg Capped	%	1103	0.008	0.022	0.039	1.784	0.000	0.250
Core Recovery	%	282	100.0	100.0	0.0	0.0	100.0	100.0
RQD	%	282	82.4	77.5	22.8	0.3	9.6	195.9
Density	g/cm ³	62	3.65	3.50	0.36	0.10	2.63	4.09

Table 14-3: Descriptive Sample Statistics of the Strieborná West Split

	Unit	Valid N	Median	Mean	Std. Dev.	CV	Minimum	Maximum
Strieborná West Split (56)								
Ag	g/t	23	77.00	164.53	246.34	1.50	3.00	1,280.00
Ag Capped	g/t	23	66.00	130.74	164.85	1.26	0.25	645.00
Au	g/t	23	0.034	0.061	0.083	1.35	0.003	0.404

	Unit	Valid N	Median	Mean	Std. Dev.	CV	Minimum	Maximum
Au Capped	g/t	23	0.034	0.061	0.083	1.35	0.003	0.404
Fe	%	23	32.600	30.684	6.761	0.22	8.060	38.400
SiO ₂	%	0						
Cu	%	23	0.359	0.669	0.988	1.48	0.012	5.020
Cu Capped	%	23	0.359	0.597	0.731	1.23	0.012	2.500
Sb	%	23	0.235	0.492	0.735	1.49	0.009	3.750
Sb Capped	%	23	0.235	0.459	0.603	1.31	0.009	2.500
Hg	%	23	0.0096	0.0167	0.0255	1.5273	0.0006	0.1310
Hg Capped	%	23	0.0096	0.0159	0.0221	1.3900	0.0006	0.1000
Core Recovery	%	23	100.0	100.0	0.0	0.0	100.0	100.0
RQD	%	23	50.9	56.2	19.1	0.3	22.0	96.3
Density	g/cm ³	0						

Table 14-4: Descriptive Sample Statistics of the Podložná Main Vein (75)

	Unit	Valid N	Median	Mean	Std. Dev.	CV	Minimum	Maximum
Podložná Main Vein (75)								
Ag	g/t	35	32.00	54.50	43.15	0.79	1.00	133.00
Ag Capped	g/t	35	32.00	54.50	43.15	0.79	1.00	133.00
Au	g/t	0						
Au Capped	g/t	0						
Fe	%	35	38.030	35.134	7.157	0.20	5.660	42.510
SiO ₂	%	35	9.110	12.943	13.460	1.040	0.250	69.820
Cu	%	35	0.267	0.415	0.298	0.72	0.003	1.110
Cu Capped	%	35	0.267	0.415	0.298	0.72	0.003	1.110
Sb	%	35	0.140	0.191	0.135	0.71	0.000	0.473
Sb Capped	%	35	0.140	0.191	0.135	0.71	0.000	0.473
Hg	%	35	0.0050	0.0084	0.0081	0.9622	0.0000	0.0250
Hg Capped	%	35	0.0050	0.0084	0.0081	0.9622	0.0000	0.0250
Core Recovery	%	0						
RQD	%	0						
Density	g/cm ³	0						

14.3 Mineral Domains

Essentially, all mineralization of consequence lies within the veins. The contacts are often abrupt, and any mineralization outside the vein is considered to be discontinuous veining and/or very weak dissemination. In both cases, these are deemed inconsequential. Furthermore, the metals behave similarly within the limits of the vein. MDA compared quantile plots of the grades, compared variograms of the different metals, and noted that at least the silver, copper, and antimony occur together, probably in the same mineral tetrahedrite. For these reasons, the veins were modelled as single domains.

The single domain, the vein, has a small high-grade population and a small low-grade population, each of which represents about five percent of the total population. The low-grades are generally unmineralized horses (pieces of country rock); the high-grades represent locations of dense, massive tetrahedrite. The same domains that MDA noted geologically and statistically, were noted, albeit more subjectively, by Jakubiak (1994), who gives good geologic descriptions of the mineralization, some of which have been restated in previous sections.

14.4 Density

Global Minerals measured density for 152 samples. Of these, 62 were coded as veins, with all of them in the Strieborná vein. Several were averaged together as they occurred within the same sample interval. In some cases, density samples fell outside sample intervals and were not included in samples coded by the modelling. Descriptive statistics of those density data are presented in Table 14-5 along with descriptive statistics of iron content, which is closely related to density. The SGS calculated a regression formula ($\text{density} = 2.5977 + 0.027071 \times \text{Fe} (\%)$) and a correlation coefficient of 0.83. Those analyses were based on 60 samples from levels 10 and 8. From Global Minerals' data, MDA calculated a regression formula ($\text{density} = 2.7897 + 0.023 \times \text{Fe} (\%)$) and a correlation coefficient of 0.48. Global Minerals' data have significant scatter. It is likely that any data set from Strieborná would have high variability because the two main gangue minerals, quartz and siderite, have quite different density values, 2.7 g/cm^3 and 3.9 g/cm^3 , respectively. MDA chose 3.55 g/cm^3 for the density to assign to the Strieborná vein. That value was decided upon considering the mean, median, and the two regression analyses, all given in Table 14-5. There are no density values for the other veins, so the two regression analyses were used to define a density value from the iron grades reported for those samples. MDA assigned density values of 3.48 g/cm^3 and 3.60 g/cm^3 for the Strieborná West Split and Podložná veins, respectively.

Table 14-5: Descriptive Statistics of Global Minerals' Vein Sample Density

	Unit	Valid N	Median	Mean	Std. Dev.	CV	Minimum	Maximum
Strieborná Main Vein (55)								
Density	g/cm^3	62	3.650	3.495	0.364	0.104	2.630	4.090
Fe	%	826	33.800	33.304	6.717	0.202	1.680	42.830
	3.55	SGS regression	3.51	33.55	3.55	Global Minerals regression	Value used	

14.5 Composites and Variography

MDA capped sampled data (Table 14-2, Table 14-3 and Table 14-4) prior to compositing to two metres. Descriptive statistics of the composites by vein are given in Table 14-6

Table 14-6: Descriptive Statistics of Composite Grades by Vein

	Unit	Valid N	Median	Mean	Std. Dev.	CV	Minimum	Maximum
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	Unit	Valid N	Median	Mean	Std. Dev.	CV	Minimum	Maximum
Strieborná Main Vein (55)								
Length	m	1165		1.63			0.00	2.00
Ag	g/t	1160	125.00	249.93	351.55	1.41	0.10	3571.75
Ag Capped	g/t	1160	124.75	232.30	291.63	1.26	0.10	2000.00
Au	g/t	178	0.062	0.124	0.185	1.488	0.000	1.363
Au Capped	g/t	178	0.062	0.124	0.185	1.488	0.000	1.363
Fe	%	851	34.66	33.28	6.12	0.18	3.04	41.89
Cu	%	1160	0.60	1.13	1.55	1.37	0.00	15.07
Cu Capped	%	1160	0.60	1.10	1.37	1.25	0.00	9.00
Sb	%	1160	0.387	0.767	1.062	1.385	0.000	10.280
Sb Capped	%	1160	0.387	0.752	0.978	1.301	0.000	7.000
Hg	%	1127	0.008	0.022	0.040	1.763	0.000	0.436
Hg Capped	%	1127	0.008	0.022	0.035	1.612	0.000	0.250
Strieborná West Split (56)								
Type	m	8		1.90			2.00	2.00
Ag Capped	g/t	8	98.26	164.14	143.63	0.88	26.25	290.05
Au	g/t	8	98.26	130.42	104.11	0.80	0.02	0.13
Au Capped	g/t	8	0.034	0.061	0.047	0.764	0.016	0.131
Fe	g/t	8	0.034	0.061	0.047	0.764	20.861	37.500
Cu	%	8	33.52	30.68	5.27	0.17	0.10	1.66
Cu Capped	%	8	0.39	0.67	0.58	0.87	0.10	1.17
Sb	%	8	0.39	0.60	0.48	0.80	0.08	1.24
Sb Capped	%	8	0.296	0.491	0.430	0.876	0.075	0.991
Hg	%	8	0.296	0.458	0.377	0.825	0.003	0.041
Hg Capped	%	8	0.009	0.017	0.015	0.887	0.003	0.035
Vein C	%	8	0.009	0.016	0.014	0.852	56.000	56.000
Podložná Main Vein (75)								
Length	m	46		1.64			0.20	2.00
Ag	g/t	46	36.46	54.50	43.07	0.79	1.00	133.00
Ag Capped	g/t	46	36.46	54.50	43.07	0.79	1.00	133.00
Au	g/t	0						
Au Capped	g/t	0						
Fe	%	46	37.18	35.13	7.15	0.20	5.66	42.45
Cu	%	46	0.27	0.41	0.29	0.71	0.00	1.03
Cu Capped	%	46	0.27	0.41	0.29	0.71	0.00	1.03
Sb	%	46	0.157	0.191	0.135	0.706	0.000	0.473
Sb Capped	%	46	0.157	0.191	0.135	0.706	0.000	0.473
Hg	%	46	0.005	0.008	0.008	0.962	0.000	0.025
Hg Capped	%	46	0.005	0.008	0.008	0.962	0.000	0.025

Correlograms of the capped grades composited to three metres were made for each metal in multiple directions. Not surprisingly, since most of the metals occur together, the calculated variograms were all very similar. Each had a nugget between 0.4 and 0.6, ranges in the strike direction (where there are by far the most pairs) between 30 m and 70 m. Down-dip ranges are shorter, but that may well be because of the sample

distribution. Correlogram models are dependent on sample distribution, which is a major factor in the Strieborná vein (Figure 14-1).

14.6 Estimation

Correlograms of the capped grades composited to three metres were made for each metal in multiple directions. Not surprisingly, since most of the metals occur together, the calculated variograms were all very similar. Each had a nugget between 0.4 and 0.6, ranges in the strike direction (where there are by far the most pairs) between 30 m and 70 m. Down-dip ranges are shorter, but that may well be because of the sample distribution. Correlogram models are dependent on sample distribution, which is a major factor in the Strieborná vein (Figure 14-1).

14.7 Estimation

The history and geologic development of the vein is very complex, with multiple stages of mineralization and deformation. Global Minerals believes that the most likely plunge to the mineralization is northeast within the vein. As a consequence, MDA instilled that bias in the estimate. However, because the sample spacing and locations (Figure 14-1) are so restricted and not well distributed, it very difficult to define or mimic, postulated metal distribution.

MDA ran estimates using nearest neighbour, ordinary kriging, and inverse distance algorithms. The reported resource employed the inverse distance to the fourth power estimate. Because of the restricted number of samples and difficult geometry of the sample spacing, MDA ran many estimates to best replicate perceived metal distributions, while honoring the underlying data. MDA also ran many estimates attempting to minimize differences between the three estimates. The chosen estimation parameters are given in Table 14-7. MDA also attempted to use shorter ranges than those finally chosen, but the results, again caused by the restricted sample distribution, cause unacceptably and unrealistically abrupt changes in grade. As a consequence, very long ranges were used and in only one single pass. The effect of such long ranges is mostly limited to Inferred material. High-grade sample grade projections were restricted.

Estimation on the Strieborná vein was done in eight “zones” with varying dips and azimuths to better reflect the changing dips and azimuths of the vein (the propeller form). An octant search was used to de-cluster the composite data, and anisotropic weighting was used to give the estimate more defined plunge.

Table 14-7: Estimation Parameters

Description	Parameter
<i>Strieborná Vein Estimation - Parameters</i>	
<i>Silver, Copper, Antimony, and Mercury</i>	
Samples: minimum/maximum/maximum per hole	1 / 16 / 2
Rotation/Dip/Tilt (searches)	40° to 70° / -30° / 65° NW to 60° SE

Description	Parameter
Search (m): major/semi-major/minor (vertical)	400 / 400 / 130
Inverse-distance power	4
Split octant search	3 comps per "octant"
High-grade restrictions (grade and distance)	1,000 g/t Ag / 50 m 4.0% Cu / 50 m 0.1% Hg / 50 m 3.5% Sb / 50 m
<i>Iron</i>	
Samples: minimum/maximum/maximum per hole	1 / 16 / 2
Rotation/Dip/Tilt (searches)	40° to 70° / -30° / 65° NW to 60° SE
Search (m): major/semi-major/minor (vertical)	400 / 400 / 130
Inverse-distance power	4
Split octant search	3 comps per "octant"
High-grade restrictions (grade and distance)	N/A
Podložná and West Split of Strieborná - Estimation Parameters	
<i>Silver, Copper, Antimony, and Mercury</i>	
Samples: minimum/maximum/maximum per hole	1 / 16 / 2
Rotation/Dip/Tilt (searches; Podložná: West Split)	55° / -30° / 90 : 45° / -30° / 80
Search (m): major/semi-major/minor (vertical)	50 / 25 / 25
Inverse-distance power	4
Split octant search	Podložná – 3 West Split - No
High-grade restrictions (grade and distance)	N/A
<i>Iron</i>	
Samples: minimum/maximum/maximum per hole	1 / 16 / 2
Rotation/Dip/Tilt (searches; Podložná: West Split)	55° / -30° / 90 : 45° / -30° / 80
Search (m): major/semi-major/minor (vertical)	50 / 25 / 25
Inverse-distance power	4
Search (composites per split octant)	3 comps per split octant at Podložná No search selection preference at West Split
High-grade restrictions (grade and distance)	N/A

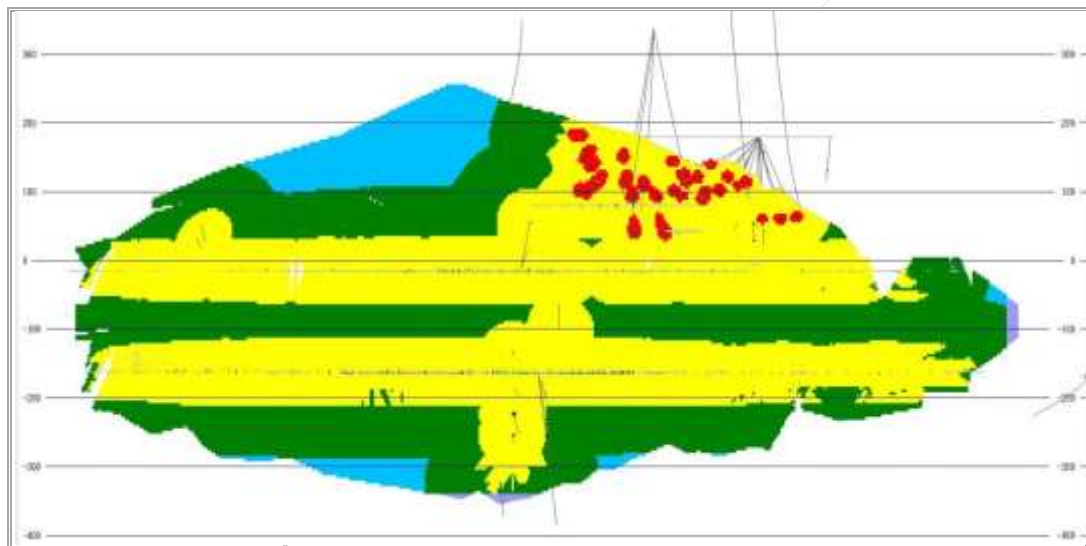
14.8 Classification

MDA classified the resources based on confidence in and/or support for underlying information (i.e., QA/QC, original certificates), number of samples and/or drill holes, and distances to the nearest sample. Different criteria were used for each vein. In the Strieborná vein, Measured, Indicated, and Inferred categories were permissible. Podložná was permitted to have only Indicated and Inferred, as that vein has not yet been drilled by Global Minerals. The West Split only has Inferred, because its continuity is not well established with only two drill hole intersections.

In the Strieborná database, a "drill hole" can be either an actual drill hole or underground sampling with as little as just one sample.

During estimation, all blocks that met the criteria for Inferred were assigned to that category. Inferred blocks that met the criteria for Indicated were reassigned to that category. For the Strieborná vein, MDA classified material as Measured where the closest sample was a Global Minerals core sample, the block was already coded as Indicated, and the closest Global Minerals composite was within 10 m of the block. Indicated blocks needed to have been classified as Indicated for both silver and copper to be classified as Indicated in the final model. To be classified as Indicated for silver or copper individually in the Strieborná vein, a block had to be within 50 m of seven “drill holes”, or within 25 m of a sample with at least four “drill holes”. Inferred material is the remaining material that is within 125 m of a sample. The distribution of the three classes of resource blocks is presented in Figure 14-2.

Figure 14-2: Resource Classification on the Strieborná Vein



Note: Red is Measured, yellow is Indicated, green is Inferred, blue is not classified or reported

For the Podložná vein, Indicated blocks needed to have been classified as Indicated for both silver and copper to be classified as Indicated in the final model. To be classified as Indicated for silver or copper individually in the Podložná vein, a block had to be within 20 m of four “drill holes.” Inferred material is the remaining material that is within and within 50 m of a sample.

14.9 Mineral Resource

The mineral resource estimates presented herein represent the third NI 43-101 compliant resource completed for Global Minerals at Strieborná, and the second Strieborná estimate performed by MDA. MDA’s resource estimates are the first to use Global Minerals’ drill data, all of which is core, and their recent channel and panel samples.

The Strieborná resource estimates have been prepared in compliance with the disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 (NI 43-101), Companion Policy 43-101CP, and Form 43-101F1, as well as with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Reserves, Definitions and Guidelines" (CIM Standards) adopted by the CIM Council on May 10, 2014.

The Strieborná block model is composed of blocks one-metre wide (across the vein), three-metres long (along strike), and four-metres high. Estimates were made using inverse distance, kriging, and nearest neighbour. The inverse distance estimate is reported, but the other two methods were used for comparison and validation.

Because of the requirement that the resource exists "in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction," MDA is reporting the resources at approximate economic cut-off grades based on operating costs estimated by AGP Mining Consultants. As such, economic considerations were used to determine cut-off grades at which the resource is presented. MDA considers the metal prices, extraction costs, and recoveries, to be reasonable, and input slight reductions of these parameters to account for material that would become economic using internal cut-off grades. For the same requirements of "...reasonable prospects of economic extraction", even though an estimate was done for mercury, MDA is not reporting mercury as a resource because, reportedly by Global Minerals, the European Union does not allow mercury to be produced for sale. At the reporting cut-off of 100 g/t Ag, the mercury grade is 0.02%. MDA does not anticipate that iron will be a saleable commodity in spite of past production of iron from siderite veins; however, the grades are being reported for information and for consistency to previous resource reporting.

At the request of Global Minerals, MDA is reporting the resource based on silver cut-off grades. The reporting cut-off grade is 100 g/t Ag. The mineral resource tabulations for Measured and Indicated, Measured, Indicated, and Inferred are presented in Table 14-8, Table 14-7, Table 14-10, and Table 14-11, respectively. These are undiluted resources although any mine plan will include reasonable mine dilution.

Table 14-8: Measured and Indicated Resources

Cut-off (g/t Ag)	Measured and Indicated							
	Tonnes	g/t Ag	%Cu	%Sb	%Fe	oz Ag	Tonnes Cu	Tonnes Sb
50	2,864,000	233	1.07	0.74	33.9	21,434,000	30,580	21,260
75	2,635,000	247	1.13	0.79	34.1	20,967,000	29,740	20,780
100	2,346,000	267	1.21	0.85	34.1	20,155,000	28,410	19,930

Cut-off (g/t Ag)	Measured and Indicated							
	Tonnes	g/t Ag	%Cu	%Sb	%Fe	oz Ag	Tonnes Cu	Tonnes Sb
120	2,105,000	285	1.28	0.9	34.1	19,304,000	27,030	18,990
140	1,879,000	304	1.36	0.96	34.1	18,360,000	25,540	18,000
160	1,661,000	324	1.44	1.02	34.1	17,312,000	23,930	16,910
180	1,484,000	342	1.52	1.07	34.2	16,340,000	22,490	15,910
200	1,309,000	363	1.6	1.13	34.2	15,273,000	20,940	14,800
250	932,000	419	1.84	1.3	34.3	12,563,000	17,130	12,120
300	677,000	474	2.07	1.47	34.2	10,323,000	14,000	9,930
400	346,000	599	2.58	1.82	34.4	6,658,000	8,940	6,310

Table 14-9: Measured Resources

Cut-off (g/t Ag)	Measured							
	Tonnes	g/t Ag	%Cu	%Sb	%Fe	oz Ag	Tonnes Cu	Tonnes Sb
50	172,000	250	1.23	0.90	30.5	1,380,000	2,110	1,550
75	155,000	270	1.33	0.98	30.8	1,347,000	2,070	1,510
100	147,000	281	1.38	1.01	31.0	1,324,000	2,030	1,480
120	135,000	295	1.45	1.06	31.0	1,282,000	1,960	1,430
140	125,000	308	1.51	1.10	31.1	1,242,000	1,890	1,380
160	114,000	323	1.53	1.11	31.2	1,189,000	1,750	1,270
180	105,000	337	1.58	1.15	31.3	1,137,000	1,660	1,200
200	92,000	358	1.67	1.21	31.3	1,056,000	1,530	1,110
250	68,000	406	1.89	1.37	31.1	887,000	1,290	930
300	54,000	438	2.03	1.46	31.1	768,000	1,100	800
400	28,000	525	2.34	1.66	31.2	475,000	660	470

Table 14-10: Indicated Resources

Cut-off (g/t Ag)	Indicated							
	Tonnes	g/t Ag	%Cu	%Sb	%Fe	oz Ag	Tonnes Cu	Tonnes Sb
50	2,692,000	232	1.06	0.73	34.1	20,054,000	28,470	19,710
75	2,480,000	246	1.12	0.78	34.3	19,620,000	27,670	19,270

100	2,199,000	266	1.2	0.84	34.3	18,831,000	26,380	18,450
120	1,970,000	285	1.27	0.89	34.3	18,022,000	25,070	17,560
140	1,754,000	304	1.35	0.95	34.3	17,118,000	23,650	16,620
160	1,547,000	324	1.43	1.01	34.4	16,123,000	22,180	15,640
180	1,379,000	343	1.51	1.07	34.4	15,203,000	20,830	14,710
200	1,217,000	363	1.6	1.13	34.5	14,217,000	19,410	13,690
250	864,000	420	1.83	1.29	34.5	11,676,000	15,840	11,190
300	623,000	477	2.07	1.47	34.4	9,555,000	12,900	9,130
400	318,000	605	2.6	1.84	34.7	6,183,000	8,280	5,840

Table 14-11: Inferred Resources

Cut-off (g/t Ag)	Inferred							
	Tonnes	g/t Ag	%Cu	%Sb	%Fe	oz Ag	Tonnes Cu	Tonnes Sb
50	1,262,000	191	0.89	0.62	33.4	7,754,000	11,250	7,770
75	1,123,000	207	0.96	0.67	33.79	7,471,000	10,730	7,480
100	986,000	224	1.02	0.72	33.91	7,090,000	10,080	7,070
120	871,000	239	1.08	0.76	33.89	6,686,000	9,400	6,610
140	764,000	254	1.13	0.80	33.87	6,236,000	8,650	6,140
160	672,000	268	1.19	0.84	33.93	5,795,000	7,970	5,670
180	578,000	284	1.24	0.89	33.96	5,284,000	7,190	5,130
200	485,000	302	1.31	0.94	34.01	4,713,000	6,350	4,560
250	296,000	353	1.49	1.08	34.18	3,365,000	4,420	3,210
300	188,000	399	1.64	1.20	34.1	2,410,000	3,070	2,240
400	87,000	470	1.9	1.32	33.31	1,316,000	1,650	1,150

Long sections of the silver grade, silver grade times thickness, and thickness are given in Figure 14-3, Figure 14-4, and Figure 14-5. Three cross-sections of the silver model showing undiluted grades are given in Figure 14-6, Figure 14-7, and Figure 14-8. Figure 10-1 shows the locations of those sections.

In the long sections, Measured, Indicated, and Inferred blocks are shown. The Cyan colour shows projected vein material not classified.

Figure 14-3: Long Section of Silver Grades on the Strieborná Vein – Looking Southeast

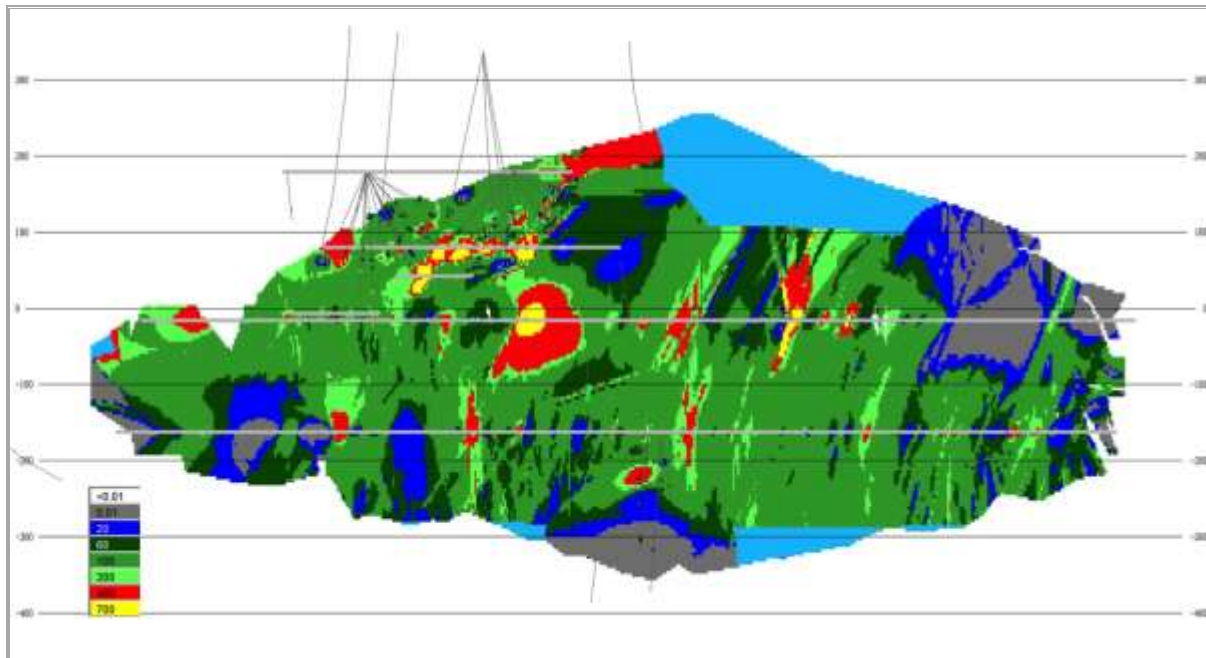


Figure 14-4: Long Section of Silver Grades Times Thickness on the Strieborná Vein - Looking Southeast

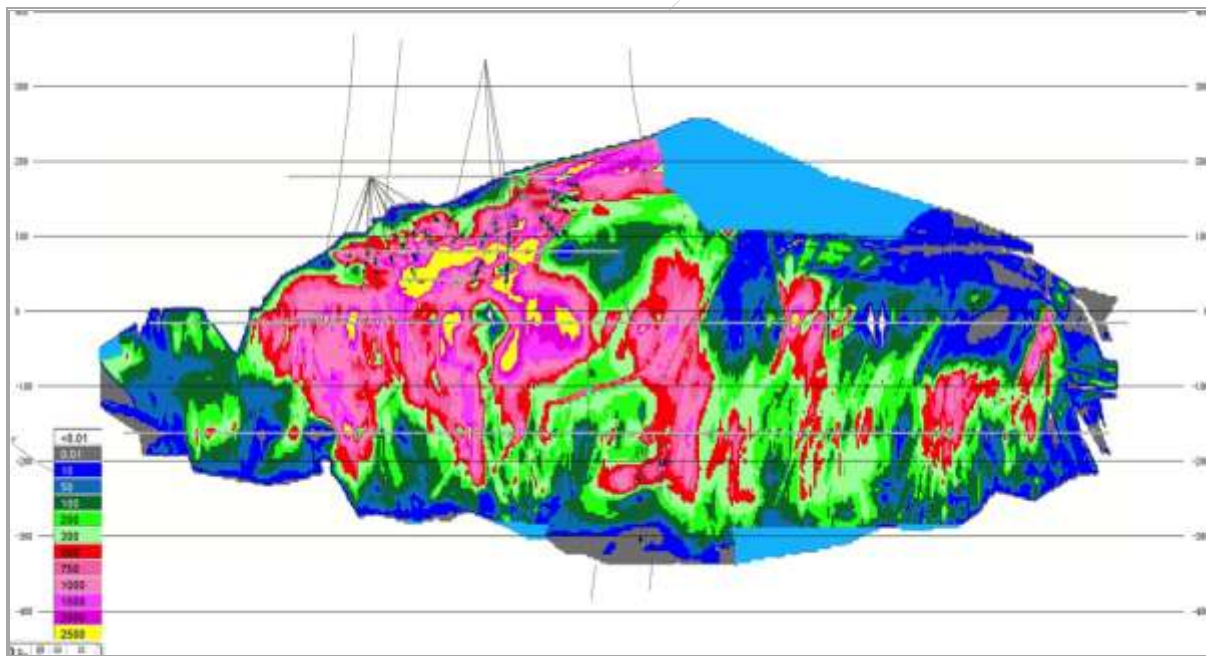


Figure 14-5: Long Section of Vein Thickness on the Strieborná Vein – Looking Southeast

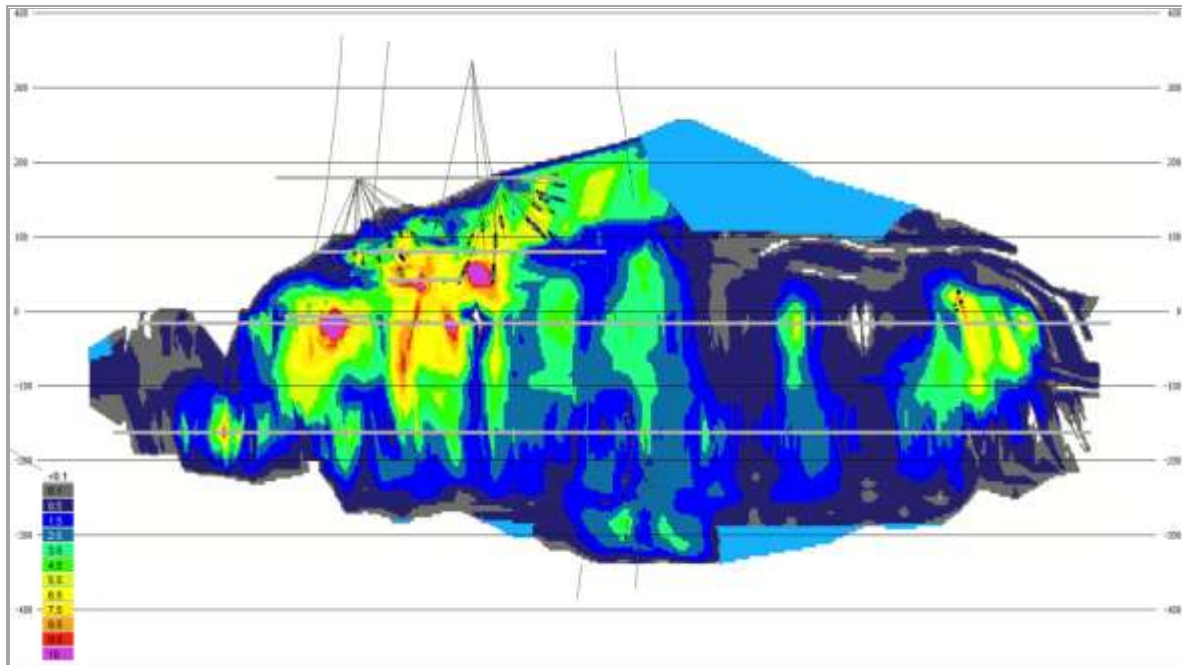


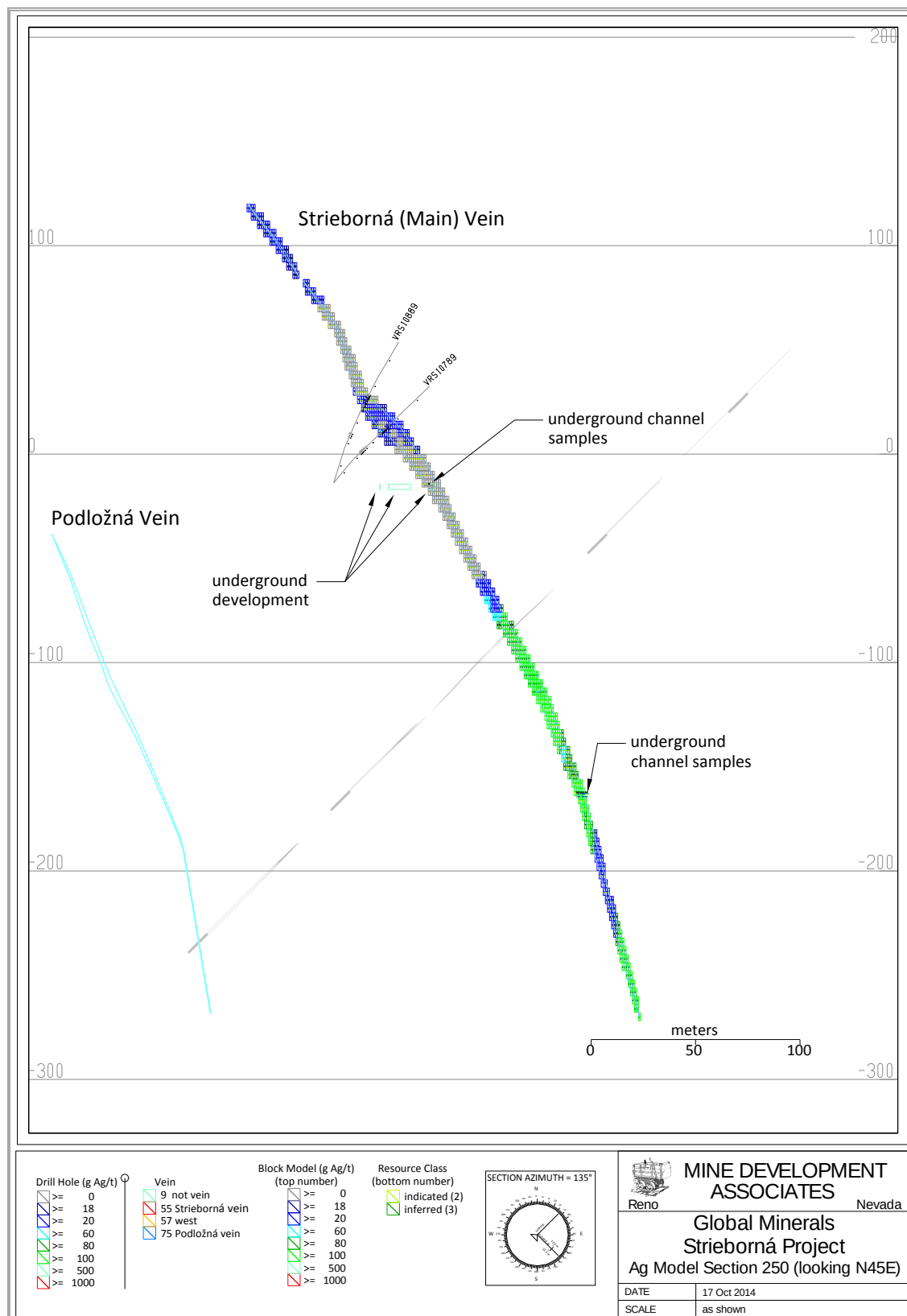
Figure 14-6: Cross-Section of the Undiluted Silver Model Grades – Section 250

Figure 14-7: Cross-Section of the Undiluted Silver Model Grades – Section 800

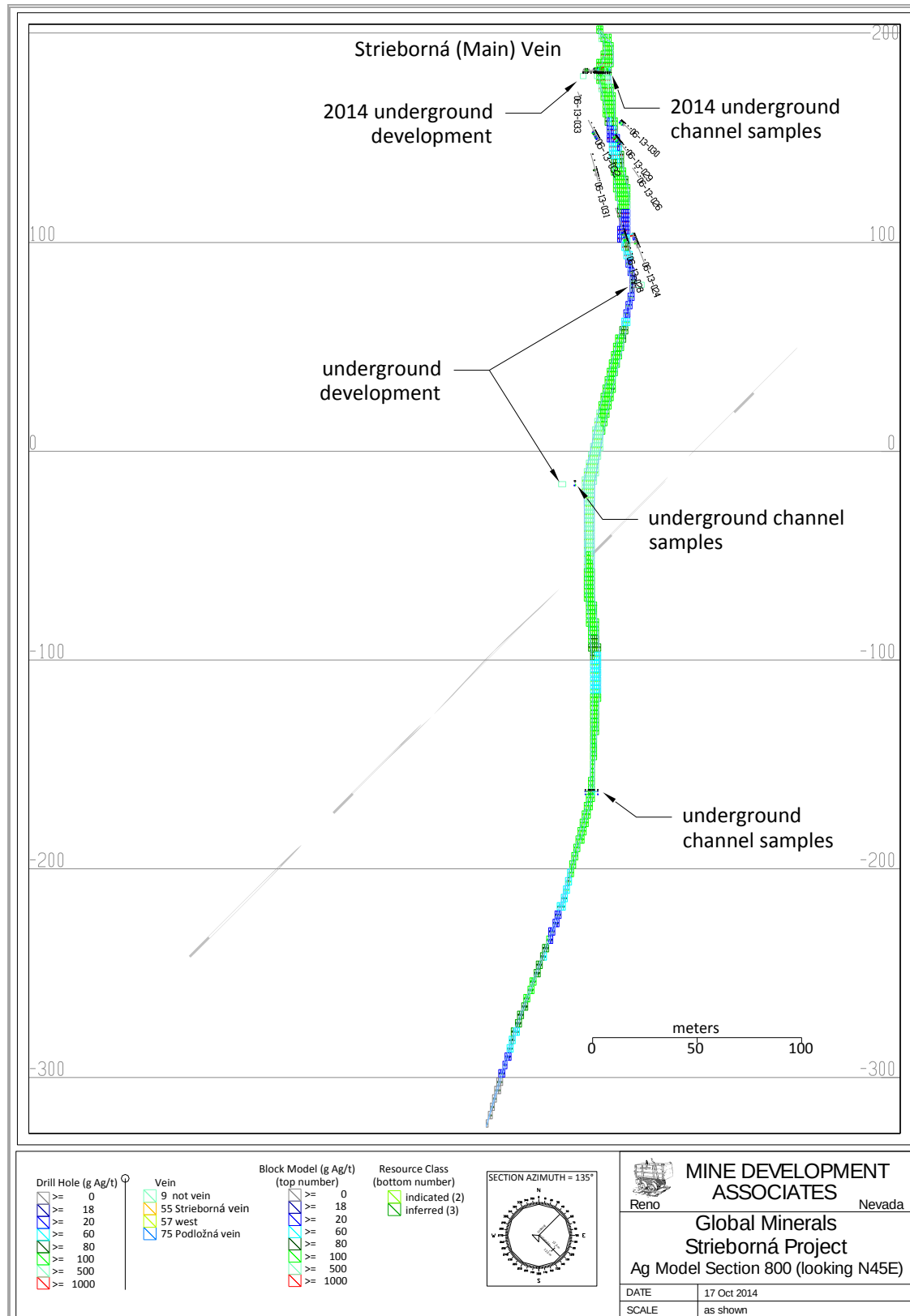
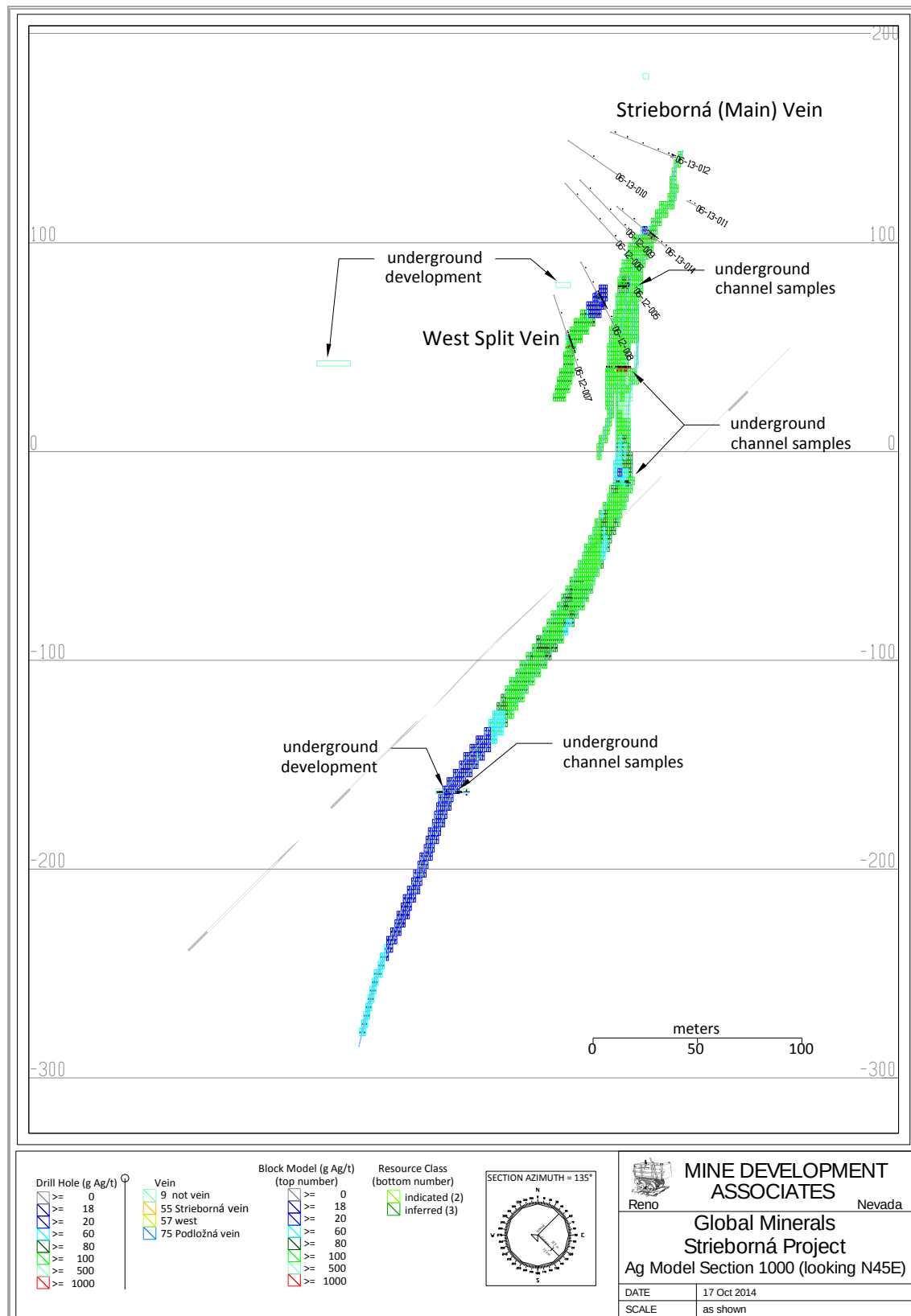


Figure 14-8: Cross-Section of the Undiluted Silver Model Grades – Section 1000



14.10 Discussion of Resources

Overall, Global Minerals' work done to date is very good. While their work only defines a small part of the deposit by drilling in the upper northeast portion of the Strieborná vein, it confirms historical work, perceptions, and interpretations of the veins. MDA cannot state with first-hand experience that the historical work is of similar caliber to Global Minerals, but during interpretation and evaluation of the historical data, MDA has been impressed with both the SGS' and CMX's work, documentation, and understanding. Duplicate sampling by CMX on the SGS underground sampling did show some, as yet not understood, high and low biases between the two sets of data, but the work demonstrated reasonableness for both sets of data.

A very important aspect of the Strieborná resource is that the previous resource estimates done over the years, including the one by MDA in December 2012/2013 for internal decision-making purposes, generated substantially similar results.

The differences in global tonnes and grade between the 2008 AMC estimate and the 2014 MDA estimate are minimal; however, the procedures and parameters used are substantially different. AMC made a grade-thickness model, MDA made a three-dimension block model; both can be appropriate for a deposit of this kind. For Measured, MDA required Global Minerals data to have been used to estimate a block; consequently, Measured material is only in the northeast and upper part of the vein. The 2008 estimate reported all material lying within the vein "without a cut-off grade applied as the form of the vein and the mean silver, copper, antimony, and iron grades are such that the Strieborná vein has reasonable prospects for economic extraction in its entirety." The 2014 estimate's total volume is substantially larger (29%) but the vein carries material below cut-off grade, which is not currently economic; the 2014 estimate does not report material below cut-off grade as a resource. AMC did not report the resource at higher cut-off grades so a direct comparison cannot be made. However, comparing the 2014 resource at 100 g/t Ag to the AMC resource (at no cut-off, as it was reported), the total tonnes dropped by 8%, the silver grade stayed the same, the copper grade increased by 11%, antimony grade by 25%, and iron grade by 5%.

As more work is done at Strieborná, more splays and structural complications will become evident and high-grade shoots will be better defined. The available data, except in the areas of most concentrated sampling, do not allow for defining and modelling what is expected to be more complicated than a simple vein.

The restricted sample distribution defining the Strieborná mineralization made this a very complicated estimate, ameliorated only by the continuity of the mineralization. MDA performed tens of different estimates attempting to impart grade distributions that were both reasonable and without artifacts of the restricted sample distribution. The chosen estimate also was required to, and did, honor the underlying sample grades and geology while not imparting a bias. MDA believes that this was achieved.

However, MDA also accepts and conveys to the reader that grade distribution *will* be different once additional sampling is completed. While MDA made every attempt to make a reasonable estimate and it is classified accordingly, the estimate did require extraordinarily long searches. MDA expects that the total metal content and grades are reasonable, but away from the samples, grade distribution will be different. Note the difference in estimated grade distribution in the northeast part of the Strieborná vein (upper left in Figure 14-3) where there is substantial drilling by Global Minerals (upper right in Figure 14-1), as compared to the rest of the deposit, which is by far dominated by underground sampling along principally two levels.

The clustering of close-spaced sampling only on development levels does not permit modelling higher grades separately within the vein for the majority of the deposit. Furthermore, it remains unclear if these higher grades form continuous predictable zones within the vein. As a consequence, the estimated grades in the block model likely do not, and probably cannot, reflect expected variable grade changes within the plane of the vein. The Strieborná resource estimate block model therefore is, and can only be, a global estimate of total metal, without the precision or accuracy for final stope design and planning, except where there is detailed sampling and drilling.

There is the potential for additional mineralization beyond Inferred classification within the interpreted veins, but it is not reported. There is also very good potential to increase the resources by additional drilling on Strieborná and also on Pallag, but particularly on Podložná.

15 MINERAL RESERVE ESTIMATES

Mineral Reserves have not yet been estimated for the Project.



16 MINING METHODS

16.1 Underground Mining

16.1.1 Introduction

The mine design work and cost estimation was undertaken by AGP with assistance from BGC, which provided the geotechnical inputs and ground support recommendations.

16.1.2 Previous Mining

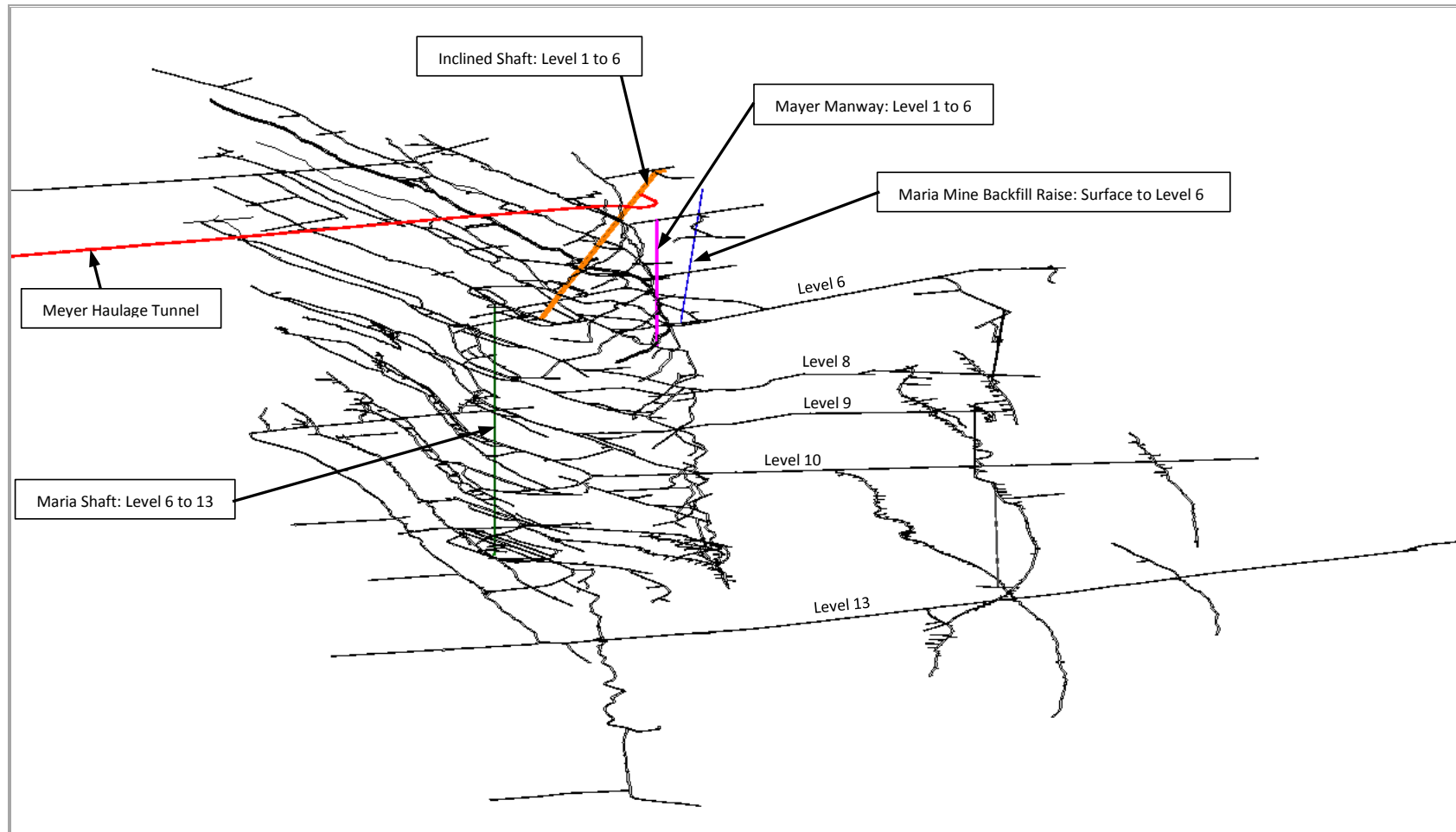
The Mária Mine, adjacent to the Strieborná vein, has been mined close to surface for many centuries. More recently, from 1951 to 1992, the underground mine produced around 2.3 Mt of ore before it was closed in the mid-1990s.

In addition, development drifts from a number of levels in the Mária Mine were driven over to the Strieborná deposit, and development was at least partially conducted along the vein in order to prepare the vein for future stoping. The existing Strieborná levels developed are Levels 6, 8, 9, 10, and 13. Figure 16-1 shows a layout of the existing Mária Mine development (exclusive of stoping) and the existing Strieborná development, with a number of relevant developments noted. In order to aid description, Table 16-1 shows the elevations of the mining levels.

Table 16-1: Elevations of Mining Levels

Level	Elevation
Top of Inclined Shaft	+360 m
Meyer Haulage Tunnel	+330 m (avg.)
Level 6	+180 m
Level 7	+130 m
Level 8	+80 m
Level 9	+30 m
Level 10	-20 m
Level 11	-70 m
Level 13	-160 m

Figure 16-1: Existing Mária Mine and Strieborná Vein Development (excludes Mária Mine Stopping)



Access from surface was via a 4.0 m wide x 3.0 m high shaft, inclined at 43 degrees, from the side of the east slope of Kalvária-Tri Vrchy ridge at Drazus, adjacent to outer dwellings of the town of Rožňava. The mine buildings and surface infrastructure were located around the shaft head. The inclined shaft was mined to Level 6. From Level 6 a 4.5 m diameter subvertical shaft was sunk down to Level 13. This shaft, known as the Mária Shaft, was lined with concrete bricks and furnished with steel and wooden internal structure. Extensive development of the Mária vein area was undertaken on Levels 6 to 13. Shrinkage and open-stoping mining methods, with some cut-and-fill, were used, using non-mechanized pneumatic mining equipment with a local workforce. Production operations progressed downwards, and the lowest level of stoping operations was at Level 10 when the mine ceased operation.

The Strieborná vein was discovered during the mining of long exploration cross-cuts, approximately 450 m in length, beyond the Mária vein hanging wall on Levels 6, 8, 9, 10, and 13. On each of these levels, development was extended along the strike of the Strieborná vein to varying degrees. This in-vein development was sampled, but no production stoping was undertaken. A number of other veins were also intersected, and in-vein development was carried out along these veins on a number of levels. Only the Strieborná vein has been targeted for further exploration and potential exploitation.

Level haulage development was typically excavated approximately 2.5 m x 2.5 m. Rail haulage with narrow gauge rail was used, with small locomotives and 0.85 m³ rigid-body rail cars. The rail cars were trammed by diesel locomotives to the Mária shaft from where they were hoisted individually to Level 6 and trammed 100 m to the incline shaft, and hoisted individually to surface and pushed 150 m to a rotary dump. The small capacity rail cars and the relatively inefficient materials handling arrangement constrained mine production to less than 500 t/d.

During the later years of the Mária Mine operation, as part of a plan to relocate the surface ore-handling facilities to a more remote location, a haulage tunnel was mined from the west slope of Kalvária-Tri Vrchy ridge, 1.5 km to the inclined shaft. The Meyer haulage tunnel was mined to approximately 4.0 m diameter by tunnel borer for the first 260 m, and then by drilling and blasting, with nominal dimensions of 3.0 m wide x 2.5 m high. Steel sets were installed at approximately 1 m spacing. The Meyer haulage tunnel was tracked, and a run of ore pad created immediately outside the portal on the west slope of Kalvária-Tri Vrchy ridge. Mining of the haulage tunnel was completed as mining operations were coming to an end, and the haulage tunnel was never actually used to support production.

On the lowest level of the mine, Level 13, a tunnel was extended some 500 m to the adjacent Sadlovsky Mine. Global Minerals reports that prior to the closure of operations at Mária Mine, a series of three watertight doors/bulkheads were installed in this drift. When both Mária and Sadlovsky mines closed, the workings were allowed to flood.

16.1.3 *Current Status*

At the time of the site inspection in February 2013, the incline shaft was in working order and equipped with a 110 kW hoist. Global Minerals had dewatered and was maintaining the water level in the Mária Mine at just below Level 8, using a 185 kW pump located at the top of the Mária shaft on Level 6. The internal infrastructure within the Mária shaft appeared to be in very poor condition and in a state of collapse. It was not possible to gain entry to the shaft. Global Minerals had rehabilitated those areas on Level 6 necessary to access the Strieborná vein, and had extended development on that level to facilitate additional exploration.

The Strieborná vein contains high copper and silver values in a single vein. While the vein pinches, swells, and varies in dip in places, the vein can be generally categorized as exhibiting regular geometry, with dips typically ranging from 65° to 75°. Stope widths vary from 1.7 m to 9.7 m, averaging 4.9 m. The potentially economic strike length of the vein is around 300 m. Ground conditions are expected to be Fair to Good (Bieniawski, 1976).

The proposed mining methods described in this study are variants of longhole open-stopping techniques, with permanent rib and sill pillars. The designs assume no backfilling will be necessary of the open stopes created. The Mineral Resource block model primarily contains Indicated and Inferred resources, but also includes a very small proportion (4,000 t, equivalent to 0.4% of total potential mill feed) of Inferred Mineral Resources. The Indicated resources extend to around -300 m elevation, a depth of approximately 650 m from surface. The production infrastructure described previously in Section 16.1.2 was considered very inefficient by Global Minerals, and AGP was requested to consider mechanized mining techniques with truck haulage. Global Minerals seeks to mine the Strieborná vein at an average rate of 300 t/d (approximately 110,000 t/a), a production rate that is included in its current permits. During the first two years of operation Global Minerals plans to seek permission to increase production to around 180,000 t/a for the remainder of the LOM.

Global Minerals reports that government permitting requires that the water level in the subvertical shaft adjacent to the Mária Mine must be maintained at a level in excess of one production level (50 m) below the deepest operating production level. The watertight seals located in the tunnel leading to Sadlovsky Mine (Section 16.1.2) cannot be inspected. The current water levels in Sadlovsky Mine are unknown, as is the volume of water it currently contains. Failure of these seals may result in rapid flooding of Mária Mine and the Strieborná vein workings to an unknown elevation. The government requirements for the maintenance of the water level in Mária/Strieborná are designed to provide sufficient spare sump capacity and time for the safe emergency evacuation of Strieborná Mine. Due to these issues, Global Minerals requested that AGP confine the study scope to identifying the potential for mining the Strieborná vein down to Level 11.

16.2 Geotechnical

16.2.1 Introduction

AGP, on behalf of Global Minerals, retained BGC to provide preliminary geotechnical evaluations to support the development of mining designs and cost estimates for the project. Specifically, geotechnical input was required for preliminary designs for underground excavations, mine pillars, and ground support.

16.2.2 Site Visit and Observations

BGC visited the project site to meet with Global Minerals personnel, review the project site, inspect the existing mine workings, inspect core logging procedures, and review core from the current exploration drilling program.

During the site visit the following tasks were undertaken:

- toured the project area; both surface and underground areas were visited
- collected and reviewed relevant project maps, level plans, and geological mapping
- reviewed rock and alteration types in the underground excavations
- inspected the project drilling and core logging procedures with respect to collection of suitable geological and geotechnical data
- carried out geotechnical review of select core and select underground exposures
- received core logging database and core photos
- met with AGP and Global Minerals to discuss the potential mine plan and project objectives.

BGC reviewed Global Mineral's geotechnical core logging procedures to confirm compliance with industry standards. In BGC's opinion, the core technicians and geologists are collecting and recording data with suitable accuracy and reliability.

At the time of the site visit, the underground workings had been dewatered to Level 8. The Meyer haulage tunnel, internal inclined shaft, and Level 6 were inspected with respect to ground support and rock mass conditions. The following summary of observed geotechnical conditions is based on those inspections:

- The rock mass remains stable within all drifts visited. Generally, only fault zones and large infrastructure excavations appeared to require ground support.
- The phyllites and psammities are strongly foliated, with foliation spacing varying, but consistently less than 10 cm.
- Foliation orientations vary significantly, but are predominantly steeply dipping.
- Small-scale folding is present in the rock mass.

- The drift profiles in the psammities are generally more rounded than in the phyllites. This may be caused by the increased fissile nature of the phyllites.
- The porphyries/pyroclastics are generally massive, but foliated.
- The rock mass is dry to damp.
- Portal steel sets are installed on 1 m spacing and show no signs of deterioration or loading.
- Although only Level 6 was visited, the site geologists report that the dominant lithology changes from metasediments to porphyries below approximately Level 10 to Level 13.
- No faults were observed during the underground inspection. Although faulting is common within and proximal to the vein, most of the areas visited during the underground inspection were development areas away from the vein.
- No indications of over-stressed excavations or pillars were observed during the underground inspection. The only ground control difficulties reported by the staff were localized caving ground around small faults. The loose ground was removed and ground support was installed without significant difficulty.

16.2.3 *Rock Mass Characterization*

Observations from the site visit were incorporated into the rock mass characterization component of the study. Other sources of data used for the rock mechanics assessment include:

- Database of core logging observations completed by Global Minerals geologists on exploration core. Approximately 3,700 m of core logging has been completed. This data includes run length, depth from/to, core recovery, Rock Quality Designation (RQD) (Deere and Deere, 1988), discontinuity count, mechanical breaks, average Joint Condition (Bieniawski, 1976), field strength index (ISRM, 1978), weathering grade, JRC, joint roughness, and joint planarity. Core orientation data is not currently being collected, although some structural orientation data is available in the historical structural mapping data provided by Global Minerals. The type of information available within the database, although limited, is considered sufficient for the current level of study.
- Observations of drill core and underground exposures collected during the site visit.
- Historical structural mapping provided by Global Minerals.
- Discussions with Global Minerals geologists.
- Relevant literature.

No geotechnical drilling was completed for this study. Neither laboratory testing nor point load testing has been completed on rock core samples. Recommendations for work at the next level of study are provided in Section 26.

The rock mass for the Strieborná Project was initially assessed according to the lithologies logged by the Global Minerals geologists, including the following units:

- phyllites
- psammities
- phyllites/psammities
- pyroclastics/porphyries
- quartz vein
- siderite quartz vein
- siderite vein
- stibnite quartz veins.

Variations in geotechnical parameters according to lithology were investigated and correlated to underground observations. Lithologies with similar geotechnical characteristics have been combined into the following three geotechnical domains:

- D1: phyllites, psammities, phyllites/psammities, pyroclastics (porphyroids). Blocky to foliated; foliation spacing less than 10 cm, joint spacing typically 0.3 m to 1.0 m; foliation generally steeply dipping; 2 + random to 3 + random joint sets; discontinuity surfaces are planar to undulating and rough. This unit was observed to be generally stable for unsupported spans up to 4 m.
- D2: veins. Includes siderite quartz veins, siderite veins, quartz veins, and stibnite quartz veins; generally massive with undulating and rough discontinuities. Average RQD and fracture intercepts indicated by the geotechnical logging suggest tighter fracture spacing than indicated by underground observations. Extensive shearing and faulting near the boundary of the deposit affects the vein at the abutments of the mine. Although a structural model has not been developed, the geologists reported noting a major structural zone, indicated by decreasing RQD and recovery, parallel to and adjacent to the NW side of the vein (footwall).
- D3: Fault Zones. Tend to be “disintegrated” as opposed to “decomposed” (i.e., brittle faults are more common than gougey faults); blocky assemblage; broken rock less than 7 cm in diameter; gouge is rarely present, but consists of silty matrix, and discontinuities are polished and slickensided. As reported by Global Minerals staff, the most common failure mode observed in fault zones encountered underground is caving, suggesting minimal larger blocks are present to support the small broken rock fragments.

Table 16-2 summarizes the “geotechnical unit” properties determined by BGC for the rock mechanics assessments.

Table 16-2: Geotechnical Domains for Strieborná PEA

Unit	Interval Count	% RQD	Joint Condition (Bieniawski, 1976)	Fracture intercept (m)	Field Strength Index Grade	UCS Estimate ⁽¹⁾ (MPa)	Q'	Q' Descriptor
D1	3,363	60	12	0.11	3.9	50 – 100	7.6	Fair
D2	292	77	25	0.14	4.1	50 – 100	29.5	Good
D3	581	0	12	0.05	3.0	25 – 50	1.3	Poor

Notes: ⁽¹⁾ UCS estimate based on Field Strength Index Grade estimated during core logging.

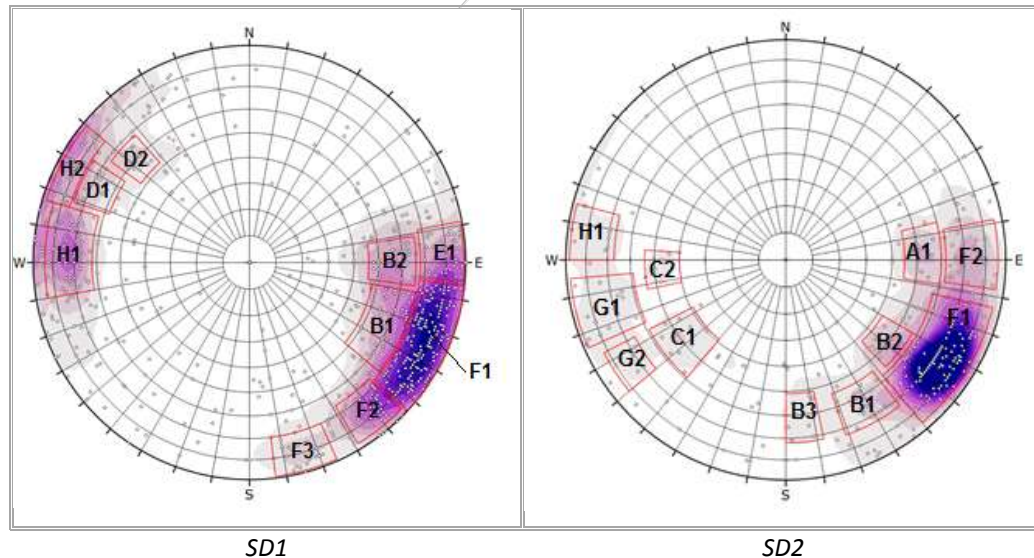
⁽²⁾ The ground above Level 6 is reported to have slightly poorer ground conditions than the levels below; this observation could not be confirmed. The underground observation that the relative competency of the rock mass increases from the phyllites to the psammities to the pyroclastics was not confirmed by the drill hole dataset, and therefore is not accounted for in the geotechnical designs.

⁽³⁾ 50th percentile values are shown for RQD, Joint Condition, Fracture Intercept, and Q'.

⁽⁴⁾ Average values are shown for Field Strength Index Grade.

Historical mine mapping data provided by Global Minerals was used to determine structural domains for the current study. As shown in Figure 16-2, there are generally two prominent discontinuity sets. Between Levels 9 and 10, one of the two prominent discontinuity sets is no longer prevalent. This may be a result of the offset from faulting reported by Jakubiak (2008), which was inferred to begin at Level 10 and continue to Level 13.

Figure 16-2: Structural Domain Stereonets from All Available Data



Because Level 10 appears to represent a structural domain boundary across which one of the two prominent discontinuity sets disappears, BGC has identified two structural domains: one includes mining depths from 360 m to 0 m elevation (SD1) and the other

includes mining depths from 0 m to -240 m elevation (SD2). Depths refer to distance below ground surface.

16.2.4 Mining Rock Mechanics Assessment

Stope Design

Preliminary stope designs were developed using the Modified Mathews/Potvin Stability Graph method (adapted from Hutchinson and Diederichs, 1996).

The Stability Graph was used to determine the maximum allowable unsupported and supported hydraulic radii for stope hanging walls, footwalls, and backs. Hydraulic radius (HR) is defined in Figure 16-3.

The results of the Stability Graph method are summarized in Table 16-3 and Figure 16-4.

It should be noted that the recommendations provided in Table 16-3 are best estimate design guidelines. If larger unsupported stope dimensions are desired, stope dimensions similar to those used in the Mária Mine could be used as an upper-bound design limit.

This recommendation is contingent on the rock quality at Strieborná being similar in quality to the Mária rock mass, which cannot be confirmed at this time. If the Strieborná rock mass is consistent with that indicated by the current dataset, but Mária Mine stope dimensions are used, there is potential for stope instability and increased dilution.

Figure 16-3: Definition of Hydraulic Radius

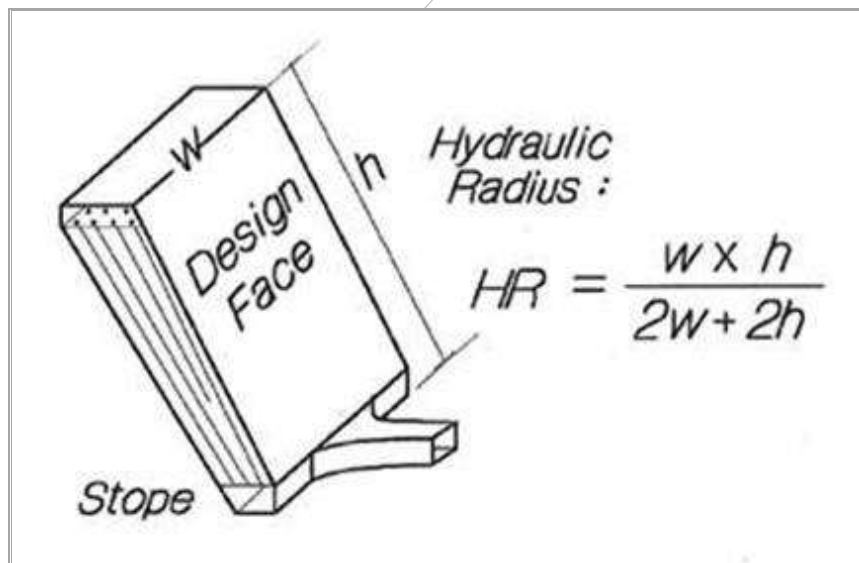
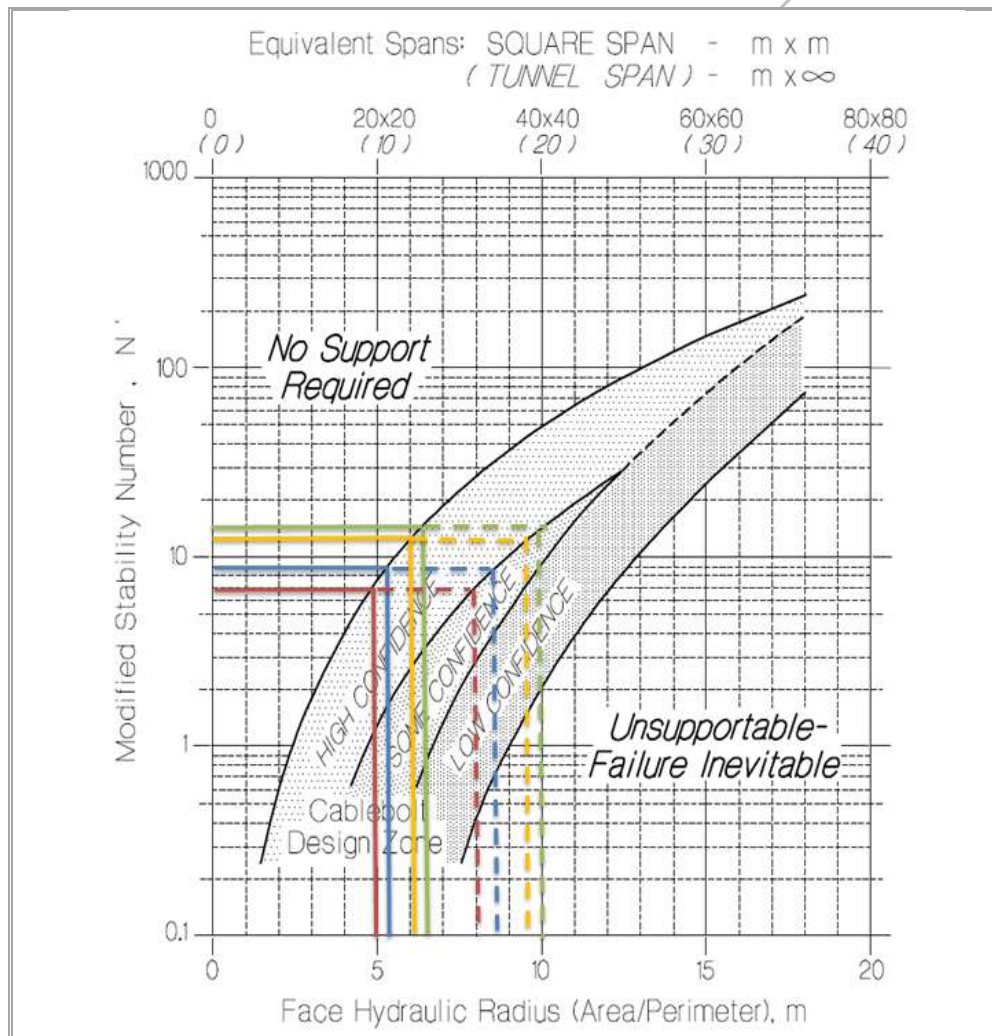


Table 16-3: Preliminary Stability Graph Results and Hydraulic Radius Recommendations for Both Structural Domains

Face	Q'	A Factor	B Factor	C Factor	N' Stability Number	A Factor	Max. HR (Unsupported)	Max. HR (Supported)
Hanging wall	7.6	1	0.2		6	9	5.4	8.5
Footwall	7.6	1	0.3		6	14	6.1	9.5
Vertical ends	7.6	1	0.5		4	15	6.5	10.0
Back	29.5	0.1	0.8		3	7	5.0	8.0

Figure 16-4: Recommended Maximum Unsupported and Supported Hydraulic Radii for Both Structural Domains



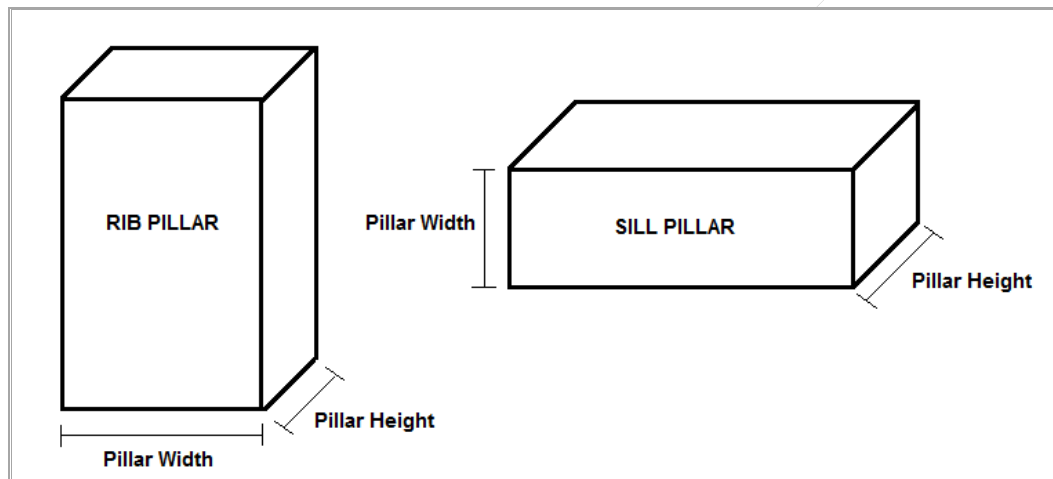
Note: Back (red), hanging wall (blue), footwall (yellow) and vertical ends (green). Solid lines represent unsupported hydraulic radii, and dashed lines represent supported hydraulic radii.

Pillars

Figure 16-5 shows the industry-standard geometrical definitions for the terms used in this report to describe pillar dimensions.

The PEA mine plan incorporates rib pillars in the mine layout to segregate stoping areas, and improve stability conditions and mining outcomes. Minimum rib pillar dimensions have been evaluated using the methods of Lunder (1994), and Brady and Brown (2004). Based on an estimated intact UCS of 75 MPa, and induced loads indicated by preliminary stress modelling, the recommended rib pillar width has been set as equivalent to the thickness of the vein (resulting in a width to height ratio of 1:1), with a minimum rib pillar width of 3 m.

Figure 16-5: Geometrical Definition for Pillar Dimensions



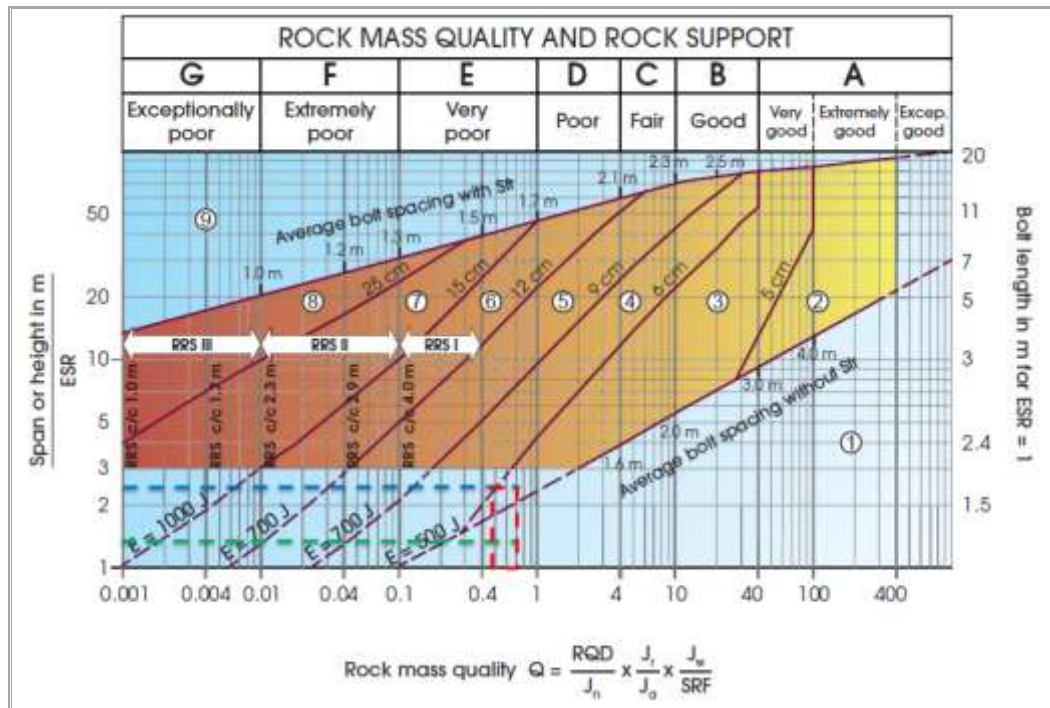
Note: Pillar height is equivalent to vein thickness for the longitudinal stoping method proposed for the current study.

Minimum recommended sill pillar dimensions have been derived based on the methods of Lunder (1994). An intact UCS of 75 MPa was assumed based on the field strength index test data within the core logging dataset, and induced loads were derived using preliminary stress modelling. The recommended sill pillar width has been set as equivalent to the thickness of the vein (resulting in a width to height ratio of 1:1), with a minimum sill pillar width of 5 m of intact rock separating adjacent stoping blocks.

Ground Support

Ground support requirements for standard development excavations have been estimated using the methods of Grimstad and Barton (1993) shown graphically in Figure 16-6. It is assumed that all openings will be excavated using drill and blast methods, and that all support will be installed at the face as mining advances.

Figure 16-6: Estimated Support Requirements – Grimstad and Barton (1993), updated by NGI (2013)



Note: Support class numbers are provided by NGI (2013). Blue dashed line represents permanent excavations, and green dashed line represents temporary excavations.

Table 16-4 presents a summary of resulting ground support recommendations for use in the preliminary mine designs.

Intersections greater than 6.0 m to 8.0 m in span will require 5 m cable bolts (or coupled grouted rebar) on a 2.5 m x 2.5 m square spacing in the back of the intersection.

These support recommendations do not apply to underground infrastructure excavations. Custom ground support guidance for these should be developed once excavation dimensions and orientations are determined.

Where possible, drifts should be planned perpendicular to the strike of the foliation, and driven with the dip so that the foliation planes can be supported prior to daylighting.

The ground support recommendations provided above are intended for guidance and cost estimating purposes only. The recommendations do not account for geological complexities that may be encountered during excavation or for stress conditions or changes in stress conditions that will result as mining progresses.

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Table 16-4: Ground Support Recommendations

Development Type	Dimensions		2.4 m Rebar		1.8 m Split Set		Mesh Proportion of Drift (%)	5 m Cablebolts or Equivalent			Notes
	Height (m)	Width (m)	Proportion of Drift (%)	Spacing (m)	Proportion of Drift (%)	Spacing (m)		Proportion of Drift	No. of Bolts/Ring	Ring Spacing	
Trucking Ramp	4.0	3.5	40	1.8 x 1.8	-	-	20	6	3	2	Estimate 40% of new permanent development requires patterned bolting, with half of that also requiring mesh. For permanent excavations with spans greater than 6 to 8 m, such as intersections, cablebolts or equivalent are recommended.
Ramp Access X/C	4.0	3.5	40	1.8 x 1.8	-	-	20				As Above
Stope Sublevel (In vein)	2.5	2.5	-	-	50	1.8 x 1.8	25	25	4	2	Estimate 50% of in-stope sublevel development (in-vein) requires patterned bolting and half of that also requiring mesh for safety during longhole drilling and repeated stope blasting. 25% of the drift will require cablebolting for stope hangingwall support.
Waste Sublevel (In vein)	2.5	2.5	-	-	25	1.8 x 1.8	12.5				Sublevels mined in waste to access stopes will not be subject to repeated close stope blasting concussion. Estimate 25% of stope access in waste sublevel development requires patterned bolting and half of that also requiring mesh.
Vent Manway Raise	2.8	2.8	-	-	100	1.2 x 1.2	75	-	-	-	Patterned support with mesh required for raise development with manway installed

The recommendations do not account for failure due to weathering or deterioration of the rock mass, failure caused by moving water (erosion, dissolution, etc.), or failure due to corrosion of ground support components. At the time of the BGC site visit, dewatering had occurred down to Level 8. Some rehabilitation of historical ground support was reported to have been completed due to degradation of the rock mass, particularly around fault zones. Re-entry of the remaining historical excavations should be preceded by a geotechnical inspection that includes an assessment of the pre-existing ground support. Rehabilitation of the ground support currently installed in the flooded historical excavations may be required.

16.3 Underground Mine Design

16.3.1 Mine Design Considerations

The relevant characteristics of the Strieborná vein from a mining method selection perspective are provided below:

- It is a hydrothermal-metasomatic, silver- and copper-bearing siderite-quartz vein, hosted in metamorphic rocks consisting of a single vein and a minor splay. Ground conditions are rated as fair to good.
- The mineralized vein is quite extensive, and is about 1,300 m in strike and about 500 m in vertical extent. However, the potentially economic portion of the vein considered in this study is about 600 m in length and 300 m in vertical extent, and is somewhat discontinuous at the mine planning cut-off grade.
- It has a variable dip from 50° to subvertical, although the portion of the vein under consideration varies from about 60° to subvertical, with the majority of the vein being subvertical. Additionally, the dip direction of the vein reverses itself from one end of the vein to the other, thus the orebody exhibits a slight twist somewhat similar to that of a propeller.
- The vein has an average true thickness of 4.9 m, ranging from 0.2 m to 12 m.

16.3.2 Mining Method Selection

Following a qualitative analysis and selective economic analyses, the following were deemed unsuitable for consideration as principal mining methods:

- Captive longhole stoping with rail haulage: high operating cost; high rehabilitation costs; complex, inefficient haulage system with hoisting of small rail cars.
- Cut-and-fill stoping, either captive non-mechanized with rail haulage or mechanized with ramp access: higher labour force, lower productive rate, higher cost, shortage of fill material.

- Shrinkage mining stoping, either captive non-mechanized with rail haulage, or mechanized with and ramp access: higher labour force, lower productive rate, higher cost.
- Alimak stoping: too technically challenging, requiring specialized equipment and workforce, and relatively extensive economic vein extents.

Economic analyses completed by AGP indicated that captive mining methods, employing pneumatic jacklegs for level and raise development, rail haulage to the Mária Mine, and hoisting to surface via the existing (albeit rehabilitated) Mária Mine materials handling infrastructure, were undesirable. The relative location of the potentially economic portion of the Strieborná vein, and the regular, relatively steeply dipping geometry of the vein, indicates the most appropriate mining and materials handling method would be a longhole open-stoping method employing mechanized rubber-tired mining equipment and trucking of material direct to. The longhole open-stoping method selected is up-hole retreat without backfill.

16.3.3 Longhole Stoping

The narrow nature of the vein dictated a small hole size for production blasting, with a 64 mm diameter hole size selected. The small hole size selected and relatively narrow vein thickness limited the sublevel spacing to 17 m floor-to-floor, to limit the impact of hole deviation on external dilution. Additionally, the sublevel spacing was also appropriate to the shape of the potentially economic mineralization, allowing for less internal dilution compared to larger sublevel spacing. The geotechnical hydraulic radius recommendations (Table 16-3) provided guidance on the stope size selected: a nominal 30 m strike length and three individual sublevel stopes in height, for a total open stope height of approximately 47 m, taking the required sill pillar into consideration. In areas where the geometry of the targeted mineralization allows for the creation of stopes only two sublevels high a maximum stope strike length of 45 m is applied, while stopes only one sublevel high are limited to no more than 60 m along strike. Each open stope, be it comprising one, two, or three individual sublevel stopes is referred to as a “stope unit” and is subsequently scheduled as a single entity.

A longitudinal retreat stope mining sequence, along the strike from vein extremities to ramp access points, is planned. The mine layout incorporates a single ramp system through the body to access all sublevels. This centrally located ramp bisects the 600 m strike length, in order to minimize mucking distances and provide additional independent mining fronts.

Ore development on vein will be limited to a width of 2.5 m. Blast holes will consist of parallel up holes 14 m to 16 m in length. Some drill holes will be fanned out where vein widths exceed 2.5 m. Drill factors range from 6 t/m at the minimum stope width of 1.5 m to 13 t/m in stopes over 5 m wide, including drilling of the slot and slot raise and allowances for re-drilling.

Blast holes will be loaded with either ANFO (assumed 90%) or emulsion (assumed 10%), depending on local water conditions. Powder factors vary from 0.46 kg/t in narrow stopes less than 3 m wide to 0.23 kg/t in stopes greater than 5 m wide.

Standard stope dimensions for a typical three sublevel high stope are 30 m on strike, 47 m high, and vein width. At an average vein width of 4.9 m, a standard stope will yield approximately 24,400 tonnes, including the development tonnes.

Production mucking will be undertaken by 6.7-tonne capacity LHDs for 83% of the planned stope tonnage. Smaller 3.5-tonne capacity LHDs will be used for stopes in the narrow sections of the vein. Ore mucked from the stope drift will be trammed to remuck bays located on each sublevel close to the main ramp. Production haulage to surface will be undertaken by 15-tonne capacity diesel haul trucks from each active sublevel.

The use of rib and sill pillars to separate each longhole open stope from its neighbour will eliminate the requirement for backfill to provide stope support. However, it will be possible and practical to target a number of easily accessible minedout (thus open) stopes during the LOM in order to dispose of some ongoing development waste underground, rather than hauling this material to surface.

The dimensions of the rib and sill pillars and the somewhat discontinuous nature of the targeted mineralization are intended to create sufficient pillar area to provide local and regional stope ground support for LOM. All pillars are deemed to be unrecoverable, and are not included in the mill feed schedule.

The formation of “standard” stope units comprising three individual sublevel stopes effectively creates a series of main levels throughout the Strieborná mine spaced every 50 m. All stopes within a given 50 m level interval are referred to as a “stopping block.” These main levels also coincide with the existing in-vein development completed as part of the original Strieborná exploration/exploitation program undertaken when the Mária Mine was in operation. As such, AGP planned the main levels at the following elevations:

- 180 m elevation (coincides with existing Level 6 development)
- 130 m elevation (where Level 7 would historically have been located)
- 80 m elevation (coincides with existing Level 8 development)
- 30 m elevation (coincides with existing Level 9 development)
- -20 m elevation (coincides with existing Level 10 development)
- -70 m elevation (where Level 13 would historically have been located).

Stopping blocks and individual stope units within a stopping block are identified by the particular main level that runs along the bottom of the stopping block, and are identified in this report by level name rather than by elevation.

Within each stopping block, mining will progress along strike in a retreating sequence from the vein extremities to the centrally-located ramp access point, moving from the

uppermost sublevel stope of a particular stope unit to the lowermost sublevel stope of the same stope unit (i.e., the sublevel stope located along the main level associated with each particular stope unit).

Per Section 16.2.4 the rib pillar thickness between horizontally adjacent stope units along the same main level has been set as a function of the width of the vein (open stope) at the required rib pillar location. The rib pillar will be created by establishing a new stope slot for the adjacent stope the required distance from the previous stope. Sill pillar dimensions between vertically adjacent stoping blocks are also a function of open stope width, and will be created by holding short the up holes drilled from the uppermost sublevel of each stope unit, located below the mined-out stoping block above. As such, blast holes for the affected uppermost sublevel stopes will consist of up holes only 8 m to 13 m in length.

Stoping will start from the upper areas of the deposit and progress downwards, stoping block by stoping block (i.e., main level by main level). The mining method for the deposit as a whole as well as within each stoping block, can therefore be described as a top-down mining method. As such, there will be opportunity to test the local ground conditions and variations of stope layout. In the event that ground conditions are better or worse than currently anticipated, stope unit strike lengths and heights, as well as rib and sill pillar dimensions, may be increased or decreased without affecting the location of the main levels.

The resource model indicated a number of very high-grade areas. Wherever possible AGP sought to minimize high grade lost in pillars, as well as scheduling these areas early in the mine life; however, the continuous retreat style of mining and development scheduling constraints provided little flexibility.

16.3.4 *Economic Cut-off Grade*

A preliminary cut-off grade analysis was undertaken at the commencement of the study using broadly estimated costs and factors to determine the appropriate cut-off grade to apply at an assumed mine rate of 110,000 t/a. The initial stope design was based on a cut-off grade defined at a silver price of \$20.00/oz, a copper price of \$3.00/oz, a process recovery of 96%, and a site operating cost of \$99.50/t. Table 16-5 shows a summary of the breakeven analysis.

As a result of this analysis, a mine planning resource cut-off value per tonne equivalent to the total operating cost (\$100/t) was used to identify the potentially mineable portion of the mineral resource. This was accomplished by calculating the net value of each resource block and eliminating resource blocks below the mine planning cut-off grade. The resultant resource blocks were wireframed together to form a “value shell” to assist in the stope definition process.

As can be seen in Table 16-5, a resource cut-off value of \$100/t is equivalent to an in situ resource value of approximately \$190/t.

Table 16-5: Breakeven Cut-off Grade Analysis

	Factor or Value	Unit	Value/Tonne
<i>In situ Cut-off Grade</i>			
Silver Grade		g/t	210
Copper Grade		%	0.86
Antimony Grade		%	0.54
In situ Contained Metal Value		US\$	191.46
Silver Equivalent In situ Cut-Off Grade		Ag Eq g/t	298
Mining Dilution (Zero Grade)		%	13.0
<i>Process Plant Feed</i>			
Silver Grade		g/t Ag	182
Copper Grade		%Cu	0.74
Antimony Grade		% Sb	0.47
Contained Metal Value		US\$	166.49
Silver Equivalent Grade		Ag Eq g/t	259
Metal Prices	Payable		
Copper	96.2%	US\$/lb	3.00
Silver	95%	US\$/oz	20.00
<i>Flotation Concentrate</i>			
Copper Flotation Recovery		%	96.0
Copper Concentrate Grade		% Cu	28.0
Silver Flotation Recovery		%	96.0
Antimony Flotation Recovery		%	95.0
Copper Concentrate Gross Revenue		US\$	152.41
<i>Copper Concentrate Charges</i>			
Concentrate Land Freight	73.80	US\$/t Wet Concentrate	2.03
Port Charges	5.00	US\$/t Wet Concentrate	0.14
Concentrate Sea Freight	40.00	US\$/t Wet Concentrate	1.10
Mercury Roasting	520.00	US\$/dmt Concentrate	13.27
Smelter Treatment Charge	150.00	US\$/dmt Concentrate	3.83
Price Participation		US\$	3.79
Copper Refining	0.15	US\$/lb Cu	2.27
Silver Refining	0.60	US\$/oz Ag	3.21
Arsenic Penalty		US\$	1.12
Antimony Penalty		US\$	22.15
Copper Concentrate Charges		US\$	52.91
Net Revenue		US\$/t Processed	99.50
<i>Estimated Operating Costs</i>			
Mining Costs	60.00	US\$/t Processed	60.00
Concentrator	19.50	US\$/t Processed	19.50
G&A	20.00	US\$/t Processed	20.00
Total Operating Cost		US\$/t Processed	99.50

16.3.5 *Application of Modifying Factors*

Strieborná stope design generally followed the following steps:

- a mine planning resource cut-off value per tonne equivalent to the total operating cost (\$100/t)
- geotechnical assessment to determine stable stope and pillar dimensions (refer to Section 16.2.4)
- selection of mining sublevel spacing and elevations
- designing the practical mining shapes (separate development and stope wireframes) using the selected cut-off value as a guide
- applying a minimum stope width of 1.5 m in some narrow areas
- removing portions of the mining shapes to account for rib and sill pillar locations
- reporting of in situ quantities inside the mining shapes using block modelling software
- checking internal dilution and optimizing mining shapes
- application of estimates for external dilution, dilution grades, and mining losses, in order to determine the expected mill feed tonnes and grades
- removal of any mining shapes that do not generate a positive value after the application of dilution and recovery factors; and
- categorizing development and stope mill feed tonnes and grades for scheduling.

In order to understand the full extent of the mining areas, longhole stopes were designed as zones of contiguous stoping areas on each sublevel (i.e., without recognizing the requirement for rib or sill pillars). The resultant stope shapes within a stoping block were reviewed with respect to the grade, vein width, and the number of sublevel stopes (from one to three) making up a particular stoping block in order to best determine the location and dimensions of the required rib and sill pillars while minimizing tonnage and grade losses to the extent possible. This resulted in the generation of separate stoping units within the stoping blocks, which honoured the geotechnical hydraulic recommendations based on the number of sublevel stopes in each individual stope unit, as well as honouring the recommended pillar dimensions and locations. There are some areas where the standard stope lengths needed to be reduced in order to better fit in the geometry of the value shell, generally around the perimeter of the deposit. Stope unit widths before dilution varied from 1.7 to 9.7 m with an average of 4.9 m.

The mining shapes followed the hanging wall and footwall of the mineralized vein structure as well as the sublevel height would allow. Very few potentially economic mining areas did not contain mineralization above the selected cut-off grade across the entire vein width.

Mining shapes exhibiting high internal dilution were checked to determine if optimization of the shape was possible.

The modifying factors to estimate mill feed tonnage from the in situ stope resources are summarized in Table 16-6.

Table 16-6: Modifying Factors

Mill Feed Components	Dilution Criteria	Avg. Overall External Dilution ⁽¹⁾	Tonnage Recovery
Development	- Drift dimensions 2.5 m high x 2.5 m wide with 0.1 m overbreak - Waste applied at zero grade	6.0%	100%
Narrow Vein Stopping	- Internal waste due to vein irregularity included in stope in situ resource - Assumed stope overbreak by 0.3 m on both hangingwall and footwall (Total 0.6 m with zero grade)	14.5% (Avg. vein width 4.9 m)	98%

Note: ⁽¹⁾ Dilution defined as waste tonnes/mill feed tonnes.

16.3.6 Mineral Resources for the Mine Plan

At the selected mine planning cut-off value of \$100/t, the potentially economic resource was considerably smaller and more discontinuous than the vein extents as shown in Figure 16-7. Figure 16-8 shows a long section vertical projection looking southeast of the planned stopes and pillars, in-vein development, and the current extent of workings.

Table 16-7 shows the comparison of in situ Mineral Resources contained within mining shapes to mill feed, and the distribution of mill feed tonnes and grade by stopping block (level). Initial production is scheduled from Level 6, followed by Level 8. The lower grade stopes on Level 7 are not mined until the completion of mining on Level 8. The remaining levels are mined sequentially to the bottom of the mine.

Table 16-7: Comparison of PEA Mine Plan Stope In Situ Resources and Estimated Mill Feed

Mining Level	In Situ ⁽¹⁾				Estimated Mill feed				Avg. Thickness (m)	External Dilution (%)
	(t)	Value	Ag g/t	% Cu	(t)	Value	Ag g/t	% Cu		
6	52,671	208	434	1.87	59,602	180	376	1.62	3.9	15.5
7	69,221	113	240	1.02	77,147	99	211	0.90	4.4	13.7
8	99,309	156	338	1.65	109,819	138	299	1.46	5.0	13.0
9	173,145	184	395	1.74	190,190	164	353	1.56	6.3	12.6
10	236,236	156	330	1.40	265,481	136	287	1.22	4.6	15.0
11	145,224	177	373	1.59	166,054	151	320	1.36	3.9	16.9
Development	108,130	144	309	1.36	114,668	136	291	1.28		6.0
Total	883,936	163	347	1.52	982,962	144	307	1.34	4.9	13.5

Note: ⁽¹⁾ Includes internal dilution within planned stope outlines.

Mill feed totals 983 kt grading 307 g/t Ag and 1.34% Cu over the LOM. Of that tonnage 979 kt is classified as Measured or Indicated, and the remaining 4 kt is classified as Inferred.

Figure 16-7: Long Section of Strieborná Vein, Highlighting Material above Cut-off Value

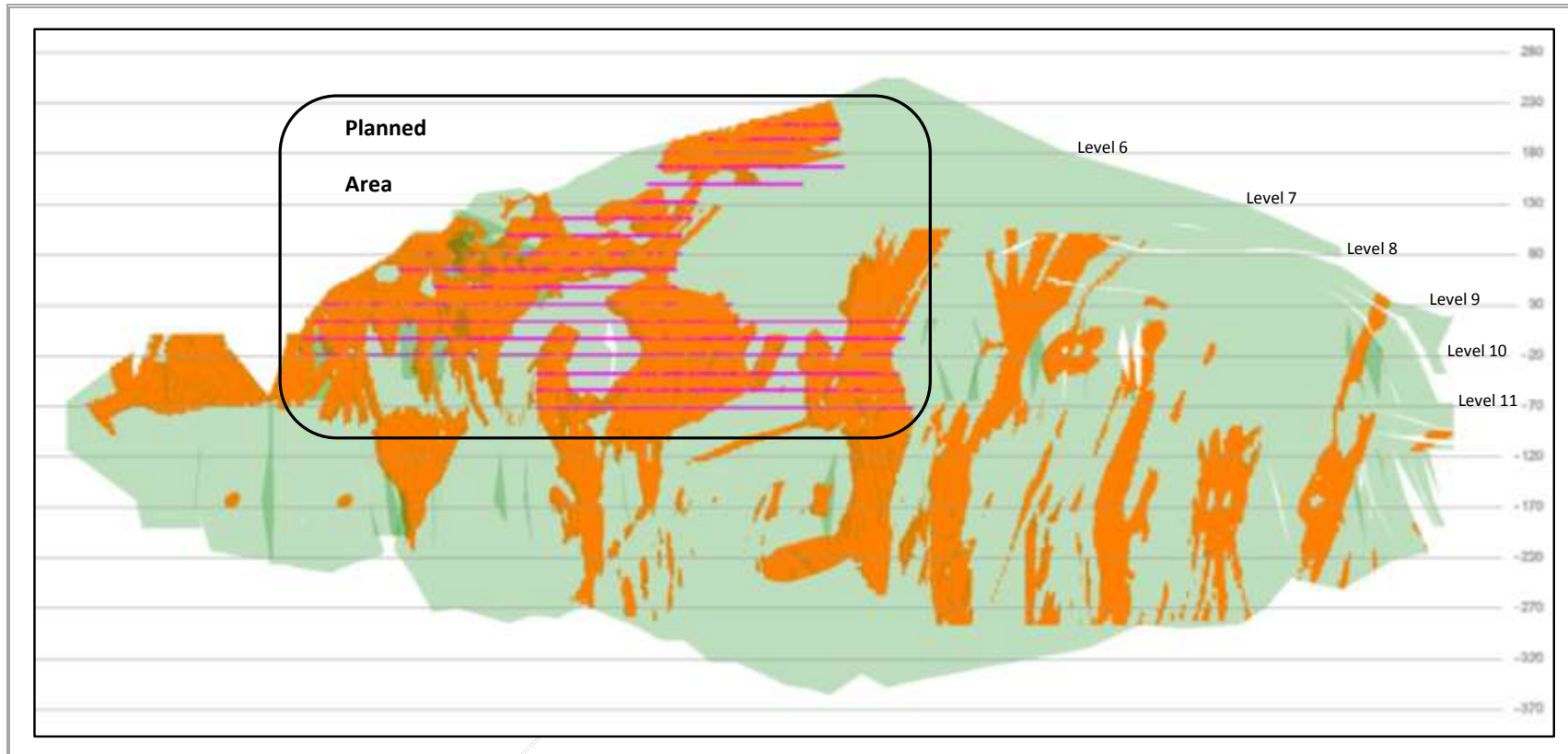
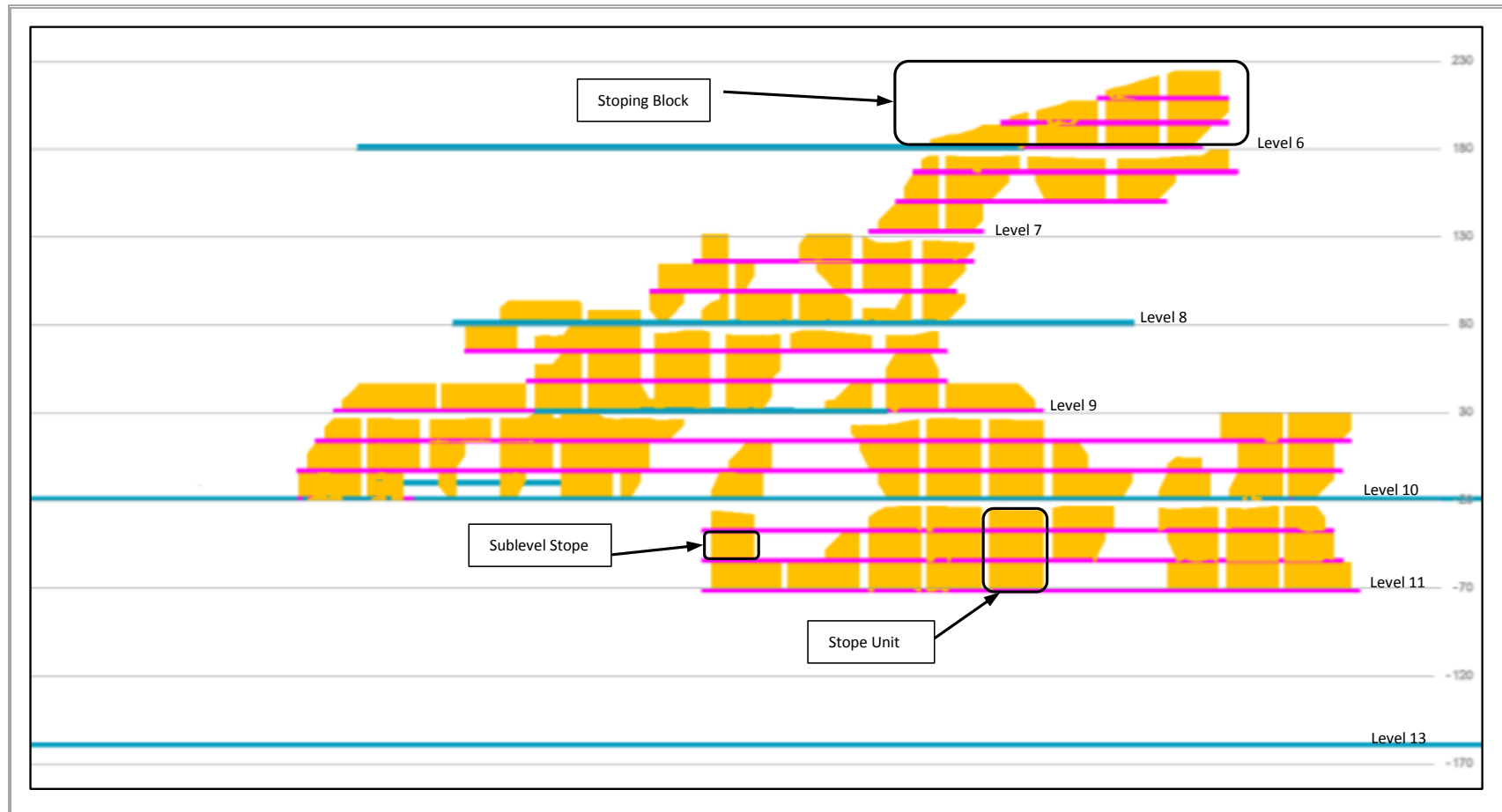


Figure 16-8: Long Section of Planned Stopes



16.3.7 *Underground Mine Layout*

Access to the underground mine is planned by a -15.0% gradient ramp, the beginning of which will be located along the existing Meyer haulage tunnel, approximately 1,250 m from the haulage tunnel portal. The ramp location along the haulage tunnel (termed the Haulage Tunnel Connection Ramp (HTCR)) is such that the ramp will target the top sublevel of the Strieborná mine on what could be described as the hanging wall side of the Strieborná vein (i.e., on the Mária Mine side). From here, a single ramp (termed the Strieborná Ramp) will spiral down along the centre of the Strieborná vein, in order to provide access to each of the sublevels such that the ramp will bisect the strike extent of the various sublevels in two roughly equal segments. The Strieborná Ramp will also provide access to existing access drifts between the historical Mária Mine and the Strieborná vein on Levels 6, 8, 9, and 10. This will create a number of direct connections between the Strieborná Ramp and the Mária Mine, providing access for small mobile equipment to the Mária Mine on each main level, and establishing the required ventilation connections between the two mines. Slashing of the existing Mária Mine drifts is planned only for those drifts adjacent to the Strieborná vein that are required to provide access for all mobile equipment to the Strieborná infrastructure, such as underground workshop facilities, as well as vein access and ventilation access requirements.

A short access drift from the Strieborná Ramp to each sublevel will provide access to the Strieborná vein. Typically, each ramp spiral will cover 34 m vertically, such that short and longer access drifts from ramp to vein will alternate from level to level. All ramp and vein access drift dimensions are planned to be 3.5 m wide x 4.0 m high.

During the first year, the existing incline shaft will be used for access of workers and materials. After the first year of mine production, a 220 m long ramp connecting the haulage tunnel to the historical Drazus site is planned, in order to provide better vehicle access between the Drazus site and the Strieborná workings.

The HTCR will traverse through the plane of the existing Mária Mine in order to establish the shortest haulage route between the Strieborná Mine and surface. The ramp pierce point through the historical Mária Mine has been selected in order to pass through an unmined portion of the Mária Mine. A historical long section projection of the Mária Mine showing the location of the pierce point of the Meyer haulage tunnel is shown in Figure 16-9. Figure 16-10 shows an isometric view of the planned access to the Strieborná vein, while Figure 16-11 shows a long section of the design.

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Figure 16-9: Pierce Point of the Haulage Tunnel Connection Ramp

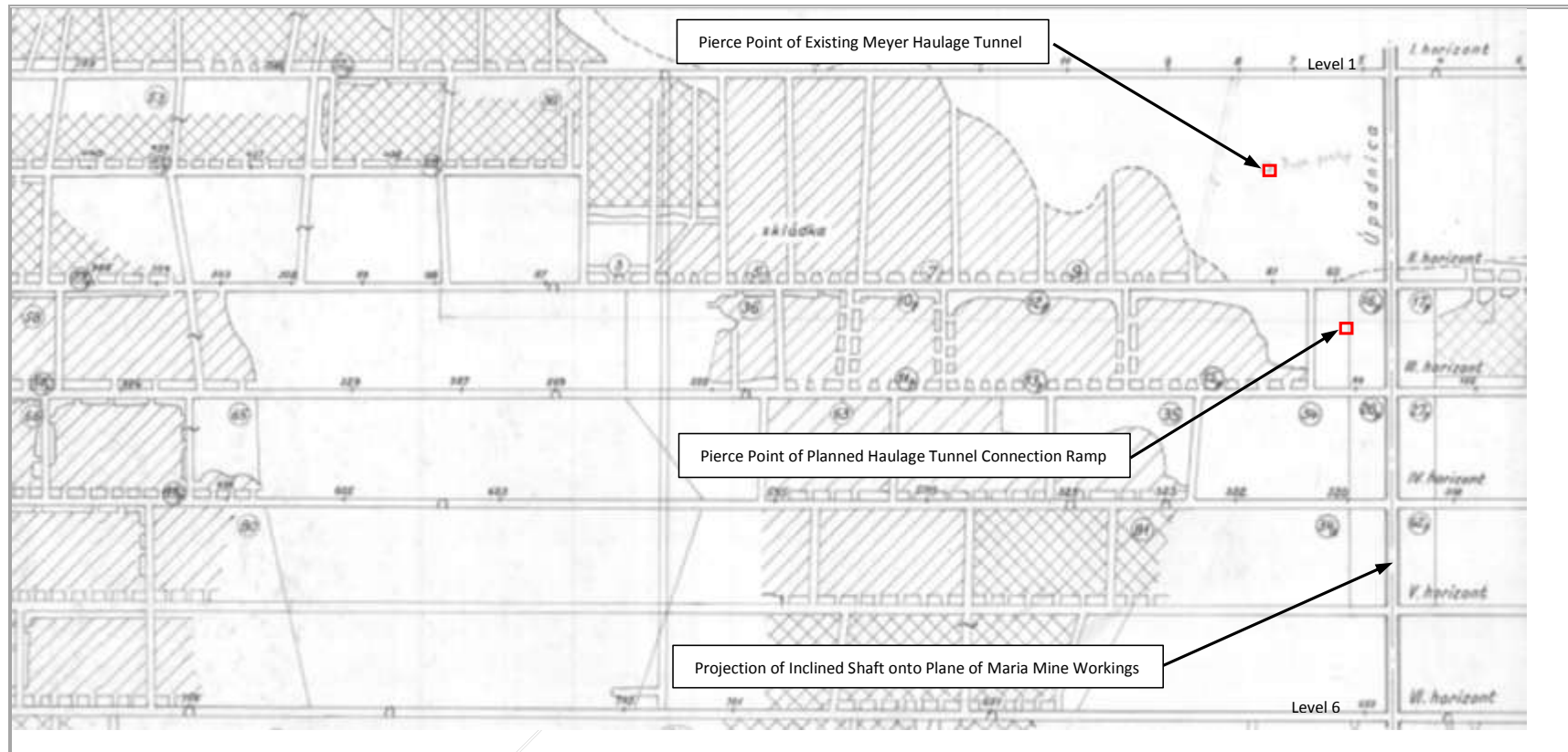


Figure 16-10: Isometric View of Planned Access to the Strieborná Vein

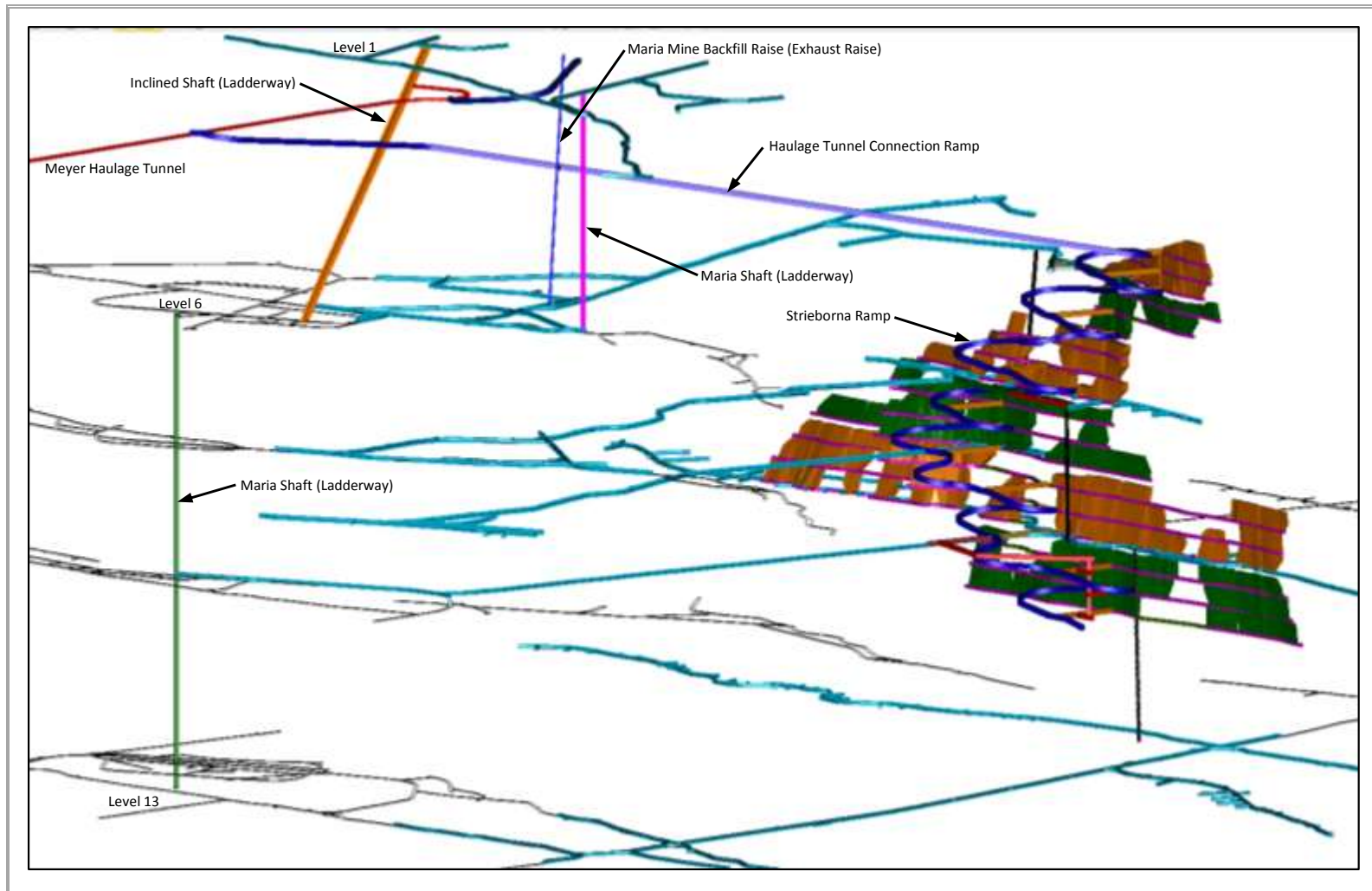
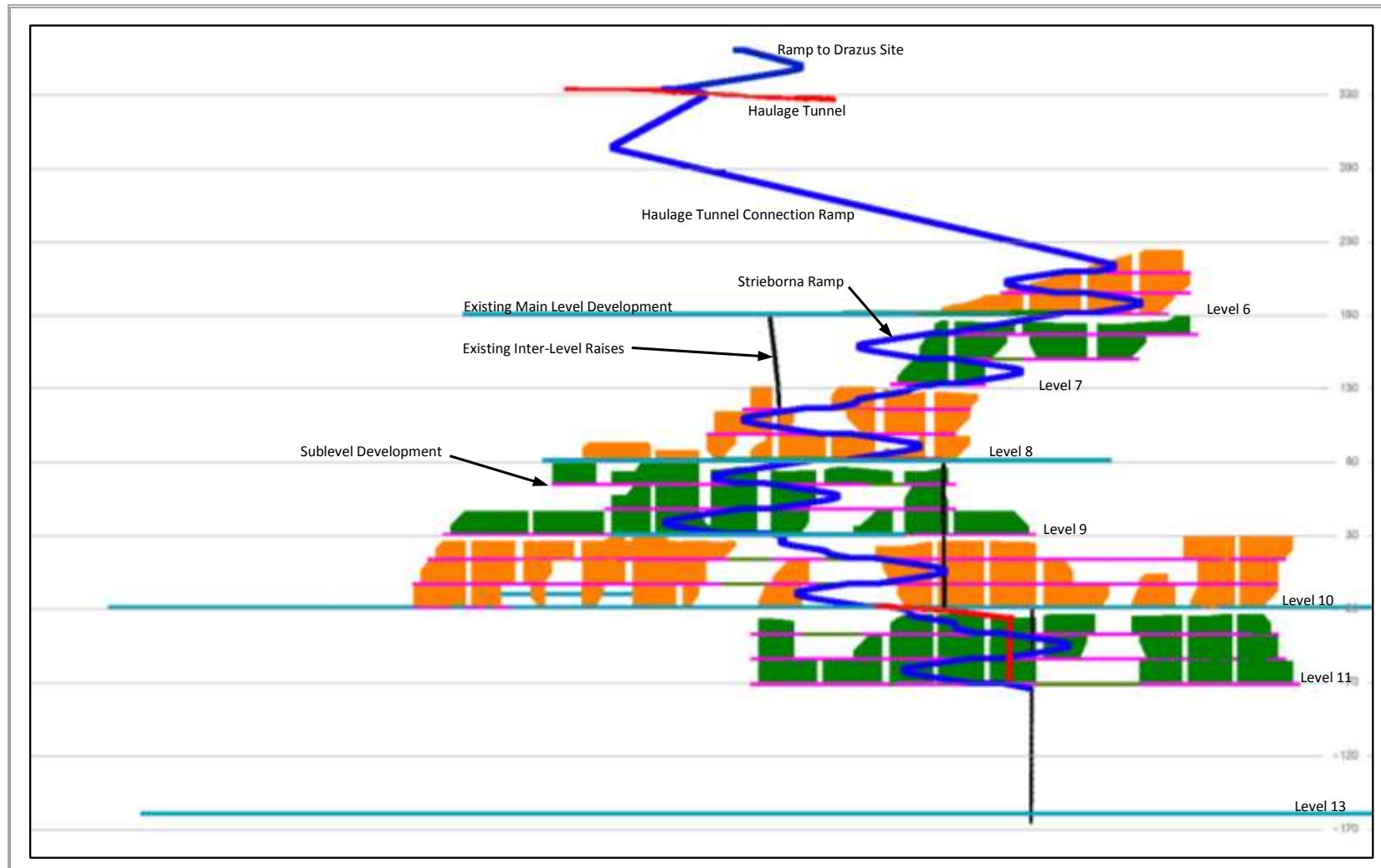


Figure 16-11: Long Section of Strieborná Vein Access



In-vein development along each sublevel is planned to be 2.5 m wide x 2.5 m high. Vein development material has been identified as waste and removed from the mill feed quantities and mine schedule. It has been found that in most cases, a portion of the existing historical in-vein development either does not coincide with the required location necessary for longhole stoping, or does not extend the required distance along strike. As such, additional in-vein development will be required along the historical in-vein sublevels (Levels 6, 8, 9, and 10). Figure 16-12 shows the development requirements on Level 9. Figure 16-13 shows Level 10 where there is more existing development on strike. Some of this development can be utilized for stope extraction, but off-vein development will not be used and parts of this will need some re-support to ensure stope stability.

The Strieborná Ramp system will be developed over the LOM such that there will always be an alternate means of egress out of the Strieborná mine during production, either up through the Strieborná ramp system, and out the Meyer haulage tunnel to surface, or down through the Strieborná ramp system to the nearest Mária Mine connection drift and up through a manway installed in the Mária shaft.

A small mobile equipment workshop will be provided on Level 6, and will be created by enlarging existing drifts with a small amount of additional level development. Major maintenance will be undertaken at a surface workshop that will be located in an existing mine building at the Drazus mine site.

Figure 16-12: Level 9 Development

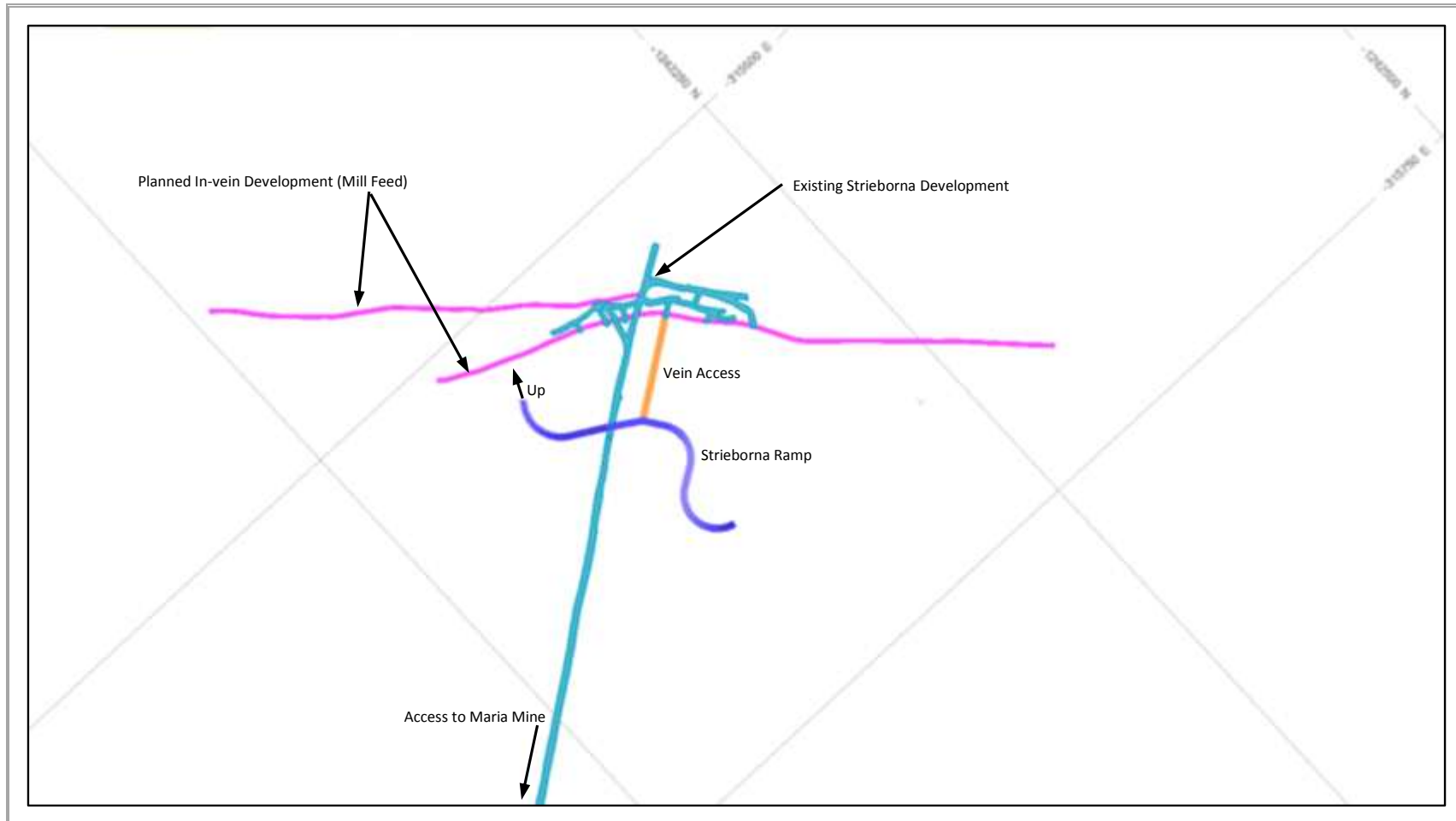
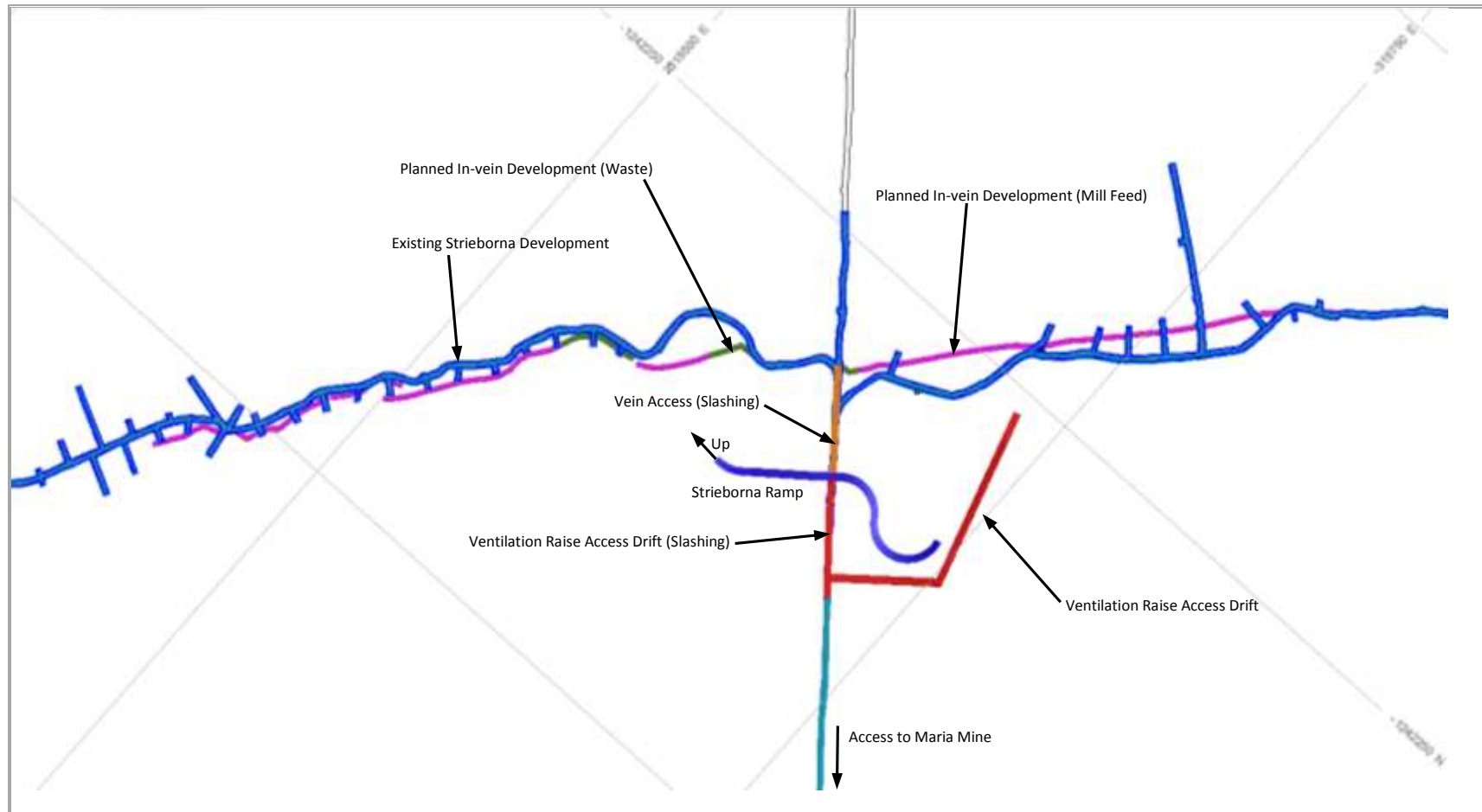


Figure 16-13: Level 10 Development



16.4 Mine Scheduling

16.4.1 Rehabilitation and Dewatering Schedule

In order to undertake exploration work for the Strieborná vein, Global Minerals dewatered the mine to below Level 8 and rehabilitated the incline shaft, haulage transfer drift and Level 6 accesses to Strieborná. Global Minerals therefore has experience of rehabilitation in the mine. As development and production from Strieborná vein progresses, additional rehabilitation of existing old workings will be required.

In order to continue to dewater the Mária and Strieborná workings, the subvertical (Mária) shaft will be cleared of debris, made safe, and an emergency ladderway will be installed. A small hoist will be used to lower pumps down the shaft, and rising water main pipes installed. There will be no hoisting of workers or materials.

Old tracks will be removed from the haulage tunnel. Some minor rehabilitation may be necessary in the haulage tunnel if the existing steel arches create tight spots for the planned 15-tonne trucks. Some arches may be replaced by bolting, or the floor level may be lowered.

Levels 8, 9, and 10 will be rehabilitated to provide access to the subvertical shaft emergency manway and for ventilation purposes.

The rehabilitation work described above does not need to be completed for operations to begin. Because mining progresses top-down, the rehabilitation work can be scheduled on an as-needed basis, in line with the requirements of the development and production schedule. The rehabilitation schedule is shown in Figure 16-14. It should be noted that Year 1 in the Mining Schedule is equivalent to Year -1 in the cash flow analysis, as it represents preproduction work.

Figure 16-14: Rehabilitation Schedule

	Year -1				Year 1				Year 2			
	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Dewater 8L - 9L (Pump hoist on 7L)												
Dewater 9L - 10L (Pump hoist on 8L)												
Dewater 10L - 11L (Pump hoist on 9L)												
Clear & Limited Re-equip Sub Vertical Shaft 6L - 7L												
Clear & Limited Re-equip Sub Vertical Shaft 7L - 8L												
Clear & Limited Re-equip Sub Vertical Shaft 8L - 9L												
Clear & Limited Re-equip Sub Vertical Shaft 9L - 10L												
Rehabilitation of existing 8L												
Rehabilitation of existing 9L												
Rehabilitation of existing 10L												

16.4.2 Development

A series of development factors were applied to the mine design in order to account for the additional development required for remuck and electrical equipment bays, temporary dewatering sumps, etc. The factored mine design lengths were subsequently imported into the mine scheduling spreadsheet for scheduling. The development factors used to convert mine design centreline lengths from MineSight to scheduled lengths in Excel are shown in Table 16-8.

Table 16-8: Development Factors for Mine Scheduling

Mine Design Item	Development Factor
Main Ramp	1.15
Level Access Drift	1.00
Ventilation Drift	1.05
Miscellaneous Drift	1.05
Raises	1.00

Owner employed crews will develop the mine. The monthly jumbo development rates used for scheduling are shown in Table 16-9. Separate development rates were derived for the large waste development headings and the smaller in-vein headings, assuming different sizes of equipment. In addition, development rates were determined on single heading, double heading, and multiple (three or more) heading bases. A contractor will excavate the small quantity of internal ventilation/egress raises.

Table 16-9: Monthly Scheduled Mine Development Rates

Heading Type	Single Heading (m)	Double Heading (m)	Multiple Heading (m)
Waste Heading	130	180	220
In-vein Heading	103	142	174

As scheduling was undertaken on a monthly basis, it was possible to visualize the number of available headings that could be scheduled by each development crew for any particular monthly period. This permitted the application of the most appropriate development rate (either single, double, or multiple) for each development crew on a monthly basis over the entire LOM.

A total of two development crews and associated mobile equipment were required in order to maintain the production schedule. The second development crew was added to the schedule as headings became available, or as the stoping schedule demanded, and subsequently removed from the schedule as development reduced in the later years of the LOM. An attempt was made to fully utilize each development crew in each monthly period by scheduling mine development at the maximum development rate based on the

number of available headings. However, this was not always possible and further optimization is possible.

The scheduling of preproduction mine development focussed on the following three priorities:

- 1) Development of the HTRC and the upper portion of the Strieborná ramp down to Level 6 and establishing a connection between the Strieborná ramp and the Mária Mine.
- 2) Early establishment of a primary airflow between the upper portion of the HTRC and the return air raise by completing the early installation and commissioning of the surface exhaust fan.
- 3) Introducing the second development crew as soon as practical after access to Level 6 is achieved, in order to develop the Strieborná ramp down to Level 8 and establish stoping operations on Level 8 prior to the completion of stoping on Level 6. This will allow for the deferral of the lower grade stopes on Level 7 until the increase in mine production in early Year 3.

All in-vein development was categorized as either mill feed or waste. The in-vein development on each sublevel was attributed to each associated stoping unit, and diluted and recovered mill feed tonnes and grade separately estimated for each segment, and then scheduled as required to suit production schedule requirements. The longitudinal retreat longhole mining method requires that longhole stoping commence at the extremities of the orebody and retreat longitudinally towards the centrally-located level access. As such, all in-vein development in a stoping block must be completed prior to the commencement of the associated longhole stopes.

16.4.3 Mill Feed Scheduling

The mill feed schedule comprises both in-vein stope development and longhole production stoping, with separate mill feed schedules generated for each. The scheduling of stope development mill feed is a function of the appropriate mine development rate, while scheduling of longhole production stoping is a function of the overall longhole stope production rate (drilling, explosives loading, blasting, mucking and hauling). Fifty separate stoping units were scheduled across the six underground levels as the development schedule allowed. Production commenced on Level 6 progressing sequentially downward through the mine, with the exception of Level 7, which was postponed to later in the mine schedule due to the level's lower average grade. An average stope production rate (per standard stoping unit) of 150 to 170 tonnes per operating day was targeted in the mine schedule. The overall average stope production rate per stoping unit achieved in the schedule was in the order of 162 tonnes per operating day.

16.4.4 Development and Production Schedule

The derivation of the production schedule comprising individual longhole stoping units and associated development allowed for a rather detailed stope-by-stope, level-by-level, and month-by-month schedule to be generated. The mine physicals captured in the development and production schedule included:

- lateral waste development (metres of development and waste tonnes)
- raise development (metres of raise and waste tonnage)
- stope development mill feed (tonnes and grade)
- longhole stope mill feed (tonnes and grade)
- total mill feed (tonnes, grade).

Table 16-10 shows an overall summary of the above-listed mine physicals by mine development year. All other mine physicals required for costing purposes were derived from these main items as part of the subsequent mine costing work.

Table 16-10: Development and Production Schedule

	Unit	Total	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7
Development										
Ramp	m	3,787	1,385	1,261	947	194	-	-	-	-
Ramp to Vein Access	m	1,136	150	623	200	163	-	-	-	-
Other Waste Drive	m	565	34	233	218	80	-	-	-	-
Sublevel in Ore	m	4,976	150	1,166	1,095	1,861	704	-	-	-
Sublevel in Waste	m	338	-	20	87	191	40	-	-	-
Raise	m	36	-	-	-	36	-	-	-	-
Existing Development Slashing	m	125	92	-	33	-	-	-	-	-
Production										
Ore Production	t	982,963	3,515	94,617	109,421	176,375	182,392	182,412	179,111	55,120
Ag	g/t	307	396	343	301	275	330	268	336	306
Cu	%	1.34	1.71	1.55	1.43	1.21	1.44	1.14	1.42	1.30
Sb	%	0.92	1.10	1.02	1.01	0.84	1.03	0.76	0.96	0.89

This assessment is preliminary in nature, as it includes Inferred Mineral Resources that are presently considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the assumptions used in this PEA will be realized.

16.5 Ventilation

The ventilation arrangement uses the historical Mária Mine and the existing connections between the two mines as the fresh air source for Strieborná. Intake air will be drawn through the Meyer haulage tunnel and the inclined shaft. The fresh air will then downcast from Level 6 down the Mária shaft, be drawn to the Strieborná vein workings via the Mária Mine connecting main levels, and thereafter upcast through the Strieborná

vein area. An auxiliary ventilation fan and flexible ducting will be used to force fresh air from the Strieborná Ramp into each active sublevel.

A surface exhaust fan will be located on top of what was previously the Mária Mine backfill raise. A connection will be made between the raise and the HTRC and the bottom of the raise will be sealed to prevent short circuiting of the ventilation system. When in operation, the exhaust fan will draw fresh air down through the Mária Mine via a number of direct airway connections to surface, flow across to the various drift connections to the Strieborná Ramp, flow up the Strieborná Ramp, and up along the HTRC to the ventilation raise connection, where it will be exhausted to surface.

Only the upper 85 m portion of the existing Mária Mine backfill raise from the HTRC to surface will be utilized to exhaust mine air. This raise could easily be twinned with an addition ventilation raise during the latter half of the LOM depending on ventilation requirements in the lower portion of the mine.

Ventilation doors will be required along the upper portion of the HTRC in order to regulate the airflow quantity along the Haulage Tunnel in order to preferentially draw return air up through the Strieborná Ramp system.

The portion of the Strieborná Ramp located below the Level 10 drift connection to the Mária Mine will require a ventilation raise and an incorporated ladderway in order to provide fresh air to the bottom of the Strieborná ramp system and an alternate means of egress from Level 11.

A schematic long section showing the planned ventilation system is shown in Figure 16-15.

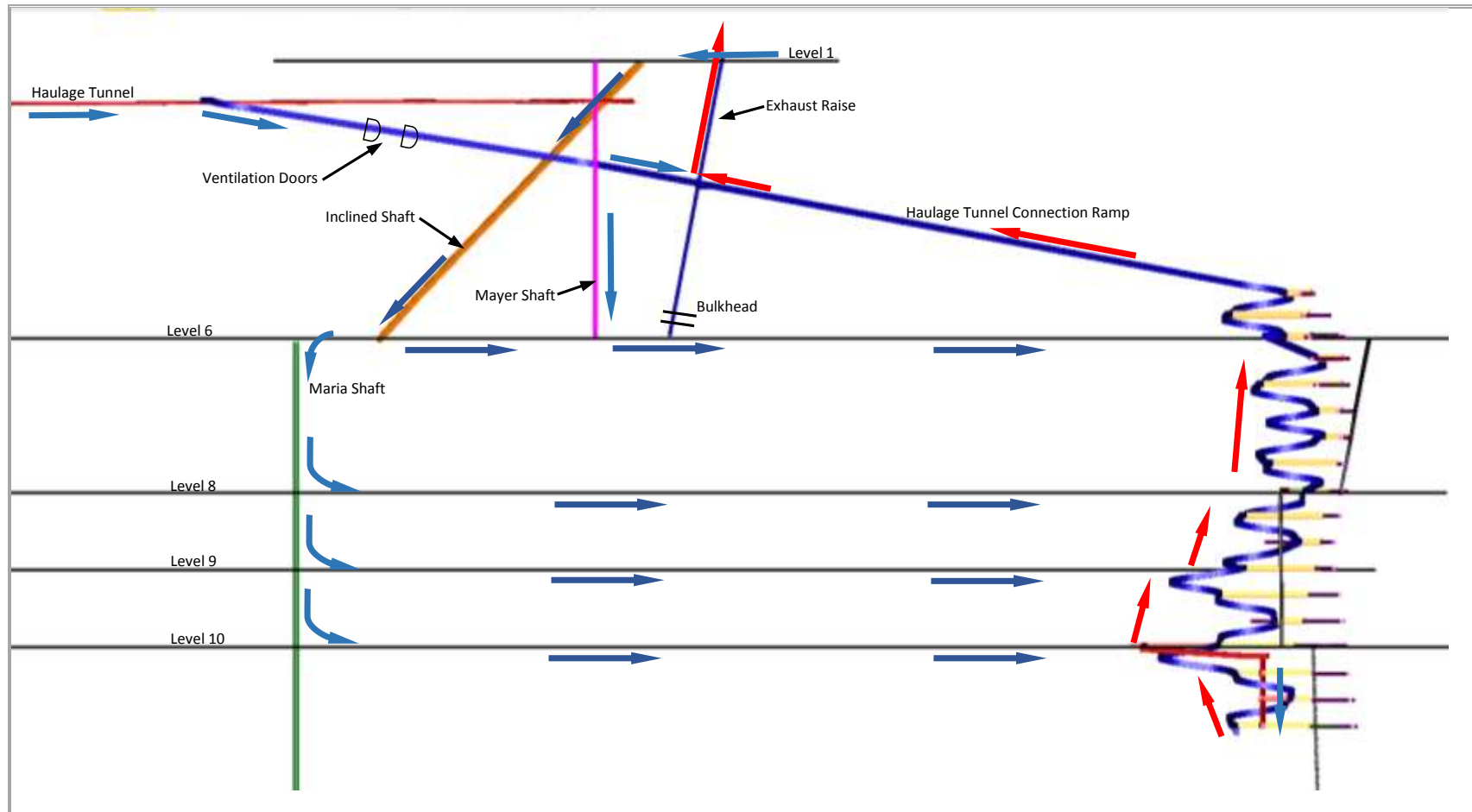
Diesel equipment fleets were estimated by modelling, and the required maximum ventilation flows were estimated using an air requirement of $0.06 \text{ m}^3/\text{s}/\text{kW}$, as shown in Table 16-11. Definitive definition and design of the ventilation system will be required in future studies.

Table 16-11: Estimated Maximum Ventilation Requirements

Equipment	kW/Unit	Max. Fleet	Installed kW	Estimated Usage	Air Requirement at $0.6 \text{ m}^3/\text{s}/\text{kW}$
3.5 t Scoop	72	1	72	80.0%	3
6.7 t Scoop	150	4	805	80.0%	39
15 t Truck	164	3	660	80.0%	32
Single Boom Jumbo	55	2	148	80.0%	7
Grader	103	1	138	50.0%	4
Fitters Vehicle	56	1	75	50.0%	2
Supervisor Transport	15	5	101	50.0%	3
Forklift	37	1	50	50.0%	1
Flat Deck and Boom	69	1	93	50.0%	3

Equipment	kW/Unit	Max. Fleet	Installed kW	Estimated Usage	Air Requirement at 0.6 m ³ /s/kW
Scissors Lift	69	2	185	50.0%	6
Sanitation Vehicle	69	1	93	50.0%	3
Rescue/Ambulance	96	1	129	1.0%	0
Bobcat	40	1	54	50.0%	2
Minimum per Equipment Fleet	-	-	-	m ³ /sec	105
	-	-	-	CFM	221,464
Leakage & Instantaneous Equipment Usage	-	-	-	-	15%
Contingency	-	-	-	-	5%
Design CFM	-	-	-	-	265,757
Design m ³ /sec	-	-	-	-	125

Figure 16-15: Ventilation System



16.6 Hydrogeology

The dewatering strategy for the existing mine was described in Section 16.4. In general, the hydrogeological regime below Level 6 at Strieborná vein is unknown at this time. Visual inspection suggests that the rock is quite tight, and inflow should be limited to service water and general seepage. Strieborná mine water drainage will be accomplished by gravity flow back to the Mária Shaft along the various connection drifts, and will be pumped to surface via the Mária Mine dewatering system. A small pump station will be required at the bottom level of the Strieborná Mine in order to discharge water reporting to Level 11 onto the existing Level 10. The water will be channeled to the Mária shaft on each main level (Levels 8, 9 and 10), where it will be piped down to the un-dewatered area of the mine below Level 11.

The water level in the subvertical shaft will be automatically monitored, and pumped when necessary to maintain the permit requirement that water levels remain more than one level (50 m) below the lowest production level. Alarms will be installed to indicate rising water levels. Pump stations will be installed on Level 10 and Level 6 for staged pumping of water to surface.

Cost allowances were provided in estimation, but definitive design of the system will be required in future studies once additional data are available.

16.7 Mine Safety

16.7.1 Fire Prevention

All diesel equipment (light vehicles and heavy-duty mobile equipment) will be equipped with automatic fire suppression systems and hand-held fire extinguishers. Hand-held fire extinguishers will be located throughout the mine at refuelling bays, workshops, explosives and detonator magazines, refuge bays, and lunchrooms. Refuelling bays, workshops, and explosives and detonator magazines will be equipped with automatic deluge systems.

A mine-wide stench gas system will be installed at the fresh air intakes to alert underground workers in the event of an emergency.

16.7.2 Mine Rescue

A mine rescue team will consist of members selected and trained from the Strieborná Mine site, and at least two teams would be available on rotation for rescue efforts. Surface and underground training facilities will be necessary for ongoing employee training and refresher training programs. Dedicated mine rescue equipment, including a rescue vehicle and all supporting testing and maintenance equipment for mine rescue purposes, will be available; specific underground mine rescue equipment would include self-contained breathing apparatus.

16.7.3 *Refuge Stations*

Each lunchroom will automatically be designed and designated as a refuge station. Portable refuge chambers accommodating from 12 to 16 people are planned for the initial development period and for subsequent isolated areas. The refuge stations will be equipped with CO and CO₂ scrubbers, medical oxygen cylinders, oxygen candle, air conditioning, first aid kit, radio, telephone, and a toilet. They would provide protection for at least 36 hours as a self-contained unit. Access will be through an air lock door and they will be equipped with compressed air line cylinders and sealant to hermetically seal the doors in order to prevent the entry of harmful gases.

16.7.4 *Emergency Egress*

In addition to the main trackless mining ramp accesses, independent emergency egress to Level 6 from each main level will be provided by the ladderway in the subvertical shaft raise. From Level 6, the incline shaft manway and a fresh air raise manway will provide separate emergency egress to surface. The haulage tunnel may also be used for emergency egress as necessary.

16.7.5 *Dust Control*

Spray nozzles operated by the mobile equipment drivers will be installed at all material loading points for dust control.

16.8 **Cost Estimation Methodology**

A series of unit cost models was adapted to reflect the direct activities at the mine. Each of the models was developed reflecting the mine design criteria and other general engineering estimates of performance. All operations will be undertaken by owner-employed crews, apart from raising, which will be completed by a mining contractor. For the first three years, the mine was assumed to work on three 8 hour shifts per day, 5 days per week; a total of 251 d/a at a mine production rate of 430 t/d, and total annual production of 110 kt. After Year 3 month 1, a seven days per week working schedule was adopted at a mining rate of 500 t/d, thereby increasing the mill feed tonnage rate to 182 kt/a. A 50-minute effective work-hour was used. All costs were modelled as end 3rd quarter 2014 United States Dollars. The cost models included ground support assumptions provided by the geotechnical study. The unit rates were applied to the scheduled quantities in order to estimate the direct costs.

Additional models were designed to reflect overhead-type activities at the mine:

- Mine Services and Fixed Plant (including labour, supplies, and equipment for construction, materials transport, road maintenance, and sanitation); diesel maintenance labour costs are also included
- Owners Mine Management and Technical (including mine supervision, mine technical and safety staff)

- Mine Power (developed from aggregation of mine loads and estimated usage).

Overheads were estimated on an annual basis and applied as a fixed daily cost. The overheads for each period were split between operating and capital development estimates in the ratio of the respective direct costs. The estimate was completed on a quarterly basis from Year -1 to the end of Year 4, and annually thereafter.

Consumables and material unit pricing for underground mining activities, were applied using local data provided by Global Minerals and data from other recent AGP mining projects. Where necessary, costs were escalated at the rate of 4% per year from the date of information to reflect end 3rd quarter 2014 costs.

Details of the material costs used in the analysis are provided in Table 16-12.

Table 16-12: Major Material and Consumables Cost Assumptions

Item	Unit	Cost US\$
Emulsion Explosive (Cartridge)	kg	2.39
Trim Product (Cartridge)	kg	12.59
ANFO (Bagged)	kg	1.06
Dev Detonator	ea.	5.16
Stoping Detonator	ea.	5.86
2.4 m No. 6 Rebar (complete)	ea.	33.95
1.8m Split Set (complete)	ea.	11.83
0.6m Splitset	ea.	3.19
6m Cablebolt (complete)	ea.	72.70
mesh, 4" x 4", 1.5 m x 3 m, 6 gauge	ea.	51.46
43 mm Development Face Drill Hole (Consumables)	m	1.27
43 mm Development Support Drill Hole (Consumables)	m	0.99
48 mm Development Cablebolt Drilling	m	3.11
64 mm Stopping Longhole (Consumables)	m	3.55
36" Ventilation FlexiDuct Installed	m	23.36
30" Ventilation FlexiDuct Installed	m	20.36
6" Pipe Installed	m	84.06
4" Pipe Installed	m	44.87
Diesel	l	2.06
Power	kWh	0.17

Global Minerals provided an estimate of local labour rates.

The models were also used to track labour and equipment hours, to identify annual requirements in each labour category and equipment type.

A summary of the unit costs derived during the modelling process is shown in Table 16-13.

Table 16-13: Summary of Underground Unit Costs

Activity	Unit	Cost
Direct Costs		
Ramp - 3.5 m wide x 4.0 m high	\$/m	1,490
Waste Access - 3.5 m wide x 4.0 m high	\$/m	1,333
Other Waste - 3.0 m wide x 3.0 m high	\$/m	1,010
Slashing From 2.4 m x 2.4 m to 3.5 m wide X 4.0 m high	\$/m	950
Sublevel In Ore - 2.5 m wide X 2.5 m high	\$/m	927
Sublevel In Waste - 2.5 m wide X 2.5 m high	\$/m	898
Conventional Raise - 2.5 m wide X 1.8 m high (Contractor)	\$/m	2,121
Conventional Raise - 2.8 m wide X 2.8 m high (Contractor)	\$/m	1,758
Stope Drilling and Blasting - 1.5 to 3.0 m Thick Orebody	\$/t	9.33
Stope Drilling and Blasting - 3.0 to 5.0 m Thick Orebody	\$/t	5.06
Stope Drilling and Blasting - Over 5.0 m Thick Orebody	\$/t	3.81
Production Mucking – 3.5-t LHD From Stopes less than 3 m Thick to Remuck	\$/t	4.25
Production Mucking – 6.7-t LHD From Stopes more than 3 m Thick to Remuck	\$/t	3.08
15-t Truck Loading and Haulage To Haulage Transfer Portal from:		
Haulage Transfer To Level 6	\$/t	8.07
Level 6	\$/t	11.08
Level 7	\$/t	13.08
Level 8	\$/t	15.08
Level 9	\$/t	17.09
Level 10	\$/t	19.09
Level 11	\$/t	21.09
Truck Dev Waste To Completed Stope	\$/t	6.02
Overheads		
Mine Services ⁽¹⁾	\$/d	7,976
Owners Supervision and Technical ⁽¹⁾	\$/d	6,155
Power Costs ⁽¹⁾	\$/d	3,688

Note: ⁽¹⁾ Overheads provided indicate scale of expense. Modelled estimated overheads vary in each estimate period depending on mine activity.

16.8.1 Equipment

Modelled equipment requirements are based on operational hours. Recent quotations for other AGP projects were used for the equipment types selected. These quotations were escalated by 4% per year from the date of quotation where necessary. Mechanical availability, utilization, and operational life were estimated by AGP for each equipment type, and the hourly operating costs were assessed. Equipment was assumed to be leased at a monthly rate of 4% of the purchase price, and these charges were included in the hourly operating costs. Purchase prices were taken from AGP's database of recent quotations, inflated by 4% per year from the quotation date to 3rd quarter 2014. Table 16-14 shows the equipment data used for modelling.

Table 16-14: Equipment Costs and Operational Data

Cost	Utilization (%)	Hourly Cost (\$)	Equip. Life (\$)
Scoop 3.5 t	55	109.39	20,000
Scoop 6.7 t	55	192.58	20,000
15 t Truck	55	126.55	25,000
1 Boom Jumbo	52	121.91	20,000
Longhole Drill	55	46.39	20,000
Grader	55	97.72	20,000
Toyota Fitters	55	38.62	16,000
Run Around	55	19.17	12,000
Forklift	55	44.42	12,000
Boom truck	55	70.38	20,000
Scissors Lift	55	73.37	20,000
Sanitation Truck	55	78.06	20,000
Toyota Rescue	55	63.81	16,000
Bobcat	55	46.26	20,000

The mine diesel fleet will pose a fire hazard. All vehicles will be fitted with on-board detection and suppression systems; in addition, a mine-wide fire detection system is recommended. The underground diesel workshop will be ventilated directly to the main return air system.

As the activities vary, the equipment fleet requirements change. Table 16-15 shows the fleet requirements for example periods.

Table 16-15: Modelled Equipment Requirements

Fleet Type	Y-1 Q4	Y1 Q4	Y2 Q4	Y3 Q4	Y4 Q4	Y6
Scoop 3.5 t	-	1	1	1	1	1
Scoop 6.7 t	1	4	3	3	3	3
15 t Truck	1	3	3	3	3	3
1 Boom Jumbo	1	2	2	1	-	-
Longhole Drill	-	1	1	1	1	1
Grader	1	1	1	1	1	1
Toyota Fitters	1	1	1	1	1	1
Run Around	5	5	5	5	5	5
Forklift	1	1	1	1	1	1
Boom truck	1	1	1	1	1	1
Scissors Lift	1	2	2	2	2	2
Sanitation Truck	1	1	1	1	1	1
Toyota Rescue	1	1	1	1	1	1
Bobcat	1	1	1	1	1	1

16.8.2 Labour

Global Minerals plans to conduct its own mining operations, with contractor involvement limited to raise mining. Historically, the local area has a rich history of mining

experience; however, much of this experience was gained under previous communist regimes, using non-mechanized equipment and labour-intensive methods. Modern, highly technical, computerized equipment is not planned for use at Strieborná; however, even the more basic trackless equipment planned for use at the mine will require operational expertise, which is currently lacking in the local workforce. Global Minerals will need an effective plan for the training of its local workforce, and may initially need to employ experienced, non-local operators during the early years of operation.

Annual labour force plans were developed to support the LOM plan and the activities scheduled to meet production objectives. The labour force tables provided reflect the underground workforce required to support development and production activities.

Details of the basic unit labour rates for production and mine supervision/technical staff salaries used in the analysis are provided in Table 16-16 and Table 16-17. Global Minerals provided the labour rates, which include bonuses and benefits.

Table 16-16: Hourly Paid Labour Cost Assumptions

Job	Owners Total (\$/h)
Drilling, Blasting, Ground Support, Shaft Personnel	18.45
Shift Boss	16.32
LHD/Truck Drivers	15.37
Construction, Auxiliary Equipment, and Pump Operators	14.76
Labourer	11.68
Mechanic/Electrician I	15.99
Mechanic/Electrician II	14.76

Table 16-17: Owner and Contractor Staff Cost Assumptions

Position	Annual Cost to Company (\$)
Mine Superintendent	162,240
Mine Captain	50,545
Mine Dry	20,007
Safety/Trainers Supervisor	42,121
Secretary/Clerk/Stores	20,007
Chief Engineer	34,750
Senior Geologist	33,697
Mine Geologist	31,591
Geology Technician	29,485
Senior Mine Engineer	33,697
Mine Engineer	31,591
Mine Technician	29,485
Chief Surveyor	35,803
Survey Helper	26,325

Position	Annual Cost to Company (\$)
Ventilation /Samplers/Rock mechanics Assistant	26,325
Maintenance Superintendent	50,545
Maintenance Foreman	31,591
Assistant Foreman	26,325
Maintenance Planner	26,325

Before Year 3 month 2, hourly paid employees will work on three shifts of 8 hours each, five days per week, for 251 mine-operating days per year. Including 26 days' vacation, public holidays, and allowances for overtime and absenteeism, each hourly paid employee will work 2,007 hours per year. Depending on their operational role, hourly employees will work on a one, two, or three shifts per day basis. After Year 3 month 2, the hourly paid workforce will work seven days per week, which will require a 40% increase in hourly paid employees.

Staff will work on an 8 hours per day, 5 days per week basis throughout the LOM plan working 2,000 hours per year. Production support will be covered by changes in work rosters and call-out when necessary. Job categories will be manned mainly on one shift per day, but infrequently on two or three shifts per day, depending on the position.

Table 16-18 provides the estimate of personnel employed by the owner for selected periods during the development and production phases of the LOM.

Table 16-18: Employed Labour

Position	Y-1 Q3	Y1 Q3	Y2 Q3	Y3 Q3	Y4 Q3	Y6
Hourly Paid						
Development Miner	13	21	14	8	-	-
Scoop Driver	2	3	3	3	2	2
Stope Blasting	-	2	2	3	2	3
Construction	3	4	4	6	6	6
Stope Drilling	-	4	4	6	5	6
Materials	2	4	4	6	6	6
Truck/Scoop Driver	3	12	9	10	9	11
Road Maintenance	-	1	1	1	1	1
Mechanic I	4	7	6	8	7	8
Mechanic II	4	8	7	9	8	9
Electrician I	1	1	1	1	1	1
Electrician II	1	2	2	3	3	3
Total Hourly Paid	33	68	57	64	50	56
Staff						
Maintenance Superintendent	1	1	1	1	1	1
Maintenance Foreman	1	1	1	1	1	1
Assistant Foreman	1	1	1	1	1	1

Position	Y-1 Q3	Y1 Q3	Y2 Q3	Y3 Q3	Y4 Q3	Y6
Maintenance Planner	1	1	1	1	1	-
Mine Superintendent	1	1	1	1	1	1
Mine Captain	-	1	1	1	1	1
Shift Boss	3	6	6	8	8	8
Mine Dry	2	2	2	2	2	2
Secretary/Clerk/Stores	2	4	4	4	4	4
Safety/Trainers Supervisor	2	2	2	2	2	2
Chief Engineer	1	1	1	1	1	1
Senior Mine Engineer	1	1	1	1	1	1
Senior Geologist	1	1	1	1	1	1
Mine Geologist	-	1	1	1	1	1
Mine Technician	2	2	2	2	2	2
Geology Technician	2	2	2	2	2	2
Mine Engineer	-	1	1	1	1	1
Chief Surveyor	1	1	1	1	1	1
Survey Helper	2	2	2	2	2	2
Ventilation /Samplers/Rockmechanics Assistant	2	4	4	4	4	4
Total Staff	26	36	36	38	38	37
Grand Total	59	104	93	102	88	93

16.8.3 Power

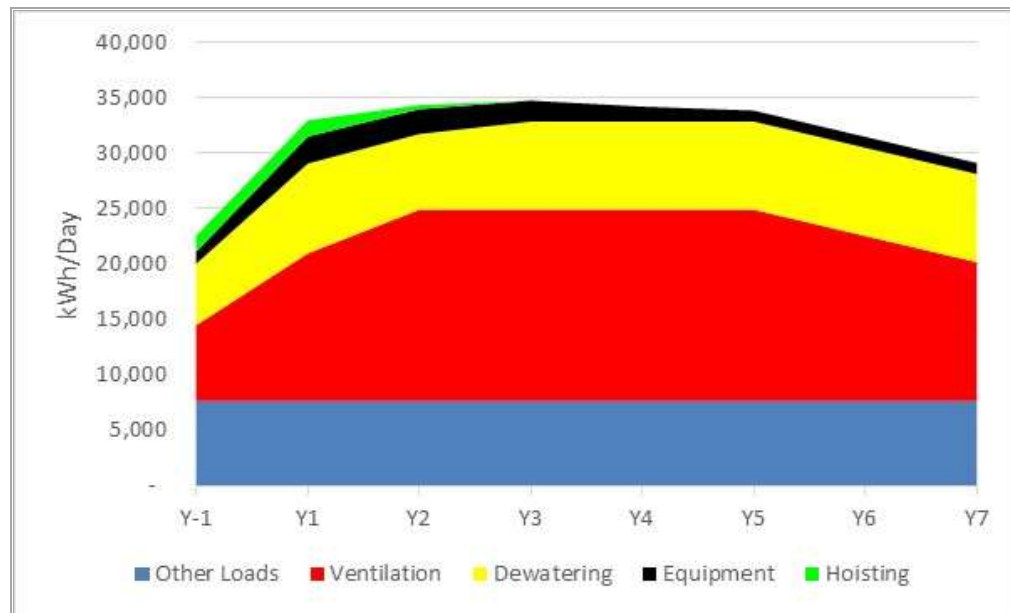
A load list was estimated for the underground mine, which is summariszd in Table 16-19.

Table 16-19: Summary Total Mine Power Requirements

	Installed kW	Average kW in Operation
Ventilation	690	495
Hoisting	121	42
Dewatering	565	228
Equipment	185	70
Other Loads	350	219
Total	1,911	1,053

The load list was examined by period to estimate the annual power requirements, as shown in Figure 16-16.

Figure 16-16: Power Usage



16.8.4 Mine Infrastructure

Estimates for rehabilitation were based on local contractor quotations received by Global Minerals for early rehabilitation plans, since superseded. AGP recommends that Global Minerals seeks new quotations for the rehabilitation plans arising from this study. Allowances for the other mine infrastructure items were included in the capital estimate:

- Rehabilitation
 - dewatering of the existing mine
 - removal of debris and limited rehabilitation of the subvertical shaft
 - subvertical winch hoist
 - inclined shaft transfer points
 - removal of haulage level tracks and minor rehabilitation
 - rehabilitation of existing Levels 8, 9, and 10
- UG Infrastructure
 - UG Lighting
 - UG service water system
 - dewatering pumps and controls
 - face pumps
 - UG fuel storage
 - workshop concrete and equipment
 - backfill cement mixing system
 - leaky feeder/communications/automation controls

- ventilation doors, seals, and regulators
- auxiliary fans
- temporary portable 16-man refuges
- mobile compressors
- UG explosives storage
- permanent refuge stations/lunchroom/shift bosses office
- Surface Infrastructure
 - surface mill feed tip
 - main fans, buildings, and ducting
 - stench gas system
 - centralized blasting system
 - respirators, cap lamps, and chargers
 - mine rescue equipment
 - first aid supplies and oxygen treatment facilities
 - technical equipment and software
 - survey equipment, office equipment, and supplies
 - fire extinguishers and refilling equipment
 - mine office
 - change house
 - surface stores
 - surface workshops
 - yard distribution
 - infrastructure EPCM.

16.9 Comments on Section 16

16.9.1 Geotechnical

The data available at this stage of study varies in reliability. Estimates of engineering properties are provided with ranges where possible, and sensitivity analyses are encouraged for mine design based on these data. The interpretations presented within this report are preliminary in nature, and require additional validation and testing with higher quality data before they can be applied to more intensive design studies.

In Section 16.2.4, recommendations were provided for stope dimension design (in the form of hydraulic radii), rib and sill pillar dimensions, and development and stope support. These recommendations have been adopted in the mine design and mining cost estimates for this study.

The current design factors are provided for general guidance to the PEA-level mine design and for cost estimating purposes. As additional geological and geotechnical work is completed, the current designs should be reviewed and updated as required. Concurrently, if the proposed mining method or mine plan changes considerably from the one assumed in this report, the assumptions inherent within the rock mechanics assessment should be reviewed and updated as required.

16.9.2 Mining

- The mine plans described in this study indicate the potential to produce 983 kt of mill feed grading 307 g/t Ag and 1.3% Cu for the process plant over an operating period of seven years. Five days per week operation will be increased to seven-day operation early in Year 3, when planned production rates increase from 430 t/d to 500 t/d.
- 99.6% of mill feed will be drawn from material classified as Measured and Indicated Resources.
- Average stope width is 4.9 m, and unplanned dilution in plant feed, defined as waste tonnes/mill feed tonnes, is estimated to be 13.5%.
- With the benefit of prior mining experience, and under test conditions, it may be possible to increase stope dimensions, thereby increasing resource recovery and enhancing project economics.
- Backfill will not be routinely placed in completed stopes; however, once available, completed stopes will be utilized for the disposal of development waste.
- It is planned that owner crews will undertake the underground mining operations using a fleet of leased mobile equipment. Relatively small-sized equipment will be used, in line with the limited planned production rate.
- Historically, the local area has a rich history of mining experience; however, much of this experience was gained under previous communist regimes using non-mechanized equipment and labour-intensive methods. Global Minerals will need an effective plan for the training of its local workforce, and may initially need to employ experienced, non-local operators during the early years of operation.
- A significant program of rehabilitation and dewatering of the existing workings of the original Mária Mine will be required during the early years of the adjacent Strieborná vein development and production operations.
- The original Mária Mine was linked at depth to the now-flooded Sadlovsky Mine. Three watertight doors/bulkheads were installed in the connecting drift, but these will remain below water levels, and no inspection will be possible. The volume of water in the Sadlovsky Mine is not known, but is considered by Global Minerals to be significant. Government permitting requires that water levels in the Mária Mine be maintained at least one production level (50 m) below the

planned operations in order to provide sufficient time for the emergency evacuation of the Strieborná workforce. Notwithstanding, there is therefore risk that a major inundation of water could significantly disrupt the planned mining operations at Strieborná.

- This assessment is preliminary in nature, as it includes Inferred Mineral Resources that are presently considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the assumptions used in this PEA will be realized.



17 RECOVERY METHODS

17.1 Introduction

The proposed method for recovering silver and copper from the Strieborná deposit consists of crushing, grinding, flotation and dewatering to produce a flotation concentrate for intermediate treatment at an off-site roaster and subsequent smelting of the calcine.

This section describes the process flowsheet, preliminary design criteria, and process description for a 500 t/d concentration plant to be located just west of the old Mária Mine tailing storage facility. The downstream roasting treatment is described in Section 19.

The mining schedule developed for this PEA predicts an average daily throughput of 300 tonnes during the first 24 months of operation, with production ramping up to the full 500 t/d capacity after that period. Average LOM headgrade to the plant is estimated to be 307 g/t Ag, 1.34% Cu, 0.92% Sb. Y1-3 headgrade estimates are very close to the LOM average – 300 g/t Ag, 1.36% Cu, 0.93% Sb.

17.2 Process Flowsheet

Using the 2012-2014 testwork as a guideline, AGP developed a process flowsheet for this PEA consisting of two-stage (jaw and cone) crushing, ball mill grinding, froth flotation, concentrate dewatering, and tailings dewatering. A schematic of the proposed flowsheet is presented in Figure 17-1.

17.3 Interpretation of Testwork

AGP has estimated the average recovery of copper and silver ultimately achievable using the crushing/grinding/flotation flowsheet described in this section. Further enhancements are made to the flotation concentrate product by an external roasting process as described in Section 19.

The flotation average performance prediction is based on results of the two locked-cycle flotation tests presented in Section 13, but also makes adjustments for the plant feed grades estimated in the mine production schedule. Table 17-1 provides a summary of AGP's projection.

Table 17-1: Metallurgical Projection for the Life of Mine Average Mill Feed

Product	Weight %	Assays	% Distribution
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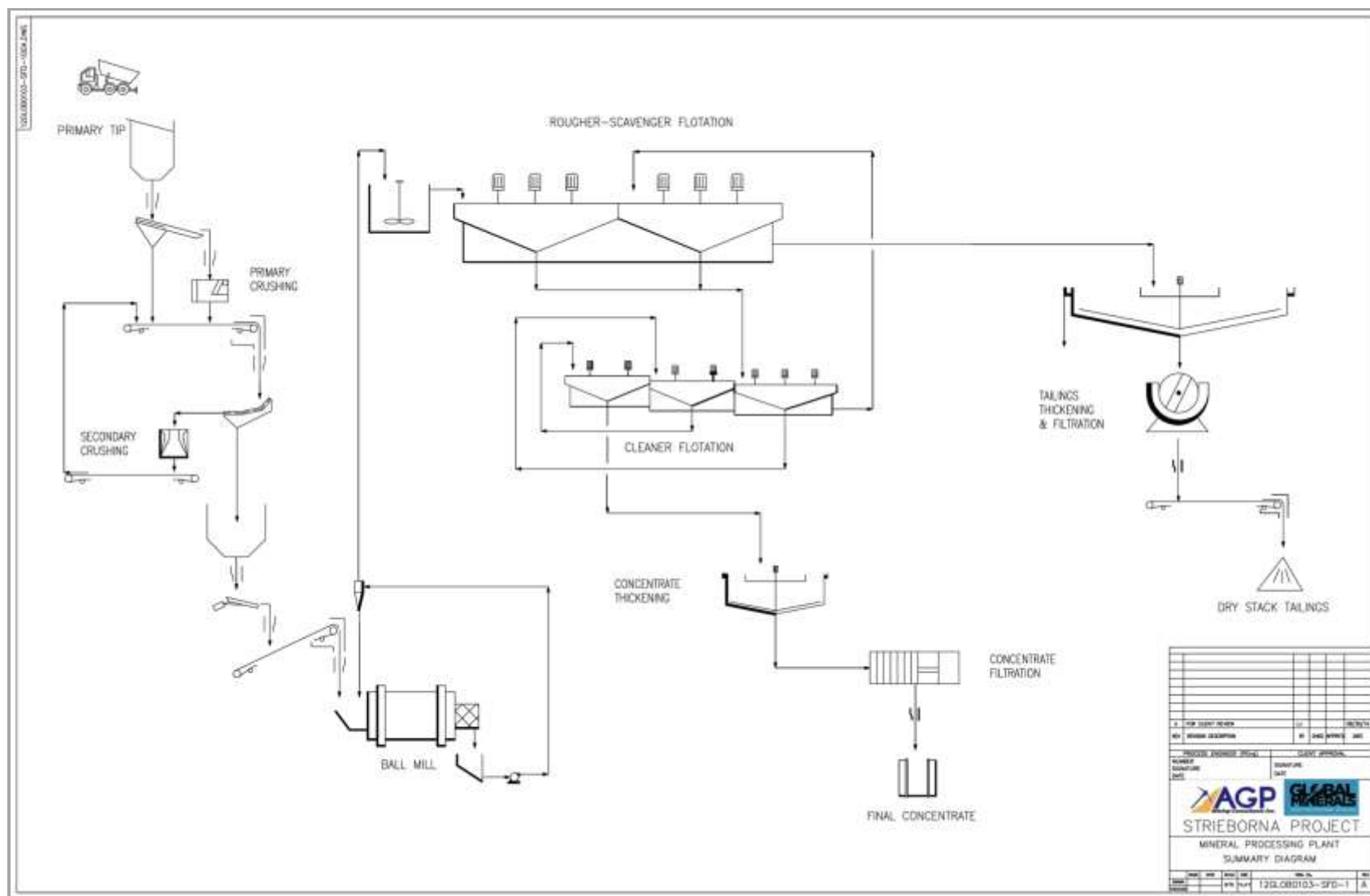
Product	Weight %	Assays			% Distribution		
		% Cu	% S	g/t Ag	Cu	S	Ag
Cleaner Concentrate	4.6	27.8	22.6	6570	98.1	95.6	98.6
Cleaner Tailings	0.9	0.58	1.07	129	0.2	0.4	0.2
Rougher Tailings	95.0	0.02	0.03	3.5	1.8	3.0	1.3
Feed	100	1.30	0.99	307	100	100	100

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NI 43-101 PRELIMINARY ECONOMIC ASSESSMENT FOR THE
STRIEBORNÁ PROJECT, EASTERN SLOVAKIA



Figure 17-1: Flowsheet for the Strieborná Concentrating Plant



In AGP's opinion, the flotation testwork conducted at Blue Coast Research suggests that a relatively simple flotation circuit consisting of rougher flotation followed by three stages of cleaning, without regrind, is sufficient to generate a flotation concentrate with a LOM average copper grade of 27.8%. Copper and silver recoveries of 98.1% and 98.6%, respectively are indicated, although for the purposes of the PEA, these have been rounded down to 98.0%. For the PEA, silver grade in flotation concentrate has been calculated using the scheduled headgrade, the process recovery of 98.0% and the calculated flotation concentrate mass.

The locked-cycle results demonstrate good metallurgical stability during the testwork, an indication of the robustness of the flowsheet.

Importantly, the flotation concentrate produced by this flowsheet contains significant levels of mercury and antimony in addition to the copper and silver paymetals. The "deleterious elements" are so named for their impact on the smelting process and normally attract penalties in toll-smelting contracts. High mercury concentrations can render the product unsaleable. To mitigate this issue, an offsite roasting step has been included in the overall processing strategy and this is discussed in Section 19.

The flotation conditions (grind, flotation time, reagents, etc.) used in the locked-cycle testwork form the basis for the design criteria outlined below.

17.4 Design Criteria

Based on the available testwork, a set of preliminary plant design criteria was developed which provides all the specific unit operation process detail required for the equipment sizing and selection. A summary of some of the key criteria is presented in Table 17-2 and Table 17-3.

Table 17-2: Process Design Criteria Fundamentals

Design Criteria	Unit	Design Data
Project Fundamentals		
Daily Throughput	dmt/d	750
Nominal Feed Rate for Crusher Circuit	dmt/h	29.1
Nominal Feed Rate for Mill Circuit	dmt/h	22.6
Physical Characteristics		
Feed Moisture	%	5
Solid Particle Density	t/m ³	2.9
Abrasion Index (Estimated)	g	0.15
Bond Ball Work Index	kWh/t	8.7
Angle of Repose	deg.	45
Head Grade (Mill Feed)		
Cu	%	1.3
Ag	g/t	307
S _{total}	%	1.1

Table 17-3: Process Design Criteria – Equipment Details

Design Criteria	Unit	Design Data
Process Plant		
Primary Crusher	Metso C80 Jaw	
CSS	mm	50 mm
Product P ₁₀₀	mm	90
Product P ₈₀	mm	50
Secondary Crusher	Metso HP100 Cone	
CSS	mm	13
Product P ₁₀₀	mm	25
Product P ₈₀	mm	15
Grinding Mill	8 ft x 11 ft Overflow Discharge	
Applied Power @ Pinion	kWh/t	8
Pinion Power Consumption	kW	199
Recommended Installed Power	kW	275
Mill Discharge % Solids	%	65
Nominal Cyclone Overflow P ₈₀	µm	130
Flotation Circuit	Rougher, Scavenger, 1 st Cleaner	
Bulk Circuit Feed Flow	m ³ /h	63
Bulk Conditioning Tank Residence Time	min	5
Rougher Residence Time	min	8
Scavenger Residence Time	min	12.5
1 st Cleaner Residence Time	min	15
2 nd Cleaner Residence Time	min	12
3 rd Cleaner Residence Time	min	9
Concentrate Thickening	3 m High Rate	
Thickener Unit Area	t/d/m ²	3.4
Thickener Rise Rate	m/h	0.6
Concentrate Filter	Horizontal Pressure Filter	
% Moisture	%	9
Bulk weight	kg/L	3.40
Cake volume	m ³ /h	0.45
Tailings Thickener	8 m High Rate	
Thickener Unit Area	t/d/m ²	10.3
Thickener Rise Rate	m/h	0.7
Tailings Filter	Continuous Belt Filter	
% Moisture	%	18
Bulk weight	kg/L	3.10
Cake volume	m ³ /h	6.98

The fundamental design criteria have been developed using limited testwork, and should be considered preliminary.

17.5 Process Description

This section describes the plant design for a new copper/silver (tetrahedrite) concentrator at the Strieborná Project in Slovakia, at an operating throughput of 500 t/d.

17.5.1 Process Summary

The Strieborná concentrator is designed as a nominal 15,000 t/month plant. Mine haul trucks will tip into a surge bin feeding a primary jaw crusher designed for 86% availability. The primary crushed rock will be conveyed to a 15 mm sizing screen, with the oversize reporting to the secondary crusher, and the undersize fed forward to the ball mill circuit.

A 125-tonne capacity mill feed bin provides surge capacity and ensures a constant feed rate to the grinding circuit. The ball mill operates in closed circuit with a cyclone cluster. The cyclone pack cuts at a P_{80} of $\sim 130 \mu\text{m}$, providing the required liberation for good flotation.

The cyclone overflow reports to the feed box of the rougher flotation circuit. The rougher/scavenger flotation plant consists of eight trough cells in series. The rougher/scavenger combined concentrate is pumped to the cleaner flotation circuit.

Three stages of cleaner flotation are carried out with the cleaner tailings from each stage returning to the feed box of the previous stage. The 3rd Cleaner concentrate is the final product of the process plant and goes to the concentrate dewatering circuit. Cleaner tailings are pumped back to the head of the scavenger flotation bank.

Flotation tailing from the rougher-scavenger circuit is thickened and belt-filtered prior to dry stacking.

Reagents are stored, mixed, and distributed from a central reagents area. Frother, collector, and lime are pumped from the reagents area to the flotation section using peristaltic reagent pumps to accurately dose to the process.

17.5.2 Detailed Process Description

Crushing – Area 100

Feed will be delivered to the primary tip by 20-tonne haul trucks at a frequency averaging one to two trucks per hour. Peak delivery rate is assumed to be 40 t/h. Ore is discharged directly into a rail-lined, 30-tonne surge bin.

A vibrating grizzly feeder at the bottom of the surge bin discharges $>60 \text{ mm}$ oversize into the primary crusher feed chute. Grizzly undersize drops onto the classification screen feed conveyor. The primary crusher consists of a 510 x 800mm jaw crusher with a closed side setting of 50 mm. Crushed ore discharges onto a sacrificial conveyor, which in turn discharges onto the classification screen feed conveyor. The screen feed conveyor is

equipped with an overhead magnet for removing tramp metal ahead of the secondary crusher.

A 1.2 m x 2.4 m vibrating screen, inclined at 5°, is used to classify the primary crushed ore and close the circuit around the secondary crusher. The screen deck consists of 15 mm slots, with the screen undersize chute feeding the fine ore bin conveyor. The fine ore bin conveyor is equipped with a weightometer for recording crushing circuit production. Oversize from the classification screen is fed by a lined chute into the secondary crusher.

The secondary crusher consists of a 100 hp cone crusher with a closed side setting of 13 mm. Crushed product discharges via a chute onto the recycle conveyor, which returns the material to the classification screen feed conveyor.

The crushing circuit is expected to run at 86% availability, seven days per week. A dust collection system is included in the classification screen and secondary crushing area.

Ball Milling – Area 200

The ball mill circuit and all subsequent areas of the plant operate at 92% availability on a 24/7 schedule. Crushing circuit product is stored in a 125-tonne fine ore bin with ≈6 hours operating capacity. An overflow chute at the top of the bin feeds a stockpile with an additional six hours of capacity. The stockpile material can be returned to the fine ore bin conveyor by front-end loader, as needed.

A vibrating pan feeder draws material from the bottom of the fine ore bin onto the mill feed conveyor. The conveyor is equipped with a weightometer to measure mass flow and allow for control of the mill feed rate.

The grinding mill consists of a 2.13 m diameter by 3 m long overflow discharge ball mill with a 275 kW, single pinion drive. The discharge end of the mill is fitted with a trommel screen with 13 mm openings. Oversize tramp material, woodchips, etc., drop through a chute into a drum or skip. The material passing through the trommel is collected in a chute and fed to the cyclone pump feed box.

The mill discharge is diluted with process water in the cyclone feed pump box and is then pumped to a cyclone cluster consisting of two 381 mm (15") diameter cyclones, one operating and one standby.

Cyclone underflow gravitates to the feed chute of the ball mill. The cyclone overflow reports to a sampling station that consists of a sampling launder and an automatic sampler. Spillage contained in the ball mill area is pumped to the mill discharge sump for retreatment.

Ball mill grinding media is delivered to the plant in 200 L drums, which are loaded up to the mill mezzanine by the overhead hoist. As required, grinding media is added manually by the operator, via a ball-loading chute.

Rougher Flotation – Area 300

Cyclone overflow flows by gravity to an agitated conditioning tank with five minutes residence time. Slurried lime is added to adjust the pulp pH, and collector (Aerofloat 3477) is introduced. The conditioning tank overflows into the feed box of a bank of three DR30 type rougher flotation cells with a combined residence time of eight minutes. Flotation air to each cell is supplied by flotation blowers via a low pressure manifold and is manually controlled. Pulp level is maintained by modulating dart valves.

Rougher concentrate is collected in a common launder and flows by gravity to the rougher concentrate froth pump, which pumps to the first cleaner feed box. The rougher tailings flow by gravity into a scavenger bank that consists of five DR30 type flotation cells with an additional 12 minutes of residence time. The concentrate collected from the scavenger float is collected in a common launder and flows by gravity to the same froth pump as the rougher concentrate, to be pumped to the cleaner circuit. Scavenger tailings are the final tailings from the flotation circuit and report to the tailings pump feed box, and are then pumped to the tailings dewatering circuit.

Spillage in the rougher section is collected in a common sump and pumped back into the first rougher cell using a submersible spillage pump.

Cleaner Flotation – Area 320

Feed to the cleaner flotation circuit, in the form of rougher/scavenger concentrate, is fed to the feed box of a 1st Cleaner bank made up of three DR18 type trough flotation cells in series. The residence time in the 1st Cleaner circuit is estimated at 15 minutes, with the concentrate collected in a common launder that reports to a dedicated froth pump. 1st Cleaner tailings are pumped back to the feed box of the scavenger bank.

The 1st Cleaner concentrate is pumped to the feed box of the 2nd Cleaner, which consists of two DR18 type flotation cells in series. The 2nd Cleaner concentrate is pumped to the 3rd Cleaner, while the 2nd Cleaner tailings flow by gravity to the 1st Cleaner. Similarly, the 3rd Cleaner consists of two DR18 cells, with the 3rd Cleaner tailings reporting to the 2nd Cleaner. Residence times in the 2nd and 3rd Cleaner stages are estimated at 12 minutes and 9 minutes, respectively.

The 3rd Cleaner concentrate is the final concentrate product for the flotation circuit, and is pumped to the concentrate dewatering area.

The cleaner area spillage is collected in bermed areas and directed into the cleaner area spillage pump, which pumps back to the 1st Cleaner feed box.

Concentrate Dewatering – Area 400

Final copper/silver concentrate is pumped to the concentrate thickener sampling box and sampler before entering the 3 m diameter concentrate thickener for dewatering. The thickener is equipped with rake lift, bed level detection, and bed mass monitoring.

Thickener overflow gravitates to the spray water tank for recycling, while the thickener underflow is withdrawn from the cone by a centrifugal underflow pump and pumped forward to the concentrate storage tank, or recycled to the thickener feed if of insufficient density.

The concentrate is pumped from the mechanically-agitated storage tank to a horizontal pressure filter for dewatering. Filtrate from the press is recycled back to the concentrate thickener feed launder.

Concentrate filter cake is discharged from the press via two cake discharge chutes onto the cake transfer belt which transfers cake to the concentrate storage shed. A front-end loader serves the concentrate stockpile and loads cake into side tipping trucks which then transport the concentrate to the off-site roasting facility. Trucks are weighed and auger-sampled at the weighbridge prior to dispatch.

Concentrate dewatering area spillage is recovered by pumping back to the respective concentrate thickener.

Tailing Dewatering – Area 450

The scavenger tailing pumped from the flotation area discharges into the feedwell of an 8 m diameter high-rate thickener. The thickener is equipped with rake lift, bed-level detection, and bed mass monitoring. Thickener overflow gravitates to the process water tank, while the thickener underflow is withdrawn from the cone by a centrifugal underflow pump and pumped forward to the tailing filter.

The tailing filter consists of a continuously-operated vacuum belt filter that produces a tailings product at 18% moisture. Filtrate is returned to the feed launder of the tailings thickener. The filter cake discharges from the press via a chute onto the tailing conveyor, which transfers the tailing cake to a storage pile within a three-sided, concrete-walled shed. A front-end loader is used to load a 20-tonne dump truck to transport the tailing cake to the dry stack area, at an average rate of approximately one truckload per hour.

Services – Area 500

Process water is stored in a 200 m³ tank and is distributed to the plant by a process water pump. The hose-down water supply pumps provide plant hosing/flushing water. The process water tank is also used to feed the diesel-powered firewater pump from a separate (lower) offtake, thus guaranteeing availability.

Clean water is piped into the plant from the nearby river, and stored in the plant clean water tank. From the storage tank, water is pumped around the plant for use as reagent mixing water, slurry pump gland seal water, and as required for mill lubrication system cooling.

Two compressors provide plant and instrument air. A filter maintains air quality. Instrument air is dried using a refrigeration drier. An air receiver is provided for compressed and instrument air lines, to allow for surges in demand.

Two separate blowers supply low-pressure air to the flotation plant. The blowers are fixed speed, with manifold pressure controlled by a modulating valve on an exhaust line.

Reagents – Area 600

PH Modifier – Calcium Hydroxide

Lime (calcium hydroxide) is delivered to the plant in one tonne bulk bags and loaded into the lime hopper. Dry lime is metered from the hopper into the agitated mixing tank by a screw feeder and mixing plate. Mixed lime slurry, at 10% solids, is pumped to an agitated dosing tank. A circulation pump supplies lime to the flotation circuit via a ring main.

Collector – Aerofloat 3477

Flotation Collector Aerofloat 3477 is delivered to the plant in 200 L drums. Peristaltic hose pumps metre the reagent directly from the drums to several addition points throughout the plant.

Frother – MIBC

Liquid Methyl Isobutyl Carbinol (MIBC) is delivered to site in 1 m³ totes. As delivered (100% strength), MIBC is pumped directly to the dosing points by dedicated peristaltic pumps.

Flocculant – Magnafloc 10

Flocculant powder is delivered to site in one tonne bags and stored in the reagent storage area. Bags are lifted by the reagent area crane and added to the flocculant powder hopper. Powder is withdrawn by the flocculant screw feeder and blown through a venturi to a wetting head located on top of the mechanically-agitated mixing tank.

From the mixing tank, mixed flocculant can be fed forward to the storage tanks, or recycled back into the mixing tank to aid mixing. From the storage tank, flocculant is pumped directly to the tailings and concentrate thickeners.

Reagent spillage is pumped to the tailings thickener, or stored in totes for disposal. The reagent area is served by a safety shower.

18 PROJECT INFRASTRUCTURE

18.1 Infrastructure and Site Layout

The Strieborná Project infrastructure consists of the following:

- haulage tunnel to the mill feed transfer area
- a new access ramp from the existing haulage tunnel to the mining levels, existing internal inclined shaft and ramp/portal at the from the Mária Mine
- process plant, office/administration and dry complex
- mobile equipment shops at mine and primary crusher and maintenance shops at all three locations
- electrical distribution system
- fuel storage
- rehabilitated existing TSF.

This infrastructure is required to support a 500 t/d underground operation.

18.1.1 *Mine/Mill Site Operations*

The mill or process plant is to be located to the south of the haulage tunnel exit portal and west of the TSF. The primary crusher is located at the mill site. The mill feed will be transported by truck to the mill complex via an existing municipal road system. Built in close proximity to the mill building will be buildings containing offices, warehousing, welding shops, and equipment/maintenance shop. Refer to site plans Figure 18-1, Figure 18-2, and Figure 18-3.

18.1.2 *Warehouse, Office, and Dry Complex*

The main warehouse, office, and dry complex will be located near the mill building. The office section will accommodate administrative staff and warehouse personnel. The dry complex will accommodate all process plant personnel. A similar warehouse, office, and dry complex will be located at the existing Mária Mine site.

Figure 18-1 Overall Site Layout

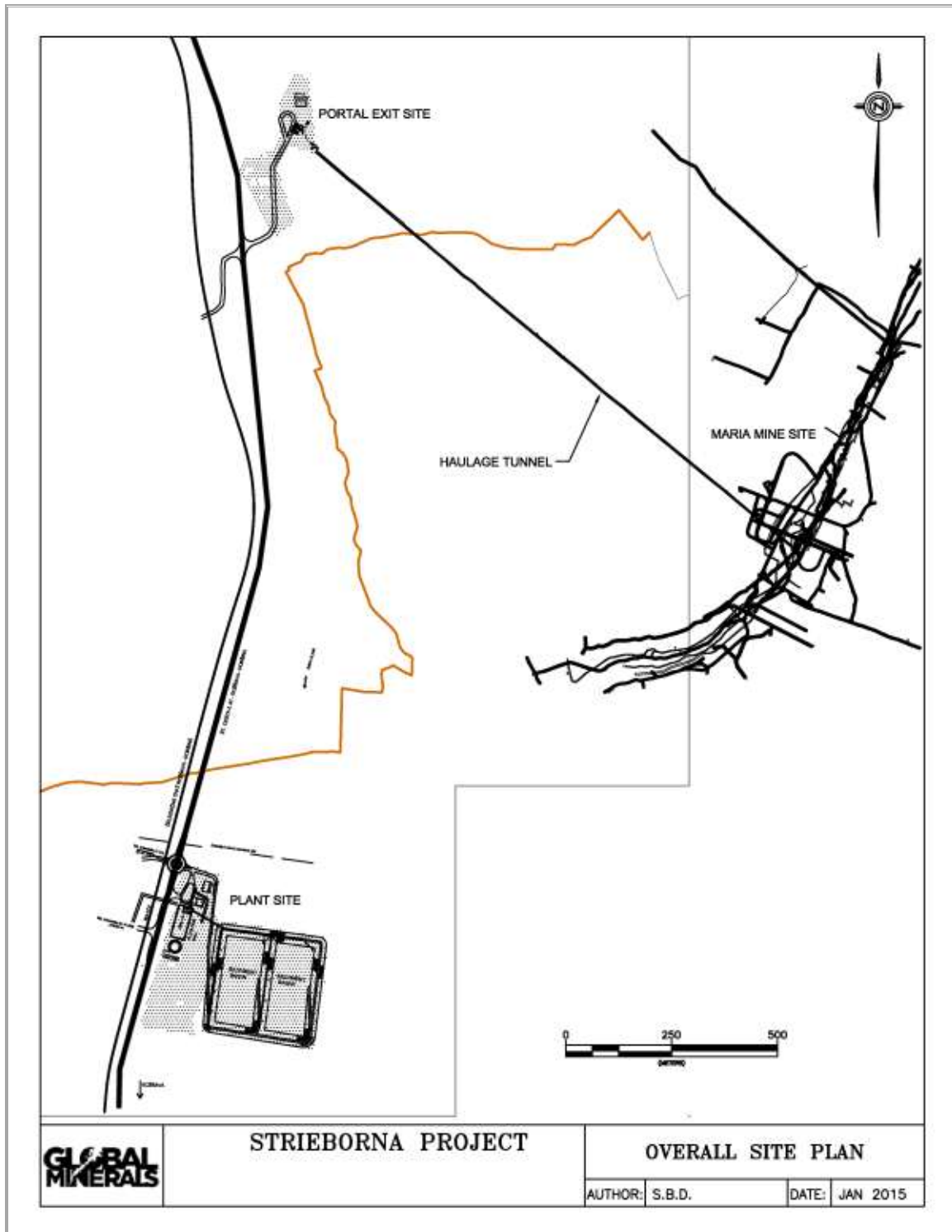


Figure 18-2 Mine Entrance

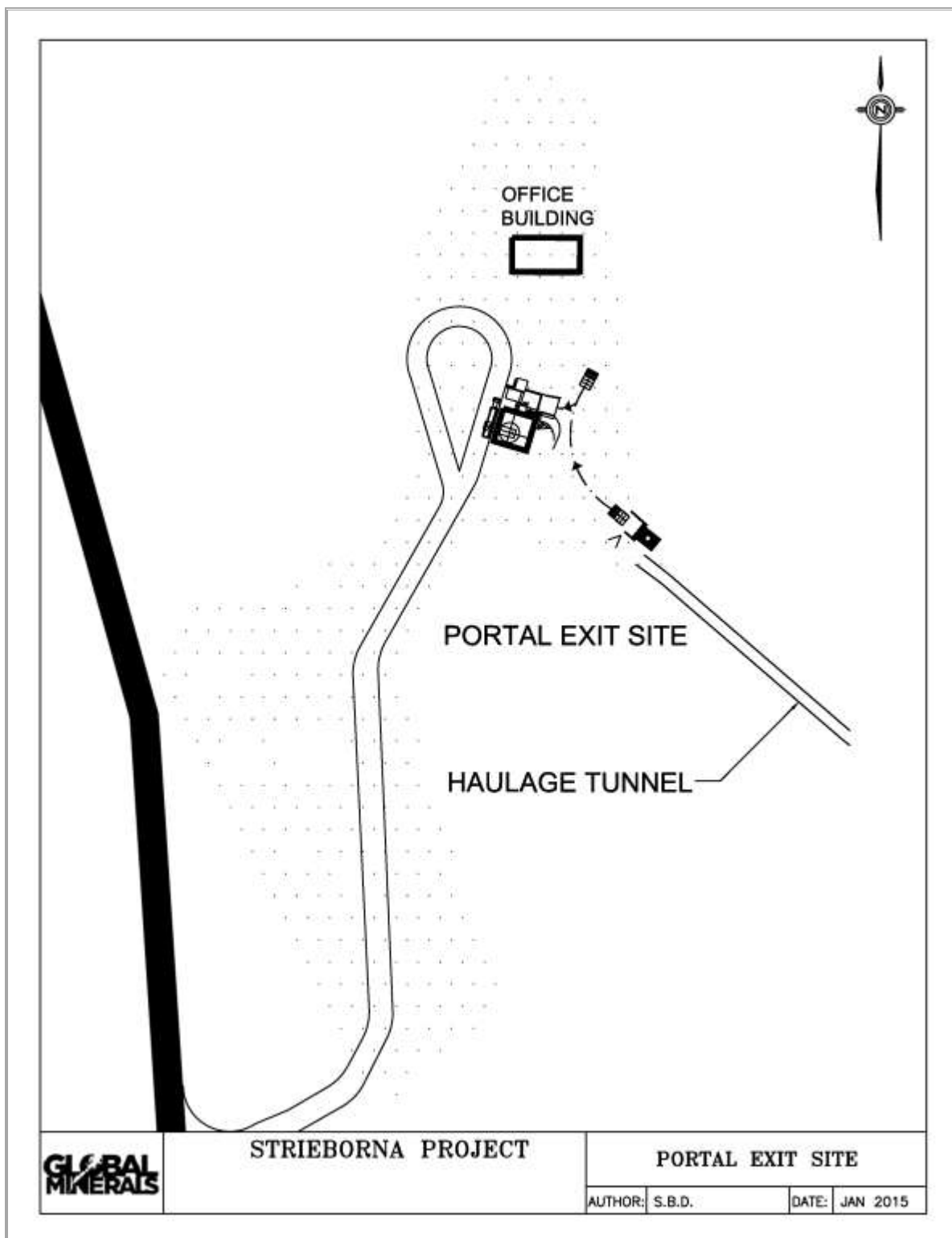
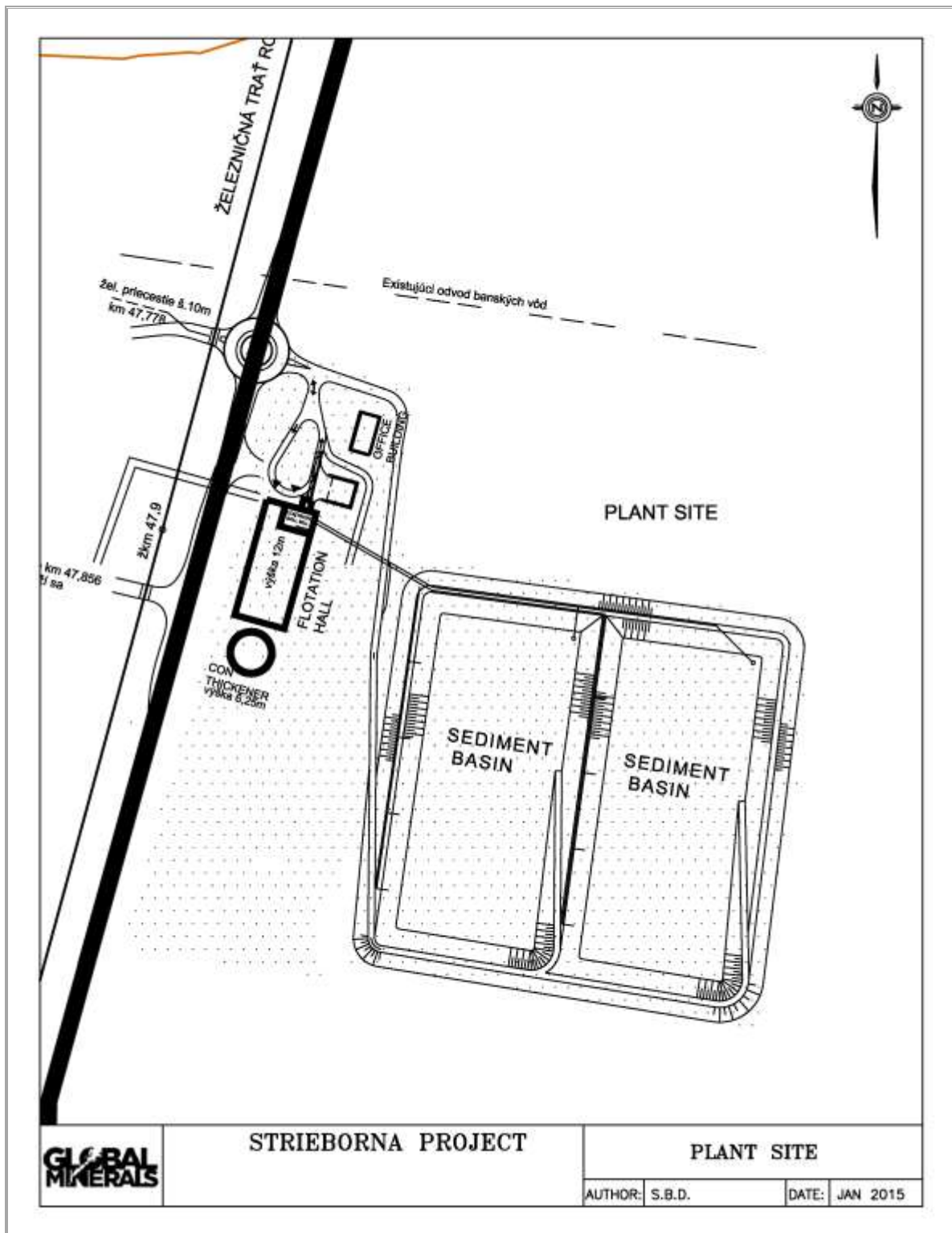


Figure 18-3 Plant Site



18.1.3 Security

The project footprint is in the existing town of Rožňava, Slovakia, within a 1 km radius, and will require three distinct security checkpoints at the process plant, haulage tunnel, and at Mária Mine. Security will consist of managing access control utilising electronic surveillance, physical and visual barriers, fencing, and lighting.

Security fencing will enclose the plant site, the mine site area, and haulage tunnel area.

18.1.4 Fuel Storage and Handling

The project fuel consumption will be primarily associated with the underground equipment. Storage on surface will be sufficient for 3 to 4 days to minimize traffic disturbance to the local populace. Fuel will be brought underground to the equipment as required with mine equipment. The fuel storage tank area will be equipped with a secondary containment.

18.1.5 Roads

The roads connecting the areas of the project will be a combination of existing municipal roads and existing site access roads, which will be upgraded.

Existing Mária Mine Site

The existing access road is in adequate condition and will not require any significant upgrade.

Haulage Tunnel Exit

The existing access road is approximately 600 m long and will require upgrading, consisting of clearing, grubbing, and re-grading.

Process Plant / Tailings Facility

Access to the primary crusher and process plant will be from a paved municipal road and will require improvements to the plant entrance to control ore haulage trucks. Also, approximately 1 km of road to access the TSF from the process plant will require upgrading.

18.1.6 Water Balance System

Approximately 110 m³/d of fresh water will be required to satisfy water demand for the process plant. Filtrate from the tailings and concentrate will be recycled to the process water tank, with make-up fresh water pumped from the Slaná River, approximately 400 m east of the process plant.

18.1.7 Service / Potable Water

Water required for all the site services including potable water will be provided by the existing municipal water system.

18.1.8 Sewage Treatment

All sewage and waste water treatment will be discharged to the existing municipal sewage system.

18.1.9 Refuse Management

A recycling policy to minimise both industrial and domestic refuse will be implemented. Recyclable products will be stored at an appropriate site storage facility and removed on a regular basis. Waste products will be transported to a suitable landfill on site.

18.1.10 Power Supply and Distribution

Additional power distribution equipment will be installed on surface in the areas of the portal and the mill. Unit substations, each with a new MCC, will be installed and commissioned to supplement the existing electrical infrastructure in these locations.

At the surface area of the underground mine, additional switchgear will be added to accommodate the new mine air compressors. Underground, two additional mine portable substations (MPS) will be added as well as cabling infrastructure to connect and distribute to and from these new substations, to accommodate the forecasted additional mining loads in the underground mine.

18.1.11 Communications

There is sufficient existing infrastructure to satisfy the communication requirements for the project.

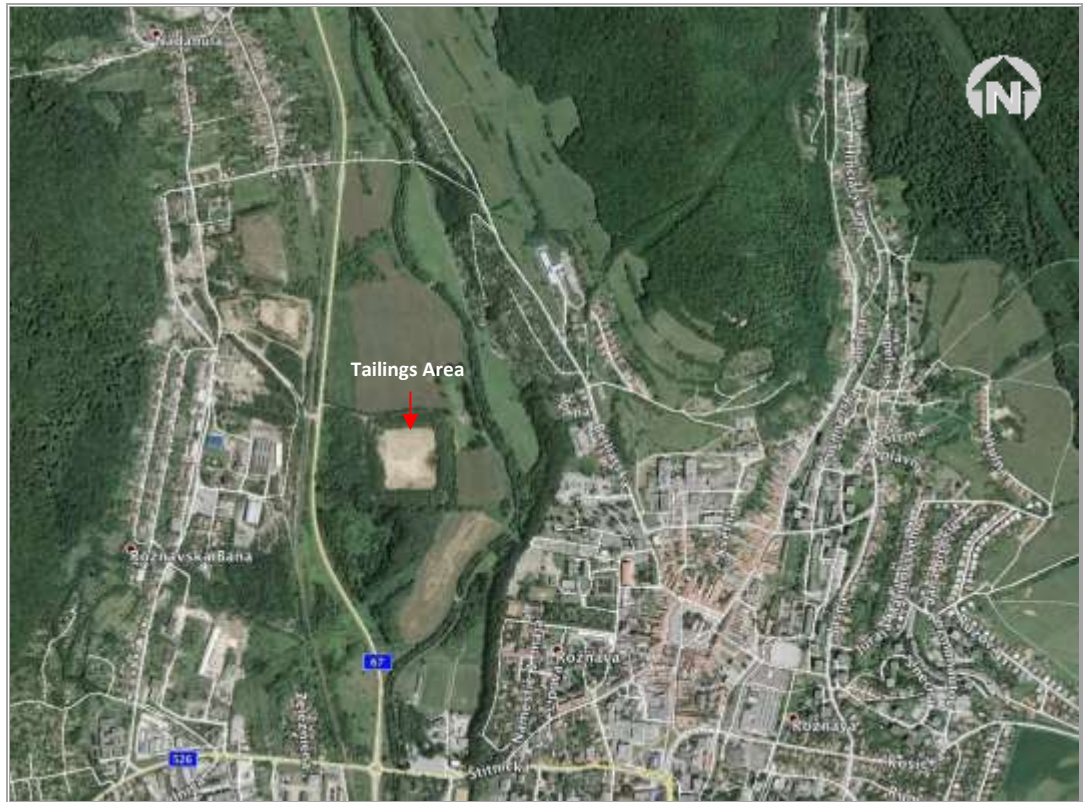
18.2 Tailings Storage Facility

18.2.1 General

The process plant and TSF is located approximately 2 km south of the mine surface infrastructure. The principal objective of the TSF is to provide secure containment of all the tailing solids generated by the milling process and to capture any run off water (whether contained within the cake or seepage arising from precipitation events) .

The TSF is an existing facility located to the east of Hwy. 67 and illustrated in Figure 18-4. This facility currently stands approximately 14 m above surrounding land and will be upgraded for the new operation via the addition of a plastic liner which will be fitted over top of a section of the current workings. The facility was geo-surveyed in 2012 by a Slovakian Engineering company (GEO Slovakia, s.r.o.) who gave an indicative assessment of dam stability. The GEO report assessed the dam to be in stable condition and given the relatively small volume of cake to be added to the facility, AGP has assumed that it is suitable for the expansion.

Figure 18-4 Tailings Facility Location



18.2.2 Design Basis

The new facility will be required to accommodate only 350,000 m to 400,000 m³ of tailing cake over the 7-year mine life. Fresh tailing cake from the new processing facility will be delivered from the plant in 20-tonne trucks and deposited onto the liner using a small loader. Seepage and runoff water collected by the plastic liner will be collected in a small containment sump and a pump set and pipeline will return water to the plant. The volume of return water is expected to be minimal.

Existing drainage channels will divert storm water around the TSF so as to avoid potential contamination. Heavy vegetation in and around the facility will prevent erosion.

19 MARKET STUDIES AND CONTRACTS

19.1 Introduction

The Strieborná Project will produce 40,600 dmt of flotation concentrate over the seven year mine life. For the PEA, AGP has used the metallurgical testwork described in Section 13 to determine the characteristics of this product, and has based all market-related assumptions and calculations on this data.

The flotation concentrate is a copper-silver concentrate that could, in theory, be marketed successfully to copper or silver smelters. However, the presence of deleterious elements (mercury, antimony, and arsenic) effectively means that some further downstream processing is required prior to sale to smelters.

The selected downstream process is a low-temperature (650°C) roasting process, using existing equipment in a German recycling plant. The roasting process reduces the mercury concentrations to levels suitable for smelters (20 ppm or less).

The anticipated flotation concentrate and roaster calcine specifications are discussed below, together with details of the anticipated flowsheet for concentrate handling, transportation, and sales.

19.2 Concentrate Specifications

Flotation concentrate is produced in the Rožňava process plant as a filter cake at a rate of approximately 20 t/d.

For financial modelling and smelter sales assumptions, AGP has fixed the flotation concentrate copper grade at 27.8%. Testwork to date suggests that this is readily achievable at very high copper and silver flotation recoveries and indications point to higher grades being possible through the addition of concentrate regrinding equipment if required by smelters.

At the 27.8% Cu grade operating point and a LOM average headgrade of 1.34% Cu, the flotation mass recovery averages 4.13%. Silver grades in concentrate vary slightly by year (in line with head grade ratios) in the range of 6,700 g/t to 7,500 g/t. Antimony grades are closely linked to copper grades, due to the mineralogical association in tetrahedrite. At the estimated copper grade point, average antimony content is estimated to be 20.2%.

As metallurgical programs and mine production schedules progress in line with project development, the metallurgical recovery model should be refined to include head grade vs recovery (or similar) performance models.

Metallurgical testwork at SGS-UK and Bluecoast Research laboratories is described in Section 13 and has generated flotation concentrates with specifications as detailed in Table 13-12 and Table 13-17.

Table 19-1: ICP Scan of Concentrate – SGS-UK Work

Element	Assay (ppm)	Element	Assay (ppm)	Element	Assay (ppm)
Ag	0.77	In	<1	Sc	<1
Al	112	K	204	Se	<1
As	6,608	La	<1	Sn	88
B	12	Lu	<1	Sr	3
Be	<1	Mg	2,289	Ta	<1
Bi	2,474	Mn	1,789	Tb	<1
Ca	1,189	Mo	12	Th	<1
Cd	32	Na	493	Ti	2
Ce	<1	Nb	<1	Tl	<1
Co	97	Ni	331	U	<1
Cr	106	P	436	V	2
Cu	246,000	Pb	1,999	W	85
Fe	86,190	Rb	<1	Yb	1
Ga	<1	S	265,000	Zn	4,896
Hf	<1	Sb	192,000	Zr	133
Hg	7,200				

Table 19-2: ICP Scan of LCT Concentrate – Bluecoast Phase 1

Element	Unit	Method	Assay	Element	Unit	Method	Assay
Ag	ppm	4A-AA	6,488	Mg	%	ICP	0.23
Al	%	Fus-XRF	<0.01	Mn	ppm	ICP	2,500
As	ppm	Fus-XRF	16,300	Mo	ppm	ICP	2,504
B	ppm	ICP	41	Ni	ppm	ICP	860
Ba	ppm	ICP	<10	P	ppm	ICP	20
Be	ppm	ICP	<0.5	Pb	ppm	ICP	220
Bi	ppm	ICP	3,098	S	%	Leco	24.38
Ca	%	Fus-XRF	0.05	Sb	%	4A-AA	20.4
Cd	ppm	ICP	26.1	Sr	ppm	ICP	<1
Co	ppm	ICP	250	Si	%	Fus-XRF	0.3
Cr	ppm	ICP	<1	Ti	%	ICP	<0.01
Cu	%	4A-AA	26.47	Tl	ppm	ICP	<10
Fe	%	Fus-XRF	15.17	V	ppm	ICP	5
Ga	ppm	ICP	16	W	ppm	ICP	<10
Hg	%	Cold fusion	0.96	Zn	ppm	ICP	6,300
K	%	ICP	<0.01	Zr	ppm	ICP	<5

Penalty elements include antimony, mercury, and arsenic. The penalties are discussed in Section 19.3.3. For the PEA financial analysis, AGP has assumed a fixed mercury concentration of 7,000 ppm and a fixed arsenic concentration of 10,000 ppm.

19.3 Concentrate Processing

19.3.1 General

With contained copper and silver valued at approximately \$6,000/dmt, the flotation concentrate cake is transported to the roasting facility in Germany by road in 1-tonne bags with an assumed moisture content of approximately 10%.

Concentrate production of 12 dmt/d is calculated for Years 1–2 (corresponding to the initial 300 t/d processing rate). From Year 2 onwards, the processing rate increases to 500 t/d and concentrate production increases to 20 dmt/d.

Concentrate transportation costs have been calculated using budget quotes from European trucking companies. Shipment of concentrate at 10% moisture from Rožňava to Dorsten plus shipment of calcine at 5% moisture from Dorsten to Rotterdam has been calculated to cost \$73.80/wmt. Transit losses (Rožňava to smelter) of 0.5% mass of copper and silver are assumed.

19.3.2 Roasting

In 2013, Global contacted a recycling company named DELA Recycling Solutions, located in Dorsten, Germany. DELA, now operating under the name of NOR, operates a commercial rotary kiln for thermal treatment of custom feed materials. It was judged that the facility has the potential to treat flotation concentrate from Strieborná and recover the contained mercury as a synthetic cinnabar product for disposal.

DELA conducted a successful program of laboratory scale testwork at their facility in 2013 and afterwards reported that “after the first test runs DELA GmbH is convinced that treatment of the material in industrial scale can be conducted successfully (Anon, 2013). Initial discussions between Global and DELA management indicated a range of fees for the roasting service. For the PEA, AGP has used an all-in processing fee of \$520/dmt tonne of concentrate.

The roasting process affects both the mass of concentrate and the chemical composition. The product is referred to as a calcine. The differences between concentrate and calcine are summarized in Table 19-3. Losses of copper and silver are assumed to be negligible through the roast.

Table 19-3: Concentrate vs. Calcine Specifications

	Unit	Concentrate	Calcine
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Mass	t	1	0.88
Copper Grade	%	27.8	31.8
Antimony Grade	%	20.2	20.2
Mercury Grade	ppm	7,000	20
Arsenic Grade	ppm	10,000	10,000

As the calcine moisture content is essentially 0%, a small quantity of water would be introduced prior to shipping to prevent dust losses and to improve handling characteristics. At 5% moisture, the calcine would be sampled and packed into 1-tonne bags for shipping to European or Asian smelters.

Transportation by road is assumed to Rotterdam then by ocean to smelters in northern Europe or Asia.

Port handling and shipping costs have been calculated at \$43.0/wmt of calcine.

19.3.3 Smelter Terms

A variety of project-specific term sheets and public domain data was reviewed to arrive at a reasonable estimate of charges payable on the Strieborná calcine product. The inclusion of 20% Sb in this product does limit the scope of sales to a handful of smelters or metals traders.

Table 19-4 shows the assumptions used in this PEA.

Table 19-4: Treatment Charges and Penalties

Item	Unit	Terms
Smelter Treatment Charge	\$/wmt	120
Refining Charge	\$/lb Cu	0.12
Refining Charge	\$/oz Ag	0.60
Price Participation	\$/lb	0.25 per lb over 2.00
Copper Payable	%	96.0
Silver Payable	%	95.0
Antimony Penalty		\$5.00 per 0.1% above 0.1%
Arsenic Penalty		\$11.00 per 0.1% above 0.1%
Mercury Penalty		\$3.00 per 1 ppm above 20 ppm

For the Strieborná calcine product, these terms result in a penalty of \$1,103/dmt, with \$1,004 of that being the antimony penalty.

With over \$1,000/t in penalties predicted for antimony alone, the importance of antimony removal steps in the Strieborná metallurgical flowsheet are readily apparent.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

This section of the report covers the principal environmental aspects of the Strieborná Project, including discussions of physical, chemical, and biological components, with a brief summary of the principal ecosystem services involved. A full environmental and social impact assessment is under preparation at the time of writing this summary report.

20.1 Environmental

20.1.1 *Physical*

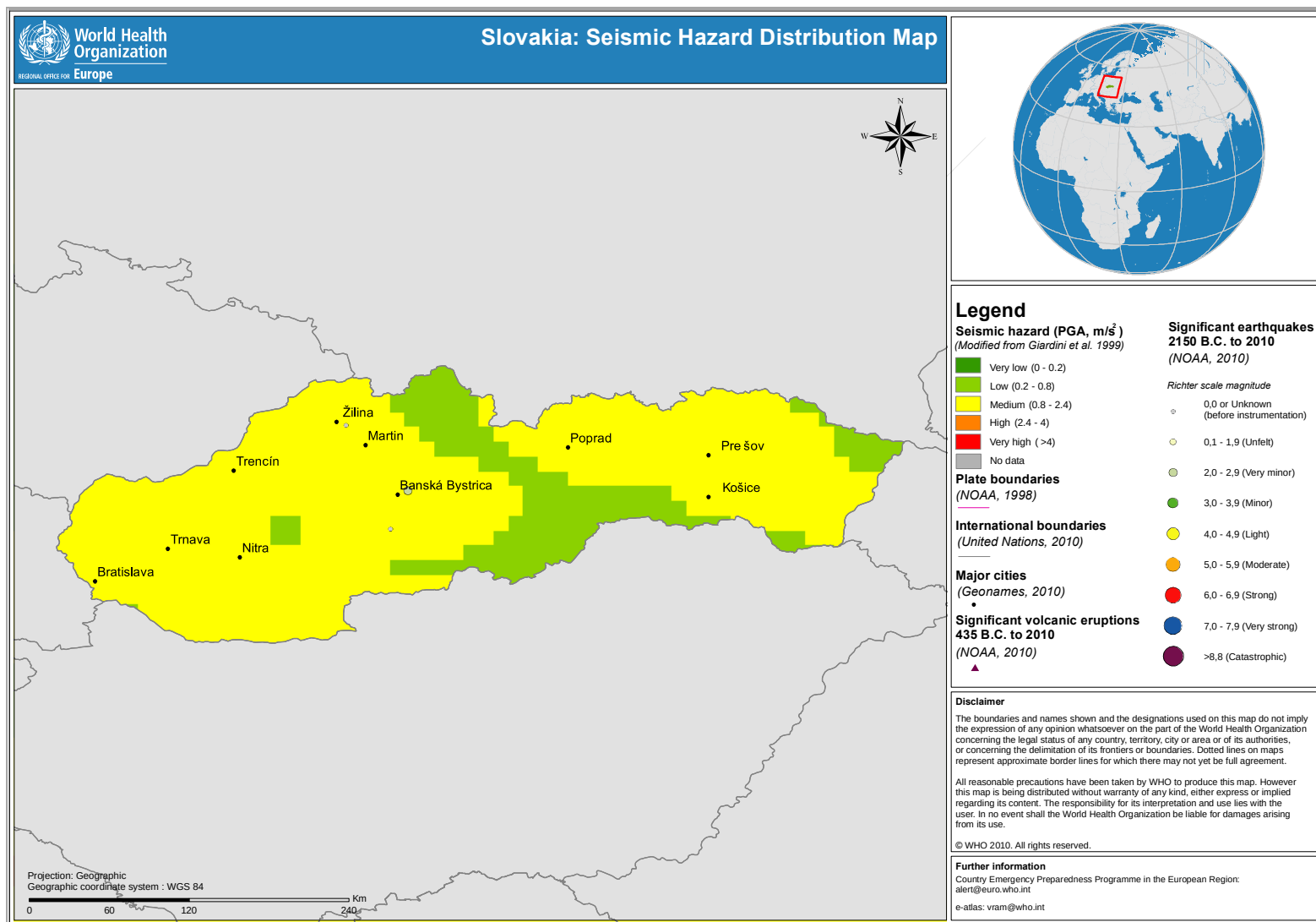
A detailed project description has already been provided, with location, geology, and general geomorphology. Figure 20-1 shows the seismic status of the area and indicates a low to very low ranking for the project site.

The regulatory environment and status of the environmental approvals and permitting of the project have been summarized previously, including the status of the local Environmental Impact Statement (EIS) process.

The Slovak Republic lies in the mild zone of sub-continental climate, with distinctive seasonal changes and microclimate depending on elevation. The low ground of the Slaná River Valley and its tributaries has a mean annual temperature of approximately 8°C, with monthly average temperatures ranging from approximately 0°C in January to 20°C in July, and annual rainfall of approximately 700 mm. The climate is not a factor that would restrict the length of the operating season for the mine (Jakubiark, Z., 2008).

The Strieborná Project exists in an area of Slovakia that has been actively mined since the early 13th century. The area is characterized by rolling hills with highly dissected valleys that support surface water streams and rivers. The mine property is located approximately 2 km north of the town of Rožňava (Figure 1-1). The town of Rožňava occupies a low spur ridge between the Slaná River (the broad valley on the left centre of the image) and its tributary Rožňavský Potok (Drazus Creek), and is flanked on the west and northeast by forested ridges Tvádova-Turecká (954 m) and Rákoš (800 m), respectively. Two high points mark the low, north-trending ridge that separates the valley of the Slaná River from Rožňavský Potok (Drazus Creek): Kalvária at 478 m, and Tri Vrchy at 581 m.

Figure 20-1: Seismic Rating Map of Slovakia



20.1.2 Hydrology and Geohydrology

The hydrology of the Slaná River and its tributaries has been documented by the Slovakian government (and precursor governments) since the early 1920s. Gauging station #7690 at Rožňava (hydrological number 1-4-31-01-031-01), located at River KM 51.9, and with a drainage area of 301.53 km², has a monitoring record dating back to 1921. There is an automated gauging station at Rožňava, which reports online with stage height and temperature data on a continuous basis (http://www.shmu.sk/sk/?page=1&id=hydro_vod_all&station_id=7690).

Flow on 12 July 2013 at 06:00 hours was 5.08 m³/sec at an average temperature of 13°C. It has been reported that the Slaná River receives much of the flow from the Slovak Karst area of the country. Table 20-1 provides two yearly summaries of average monthly Slaná River flows at Rožňava in cubic metres per second for 2009 and 2010.

Table 20-1: Slaná River Flows (m³/sec)

Year	Nov.	Dec.	Jan.	Feb.	Mar.	Apr.	May	Jun.	Jul.	Aug.	Sep.	Oct.	Avg.
2009	2.57	9.87	4.74	5.45	5.99	6.10	2.59	3.31	5.08	2.50	2.35	20.30	4.38
2010	5.67	6.53	11.06	4.50	5.39	12.20	15.20	19.00	4.17	5.21	6.69	4.88	8.38

Two major creeks drain the Strieborná mine property: the Rožňavský Potok (known locally as Drazus Creek) and the Cucmiansky Creek (Potok). Discharge of treated water from the dewatering process from the water treatment facility flows into the Slaná River, with secondary discharge into Drazus Creek. These discharges are approved, permitted, and monitored.

20.1.3 Chemical

Water Quality

The principal chemical concern of the project is the dewatering of the flooded mine workings, which has been ongoing since January 2012. The modern water treatment facility treating water from the mine dewatering activities is fully permitted, inspected, and monitored. Mine water discharge is permitted, and is monitored on a daily basis. The water is transported to the surface by submersible pumps and then treated in a modern water treatment facility, which utilizes chemical flocculation and pH adjustment (Figure 20-2). This water treatment system is computerized, self-adjusting, and continuously monitored. The water treatment volumes (flow rates) have varied between a few liters per second (L/sec) to over 70 L/sec. It has been estimated the total average inflow to the mine is in the order of 4 L/sec to 12 L/sec, with a maximum of about 20 L/sec.

Figure 20-2: Strieborná Project Water Treatment Facility

Sludge that is generated by the water treatment facility is disposed of in a monitored, third-party TSF; monitoring includes groundwater quality down gradient of the facility. The treated water discharged to the local Slaná River drainage is monitored daily, and has met Slovakian water quality standards (Figure 20-3 and Figure 20-4). Although some of the early, upper level waters pumped from the mine indicated a slightly depressed pH, there has been no indication of any latent acid rock drainage (ARD) being generated in the lower mine workings. The water treatment plant has been operated to adjust final pH to meet Slovakian water quality standards prior to discharge. A summary of water quality discharge monitoring results is documented and reported to the authorities on a regular basis.

Figure 20-4: Aerial View of Strieborná Project, Slaná River Area



Air Quality

Air quality considerations principally focus on two particular areas: fixed and mobile sources, and fugitive dust. The amount of air pollution (CO₂) anticipated to be generated by the various diesel engines used in the mining process, including crushing, is estimated to be approximately 1,800 t/a to 2,600 t/a of CO₂ based on estimated annual diesel fuel use (based on 2,681 g/L of CO₂ of diesel combusted). Extensive use of electric motors is planned, with electricity supplied from the national grid, mostly produced by nuclear power (55%), as well as coal burning power plants (18.7%), hydro power (16.0%), and natural gas (5.8%). Fugitive dust may be generated at various locations, including roads, crushers, tailings management facilities, conveyors, and stockpiles. A detailed air quality-monitoring program will be implemented to identify compliance with current air quality standards, and appropriate technologies to be applied to meet these standards.

The greenhouse gas (GHG) emissions target for Slovakia is 66.3 Mt/a of CO₂ equivalent. The general trend has been steady for Slovakia for the past several years. The anticipated contribution of the Strieborná Project to the GHG target is anticipated to be negligible, less than 0.00027% to 0.00039% of the annual target.

20.1.4 *Biological*

The local ecological biome can be characterized as secondary regrowth of beech and oak forests with some riparian habitats near the streams and rivers that bisect the area. There are no national parks, nature reserves, or other protected areas in the immediate vicinity of the project. Riparian and associated wetland habitats may represent the most sensitive areas of the region. There are no lakes, reservoirs, or ponds on the property. There are no known rare, threatened, or endangered plant or animal species occurring in the project area. Over the years, hunting, development, and agriculturalization have displaced many wildlife species. Some local recreational fishing occurs in the project area, in the Slaná River and its tributaries. There is a small fish hatchery below the mine area, which will not be impacted by the mining operations, as it takes its water from Drazus Creek above the mine site. The project will have no significant impact on the biodiversity of the area, and, as such, the project does not anticipate a biodiversity action plan being required. As an underground mining development, the amount of surface disturbance will be small (~10 ha).

20.1.5 *Ecosystem Services*

The term ecosystem services refers to ecological factors and functions that benefit the living world, such as water quality protection, air quality protection, habitat maintenance, carbon capture, drainage, and many other factors. The Millennium Ecosystem Assessment (MA) report 2005 defines ecosystem services as benefits people obtain from ecosystems and distinguishes four categories of ecosystem services, where the so-called supporting services are regarded as the basis for the services of the other three categories.

The following list presents definitions and examples of each service according to the MA:

Supporting Services – ecosystem services “that are necessary for the production of all other ecosystem services”

- nutrient dispersal and cycling
- seed dispersal
- primary production.

Provisioning Services – “products obtained from ecosystems”

- food (including seafood and game), crops, wild foods, and spices
- water
- minerals (including diatomite)
- pharmaceuticals, biochemicals, and industrial products
- energy (hydropower, biomass fuels).

Regulating Services – “benefits obtained from the regulation of ecosystem processes”

- carbon sequestration and climate regulation
- waste decomposition and detoxification
- purification of water and air
- crop pollination
- pest and disease control.

Cultural Services – “nonmaterial benefits people obtain from ecosystems through spiritual enrichment, cognitive development, reflection, recreation, and aesthetic experiences”

- cultural, intellectual, and spiritual inspiration
- recreational experiences (including ecotourism)
- scientific discovery.

Some of the ecosystem services that are recognized in the project area include the following:

- **Vegetative Cover** – primary vegetative production, seed production, prevention of erosion, filtering runoff, holding soil in place, providing shade, cover, habitat, and carbon capture and sequestration.
- **Geological Processes** – absorption of rainfall and runoff, water filtration, recharging aquifers, draining runoff, and providing input to local streams for aquatic habitat maintenance. The minerals produced from the mine can be considered as a type of provisioning service.

- **Cultural and Recreational** – outdoor recreation and religious/cultural resources. Karst topography is a tourist attraction, but is located in the national park to the south.

20.2 Social

This section of the report covers the principal social, cultural, archaeological, anthropological, and economic aspects of the Strieborná Project, including discussions of the local communities affected, demographics, land use, livelihoods, employment, taxation, expectations, sustainability, and other sociological and economic aspects of the project area. Visual, noise, and aesthetic aspects are also considered.

The property is readily accessible by the paved road linking Rožňava with the village of Čučma. Rožňava is a county town with almost 20,000 inhabitants, linked by good roads to Košice and Poprad. The distance to Košice along Road E571 is 70 km. Košice is the capital of the Košice Self-Governing Region, and the second largest city in Slovakia after Bratislava. The town of Rožňava is situated on the railway line linking Bratislava with Košice, which operates normal and fast passenger trains as well as freight trains. The nearest international airport is Košice Airport, situated 14 km west of Košice. The companies offering regular flights to and from Košice are Czech Airlines, Tyrolean Airways, SkyEurope Airlines, and Slovak Airlines; during the summer vacation season, flights are also offered by ATE, Spanair, Tunis Air, Onur Air, Air Slovakia, and Hemus Air. Other international airports within close proximity are those in Bratislava, Vienna, and Budapest (Jakubiak, Z., 2008).

The 2001 census data for Košice showed a total of 19,261 inhabitants, 69.3% Slovak, 26.8% Hungarian, 1.6% Roma, and 0.7% Czech. The religious makeup was 41.1% Catholic, 32.4% none, 12% Lutheran, and 1.3% Greek Orthodox. There are no classic “indigenous peoples.” The most vulnerable social group is likely to be the Roma population (often referred to as “gypsies”), who often lack formal education, exhibit poor health, have high unemployment, and generally poor living conditions.

Community interaction and support has been very positive throughout the process of bringing the Strieborná mine back into production. The project is overwhelmingly viewed as positive from the perspective of most local community members and elected officials, as mining has been a dominant part of the local socioeconomic structure since the early 13th century. Regular meetings and specific presentations have been made to local community officials and organizations. Tours of the facilities, including the advanced water treatment plant, have also been conducted (Figure 20-5).

Employment is the principal concern of the local communities.

Figure 20-5: Tour of Mine with Local Officials



20.2.1 Health, Safety, and Security

The mining project will be operated using current health, safety, and security procedures, which are in accordance with EU policies and procedures. These procedures include communication from the Commission to the European Parliament, the Council, the European Economic and Social Committee, and the Committee of Regions on the practical implementation of the provisions of the Health and Safety at Work Directives 89/391 (Framework), 89/654 (Workplaces), 89/655 (Work Equipment), 89/656 (Personal Protective Equipment), 90/269 (Manual Handling of Loads), and 90/270 (Display Screen Equipment).

Modern personal protective equipment will be required of all personnel in safety-controlled areas of the mine, especially underground (eye and hearing protection, hardhats, steel-toed shoes, harnesses, etc.). Training programs will include all staff, including miners, supporting contractors, vendors, and underground staff, and will be conducted according to EU health and safety training requirements.

The Strieborná Project will employ approximately 100 miners, 10 managers, 5 administrators, and 30 other employees. Secondary benefits to the local communities can be generally estimated using a multiplier of annual payroll (of three to over five times) to account for other business activities stimulated and supported by the mining project.

Potential impacts to the local communities in the areas of noise, traffic, and fugitive dust have been considered, and the project has been designed to eliminate or minimize these impacts. Use of the underground haulage tunnel is a major component of the project that will be operated to eliminate impacts to the local community. The haul distance of mineralized rock from the tunnel terminus to the plant is approximately 2 km, on a road that normally carries traffic approximately 30 times the two trucks per hour estimated traffic for the project.

In addition to the tax revenue it will generate, the Strieborná Project will also support the local communities directly with specific community development programs, including support for education, health, and social well-being.

As outlined in Table 20-2, the project has also engaged in community development activities over the past three years, contributing approximately €50,000.

Table 20-2: Summary of Community Support Activities

No.	Activity
1	M-Zone (youth free-time centre), run by the City of Rožňava; aimed at providing city youth with a safe location to gather and participate in various activities (TV, video games, ping-pong) to keep the city youth away from substance abuse and other negative behaviour
2	Annual financial support for various youth soccer and basketball teams
3	Sponsored building of a historical "Iron Route" in and around Čučma, including informational billboards, and structures along the route (in conjunction with the village of Čučma)
4	Provided financial sponsorship for a local fishery to purchase young fish to release into the Slaná river, and helped with cleanup of a section of the river
5	Sponsored local hunting association's public show of trophies
6	Provided financial sponsorship to the City of Rožňava for youth sports

There are no known cultural or archaeological sites within the mining area. There are several churches, cemeteries, and other historical buildings located in the nearby town of Rožňava, which will not be affected by the operations. Rožňava has been a mining town since at least the 13th century, and has a mining museum and an old mint founded in the 17th century, both of which serve as tourist attractions.

20.2.2 Stakeholders

Stakeholder mapping of the project area has indicated the following major and minor social groups:

- general public
- residents and land owners
- community organizations
- government organizations
- non-governmental organizations
- agricultural organizations

- students.

Stakeholders may also include:

- employees
- contractors
- industry or business associations
- environmental groups
- local councils
- government departments
- public and private schools / school systems
- colleges, universities, and trade schools
- youth groups
- senior citizens' groups
- politicians.

20.2.3 *Visual, Noise, and Aesthetics*

The mine is not expected to pose any significant visual impacts to the local community. Noise has been discussed previously in Section 20.2.1. The aesthetic nature of the local community has been dominated by mining culture since the 13th century, and certain functional elements of the mining project will tend to enhance this aspect of the local culture.

20.2.4 *Land Use*

The project area has been used for mining for hundreds of years. The surrounding area is also characterized by mixed use grazing, fruit orchards, and limited crop agriculture. Abandoned, undeveloped, and marginal lands occur throughout the area, adjacent to the roadways, railways, farms, residential, and industrial areas. Rezoning of one small area (less than two hectares) of the project was approved, and was the only land use change required.

The nearest national park is the Slovak Karst National Park (Národný Park Slovenský Kras), a national park in the Slovak Karst (Slovenský Kras) mountain range in Southeast Slovakia. It lies in the Gelnica, Rožňava, and Košice–Okolie districts in the Košice region, approximately 12 km south of the project area. The national park covers an area of 346.11 km² (133.64 mi²), and its buffer zone covers 117.41 km² (45.33 mi²). The park was established on 1 March 2002, after being a protected landscape area since 1973. Slovak Karst is the first Slovak biosphere reserve created since 1 March 1977, when it was included in the UNESCO Man and the Biosphere Programme. In 1995, 12 out of 700 caves in the park were recognized as a UNESCO World Heritage Sites. This park is located

approximately 10 km to the south of the project, and south of the town of Jovice. Rožňava is considered a principle rallying point for tourist trips into the park.

20.2.5 *Agriculture*

The principal crops of this region of Slovakia include corn, winter wheat, oats, barley, potato, canola, apples, various fruits, selected hops, and grape species (viniculture). Annual and perennial fodder, permanent pastures, and certain legumes are also produced. The mining project will not adversely affect any of the agricultural values of the area. There is no cultivated agriculture occurring on any of the mine property. The ultimate reclamation and closure of the TSF will likely increase the amount of permanent pasture or fodder production. More advanced agricultural uses may be considered for the closure phase.

20.2.6 *Waste Management / Chemical Storage*

The waste management plan for the project is well developed, and includes contracting a licensed waste disposal company that operates throughout the region. The TSF is a licensed operational project component, and will be monitored according to Slovakian environmental regulations. The sludge from the water treatment facility, which contains iron and manganese precipitates, will also be deposited in the TSF. Recycling of metal, steel, and possibly other materials will be incorporated into the operations plan. Safety measures for storage of fuel, sodium hydroxide, carbon dioxide, flocculant, and other operation supplies will be performed in accordance with current Slovakian and EU standards. Sewage treatment will be accomplished using a connection to the local municipal system.

20.2.7 *Reclamation and Closure*

The project is anticipated to be in production for approximately 10 years based on current resources. Continued exploration is expected to replace exploited resources. A detailed reclamation and closure plan is under development that will cover the mine, all mine facilities, the crusher, and the TSF. A total of approximately 10 ha will be subject to direct revegetation, including the TSF.

Reclamation will be accomplished using selected native plant and seed admixtures that reflect the local ecology. Final grading, contouring, and erosion and sediment control will be incorporated into the overall reclamation and closure plan development. Steps will be taken to avoid introduction of invasive species, noxious weeds, or other undesirable species. There will be no collecting, hunting, or fishing allowed on the property.

20.2.8 *Sustainability*

The Strieborná Project will follow recommendations of the EU Raw Materials 2020 Initiative, and general sustainability guidelines in all areas of its operations. The fact that

modern technology is being used in an environmentally and socially advanced way to rework an existing mining facility is a tribute to the application of technological advances, innovation, and engineering to an abandoned facility that can still produce valuable minerals and taxable income, significant employment, along with all of the associated community development and support activities.

Ensuring efficient and effective extraction of minerals involves a broad range of sustainability considerations, from energy supply and efficiency, to effective community engagement, social grievance mechanisms, and alignment with requirements of the International Labour Organization (ILO). Sustainability considerations will be made for the overall Strieborná mining project operations. Continuing community relations efforts will bolster broad community support, as already expressed by Rožňava and the surrounding communities. The current project grievance mechanism entails direct communication to the on-site manager, who registers, records, and responds appropriately.

20.3 Summary

The renewal of underground mining at the Strieborná Project using modern technology is expected to bring much needed employment, income, taxation, environmental protection, and social benefits to the local community and to Slovakia in general. Implementation of modern water treatment technology to dewater the flooded underground workings, treat, and discharge the water has been accomplished within the permitting framework of the Slovakian government, and is compliant with EU standards. Global Minerals operates as a valued and participatory corporate citizen in its operations. The physical, chemical, biological, social, and regulatory aspects of the project are being considered under the various applicable EU environmental policies and related Framework documents (http://ec.europa.eu/environment/index_en.htm).

A detailed environmental impact assessment process is required by the Slovakian government, and is under preparation at time of writing this report. Potential negative social and environmental impacts will be identified, and appropriate steps taken to eliminate, minimize, mitigate, or ameliorate these impacts. Appropriate controls for treatment of mine dewatering have already been put in place, and are permitted, monitored, and inspected. Detailed operational planning and engineering for air quality protection, erosion and sediment control, surface water and groundwater quality, waste management, spill prevention control and countermeasures, tailings management, emergency response, community relations, worker health and safety, security, traffic, noise, dust, vibration, and other aspects will be incorporated into the overall SEMMP to be specifically applied to the Strieborná Project. A comprehensive bankable feasibility study to international standards is anticipated. The project is expected to be operated in a safe, environmentally and socially responsible manner, within a modern sustainability framework, using advanced approaches, technology, protocols, and management.

The project has demonstrated broad community support, with a continuing stakeholder engagement program with wide reach, including governmental, non-governmental, social, and economic organizations throughout the local communities.

21 CAPITAL AND OPERATING COSTS

Capital and operating costs for the Strieborná Project have been estimated as part of this PEA. Costs are described below on an area-by-area basis.

21.1 Capital Costs

21.1.1 Summary

The capital costs for the Strieborná Project are summarized in Table 21-1. The costs are based on a 7-year project life with throughput starting at 300 t/d and expanding to 500 t/d early in Year 3.

Table 21-1: Strieborná Project Capital Cost Summary

Capital Category	Total Capital (\$M)	Preproduction Year -3 to Year -1 (\$M)	Production Year 1 (\$M)	Production Year 2+ (\$M)
Underground Mining	29.5	10.6	10.7	8.2
Processing	10.2	5.8	3.9	0.5
Infrastructure	3.7	3.1	0.5	0.1
Environmental	0.2	0.0	0.0	0.2
Indirect Costs	4.3	3.2	0.9	0.2
Contingency	5.8	2.6	2.0	1.2
Total	53.6	25.3	18.0	10.3

Initial direct and indirect capital requirements (preproduction) are estimated at \$25.3 million. Production starts in Year 1 with additional capital requirements of \$18.0 million, with the majority being a result of continued underground development. The capital requirement in Year 1 is offset by revenue generated in the same year. From Year 2 onward, a further \$10.3 million is required, again offset by production revenues.

Contingency allowances are calculated from percentages shown in Table 21-9.

Life of mine (LOM) capital requirements are estimated to be \$53.6 million.

21.1.2 Underground Mining Capital

Methodology used to describe the estimation of the capital costs for the underground mine is described in Section 16.

Table 21-2 shows a summary of the mine preproduction capital estimate.

Table 21-2: Summary of the Preproduction Mine Capital Estimate (\$ '000s)

Cost Area	Cost
-----------	------

Cost Area	Cost
Capital Development	8,585
Operating Costs Transferred to Capital	1,350
Mine Infrastructure	4,131
Total Mine Preproduction Capital	14,065

All development in outside the Strieborná vein in waste was classified as capital development.

By the end of the preproduction capital development programme, ramp access from the Haulage Transfer drift to Levels 6 and 7 will be completed together with ramp access to the sublevels above Level 6. A small mobile equipment workshop will also be commissioned on Level 6.

Operating cost activity during the preproduction period includes approximately 540 m of in-vein development and the commencement of production from the first stope above Level 6. Approximately 43,500 tonnes of mill feed will be stockpiled on surface prior to commissioning of the process plant.

Modelled unit rates (Table 16-13) were applied to the development and production schedule to provide capital and operating direct cost estimates. Mine-wide overhead estimates were divided in proportion of the respective direct cost estimates between the capital development and operating cost estimates. The capital development estimate is summarized in Table 21-3 and the operating costs transferred to capital in Table 21-4.

Table 21-3: Preproduction Capital Development Cost Summary (\$ '000s)

	Total	Year -1				Year 1 Q1
		Q1	Q2	Q3	Q4	
Development	3,137.1	633.0	605.1	575.9	571.9	751.2
Trucking	811.7	149.1	139.7	123.5	154.0	245.5
Mine Services	1,730.4	362.0	349.9	317.3	376.1	325.0
Owner's Supervision and Technical Services	1,393.5	292.1	282.5	263.9	311.7	243.2
Power	1,512.1	324.3	303.5	274.6	361.0	248.7
Total	8,584.9	1,760.5	1,680.7	1,555.2	1,774.8	1,813.6

Table 21-4: Preproduction Operating Costs Transferred to Capital (\$ '000s)

	Total	Year -1				Year 1 Q1
		Q1	Q2	Q3	Q4	
Development	454.4	-	39.7	99.3	-	315.4
Stoping	43.5	-	-	-	-	43.5
Stope Mucking	19.4	-	-	-	-	19.4
Trucking	184.1	-	11.4	27.5	-	145.2
Mine Services	252.3	-	24.0	57.5	-	170.7
Owner's Supervision and Technical Services	195.0	-	19.4	47.8	-	127.7
Power	201.3	-	20.8	49.8	-	130.6
Total	1,350.0	-	115.4	282.0	-	952.6

Mobile Equipment

The owner will lease all mine mobile equipment. The equipment leasing costs are included in the equipment hourly rates. There is therefore no mobile equipment capital cost.

Mine Infrastructure

Mine infrastructure items included in the estimate are described in Section 16. A summary of the preproduction mine infrastructure estimate is provided in Table 21-5.

Table 21-5: Summary of Preproduction Mine Infrastructure (\$ '000s)

	Total	Year -1				Year 1 Q1
		Q1	Q2	Q3	Q4	
Rehabilitation	239.9	114.7	-	-	-	125.2
Underground Infrastructure	1,262.3	411.0	67.2	-	214.7	569.5
Surface Infrastructure	2,628.4	2,359.4	250.0	-	3.0	16.0
Total Mine Infrastructure	4,130.6	2,885.1	317.2	-	217.7	710.7

Sustaining Capital

Sustaining capital comprises capital development, rehabilitation, and mine infrastructure requirements. Sustaining capital for the mine is summarized in Table 21-6.

Table 21-6: Summary of Mine Sustaining Capital (\$ '000s)

	Total	Year 1 Q2-Q4	Year 2	Year 3
Capital Development	11,092	5,448	4,346	1,298
Rehabilitation and Mine Infrastructure	4,380	1,798	2,402	180
Total Sustaining Capital	15,472	7,246	6,747	1,478

21.1.3 Process Plant Capital Cost

The capital cost estimate for the processing plant, based on the proposed flowsheet as described in Section 17, is summarized in Table 21-7. LOM direct costs for the plant are estimated at \$10.2 million with a further \$3.76 million allocated to indirect costs and contingencies. A total of \$500,000 of the direct capital allowance has been deferred for use in Year 2 as part of the expansion from 300 t/d to 500 t/d. No sustaining capital is required for the process plant facility due to its simple nature and relatively short lifespan.

Table 21-7: Summary of Direct Capital Costs for the Strieborná Process Plant

Item	Cost Y-3 to Y1 (\$)	Cost +Y2 (\$)	Cost LOM (\$)
Crushing and Ore Storage	2,062,117	-	2,062,117
Ball Milling	1,374,869	-	1,374,869
Rougher Flotation	885,702	375,000	1,260,702
Cleaner Flotation	547,238	125,000	672,238
Concentrate Dewatering	726,819	-	726,819
Tailings Dewatering	2,128,279	-	2,128,279
Services	828,367	-	828,367
Reagents	550,057	-	550,057
Building	598,994	-	598,994
Direct CAPEX Subtotal	9,702,442	500,000	10,202,442
Indirect Costs	1,904,388	211,599	2,115,986
Contingency	1,479,252	164,361	1,643,613
Total CAPEX	13,086,082	875,960	13,962,041

The process plant capital estimate was compiled using the following methods and assumptions:

- Mechanical Equipment:
 - Estimated costs for major mechanical equipment (i.e., crushers, mill, thickeners, filters, pumps, etc.) were taken from budget quotes and from database information from recent projects.
 - An installation rate for mechanical equipment was factored based on database information from similar projects.
- Civil and Earthworks:
 - An estimate of the cost of civil and earthworks was made by factoring from the mechanical equipment supply and install cost on a plant-wide basis, based on recent AGP projects of similar size and scope.

- Structural:
 - An estimate of structural steel cost was made by factoring from the mechanical equipment supply cost, based on recent projects of similar size and scope.
 - The direct cost includes transportation to site.
 - Erection costs were estimated as a percentage of the structural supply estimate on a plant wide basis.
- Platework:
 - An estimate of platework fabrication cost was made by factoring from the mechanical equipment supply cost, based on recent AGP projects of similar size and scope.
 - The direct cost includes transportation to site.
 - Platework erection costs were factored from the platework supply cost.
- Piping:
 - Piping costs were estimated using database information from other AGP studies.
 - The direct cost includes transportation to site and installation.
- Electrical and Instrumentation:
 - Electrical and instrumentation (E&I) costs were estimated using database information from other AGP studies.
 - The direct cost includes transportation to site and installation.
- Building Costs:
 - Building costs were estimated using database information from other studies.

21.1.4 *Infrastructure Capital Cost*

The infrastructure capital costs estimated for Strieborná are somewhat lower than other projects of a similar size and this is a function of the project's proximity to existing (and useable) infrastructure. Power and water tie-in effort is low and the mill site is placed within an existing industrial area.

The costs are listed in Table 21-8 and are based on information vendor quotations and from work conducted on similar operations.

Table 21-8: Summary of Infrastructure Capital Costs

Item	Cost Y-3 to Y1 (\$)	Cost +Y2 (\$)	Cost LOM (\$)
Electrical			
Substation at portal	275,000	-	275,000
Underground power – substations	1,300,000	-	1,300,000
Plant power – substation	300,000	-	300,000
Site Facilities			
Primary crushing road and security	54,000	-	54,000
Municipal roadway improvements	150,000	-	150,000
Site roads	90,000	-	90,000
Two bay garage and office/shop/ warehouse	270,000	-	270,000
Fuel storage	5,000	-	5,000
Plant mobile equipment	185,000	-	185,000
Administration offices (modular units)	290,000	-	290,000
Mill site fencing	90,000	-	90,000
Communications (general)	50,000	-	50,000
Water Storage/Supply			
Process water intake, pumps, pipeline	180,000	-	180,000
Sewage/water tie in to municipal infrastructure	75,000	-	75,000
Tailings Disposal			
Initial works	250,000	-	250,000
Sustaining costs	9,813	88,483	98,296
Total Direct Capital	3,573,813	88,483	3,662,296
Indirect costs	549,344	-	549,344
Contingency	549,344	-	549,344
Total Capital	4,672,501	88,483	4,760,984

21.1.5 Environmental

Daily environmental monitoring costs are included in the G&A cost category as an operating cost.

Mine closure costs, including removal of infrastructure and reclamation of the mine, mill and tailings sites, have been estimated to be \$150,000.

21.1.6 Owner's Cost

Owner's costs have been estimated to be \$1.64 million, consisting of start-up and construction working capital plus G&A costs in preproduction years.

21.1.7 Contingency Capital

Contingency allowances have been made in the various project areas, generally as a percentage of the direct costs for the particular area. Table 21-9 summarizes the

contingency factor applied in each area, and this result in an overall project contingency allowance of \$5.75 million.

Table 21-9: Contingency Factors by Category

Capital Category	Contingency Allowance (%)
Underground Mining	12.0
Processing	16.1
Infrastructure	15.0
Environmental	10.0

21.2 Operating Costs

21.2.1 Summary

Operating costs have been estimated for a 500 t/d mining and milling operation, with financial models also taking into account the fact that throughputs of 300 t/d are planned for the initial two years.

Project operating cost estimates are summarized in Table 21-10 on a cost per tonne milled basis during Years 1–2 (300 t/d processing rate) and Years 3–10 (500 t/d processing rate).

G&A costs are based on an average of 41 people, with 13 staff and 28 hourly employees.

Table 21-10: Operating Cost Summary for the Strieborná Project

Cost Area	Cost (\$/t)		
	300 t/d	500 t/d	LOM
Underground Mining	52.07	61.32	59.37
Processing	19.73	16.80	17.58
G&A	18.64	11.18	13.82
Total	99.69	80.05	90.77

21.2.2 Underground Mining

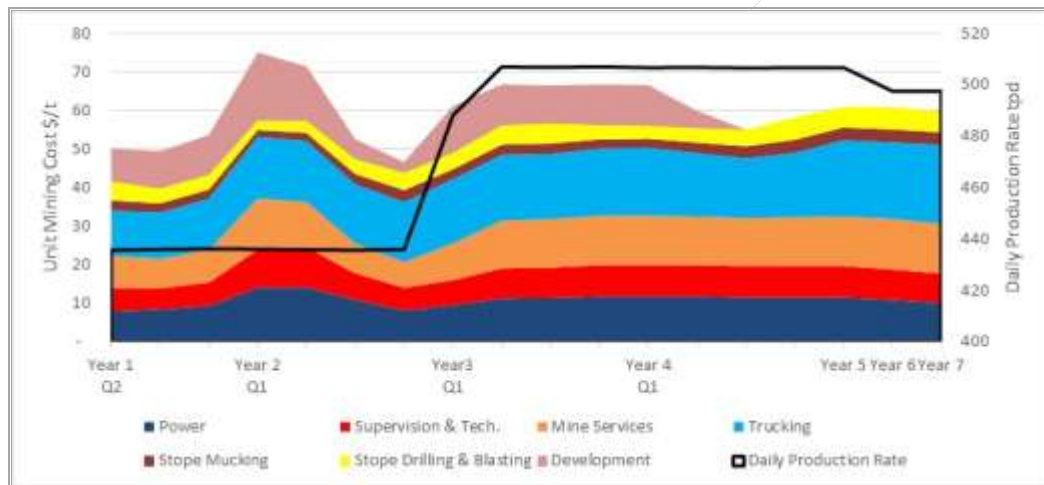
Methodology used to describe the estimation of the operating costs for the underground mine is described in Section 16.

Operating development activity was confined to in-vein development. Operating costs estimated during the project capital period were transferred into the capital cost estimate. A summary of the LOM operating costs is provided in Table 21-11. Unit costs during the operating period are shown graphically by activity in Figure 21-1.

Table 21-11: LOM Underground Operating Costs (US\$/t)

Activity	Cost (\$/t)	Total Cost (\$ '000s)	Proportion (%)
Development and Mucking	4.54	4,461.3	8
Production Drilling and Blasting	4.80	4,714.2	8
Stope Mucking	2.87	2,822.8	5
Trucking	17.32	17,021.6	29
Mine Services	11.79	11,588.6	20
Owner's Supervision	7.67	7,535.0	13
Power	10.39	10,215.2	18
Contingency	-	-	-
Total	59.37	58,358.8	100

Figure 21-1: Mine Unit Operating Costs by Activity



21.2.3 Process Plant

The process plant operating cost for the initial phase of the project, while operating at 300 t/d, is estimated at \$19.73/t processed. When the plant throughput increases to 500 t/d in Year 2, the operating cost is estimated at \$16.80/t processed. Most of these costs are attributable to labour and power. A breakdown, by major area, is provided in Table 21-12.

Details of the estimating methods in each of the major cost areas are discussed below.

Table 21-12: Operating Cost Estimate for the Strieborná Processing Plant

Cost Area	Cost (\$/t)	
	300 t/d	500 t/d
Labour – Mill	10.19	7.69
Power	4.17	4.17
Plant Maintenance Spares	1.69	1.28
Reagents	1.28	1.28
Mill Balls	0.77	0.77
Liners (Crusher + Mills)	0.52	0.52
Assay Laboratory	0.46	0.55
Safety Equipment	0.31	0.24
E&I Maintenance	0.20	0.15
Piping	0.08	0.08
Lubricants	0.06	0.06
Water	0.01	0.01
Subtotal	19.73	16.80

Labour

Labour costs have been calculated using typical plant staffing levels. Pay scales are based on recent database rates. The plant schedule is assumed to be two 12-hour shifts, with two shifts at the site and two shifts on leave at all times.

For the 300 t/d operating scenario, the mill is estimated to require a total complement of 28 persons at an estimated cost of \$10.19/t of ore processed. When the throughput increases to 500 t/d, the plant is expected to add an additional 9 personnel, but the labour operating cost will drop to \$7.69/t.

Power

Power supply is the second largest operating cost, accounting for ~21% of the overall total at start-up. A summary of the cost calculation is presented in Table 21-13.

The total connected power was taken from the mechanical equipment list, with load factors applied for each piece of equipment. The client has provided a power supply rate of \$130/MWh. This equates to an annual operating cost of \$4.17/t of ore treated, independent of the change in throughput.

Table 21-13: Process Plant Electricity Cost Estimate at 500 t/d Throughput

	Unit	Estimate
Total Connected Power	kW	1,251
Load Factor	%	60.4
Estimated Power Consumed	kW	756
Annual Running Time	h	7,750
Annual Consumption	MWh	5,856
Cost per MWh	\$	130
Total Cost	\$/a	761,271
	\$/t	4.17

Reagents

Reagent costs were estimated using unit costs from vendors and consumption rates from the lab testwork. A summary of the costs for each reagent is shown in Table 21-14. The total reagent cost amounts to \$1.28/t of mill feed.

Table 21-14: Summary of Estimated Reagent Operating Costs

Reagent	Annual Cost (\$)	Cost per Tonne (\$)
Lime	5,133	0.03
3477	94,900	0.52
MIBC	111,060	0.61
Flocculant	21,900	0.12
Subtotal	232,993	1.28

Balls and Liners

The cost of grinding balls and crusher/mill liners were estimated using database supply rates and consumption estimates based on the grindability testwork.

Maintenance and Supplies

An allowance for plant maintenance was factored from the mechanical supply cost for each area. The cost includes transportation, but the labour costs for replacement, installation, and maintenance are included in the labour allowance.

Similar amounts were factored from the overall mechanical supply cost to cover E&I maintenance and piping costs.

Fixed allowances were included to cover minor costs associated with lubricants, assay lab supplies, and safety equipment. Costs include training, monitoring equipment, first aid supplies, and personal protective equipment.

21.2.4 General and Administrative

General and administrative (G&A) costs make provision for 13 staff and 28 hourly employees. Most employees will be located in the town of Rožňava, with provision for a single ex-pat manager.

Preproduction G&A costs are included in the capital cost allowance under the owner's cost sub-category.

The LOM G&A cost is estimated to be \$13.82/t processed.

Table 21-15 summarizes the G&A expense categories applied to the estimate, together with average annual costs for each.

Table 21-15: G&A Costs, LOM Average

Category	Average Cost (\$/a)
Salaried Staff	477,000
Hourly Personnel	301,000
Site Operation and Maintenance Supplies	100,000
Site Power	100,000
Local Office	45,000
Information Systems (Hardware/Software)	20,000
Communications	40,000
Public/Community Relations	40,000
Recruitment and Training	80,000
Safety and Medical Supplies	50,000
Consultants	100,000
Legal and Audit Fees	50,000
Bank Charges	6,000
Permits and Licences	40,000
Taxes and Insurance	200,000
Logistics	30,000
Office Supplies	20,000
Warehouse Supplies	30,000
Equipment Rental and Maintenance	35,000
Employee Bussing	100,000
Security	50,000
Environmental Monitoring	60,000
Subtotal	1,974,000
Sustaining Capital at 2% of Operating	39,000
Total	2,013,000

22 ECONOMIC ANALYSIS

22.1 Summary

Detailed results of the analysis of the Strieborná Project are shown in Table 22-1.

Table 22-1: Discounted Cash Flow Results

Cost Category	Unit	Low Case	Base Case	High Case
Silver Price	\$/oz	18	20	22
Copper Price	\$/lb	2.70	3.00	3.30
Operating Costs				
Underground Mining	\$M	58.4	58.4	58.4
Processing, Tailings, and ROM Trucking	\$M	18.5	18.5	18.5
G&A	\$M	13.6	13.6	13.6
Subtotal On-site Costs	\$M	90.5	90.5	90.5
Transportation, Port Costs, Shipping	\$M	5.0	5.0	5.0
Total Operating Costs	\$M	95.5	95.5	95.5
Capital Costs				
Underground Mining	\$M	29.5	29.5	29.5
Processing	\$M	10.2	10.2	10.2
Infrastructure	\$M	3.7	3.7	3.7
Environmental Costs	\$M	0.2	0.2	0.2
Owners and Indirect Costs	\$M	4.3	4.3	4.3
Contingency	\$M	5.7	5.7	5.7
Total Capital Costs	\$M	53.6	53.6	53.6
Copper Revenue	\$M	68.7	74.9	81.0
- Less fuming, smelting, refining, penalties	\$M	76.6	76.6	76.6
Net Copper Revenue	\$M	-7.8	-1.7	4.4
Silver Revenue	\$M	161.6	179.6	197.5
- Less refining	\$M	5.4	5.4	5.4
Net Silver Revenue	\$M	156.2	174.2	192.1
Net Copper and Silver Revenue	\$M	148.4	172.5	196.5
Pre-Tax Cash Flow	\$M	-0.7	23.3	47.4
Total Tax	\$M	3.2	8.4	13.6
Post-Tax Cash Flow	\$M	-3.9	14.9	33.8
Pre-Tax NPV @ 5%	\$M	-7	11	29
Pre-Tax IRR	%	-0.4	12.4	23.6
Pre-Tax Payback Period	years	-	4.7	3.7
Post-Tax NPV @ 5%	\$M	-10	5	19
Post-Tax IRR	%	-2.3	8.3	17.9
Post-Tax Payback Period	years	-	5.1	4.0

The project as envisioned provides positive results at the base case metal prices of \$20/oz Ag and \$3.00/lb Cu. The pre-tax Net Present Value (NPV) with a 5% discount rate

is \$11 million using the base case metal prices. Post-tax, with the same 5% discount rate and metal prices, the NPV is \$5 million. Internal rates of return (IRR) are 12.42% pre-tax, and 8.3% post-tax. Payback on the project capital is expected to be 4.7 years pre-tax, and 5.1 years post-tax.

The results also show the project's leverage to the higher metal prices, as indicated in the high case, with silver at \$22/oz and copper at \$3.30/lb. These higher metal prices increase the post-tax NPV at a 5% discount rate to \$19 million, with a 17.9% after-tax IRR, more than double the base case.

This assessment is preliminary in nature as it includes Inferred Mineral Resources that are presently considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the assumptions used in the preliminary assessment will be realized.

Metal recoveries of 98% were applied to all tonnes milled.

Costs for the Strieborná Project are shown in terms of dollars per tonne (\$/t) and dollars per dry metric tonne (\$/dmt) of silver-copper concentrate (Ag-Cu Conc.) in Table 22-2.

Table 22-2: Base Case Operating Cost Estimates

Category	Mill Feed (\$/t)	Ag-Cu Conc. (\$/dmt)
Mining (mill feed and waste)	59.37	1,436.67
Processing, Tailings, and ROM Trucking	18.83	455.73
G&A	13.82	334.49
Subtotal On-site Costs	92.02	2,226.89
Transportation, Port Costs, Shipping	5.08	122.95
Total Operating Costs	97.10	2,349.84

22.2 Mill Feed Schedule

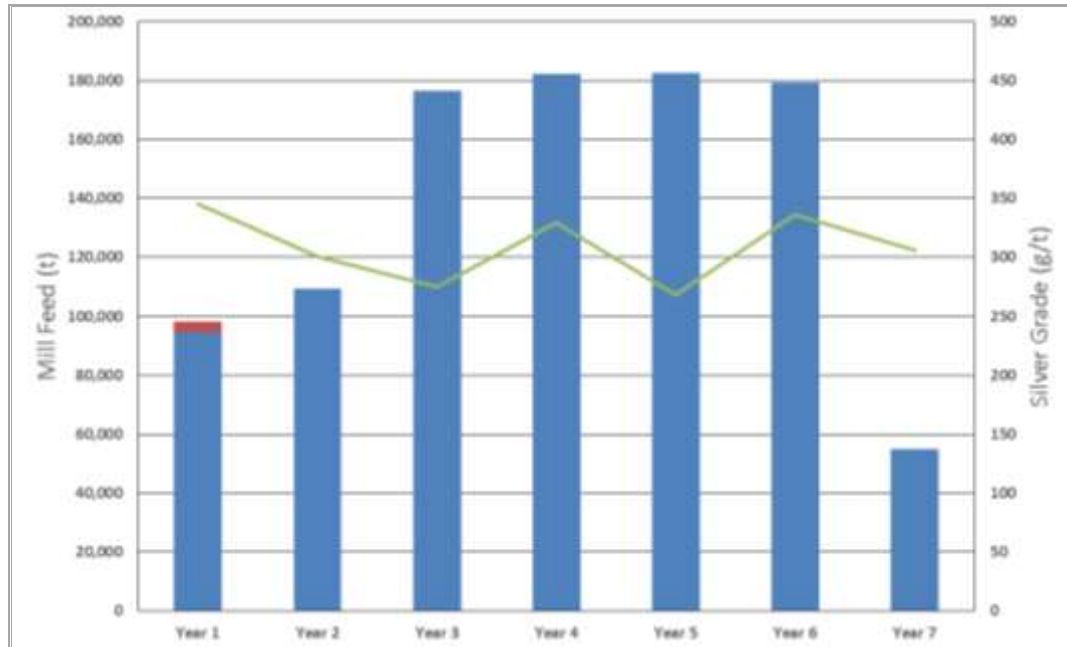
The mining schedule discussed in Section 16 of this report was condensed into a mill feed schedule. This schedule adjusted the mining schedule to match the start of mill production, and made use of temporary stockpiling prior to mill commissioning.

Life-of-mine (LOM) mill feed totals 983,000 tonnes grading 307 g/t Ag, 1.34% Cu, and 0.92% Sb. Of that, 979,000 tonnes are classified as Measured and Indicated with the remaining 4,000 tonnes classified as Inferred.

A preproduction stockpile of 3,500 tonnes will be created from underground development in Year -1. This material will be available for mill start-up, and reclaimed in Year 1.

The initial production rate is 300 t/d, ramping up to 500 t/d in Year 3, and sustained until the LOM end in Year 7. The mill feed schedule is shown graphically in Figure 22-1.

Figure 22-1: Mill Feed Schedule



22.3 Metal Pricing Assumptions

Typical metal pricing for projects is based on a three-year trailing average price. Using that criteria, the silver price would be \$24.84/oz, and copper would be \$3.36/lb (December 2011 to 2014). Silver price volatility in the last year with a downward trend has caused concern about using the three-year trailing average for projects. Copper has shown more stability, but has recently also succumbed to market pressure downwards from the three-year average. Until stability is restored in the metal markets, a more conservative approach to metal price selection is being adopted by AGP for the Strieborná Project.

While the spot metal prices are lower than the base case price applied, the PEA is also illustrating the impact of higher metal prices, in particular silver, which would be required for improved economics. The use of \$20/oz Ag and \$3.00/lb Cu is not considered unrealistic in light of recent history.

To set the low and high price cases, a 10% variation on either side was applied to the base prices.

The copper concentrate is carrying all of the silver that is payable. It is also carrying all of the penalty items. The terms were determined from indicative pricing with the concentrate specifications expected from the process.

The copper concentrate contains significant penalties associated with mercury, arsenic, and antimony. The mercury penalty will be eliminated by fuming of the copper concentrate in a licenced facility in Germany, allowing proper stabilization and storage of the resulting mercury. This is estimated to cost \$520/dmt of copper concentrate.

The arsenic and antimony remain as significant penalty items in the resulting calcine from the fuming process. Arsenic grades 1% in the calcine, drawing a penalty of \$99/dmt of calcine. The antimony grade is 20.2% in the calcine, and suffers with a significant penalty of \$1,004/dmt.

Smelting of the calcine is estimated to cost \$120/dmt, and a price participation reduction of \$175/dmt is added to this. Payables for the copper are 96% after a minimum deduction of 1%. Refining terms are \$0.12/lb of payable copper.

Silver is estimated to be 95% payable in the calcine, with a minimum deduction of 30 g/t, and a refining charge of \$0.60/oz.

22.4 Taxes and Royalties

Various taxes and royalties are included in the economic evaluation. These include the Government Royalty, Corporate Income Tax, an estimation of depreciation and also the Value Added Tax (VAT). These have been applied.

22.4.1 Government Royalty

The Government Royalty works on a ratio of the cost of mining the minerals to the total cost of the final product. The royalty rate varies for each metal, and in discussion with Global Minerals, AGP applied a royalty of 0.1% for each metal using the ratio concept.

Due to the complex nature of the final product, detailed calculations will be required to accurately determine the exact royalty payable in subsequent studies.

22.4.2 Depreciation

For this PEA study, a rudimentary estimate of depreciation was completed on an annual basis. Two assumptions were made: underground development would use a 30% declining balance; all other capital would be depreciated at a 20% declining balance.

For the depreciation, no set time period was applied, and salvage was considered to be zero dollars.

Depreciation was calculated on all capital, but was not applied until mill production commenced in Year 1. The calculation was completed for all years of mining.

22.4.3 Corporate Income Tax

A Federal tax rate of 23% was applied on income after accounting for depreciation and carried forward losses. This was a simple calculation to determine the impact, and AGP considers this to be reasonable for this level of study.

Carry forward costs associated with rehabilitation and exploration were included in Year 1, and are permitted under Slovakian regulations to be used to offset initial profits.

22.4.4 Value Added Tax (VAT)

Slovakia has a VAT rate currently set at 20%. Detailed estimation of VAT for each item has not been completed for this study. A broad-brush approach to estimate applicable VAT was made, based on capital and operating cost estimates. VAT was applied to 80% of the operating cost, and to 50% of the capital costs. This was based on the assumption that some of the costs would be net of VAT.

An estimate of 15% of the VAT was assumed to be non-refundable. The total VAT payable by the company was added to the corporate tax payable to arrive at the total tax payable on an annual basis.

22.5 Discounted Cash Flow Analysis

This assessment is preliminary in nature, as it includes Inferred Mineral Resources that are presently considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the assumptions used in the preliminary assessment will be realized.

LOM mill feed totals 983,000 tonnes grading 307 g/t Ag, 1.34% Cu and 0.92% Sb. Of that, 979,000 tonnes are classified as Measured and Indicated, with the remaining 4,000 tonnes classified as Inferred.

The initial production rate is 300 t/d, ramping up to 500 t/d in Year 3, and sustained until the LOM end in Year 7.

The Base Case metal prices are \$20/oz Ag and \$3.00/lb Cu. A 5% discount rate is used as the basis for all the cases, with examination at 0% and 7% also included.

The results of the analysis of the Strieborná Project Base Case, as it is currently envisioned, show positive economics. The pre-tax NPV with a 5% discount rate is \$11 million. Post-tax, with the same 5% discount rate, the NPV is \$5 million. IRR are 12.4% pre-tax, and 8.3% post-tax. Payback on the project capital is expected to be 4.7 years pre-tax, and 5.1 years post-tax. The detailed results are shown in Table 22-3 and Table 22-4.

Table 22-3: Discounted Cash Flow Results Summary

	Unit	Pre-Tax			Post-Tax		
		Low Case	Base Case	High Case	Low Case	Base Case	High Case
Silver Price	\$/oz	18	20	22	18	20	22
Copper Price	\$/lb	2.70	3.00	3.30	2.70	3.00	3.30
NPV (0%)	\$M	-1	23	47	-4	15	34
NPV (5%)	\$M	-7	11	29	-10	5	19
NPV (7%)	\$M	-9	7	23	-11	2	14
IRR	%	-0.4	12.4	23.6	-2.3	8.3	17.9
Payback Period	years	-	4.7	3.7	-	5.1	4.0
LOM Metal Revenue ⁽¹⁾	\$M	148	172	196	148	172	196

Note: ⁽¹⁾ After fuming, smelting, refining, payables, and royalties

Table 22-4: Discounted Cash Flow Detailed Results

Cost Category	Unit	Low Case	Base Case	High Case
Silver Price	\$/oz	18	20	22
Copper Price	\$/lb	2.70	3.00	3.30
<i>Operating Costs</i>				
Underground Mining	\$M	58.4	58.4	58.4
Processing, Tailings, and ROM Trucking	\$M	18.5	18.5	18.5
G&A	\$M	13.6	13.6	13.6
Subtotal On-site Costs	\$M	90.5	90.5	90.5
Transportation, Port Costs, Shipping	\$M	5.0	5.0	5.0
Total Operating Costs	\$M	95.5	95.5	95.5
<i>Capital Costs</i>				
Underground Mining	\$M	29.5	29.5	29.5
Processing	\$M	10.2	10.2	10.2
Infrastructure	\$M	3.7	3.7	3.7
Environmental Costs	\$M	0.2	0.2	0.2
Owners and Indirect Costs	\$M	4.3	4.3	4.3
Contingency	\$M	5.7	5.7	5.7
Total Capital Costs	\$M	53.6	53.6	53.6
Copper Revenue	\$M	68.7	74.9	81.0
- Less fuming, smelting, refining, penalties	\$M	76.6	76.6	76.6
Net Copper Revenue	\$M	-7.8	-1.7	4.4
Silver Revenue	\$M	161.6	179.6	197.5
- Less refining	\$M	5.4	5.4	5.4
Net Silver Revenue	\$M	156.2	174.2	192.1
Net Copper and Silver Revenue	\$M	148.4	172.5	196.5
Pre-Tax Cash Flow	\$M	-0.7	23.3	47.4
Total Tax	\$M	3.2	8.4	13.6
Post-Tax Cash Flow	\$M	-3.9	14.9	33.8
Pre-Tax NPV @ 5%	\$M	-7	11	29
Pre-Tax IRR	%	-0.4	12.4	23.6
Pre-Tax Payback Period	years	-	4.7	3.7
Post-Tax NPV @ 5%	\$M	-10	5	19
Post-Tax IRR	%	-2.3	8.3	17.9
Post-Tax Payback Period	years	-	5.1	4.0

22.6 Sensitivity Analysis

The project sensitivity to various inputs was examined on the Base Case (post-tax). The inputs that were analyzed for sensitivity were:

- silver grade
- silver price
- total capital cost
- total operating cost
- mercury treatment cost
- copper treatment cost
- antimony concentrate grade.

The results of the analysis are shown in Table 22-5.

Table 22-5: NPV Sensitivity (\$ millions) – Post-Tax

Sensitivity Item	-20 %	-10 %	Base	10 %	20 %
Silver Grade	-18	-6	5	15	25
Silver Price	-19	-6	5	15	24
Total Capital Cost	12	8	5	1	-2
Total Operating Cost	16	10	5	-1	-7
Mercury Treatment Cost	7	6	5	3	2
Copper Treatment Cost	6	5	5	4	4
Antimony Concentrate Grade	9	7	5	2	0

The greatest sensitivity is to silver price and silver grade, both of them being related. As the price increases, the NPV of the project also dramatically increases. A 20% increase in the price of silver results in the NPV rising from \$5 million to \$24 million, an almost fivefold increase. A 20% decrease in the price of silver also affects the project economics, resulting in a drop to -\$19 million.

Changes in the mill feed silver grade have an almost identical impact.

The least sensitive item is the copper treatment cost. A 20% increase or decrease results only in a \$1 million change in the project NPV or 20%.

The sensitivities of the NPV (5%) and IRR at various metal price combinations are shown in Table 22-6 and Table 22-7. This further highlight the significant leverage the project has to increases in the metal prices, particularly silver.

Table 22-6: NPV 5% Sensitivity – Post-Tax

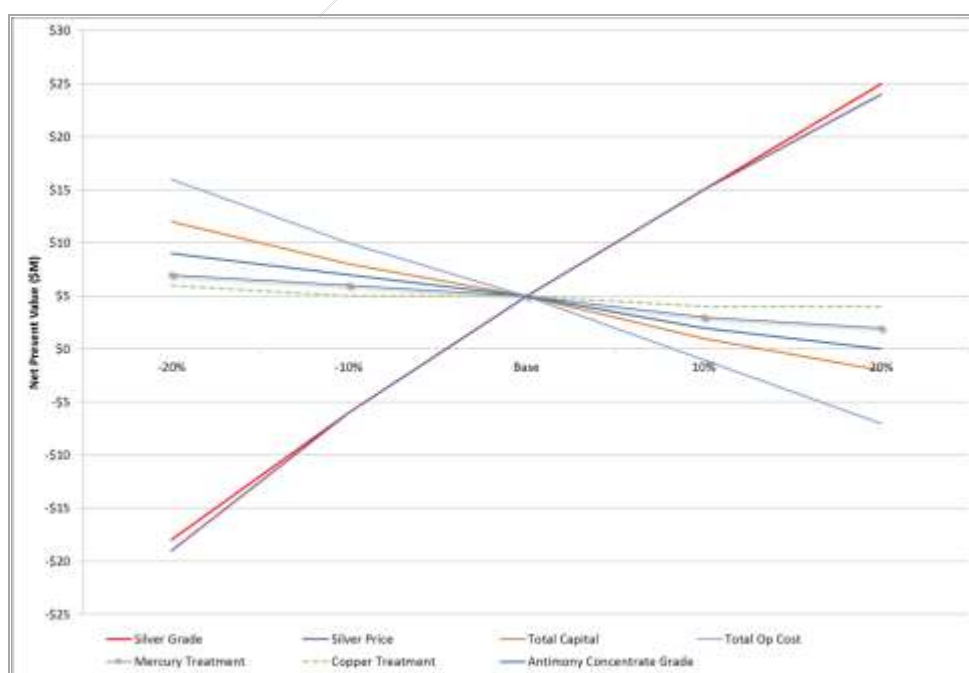
NPV (\$M)		Silver Price (\$/oz)				
		\$18.00	\$20.00	\$22.00	\$24.00	\$26.00
Cu Price (\$/lb)	\$2.50	-13	-1	9	20	30
	\$2.75	-9	2	12	23	33
	\$3.00	-6	5	15	26	36
	\$3.25	-3	8	18	29	39
	\$3.50	0	11	21	32	42
	\$3.75	3	14	24	35	45

Table 22-7: IRR Sensitivity – Post-Tax

IRR (%)		Silver Price (\$/oz)				
		\$18.00	\$20.00	\$22.00	\$24.00	\$26.00
Cu Price (\$/lb)	\$2.50	-4.6	3.9	11.5	18.5	25.0
	\$2.75	-1.8	6.1	13.5	20.4	26.8
	\$3.00	0.5	8.3	15.6	22.3	28.6
	\$3.25	2.8	10.4	17.5	24.1	30.3
	\$3.50	5.0	12.5	19.5	25.9	32.1
	\$3.75	7.2	14.6	21.4	27.7	33.8

The NPV sensitivity is also shown in a spider graph in Figure 22-2.

Figure 22-2: NPV 5% Sensitivity – Post-Tax



22.7 Cash Costs

Cash cost tabulations were made to provide the operating costs in either dollars per tonne milled or dollars per tonne of silver/copper concentrate.

Table 22-8: Summary of Base Case Operating Cost Estimates

Operating Cost Category	Mill Feed (\$/t)	Ag-Cu Concentrate (\$/dmt)
Operating Costs		
Mining (mill feed and waste)	59.37	1,436.67
Processing, Tailings, and ROM Trucking	18.83	455.73
G&A	13.82	334.49
Subtotal On-site Costs	92.02	2,226.89
Transportation, Port Costs, Shipping	5.08	122.95
Total Operating Costs	97.10	2,349.84

22.8 Detailed Cash Flow

The Base Case cash flow is shown in Table 22-9 and Table 22-10. This includes the mine schedule, copper, and silver recovery, payable calculations, operating and capital costs, revenue estimates, taxation calculations and pre-tax and post-tax cash flow. For each of the cash flows, the NPV and IRR are determined.

Table 22-9: Base Case Cash Flow – Mine Production

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Mill Production												
Mill Feed Tonnes	tonnes	982,963	-	-	98,132	109,421	176,375	182,392	182,412	179,111	55,120	-
Value per Tonne	\$/t	144	-	-	164	139	128	153	127	159	144	-
Copper	%	1.34	-	-	1.55	1.43	1.21	1.44	1.14	1.42	1.30	-
Antimony	%	0.92	-	-	1.02	1.01	0.84	1.03	0.76	0.96	0.89	-
Silver	g/t	307	-	-	345.18	301.04	275.09	329.57	267.86	336.41	306.11	-
Concentrates and Metal												
Copper (recovered)	pounds	28,495,752	-	-	3,294,401	3,371,442	4,609,755	5,676,894	4,495,823	5,496,915	1,550,522	-
Copper Concentrate	dry tonnes	40,621	-	-	4,696	4,806	6,571	8,092	6,409	7,836	2,210	-
Copper Concentrate	wet tonnes	42,759	-	-	4,943	5,059	6,917	8,518	6,746	8,248	2,327	-
Delivered to Smelter (less losses):												
Copper Concentrate	dry tonnes	40,418	-	-	4,673	4,782	6,538	8,052	6,377	7,797	2,199	-
Copper Payable	dry tonnes	38,801	-	-	4,486	4,591	6,277	7,730	6,122	7,485	2,111	-
	pounds	27,219,142	-	-	3,146,812	3,220,402	4,403,238	5,422,569	4,294,410	5,250,653	1,481,058	-
Silver (recovered)	ounces	9,497,465	-	-	1,067,258	1,037,878	1,528,738	1,893,963	1,539,513	1,898,492	531,621	-
Silver (in copper concentrate delivered)	g/dmt	7,272	-	-	7,068.6	6,716.9	7,235.9	7,279.5	7,471.6	7,535.8	7,481.1	-
Silver (payable)	ounces	8,977,478	-	-	1,006,826	981,055	1,445,040	1,790,269	1,455,225	1,794,550	502,514	-
Mine Production												
Mined Tonnes	tonnes	982,963	-	3,515	94,617	109,421	176,375	182,392	182,412	179,111	55,120	-
Operating Cost												
Underground Mining	US Dollars	58,358,790	-	0	4,155,914	6,652,392	11,460,171	10,865,407	11,085,682	10,854,204	3,285,019	0
Processing	US Dollars	17,283,410	-	-	1,936,144	2,158,876	2,963,100	3,064,186	3,064,522	3,009,065	1,087,518	-
G&A	US Dollars	13,587,500	-	-	2,013,000	2,013,000	2,013,000	2,013,000	2,013,000	2,013,000	1,509,500	0
Tailings & Trucking ROM	US Dollars	1,228,704	-	0	122,665	136,776	220,469	227,990	238,015	223,889	66,900	0
Concentrate Trucking	US Dollars	3,155,607	-	0	364,820	373,352	510,482	628,657	497,865	608,728	171,704	0
Port Costs	US Dollars	213,795	-	0	24,717	25,295	34,586	42,592	33,731	41,242	11,633	0
Shipping to Smelter	US Dollars	1,624,838	-	0	187,848	192,241	262,850	323,699	256,353	313,436	88,411	0
Subtotal Operating	US Dollars	95,452,644	-	0	8,805,109	11,551,932	17,464,658	17,165,530	17,179,168	17,063,561	6,222,685	0
Capital Cost												
Open Pit Mining	dollars	0	0	0	0	0	0	0	0	0	0	0
Underground Mining	dollars	29,537,298	0	10,588,604	10,723,164	6,747,258	1,478,272	0	0	0	0	0
Processing	dollars	10,202,442	0	5,821,465	3,880,977	500,000	0	0	0	0	0	0
Infrastructure	dollars	3,662,296	657,500	2,447,500	468,813	10,942	17,638	18,239	18,241	17,911	5,512	0
Environment Costs	dollars	150,000	0	0	0	0	0	0	0	0	45,000	105,000
Indirect	dollars	4,274,531	567,103	2,594,500	901,329	211,599	0	0	0	0	0	0
Contingency	dollars	5,752,434	164,803	2,422,046	1,999,159	974,032	177,393	0	0	0	0	15,000
Subtotal Capital	dollars	53,579,001	1,389,407	23,874,115	17,973,443	8,443,831	1,673,302	18,239	18,241	17,911	50,512	120,000

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Table 22-10: Base Case Cash Flow – Economic Calculations

		Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Revenue (after smelting, price participation, refining and losses)												
Copper - Gross Revenue (payable x price)	dollars	74,852,640	-	-	8,653,732	8,856,104	12,108,905	14,912,065	11,809,627	14,439,296	4,072,910	-
Copper - Charges/Penalties												
Smelting	dollars	4,850,142	-	-	560,726	573,839	784,607	966,240	765,215	935,607	263,908	-
Refining	dollars	3,266,297	-	-	377,617	386,448	528,389	650,708	515,329	630,078	177,727	-
Mercury (Roasting)	dollars	23,893,276	-	-	2,761,151	2,825,722	3,863,595	4,758,001	3,768,105	4,607,155	1,299,546	-
Arsenic penalty	dollars	4,001,367	-	-	462,599	473,417	647,301	797,148	631,302	771,876	217,724	-
Antimony penalty	dollars	40,567,155	-	-	4,689,979	4,799,656	6,562,545	8,081,746	6,400,348	7,825,524	2,207,355	-
Mercury penalty	dollars	0	-	-	-	-	-	-	-	-	-	-
Subtotal Charges/Penalties	dollars	76,568,237	-	-	8,852,072	9,059,083	12,386,437	15,253,845	12,080,300	14,770,240	4,166,260	-
Copper - Net Revenue (Gross - Charges/Penalties)	dollars	-1,715,596	-	-	198,340	202,979	277,532	341,779	270,673	330,944	93,350	-
Silver - Gross Revenue (payable x price)	dollars	179,549,569	-	-	20,176,517	19,621,092	28,900,798	35,805,376	29,104,500	35,890,996	10,050,290	-
Silver - Refining Charges	dollars	5,386,487	-	-	605,296	588,633	867,024	1,074,161	873,135	1,076,730	391,509	-
Silver - Net Revenue (Gross - Refining)	dollars	174,163,081	-	-	19,571,222	19,032,459	28,033,774	34,731,214	28,231,365	34,814,266	9,748,781	-
Subtotal Revenue	dollars	172,447,485	-	-	19,372,881	18,829,480	27,756,243	34,389,435	27,960,692	34,483,322	9,655,431	-
Royalty Calculation												
Cost of Mining Minerals	dollars	73,174,994	0	0	6,291,579	8,802,168	13,693,640	13,106,397	13,326,697	13,091,093	4,863,419	0
Total Cost of manufactured product	dollars	177,407,367	-	-	18,262,476	21,199,649	30,718,119	33,493,536	30,132,603	32,910,531	10,690,454	-
less Copper Royalty	dollars	9,038	-	-	895	1,144	1,594	1,716	1,508	1,648	534	-
less Antimony Royalty	dollars	0	-	-	-	-	-	-	-	-	-	-
less Silver Royalty	dollars	52,040	-	-	4,865	5,613	9,079	9,891	9,157	10,181	3,254	-
Subtotal Royalty	dollars	61,077	-	-	5,759	6,757	10,673	11,607	10,664	11,829	3,788	-
Net Revenue	dollars	172,386,408	0	0	19,367,122	18,822,724	27,745,569	34,377,828	27,950,028	34,471,494	9,651,643	0
Pre-Tax Cashflow												
Operating Expenditure	dollars	95,454,000	0	0	8,805,000	11,552,000	17,465,000	17,166,000	17,179,000	17,064,000	6,223,000	0
Capital Cost	dollars	53,578,000	1,389,000	23,874,000	19,973,000	8,444,000	1,673,000	18,000	18,000	18,000	51,000	120,000
Revenue	dollars	172,387,000	0	0	19,367,000	18,823,000	27,746,000	34,378,000	27,950,000	34,471,000	9,652,000	0
Pre-Tax Cashflow	dollars	23,355,000	-1,389,000	-23,874,000	-7,411,000	-1,173,000	8,608,000	17,194,000	10,753,000	17,389,000	3,378,000	-120,000
Pre-Tax Cumulative Cashflow	dollars		-1,389,000	-25,263,000	-32,674,000	-33,847,000	-25,239,000	-8,045,000	2,708,000	20,097,000	23,475,000	23,355,000
NPV (millions) @												
0%	\$23		-1,389,000	-23,874,000	-7,411,000	-1,173,000	8,608,000	17,194,000	10,753,000	17,389,000	3,378,000	-120,000
5%	\$11		-1,323,034	-21,654,422	-6,401,900	-965,030	6,744,593	12,828,713	7,640,935	11,767,987	2,177,198	-73,650
7%	\$7		-1,298,371	-20,852,476	-6,049,584	-894,876	6,137,385	11,454,965	6,695,187	10,118,680	1,837,068	-60,979
Payback Period	years	4.7										
IRR		12.4%										
Post-Tax Cashflow												
Depreciation												
Capital Depreciation - Underground	dollars	29,537,298	-	-	3,196,765	6,446,590	5,746,442	4,244,250	2,970,975	2,079,683	4,852,593	-
Capital Depreciation - Infrastructure	dollars	23,891,703	-	-	2,192,520	4,116,193	3,482,114	2,807,018	2,249,263	1,803,025	7,208,161	13,998
Annual Depreciation	dollars	53,429,001	-	-	5,389,285	10,562,782	9,228,557	7,051,269	5,220,238	3,882,708	12,060,754	13,998
Revenue - Operating Cost	dollars	-	-	-	10,562,000	7,271,000	10,281,000	17,212,000	10,771,000	17,407,000	3,429,000	-
less Annual Depreciation Applied	dollars	-	-	-	5,389,285	7,271,000	9,228,557	7,051,269	5,220,238	3,882,708	3,429,000	-
Pre-Tax less Annual Depreciation Applied	dollars	-	-	-	5,172,715	-	1,052,443	10,160,731	5,550,762	13,524,292	-	-
Carried Forward Depreciation	dollars	-	-	-	8,953,000	3,780,285	7,072,067	6,019,624	-	-	-	8,631,754
Carried Forward Depreciation Applied	dollars	-	-	-	5,172,715	-	1,052,443	6,019,624	-	-	-	-
Pre-Tax less Depreciation(Annual & Carried forward)	dollars	-	-	-	-	-	-	4,141,107	5,550,762	13,524,292	-	-
Depreciation Carried Forward	dollars	-	-	-	3,780,285	7,072,067	6,019,624	-	-	-	8,631,754	8,645,753
Corporate Tax												
Corporate Tax	dollars	5,339,717	-	-	-	-	-	952,455	1,276,675	3,110,587	-	-
VAT												
VAT on Operating	dollars	2,290,863	-	-	211,323	277,246	419,152	411,973	412,300	409,525	149,344	-
VAT on Capital	dollars	803,685	20,841	358,112	269,602	126,657	25,100	274	274	269	758	1,800
Total VAT Payable	dollars	3,094,548	20,841	358,112	480,924	403,904	444,251	412,246	412,574	409,794	150,102	1,800
Total Tax	dollars	8,434,266	20,841	358,112	480,924	403,904	444,251	1,364,701	1,689,249	3,520,381	150,102	1,800
Post-Tax Cashflow	dollars	14,920,734	-1,409,841	-24,232,112	-7,891,924	-1,576,904	8,163,749	15,829,299	9,063,751	13,868,619	3,227,898	-121,800
Post-Tax Cumulative	dollars		-1,409,841	-25,641,953	-33,533,877	-35,110,781	-26,947,032	-11,117,733	-2,053,982	11,814,637	15,042,534	14,920,734
NPV (millions) @												
0%	\$15		-1,409,841	-24,232,112	-7,891,924	-1,576,904	8,163,749	15,829,299	9,063,751	13,868,619	3,227,898	-121,800
5%	\$5		-1,342,885	-21,979,240	-6,817,341	-1,297,323	6,396,511	11,810,488	6,440,578	9,385,572	2,080,454	-74,755
7%	\$2		-1,317,853	-21,165,265	-6,442,161	-1,203,012	5,820,640	10,545,775	5,643,402	8,070,166	1,755,437	-61,894
Payback Period	years	5.1										
IRR		8.3%										

23 ADJACENT PROPERTIES

There are no known adjacent mineral properties with the type of mineralization associated with the Strieborná deposit that are currently either under exploration, under development, or in production.



24 OTHER RELEVANT DATA AND INFORMATION

AGP is not aware of any other relevant data or information that has not been previously stated in the report that might affect the continued exploration activity on the Strieborná Project.

25 INTERPRETATION AND CONCLUSIONS

25.1 Resource Estimate

While Global Minerals' work only defines a small part of the deposit by drilling in the upper northeast portion of the vein, it confirms historical work, perceptions, and interpretations of the veins. Furthermore, and a very important aspect of this resource estimate, is that it is similar to previous resource estimates done for Strieborná over the years.

As more work is done at Strieborná, more splays and structural complications will become evident, and high-grade shoots will be better defined. The available data, except in the areas of most concentrated sampling, do not allow for defining and modelling what is expected to be more complicated than a simple vein. Furthermore, the restricted sample distribution defining the Strieborná mineralization made this a very complicated estimate, ameliorated only by the apparent continuity of the mineralization.

The clustering of close-spaced sampling only on development levels does not permit modelling higher grades separately within the vein for the majority of the deposit. Furthermore, it remains unclear if these higher grades form continuous predictable zones within the vein. However, where Global Minerals has drilled and sampled in the upper northeast portion of the deposit, continuity of high-grade zones is becoming better established.

There is the potential for additional mineralization beyond Inferred classification within the interpreted veins, but it is not reported. There is also very good potential to increase the resources by additional drilling on Strieborná and also on Pallag, but particularly on Podložná.

25.2 Geotechnical

BGC Engineering Inc. was retained by AGP, on behalf of Global to provide preliminary geotechnical evaluations to support the development of mining designs and cost estimates for the Strieborná Project.

Review of rock mechanics data, together with inspection of the accessible areas of the existing mine, allowed the development of initial design criteria for the sizing of stopes and pillars together with recommended support assumptions.

The design criteria were provided for general guidance to the PEA-level mine design and for cost estimating purposes. As additional geological and geotechnical work is completed, the current designs will be reviewed and updated as required.

25.3 Underground Mining

The Strieborná vein will be accessed from the upper levels of the existing Mária Mine. A mechanized method with truck haulage to surface has been adopted in order to avoid the complicated, low productivity materials handling system used historically. Rehabilitation of existing workings has already been completed to Level 6. Further rehabilitation of the subvertical shaft and mining levels below Level 6 will be required. The mine is currently dewatered to below Level 8, but additional pumping will be required to lower the water level to Level 11.

Although the Strieborná vein pinches and swells, the generally regular, steep dipping geometry of the vein, and average 4.9 m width indicates longhole stoping methods to be appropriate. To minimize potential stope dilution resulting from longhole drilling the vertical sublevel interval was limited to 17 m. AGP assumed a minimum planned stoping width of 1.5 m, adjusting the planned development and stope grades to include the non-vein dilution material. Preliminary geotechnical guidelines were used to design the longhole stope and pillar dimensions.

The mine plans described in this study indicate the potential to produce 983 kt of mill feed grading 307 g/t Ag and 1.34% Cu to the process plant over an operating period of six to seven years. It is planned that Global will employ its own workforce to undertake the majority of the mining operations. Mobile mining equipment will be leased. Dilution in plant feed, defined as waste tonnes/resource tonnes, is estimated to be 13.5%.

The mine will produce 110 kt/a of mill feed operating on a three 8 h/d shift system, five days per week until Q1 Year 3 when mine operations will be conducted on a seven day per week basis and annual production will increase to 180 kt/a.

Mine cost estimates were developed from first principles. The mining costs are estimated to include initial project capital of \$14.1 million during a fifteen month period prior to the commissioning of the process plant. Thereafter, mining costs were estimated to comprise operating costs of \$59/t and sustaining capital of \$15.5 million.

25.4 Metallurgy and Processing

25.4.1 Mineral Processing

Samples selected for the metallurgical testwork programs described in this technical report are considered preliminary in nature. Additional work would include a structured sampling program and larger sample masses to ensure more complete representation of mineralization variability and spatial distribution.

Mineral processing testwork conducted to date has been very encouraging. The following conclusions are drawn:

- The tested sample has a BWi of 8.7 kWh/t and is therefore considered to be softer than average. This will have a positive impact on the capital/operating cost of grinding.
- At a relatively coarse primary grind, the Strieborná Phase 1 and main deposit composites contained free, fast-floating tetrahedrite copper and silver, excellent recoveries of silver and copper (~98%) were achieved using a conventional, simple and low-cost flotation process.
- A high-grade copper/silver concentrate (~27% Cu and 6,500 g/t Ag) was produced in locked-cycle testing at over 98% recovery of silver and copper. Further improvements are expected through the inclusion of regrinding in the circuit. However, the concentrates produced are especially rich in antimony and mercury, which is likely to reduce the scope for their marketability as a direct-sale product, without pre-processing to remove some or all of these impurities.
- Cinnabar is fast floating and associated with strongly floatable minerals therefore mercury is also significantly upgraded in flotation.

25.4.2 *Pyrometallurgical Testing*

Near complete mercury elimination from Strieborná concentrate can be achieved by roasting up to 750°C, with the likely required elimination of 99.6% (~30 ppm Hg for a nominal level of 7,000 ppm in concentrate) achieved at 650°C. While this result was achieved by roasting in a nitrogen atmosphere, thermodynamic calculations and Gortdrum Mines experiences suggest the same result would have been achieved by roasting with a small amount of oxygen present (about 0.5% O₂). At this stage, any possible sintering effect was not determined.

The testwork completed to date suggests a one-step roasting process would be sufficient in a commercial operation. The specific roasting times and temperatures require optimization, but would be in the order of 170 minutes and 650°C.

Amenability studies by a third party processor for removal of the mercury have been conducted and the third party has indicated the product is highly amenable to mercury removal and safe mercury disposal through their process.

25.4.3 *Hydrometallurgical Testing*

Historical testwork, past operations at the Sunshine Mine (Idaho) and more recent testing at UBC and SGSM (Lakefield) all suggest the antimony concentration in flotation concentrates can be dramatically reduced via the use of an alkali leach process.

This testwork has shown that 99% of the antimony and 98% of the mercury can be extracted through a caustic sulphide leach. Further, preliminary testing of mercury precipitation alternatives suggests some potential for mercury removal from the pregnant liquor ahead of antimony recovery. However, this testwork must be considered as very preliminary and the reader should be cautioned that much more

testing is needed before any real confidence can be ascribed to the viability of this process.

25.4.4 Recovery Methods

Interpretation of the metallurgical testwork completed to date suggests a very simple and conventional crushing, grinding, and flotation process would be required on site at Strieborná to make a copper-silver-antimony flotation concentrate.

Metallurgical testwork indicates a relatively low hardness (BWi) and this will translate into lower than average mill capital requirements and grinding power requirements as the project progresses.

The recovery of copper and silver to a flotation concentrate has been shown to be excellent (>95%), although removal of mercury (for disposal) and antimony (for sale) from this concentrate is necessary due to elevated levels of these elements. The concentrations of mercury and antimony in the flotation concentrate would likely have a negative impact on the value and/or saleability if shipped directly to a smelter.

Several potential options exist for the downstream processing of Strieborná flotation concentrates, including roasting and alkali leaching. As yet, these processes have been tested at the laboratory scale only.

25.5 Infrastructure and Site Layout

The infrastructure and site plan designs are based on information from published data and from previous work on similar operations. Several assumptions were made in the costing of this portion of the Project, and subsequent testing will provide more accurate data for refining the site plan design and associated infrastructure costs.

The site for the mill and associated operations was chosen for its relative proximity to the ore body and to existing utilities, and for its proximity to potential water source and to the existing TSF.

For the purposes of this study, water sources required for operations will be supplied from the Slaná River.

It is anticipated that the electrical load is approximately 1.5 MW.

The TMS is an existing facility located to the west of the mill. The facility will be re-started after partial clearing/grubbing, contouring (for drainage) and placement of a impermeable liner over the existing upper surface for seepage collection. A small solution recycling pump/pipeline would be installed to collect and pump any recovered water back to the process plant. The upgraded facility will be required to provide secure containment for the planned stacking of 300,000 m³ to 400,000 m³ of tailing filter cake. A tailings density of 3.0 t/m³ was used for the volume calculation and slopes of 1:2 (H:W).

Tailing filter cake will be trucked via a 500 m haulage road from the plant to the storage facility.

25.6 Environmental and Social/Community Impact

The renewal of underground mining at the Strieborná Project using modern technology is expected to bring much needed employment, income, taxation, environmental protection, and social benefits to the local community and to Slovakia in general. Implementation of modern water treatment technology to dewater the flooded underground workings, treat, and discharge the water has been accomplished within the permitting framework of the Slovakian government, and is compliant with the EU standards. Global Minerals operates as a valued and participatory corporate citizen in its operations. The physical, chemical, biological, social, and regulatory aspects of the project are being considered under the various applicable EU environmental policies and related framework documents.

Appropriate controls for treatment of mine dewatering have already been put in place, and are permitted, monitored, and inspected.

25.7 Markets and Smelters

A flotation concentrate grading 27.8% Cu has been assumed for this PEA. Testwork shows that this grade is achievable.

Due to high levels of minor elements such as mercury (7,000 ppm to 10,000 ppm), Antimony (~20%) and arsenic (~1%), direct shipping to a copper smelter is considered uneconomic. Instead, the concentrate is shipped to an industrial recycling facility for low temperature (650°C) roasting. This effectively lowers the mercury concentration to 20 ppm and provides a slight increase in copper and silver grade. The incremental cost of this roasting stage has been estimated to average \$520/t of flotation concentrate, or \$21.50/t of mill feed.

For the PEA, several copper smelters and metals traders were approached for indicative term sheets and together with Q4 2014 market research a set of charges and penalties was constructed for use in financial models and cut-off calculations. Of note, the very high antimony concentration has attracted a very large penalty – equivalent to \$1,004/t of concentrate or \$41.50/t of mill feed. The very high financial penalties for antimony in the calcine means that any improvement in the penalties clause on term sheets from smelters who have expressed an interest in Strieborna concentrate could have a positive impact on the project's economics. Testwork designed to generate flotation concentrate and then calcine is outlined in the recommendations for further work (Section 26.9).

25.8 Economic Analysis

The results of the analysis of the Strieborná Project are encouraging. The project as envisioned provides positive results at the base case metal prices of \$20/oz Ag and

\$3.00/lb Cu. The pre-tax NPV with a 5% discount rate is \$11 million using the base case metal prices. Post-tax, with the same 5% discount rate and metal prices, the NPV is \$5 million. IRR are 12.42% pre-tax, and 8.3% post-tax. Payback on the project capital is expected to be 4.7 years pre-tax, and 5.1 years post-tax. Initial project capital is estimated at \$25.3 million.

The results also show the project's leverage to the higher metal prices, as indicated in the high case, with silver at \$22/oz and copper at \$3.30/lb. These higher metal prices increase the post-tax NPV at a 5% discount rate to \$19 million, with a 17.9% after-tax IRR, more than double the base case.

This assessment is preliminary in nature, as it includes Inferred Mineral Resources that are presently considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the assumptions used in this PEA will be realized.

25.9 Project Risks

Project risks are identified in Table 25-1 along with impact, and mitigation information. In general, identified risks will be mitigated through normal development channels, including the design and execution of more thorough testwork and engineering programs.

Table 25-1: Project Risks, Impact, and Mitigation

Risk	Explanation	Potential Impact	Possible Risk Mitigation
Geology			
Underground sampling	- Almost all – except the recent Level 6 – underground sampling pre-dates Global Minerals's involvement in the project. - There is no current way to verify the results of historic sampling until the lower levels have been pumped out.	There are indications of historic checking that show historic grades to be low.	- Jakubiak (1994) for CMX did a very detailed check sampling program. MDA has been impressed with both the SGS' and CMX's reported work, documentation, and understanding. - Duplicate sampling by CMX on the SGS underground sampling did show some, as yet not understood, high and low biases between the two sets of data. The more predominant bias suggests the sample assays used in this estimate are biased low.
All mineralization of consequence lies within the veins.	- A single vein-domain was used to control the estimate. - Different styles and grades of mineralization exist within the vein, which are difficult to define spatially with the existing data.	- There is inevitable mixing of domains, which will distort grade distributions in any estimate until more detailed sampling is obtained throughout the vein. - The single vein domain has a small high-grade population and a small low-grade population, each of which represents about five percent of the total	- The high-grade sample values' influence in the estimate was restricted. The low-grade samples were not. - MDA performed tens of different estimates attempting to arrive at modelled grade distributions that were both reasonable and without artifacts of the restricted sample distribution.

Risk	Explanation	Potential Impact	Possible Risk Mitigation
Estimation is difficult because of the geometry of the sampled areas.	There is very detailed sampling limited to several levels, and there is good sample spacing in drilling restricted to one area of the vein. The inconsistency of this sampling makes estimation very difficult to portray <i>in situ</i> grade distributions.	population. As a consequence, the estimated grades in the block model likely do not, and probably cannot, reflect expected variable grade changes within the plane of the vein on a local scale.	- The clustering of close-spaced sampling only on development levels presents a good display of distribution of mineralization locally. - MDA ran many estimates to best replicate perceived metal distributions, while honoring the underlying data, and attempting to minimize differences between the three estimates. - Previous resource estimates done over the years generated substantially similar results
Geotechnical			
Definitive geology model has not yet been developed for the project.	- Lack of information outside of immediately accessible mineralized zone - Only the vicinity of Level 6 is currently available for inspection. - Unknown geological structure and changes in ground conditions may be encountered. - This is mitigated to an extent by previous historic mining.	- Increased support costs associated with poorer ground conditions and/or structure that have not been accounted for in the study - Increased development time to sort out geotechnical issues encountered underground delaying the project	- The continued gathering of geotechnical data as described in the geotechnical recommendations. - Enhanced research and review of historic mining documentation.
In Situ Stress regime is currently undefined	The current 'locked-in' ground stress conditions in the vicinity of the mine have not been established.	Elevated, concentrated, and/or varying stress conditions can negatively impact worker safety, ground performance and mining outcomes.	The continued gathering of geotechnical data as described in the geotechnical recommendations.
Groundwater regime is currently undefined	- With increased ground water, instability on discontinuities is more likely - Higher pumping requirements and costs than assumed	- With higher pore pressures, wall stability is affected which may require more rib pillars or larger sill pillars resulting in feed material losses - Increased costs	- Implement a hydrogeological program to better understand regional groundwater conditions - Design dewatering system to reduce risk
Mining			
Rehabilitation Programme	- The rehabilitation and Mária Mine dewatering programme has only recently been defined. - There is risk in the cost and time estimates provided in this study.	Higher capital costs and schedule delays.	Fully define the requirements of the program to identify work content and manpower planning to confirm costs and schedule, including new contractor quotations where necessary.
Employee Training	While the local workforce has experience in	Failure to meet productivity targets.	Consider the temporary employment of skilled non-local mechanized miners.

Risk	Explanation	Potential Impact	Possible Risk Mitigation
	underground mining little of this includes the mechanized equipment and methods used in this study.		- Consider the temporary employment of suitable contractors. - Develop and make cost provisions for an adequate program of training to increase local skill levels. - Select basic mechanized equipment requiring minimum skill levels.
Failure of water tight seals between the Mária and Sadlovsky Mines.	- The water tight seals installed between the Mária and Sadlovsky Mines on Level 13 will remain submerged throughout the life of mine. - These seals therefore cannot be inspected or upgraded. - The quantity of water in Sadlovsky Mine has not been identified but is believed to be substantial. - The after effects on the life of mine programme are unknown.	- Major water inundation of the Mária and Strieborná Mines potentially temporarily halting mine production. - Permit conditions consider only the emergency evacuation of the workforce.	- Remote monitoring of water levels in Mária Mine and automation of effective water level alarm systems. - Attempt to quantify the amount of water in Sadlovsky mine and attempt to estimate the effects of uncontrolled flooding. - Review any potential mitigation methods. - Identify emergency pumping resources.
Infrastructure designs	Infrastructure designs including ventilation and dewatering are not fully defined.	Changes in cost estimates	Address issues in subsequent planning and detailed design
Surface waste dumps	Prior to mid Year 2 all developed waste will be hauled to surface waste dumps.	Any acid generating waste material may require special storage considerations.	Identify any potential ARD material and plan accordingly
Metallurgy			
Sample representativity is limited.	Met predictions based on testing of limited samples.	Metallurgical recoveries may vary from those predicted.	Small variability met program on any future newly drilled material will add confidence to predicted metallurgy
Hg roasting process is based on preliminary work	Small samples were tested in lab-scale equipment	The mercury roasting step could be more expensive or may not work at all	Larger scale work is required. Further testing will gather a larger sample
Hydrometallurgical program is unsuccessful	Testwork is ongoing and subject to normal development uncertainty	Antimony will remain in concentrate/calcine as a penalty element	Additional work must continue.
Processing			
Mill feed grade variability	High grade vein deposit may result in large variations in pay metal grades in mill feed	Flotation circuit may be difficult to control, surging in cleaner circuit, underdosing of reagents, and lower overall recoveries	- Additional variability testwork to be conducted to determine effect of head grade on recovery - Stockpiling and blending of high grade feed to achieve target mill feed grade
Mill feed hardness	Some areas of the	Target grind size may not	Additional grindability testwork

Risk	Explanation	Potential Impact	Possible Risk Mitigation
variability	deposit may contain harder material which is not represented in the composites already tested	be achieved at design throughput	to identify hardness variability in the deposit • Testwork to date indicates that recovery/grade is not strongly influenced by grind size
• Concentrate grade	• Plant may not achieve target concentrate grade	• Higher shipping costs • Lower value at smelter if additional penalties apply	• Potential exists to further optimize the cleaner circuit, including the addition of a regrind to increase concentrate grade
• Tailings filterability	• No testwork completed to date on filtration of the tailings	• Tailings filtration rate may be slower than estimated, creating a bottleneck in the plant • Moisture levels in filter cake may not be achieved, rendering tailings difficult to handle	• Vendor testwork to effectively characterize flotation tailings and size appropriate filtration equipment
• Mercury roasting facility closes	• The roasting facility is unique in Europe	• The concentrate would become less attractive to smelters	• Options for on-site Hg roaster will be looked at as part of project development
Infrastructure			
• Tailing impoundment is not suitable	• Preliminary survey of existing tailing facility only.	• May need to find an alternative site, or • May need to allocate capital to further improvements	• Additional studies and engineering work to better describe the facility
• Foundation conditions are worse than expected	• As the project develops, we become aware of additional complexities	• More expensive/complicated designs • Increased earthworks costs	• Geotechnical work is important and should be budgeted at an early stage
Economic Analysis			
• Metal Price volatility	• Metal prices change with global markets	• Prices could drop as the project comes on line • Revenues will reduce	• Sensitivity analysis • Market research
Environmental/Social			
• Changes in environmental (air, water, soil, noise, vibration, waste, reclamation, etc.) regulations or regulatory interpretation, stricter application of EU Standards, or other more stringent requirements.	• There is always a risk that more requirements may be placed on a project. The fact that the project has been permitted, operated, and monitored for treatment and discharge of mine water is a solid basis for optimism.	• Additional costs may be required to meet increased regulatory requirements.	• Maintain modern monitoring and reporting, keep regulators informed, be aware of potential changes in regulations and regulators.
• Public concerns or demands regarding mining activities and impacts to the community.	• There has been a generally positive response to the proposal to reopen and operate the mining activities, but anti mining forces can come from local, regional, and national centers that do not	• Engagement of anti-mining NGOs or the public may interfere with orderly and efficient start up of mining activities. Road and supply line blockages are not uncommon in the region.	• Keeping key governmental and other public figures informed of the mining plans for environmental and social impact mitigation, operations, economic benefits, and other key aspects. Continued vigilance and interaction with key public figures, administration, regulators,

Risk	Explanation	Potential Impact	Possible Risk Mitigation
	see direct benefits.		reasonable NGOs. Maintain open channels of communication and information flow, and dedication to excellence in modern mining practices.
Violation of discharge standards: changes in quality causing exceedances of trace element levels or other parameters in the mine dewatering discharge, upset of the mine water treatment processing equipment, failure to meet discharge criteria.	There is a risk that the mine dewatering may encounter difficult to treat water, or that the treatment process is not able to maintain proper discharge quality.	Discharge exceedances would cause increased concern over the normal operations of the project, especially the dewatering of the mine over the long term.	Careful monitoring, inspections of equipment, diligent operations of the mine dewatering process, identification and use of additional treatment alternatives.
Smelters			
The flotation concentrate carries high levels of penalty elements	- Mercury can be as high as 1% - Antimony is close to 20% - Arsenic is high, but not problematic	Significant penalties charged by toll smelters will reduce revenues.	- Roasting concentrate will reduce mercury to <20 ppm - Hydrometallurgical or pyrometallurgical options for processing can turn Antimony into a product

25.10 Project Opportunities

At this early stage in the project life cycle, several opportunities are normally identified for potential exploitation in downstream studies. A preliminary list of opportunities for Strieborná is given Table 25-2.

In AGP's opinion, a key opportunity for the Strieborná Project stands out as being very valuable if realized. If the hydrometallurgical testwork program described in Section 13 can pilot and prove conditions for an antimony removal step prior to smelting, then project economics will see dramatic improvement. The project cash flow models currently include a smelter penalty attributable to antimony of \$41.50/t of ore processed. If this could be reduced through partial antimony removal, this penalty would decrease, but in addition the revenues received for an antimony product would further improve project profitability.

As an alternative to the above, the current low temperature roasting process (used to reduce mercury concentrations) could be developed to include a higher temperature roasting step to drive off sulphur and antimony. The antimony and sulphur oxide vapours could be collected and processed to provide similar effects to the hydrometallurgical option—lower smelter penalties and higher metals revenues.

Table 25-2: Project Opportunities

Opportunity	Explanation	Potential Benefit
Geology		
Potential to increase the	Additional veins are identified by historic	The potential to increase resources is

Opportunity	Explanation	Potential Benefit
resources	work, but these are either not included, or are included as Inferred due to insufficient data.	considered very good. Once done, the conversion of these to reserves will improve the economics of the project. The similarities to the Couer D'Alene district suggest that the potential could be quite large.
Geotechnical		
Increased rock mass competency and strength	Increased rock mass competency and strength may allow for larger mining geometries and increased spacing on rib pillars	Higher mine recovery
limited occurrence of discontinuities, faults, intersecting mine workings, stopes	If the rock has fewer fractures or shears/faults the stability of the underground workings will be improved	- Improved safety for workers due to rockfalls - Potential reduced dilution from hangingwall and footwall material
Mining		
Identification of additional mineable resources	The Strieborná vein extends beyond the currently planned area both on strike and at depth.	There is potential to extend mine life or rate of production.
Increase rate of production	Planned production rates were requested by Global. With additional planning the deposit may be capable of a higher rate of production, subject to mine permitting conditions.	Improved economics
Change shift schedule	Eight hour working shifts are considered in this study, longer shifts may provide benefits.	- More underground working time per shift - Reduction in the number of employees required. - Reduction in training requirements.
Metallurgy and Processing		
Gravity pre-concentration	Strieborná type mineralization is often amenable to pre-concentration at a size that only needs crushing.	Could reduce mill size, could allow for concentration underground hence circumventing any hoisting capacity limitations. Could drop cut-off grade.
On-site concentrate processing	On-site processing to create a marketable antimony product could also allow for relatively cheap mercury removal.	Could turn antimony from smelter charge into a revenue stream. Could allow for cheaper mercury removal than assumed at this point.
Regrind optimisation	Regrind size and effect of regrinding has not been fully demonstrated.	Slight improvement to metallurgy could arise from better defined regrinding.
Hydrometallurgical testwork	The hydromet program has examined the leaching of antimony from flotation concentrates. If successful, this transforms antimony from a penalty element to a by-product revenue item.	- Additional revenues - Lower smelter penalties - High grade copper product
Pyrometallurgical testwork	- The Hg roasting process could be expanded to attempt to remove antimony from concentrates after mercury removal. - Antimony oxide could be sold as a by-product - The lower antimony % in concentrate would attract lower penalties in the smelter	- Additional revenues - Lower smelter penalties - High grade copper product
Infrastructure		
Foundation conditions for infrastructure	Foundation conditions could be better than expected.	Reduced capital costs
Economic Analysis		
Metal Prices	Metal prices could be much higher as the project begins operation	Additional revenues

Opportunity	Explanation	Potential Benefit
Markets and Smelter		
Antimony in flotation concentrates	Currently this element results in a significant processing penalty	Hydrometallurgical or Pyrometallurgical processes under development could turn this element into a saleable product, thereby increasing revenue.
Term sheets from other smelters	Other smelters may offer better terms	Reduced charges and penalties

26 RECOMMENDATIONS

26.1 Introduction

Based on the studies completed for this PEA, AGP recommends that Global Minerals proceed to the next level of study for the Strieborná Project.

Specific recommendations by area are discussed in the following sections.

26.2 Exploration and Resource Development

From an exploration and resource development standpoint, Global Minerals should expand the resources along strike on the Strieborná and Podložná veins, and continue to infill drill for better definition.

26.2.1 Exploration

While the Strieborná vein is open to the northeast, southwest, up and down dip, drilling is complicated, to impossible, due to limited access and a steep dip. Global Minerals could drill four holes of 500 m each to intersect down-dip extensions of Strieborná, or strike extensions of Podložná, for a cost of \$300,000. Regardless, once access is constructed for production or for additional development, a large exploration-drilling program to test the extensions of Strieborná and Podložná is warranted.

26.2.2 Infill and Development

The next of several steps is to establish two new drill stations at 100 m to 200 m to the southwest of the existing drill stations on Level 6. The “all-in” cost for drifting (\$250,000) and drilling (\$450,000) is \$700,000. Also, the up-dip extension of the vein, above Level 6, could easily be targeted from the surface with holes ranging from 150 m to 200 m in length. The company reports that they have permitted three drill stations to test the up-dip extension of the vein along strike and to the southwest. This exercise would increase the Measured material substantially and increase the Indicated material slightly. Also, Global Minerals should continue to develop the Strieborná vein from Level 6 to explore along the vein. Global Minerals should continue to dewater and rehabilitate the shaft to access Level 8 down to Levels 10 and 13, which would allow for additional infill and exploration drilling opportunities.

26.3 Geotechnical

The data available at this stage of study varies in reliability. Estimates of engineering properties are provided with ranges where possible and sensitivity analyses are encouraged for mine design based on these data. The interpretations presented within this report are preliminary and require additional validation and testing with higher

quality data before they can be applied to more intensive design studies. Recommended data collection and interpretation tasks are outlined below.

Geological Model

The geological model was incomplete at the time of this study, and contained limited information regarding geological structures; particularly the location, orientation, and geotechnical characteristics of major structures (fault and shear zones). The completion of core orientation and down hole geophysics would significantly improve the structural interpretations for the project area. A complete geological model and structural model should be considered necessary components at the next level of study.

Geotechnical Core Logging

Dedicated geotechnical core holes should be drilled targeting the proposed mining excavations. It is recommended that all geotechnical core holes be logged by a suitably qualified person with experience in geotechnical logging. These drill holes should target hanging wall, footwall, and mineralized rock masses, and should be drilled using a triple tube core barrel to minimize drilling disturbance. Core orientation techniques (i.e. ACTIII and/or Borehole televiewer) should be employed to determine geologic discontinuity orientations. A detailed review of the alteration types should be completed prior to initiation of the geotechnical core logging program, so that the logging techniques employed can allow review of the potential influence of alteration and lithology on the rock mass.

Point Load Testing

Experience at other mining properties has indicated that alteration may have a significant impact on the intact rock strength. The potential for further division of the geotechnical units according to alteration may be evaluated through a point load testing program. The point load test is a simple and rapid method of determining an index value which can be related to the intact strength of a rock sample. Global could undertake a point load testing program as part of its next exploration drilling program. The additional effort is minimal, and would require the rental or purchase of a point load test machine, selection of samples according to lithology or alteration, and completion of testing according to established procedures. BGC can provide further support in developing the testing specifications and database to store the results. Staff can also be supplied to undertake the required work.

Laboratory Testing

Uniaxial compressive strength (UCS) testing, Brazilian tensile strength testing, triaxial compressive strength testing, direct shear testing of discontinuities, and index testing of discontinuity infill should be conducted on select samples to support rock mass classification and derivation of design strengths. These samples should be collected from

dedicated geotechnical drill holes to ensure appropriate materials are sampled, and to avoid conflicts with exploration sampling and assaying.

Hydrogeologic Evaluations

Hydrogeological testing (e.g., packer testing) and instrumentation (e.g. piezometers) should be installed in select drill holes to provide data for groundwater modelling and excavation dewatering simulations. This information will be used in subsequent geotechnical evaluations to determine the feasibility of dewatering the proposed excavations.

In Situ Stress Study

There is currently very limited information available with regards to the in situ stresses in the project area. In situ stress testing is generally expensive and it can be difficult to collect enough data for a statistically significant dataset.

Completion of Borehole televiewer surveys will allow inspection of the survey results for borehole breakouts. Borehole breakouts, if present, can provide information on the in situ stresses and could be used to refine the in situ stress estimates assumed for designs.

Revision of Current Designs

The current designs are provided for general guidance to the PEA-level mine design and for cost estimating purposes. As additional geological and geotechnical work is completed, the current designs should be reviewed and updated as required. Concurrently, if the proposed mining method or mine plan changes considerably from the one assumed in this report, the assumptions inherent within the rock mechanics assessment should be reviewed and updated as required.

26.4 Mining

AGP believes that the preliminary mine plans and cost estimates in this study demonstrate the potential for economic development of the Strieborná deposit. This study includes the potential to mine inferred resources. AGP recommends that Global Minerals undertakes additional studies to a prefeasibility or feasibility level of study that allows the estimation of a mineral reserve, confirmation of economic viability, and an investment decision on deposit development. AGP recommends that these additional studies include:

- potential to adapt stope designs based on updated geotechnical data with the aim of increasing resource recovery
- undertake studies that will allow definition of the mine ventilation
- better identify the dewatering requirements and methodology for the currently flooded areas of the existing Mária Mine workings

- new quotations be sought for rehabilitation of the existing workings and shafts in line with the requirements and schedule outlined in this report
- attempt to quantify the water contained in the adjacent Sadlovsky Mine and mitigate the potential effects of water incursion from this property
- as the design of mine systems and infrastructure requirements are more clearly identified, quotations should be sought to confirm the adequacy of allowances included in this study
- confirm the suitability of equipment selection and costs used in this study in order that quoted cost rates can be used in ongoing cost estimation
- investigate and identify adequate training systems to provide the necessary skill levels required for mechanized mining
- cost estimates were prepared assuming the underground mine would operate on a three 8-hour shifts per day basis; the potential to increase shift length on a two shifts per day basis should be investigated.

26.5 Metallurgy and Process

26.5.1 Metallurgy

The preliminary flotation program described in this report has generated excellent results and is therefore considered to represent little risk to project success. Further, base case downstream processing options which involve third party treatment for mercury removal and subsequent sale of the mercury-free product to smelters, appears viable based the preliminary level of marketing conducted to date. However, more definitive terms should be sought. Further, alternative pyrometallurgical and/or hydrometallurgical processing options represent potential for substantial upside and must be investigated further.

The following metallurgical testwork on the Strieborná deposit is recommended as the project advances into the next phase of development:

- Limited further mineral processing testing to arrive at the optimum flowsheet for the available bulk sample of Phase 1 material. This work should include regrind optimization and iron removal testwork.
- Pilot plant testing to generate a large amount of product for more definitive third-party evaluation, solid-liquid separation, and downstream hydromet/pyromet testing.
- Further testwork by Hg-removal contractors, specifically to (a) reduce the cost and simplify the process so allowing for a reduction in toll treatment charges and (b) to reduce the chemical effect on the remainder of the tetrahedrite concentrate during mercury removal. This would potentially allow for antimony recovery by the project after mercury removal. Specifically nitrogen-based roasting should be explored to see if the mercury can be fumed off, without

forming the insoluble antimony oxide product tripuhyite. This should allow for the use of the proven and well-understood Sunshine antimony removal process on the mercury-free product.

- Should on-site roasting be deemed an option, further work would be needed to evaluate this.
- Further development of the on-site processing option, including antimony/mercury leach with subsequent precipitation of mercury for third party disposal and electrowinning of antimony metal for sale.
- Should either of the above show potential to enhance project economics, they would need closed circuit testing to develop a complete water-balance and to tighten down process economics.

26.5.2 Processing

The PEA processing flowsheet and process design criteria is based on preliminary testwork. As the metallurgical testwork program progresses to include larger, more representative samples and more detailed scope, then the process designs and equipment selections can also be developed.

The next process plant design should include mechanical equipment lists, multiple process flow diagrams, more detailed process design criteria, and mass and water balance calculations. More advanced equipment specifications and selection criteria plus layout drawings and material take-off schedules will assist in reducing capital and operating cost estimation error.

26.6 Infrastructure and Site Layout

Additional testing and data collection is required to define the infrastructure and site layout requirements and associated costs in these areas:

- geotechnical testing and data to further develop construction requirements for the mill, water supply, and services sites
- hydrogeology and water quality testing on water sources surrounding the mine site to confirm available volumes and quality of water required to support the mill and services infrastructure
- materials testing and quarry location scoping for design and construction cost estimates
- foundation testing and material suitability studies need to be completed for the plant site and water retention pond locations

- archeological studies
- verification that existing municipal sewer and water lines have sufficient capacity for the project.

26.7 Water Management

A complete hydrogeological review of the mine and process plant areas is required as the project advances. The review should cover the following topics:

- water quality and quantity for processing facilities and services infrastructure
- mine seepage evaluations
- determination of precipitation and drainage areas for estimation of drainage/diversion facilities
- hydrogeological evaluation of the tailing storage facility, including surface runoff, ground water and seepage evaluations
- creation of a project-wide water balance.

26.8 Environmental and Social / Community Impact

A detailed environmental impact assessment process is required by the Slovakian government and is under preparation at time of writing this report. Potential negative social and environmental impacts will be identified, and appropriate steps taken to eliminate, minimize, mitigate, or ameliorate these impacts. Appropriate controls for treatment of mine dewatering have already been put in place, are permitted, monitored, and inspected. Detailed operational planning and engineering for air quality protection, erosion and sediment control, surface water and groundwater quality, waste management, spill prevention control and countermeasures, tailings management, emergency response, community relations, worker health and safety, security, traffic, noise, dust, vibration, and other aspects will be incorporated into the overall SEMMP to be specifically applied to the Strieborná Project. A comprehensive bankable feasibility study to international standards is anticipated. The project is expected to be operated in a safe and environmentally and socially responsible manner within a modern sustainability framework using advanced approaches, technology, protocols, and management.

26.9 Markets and Contracts

Given the financial penalties for antimony in the concentrate are so high, any improvement in the quotes from smelters who have expressed an interest in Strieborna concentrate could have a positive impact on the project's economics.

Global has approximately 600 tonnes of mineralized material stockpiled at the Maria mine site remaining from the 2014 bulk sample exercise. A portion of this should be crushed and processed by pilot plant to make a flotation concentrate. This concentrate

sample should be used to conduct large scale mercury and antimony roasting tests and the subsequent calcines should be sent to smelters for internal laboratory testing to establish commercial-level treatment terms.

The budgets given in Section 26.10 for testwork to support project development include the cost of pilot processing.

Of note, if the hydrometallurgical testwork program is able to develop further inroads into the antimony recovery process, then this metal could switch from being a penalty element to a revenue stream. Development of this process option is recommended.

26.10 Recommendations and Estimated Budget

Table 26-1 shows a summary of recommendations per discipline and the total costs to advance to the next stage of development.

Table 26-1: Summary of Recommendations and Total Costs

	Recommended Budget (\$)
Geology	1,000,000
Geotechnical	700,000
Mining	600,000
Metallurgy	500,000
Infrastructure	200,000
Water Management	100,000
Environmental	200,000
Total	3,300,000

27 REFERENCES, UNITS OF MEASURE, ABBREVIATIONS AND ACRONYMS

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27.2 Units of Measure

Above mean sea level	amsl
Acre	ac
Annum (year)	a
Centimetre cubic	cm ³
Centimetre	cm
Coefficients of Variation	CVs
Day	d
Degree	°
Degrees Celsius	°C
Dollar (American)	US\$
Dry metric tonne	dmt
Foot	ft
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m ²)	ha
Hour	h
Hours per day	h/d
Inch	"
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre	kg/m ³
Kilograms per hour	kg/h
Kilograms per square metre	kg/m ²
Kilometre	km
Kilometres per hour	km/h
Kilopascal	kPa
Kilovolt	kV
Kilovolts	kV
Kilowatt hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Kilowatt	kW
Less than	<
Litre	L
Litres per minute	L/min
Megawatt	MW
Metre cubic per second	m ³ /sec
Metre cubic	m ³
Metre	m
Metres above sea level	masl
Metres per minute	m/min
Metres per second	m/sec
Metric ton (tonne)	t

Microns.....	µm
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million tonnes.....	Mt
Million.....	M
Minute (plane angle)	'
Minute (time)	min
Ounce	oz
Parts per million	ppm
Percent	%
Revolutions per minute	rpm
Second (plane angle)	"
Second (time)	sec
Specific gravity.....	SG
Square kilometre	km ²
Square metre	m ²
Tonne (1,000 kg).....	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a

27.3 Abbreviations and Acronyms

Acid Base Accounting	ABA
Acid Rock Drainage.....	ARD
AGP Mining Consultants Inc.	AGP
Atomic Absorption Spectrophotometer.....	AAS
Atomic Absorption.....	AA
Canadian Institute of Mining, Metallurgy, and Petroleum.....	CIM
Carbon-in-Leach	CIL
Coefficient of Variation	CV
Environmental Management and Monitoring Program	SEMMP
General and administration.....	G&A
Global Minerals Limited	Global Minerals
Inductively Coupled Plasma Atomic Emission Spectroscopy.....	ICP-AES
Inductively Coupled Plasma.....	ICP
Methyl Isobutyl Carbinol	MIBC
Mine Development Associates	MDA
National Instrument 43-101	NI 43-101
Qualified Persons.....	QPs
Quality assurance	QA
Quality control.....	QC
Rock Quality Designation.....	RQD
The Slovak Geological Survey	The SGS

X-Ray Fluorescence Spectrometer	XRF
Acid Base Accounting	ABA
Acid Rock Drainage	ARD
ALS Minerals	ALS
Atomic Absorption	AA
Blue Coast Research	BCR
Bond Ball Work Index	BWi
CIM Definition Standards for Mineral Resources and Reserves, Definitions, and Guidelines	CIM Standards
CMX Resource Limited	CMX
Controlled Source Audio-Frequency Magnetotellurics	CSMAT
Feichtner Consulting Engineers GmbH	FCE
GEO-TECHNIC Consulting spol. s.r.o.	Geo-Technic
Greenhouse Gas	GHS
Induced Polarization	IP
International Labor Organization	ILO
Kingston Process Metallurgy Inc.	KPM
Liquid Methyl Isobutyl Carbinol	MIBC
National Instrument 43-101	NI 43-101
Process Mineralogical Consulting	PMC
Rock Quality Designation	RQD
SAMAX Resources Ltd.	SAMAX
Semi Quantitative XRD	SQ-XRD
Sunshine Mining and Refining Company	Sunshine Mining
United States Dollars	US\$
University of British Columbia	UBC
Východoslovenská energetika	VSE
Xstrata Process Support	XPS
Železorudné Bane š.p.	ŽELBA

28 CERTIFICATES



28.1 Steven Ristorcelli

To Be inserted as PDF



28.2 Andrew Holloway, CEng, P.Eng.

I, Andrew Holloway, CEng., P.Eng., of Peterborough, Ontario, do hereby certify that as one of the authors of this updated technical report titled “Technical Report and Preliminary Economic Assessment for the Strieborná Project, Slovakia” dated January 26, 2015 I hereby make the following statements:

- I am a Principal Process Engineer with AGP Mining Consultants Inc. with a business address at 92 Caplan Ave., Ste. #246, Barrie, Ontario, Canada, L4N 0Z7.
- I am a graduate of the University of Newcastle upon Tyne, England, B.Eng. (Hons.), 1989, and I have practised my profession continuously since then.
- I am a Professional Engineer licensed by Professional Engineers Ontario (Membership Number 100082475).
- I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101 or the Instrument) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.
- My relevant experience with respect to metallurgy and mining project management includes 23 years’ experience in the mining sector covering mineral processing, process plant operation, design engineering, operations and project management. I have been involved in numerous projects around the world in both base metals and precious metals deposits.
- I am responsible for the content of Sections 1.1, 1.2, 1.9.1, 1.9.2, 1.12.6, 1.12.8, 2, 3, 4, 5, 6, 15, 19, 21.1.1, 21.1.5, 21.1.6, 21.1.7, 21.2.1, 21.2.4, 23, 24, 25.7, 25.9, 25.10, 26.1, 26.7, 26.9, 26.10, and 27 of this technical report.
- I have not conducted a site visit of the property.
- I am independent of the issuer as defined by Section 1.5 of the Instrument.
- I have read NI 43-101 and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of this Technical Report, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Signed and dated this 23rd day of January 2015, at Barrie, Ontario.

“Original Document Signed and Sealed”

Andrew Holloway, CEng., P.Eng.

28.3 Geoffrey Challiner, CEng.

I, Geoffrey Challiner, CEng. of Cheshire, UK, do hereby certify that as one of the authors of this updated technical report titled “Technical Report and Preliminary Economic Assessment for the Strieborná Project, Slovakia” dated January 26, 2015 I hereby make the following statements:

- I am an Associate Mining Engineer with AGP Mining Consultants Inc. with a business address at 92 Caplan Ave., Ste. #246, Barrie, Ontario, Canada, L4N 0Z7.
- I am a graduate of Royal School of Mines, London, UK (1975), and I have practised my profession continuously since then.
- I am a member in good standing of the Institute of Materials, Minerals & Mining, UK (Registration Number 45202).
- I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101 or the Instrument) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.
- My relevant experience with respect to the project comprises over thirty years’ experience in mining engineering and operations, specializing in underground mines, including gold and precious metals deposits.
- I am responsible for the content of Sections 1.5, 1.6, 1.12.2, 1.12.3, 16, 21.1.2, 21.2.2, 25.2, 25.3, 26.3, and 26.4 of this Technical Report.
- I have conducted a site visit of the property on March 11 to 14, 2013.
- I am independent of the issuer as defined by Section 1.5 of the Instrument.
- I have read NI 43-101 and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of this Technical Report, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Signed and dated this 23rd day of January 2015, at Cheshire, UK.

“Original Document Signed and Sealed”

Geoffrey Challiner, CEng.

28.4 Gordon Zurowski, P.Eng.

I, Gordon Zurowski, of Stouffville, Ontario, do hereby certify that as one of the authors of this updated technical report titled “Technical Report and Preliminary Economic Assessment for the Strieborná Project, Slovakia” dated January 26, 2015 I hereby make the following statements:

- I am a Principal Mine Engineer with AGP Mining Consultants Inc., with a business address at 92 Caplan Avenue, Suite 246, Barrie, Ontario, L4N 0Z7.
- I am a graduate of University of Saskatchewan, B.Sc. Geological Engineering, 1989, and I have practised my profession continuously since then.
- I am a Professional Engineer licensed by Professional Engineers of Ontario (Membership #100077750).
- I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101 or the Instrument) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.
- My relevant experience with respect to mining project engineering and management includes the design and evaluation of open pit mines for the last 20 years. I have been involved in numerous projects around the world in both base metals and precious metal deposits.
- I am responsible for the content of Sections 1.10, 22, and 25.8 of this technical report.
- I have not conducted a site visit of the property.
- I am independent of the issuer as defined by Section 1.5 of the Instrument.
- I have read NI 43-101 and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of this Technical Report, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Signed and dated this 23rd day of January 2015, at Stouffville, Ontario.

“Original Document Signed and Sealed”

Gordon Zurowski, P.Eng.

28.5 Mario Colantonio, P.Eng.

I, Mario Colantonio, P.Eng, of Timmins, Ontario, do hereby certify that as one of the authors of this updated technical report titled “Technical Report and Preliminary Economic Assessment for the Strieborná Project, Slovakia” dated January 26, 2015 I hereby make the following statements:

- I am an Engineering Manager/Principal with Porcupine Engineering Services Inc. with a business address at 316 Spruce St. South, Timmins, Ontario, P4N 2M9.
- I am a graduate of Queens University, Civil Engineering Program, 1985, and have practised my profession continuously since then.
- I am a Professional Engineer licensed by Professional Engineers of Ontario (Membership #8869554).
- I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101 or the Instrument) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.
- My relevant experience with respect to project infrastructure includes the design and construction review of mining and milling infrastructure systems working as a consulting engineer for 28 years in the mining industry. I have held the position of engineering manager for consulting engineering firms in Northern Ontario for the last 18 years and in that role I have overseen the work of teams of mechanical, electrical, civil, and structural engineers on several mine and mill projects.
- I am responsible for the contents of Sections 1.8, 1.12.5, 18, 21.1.4, 25.5, and 26.6 of this Technical Report.
- I have conducted a site visit of the property on August 2 to 3, 2014.
- I am independent of the issuer as defined by Section 1.5 of the Instrument.
- I have read NI 43-101 and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of this Technical Report, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Signed and dated this 23rd day of January 2015, at Timmins, Ontario.

“Original Document Signed and Sealed”

Mario Colantonio, P.Eng.

28.6 Lyn Jones, P.Eng.

I, Lyn Jones of Peterborough, Ontario, do hereby certify that as one of the authors of this updated technical report titled “Technical Report and Preliminary Economic Assessment for the Strieborná Project, Slovakia” dated January 26, 2015 I hereby make the following statements:

- I am a Senior Metallurgical Associate with AGP Mining Consultants Inc. with a business address at 92 Caplan Avenue, Ste. #246, Barrie, Ontario, L4N 0Z7.
- I am a graduate of the University of British Columbia, Metals and Materials Engineering, 1998 and I have practised my profession continuously since then.
- I am a Professional Engineer licensed by Professional Engineers of Ontario (Membership Number 100067095).
- I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101 or the Instrument) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.
- My relevant experience with respect to metallurgy and process design includes 14 years’ experience in plant operations, metallurgical testing, flowsheet development, and process engineering.
- I am responsible for the contents of Sections 1.7.4, 17, 21.1.3, 21.2.3, 25.4.4, and 26.5.2, of this Technical Report.
- I have conducted a site visit of the property on April 12 to 13, 2014.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have read NI 43-101 and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of this Technical Report, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Signed and dated this 23rd day of January 2015, at Peterborough, Ontario.

“Original Document Signed and Sealed”

Lyn Jones, P.Eng.

28.7 Christopher J. Martin, M.Eng, CEng.

I, Christopher J. Martin, M.Eng., CEng., of Nanoose Bay, British Columbia, do hereby certify that as one of the authors of this updated technical report titled “Technical Report and Preliminary Economic Assessment for the Strieborná Project, Slovakia” dated January 26, 2015 I hereby make the following statements:

- I am a Principal Metallurgist with Blue Coast Metallurgy Ltd. with a business address at 1020 Herring Gull Way, Parksville, British Columbia Canada, V9P 1R2.
- I am a graduate of the Camborne School of Mines, Camborne, England, B.Sc. (Hons.), 1984 and McGill University, Montreal Canada (M.Eng), 1988, and I have practised my profession continuously since then.
- I am a Chartered Engineer registered with the Engineering Council of the United Kingdom.
- I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101 or the Instrument) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.
- My relevant experience with respect to mineral processing and metallurgy includes 27 years’ experience in the mining sector covering mineral processing and hydrometallurgy, process plant operation, design engineering, and management. I have been involved in numerous projects around the world in both base metals and precious metals deposits.
- I am responsible for the contents of Sections 1.7.1, 1.7.3, 1.12.4, 13 (except 13.3.5), 25.4.1, 25.4.2, 25.4.3, and 26.5.1 of this technical reports.
- I have not conducted a site visit of the property.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have read NI 43-101 and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of this Technical Report, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Signed and dated this 23th day of January 2015, in Parksville, British Columbia.

“Original Document Signed and Sealed”

Christopher J. Martin, M.Eng, CEng.

28.8 Phillip Mackey, Ph.D., P.Eng.

I, Phillip Mackey, Ph.D., P.Eng., of Kirkland, Quebec, do hereby certify that as one of the authors of this updated technical report titled “Technical Report and Preliminary Economic Assessment for the Strieborná Project, Slovakia” dated January 26, 2015 I hereby make the following statements:

- I am President of P.J. Mackey Technology Inc., with a business address at 295 Kirkland Blvd, Kirkland, Québec, H9J 1P7.
- I am a graduate of the University of New South Wales, Australia, Ph.D. in 1969, and I have practised my profession continuously since then.
- I am a Professional Engineer licensed by Professional Engineers Ontario (Membership Number 28236016).
- I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101 or the Instrument) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.
- My relevant experience with respect to mineral processing includes over 40 years’ experience in the mining and metallurgical sector covering: copper smelting operations, pyrometallurgical plant design, research and development work in non-ferrous metallurgy, and work on nickel (both sulphides and laterites), zinc and lead metallurgy, as well as precious metals. I have been involved in a number of metallurgical projects around the world in copper and nickel and have been involved in several due diligence activities.
- I am responsible for the content of Sections 1.7.2 and 13.3.5 of this Technical Report.
- I have not conducted a site visit of the property.
- I am independent of the Issuer as defined by Section 1.5 of the Instrument.
- I have read NI 43-101 and this Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- As of the effective date of this Technical Report, to my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Signed and dated this 23rd day of January 2015, in Kirkland, Québec.

“Original Document Signed and Sealed”

Phillip Mackey, Ph.D., P.Eng.

28.9 John G. Aronson, B.S., M.S., CFS, CSE, CEP

I, John G. Aronson, B.S., M.S., CFS, CSE, CEP of Fort Collins, Colorado, USA, do hereby certify that as one of the authors of this updated technical report titled “Technical Report and Preliminary Economic Assessment for the Strieborná Project, Slovakia” dated January 26, 2015, I hereby make the following statements:

- I am a President of AATA International Inc. with a business address at 1400 Wewatta Street, Suite 310, Denver, Colorado, USA 80202.
- I am a graduate of the Nebraska Wesleyan University, Lincoln, Nebraska (B.S. Biological Sciences, With Highest Distinction), and the University of Nebraska-Lincoln (M.S. with Minor in Civil Engineering).
- I am a Certified Environmental Professional (CEP) with the Academy of Board Certified Environmental Professionals (<http://www.abcep.org>) a Certified Senior Ecologist (CES) with the Ecological Society of America, and a Certified Fisheries Scientist (CFS) with the American Fisheries Society.
- My relevant experience with respect to environmental and social sciences and impact assessment includes 40+ years’ experience in all types of mines and mineral processing facilities around the world. I have been involved in numerous projects around the world in both base metals and precious metals deposits.
- I have practised my profession continuously since 1973.
- I am responsible for the content of Section 1.11, 1.12.7, 20, 25.6, and 26.8 of this Technical Report.
- I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101 or the Instrument) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purpose of NI 43-101.
- I have not conducted a site visit of the property.
- I have no prior involvement with the property that is the subject of this Technical Report.
- I am independent of the issuer as defined by Section 1.5 of the Instrument.
- I have read National Instrument 43-101 and the Technical Report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1.
- As of the effective date of the Technical Report, to my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 23rd day of January 2015 in Washington, D.C., USA.

“Original Document Signed and Sealed”

John G. Aronson, B.S., M.S., CFS, CSE

