Report to:

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KREMNICA GOLD PROJECT PRE-FEASIBILITY STUDY

Kremnica, Slovak Republic, Europe

Prepared by:

BEACON HILL CONSULTANTS (1988) LTD.
July 5, 2007

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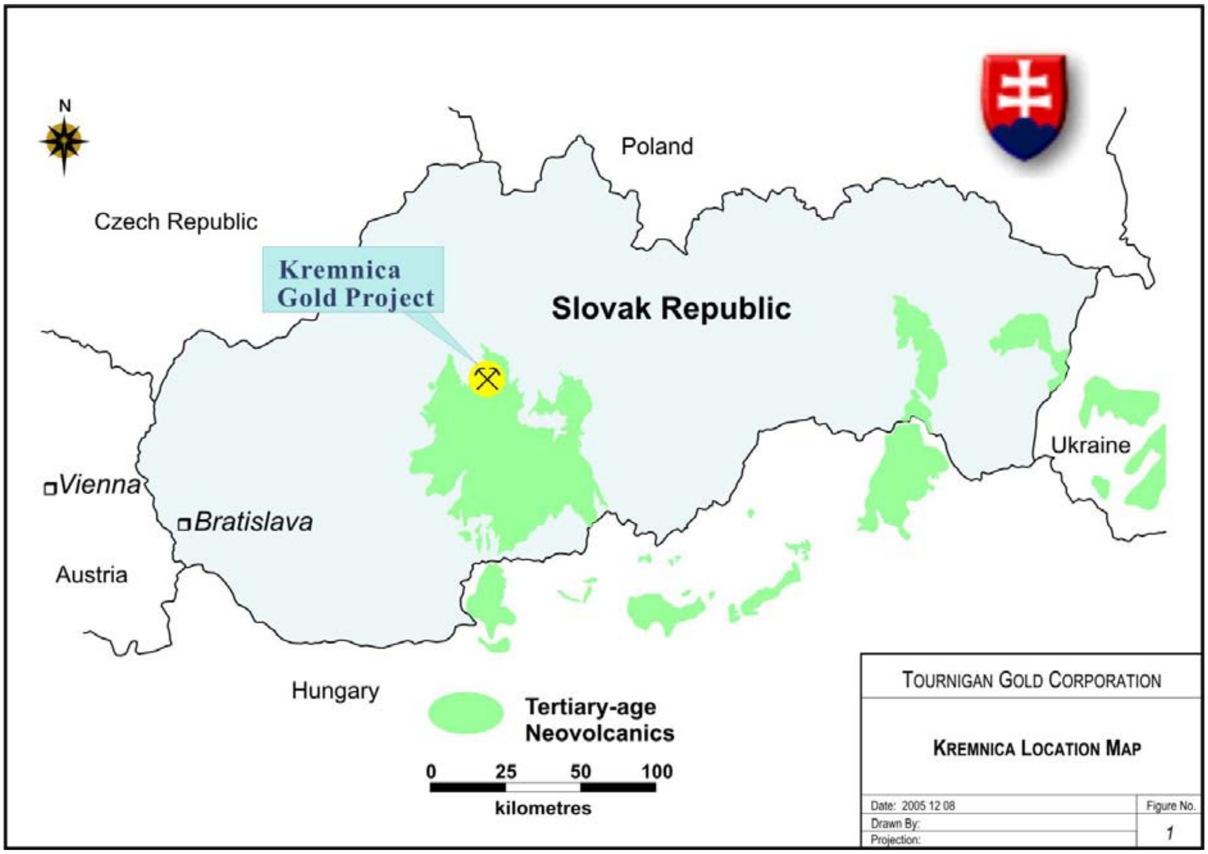
SECTION 1.0 – SUMMARY

1.1 PROJECT DESCRIPTION

The Kremnica project is located in the central Republic of Slovakia, as shown on Figure 1.1. The town of Kremnica lies 17 km west of central Slovakia's largest city, Banská Bystrica.

The Kremnica project comprises two contiguous properties: the Kremnica Mining License and the Lutila Exploration License. These licenses are 11.79 km^2 and 86.38 km^2 in area, respectively.

Figure 1.1: Kremnica Location Map



Tournigan Gold Corporation (Tournigan) commissioned Beacon Hill Consultants (1988) Ltd. (Beacon Hill) to complete a study on the Šturec deposit and prepare a pre-feasibility report on which the viability of the Kremnica project could be evaluated. The study established the reserves exploitable by open pit mining methods, as well as the waste-to-ore ratio and process feed grades, as shown on Table 1.1.

Table 1.1: Kremnica Production Schedule – 6,000 tpd Conveyor

Kremnica Gold Corp
Šturec Deposit
Production Schedule
6000tpd Conveyor

	Pre-strip	1	2	3	4	5	6	7	8	Total
Ore Mined tonnes	203,844	2,296,024	3,019,971	2,909,891	2,731,668	2,373,270	2,247,535	446,955		**16,229,158**
High Grade Stockpile tonnes	145,656									**145,656**
Low Grade Stockpile tonnes	58,188	196,024	919,971	809,891	631,668	273,270	147,535	27,576		**3,064,123**
HG Stockpile Processed tonnes								145,656		**145,656**
LG Stockpile Processed tonnes								1,534,965	1,529,158	**3,064,123**
Ore Processed tonnes		**2,100,000**	**2,100,000**	**2,100,000**	**2,100,000**	**2,100,000**	**2,100,000**	**2,100,000**	**1,529,158**	**16,229,158**
Waste tonnes	2,472,115	3,154,978	2,613,322	3,526,495	7,141,734	5,692,427	1,519,381	53,426		**26,173,878**
Total Tonnes Mined	**2,675,959**	**5,451,002**	**5,633,293**	**6,436,386**	**9,873,402**	**8,065,697**	**3,766,916**	**500,381**		**42,403,036**
Waste: Ore Strip Ratio	12.13	1.37	0.87	1.21	2.61	2.40	0.68	0.12		**1.61**
Gold Grade Mined gAu/t	1.11	1.74	1.13	1.12	1.19	1.56	1.76	1.99		**1.40**
Silver Grade Mined gAg/t	5.06	16.00	8.27	7.45	9.27	12.08	14.90	17.69		**11.08**
Gold Grade Stockpile gAu/t	1.11	0.52	0.55	0.54	0.53	0.52	0.50	0.48		**0.53**
Silver Grade Stockpile gAg/t	5.06	6.47	4.45	4.75	5.82	7.25	7.95	8.68		**5.39**
Gold Grade Processed gAu/t		**1.85**	**1.39**	**1.34**	**1.39**	**1.69**	**1.85**	**0.90**	**0.53**	**1.40**
Silver Grade Processed gAg/t		**16.89**	**9.94**	**8.49**	**10.31**	**12.70**	**15.39**	**7.97**	**5.39**	**11.08**

The overall design of the mine and site facilities was completed with input from local inhabitants to ensure it is acceptable to all parties and adheres to all relevant safety and environmental requirements. To the extent possible, the facilities were designed to blend into the landscape to minimize the visual effect of the mine on the surrounding community.

The daily production rate of the mine is 6,000 tpd ore. The mine operates on 6 days per week, with two 8-hour shifts per day, for a total of 302 days per year (312 days less 10 days statutory holidays). The process plant operates 24 hours per day for 350 days per year.

Ore and waste are excavated using conventional drilling, blasting, and hauling techniques with diesel-powered equipment. The ore is trucked to the primary crusher where it is dumped as feed for the process plant. High grade ore, above 0.75 gAuEq/t, produced during the pre-production period will be placed on a high-grade stockpile for processing at a later date. Low grade material, above 0.5 gAuEq/t will be mined separately and placed on a low grade stockpile to be processed after the high grade ore has been processed.

The process plant utilizes a conventional gravity-cyanide leach process to recover gold and silver in the form of a doré bar containing on average 15% gold and 83% to 84% silver. Minor impurities such as copper are not expected to exceed 1% to 2% of the doré bar weight.

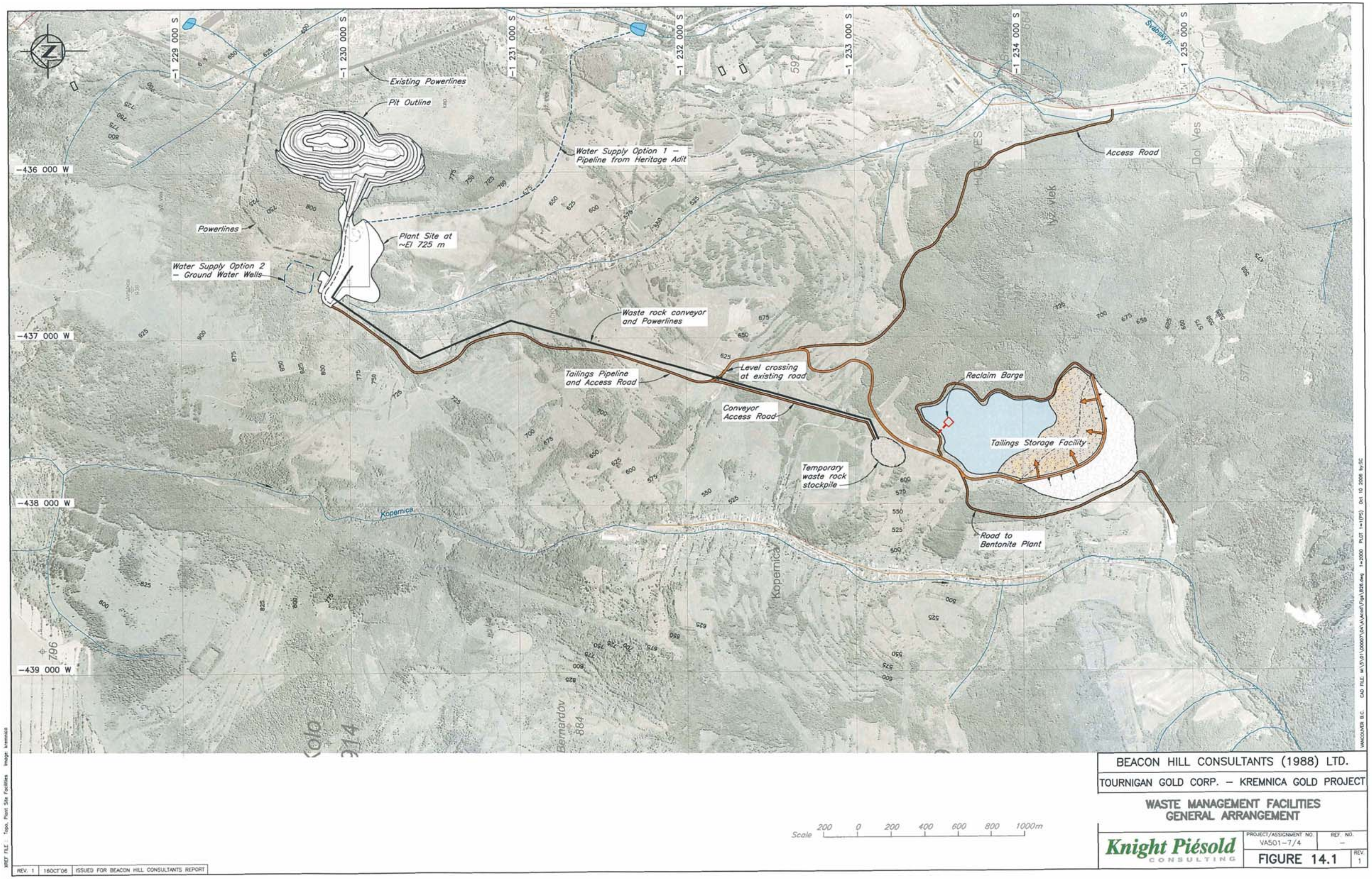
Tailings from the plant passes through a safety screen that captures tramp carbon, and then flows to the 25 m diameter tailings thickener where cyanide-containing solutions are recovered and recycled. The thickener underflow is treated to destroy its residual cyanide.

A tailings storage facility (TSF) will be constructed approximately 5 km from the process plant to provide secure containment of the tailings. Waste rock that is not potentially acid generating (NAG) will be used to build the tailings impoundment dam. Water will be recirculated back to the process plant and used as process water, any water that may be released from the TSF will be treated, and its resulting quality will be equal to, or exceed, the quality of the receiving waters.

Power supply to the project will originate at an existing SSE (Stredo Slovenska Energetika) 22 kV power line located approximately 1 km from the process plant. From this line tap, a new 22 kV three-phase overhead line will extend west and south to the proposed main Kremnica Gold plant substation. The substation will consist of two similar primary transformers, each with the capacity to serve the load alone on a standby rating. The site distribution voltage will be 6.6 kV, which will be utilized to transmit power to the various load centers around the site.

In addition to the process plant, site infrastructure will consist of an administration building, warehouse, maintenance shops, security/gatehouse/emergency vehicle building, fuel storage facilities, and an explosives storage magazine. Surface access roads will be constructed to minimize passage of large vehicle traffic through the town of Kremnica or the surrounding villages. The surface roads from the TSF to Route 65 will also be available for use by the vehicular traffic from the Kopernica bentonite facility. The overall site facilities are shown on Figure 1.2.

Figure 1.2: Waste Management Facilities General Arrangement



Initial capital expenditures have been estimated at $106.2[1] million with ongoing capital at $24.9 million. Average operating costs are estimated to be $10.36/tonne of ore processed.

It is expected that 674,500 ounces of gold and 3,739,000 ounces of silver will be produced over the eight-year mine life.

The mine is expected to directly employ 160 full-time persons, and create approximately 320 to 480 additional indirect jobs.

1.2 CONCLUSIONS AND RECOMMENDATIONS

The economic evaluation of a 6,000 tpd open pit operation at the Šturec deposit indicates that based on an initial capital expenditure of US$106.2 million and metal prices of US$525/oz Au and US$9.25/oz Ag, the project will generate an after-tax Internal Rate of Return (IRR) of 13.08% and a Net Cash Flow of US$75 million (US$20.1 million discounted at 8%). With this scenario, payback of initial capital can be achieved in 4.37 years.

The mineral reserves established in this study based upon a cut-off of 0.5 gAuEq/t are 16.23 million tonnes grading 1.4 g/t for gold and 11.08 g/t for silver. These reserves include material mined as low grade ore, stockpiled and processed after the beneficiation of the higher grade ore. The base case spreadsheet is provided in Table 1.2. Sensitivities to gold price, mining grade, capital cost, operating cost, and silver price at March 2007 are shown in Table 1.3 and Figures 1.3 to 1.7. The sensitivity results indicate that the project is most sensitive to gold price and capital cost changes, and least sensitive to silver price.

The foregoing results indicate that the Šturec deposit at the Kremnica Gold Property can be mined in a viable manner. It is recommended that certain criteria be further investigated in an attempt to improve the IRR and cash flow. These are;

- A number of incentives are available through the European Union and Slovakia that could reduce taxes and provide grants.
- Capital costs are based in part on North American costs. It is recommended that the drawings and data in subsequent studies be developed to a level where Slovakian companies will be willing to provide bids on the proposed work. It is expected that the quotes from Slovakian companies will be lower than North American costs, resulting in a reduction of initial capital costs.
- A review of the working week to determine if a 7 day 24 hour per day schedule can be used.
- A review of waste disposal options to determine if there is potential to placing waste close to the open pit and or design of the open pit for waste disposal within the pit.
- Determine if TSF5a has more potential for cost saving than the alternative used within this study site TSF4.
- Relocation of the surface facilities to eliminate the conveyor or any long haulage of waste rock.
- A preliminary study, commissioned by the company, indicated that there may be regions of higher grade within the deposit that may offer opportunities for increased grade in localized regions. This study utilized indicator models and resulted in a number of grade shells based on

[1] Unless specified other wise, all costs in this report are in US dollars.

probability of occurrence. This approach should be considered for future grade modeling in an effort to accurately identify and locate these areas of elevated grade and to possibly increase grade overall.

- Analyze assay results for screen metallic's as there may be opportunities for grade increases due to the nature of the coarse gold within the vein.
- A model of the cross-cutting vein structure within the footwall and hangingwall andesites which are currently inferred should result in opportunities for localized high grade areas during mining.
- Conduct additional drilling to raise the in-pit inferred material to indicated classification so that it may be included in the cashflows.
- Conduct pit optimization studies at a higher gold price (current pit optimizations were run at a gold price of $385 per ounce). The use of a higher gold price will result in a larger pit containing more reserves, extending the life of mine, and improving the cashflows.

The results of the pre-feasibility study show that the Šturec deposit at the Kremnica Gold property has the potential to be mined in a viable manner. Thus, it is recommended that investigations into a number of possible improvements to the project as shown above are completed and a positive result from those investigations prior to a full feasibility be completed on the project. The total estimated cost, inclusive of a 10% contingency, to complete a full feasibility study is estimated at $2.38 million. A description of the work and breakdown of the estimate is provided in Section 20.

Table 1.2: Base Case Financial Analysis of the Šturec Deposit – 6,000 tpd

KREMNICA PROPERTY
ŠTRUREC DEPOSIT
PRE-FEASIBILITY
BASE CASE
2.1MTPY (6000 TPD)
Financial Analysis Base Case
$(000)s

Resource Tonnes (000)'s	16,229							YEAR					15-Mar-07
Description	**2007**	**2008**	**2009**	**2010**	**2011**	**2012**	**2013**	**2014**	**2015**	**2016**	**2017**	**2018**	**Total**
	-3	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	
Production Tonnes/year				2,100,000	2,100,000	2,100,000	2,100,000	2,100,000	2,100,000	2,100,000	1,529,158		**16,229,158**
Au Grade g/t				1.85	1.39	1.34	1.39	1.69	1.85	0.90	0.53		**1.40**
Recovery Au %				93.00%	93.00%	93.00%	93.00%	93.00%	93.00%	93.00%	93.00%		**93.00%**
Ag Grade g/t				16.89	9.94	8.49	10.31	12.70	15.39	7.97	5.39		**11.08**
Recovery Ag %				66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%		**66.00%**
Au ounces payable				115,825	86,727	83,570	86,616	105,778	115,470	56,185	24,266		**674,437**
Ag ounces payable				737,392	434,194	370,935	450,073	554,797	672,170	347,952	171,410		**3,738,922**
Gross Revenue				$67,318	$49,351	$47,129	$49,434	$60,417	$66,549	$32,568	$14,255		**$387,021**
Operating Costs				($21,947)	($22,182)	($23,299)	($28,073)	($25,573)	($19,611)	($15,964)	($11,416)		**($168,065)**
Income Tax				($1,231)		($259)	($1,502)	($4,043)	($6,575)	($2,654)	($311)		**($16,576)**
Royalty 3%				($896)	($666)	($673)	($838)	($947)	($754)	($230)	($95)		**($5,098)**
Waste Fees and Mining Licence				($219)	($219)	($219)	($219)	($219)	($219)	($219)	($219)		($1,752)
Revenue Before Capital Exp.				$43,025	$26,284	$22,679	$18,802	$29,634	$39,390	$13,501	$2,215		**$195,530**
Capital Expenditures													
- Development/Construction	($2,318)	($62,983)	($40,919)										**($106,220)**
- On-Going Capital					($4,606)	($1,035)	($5,822)	($1,035)	($7,372)	($1,035)			**($20,905)**
Working Capital Change				($3,697)							$3,697		
Reclamation												($4,000)	**($4,000)**
Salvage												$10,622	**$10,622**
Total Capital	($2,318)	($62,983)	($40,919)	($3,697)	($4,606)	($1,035)	($5,822)	($1,035)	($7,372)	($1,035)			**($130,821)**
Net Cashflow	**($2,318)**	**($62,983)**	**($40,919)**	**$39,328**	**$21,678**	**$21,644**	**$12,981**	**$28,599**	**$32,018**	**$12,466**	**$5,912**	**$6,622**	**$75,028**
Discounted NCF 5%	**($2,318)**	**($59,984)**	**($37,115)**	**$33,973**	**$17,835**	**$16,959**	**$9,686**	**$20,325**	**$21,671**	**$8,036**	**$3,629**	**$3,872**	**$36,569**
Discounted NCF 8%	**($2,318)**	**($58,318)**	**($35,081)**	**$31,220**	**$15,934**	**$14,730**	**$8,180**	**$16,687**	**$17,298**	**$6,236**	**$2,738**	**$2,840**	**$20,148**
Discounted NCF 10%	**($2,318)**	**($57,257)**	**($33,817)**	**$29,548**	**$14,806**	**$13,439**	**$7,327**	**$14,676**	**$14,937**	**$5,287**	**$2,279**	**$2,321**	**$11,228**
Rate of Return	**13.08%**												

Notes: *Payback* *4.37* *years*

1. Metal Prices US $ Au/oz 525.00 Ag/oz 9.25 — **Au first full production year 115,825 ozs**

2. Capital requirements based on 100% equity. **4. Taxes are approximate.** — **Ag first full production year 737,392 ozs**

3. All funds are in US$ except where noted. — **NSR/tonne $23.85**

Table 1.3: Base Case Sensitivity Analysis of the Šturec Deposit – 6,000 tpd

KREMNICA GOLD PROJECT
ŠTUREC DEPOSIT
SENSITIVITY ANALYSIS
BASE CASE 6000tpd

UPDATED AS OF 15-Mar-07

Case	Description of Sensitivity	Net Cash Flow Dis.0%	NPV Dis.5%	NPV Dis.8%	IRR
		US$(000)s	US$(000)s	US$(000)s	%
CASE 1	**Base Case**	**$75,028**	**$36,569**	**$20,148**	**13.08%**
Case 2	Gold Price $375	($12,617)	($29,640)	($36,425)	-2.55%
CASE3	Gold Price $425	$17,198	($7,066)	($17,112)	3.29%
CASE4	Gold Price $475	$46,802	$15,320	$2,029	8.53%
CASE 1	**Base Case**	**$75,028**	**$36,569**	**$20,148**	**13.08%**
CASE5	Gold Price $575	$101,895	$56,748	$37,334	17.07%
CASE6	Gold Price $625	$128,762	$76,900	$54,483	20.79%
CASE7	Gold Price $675	$169,063	$107,129	$80,207	26.00%
CASE8	Grade -10%	$42,567	$12,130	($692)	7.82%
CASE9	Grade -5%	$58,951	$24,472	$9,835	10.54%
CASE 1	**Base Case**	**$75,028**	**$36,569**	**$20,148**	**13.08%**
CASE10	Grade +5%	$90,447	$48,151	$30,013	15.40%
CASE11	Grade +10%	$105,867	$59,723	$39,865	17.63%
CASE12	Capital Cost -20%	$102,285	$61,067	$43,247	20.75%
CASE13	Capital Cost -10%	$88,959	$49,040	$31,883	16.65%
CASE 1	**Base Case**	**$75,028**	**$36,569**	**$20,148**	**13.08%**
CASE14	Capital Cost +10%	$60,454	$23,627	$8,017	9.89%
CASE15	Capital Cost +20%	$45,124	$14,042	$934	8.25%
CASE16	Operating Cost -20%	$103,883	$58,267	$38,638	17.36%
CASE17	Operating Cost -10%	$89,365	$47,354	$29,340	15.25%
CASE 1	**Base Case**	**$75,028**	**$36,569**	**$20,148**	**13.08%**
CASE18	Operating Cost +10%	$60,218	$25,408	$10,626	10.74%
CASE19	Operating Cost +20%	$45,124	$14,042	$934	8.25%
CASE20	Silver Price -US$6.75	$67,494	$30,917	$15,335	11.91%
CASE21	Silver Price -US$8.00	$71,321	$54,862	$17,783	12.51%
CASE 1	**Base Case**	**$75,028**	**$36,569**	**$20,148**	**13.08%**
CASE22	Silver Price -US$10.50	$78,734	$39,348	$22,513	13.64%
CASE23	Silver Price -US$11.75	$82,441	$42,126	$24,878	14.20%

Figure 1.3: Cash Flow vs. Gold Price – 6,000 tpd

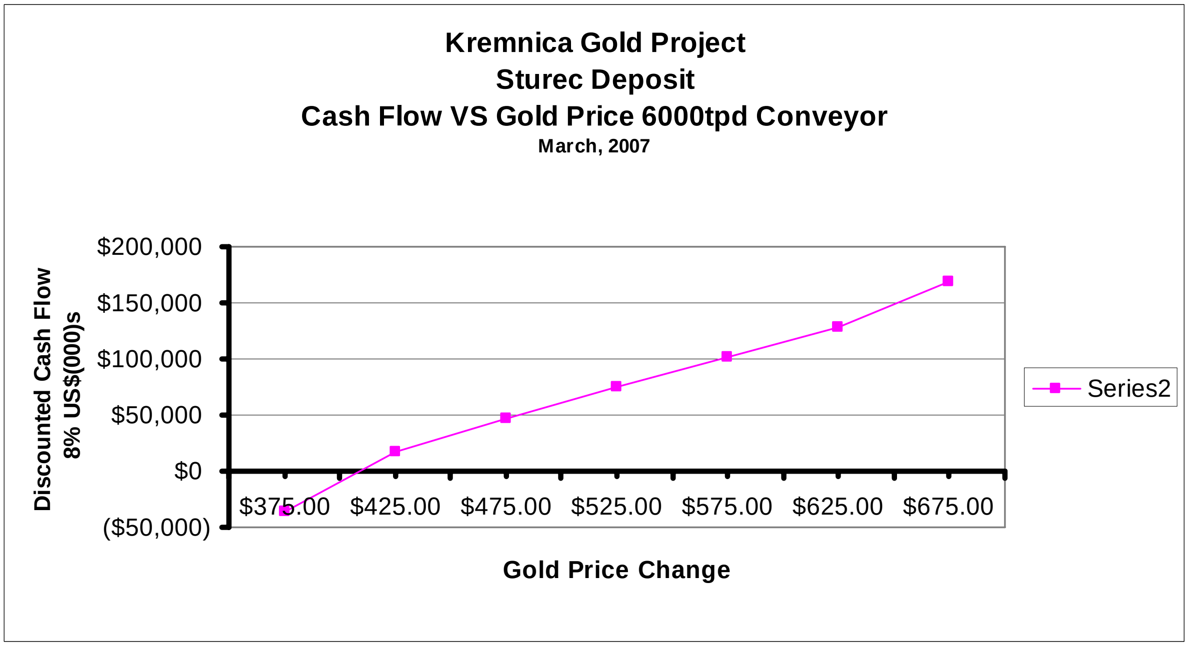


Figure 1.4: Cash Flow vs. Head Grade Change – 6,000 tpd

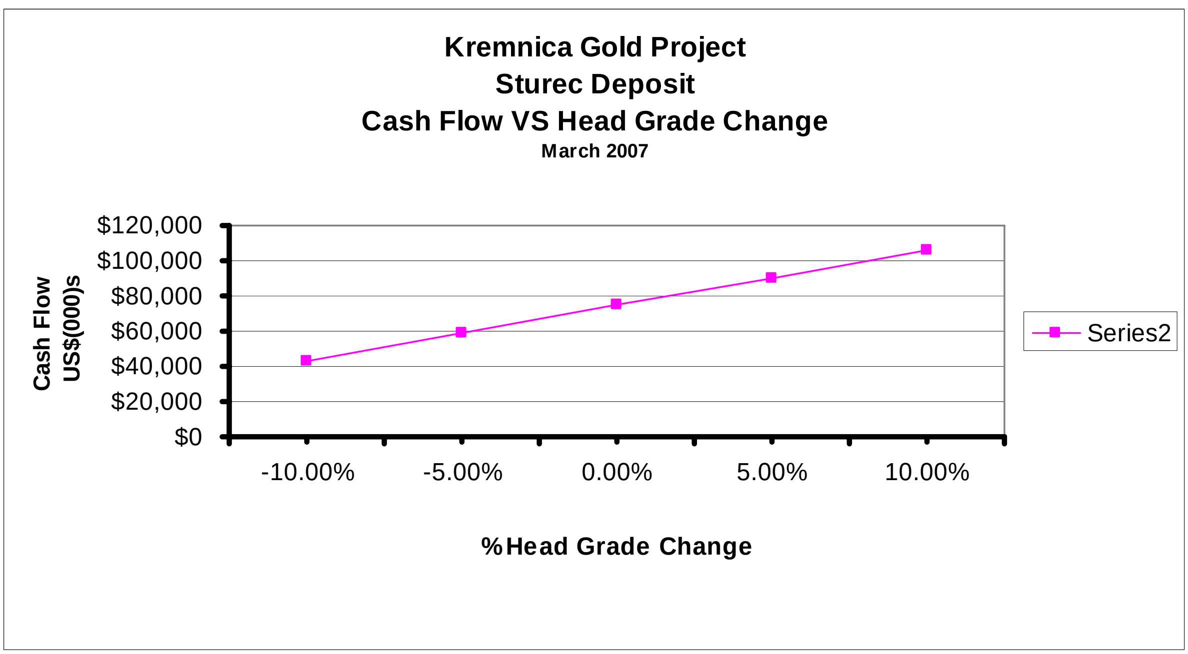


Figure 1.5: Cash Flow vs. Capital Cost Change – 6,000 tpd

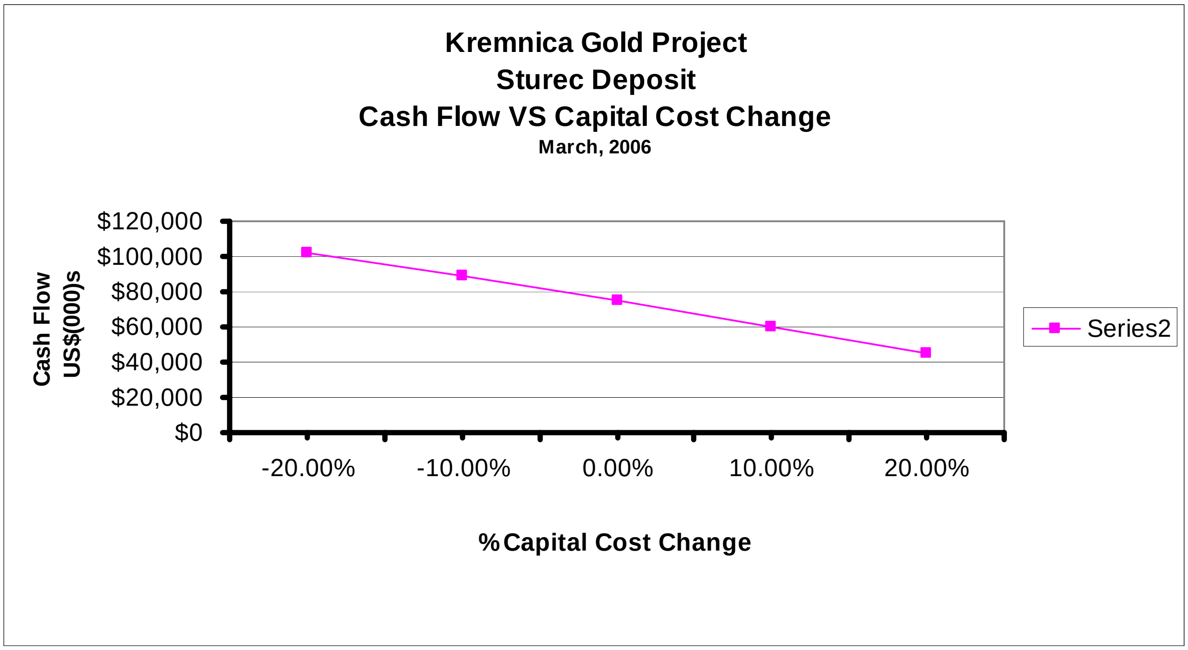


Figure 1.6: Cash Flow vs. Operating Cost Change – 6,000 tpd

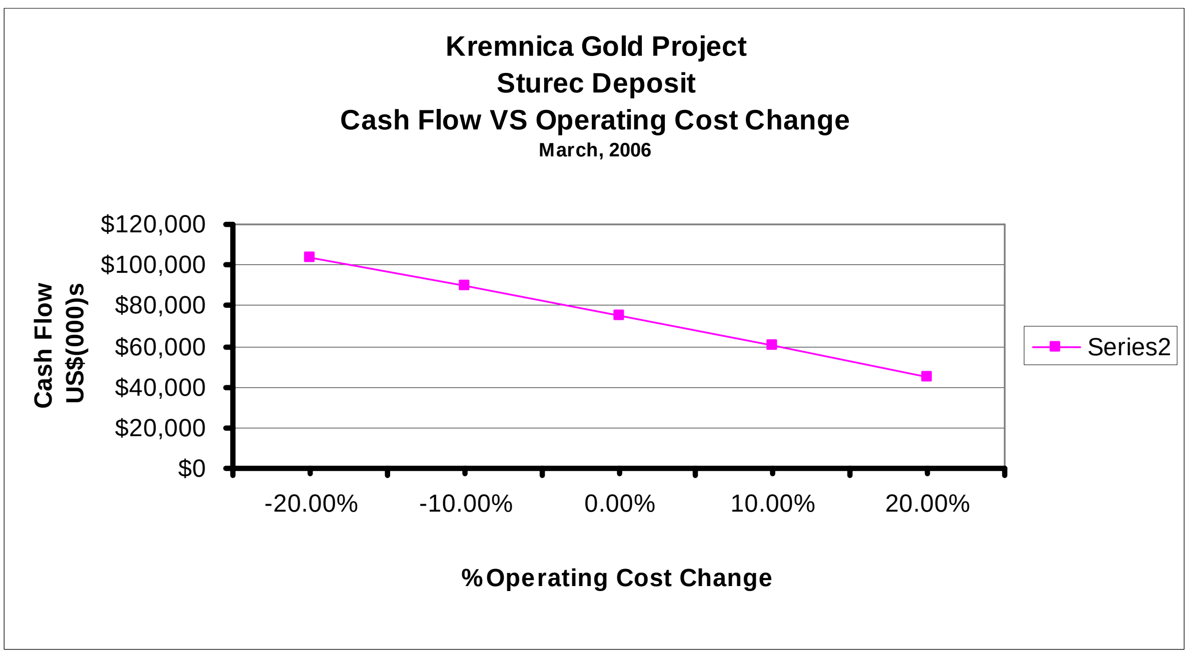
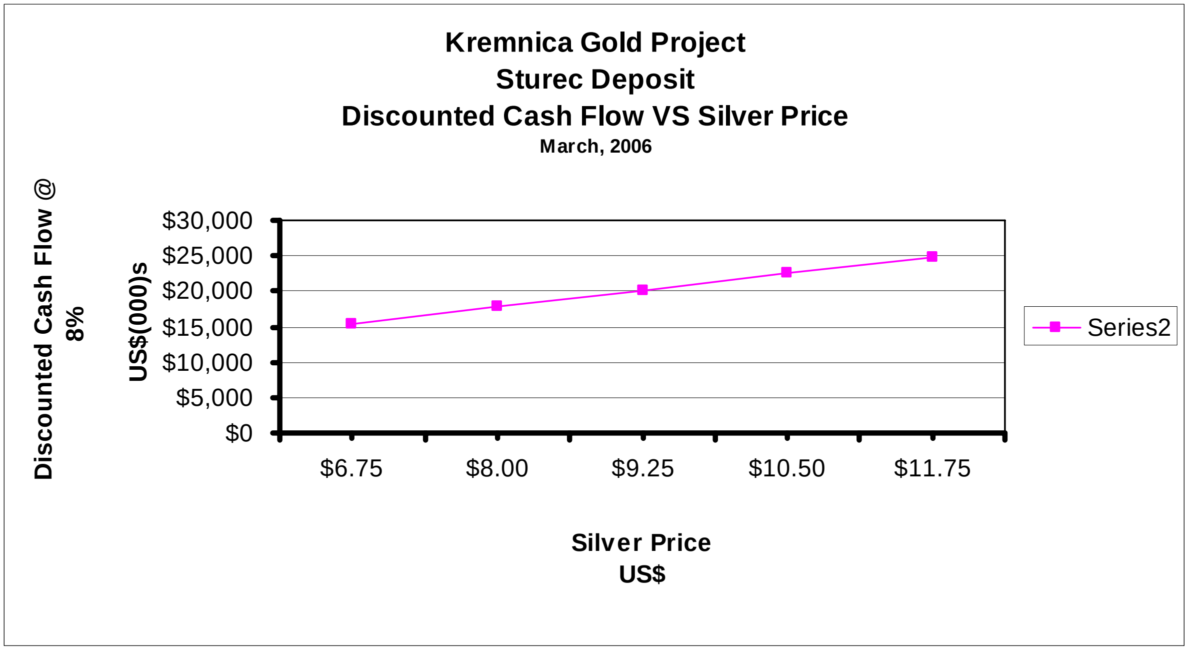


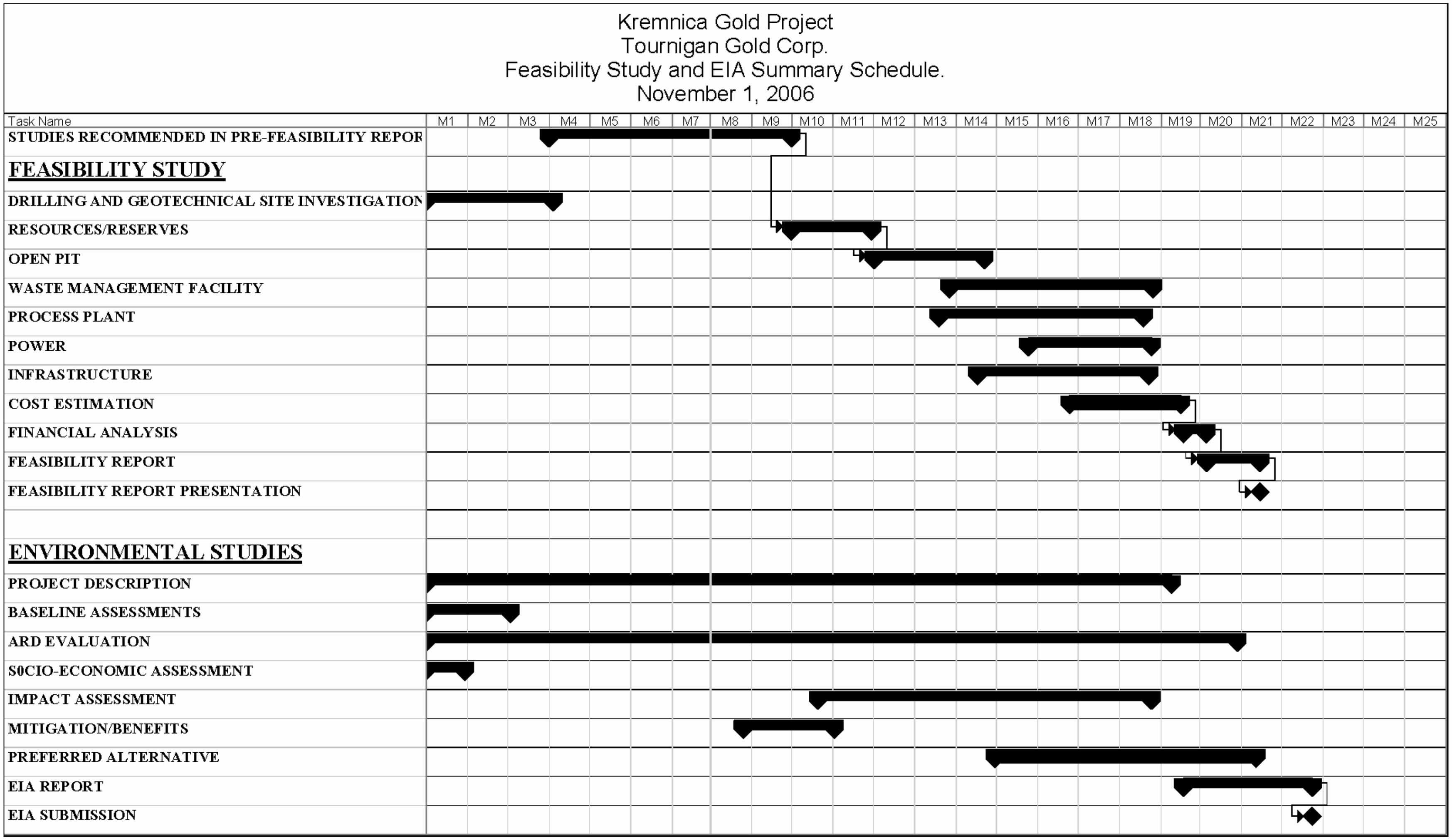
Figure 1.7: Discounted Cash Flow vs. Silver Price



1.3 PROPOSED FULL FEASIBILITY AND EIA SCHEDULE

1.3.1 Overview

The Feasibility Study Summary Schedule shown below describes the activities and time required that are estimated to complete the full feasibility and EIA reports. The subsequent events to take the project to construction are not shown.


Kremnica Gold Project
Tournigan Gold Corp.
Feasibility Study and EIA Summary Schedule.
November 1, 2006
Task Name
M1 M2 M3 M4 M5 M6 M7 M8 M9 M10 M11 M12 M13 M14 M15 M16 M17 M18 M19 M20 M21 M22 M23 M24 M25
STUDIES RECOMMENDED IN PRE-FEASIBILITY REPOR
FEASIBILITY STUDY
DRILLING AND GEOTECHNICAL SITE INVESTIGATION
RESOURCES/RESERVES
OPEN PIT
WASTE MANAGEMENT FACILITY
PROCESS PLANT
POWER
INFRASTRUCTURE
COST ESTIMATION
FINANCIAL ANALYSIS
FEASIBILITY REPORT
FEASIBILITY REPORT PRESENTATION
ENVIRONMENTAL STUDIES
PROJECT DESCRIPTION
BASELINE ASSESSMENTS
ARD EVALUATION
S0CIO-ECONOMIC ASSESSMENT
IMPACT ASSESSMENT
MITIGATION/BENEFITS
PREFERRED ALTERNATIVE
EIA REPORT
EIA SUBMISSION

SECTION 2.0 -INTRODUCTION

2.1 PURPOSE

The purpose of this report is to review the potential viability of the Šturec deposit on the Kremnica property at a pre-feasibility level of confidence and to present a program to further develop the property to a full feasibility level.

The Šturec deposit is considered amenable to surface methods of extraction and does not lend itself to underground mining. Metallurgical testwork indicates that mineral processing by whole ore cyanidation provides the highest recovery; this method has therefore been incorporated into this report.

This pre-feasibility study has used measured and indicated resources to evaluate the project. Inferred resources[2] are shown separately and are not included in the evaluation other than to shown the potential of the project should they be enhanced to a higher level of confidence. The inferred resources have been included for completeness only.

2.2 REPORT COMPILATION

This report has been directed and compiled by W. Peter Stokes, P.Eng., of Beacon Hill Consultants (1988) Ltd.

The geological and resource sections were extracted from the 11 May 2006 report entitled "Revised Mineral Resource Estimate, Kremnica Gold Project – Kremnica, Slovak Republic, Europe" prepared for Tournigan Gold Corporation by Garth D. Kirkham, P.Geoph., and W. Peter Stokes, P.Eng. of Beacon Hill Consultants (1988) Ltd.

The pit optimization section was prepared by Garth Kirkham; the processing section by John Fox, P.Eng.

The tailings and waste management section was prepared by Knight Piésold Ltd. under the direction of Ken. Brouwer, P.Eng.

W. Peter Stokes, who also provided project management and report coordination, prepared all other sections.

[2] Due to the uncertainty that may be attached to an inferred mineral resource, it cannot be assumed that all or any part of an inferred mineral resource will be upgraded to an indicated or measured resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred resources must be excluded from estimates that form the basis of feasibility or other economic studies.

2.3 ACKNOWLEDGEMENTS

This report was completed with the help of a number of companies who provided budget pricing, equipment descriptions, and other information. The authors of this report wish to thank these companies for their assistance. We would also like to thank the following individuals:

- J. Walchuck, P.Eng., President and CEO of Tournigan Gold Corporation
- J. Ringwald, P.Eng., VP Technical Services of Tournigan Gold Corporation.
- S. Stine, P.Eng., precseda predstavenstva, Kremnica Gold, a.s.
- G. Stock, Corporate Development and Communications, Tournigan Gold Corporation
- K. Ausburn, VP Exploration, Tournigan Gold Corporation
- B. Bartalsky, General Director Kremnica Gold, a.s.
- J. Cuthill, Senior Geologist, Tournigan Gold Corporation
- J. Crummy, Geologist, Kremnica Gold, a.s.
- Kremnica Gold staff at Kremnica a.s. for their help and cooperation throughout the course of this study.

2.4 TERMS OF REFERENCE

Beacon Hill Consultants (1988) Ltd. (Beacon Hill) was commissioned by Tournigan Gold Corporation (Tournigan) to complete a study on the Šturec deposit located close to the town of Kremnica, Slovakia. The purpose of the study was to prepare a pre-feasibility report on which the viability of the project could be evaluated.

Previous work on the project by Beacon Hill in February 2004 indicated that at the preliminary assessment level the project had the potential to be viable. As a result of this work, Tournigan completed a drilling program in 2005 that formed the basis for a revised mineral resource estimate, on which this pre-feasibility study report is based. The work completed under Beacon Hill included all the technical aspects of the project. Environmental baseline work, public consultation, and socio-economic studies were completed by others.

A number of criteria as established by Tournigan formed part of the scope of work. These consisted of:

- Elimination of the waste rock disposal facility. To achieve this waste rock excavated from the open pit suitable for construction was used to build the Tailing Storage Facility dam.
- The location of all surface facilities so that they can be either not seen from the town of Kremnica and villages or camouflaged with berms and vegetation.
- Restrict hours of operation for the open pit to 16 hours per day 6 days per week and not operate on statutory holidays.
- Establish transportation corridors that reduces to a minimum the movement of large vehicular traffic through the town of Kremnica and surrounding villages.

In order to complete the pre-feasibility study, Beacon Hill utilized the services of associates for the following areas or tasks: geology, resource estimation, open pit layouts, metallurgical testwork, and plant design. Beacon Hill commissioned Knight Piésold to complete the geotechnical requirements for the open pit, tailings storage facility, and site infrastructure. Merit Consultants International Inc. was commissioned by Beacon Hill to complete the cost estimation with input from the staff at Kremnica Gold a.s.., Beacon Hill's Associates, and Knight Piésold. Kevin Morin of MDAG was commissioned to establish the ARD investigations.

SECTION 3.0 - RELIANCE ON OTHER EXPERTS

Mineral Processing and Metallurgical Testing; the authors have relied mainly on the reports written by Hazen Research Inc. Hazen Research is a well known and reputable company currently active in the mining industry. The authors have no reason to doubt the validity of the Hazen reports but the authors did not commission any metallurgical tests themselves and are not experts in this field. The reports relied on are as follows:

Hazen Research Inc. 4601 Indiana St. Golden, Colorado, 80403.

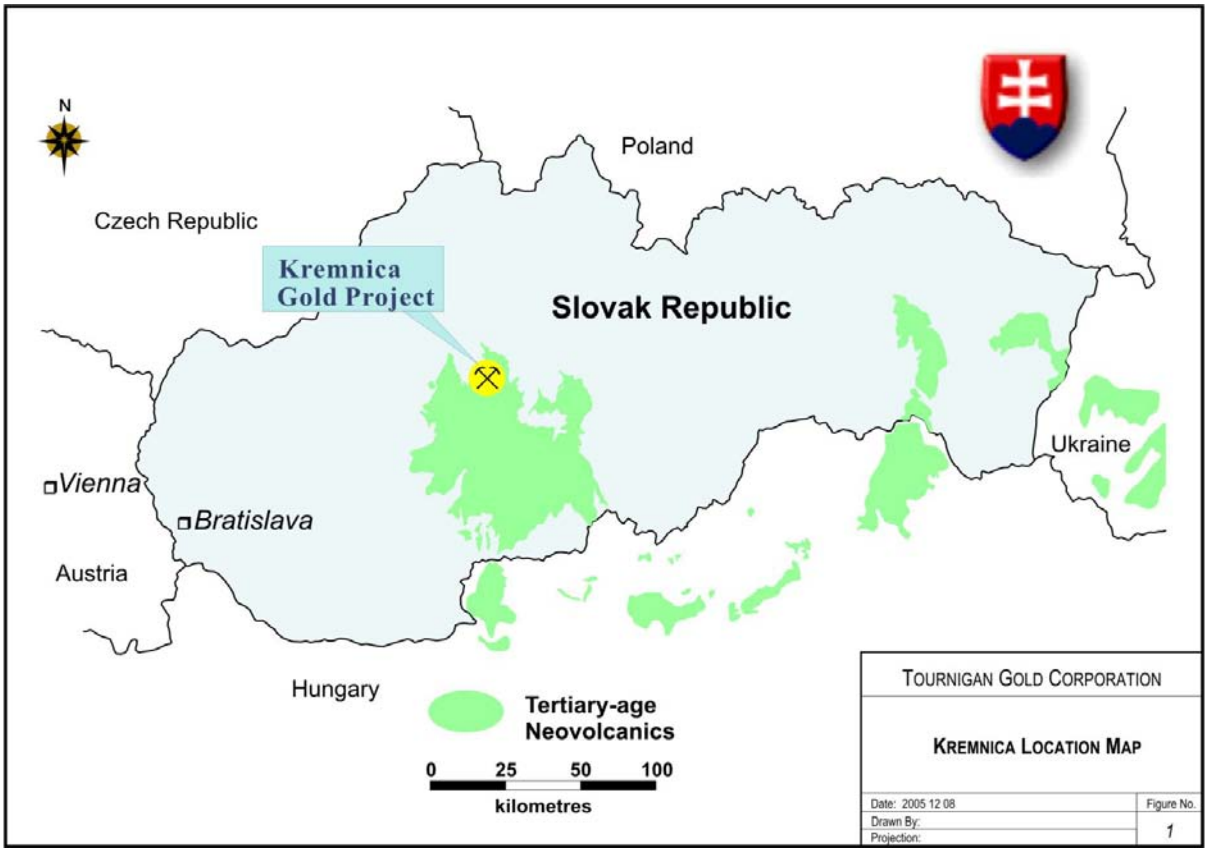
1) Metallurgical Testing of Kremnica Gold Ore Samples from Slovakia. April 17, 1997.
2) Metallurgical Testing of Additional Kremnica Gold Ore Samples from Slovakia. February 2, 1998.

The authors have relied on the experience and expertise of the on-site geology personnel for input with respect to the interpretation of recovery data and also the interpretation of geology, mineralization, and specific gravity data. In particular, the authors relied upon John Cuthill’s sectional interpretation of the geology, sulfide/oxide interface, collapse zone and voids utilized within this report. The authors believe these interpretations to be a current and accurate representation of the deposit.

SECTION 4.0 – PROPERTY DESCRIPTION AND LOCATION

The Kremnica project is located in central Republic of Slovakia as shown on Figure 4.1. The town of Kremnica lies 17 km west of central Slovakia's largest city, Banská Bystrica. The project area is accessible from Vienna, Austria by driving east across the border into Slovakia and then northeast through Bratislava, Nitra, Zlaté Moravce, and Ziar nad Hronom. The trip takes approximately four hours. Kremnica has a population of about 6,000 and is accessible by train from Bratislava, the capital and largest city in Slovakia.

Figure 4.1: Kremnica Location Map



The Kremnica project comprises two contiguous properties: the Kremnica Mining License and the Lutila Exploration License, which, based on the Slovak JTSK coordinate system, are 11.79 km^2 and 86.38 km^2 in area, respectively. Boundary coordinates are listed in Tables 4.1 and 4.2; the properties are depicted in Figure 4.2 and 4.3.

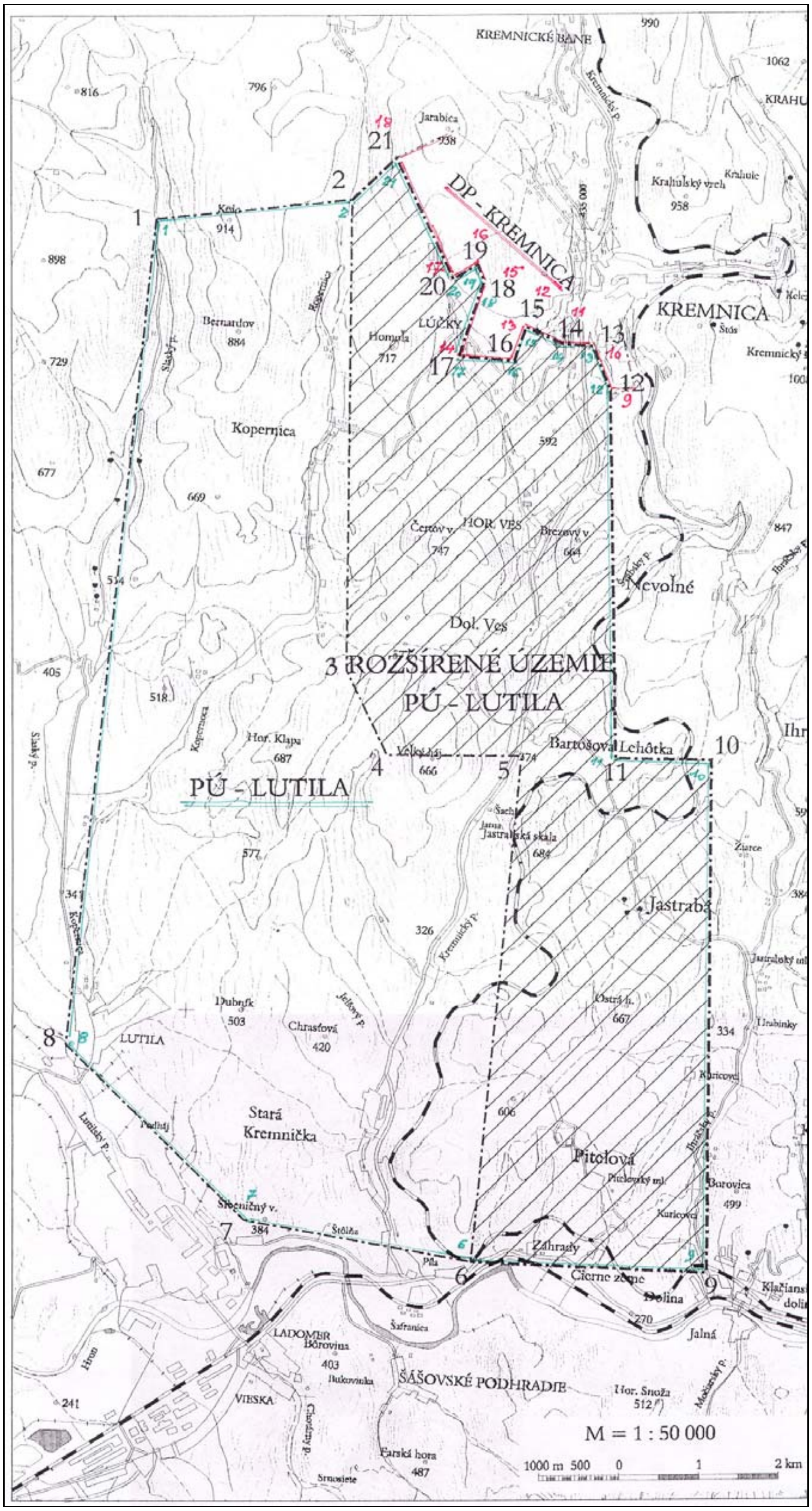
Table 4.1: Kremnica Mining License (MHD-D.P. 12)

	X	Y
1	434 704,48	1 226 335,58
2	434 057,01	1 230 565,93
3	434 154,93	1 230 587,40
4	434 109,63	1 230 808,98
5	434 396,33	1 230 874,66
6	434 346,99	1 231 088,05
7	434 582,61	1 231 139,37
8	434 390,93	1 232 006,62
9	434 689,39	1 232 050,70
10	434 902,80	1 231 494,23
11	435 286,16	1 231 462,16
12	435 743,01	1 231 239,45
13	435 892,07	1 231 669,13
14	436 587,24	1 231 609,33
15	436 268,56	1 230 656,54
16	436 347,27	1 230 437,13
17	436 650,90	1 230 622,04
18	437 358,63	1 229 065,52
19	436 573,69	1 228 696,69
20	436 497,30	1 226 635,16

Table 4.2: Lutila Exploration License

	Y	X
1	440 360,00	1 229 840,00
2	437 914,94	1 229 598,84
21	437 358,63	1 229 065,52
20	436 650,90	1 230 622,04
19	436 347,27	1 230 437,13
18	436 268,56	1 230 656,54
17	436 587,24	1 231 609,33
16	435 892,07	1 231 669,13
15	435 743,01	1 231 239,45
14	435 286,16	1 231 462,16
13	434 902,80	1 231 494,23
12	434 689,39	1 232 050,70
11	434 635,00	1 236 775,00
10	433 430,00	1 236 810,00
9	433 495,00	1 243 385,00
6	436 450,00	1 243 220,00
7	439 279,63	1 242 677,00
8	441 500,00	1 240 430,00

Figure 4.2: Kremnica Project Licenses



The Kremnica Mining License (MHD-D.P. 12) was issued 21 January 1961, and remains valid in perpetuity. The mining license is shown on Figure 4.3. On 6 March 2006, the license was transferred from Kremnica Gold a.s. to Ludovika Holding, s.r.o., both of which are 100% owned by Tournigan Gold Corporation. According to the mining license, the Mining Bureau must permit new mining projects. The current permit allows only for protection and preservation of the old mine workings and is valid until December 2010, after which a new permit for work must be obtained. The mining license fee is 60,000 Slovak korunas (SKK) or approximately CAD$2,222 per year. According to Slovak law, work must be done within the license area or the license can be withdrawn; however, the law is not specific on when, what, or how the work must be completed. All known mineral resources, as well as potential areas for future operations, waste rock disposal, tailings dam sites, and processing plant sites, are contained within the mining license.

Kremnica Gold a.s registered the Lúĉky Exploration License on 5 June 2002, and it became effective on 23 September 2002. The license was subsequently closed.

The original Lutila Exploration License was registered on 1 October 2003 by Kremnica Gold a.s. and became effective on 5 March 2004. The license was extended to include the Lúĉky Exploration License on 19 August 2004. The current license fee for the Lutila Exploration License is SKK261, 000 (approximately CAD$9,700) per year.

The mining license and exploration license have been legally surveyed. As shown in Figure 4.2, the two licenses are contiguous, covering all known mineralized areas within the Kremnica district. There are no other metal claims or other companies working within the district. However, within the exploration license area there are several registered clay deposits, and two small clay pits are in operation. These clay licenses are not expected to have any material effect on the subject properties.

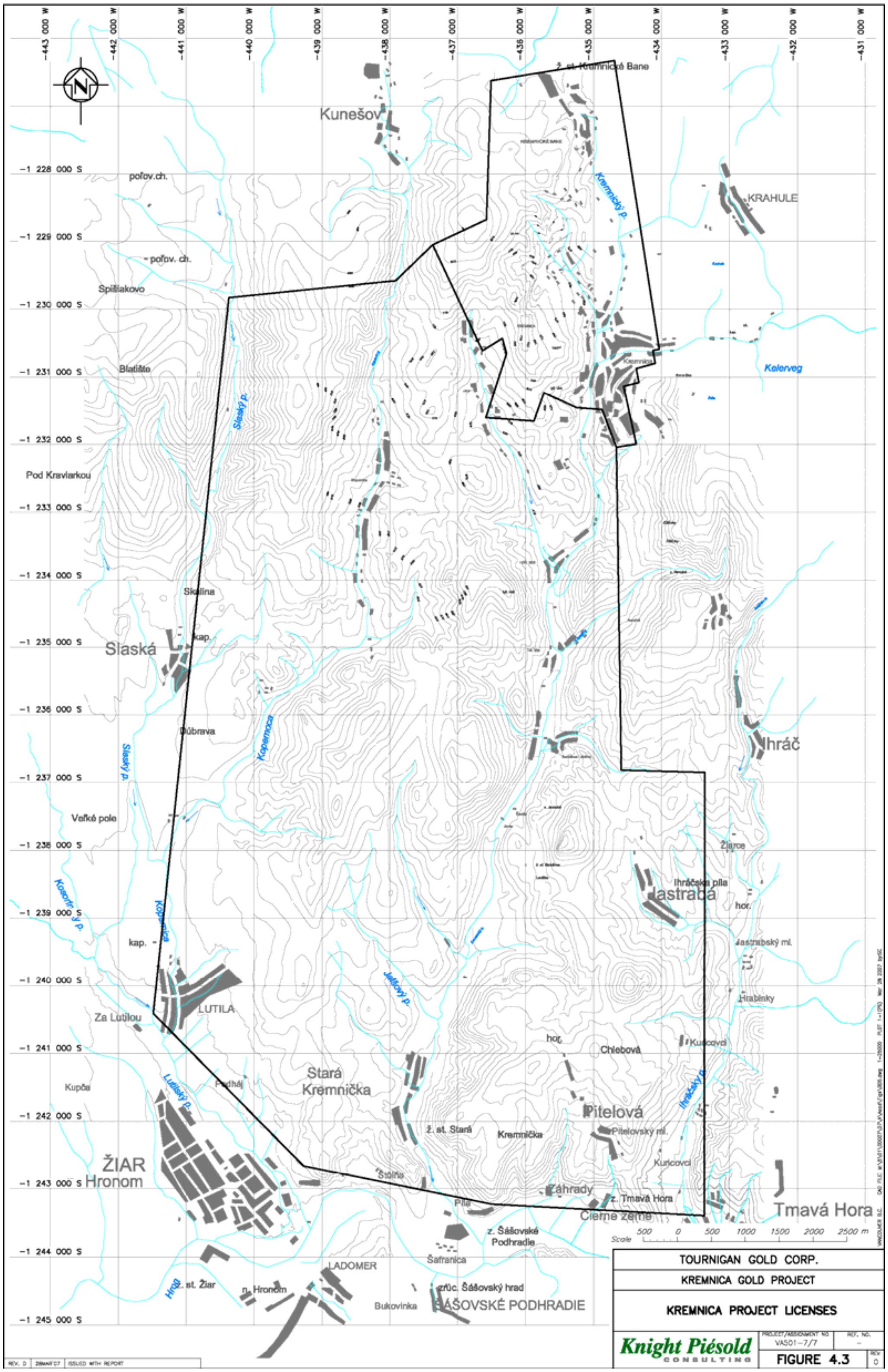
With the exception of a royalty payment to the Slovak State, the Kremnica project is not known to be subject to any royalties, back-in rights, payments, or other agreements and encumbrances. The Slovak State is entitled to a royalty to be paid as a percentage of net profits. The rate of payment can be between 0% and 10% of net profit, but is open to negotiation. In the case of Kremnica, it is expected that the payment would be close to 3% of net profit.

4.1 SURFACE RIGHTS

The Kremnica Mining License (Figure 4.3) covers 11.79 km^2, and includes within its boundaries the township of Kremnica (population approximately 6,000) and several hundred privately owned properties. The core of the mining area has been set aside by the township for the purpose of mining and a very limited number of houses exist in this area. The author of this report has reviewed maps of land ownership under the mining license and found that virtually all of the land required for the proposed project belongs to the State Land Fund, or the State Department of Forestry. Negotiations with the State for mining rights to these lands are expected to be straightforward. It is likely that a future mining operation would consider purchasing some privately owned land. Current legislation in Slovakia provides for State resumption of surface rights for the purpose of mine development. This court-ordered process is intended to prevent a landowner(s) from halting the development of a mining project that is beneficial to the State.

When a company applies for an exploration license in Slovakia, all affected parties are contacted and asked to comment on the proposed exploration activities. The company applying for the exploration license meets with all affected parties and an agreement is reached as to how exploration will

Figure 4.3: Mining License Area



proceed. Exploration work sufficient to define a resource is then carried out under the agreement. A new agreement must be reached before mining can begin. The permitting process is outlined in the agreement and described in straightforward regulations.

At present, there is opposition to mine development by NGO's (non-governmental organizations) and a number of residents of the town of Kremnica. Tournigan is addressing these concerns through public consultation and education. Tournigan is committed to acquiring a social license to operate and to ensuring sustainability of mining in the area.

4.2 ENVIRONMENTAL LIABILITIES

The previous owners of Kremnica Gold, Argosy Mining Corporation, conducted a large exploration and drilling program in the project area. At the end of the active drilling phase, Kremnica Gold received letters from the State Forestry, Kremnica Municipality, and landowners (land users) stating that there were no environmental problems resulting from exploration activities; hence, Tournigan Gold, the new owners of Kremnica Gold, is limited from any liability for past exploration activities.

Kremnica Gold has no environmental liabilities for past mining activities and would only assume liability for any environmental problems related to the future operation of a mine. Slovak law states that organizations conducting mining operations are responsible for their actions and that those responsibilities are not transferable. Any liability for environmental damage resulting from past mining activities by the State Company, Rudne Bane, would fall to the State.

SECTION 5.0 – ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

The Kremnica project is situated in the rolling hills of the Rudnice valley at an elevation of between 958 and 619 masl (meters above sea level). Old mine workings, mounds, excavations, pits, and adits can be found on most parts of the mining license, reflecting the 1,000-year mining history of the Kremnica area. The majority of the target areas consist of meadows and mixed regenerating coniferous and broadleaved forest.

The town of Kremnica is situated within the mining license. There are also some scattered residential areas within the license area, but these are not expected to affect future development. The core areas of the mining license are designated for mining activity as part of the 10-year plan for the city of Kremnica.

Paved roads and a network of old mining and forestry tracks service the property. There is also a regularly operating rail line to the town of Kremnica. High voltage power lines pass through the margins of the mining lease, and connection to the national grid is possible. A network of historic water storage impounds associated with the mining history of the area would ensure an adequate water supply.

Climate is typical of the mountains of central European, with mild summers and cold winters. Average temperature throughout the year is approximately 8°C, with highs around 30°C and lows around –30°C. Precipitation averages around 920 mm per year. Snow cover can be expected three to four months per year.

Potential sites for waste rock disposal, tailings dam sites, and potential processing plant sites have previously been proposed as described in the report, "Preliminary Assessment, Kremnica Gold Project, Šturec Deposit" (Beacon Hill, February 2004). These sites and others are evaluated as part of this pre-feasibility study.

With a 1,000-year mining history, a well-educated population, and high unemployment in the region, a future mine can expect to benefit from a highly skilled, professional, and inexpensive labour force.

SECTION 6.0 – HISTORY

6.1 MINING OPERATIONS PRIOR TO 1970

Gold mining commenced at Kremnica as early as the 8th century and is relatively well documented from the year 1328 forward. Production has totalled 46,000 kg (1.5 million ounces) gold and 208,000 kg (6.7 million ounces) silver. Production was from open pits and underground mine workings. Extensive underground mining occurred within an area 4 km long x 2 km wide. The largest relic of past mining is a surface depression in the Šturec area measuring 600 m long x up to 200 m wide x 100 m deep. The district is pockmarked with hundreds of pits, collapsed shafts, and adits.

6.2 MODERN SLOVAK EXPLORATION

The Slovak Geological Survey and Rudne Bane (the State mining company) explored the Kremnica district intermittently between 1962 and 1990. This exploration work included driving four major exploration adits, more than 20 underground crosscuts, and both surface and underground drilling. Exploration defined near-surface, low-grade deposits at Šturec and Vratislav. No modern exploration was undertaken in the Wolf area, located one kilometer to the north. In the early 1990s, based on the above work, the Czechoslovakian government calculated a resource totalling 7,570,000 tonnes at a gold grade of 1.53 g/t, or 370,000 ounces of gold and 2,811,000 ounces of silver. This resource is considered historical.

6.3 MINING FROM 1987 TO 1992

Beginning in 1987, Rudne Bane mined 50,028 tonnes averaging 1.54 g/t gold from a small open pit located in the Šturec deposit. The ore was treated in a cyanide mill, located near the Ludovika shaft that operated at about 30 tpd. The operation was not profitable and was therefore abandoned in 1992.

6.4 ARGOSY MINING CORPORATION

Kremnica Banska Spolocnost (KBS), an investment company composed of former mine managers, obtained the title to the Kremnica Mining Lease (MHD-D.P. 12) from the Slovak government on 1 April 1995. In 1995, Argosy Mining Corporation of Vancouver formed a 100% owned Slovak Subsidiary, Argosy Slovakia s.r.o., which entered into a joint venture with KBS on 6 October 1995. Argosy Slovakia purchased KBS's share of the joint venture on 24 April 1997 to control 100% of the mining license through its subsidiary, Kremnica Gold a.s. Argosy completed a core drilling program in 1996 and a combined core and reverse-circulation (RC) drilling program in 1997 for a total of 79 holes (12,306 m).

Argosy commissioned Western Services Engineering Inc. of California (Western Services) to produce a preliminary resource estimate for the Šturec South deposit at Kremnica. This 1997 resource estimate is considered historical. The resource estimate was based on a block containing mineralized envelopes with a 0.5 g/t Au cut-off grade. The stated resource was 1.1 million ounces of gold and 9 million ounces of silver. At a 1.0 g/t gold cut-off grade, the main Šturec zone was calculated to contain an open pit resource (including ore loss and dilution) of 11.26 million tonnes at a grade of 1.8 g/t gold and 12.5 g/t silver, at a waste-to-ore ratio of 2.4:1.

6.5 TOURNIGAN GOLD CORPORATION

Tournigan acquired the rights to the Kremnica project by purchasing Kremnica Gold a.s. from Argosy Mining Corporation in July 2003. Tournigan owns 100% of Kremnica Gold a.s. and states that it has no long-term debt.

In 2004 Tournigan conducted diamond drilling programs north of Šturec at the Wolf and Vratislav deposits, and large soil geochemical surveys south of Šturec over much of the Kremnica South deposit. Test diamond drilling was carried out at the Certov and Bartasova Lehotka areas at Kremnica South, and limited exploration trenching was done south of the town of Lúĉky .

During 2004, Tournigan commissioned an independent technical report to establish a resource estimate for the Šturec deposit. In the summer and autumn of 2005, Tournigan executed a 36-hole program of RC drilling as infill of Argosy's and Tournigan's earlier core drilling programs. Tournigan drilled five additional holes as twins of Argosy's previous core holes. This 41-hole program resulted in the deposit being drilled off on approximate 50-meter centers (earlier drilling had been on approximate 100 x 50 meter centers). The RC program provided a correlation between core and RC holes: they were mutually supportive with respect to the location of ore zones and their grades.

The holes and assay results were displayed on cross-sections and recorded on logs. Samples were collected at 1-meter intervals under the immediate supervision of a geologist, sealed in plastic bags, and submitted for analysis and check analyses according to the required formal protocols.

The holes were logged on site by the drill geologists and again in the laboratory where qualitative samples were taken and inventoried as geological reference samples. Bulk rejects were stored on the property and are available for any further testwork.

The results confirmed the geology and ore outlines that were previously established by core drilling (e.g., rock types and alteration, location of zones of oxidation, location of ore-bearing veins and stockworks, hanging walls, footwalls, thicknesses, strikes, dips, and grades).

For metallurgical testwork, Tournigan collected and submitted representative suites of different ore types and wallrock types from RC samples (see Section 13 for results).

In 2005, Tournigan commissioned Beacon Hill to conduct a pre-feasibility study based on the revised mineral resource estimate to assess the economic viability of developing a mine at Kremnica.

6.6 PREVIOUS RESOURCE ESTIMATES

Three resource estimates have been previously calculated. The first was based on work completed prior to 1996 and was performed by the government of Czechoslovakia for the Šturec deposit and the combined Vladimir and Vratislav deposits. This work used a 0.5 gAu/t cut-off grade and is summarized in Table 6.1.

Table 6.1: Historical Resource Estimates; Government of Czechoslovakia

Šturec	Tonnes	Au g/t	Ag g/t	Au Grams	Au Ounces	Ag Grams	Ag Ounces	Au Eq*
Rudne Bane Area	6,397,000	1.45	10.84	9,275,650	298,214	69,343,480	2,229,407	327,167
North End – Survey	291,000	2.72	14.51	791,520	25,448	4,222,410	135,751	27,211
Vratislav-Vladimir	882,000	1.67	15.74	1,472,940	47,355	13,882,680	446,331	53,152
Combined Total	7,570,000	1.52	11.55	11,540,110	371,017	87,448,570	2,811,490	407,530

Note: ***Au Eq based on Au=77 Ag.

An additional geologic cross-sectional polygonal resource estimate was independently calculated. The results of this work indicated a total of 16,000 ounces of gold for Vladimir and 74,000 ounces for Vratislav combined.

In 1996 and 1997 Argosy Minerals Corp., the previous owner of the properties performed an additional 12,306.7 m of drilling, 9,382.4 m of which was in the Šturec deposit. This work significantly defined the Šturec deposit at depth, to the south, and to the north, where it now connects with the Vladimir deposit. The resource calculated by Western Services in their March 1998 study, which is the primary reference and source of data for the technical report by Smith and Kirkham is shown in Tables 6.2 and 6.3.

Table 6.2: 1998 Resource Estimates; Indicated Resources

Cut –off Grade g/t AuEq	Total Tonnes	AuEq	Au	Ag	Grams	Ounces
0.0	3,768,000	1.71	1.48	11.6	6,443,280	207,153
0.5	3,655,000	1.75	1.51	11.9	6,396,250	205,641
1.0	2,652,000	2.12	1.83	14.2	5,622,240	180,756
1.5	1,719,000	2.59	2.25	16.9	4,452,210	143,139
2.0	1,021,000	3.19	2.79	19.8	3,256,990	104,713
2.5	644,000	3.74	3.29	22.6	2,408,560	77,436
3.0	401,000	4.36	3.85	25.7	1,748,360	56,210
3.5	236,000	5.16	4.58	29	1,217,760	39,151
4.0	165,000	5.79	5.15	31.8	955,350	30,715
4.5	122,000	6.35	5.66	34.5	774,700	24,907
5.0	86,000	7.02	6.26	37.9	603,720	19,410

Source: Western Services Engineering, Inc. **Note**: AuEq is based upon 50:1 Au:Ag.

Table 6.3: 1998 Resource Estimates All Resource Classes

Cut –off Grade g/t AuEq	Total Tonnes	AuEq	Au	Ag	Grams	Ounces
0.0	19,645,000	1.71	1.49	10.9	33,592,950	1,080,020
0.5	19,063,000	1.75	1.53	11.1	33,360,250	1,072,539
1.0	13,303,000	2.17	1.9	13.8	28,867,510	928,096
1.5	8,477,000	2.7	2.37	16.9	22,887,900	735,851
2.0	5,181,000	3.33	2.93	20.2	17,252,730	554,679
2.5	3,281,000	3.97	3.5	23.2	13,025,570	418,775
3.0	2,184,000	4.6	4.07	26.4	10,046,400	322,994
3.5	1,406,000	5.36	4.77	29.2	7,536,160	242,289
4.0	986,000	6.06	5.43	31.5	5,975,160	192,103
4.5	705,000	6.79	6.1	34.4	4,786,950	153,901
5.0	546,000	7.39	6.66	36.6	4,034,940	129,724

Source: Western Services Engineering, Inc. **Note**: AuEq is based upon 50:1 Au:Ag.

In 2004, Beacon Hill (1988) performed a preliminary assessment of the Kremnica property and Smith and Kirkham authored a NI 43-101 compliant report. The primary differences in approach between Smith and Kirkham's report and previous reports lay in the methods for controlling the interpolation and for masking the model; in Smith and Kirkham's report, these were based on the geology as defined by the geologic sections, which were deemed to accurately represent the deposit. In addition, no cut-off grade was applied to the raw assay data or the composites prior to interpolation. This methodology differs from those of previous studies, which employed a 0.5 g/t gold equivalent mineralization envelope for interpolation along with a 0.5 g/t preprocessing cut-off (applied to the composites). Previous methods tended to be biased, producing a higher grade and lower tonnage by eliminating low grade or diluted zones.

In addition, the calculation utilized for gold equivalent in this report is based on a price of US$385/oz for gold and US$5.00/oz for silver, whereas the Western Services report is based on a price of US$300/oz for gold and US$5.00/oz for silver.

Due primarily to the direction of continuity (i.e., search ellipse direction), the mineralized blocks extend further to the north and south, as well as deeper into the vein. In addition, in previous modelling exercises, it appears that the quartz breccia unit was masked out as waste, whereas it is now evident that this unit contains mineralization and should be included. Tables 6.4 and 6.5 list the 2004 resources for the Šturec deposit.

Table 6.4: 2004 Indicated Resources for the Šturec Deposit

Cut-off Grade g/t AuEq	Indicated						
	Tonnes	**AuEq**	**SG**	**Au**	**Ag**	**Grams**	**Ounces**
0.5	9,820,697	1.64	2.31	1.49	11.46	16,124,819	518,416
1.0	5,663,941	2.31	2.28	2.11	15.43	13,111,593	421,540
1.5	3,549,685	2.97	2.27	2.72	19.13	10,533,980	338,670
2.0	2,238,598	3.69	2.26	3.40	22.19	8,260,140	265,565
2.5	1,536,456	4.35	2.27	4.02	25.80	6,689,576	215,071
3.0	1,074,301	5.05	2.28	4.67	29.05	5,427,490	174,495
3.5	744,439	5.86	2.30	5.44	32.44	4,360,903	140,204
4.0	551,892	6.61	2.31	6.14	36.12	3,645,457	117,202
4.5	430,437	7.27	2.30	6.75	40.18	3,129,394	100,611
5.0	361,437	7.75	2.28	7.20	43.00	2,802,439	90,099

Source: Smith and Kirkham. **Note**: AuEq is based upon 77:1 Au:Ag.

Table 6.5: 2004 Inferred Resources for the Šturec Deposit

Cut-off Grade g/t AuEq	Inferred All						
	Tonnes	**AuEq**	**SG**	**Au**	**Ag**	**Grams**	**Ounces**
0.5	16,109,256	1.49	2.33	1.35	11.32	24,072,980	773,951
1.0	8,367,508	2.22	2.30	2.01	16.06	18,568,081	596,968
1.5	4,924,526	2.92	2.27	2.66	20.26	14,375,273	462,168
2.0	3,153,432	3.59	2.25	3.29	22.93	11,309,250	363,595
2.5	2,154,214	4.22	2.25	3.90	25.08	9,095,013	292,407
3.0	1,301,835	5.18	2.25	4.81	28.81	6,745,498	216,869
3.5	898,651	6.06	2.27	5.64	32.40	5,450,094	175,222
4.0	681,989	6.82	2.27	6.37	34.32	4,649,751	149,490
4.5	527,775	7.57	2.28	7.14	32.89	3,993,118	128,380
5.0	439,473	8.13	2.29	7.69	33.58	3,572,640	114,861

Source: Smith and Kirkham. **Note**: AuEq is based upon 77:1 Au:Ag.

The resources shown in Tables 6.4 and 6.5 meet the requirements for the categories of "indicated" and "inferred" resources, as defined by the CIM. The reliability of these resources is very good; however, a more recent estimate has been calculated for this report and is described in Section 17.

SECTION 7.0 – GEOLOGICAL SETTING

7.1 REGIONAL GEOLOGICAL SETTING

Much of the gold produced in central and eastern Europe is associated with Tertiary volcanic rocks that were erupted along the Carpathian Arc subduction zone. Slovakia occurs at the northern apex of the Carpathian Arc. Volcanic sequences associated with the Carpathian Arc occur in central and eastern Slovakia. The largest of these is the Central Slovak Volcanic Field, which is about 60 km in diameter.

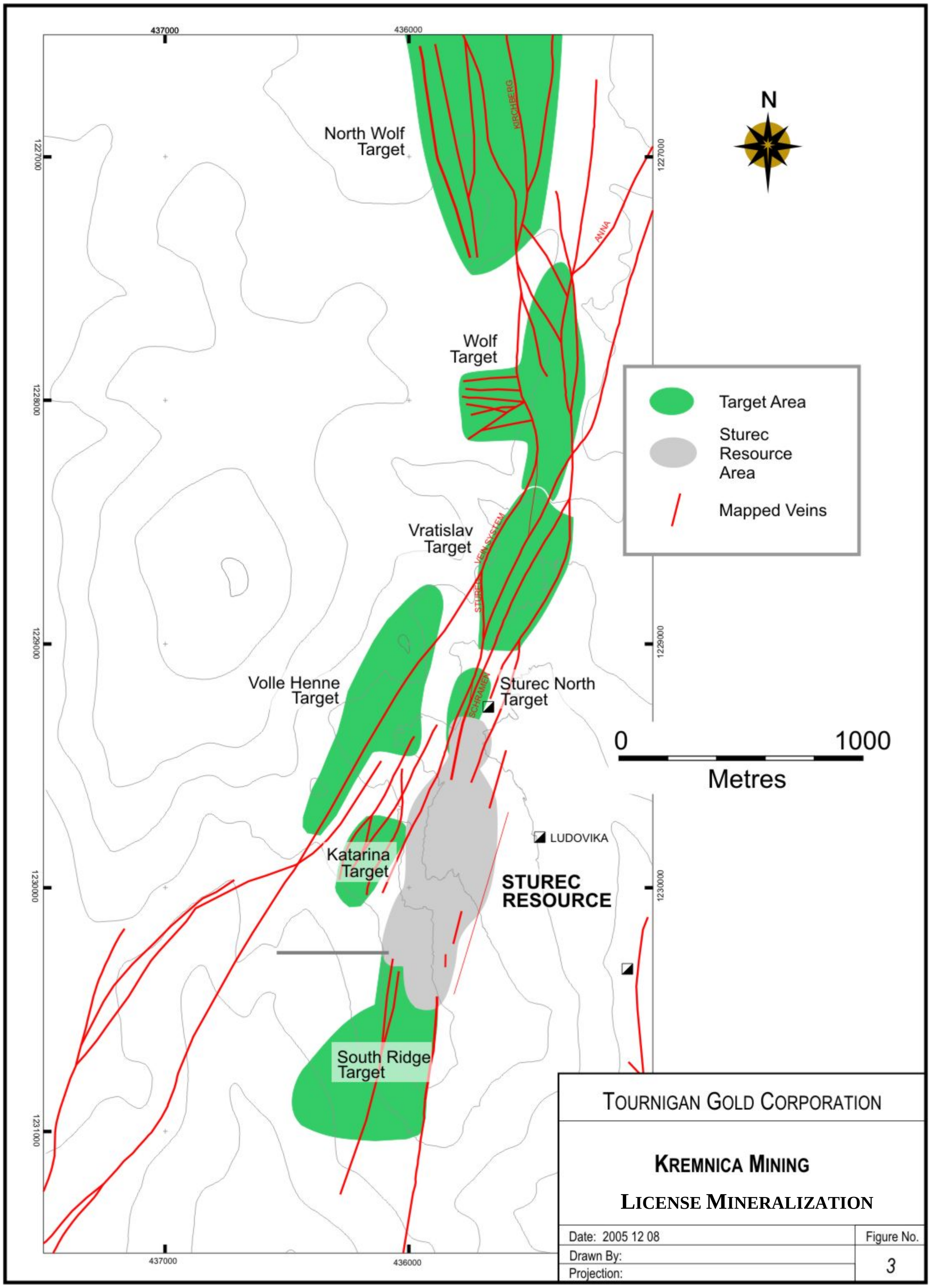
The Kremnica district is located near the northern margin of the Central Slovak Volcanic Field within an area dominated by north to north-northeast trending faults and post-andesite resurgent rhyolite domes and dikes. The faults are normal, extensional, and form a series of horsts and grabens that are extensions of the Banska Stiavnica caldera complex that lies several kilometers to the south. Older east-west-striking, low-angle thrust faults associated with the closure of the Carpathian Arc have been mapped peripheral to the volcanic field and are projected beneath it.

7.2 KREMNICA REGIONAL GEOLOGY

The predominant host rock in the Kremnica district is Tertiary andesite. It occurs as flows with minor interbedded tuffs and breccias. Diorites have been intersected in some drill holes and are thought to be both pre- and post-andesite. Rhyolite dikes are localized in north-south and southeast-striking structures. Rhyolite is relatively rare in the Šturec deposit, occurring as narrow dikes at the north end of the deposit and at depth. The Tertiary volcanic sequence overlies Mesozoic limestone, which has been detected in some of the deeper drill holes in the district. These rocks are cut by north- to northeast-striking, steeply dipping faults that form a series of horsts and grabens.

As shown on Figure 7.1, there are two major vein systems at Kremnica. The principal system, called the “First Vein System,” strikes north to north-northeast through the center of the district, and is the focus of the exploration activity. The “Second Vein System” is near the town of Kremnica and consists of north- and northwest-striking veins. Due to its location beneath the town of Kremnica, the second vein system is not considered a viable exploration target.

Figure 7.1: Kremnica Mining License Mineralization



SECTION 8.0 - DEPOSITS TYPES

As reported by Smith and Kirkham (2004), gold-silver mineralization at Kremnica is part of a large low-sulphidation quartz-sericite-adularia epithermal-hydrothermal system hosted in Tertiary andesite volcanic flows and tuffs and lesser diorites and rhyolite dikes. Host volcanic rocks are part of the northwest extremity of the relict Tertiary Carpathian Volcanic Arc that extends irregularly south from Slovakia to Turkey.

Mineralization occurs in large banded to massive quartz veins, smaller quartz veins and sheeted veins, quartz stockwork veining, and silicified hydrothermal breccias. Geological work completed by Tournigan in 2005 has demonstrated that gold and silver mineralization within the sheeted veins and stockwork veining zones is primarily localized in areas immediately adjacent to the main vein zones.

Vein mineralogy consists of quartz, calcite, adularia, sericite-illite, and lesser chalcedony. Vein-calcite is typically evidenced by quartz-after-calcite pseudomorph textures.

Alteration consists of a core of intense silicification (abundant quartz veining and silica flooding of vein wall rock), and large zones of argillic and propylitic clay alteration, which can include minor disseminated pyrite. Silicification is primarily quartz with lesser chalcedony.

Gold occurs freely and in non-refractory association (coatings, etc.) with sulphides and with silver as electrum. Besides electrum, silver occurs in the minerals polybasite, pyrargyrite, and argentite. Sulphide minerals consist predominately of pyrite and marcasite with much lesser amounts of chalcopyrite, arsenopyrite, stibnite, sphalerite, and galena. Sulphide concentrations rarely exceed 2% and average 0.5%. Average gold grades throughout the deposit are approximately 2 g/t, but high-grade zones can exceed 30 g/t. Silver/gold ratios vary, but average approximately 8:1.

Large mineralized banded to massive quartz veins and associated silica, argillic, and propylitic alteration zones are localized along a major, broad structural zone that strikes approximately north to northeast and is mineralized for a length of at least 6.5 km. Some 80 veins are documented within the Kremnica vein system, with individual vein groups being up to 100 m thick.

The Kremnica vein system is described as occurring within a horst structure that is bounded by a larger, district-wide graben structure. Horst-graben structures imply normal and/or reverse faulting, but earlier, pre-mineralization strike-slip movement within the structural zone may have occurred. This relation could have implications for future exploration within and outside the documented areas of mineralization.

As reported by Smith and Kirkham (2004), a large area of argillic/propylitic alteration associated with a rhyolite flow-dome complex containing broad, low-level Au, Ag, and As-Sb-Hg-Tl soil geochemical anomalies occurs approximately 5 km south of, and on strike with, the main part of the Kremnica deposit. This area may represent the Kremnica mineralized volcanic/hydrothermal system at higher volcanic-stratigraphic and epithermal-hydrothermal levels.

SECTION 9.0 - MINERALIZATION

9.1 ŠTUREC, WOLF AND VRATISLAV

9.1.1 Šturec Deposit

The Šturec deposit (Figure 7.1) occurs in the southern part of the central First Vein System. The Šturec deposit is continuously mineralized for 1,200 m along strike, is typically 100 to 150 m wide, and extends to a known depth of at least 300 m. The deposit is open to extension both at depth and to the north and south. The heart of the deposit is the Schramen massive to sheeted quartz vein, which is up to 100 m wide along a 500 m strike section. It strikes almost due north, generally dips steeply to the east, and thins to the north, south, and at depth.

The second important element of the Šturec deposit is a northeast-striking quartz vein system that joins with the northern part of the Schramen vein. This vein system projects southwest away from the Schramen vein where it outcrops approximately 100 m west of the Schramen vein. It then bends to the south and strikes parallel to the Schramen vein. This vein system dips 40° to 55° east, re-joining with the Schramen vein at depth. Zones of stockwork gold mineralization occur between the two principal veins. There are also numerous late crosscutting veins.

The effects of past mining are an important feature in the Šturec deposit. Substantial portions of the two main vein systems have been mined from both the surface and underground. A large area of subsidence now exists in the upper part of the deposit, and parts of the veins that were mined are filled with material that in many places could be considered “ore” today. When inspecting drill core or RC chips it is frequently difficult (or impossible) to distinguish between subsidence blocks and old mine backfill.

Hydrothermal breccias are closely associated with the principal vein systems in the district. They are usually composed of quartz vein material, strongly silicified andesite, and (rarely) rhyolite clasts cemented by iron-sulphide-bearing silica. They occur predominately adjacent to the veins but breccias can merge into veins vertically and/or along strike.

9.1.2 Wolf Target

As reported by Smith and Kirkham (2004), the country rock at Wolf is similar to that at Šturec, with a significant increase in the volume of rhyolite. Two large, north- to northeast-striking rhyolite dikes have intruded the andesites along predominately north-south structures. The rhyolites are very well mineralized in areas where they are intersected by, or run parallel to, the veins. This mineralization takes the form of silicification, quartz veining, and silicified hydrothermal breccias. The Wolf target also contains numerous voids and rubble zones, similar to those encountered at Šturec.

As at Šturec, gold mineralization occurs primarily in quartz veins that have the same north to northeast strike as the rhyolite dikes, and dip moderately to steeply to the east. At depth these veins are interpreted to merge into one moderately east-dipping structure. At the Wolf target, mineralization is defined for 300 m along strike, and is at least 50 m wide and extends to a depth of at least 50 m. The widest vein is the Kirchberger at approximately 30 m. The mineralogy of the deposit is similar to the Šturec deposit, although considerably more rich in silver.

A second sequence of veins at the Wolf target strikes east-west, bisecting the rhyolite dike on the footwall of the Kirchberger vein and projecting into andesite wall rock. Pits that exploited the veins in historic times become shallower to the west. Thin, sparse stockwork veins have also been observed and sampled within the rhyolite. Seven outcrop samples collected in 1996 and 1997 returned assay values of up to 19.0 g/t gold, illustrating the high-grade nature of these veins. During 1997 drilling of the Kirchberger structure only one drill hole intercepted significant mineralization in the footwall. Hole AS-134 intercepted thin, amethyst-quartz veins sub-parallel to the core axis, with minimal hydrothermal alteration in the surrounding andesite. The veins assayed 8.0 m at 2.82 g/t gold from the downhole interval 81.5 to 89.5 m.

9.1.3 Vratislav Target

As reported by Smith and Kirkham (2004), the Vratislav target is located between the Šturec deposit and Wolf target. Three major veins have been identified underground by previous historic mine operations. The veins all strike north-south and are splays off the Schramen vein. The Schramen vein is the easternmost structure and the Schindler vein the westernmost splay, dipping back to the east at 40° to 50° and intersecting the Schramen vein at depth. The Schindler is sometimes referred to as the Vladimir vein in this area. A second major vein, the Teich vein, splays off the Schindler vein in the Vratislav area. The Teich vein is steeply dipping and occupies the same spatial position as the major structure in the Šturec resource. The veins are surrounded by low-grade stockwork mineralization.

Recent and historic underground data indicate the Schindler vein is 4 to 10 m thick and grades from 1.5 to 2.5 g/t gold. In outcrop the vein is predominately quartz, banded, porous and vuggy, and colored yellow-brown-black from limonite Mn-oxide alteration. To the north this vein is thought to be the same as the Kirchberger vein. The Teich vein branches off the Schindler at a steep angle and becomes approximately vertical. Historic sampling by the Slovakian Geologic Survey adjacent to the veins indicates the presence of low-grade stockwork mineralization with gold grades in the 0.25 to 2.1 g/t range.

The Vratislav area has been extensively mined, as evidenced by old surface workings. A large pit-wall scarp exists where the Schindler vein was stripped from the hillside, similar to, although not as large as, the scarp at Šturec. An elongated, 20 m deep pit indicates where some of the Teich veins were exploited at the surface. It is not known how extensive the historic underground mining was along these veins.

9.1.4 OTHER MINERALIZED AREAS WITHIN THE KREMNICA MINING LICENSE

9.1.5 South Ridge Target

As reported by Smith and Kirkham (2004), geologic mapping indicates that the main structure, the Schramen vein, continues to the south as depicted in Figure 7.1. However, it changes character from the typical white, vuggy banded epithermal quartz vein exemplified at the Šturec depression to a more chalcedonic-pyrite silicified breccia to the south. Several splays also occur and the average grade of the mineralization drops. The western footwall side of the South Ridge zone is defined as a moderately east-dipping (±45°) mineralized structure (vein system) that converges with the Schramen vein at depth and along strike to the north. At 1,230,470S on the mine grid, a major cross structure diverges to the southwest. It branches off the Schramen vein and can be traced on the surface over the ridge crest and down the west flank of the ridge to the field above the village of Lúĉky . The vein system appears to be composed of three or more sets of veins striking between 35° to 80° northeast, and dips at between 65° to 70° to the south-southeast. The veins are surrounded by zones of intense stockwork veining and silicification. Between these two major structures is a wedge-shaped block that contains stockwork vein mineralization and large crosscutting (ladder) veins

Seven reconnaissance samples collected by Argosy in 1996 and 1997 contained gold grades ranging from 0.53 to 5.26 g/t and averaging 1.78 g/t gold. The South Ridge target is about 200 m wide at the surface where it abuts the Šturec resource and narrows to the south along the projections of the Schramen and footwall vein systems. The major cross structure, which appears to diverge from the Schramen vein at 1,230,500S, is about 20 m wide near the ridge crest and appears to fan out for the next 200 m. Soil survey data indicate that the target may extend 500 m further southwest toward the village of Lúĉky.

9.1.6 North Šturec Target

As reported by Smith and Kirkham (2004), the North Šturec target occurs north of the Šturec deposit and along a portion of the vein system extending north and west of the areas drilled by Argosy (Figure 7.1). The area may contain a faulted-off portion of the Šturec deposit. The target has been defined by the coincidence of mineralized outcrops and geochemical anomalies. A bend occurs in the Schramen vein, as illustrated on the Argosy level plans used in the construction of the 1997 resource model. Two outcrops of quartz vein have been found in the target area. The vein contains alternating bands of solid white and porous vuggy limonitic quartz and is estimated to be up to 10 m wide. Two samples from the vein, collected within old workings, contained gold (0.35 and 1.05 g/t) and silver (42.1 and 24.3 g/t).

9.1.7 Volle Henne Target

As reported by Smith and Kirkham (2004), the Volle Henne target is located northwest of the Šturec resource, as illustrated in Figure 7.1. The target was identified by old underground and surface workings, soil geochemistry, and rock chip geochemistry from outcropping quartz veins. The area of surface and underground workings is approximately 200 m wide x 300 m long; however, mineralization may continue both southwest and northeast to join the Katarina and Vratislav targets.

Although only reconnaissance work has been completed, the geology of the area appears to be typical of many of the Kremnica vein systems. It is characterized by a large dominant north- to northeast-striking, east-dipping structure, and thinner, steeply dipping vein offshoots and stockwork vein mineralization. The dominant structure appears to be a ±3 m wide, low-grade quartz vein that strikes between 20° north to 50° northeast. It dips steeply (70°) to the east and marks the western edge of the soil anomaly and significant old surface workings. Alteration in the area is comprised of weak argillization of the host andesite with local areas of silicification.

The extensive areas of underground and surface workings and the occurrence of stockwork zones in outcrop indicate the possibility of finding another stockwork vein resource similar to the South Ridge area. The historic miners were probably following relatively thin (up to a few meters wide), high-grade veins containing coarse (visible) gold. These veins were ideal for them, as they could follow the veins easily and recover the gold without difficulty. Zones of relatively low-grade stockwork mineralization have been observed around the veins that were mined.

9.1.8 Katarina Target

As reported by Smith and Kirkham (2004), the Katarina target is located west of the Šturec resource, as illustrated in Figure 7.1. The Katarina target lies beneath an ancient open pit. Old adit plans also show a dense network of tunnels under the target area. The size of this target has been reduced in size due to poor results from drilling five holes in the central and northern parts of the Katarina system. There is now a 150 m x 100 m area where it may be possible to find near-surface mineralization.

The Katarina system contains discrete, narrow (up to a few meters wide), high-grade quartz (±carbonate) veins, commonly with visible gold. The veins strike in a north-northeast direction and appear to be near vertical or dipping steeply to the west. Geological mapping suggests that the vein system splays and weakens to the north, converging into larger structures in the south. Some diffuse stockwork mineralization has also been observed.

A soil sampling program conducted during 1997 produced a 150 m x 400 m anomaly. The majority of the soil samples contained concentrations of over 100 ppb gold with maximum values over 1,000 ppb gold. Five reconnaissance rock chip samples contained up to 9.5 g/t gold.

SECTION 10.0 - EXPLORATION

As described in Section 6.2, modern exploration in the Kremnica district was conducted by the Slovak Geological Survey and Rudne Bane, the State mining company responsible for mining in the district. Exploration work was initiated in 1962 and conducted intermittently through to 1990. This work included driving four major exploration adits, more than 20 crosscuts, and both surface and underground drilling. This exploration is detailed in Section 11 “Drilling.”

In 1997, Argosy conducted soil sampling within the mining license boundary in the Katarina and Volle Henne areas. A total of 135 samples were collected on 25 m intervals along gridlines 200 m apart. Samples were assayed for both gold and silver, although the silver results were not available to this author. The program defined a strong (+250 ppb) gold-in-soil anomaly 150 m wide x 800 m long, striking north-northeast and open to the north and south. Silver results were not reviewed by this author.

During 2004, Tournigan completed a drilling program in the Kremnica South area. Nine drill holes totalling 2,037 m were completed at Certov vrchy and two holes totalling 421 m were completed at Bartasova Lehotka. Based on encouraging results from the drilling program, Tournigan applied for, and acquired, additional exploration concessions southwest of the Kremnica South area.

During 2004 to 2005, Tournigan also completed a large soil geochemistry survey over the Kremnica South area along a 40 m x 200 m grid.

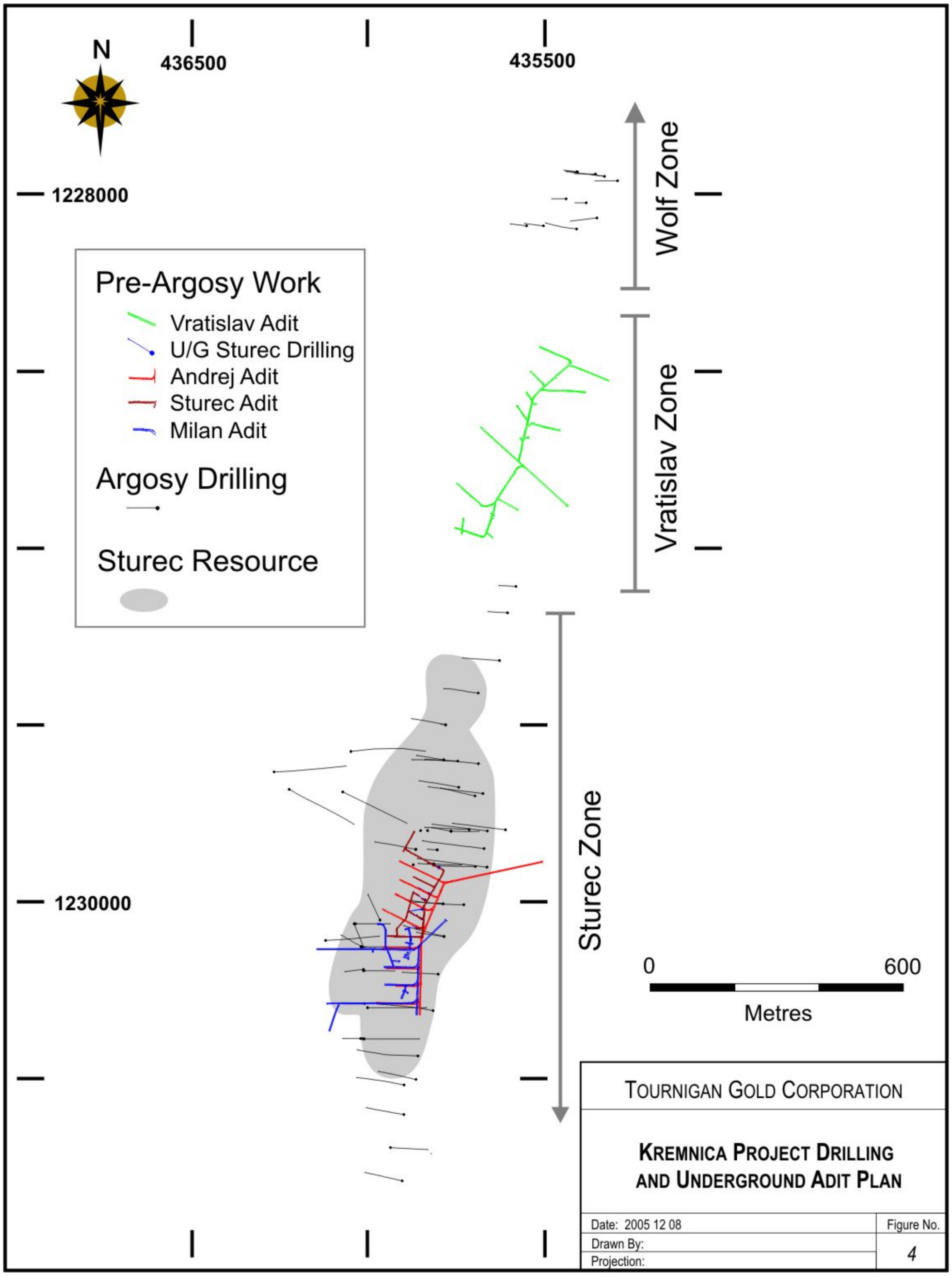
SECTION 11.0 - DRILLING

The drilling, surface, and underground sampling database is derived from four main sources:

- underground drilling and channel sampling of various crosscuts by Rudne Bane and Slovak Geological Survey
- Argosy Mining Corporation diamond core drilling program; throughout the property there are a total of 174 drill holes with 10,572 assay intervals and 40 crosscuts with 3,148 assay intervals
- drilling from the 2005 Tournigan campaign that resulted in 41 infill RC holes including five twin holes increasing the assay and lithology database by 3,989 sample intervals
- nine bench channels incorporating a total of 317 sample intervals were also collected during 2005.

Figure 11.1 shows the drill hole locations from the Argosy Mining diamond core drilling program, as well as underground adits with continuous channel sampling.

Figure 11.1: Drilling and Underground Adit Plan



11.1 RUDNE BANE AND SLOVAK GEOLOGICAL SURVEY PROGRAMS

During the extensive mining history within the Kremnica District mine production was conducted using both open pit and underground techniques. During and subsequent to the late phases of mining the Slovak Geological Survey and Rudne Bane conducted intermittent exploration within the Šturec deposit above the 500-m elevation level. This work consisted of driving a series of underground crosscut adits along and across strike of the Šturec deposit. Underground fan drilling of diamond drill core drilling from drill stations within the crosscuts was also completed. A series of diamond drill holes collared at the surface were completed as well.

11.1.1 Rudne Bane Programs

Rudne Bane was the last company to mine the Kremnica deposit prior to acquisition by Argosy Mining Corporation in 1995. During limited production Rudne Bane conducted underground sampling of the larger mineralized portions of the Šturec deposit and underground fan drilling of the northern-most known limits of the deposit.

Underground Rib Channel Sampling

Rudne Bane drove a series of crosscuts through the Šturec deposit in order to obtain continuous channel samples collected at approximately one-meter intervals through the core of Šturec mineralization. A total of 40 lines of continuous channel samples were collected. The forty lines included a total of 2,709 assayed channel sample intervals. The samples were analyzed at Rudne Bane's internal laboratory using fire assay with a gravimetric finish. The resultant assays were compiled on surveyed plan view maps. Pulp rejects are currently stored in a locked, secure building on site at the Kremnica mine. As part of the site visit the author collected a suite of 37 pulp rejects from the crosscut channel samples. Results of this program are detailed in the following sections.

Underground Diamond Core Drilling Program

As reported by Smith and Kirkham, 2004, Rudne Bane drilled a total of twelve diamond drill core holes from the underground crosscuts at the northern limits of the mining area. A total of 226 sample intervals were assayed for gold and silver. The total drilled meters was 425.3m. Samples were analyzed by Rudne Bane at their internal laboratory using fire assay with a gravimetric finish. The drilling equipment was Russian designed core drilling systems. The results from these drill holes could not be verified however based on the results of the verification sampling completed on the underground crosscut channel sampling program which are of the same vintage as the channel sampling data, data from these drill holes was considered to be reliable. Therefore, these holes were utilized within the resource estimate as they were in the previous estimates.

11.1.2 Slovak Geological Survey Programs

Underground Diamond Core Drilling Program.

As reported by Smith and Kirkham, 2004, a total of 48 underground drill holes were completed by the Survey from underground drill stations within the Rudne Bane crosscuts. Of these, 38 holes intersected intervals that were assayed for gold at the Survey's lab using wet chemistry analytical analysis. A total of 1003 intervals were analyzed. The total underground-drilled meters were 3362.4m. The drilling equipment was Russian designed core drilling systems. The results from these drill holes could not be verified however 9 of the holes namely VKB-3, VKB-3R, VKB-4, VKB-4A, VKB-4B, VKB-2, VKB-2A, VKB-2B, and VKB-7 have intersections within close proximity with RC holes and Argosy surface diamond drillholes. For these underground drillholes, it is impossible to twin them with surface holes, however the intersection with the selected RC and Argosy data allow a virtual twin as shown in Figures 11.2 through 11.5. This allows confirmation of similar grades within the areas of intersection and is sufficient to compare assay intervals. The remaining 39 of the 48 holes are outside the modeling area and the assay data is not included in the Kremnica resource estimate.

Figure 11.2: Intersection of KGST-2R and VKB-7 for Assay Comparison and Confirmation

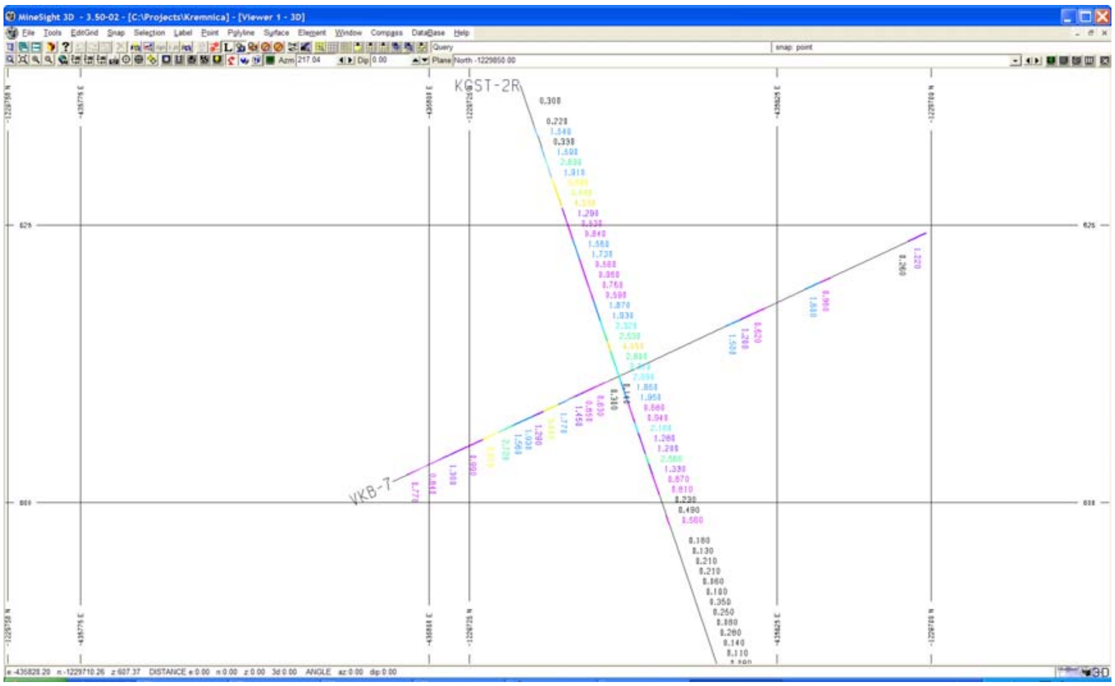


Figure 11.3: Intersection of AS-8, AS-8.1.B and VKB-2. VKB-2A, VKB-2B for Assay Comparison and Confirmation

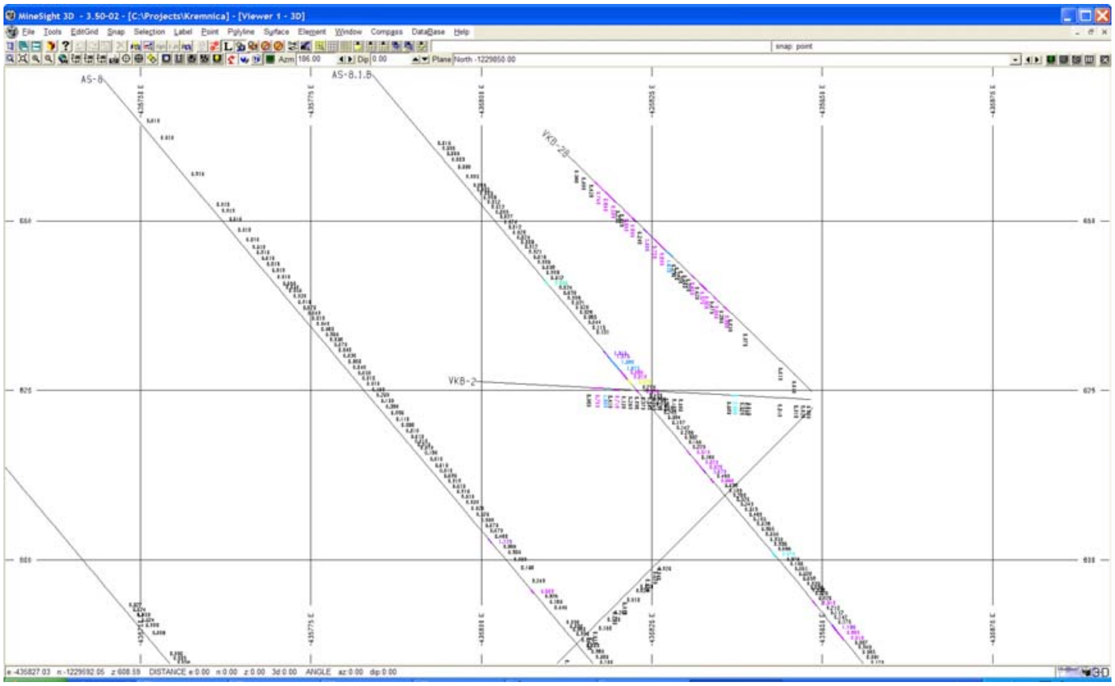


Figure 11.4: Intersection of KGST-3R and VKB-3 and VKB-3R for Assay Comparison and Confirmation

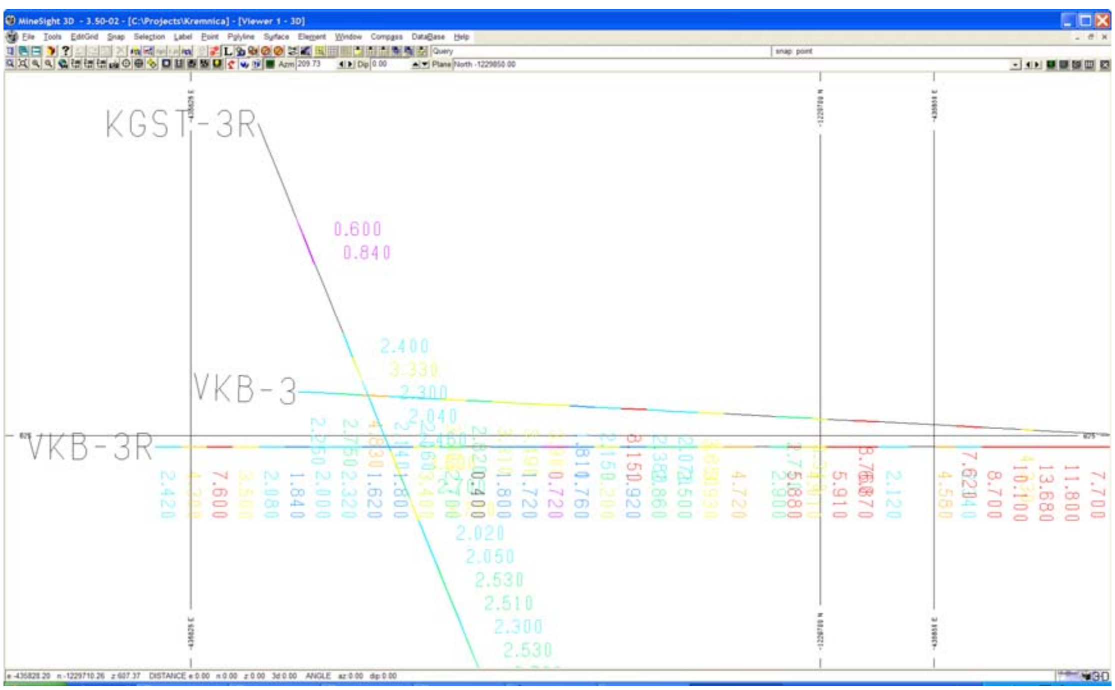
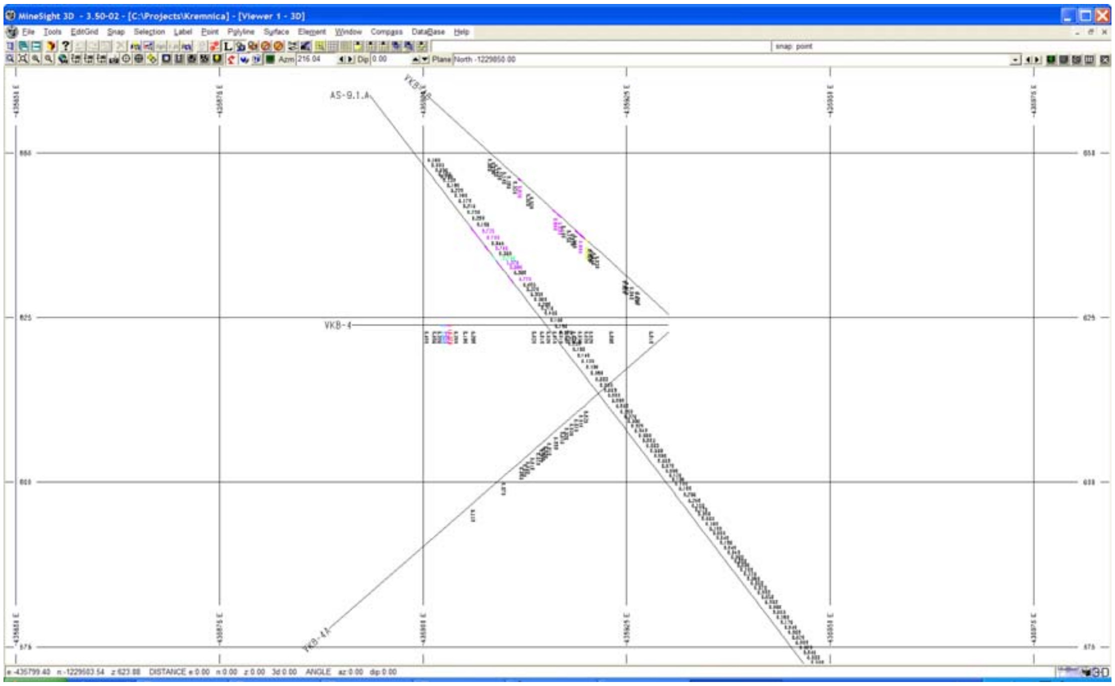


Figure 11.5: Intersection of AS-9.1.A and VKB-4, VKB-4A, VKB-4B for Assay Comparison and Confirmation



Surface Diamond Core Drilling Program.

As reported by Smith and Kirkham, 2004, the Slovak Survey also drilled a total of 35 surface collared diamond core holes. The only information available up until this time is drill collar locations and orientations. These holes are not within the area defined for the calculation of resources for the Sturec deposit.

11.2 ARGOSY MINING CORPORATION DRILL PROGRAMS

Argosy completed drilling programs in 1996 and 1997. A Longyear 44D diamond drill rig completed the 1996 drill program. The 1997 program was completed using two Acker 5 Uni-drills. One was Austrian-owned, crawler mounted and capable of drilling both reverse circulation and diamond core drilling. The second was Slovak-owned, and capable of drilling core only.

The 1996 and 1997 Argosy programs were completed in accordance to western mining standards. The drill core was logged by Argosy geologists and specific sample intervals were identified for splitting and subsequent assaying. Down hole surveys were obtained using gyroscopic down hole survey camera, and all drill hole collars surveyed by theodlite. Table 11.1 summarizes the Argosy drill programs.

Table 11.1: Argosy Mining Corporation Drill Hole Summary for Kremnica Project

Deposit	1996 Program		1997 Program		Total	
	Holes	Meters	Holes	Meters	Holes	Meters
Šturec	37	5161.2	25	4221.2	62	9382.4
Katarina	2	607	3	703.5	5	1310.5
Wolf	0	0	12	1613.8	12	1613.8
Total	**39**	**5768.2**	**40**	**6538.5**	**79**	**12306.7**

11.3 RESULTS OF HISTORIC DRILLING AND UNDERGROUND SAMPLING

As reported by Smith and Kirkham, 2004, the main result of the drilling and underground sampling programs completed prior to 2005 has been the definition of an Indicated Resource of 518,416 oz gold equivalent and an Inferred Resource of 773,951 oz gold equivalent, (assuming 0.5g/t cut-off) contained within the Šturec deposit. The geological resource has been defined in accordance with NI 43-101 regulations.

SECTION 12.0 – SAMPLING METHOD AND APPROACH

12.1 SAMPLING METHOD

Prior to completion of the 2005 program there was a combined total of 214 drill-holes and crosscuts, 3,540 surveys, 1,414 lithologies and 13,720 assay intervals which make up the Kremnica data base. All drill hole locations have been marked in the field with concrete covers with metal rods showing the drill orientations and drill hole numbers. Some have been lost to erosion or vegetation over-growth, but most are still clearly identifiable. Detailed descriptions of sample location, nature of material sampled, representative characteristics of the sample, type of lithology, alteration, structure, and mineralization, if any, were recorded from drill core and crosscut continuous channel samples, and were supplied to, and reviewed by, the author of this report. Most drill core was sawed or split, with half cores sampled to geological boundaries, preferentially at meter intervals. Underground crosscuts were sampled by continuous channel sample using hammer and chisel, as is commonly used by industry. Again crosscuts were sampled to geological boundaries preferentially at meter intervals.

The 2005 program completed 41 hole infill RC drill program added a total of 3,989 assay intervals and lithologies to the database for the project.

12.2 DRILL HOLE ORIENTATIONS

According to Smith and Kirkham, 2004, recent work by Boris Bartalsky, resident Slovak Geologist and Project Manager for both Argosy and Tournigan, suggests that a great deal of early historic Au production at Kremnica was from sets of narrow northeast-striking veins localized in the hanging wall of the main east-dipping Šturec vein. These veins are abundant and apparently occur throughout the hanging wall block and are typically higher grade than the other veins. Drilling by Argosy was uniformly oriented with east or west azimuths and therefore may have systematically failed to sufficiently account for these vein sets. The 2005 RC drilling program was designed to not only supply in-fill drilling data in order to upgrade the resources but also oriented so as to intersect areas within the hanging wall and footwall stockwork and andesite thereby delineating the areas of higher grade mineralization.

12.3 CORE RECOVERY

There were no records of recovery within the core holes drilled by the Slovak Geological Survey. The Rudne Bane underground drift and cross-cut sampling data has no recovery data for the obvious reason that this sampling method (hammer and chisel) is assumed to be 100% recovery.

According to the Western Services 1997 Report during the 1996 exploration program, Argosy did not record the percentage of core recovery. Western Services Engineering later recommended that Argosy record the core recovery within all Argosy holes to determine if a bias was introduced into the grade. In 1997, Argosy logged core recovery of the 1996 diamond drill core as well as the ongoing 1997 diamond drill drilling.

Examination of the newly calculated recoveries by Western Services reveals that the 1997 drill holes were not logged by individual core run but rather recovery over a specific interval. The 1996 data was logged by grouping large runs of core into a single interval and assigning it the average recovery of the run. Core recovery data was available for the Rudne Bane underground drilling program and is assumed to be calculated on the same basis as the 1996 Argosy data. Both practices are atypical of core recovery logging. Analysis of the available logged recovery data was completed for those intervals that were mineralized (0.5g Au/t equivalent). The examination was split into two specific analyses. The first was comprised solely of the 1997 drill intercepts that better approximated the core run intervals. This group contained a total of 587 intervals. The second was an evaluation of all Argosy holes and contained a total of 1,290 intervals. The average core recovery was 86.5% for the 1997 drill core. According to Western Services the global mineralized data set had an average core recovery of 85.7%.

According to the Western Services 1997 Report an examination of plots of gold and silver vs. core recovery indicates that as core recovery decreases the grades also decrease. In particular drill-core recovery through some broken and brecciated zones within mineralization was often poor. Drill core inspected by the author revealed several intervals of strongly silicified and brecciated and/or broken material that would likely be well mineralized had core recoveries of only 5% to 10%. Poor recovery of well-mineralized zones can potentially result in mineralized material not being collected and analyzed and therefore being unreported or under-reported.

As part of the 2005 program, Tournigan personnel examined the Argosy drill core stored on site in an effort to compare historically recorded recoveries with material remaining within the core boxes. As an integral part of this exercise, facilities for storing all the core boxes were constructed for easy access and identification in the future. In addition, a map of the facilities was created along with a report for each drillhole listing intervals and approximate material remaining. Finally, digital photographs were taken of each core box and are available on CD.

SECTION 13.0 - SAMPLE PREPARATION, ANALAYSIS AND SECURITY

There are no good records of sample preparation and analysis methods for the early work done by Rudne Bane and the Slovak geological survey. Re-analysis of the Rudne Bane channel pulps by Argosy confirms their validity however. The author visited the sample storage facilities at the Kremnica site. Based on observations there a methodical approach by the Slovaks to sample preparation is suggested. All the samples are stored in a dry locked storage within the old mine buildings. The underground channel pulps are in very good condition, well laid out, clean, clearly marked and neatly stored. The Kremnica mine buildings also have 24-hour security.

During the 1996 drilling program conducted by Argosy, all sample intervals were shipped for sample preparation and analyses by either SGS France (internationally certified laboratory) or the Slovak Geological Survey (uncertified internal laboratory). Standardized checks, blind assays, and blanks were not implemented in the 1996 program.

During the 1997 program, Chemex set up a certified sample preparation facility and trained staff on the Kremnica site. Mr. Ken Bright (Chief Geochemist) of Chemex's Vancouver office inspected the facility. Mr. Bright confirmed that the facility and defined sample preparation procedures were acceptable. The facility was not ready at the start of the 1997 program and Argosy had to utilize the Geological Survey's sample prep facilities during the early stages of the program. The Survey also assayed the early samples. The Survey results were used by Argosy to identify mineralized intervals ($\geq$0.5 g/t Au). These intervals were shipped to Chemex for determination of the final assay value. Argosy submitted standards and blind duplicates to the Survey and Chemex to determine the quality of the results.

During the 2005 program Tournigan utilized the on site sample preparation facility to process all of the reverse circulation drill samples. RC samples were shipped to OMAC Laboratory in Loughrea, Ireland, a subsidiary of Alec Stewart Laboratories.

Today, all remaining pulps from the Rudne Bane underground sampling program, all remaining core splits and sample pulps from the Argosy programs and all coarse rejects and pulps from the 2005 program are stored in secure buildings on the Kremnica mine site. Many drill core pulps have been removed during a series of re-sampling programs. Several mineralized intervals in the core have been completely removed and sampled for metallurgical or re-sample purposes.

SECTION 14.0 - DATA VERIFICATION

This section is divided into two parts; namely a small control sampling program and a systematic evaluation of the global data set discussed in detail in section 17.2 Assay Database.

14.1 CONTROL SAMPLING

During July 2005 the drilling of Drill Hole Number KG-ST-7R and KG-ST-8R was observed, samples extracted and sealed in sample bags containing all of the material recovered from each one meter interval drilled. A total of 67 sample splits from the RC drilling program were submitted to ALS Chemex for comparison with the data generated by the lab used by Tournigan for analysis of all samples from Kremnica, OMAC Assay Laboratory in Loughrea, Ireland (subsidiary of Alex-Stewart Laboratories). Figures 14.1 and 14.2 shows the results for gold and silver, respectively which exhibit an excellent correlation between the two laboratories resulting correlation coefficients of 0.9304 and 0.964 for gold and silver, respectively.

Figure 14.1: Au Check Assay Results; OMAC vs. Chemex

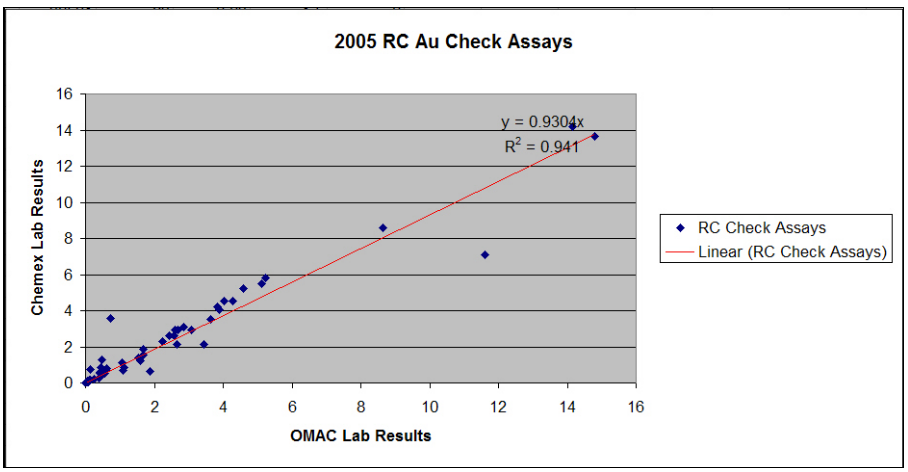
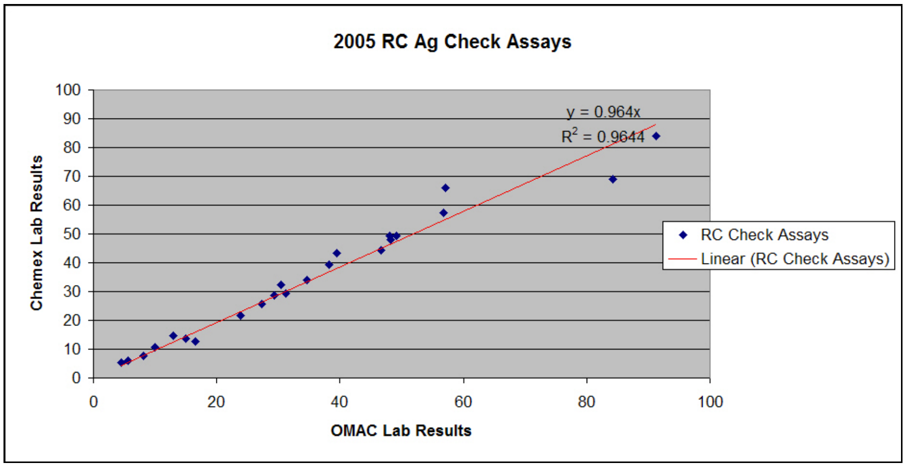


Figure 14.2: Ag Check Assay Results; OMAC vs. Chemex



In addition, 97 pulp duplicates were run during the RC drilling program and with the exception of 3 Au and 3 Ag duplicates above cut-off, the results are within acceptable tolerances as shown in Figures 14.3 and 14.4.

Figure 14.3: Au RC Duplicates

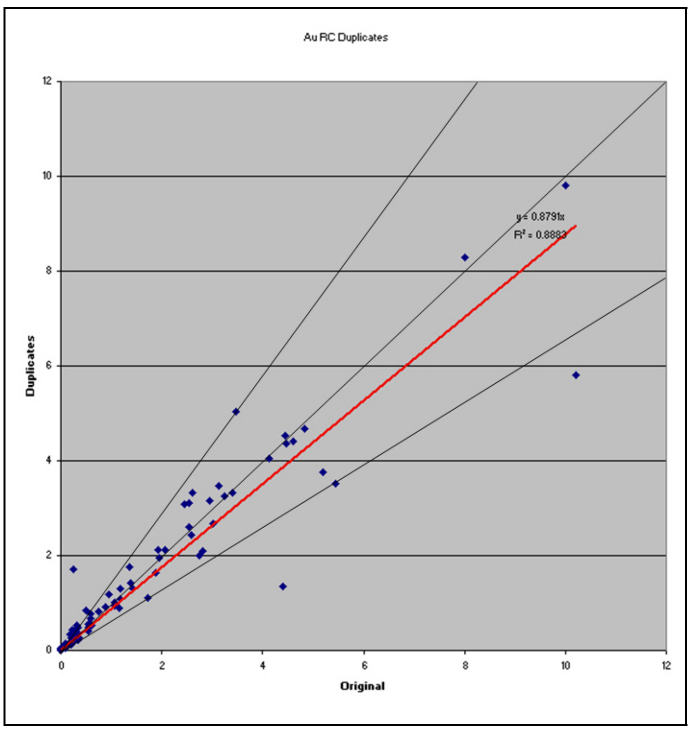
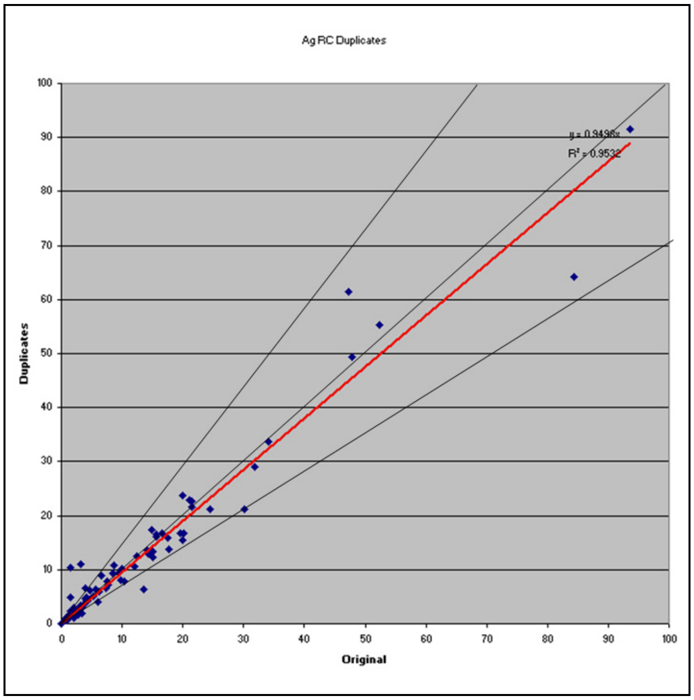


Figure 14.4: Ag RC Duplicates



In addition to the control sampling program completed to assess the relative accuracy of the OMAC assay facility in Ireland a suite of 38 pulps from the Rudne Bane underground crosscut sampling program was collected. Figures 14.5 and 14.6 illustrate the correlation between the historic Rudne Bane underground channel sample assay data for Au and Ag, respectively. Overall, the Chemex check sampling results in an average grade of 2.41 g/t for gold and 12.01 g/t for silver. This equates to a 0.13 g/t differential or 5% higher grade gold than the historic Rudne Bane sampling while the silver values show a 1.47 g/t or 12% lower value for the Chemex results versus the historic Rudne Bane data. Considering the nature of the deposit, variability and the relatively high nugget effect evident within the dataset, these results are within reasonable and acceptable limits.

Figure 14.5: Au Check Assay Results; Rudne Bane vs. Chemex

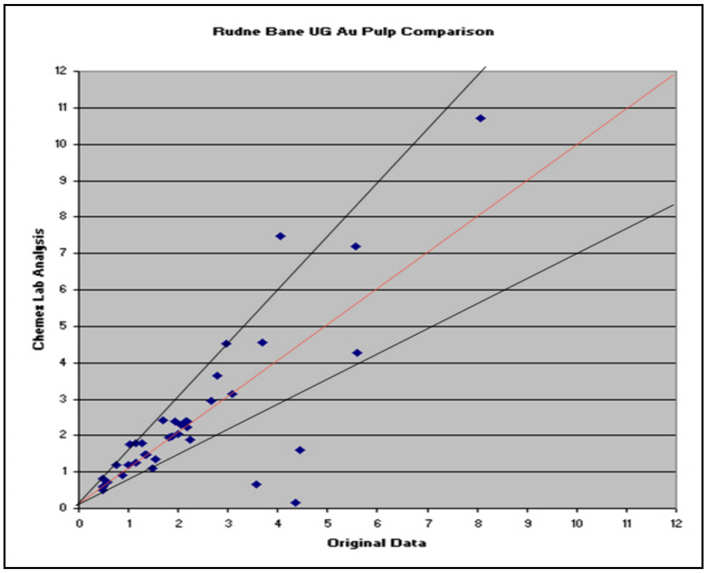
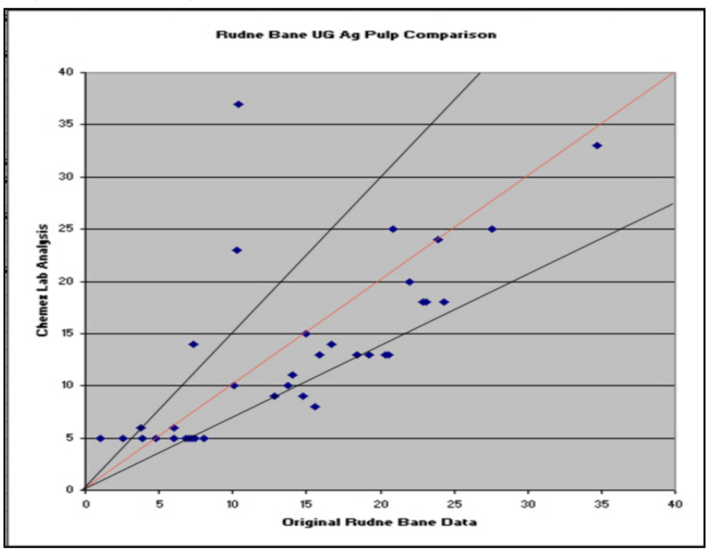


Figure 14.6: Ag Check Assay Results; Rudne Bane vs. Chemex



The results of the ALS Chemex assays confirm the results of the OMAC laboratory in Ireland. The results from ALS Chemex were approximately 5% higher than the Rudne Bane database. This is a further endorsement of the validity of the historic data and is consistent with previous assessments of the comparison between Rudne Bane original assays and verification assays completed by western laboratories.

According to Smith and Kirkham, 2004, as part of a seven-day site visit in October and November 2003 by Mr. Smith, Mr. Smith collected 23 check samples detailed in Table 14.1. The samples include rock chips taken from the major mineralized zones within the project and re-samples of drill core and underground channel sample pulps. The samples are not useful as systematic verification of previous work, but only as an independent control to ensure that the orders of magnitude of mineralization stated in various reports exists. With consideration for the small size of the check sample program collected by the author of this report, the samples show an acceptable correlation with sampling by prior workers.

Table 14.1: Kremnica Project Control Sampling

Bruce Smith Control Samples, November 03						Control No.	Control Assay	
Rock chip field sampling						(OMAC Ireland analysis)		
Location					Sample type		Au g/t	Ag g/t
Main Schramen vein, sampled underground adit P2					5m chip.	BS001	9.32	23.7
Main Schramen vein, sampled underground adit P4					5m chip.	BS002	3.64	101.0
Volle Henne, vein material.					Waste from old adit.	BS003	0.30	1.9
Main Šturec pit.					Waste vein material.	BS004	3.36	29.7
Main Šturec pit, outcropping vein at south end of pit.					5m channel.	BS005	1.63	24.0
West Wolf, east-west crosscutting vein, 1m wide.					1m chip.	BS006	14.84	80.8
Wolf, main NS Kirchberger vein.					Grab sample.	BS007	0.07	2.4
Wolf upper west, brecciated rhyolite dyke.					Grab sample.	BS008	2.79	16.7
Vratislav, Schindler vein outcrop.					4m chip.	BS009	1.46	9.3
Lucky exp lic, south ridge silicified veins and selvage.					Float.	BS010	0.03	0.3
Lucky exp lic, south ridge silicified rhyoite and veins.					3m chip.	BS011	0.05	0.5
Re-sampling of Stored Pulps.								
Underground cross-cut check samples.				(Bondar Clegg analysis)				
XCUT_ID	Au g/t	Ag g/t	LENGTH	FROM	TO			
P-1-79	2.94	28.1	0.94	78.72	79.67	BS012	3.64	32.3
P-1-74	3.21	13.1	1.03	73.58	74.60	BS013	3.37	13.2
P-2-20	3.91	6.1	0.95	21.90	22.85	BS014	4.13	6.1
P-2-76	2.22	11.4	1.03	78.07	79.10	BS015	2.24	11.4
P3-3	0.04	0.13				BS017	0.10	4.0
P3-34	2.48	16.88				BS016	2.68	13.5
Kremnica project drilling 1996 assays.				(SGS France analysis)				
HOLE	FROM	TO	SAMPLE	AU_1	AG_1			
AS-5	126.4	127.4	133742	0.740	27.00	BS019	0.60	6.2
AS-4.5.1.B	17.0	18.5	138357	0.960	10.40	BS020	0.99	10.1
AS-4.5.1.B	14.0	15.5	138355	1.570	14.30	BS021	1.51	15.1
Kremnica project drilling 1997 assays.				(Geological Survey analysis)				
Hole	From	To	SAMPLE	Au g/t	Ag g/t			
AS144	11.00	12.00	71440120	1.600	18.7	BS022	1.50	16.5
AS144	12.00	12.70	71440127	0.490	11.5	BS023	0.26	11.5
S144	13.70	14.20	71440142	3.200	18.9	BS024	3.00	19.0

As a part of the 2005 re-sampling program, 170 intervals were selected from various Argosy drillholes and sent for check assays to OMAC in Ireland. In this study the average grade for the check sampling data was 2.94 g/t for Au and 20.74 g/t for Ag, while the original data had a overall average assay's of 3.27 g/t for Au and 20.83 g/t for Ag, respectively. This equates to an approximate 10% positive bias toward the original Au data and a 6% positive bias for the original duplicates. The results of this study are shown in Figures 14.7 and 14.8. It is interesting to note that while there is a positive bias toward the original Au assay data, it appears that this is primarily within the lower grades and there appears to be a bias toward the check assay data within the higher grade population.

However, differentials within this range are not uncommon within gold deposits especially those that exhibit relatively high nugget effect.

The silver illustrates a very small positive bias (i.e. less than 1%) toward the original data versus the check assay data and offer an excellent correlation between the two datasets. This adds additional support for the validity of the original data and lends credence for its use within the resource estimation process.

Figure 14.7: Au Check Assay Results; Argosy Drillhole Data

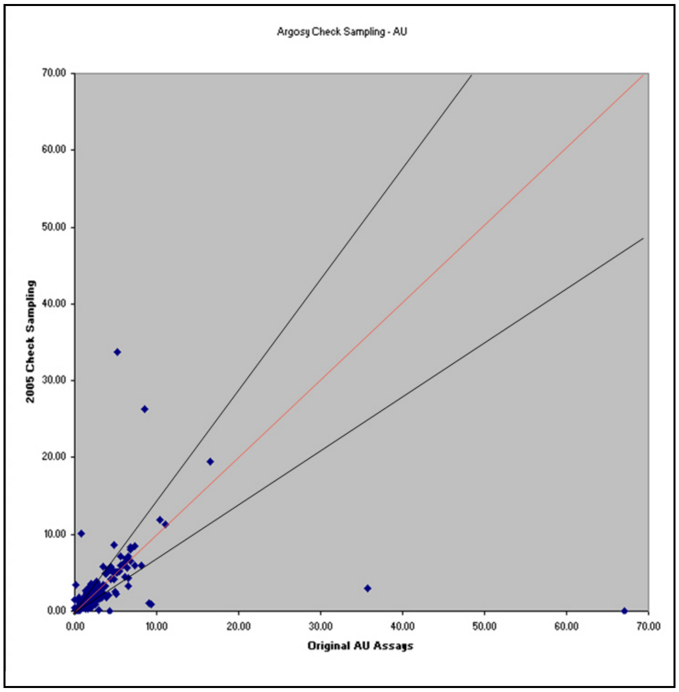
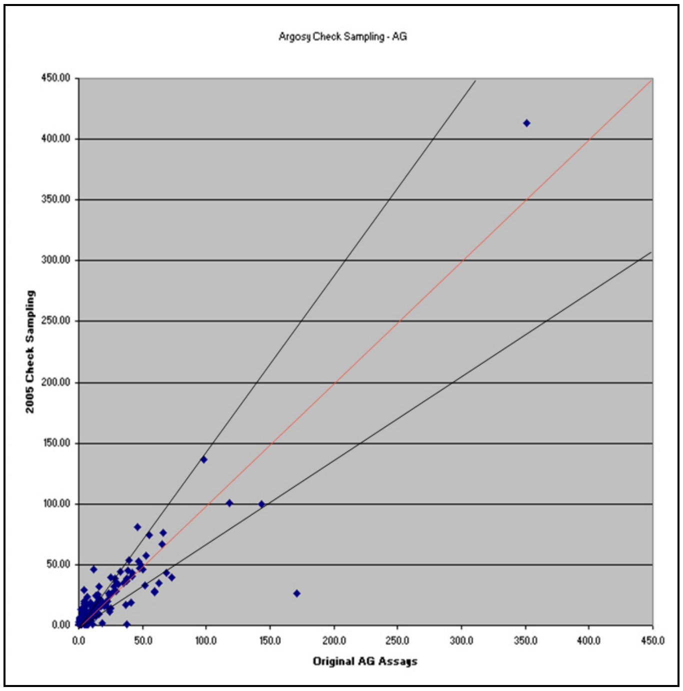


Figure 14.8: Ag Check Assay Results; Argosy Drillhole Data



A total of 98 pulp blanks, 57 standard reference material, 97 pulp duplicates and 67 check assays were included with 3,153 sampled intervals. This equates to an overall insertion rate of over 10%. All pulp blanks for gold and silver resulted in values less than five times the lower detection limit except for one value for silver which returned a value of 16.88 ppm Ag. This is likely a result of sample mislabeling which should be investigated. The assay results of the standards are within acceptable limits with the exception of one result (i.e. 56 of 57 pass rate).

As an additional check to insure the validity of the Argosy drillhole data, Tournigan commissioned 5 twin holes for a total of 235 meters within the 2005 RC drilling program. Although it is difficult to compare the results graphically, as the intervals do not match exactly and the sample intervals lengths differ between the RC holes (i.e. exactly 1 meter intervals) and the Argosy holes (i.e. 0.4 – 8 meters), it is possible to compare similar ranges within the drillholes. The RC twins were named KGST-37R through KGST-41R and the holes that they were designed to twin are as follows:

1) KGST-37R AS-5.1.1.A
2) KGST-38R AS-4.1.1
3) KGST-39R AS-4.1.B
4) KGST-40R AS-4.5.1.B
5) KGST-41R AS-4.D

Due to the fact that the assay intervals differ, a comparison of the weighted average over similar intervals was done. It is the opinion of the author that it is virtually impossible to exactly match the location, sample intervals and assay values for any twin drilling and analysis exercise. Therefore, any results from drilling twin holes should indicate values within a range. This is especially true in the case of a gold and silver deposit that displays a relatively high nugget effect.

For KGST-37R, the RC (Au of 3.47 g/t and Ag of 38.24 g/t) versus the Argosy drillhole (average for Au of 3.38 g/t and Ag of 33.83 g/t) resulted in a positive bias toward the RC results of 0.09 g/t (2.6%) and 4.41 g/t (11.4%) for Au and Ag, respectively.

For KGST-38R, the RC (average for Au of 2.06 g/t and Ag of 14.44 g/t) versus the Argosy drillhole (Au of 1.47 g/t and Ag of 9.89 g/t) resulted in a positive bias toward the RC results of 0.59 g/t (29%) and 4.55 g/t (31%) for Au and Ag, respectively.

For KGST-39R, the RC (average for Au of 0.89 g/t and Ag of 12.73 g/t) versus the Argosy drillhole (Au of 0.91 g/t and Ag of 12.39 g/t) resulted in a positive bias toward the Argosy results of 0.02 g/t (2.0%) for Au and a positive bias toward the RC results of 0.34g/t (2.7%) for Ag.

For KGST-40R, the RC (average for Au of 1.4 g/t and Ag of 9.4 g/t) versus the Argosy drillhole (Au of 1.03 g/t and Ag of 8.61 g/t) resulted in a positive bias toward the RC results of 0.37 g/t (26%) and 0.79 g/t (8.4%) for Au and Ag, respectively.

For KGST-41R, the RC (average for Au of 2.61 g/t and Ag of 19.62 g/t) versus the Argosy drillhole (Au of 1.56 g/t and Ag of 14.39 g/t) resulted in a positive bias toward the RC results of 0.95 g/t (37%) and 5.23 g/t (27%) for Au and Ag, respectively.

For all cases, with the exception of one, the new RC data has a relative higher Au and Ag values than its twinned Argosy counterpart. Smith and Kirkham, 2004 pointed out the likelihood that the previous sampling may have resulted in an underreporting of the grades. This current data also indicates that the Argosy grades may have been underreported. One of the reasons may be, as site staff have theorized, is a result of the drilling method. Namely, that diamond drilling and the subsequent sawing of the core, may be washing away limonite and thereby gold and silver bearing fluids. It was for this reason that reverse circulation was chosen for the 2005 drilling campaign. It is clear that we can consider the Argosy data to be valid and that it may be considered a conservative estimate of Au and Ag content.

SECTION 15.0 - ADJACENT PROPERTIES

There are no properties adjacent to the Kremnica property. Tournigan is actively exploring the region for precious metals, and has the only known mining license in the area.

The closest prospective areas to the Kremnica property are centered on the historic mining district Banska Stiavnica, approximately 40 km to south. At the time of writing, small-scale underground gold-silver production is ongoing at the Hodrusa mine in the Banská Bystrica district.

SECTION 16.0 – MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 HISTORIC OPERATIONS

Ore processing has been documented at Kremnica for over 700 years, and records indicate that gold recovery has been relatively straightforward using gravity, flotation, and cyanidation processing.

Throughout most of the district's history, operations involved gravity separation and amalgamation, with approximately half of the gold being recovered. A flotation plant was constructed in 1935 that utilized amalgamation to recover gold from the flotation concentrates. Recoveries are thought to have increased to approximately 70%.

In the mid 1980s, the 1935 plant was upgraded to treat ore by direct cyanidation. Oliver Finca, the mine engineer, reports that subsequent operations from 1987 to 1992 achieved gold recoveries reaching almost 80%. Typical reagent consumptions in kilograms per tonne were reported to be 0.88 for cyanide, 0.11 for zinc, 1.96 for chlorine, and 7.1 for lime.

16.2 ARGOSY SPONSORED TESTWORK

Argosy utilized Hazen Research, Inc. (Hazen) of Golden, Colorado, U.S.A to conduct metallurgical work on five mineralized samples from the Šturec deposit. All the samples were tested using both flotation and direct cyanidation. Table 16.1 provides Argosy's summary of the metallurgical results as they apply to the various rock units in the deposit model. It should be noted that none of the processes were optimized.

Whole ore cyanidation in bottle roll tests yielded gold recoveries ranging from 84.1 to 94.4%, with cyanide consumption ranging from 1.3 to 2.2 kg/t at minus 170 to 270 mesh grinds. Oxide and sulphide mineralization provided approximately the same results.

Flotation on minus 200 mesh sulphide mineralized material yielded gold recoveries generally in the 90% to 94% range. Similar tests on oxide mineralization gave results in the 65% to 82% range. Flotation concentrates from quartz vein material total only 1.41% to 6.84% of the weight by feed, while concentrates from andesite mineralization total 8.11% to 29.75% of the feed. Concentrate grades from quartz vein ore range from 26.8 to 156 g/t Au, while concentrate from andesite ore range from only 7.13 to 48.2 g/t Au.

Three cyanide leach tests were performed on flotation concentrates. Gold recoveries for the two tests on quartz vein mineralized concentrates were 96.4% and 97.7%. Gold recovery for the single test on andesite concentrate was 91.0%. Cyanide consumption in the three tests ranged from 0.49 to 0.66 kg/t of original ore.

R. MacPherson, a joint venture of Hazen and Lakefield Research, conducted Bond Work Index and abrasion tests on three Argosy samples in early 1997. The three rock types had roll mill indices ranging from 16.0 to 17.2 kW/t and abrasion indices of 0.3368 to 1.2282 lbs/kWh.

Table 16.1: Densities and Metallurgical Recoveries

Rock Type	Metallurgical Parameter	Oxide	Mixed	Sulphide
Quartz Vein	Density	2.46	2.48	(2.48)
	Whole Ore Cyanidation Recovery %	89.2 Au 73.1 Ag	(91.0) Au (73.1) Ag	91.0 Au 49.6 Ag
	Flotation Recovery %	65.5 Au 63.4 Ag	(92.3) Au (91.7) Ag	92.3 Au 91.7 Ag
	Flotation / Cyanidation Recovery %	64.0 Au 47.2 Ag	(84.0) Au (30.4) Ag	84.0 Au 30.4 Ag
Hydrothermal Breccia	Density	2.46	2.55	2.55
	Whole Ore Cyanidation Recovery %	(89.2) Au (73.1) Ag	(91.0) Au (49.6) Ag	(91.0) Au (49.6) Ag
	Flotation Recovery %	(65.5) Au (63.4) Ag	(92.3) Au (91.7) Ag	(92.3) Au (91.7) Ag
	Flotation / Cyanidation Recovery %	(64.0) Au (47.2) Ag	(84.0) Au (30.4) Ag	(84.0) Au (30.4) Ag
Silicified Stockwork Andesite	Density	2.35	2.48	2.43
	Whole Ore Cyanidation Recovery %	95.8 Au 51.6 Ag	84.1 Au 57.7 Ag	94.4 Au 45.9 Ag
	Flotation Recovery %	82.2 Au 50.7 Ag	89.6 Au 69.6 Ag	90.7 Au 74.7 Ag
	Flotation / Cyanidation Recovery %	(79.7) Au (49.2) Ag	(86.4) Au (67.1) Ag	(92.6) Au (46.3) Ag
Rubble	Density	1.84	1.84	1.84
	Whole Ore Cyanidation Recovery %	89.2 Au 73.1 Ag	(87.6) Au (53.7) Ag	(92.7) Au (47.8) Ag
	Flotation Recovery %	65.5 Au 63.4 Ag	(90.6) Au (80.7) Ag	(91.5) Au (83.2) Ag
	Flotation / Cyanidation Recovery %	64.0 Au 47.2 Ag	(85.2) Au (49.8) Ag	(88.3) Au (38.4) Ag
Void	Density	0	0	0

Note: The number in brackets represents Argosy estimates based on results of other tests.

16.3 2005-2006 METALLURGICAL TESTWORK

16.3.1 Overview

A program of metallurgical testwork was carried out by Process Research Associates (PRA) in Vancouver, BC, Canada between 2005 and 2006 on a range of ore samples. This work established that Kremnica ore presented no particular treatment problems. A number of flowsheet alternatives were examined including gravity concentration, flotation, cyanide leaching, and leaching with alternative lixiviants. The selected flowsheet of gravity concentration and cyanide leaching gave higher recoveries than the alternative processes, and had no serious drawbacks. This flowsheet is the one most commonly used to treat gold ores throughout the world.

The selected flowsheet translated into a plant design that is almost standard to the industry: primary crushing followed by stockpiling and then a SAG/ball mill grinding circuit. Gravity concentration is incorporated into the grinding circuit. The cyclone overflow is fed to a carbon-in-leach (CIL) circuit with the recovered carbon sent to a standard pressure Zadra elution and recovery circuit using electrowinning. CIL tailings are thickened to recover cyanide and minimize the amount of cyanide destruction that is required. The tailings thickener underflow is treated for cyanide destruction using the SO_2/air (Inco) method, and the treated tailings are pumped to the tailings pond. Water is recovered from the tailings pond for reuse in the ore process plant.

16.3.2 Metallurgical Testing

Metallurgical testing was carried out by PRA in Vancouver during 2005-2006. A series of 45 samples were received by PRA and made into 10 composites based on ore type and location. A single master composite was made from the 10 primary composites.

The PRA report summarizes the results on all 11 composites and is reproduced below as Table 16.2.

Table 16.2: Master Composites

Sample	Average Head, g/t		Overall Gold Recovery, %*				Overall Silver Recovery, %*			
ID	Au	Ag	GSB	CN	GSB+Flot.	GSB+CN	GSB	CN	GSB+Flot.	GSB+CN
Comp.1	0.51	5.6	70.6	76.1	77.0	88.3	30.2	45.0	73.0	44.5
Comp.2	2.60	7.8	83.5	89.8	93.9	96.1	35.6	60.3	50.4	68.8
Comp.3	2.48	15.3	56.1	89.6	82.7	92.4	20.9	44.5	83.9	52.4
Comp.4	2.49	16.8	65.4	90.2	90.0	94.6	26.3	48.7	80.3	55.7
Comp.5	2.00	18.4	37.9	92.9	72.1	95.4	12.8	64.0	57.8	74.0
Comp.6	1.33	10.0	46.4	94.1	78.0	94.9	16.2	62.7	49.5	69.6
Comp.7	1.86	14.2	66.8	90.9	85.0	94.4	18.3	57.4	45.1	62.1
Comp.8	1.83	15.8	65.3	92.8	73.8	95.2	27.0	58.3	49.8	62.3
Comp.9	2.02	14.8	37.3	82.4	64.1	85.2	16.7	48.2	55.7	55.7
Comp.10	2.04	14.0	48.2	87.9	75.9	93.5	26.1	56.0	63.6	66.0
Master	1.68	12.8	58.5	89.5	76.2	92.4	21.6	60.1	56.5	66.4

*recoveries denoted by GSB = gravity, CN = cyanide leach; Flot = flotation

The various composites ranged from 0.5 to 2.6 g/t Au and 5.6 to 18.4 g/t Ag. Not surprisingly, the master composite gave results close to the average of the other tests. Gravity gold recovery results were very promising, with recoveries ranging from 37% to 70.6%, with the highest percent recovery coming from the lowest grade sample (composite 1).

On all composites, the highest recovery was achieved using gravity followed by leaching of the gravity tailings. This process averaged 93% for the 10 composites (92.4% for the master composite) with a range of 85.2% to 95.4%. This is 3% higher than the whole ore leach circuit and over 15% higher than gravity with flotation (Note: the flotation concentrate from the master composite was leached with cyanide to give 95.8% recovery. This is equivalent to a further overall loss in recovery of about 1.5% when compared to the gravity-cyanide flowsheet). Silver recovery from the gravity-cyanidation flowsheet was about 66%.

A single thiourea leach was carried out on the master composite, resulting in recoveries of 73.8% gold and 60.4% silver. Compared to the results from the cyanide leaching process, this was more than 15% less for gold, but almost the same for silver. Kinetics in a thiourea leach are faster than in a cyanide leach, but this does not compensate for the much lower gold recovery.

The standard Bond ball mill work index was determined on the master composite. It returned a value of 15.6 (metric) indicating a medium hard ore.

Future testwork is proposed to examine a range of samples covering the more significant ore types and zones that will be mined during the initial production years. The proposed testwork will include more comminution evaluation including not only the Bond tests, but also tests that check ore competency for autogenous milling. The testwork is described in Section 20.1, "Recommended Work Program."

16.4 PLANT DESIGN

16.4.1 Introduction

The gravity-cyanide leach process that produced the best results during metallurgical testing was translated into a process flowsheet and then into a preliminary plant design. The flowsheet developed is very conventional. Figures 16.1 to 16.3 show the General Arrangement drawings and flowsheets.

16.4.2 Plant

The 6,000 tpd plant (2,100,000 tpa) will receive ore at a dump pocket (150 tonne live capacity) and crush it through a jaw crusher (1,000 x 1,500 mm) to a nominal 150 mm. The ore is fed to the jaw by a 1.2 m apron feeder. Crushed rock is collected on a 42" collection conveyor and then fed to the 36" stockpile stacking conveyor, which places it on a 6,000 tonne live stockpile.

Ore is reclaimed from the stockpile by one or two of four belt feeders and fed to the 36" SAG mill feed conveyor. This conveyor is fitted with a weightometer.

Figure 16.1: 6,000 tpd Process General Arrangement

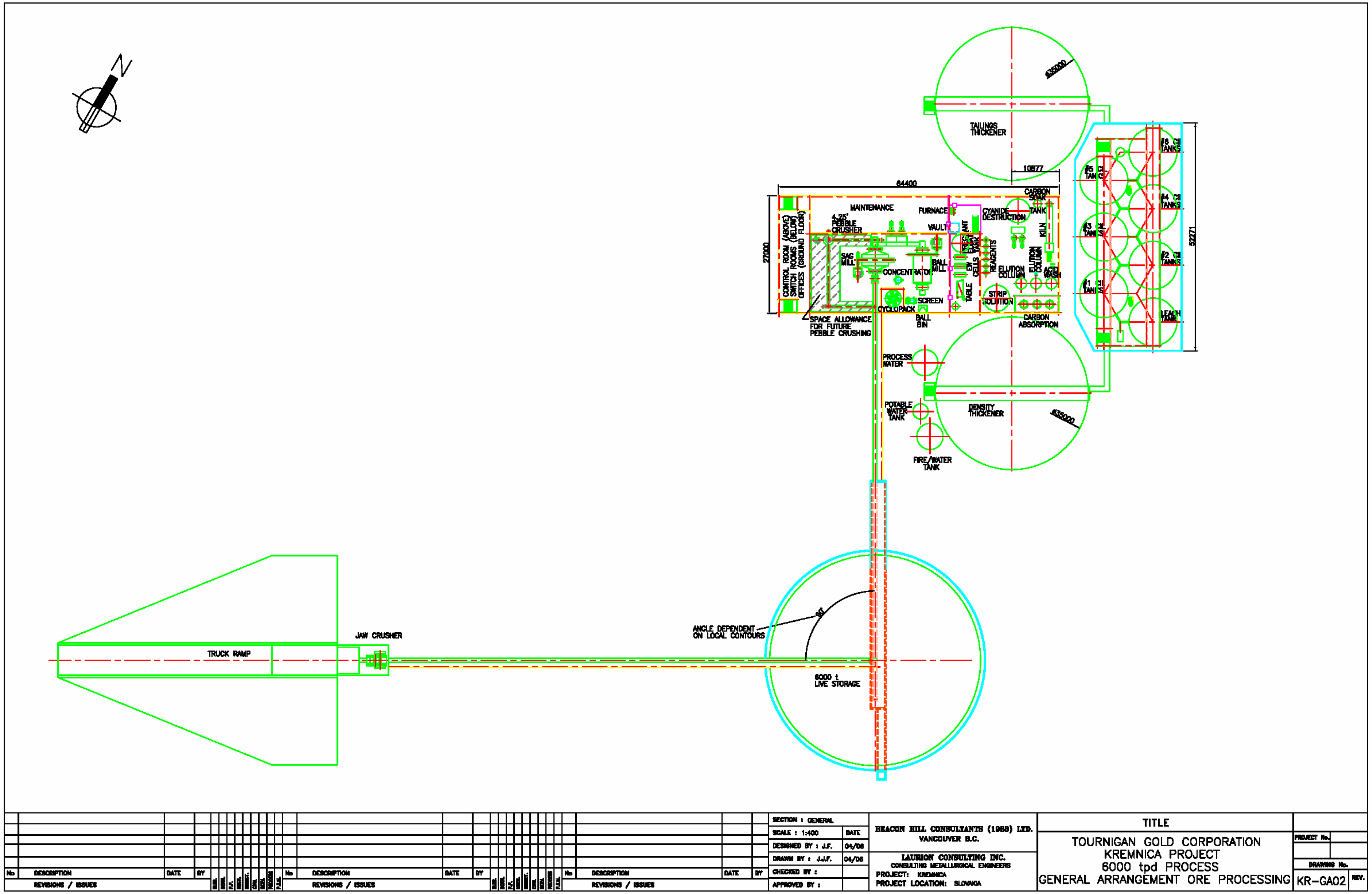


Figure 16.2: 6,000 tpd Process General Arrangement – Refinery

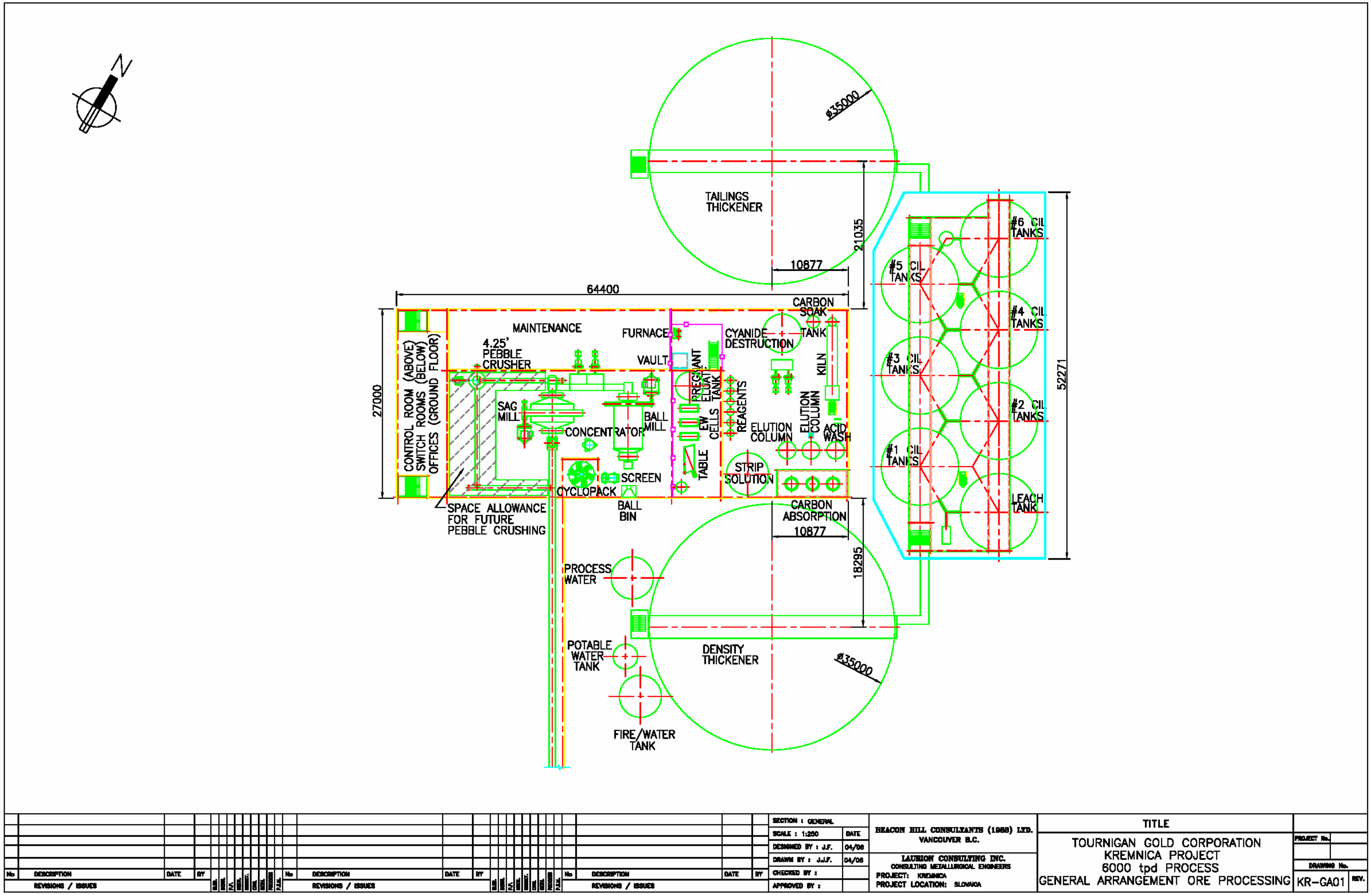
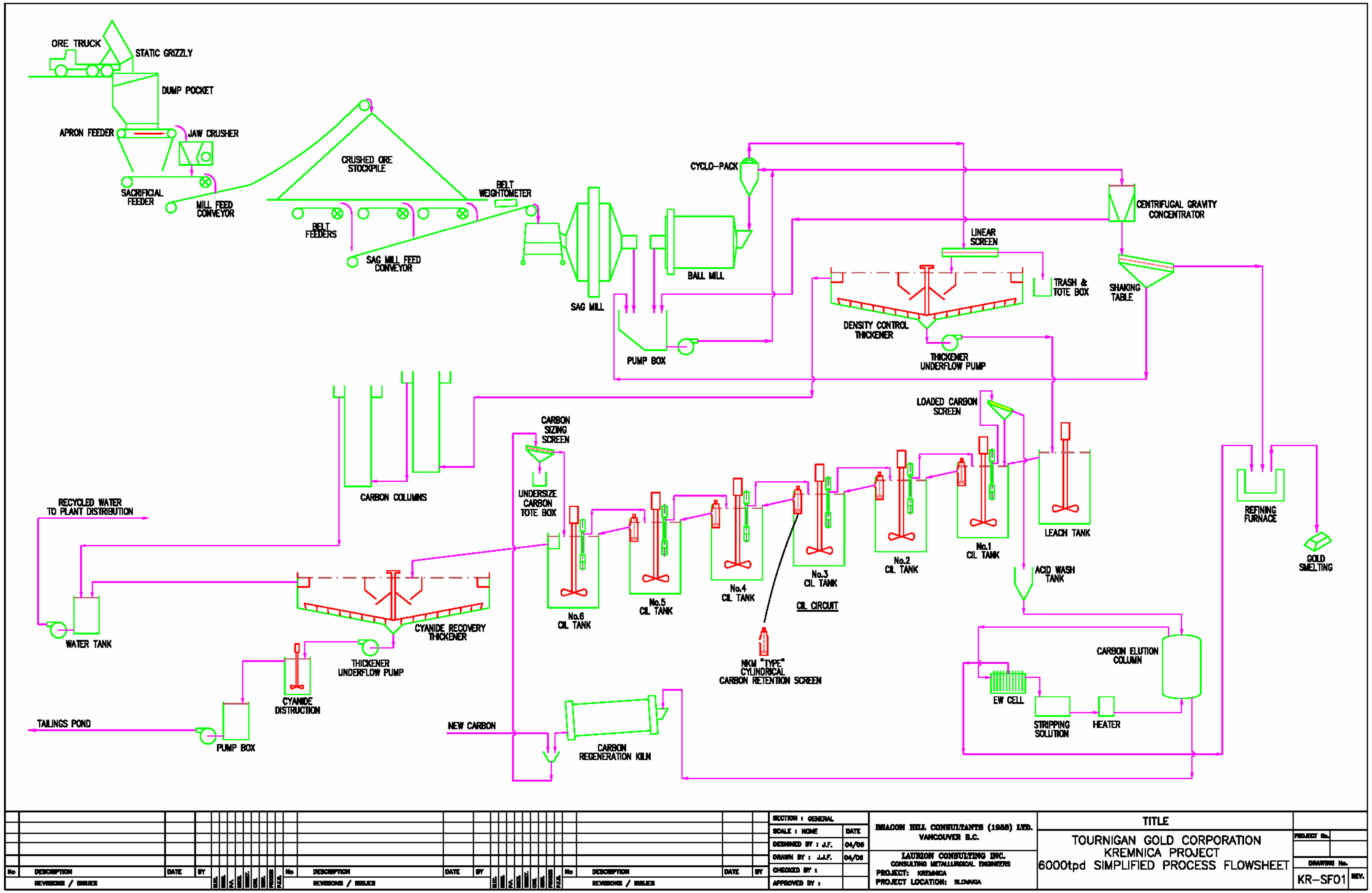


Figure 16.3: 6,000 tpd Simplified Process Flowsheet



The reclaimed ore is fed to a 7.3 m x 3.8 m EGL SAG mill operated with a 4.8 m x 7.3 m Ball mill. Both mills are fitted with 2500 kW synchronous motors to simplify the provision of spares, and both discharge to a common pump box that pumps to a cyclopack fitted with six cyclones. A bleed from the cyclone feed system is sent to a centrifugal gravity concentrator. Cyclone overflow is fed to the leach circuit. Provision will be made for future pebble crushing (SAG mill discharge), but it will not be installed in the initial plant.

The comminution circuit designed for Kremnica is typical of those used in North American process plants (as well as Australian, South African, and in many parts of South America) on an ore of this type at this capacity. Prior to the widespread use of SAG grinding, fine crushing with ball milling (or rod-ball milling) was used for smaller capacities. Beyond 6,000 tpd, however, multiple units and/or parallel lines are required, making it more cost efficient to install a SAG/ball circuit instead. There are no technical reasons why a conventional (fine crushing/grinding) circuit could not be used if suitable equipment were more readily available. A review of alternative comminution options should take place during the feasibility study.

The centrifugal gravity concentrator produces a gravity concentrate that is re-concentrated over a shaking table. This is an important part of ore processing since approximately 50% of the gold is potentially recoverable by gravity. The table concentrate is then smelted directly to doré, while the table tailings and centrifugal concentrator tailings are returned to the grinding circuit.

Cyclone overflow is sampled and fed via a linear trash screen to the 25 m CIL feed thickener used for density control. Thickener underflow is diluted to the desired density using water reclaimed from the tailings thickener and elsewhere and fed to the CIL circuit consisting of a single aerated leach tank and six aerated 14.5 m x 15 m CIL tanks. The CIL tanks are fitted with inter-tank screens (NKM-type) and recessed impellor pumps for upstream carbon movement. Thickener overflow is fed through carbon columns before being returned to the grinding circuit.

The CIL tailings pass through a safety screen that captures tramp carbon, before being sent to the 25 m tailings thickener where cyanide-containing solutions are recovered and recycled. The thickener underflow is treated to destroy residual cyanide using the copper catalyzed SO_2/air process (sometimes called the Inco process). Alternative cyanide destruction systems based on hydrogen peroxide (or its derivative, Caros acid) and chlorine have also been used. The advantages of these alternatives will be compared in the feasibility study so that the right system is selected. The availability of certain reagents in Slovakia may be a factor in selecting one alternative over another.

Loaded carbon harvested from CIL tank #1 is washed over a screen and then transferred to the batching/acid-wash tank. The tank is elutriated to remove trash and slimes and then drained, and the carbon is acid-washed with hydrochloric acid to remove lime and gypsum scale. After neutralization, the carbon is transferred to an elution column with a 4 tonne carbon capacity. Gold and silver are stripped from the carbon using the pressure Zadra process (dilute hot caustic (2%) cyanide (0.5%) solution) at 125°C. The resulting solutions are fed to electrowinning cells where the gold and silver are recovered. The solutions are recycled to the elution column after reheating, and the gold and silver harvested from the electrowinning cell are smelted to doré.

The entire process flowsheet and plant design selected for Kremnica is very conventional and similar to those used at many successful mines throughout the world.

SECTION 17.0 - MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

17.1 INTRODUCTION

The following sections detail the methods, process and strategies employed in creating the revised resource estimate for the Kremnica Gold and Silver Project. Table 17.1 lists some conventions and abbreviations that are encountered throughout the resource estimation section of this report.

Table 17.1: Report Conventions and Abbreviations

Abbreviation	Description
Au	Gold
Ag	Silver
g/t or gpt	Grams per Tonne (Gold or Silver Grade)
Kg	kilogram
Cm	centimeter
M	meters or metres
QA/QC	Quality Assurance / Quality Control
X, Y, Z	Cartesian Coordinates, also “Easting”, “Northing”, and “Elevation”
DDH	Diamond Drill Holes.
N, S, E, W	Cardinal points, North, South, East, and West, respectively, and combinations thereof.
SG	Specific Gravity
RC	Reverse Circulation
CV	Coefficient of Variation.
Tournigan	Tournigan Gold Corporation
Kremnica	Kremnica Property or Town
John Cuthill	John Cuthill, Senior Geologist, Kremnica
NI 43-101	National Instrument 43-101.
TSX-V	Toronto Venture Stock Exchange.

17.2 ASSAY DATABASE

The database consists of all holes derived from the 1996 and 1997 Argosy drilling campaigns, the underground sampling program referred to as the Rudne Bane dataset, the 2005 RC (reverse circulation) drill data and the bench channel sample data collected during the 2005 program. The selection process for assays to be utilized for this study was consistent with the 2004 Preliminary Assessment performed by Beacon Hill (Smith and Kirkham, 2004) where the selection criteria for the Argosy and Rudne Bane assay data was dependent upon confidence and assay lab considerations. The RC data and the bench channel sample data were not selectively filtered apart from addressing recovery issues related to the RC data.

However, in the 2005 campaign, additional QA/QC was performed relating to recovery data. All recovery data available from the logs was manually input for this exercise and analyzed. It was deemed that any Argosy drill core data that had a recovery of 30% or lower was not to be used in the current resource calculation. The 30% threshold was chosen based on conversations with on-site staff and upon inspection of core boxes with material remaining that were logged with varying recoveries. It was deemed that over a one meter interval, which was the most common sample interval length, that a core recovery of 30% would result in approximately a 1000 gram split sample which is sufficient to derive a representative sample. This estimation is based on the following assumptions:

- Although the diamond drilling was done with HQ core going to NQ core further down the hole, it is assumed that the drilling was NQ for the complete hole, to be conservative. Therefore, this would equate to approximately a 60mm diameter or 30mm radius and a volume of 0.002826 m^3 for a 1 meter length of core.
- Specific gravity is assumed to be 2.5 tonne/ m^3, which again are conservative for the estimates.

In addition, the 2005 RC data was selected based upon recovery namely; RC data with a recovery of less than 10% was excluded from the resource calculation. This percentage threshold was based upon conversations with site geology staff and the drill site geologists along with an estimate of the remaining material by weight and the reasonableness from which a representative sample may be attained. It should be noted that during the 2005 RC drilling campaign, that a one meter sample was deemed to have 100% recovery if 30 kg of material was collected over that interval. This was based on the assumption that all material possessed a specific gravity equal to that of the andesite which was estimated at 2300 kg/m^3 and the diameter of the bit being approximately 13 cms. However, based on the 134 specific gravity measurements as listed in the Appendix, specific gravity for the andesite can range from a minimum of 1840 kg/m^3 to a maximum of 2710 kg/m^3. This equates to a minimum sample recovery weight of 24 kg to a maximum of 36 kg. In addition, the maximum recovery, that was within altered andesite, estimated for any one sample was 177% which equates to a 53 kg sample or a specific gravity of 4071 kg/m^3. There are also 352 RC samples with a recovery of greater than 100% which highlights the issues related to calculating recoveries in this manner.

The single biggest problem, related to the RC recoveries, is that there is not a determination of what the density of any given sample of chips was when it was solid rock; therefore, it is not possible to accurately estimate what the weight of 100% recovery would be. Based on observations during drilling, it is common when drilling a given location, particularly in the collapse zone, the bit would be interpreted as having established itself in a crack more or less parallel to the direction of movement of the drill. This would cause less material to be recovered due to losses in the crack even though essentially solid rock is being encountered. It is theorized that in such cases, that a representative sample is in fact being captured as an equal amount of material is being recovered as lost through fissures. The exception is in the case of voids which were recorded during logging and accounted for. It was deemed by the author that a recovery of less than 10% or less than 3 kg would not return a representative sample and that these samples are to be set to missing. This equates to 423 RC sample intervals with a recovery of between 0% and 10%.

Table 17.2 and 17.3 summarizes statistics for the complete Au and Ag assay (i.e. after selection criteria has been applied as described above) database (i.e. between 1,230,500 South and 1,229,450 South) used for the resource evaluation. The database has 13,367 Au and 13,231 Ag values with a minimum value of 0.00 and shows that the gold and silver distribution is relatively well behaved (in comparison with other precious metals deposits), still with a few samples representing an outlier population as shown in Figures 17.1 through 17.4. The average overall Au grade (weighted by sample length) is 0.84 g/t, with a standard deviation of 2.04, resulting in a fairly high coefficient of variation[3] (CV) of 2.42. Approximately 77% of the assay data is below 1.0 g/t, and 64% of the data is below 0.5 g/t. As for Ag grades, the average overall grade (weighted by sample length) is 7.21 g/t with a standard deviation of 13.28, resulting in a fairly high coefficient of variation (CV) of 1.84. Approximately 78% of the assay data is below 10 g/t, and 62% of the data is below 5 g/t.

There are a few high-grade values, with 13 Au values being greater than 20 g/t and the maximum being 73 g/t. Silver values also have outliers with 36 Ag values being greater than 100 g/t and the maximum being 501 g/t. Figures17.1 and 17.3 show the histogram and basic statistics of all Au and Ag assays weighted by assay interval, respectively. In addition, Figures 17.2 and 17.4 show the corresponding probability plot for Au and Ag, respectively. Figure 17.5 illustrates a plan view of the drill holes for spatial reference.

[3] The coefficient of variation is defined as CV=σ/m (standard deviation/mean), and represents a measure of variability that is unit-independent. This is a variability index that can be used to compare different and unrelated distributions.

Table 17.2: Statistics of all Au samples by Cut-off Grade.

Cutoff	UnWeighted					Weighted				
	Weight	%	AU	SD	CV	Weight	%	AU	SD	CV
0.0	13367	100.0%	0.87	2.04	2.34	14838.7	100.0%	0.84	2.04	2.42
0.5	5040	37.7%	2.09	2.95	1.41	5391	36.3%	2.08	3.00	1.44
1.0	3241	24.2%	2.85	3.45	1.21	3454.3	23.3%	2.85	3.52	1.23
1.5	2275	17.0%	3.54	3.92	1.11	2407.5	16.2%	3.55	4.01	1.13
2.0	1640	12.3%	4.24	4.41	1.04	1730	11.7%	4.27	4.54	1.06
2.5	1175	8.8%	5.04	4.99	0.99	1237.1	8.3%	5.09	5.14	1.01
3.0	862	6.4%	5.88	5.60	0.95	912.9	6.2%	5.93	5.75	0.97
3.5	657	4.9%	6.71	6.18	0.92	696.5	4.7%	6.77	6.36	0.94
4.0	519	3.9%	7.50	6.74	0.90	551.8	3.7%	7.57	6.92	0.91
4.5	407	3.0%	8.40	7.36	0.88	434.5	2.9%	8.47	7.56	0.89
5.0	340	2.5%	9.12	7.86	0.86	364.3	2.5%	9.19	8.06	0.88
5.5	275	2.1%	10.05	8.48	0.84	291.8	2.0%	10.18	8.73	0.86
6.0	222	1.7%	11.08	9.14	0.83	236.2	1.6%	11.23	9.40	0.84
6.5	191	1.4%	11.87	9.63	0.81	202.6	1.4%	12.06	9.91	0.82
7.0	160	1.2%	12.86	10.23	0.80	169.9	1.1%	13.08	10.52	0.80
7.5	143	1.1%	13.53	10.63	0.79	151.7	1.0%	13.79	10.92	0.79
8.0	128	1.0%	14.22	11.04	0.78	135.7	0.9%	14.51	11.34	0.78
8.5	107	0.8%	15.39	11.72	0.76	114.8	0.8%	15.65	11.98	0.77
9.0	93	0.7%	16.40	12.27	0.75	99.7	0.7%	16.71	12.53	0.75
9.5	90	0.7%	16.64	12.40	0.75	96.5	0.7%	16.95	12.66	0.75
10.0	85	0.6%	17.05	12.64	0.74	91.5	0.6%	17.35	12.89	0.74
10.5	68	0.5%	18.76	13.62	0.73	74.1	0.5%	19.03	13.81	0.73
11.0	54	0.4%	20.86	14.59	0.70	59.6	0.4%	21.07	14.71	0.70
11.5	52	0.4%	21.23	14.74	0.69	56.8	0.4%	21.55	14.90	0.69
12.0	47	0.4%	22.25	15.16	0.68	51.8	0.3%	22.51	15.27	0.68
12.5	41	0.3%	23.73	15.71	0.66	44.5	0.3%	24.20	15.86	0.66
13.0	40	0.3%	24.00	15.81	0.66	43.6	0.3%	24.44	15.94	0.65
13.5	38	0.3%	24.56	16.03	0.65	41.5	0.3%	25.00	16.14	0.65
14.0	35	0.3%	25.49	16.39	0.64	38.6	0.3%	25.85	16.43	0.64
14.5	28	0.2%	28.32	17.22	0.61	32.2	0.2%	28.17	17.08	0.61
15.0	25	0.2%	29.94	17.55	0.59	29.7	0.2%	29.30	17.33	0.59

Table 17.3: Statistics of all Ag samples by Cut-off Grade.

	UnWeighted					Weighted				
Cutoff	**Weight**	**%**	**AG**	**SD**	**CV**	**Weight**	**%**	**AG**	**SD**	**CV**
0	13231	100.0%	7.52	13.69	1.82	14699.2	100.0%	7.21	13.28	1.84
1	9810	74.1%	10.03	15.11	1.51	10648.4	72.4%	9.83	14.78	1.50
2	8128	61.4%	11.83	16.02	1.35	8754.9	59.6%	11.67	15.71	1.35
3	6965	52.6%	13.40	16.80	1.25	7460.6	50.8%	13.27	16.50	1.24
4	5998	45.3%	15.01	17.58	1.17	6420.2	43.7%	14.87	17.26	1.16
5	5273	39.9%	16.47	18.27	1.11	5643.1	38.4%	16.31	17.94	1.10
6	4688	35.4%	17.85	18.93	1.06	5015.3	34.1%	17.68	18.58	1.05
7	4180	31.6%	19.25	19.59	1.02	4457.9	30.3%	19.09	19.25	1.01
8	3724	28.1%	20.70	20.29	0.98	3967.1	27.0%	20.54	19.93	0.97
9	3382	25.6%	21.94	20.89	0.95	3591.6	24.4%	21.81	20.54	0.94
10	3067	23.2%	23.23	21.53	0.93	3253.1	22.1%	23.10	21.17	0.92
11	2752	20.8%	24.69	22.26	0.90	2914.2	19.8%	24.57	21.89	0.89
12	2483	18.8%	26.13	22.98	0.88	2622.2	17.8%	26.04	22.61	0.87
13	2263	17.1%	27.47	23.65	0.86	2391.6	16.3%	27.35	23.26	0.85
14	2047	15.5%	28.95	24.39	0.84	2156.2	14.7%	28.88	24.01	0.83
15	1865	14.1%	30.37	25.11	0.83	1960.6	13.3%	30.32	24.72	0.82
16	1703	12.9%	31.79	25.83	0.81	1788	12.2%	31.75	25.42	0.80
17	1567	11.8%	33.13	26.51	0.80	1644.7	11.2%	33.09	26.08	0.79
18	1463	11.1%	34.24	27.09	0.79	1537.9	10.5%	34.18	26.63	0.78
19	1342	10.1%	35.67	27.85	0.78	1407.7	9.6%	35.64	27.38	0.77
20	1233	9.3%	37.11	28.61	0.77	1288	8.8%	37.15	28.16	0.76
21	1120	8.5%	38.80	29.50	0.76	1169.4	8.0%	38.85	29.02	0.75
22	1018	7.7%	40.54	30.40	0.75	1064.9	7.2%	40.56	29.86	0.74
23	945	7.1%	41.94	31.12	0.74	988.1	6.7%	41.97	30.55	0.73
24	864	6.5%	43.68	32.00	0.73	905.4	6.2%	43.67	31.37	0.72
25	800	6.0%	45.23	32.77	0.72	837.7	5.7%	45.23	32.11	0.71
26	746	5.6%	46.66	33.48	0.72	783.3	5.3%	46.61	32.77	0.70
27	698	5.3%	48.06	34.17	0.71	733.9	5.0%	47.97	33.42	0.70
28	655	5.0%	49.41	34.85	0.71	688.2	4.7%	49.34	34.07	0.69
29	619	4.7%	50.64	35.47	0.70	651.2	4.4%	50.53	34.65	0.69
30	589	4.5%	51.73	36.02	0.70	621.9	4.2%	51.53	35.14	0.68

Figure 17.1: Histogram and Basic Statistics of all Au samples Weighted by Assay Interval Length.

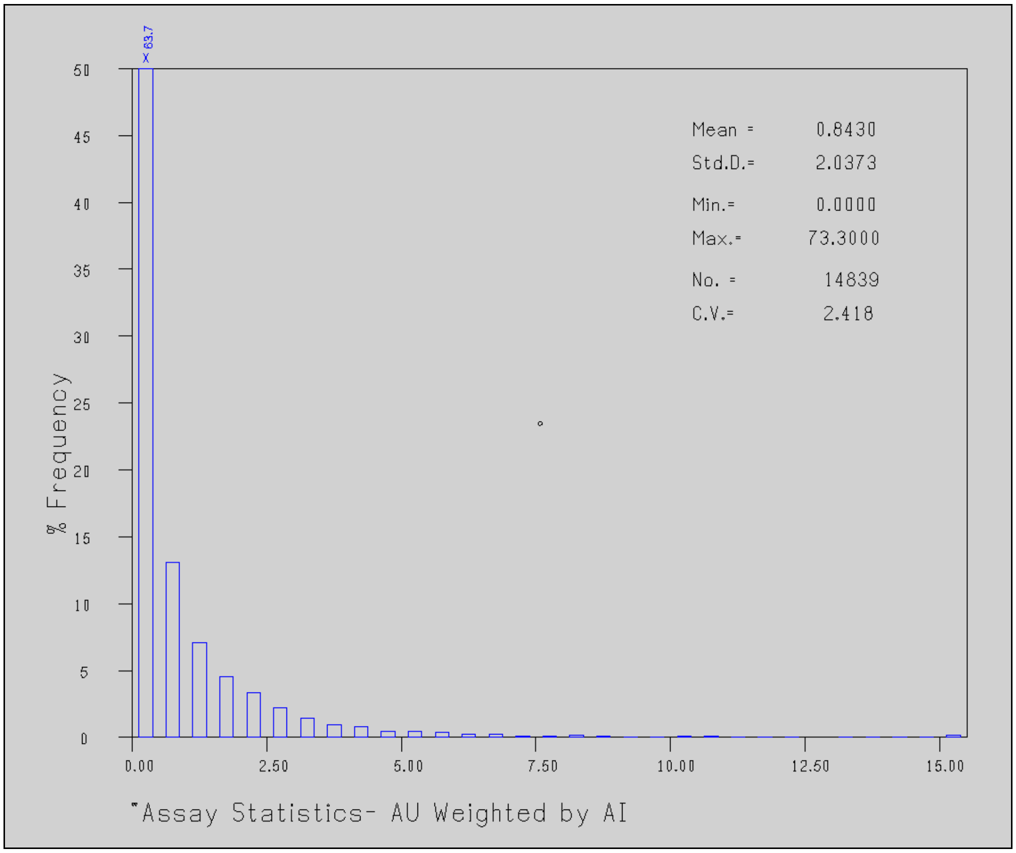


Figure 17.2: Probability Plot of all Au samples, same samples as Figure 17.1.

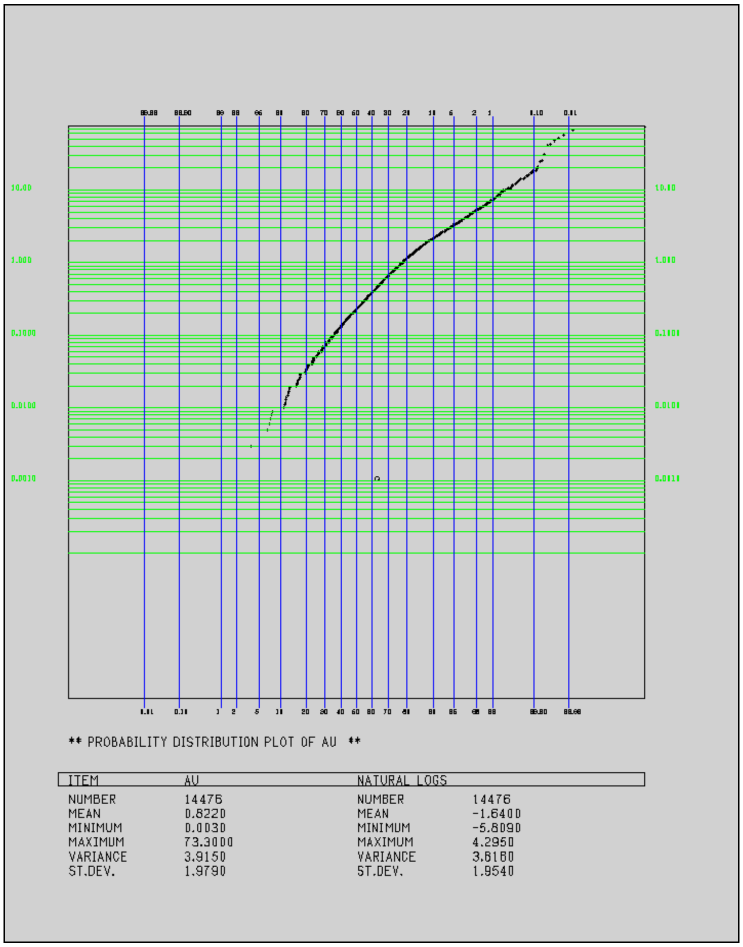


Figure 17.3: Histogram and Basic Statistics of all Ag samples Weighted by Assay Interval Length.

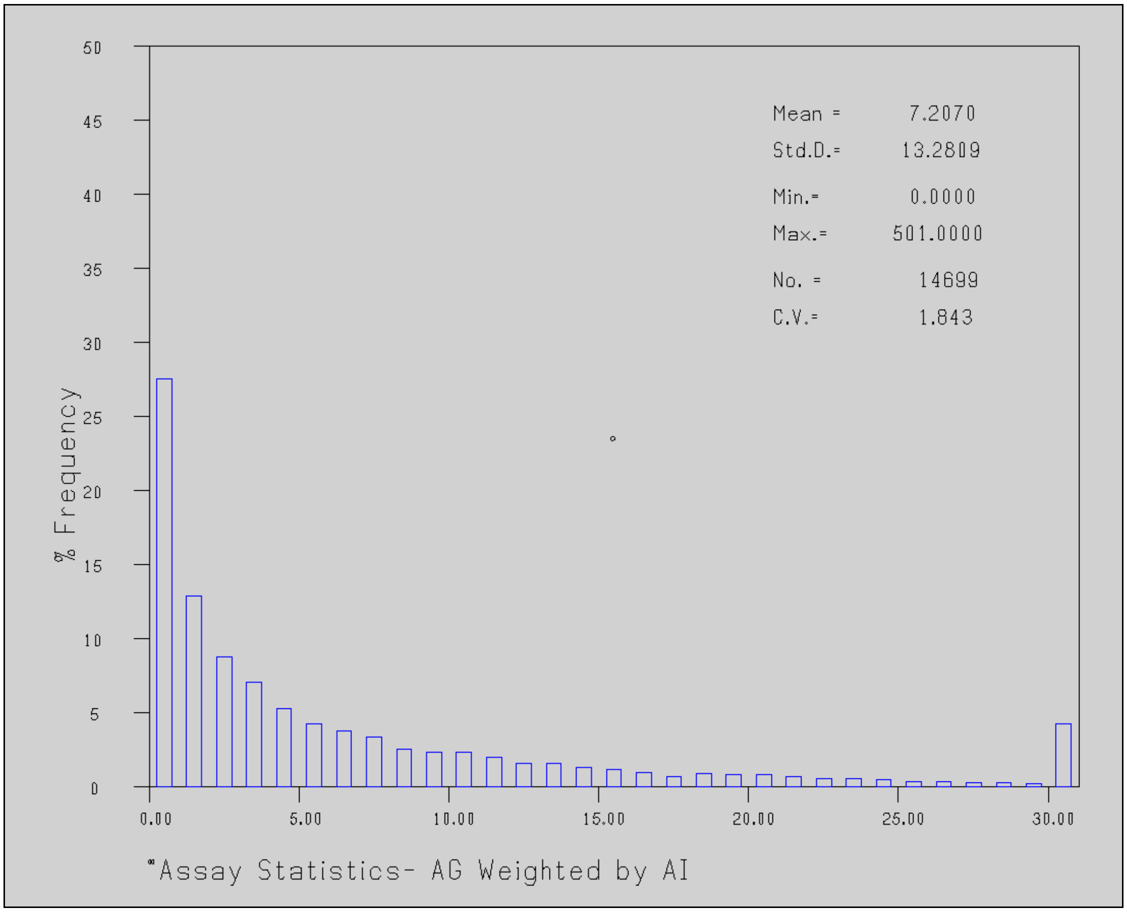


Figure 17.4: Probability Plot of all Ag samples, same samples as Figure 16.

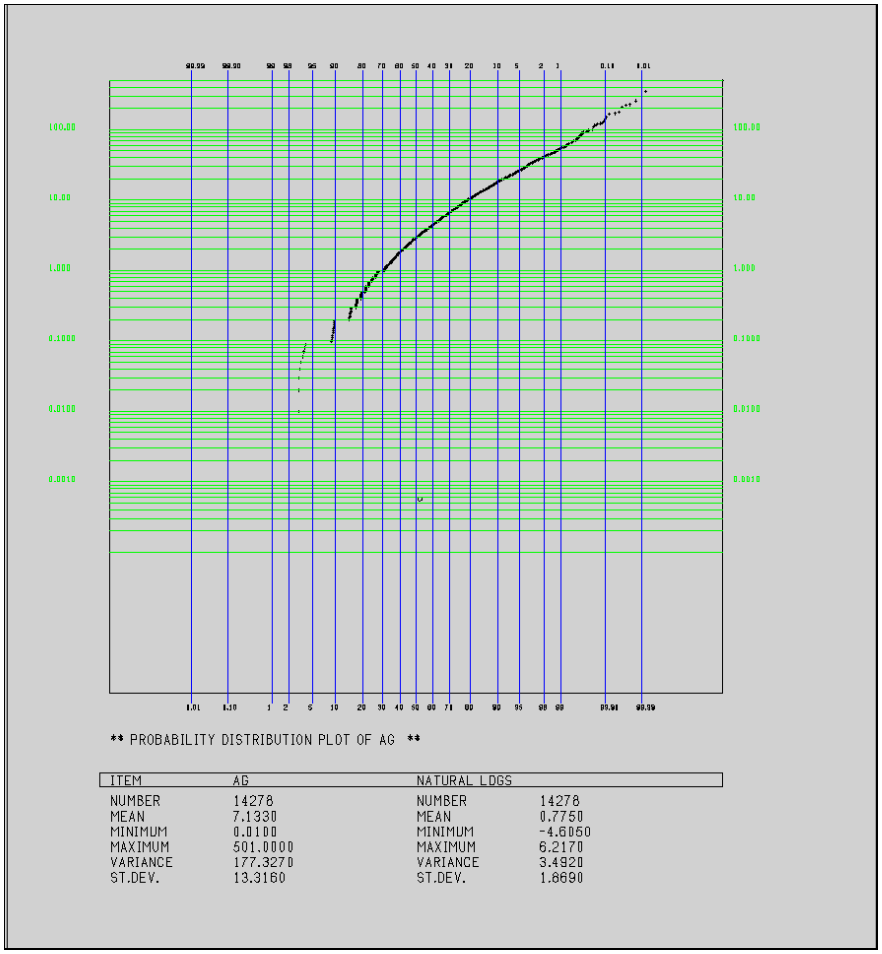
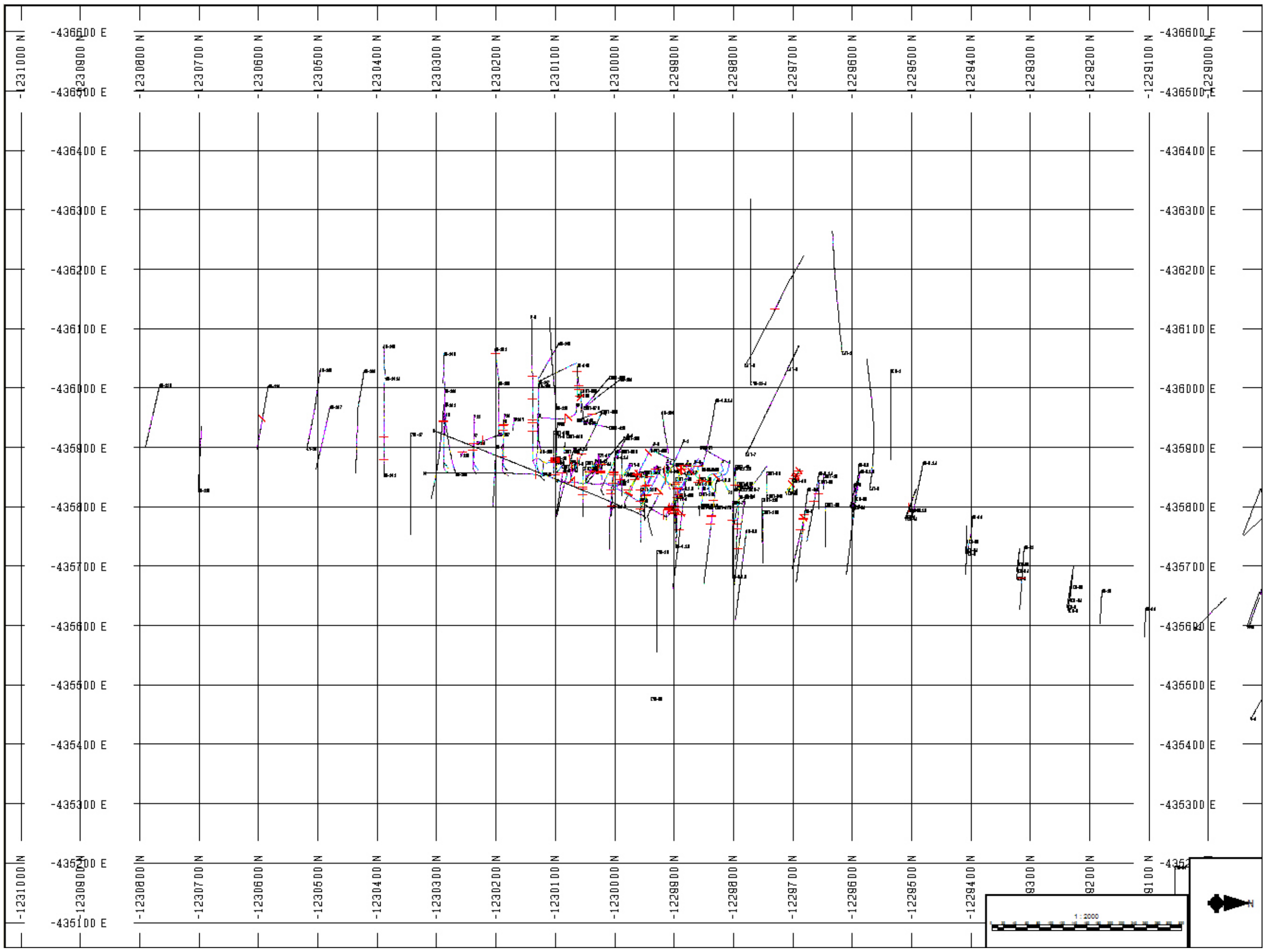


Figure 17.5: Plan view of the drill holes within the project area displaying the Slovak Grid System.



17.3 TOPOGRAPHY

Topography was imported from an AutoCAD topographic map supplied by Tournigan Gold Corp. in DXF format. Figure 17.6 shows a plan view of the topographic map and Figure 17.7 illustrates a three-dimensional rendition of the topography used, looking Northwest. Note that the elevations range between 620 and 800 meters as shown in Figure 17.8 with ridges extending from the south to the base of the historic pit and a large ridge along the west side of the pit high wall. This topographic surface was checked against drill hole collars along every section and it was determined that the drillhole collar elevations matched topography to within 2 meters with the exception of 2 drillhole outside the area of interest that are within 3 meters.

Figure 17.6: Plan View of Color-coded Topography, Kremnica Deposit with elevations ranging from 620 to 800m, approximately.

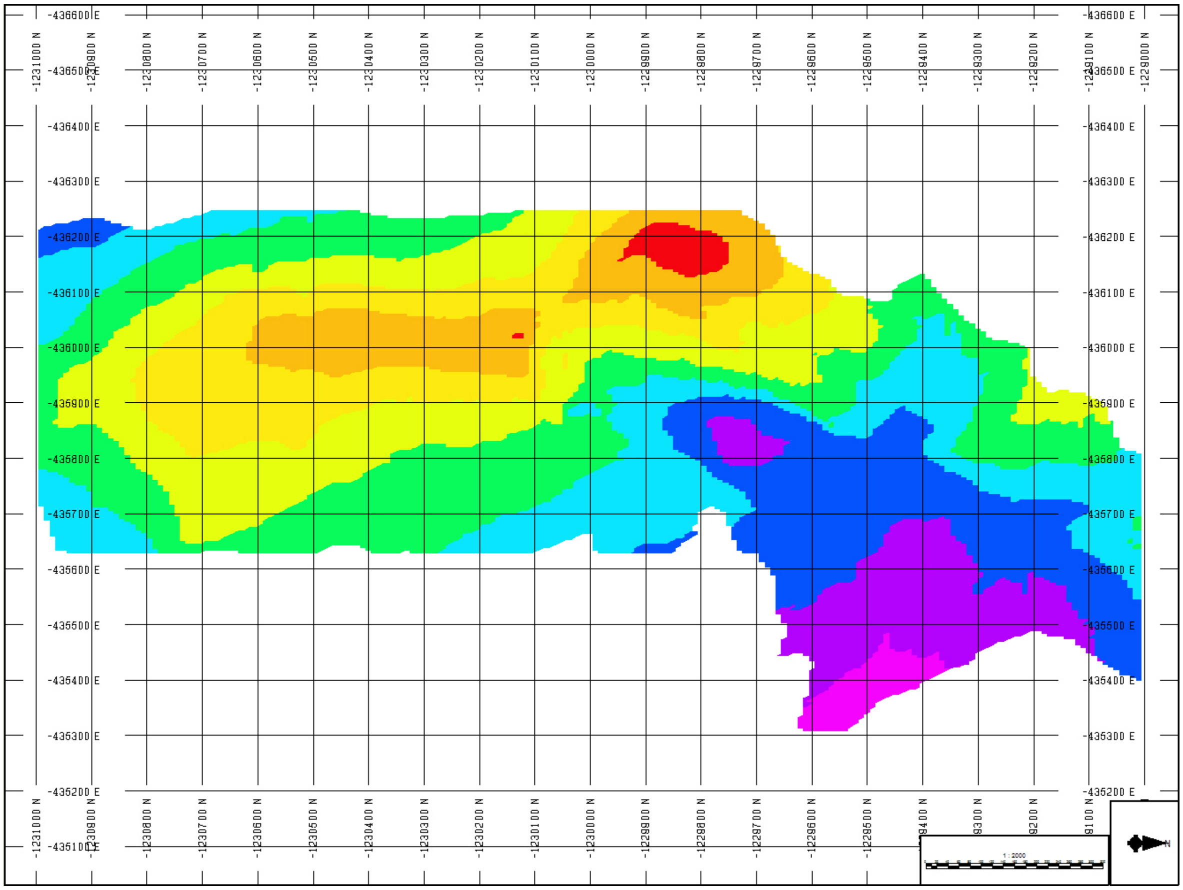


Figure 17.7: 3D Display of color-coded topography presents elevations ranging from 620m to 800m, approximately. Northwest is into page.

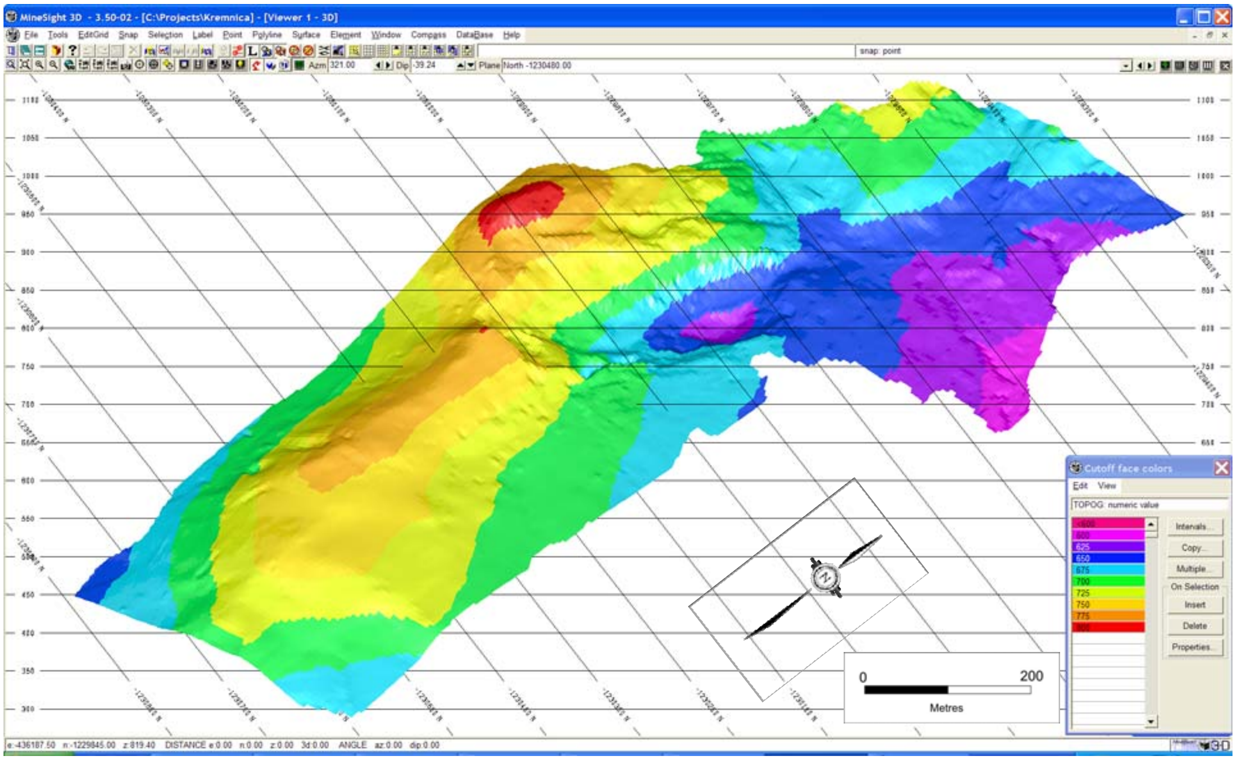
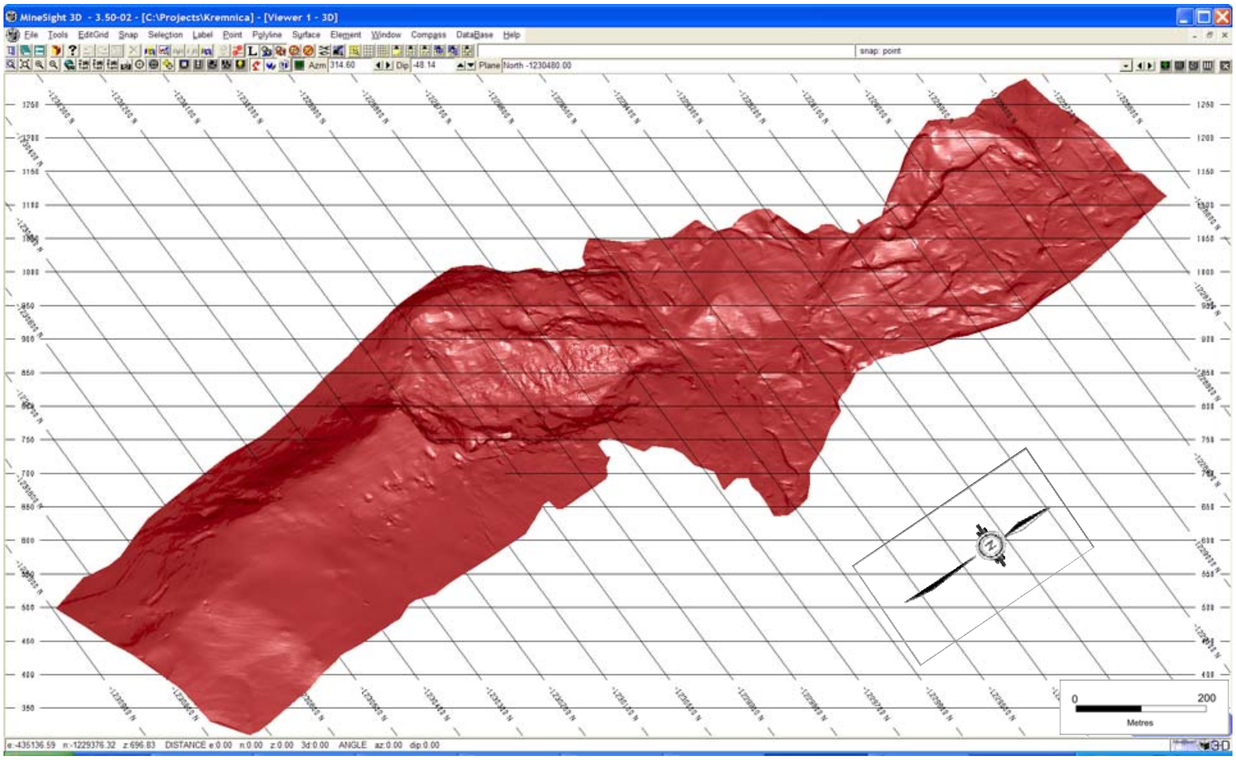


Figure 17.8: 3D Digital Terrain model. Northwest is into page.



17.4 DENSITY

A total of 134 specific gravity measurements were taken in order to accurately determine the relative density of the different materials. Table 17.4 lists the averages for each rock type encountered that is used in the specific gravity model as described in the following section.

Table 17.4: 2005 Specific Gravity Determinations for the Kremnica Deposit.

Rock Type	Sulphide/Oxide	Collapse Zone Material	Average SG for Samples g/cm^3
Vein	Oxide		2.30
Vein	Sulfide		2.37
Stockwork	Oxide		2.16
Stockwork	Sulfide		2.30
All		Collapse	2.17
Andesite-Hangingwall and Footwall			2.34

In previous studies, the specific gravity determinations varied to some degree as shown in Table 17.4. Overall, a comparison of the density values are now found to be lower than previously used for each rock type with the exception of what is considered the fill or rubble zone. The current measurements show that these areas are more competent than originally thought however the area that is affected, that being the collapse zone, is much larger than originally estimated, as described below. In addition, there does not appear to be a significant differential between the hanging wall and foot wall andesites.

Table 17.5: Density Factors - Previous Studies

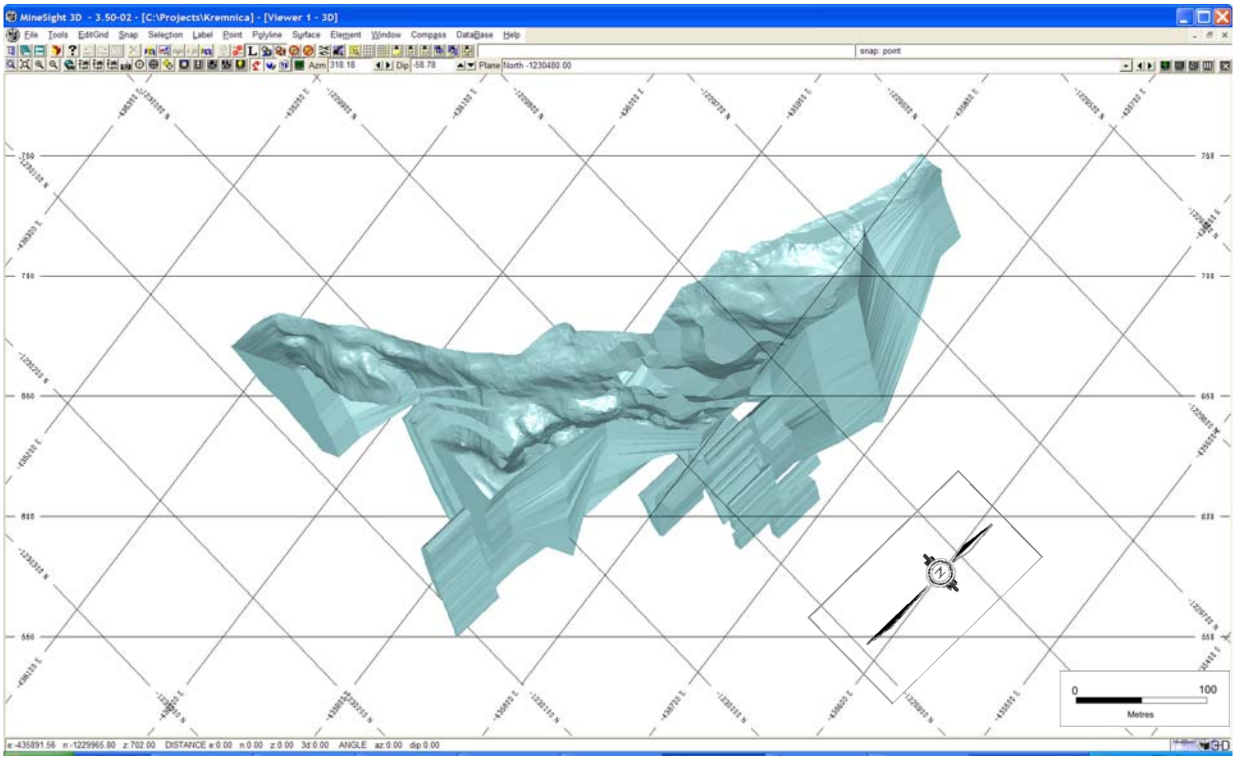
Rock Type	**Oxidation Class**		
	Oxide	**Mixed**	**Sulfide**
Vein	2.46	2.48	2.48
Stockwork	2.35	2.48	2.43
Fill / Rubble	1.84	1.84	1.84
Quartz Breccia (estimate)	2.46	2.46	2.46
Andesite	**All**		
Footwall-Milonite	2.32		
Hanging wall-Grey/Altered	2.09		

17.5 SPECIFIC GRAVITY MODELING

Specific gravity was assigned on a block-by-block basis that depended upon whether the block was within a zone predominantly composed of collapse zone material, oxide or sulphide material or within a void. Solids models of each were created in order to facilitate the coding of the specific gravity measurements into the block model.

First the collapse zone model was created by linking interpreted sections, every 50 meters through out the deposit. These sections (Figure 17-9) were derived from interpretations of drillhole data and the personal experience of John Cuthill, Senior Geologist at site. The resultant solid is then assigned an SG of 2.17 gm/cm^3 which is coded directly into the block model on a whole block basis. These were then used for tonnage calculations.

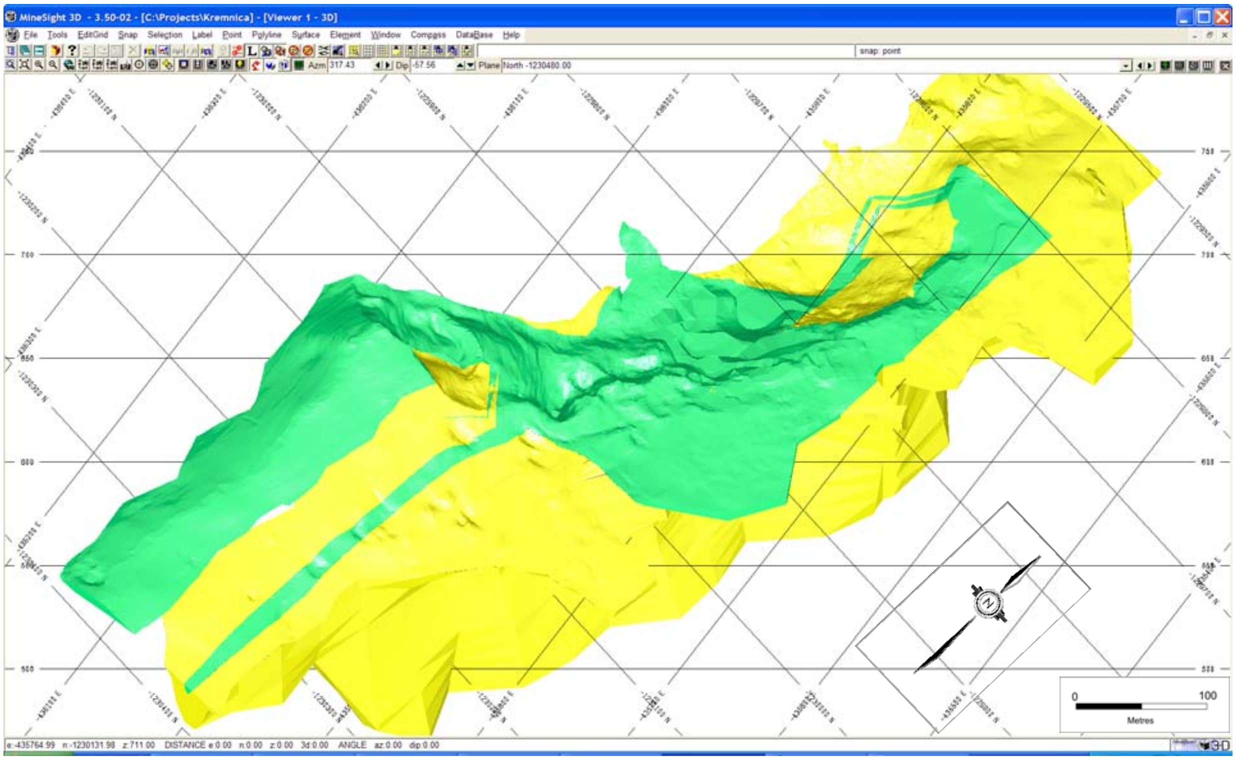
Figure 17.9: Collapse Zone solids model.



In addition to the collapse zone, a solids model, Figure 7.10, was created for the oxide/sulphide zones which were derived as separate sets of 50 meter sections and modelled individually. These sections were digitized into AutoCAD and then imported into MineSight for wireframe modelling. The resultant solids are shown below in Figure 17.10. As noted in Table 17.5, the vein and the stockwork have differing SG's depending upon whether they are oxide or sulphide. The same process of coding the specific gravity measurements into the solids and then using these to code the block model with the values was followed.

It should be noted that the interpretation of the oxide and sulphide zones vary from previous interpretations in that there appears not to be a mixed oxide/sulphide zone as previously thought. It appears that all zones have some oxide and sulphide component to them and are therefore all mixed to some extent however the oxide zone is more strongly oxide than sulphide and the sulphide zone is more sulphide than oxide. In addition, there is no evidence that would indicate that the gold and silver mineralization is controlled by the oxide and/or sulphide content. These hypotheses are primarily based on interviews with on-site geological staff, specifically John Cuthill which the author believes to be an accurate representation based on current understanding of the deposit.

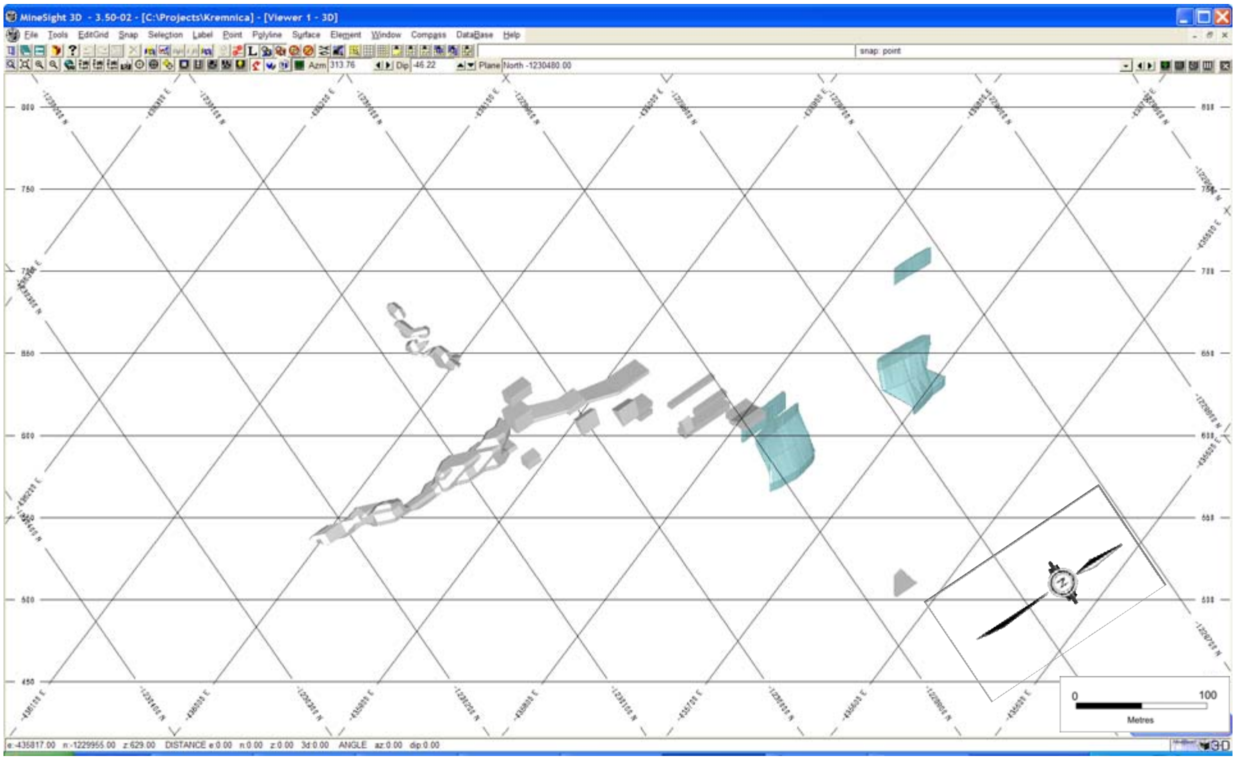
Figure 17.10: Oxide (green) and sulphide (yellow) solids model.



Lastly, a solids model for all known voids were created. These detailed models of the voids were derived from coordinates provided by site personnel; in particular Mr. Finka who has a long history with the site and is the last person who have visited the underground openings, knows their location and is the best source of knowledge. Additionally, voids were created from the source drill data and plotted on the geology sections and plans by John Cuthill. Finally, void data derived from previous studies such as the WMC Report and the 2003 Preliminary Assessment was also incorporated. The result, as shown in Figure 17.11, is a 3D solids model representing the most current and comprehensive understanding of the voids underground. This void model was then used to mask the blocks intersecting by 10% or more of its volume. In any such case the SG was then set to 0.0 to account for these volumes of air.

It should be noted that the void model was also utilized in the grade interpolation process, as described in the following sections. These solids were used for masking out these areas within the composite data base and within the block model to insure that grades were not interpolated into these regions.

Figure 17.11: Solids model of voids.



17.6 COMPUTERIZED GEOLOGIC MODELING

Solids models of the main geologic units controlling mineralization were derived from detailed sectional and plan interpretations created by John Cuthill. Sections were created every 50 meters and plans every 20 meters as shown in a representative section and plan in Figure 17.12 and 17.13. These sections and plans were then digitized into AutoCAD and then imported into MineSight for wireframe modelling. The resultant solids are shown below in Figure 17.14 and 17.15. In addition, the solids models for the voids or openings, as described above, were incorporated into the solids modelling process and subsequently used for the coding, geologic masking and grade interpolation.

As mentioned, the sectional and plan interpretations were created and the lithologic and grade intersections coded into the assay data base in plan view along with incorporating orientation of the mineralized zone observed within the drift and on surface. The solids were used to then code the drillhole assays and composites and for subsequent geostatistical analysis. In addition, these solids are assigned a numeric code which is then coded directly into the block model for geologic matching. This process entails matching the code assigned to the assays and composites with those within the block model so that the composites for any one unit are constrained to the geologic unit from which they occur. This process is described further within the interpolation section.

Solids of the vein and stockwork were created from linking polylines of the geologic units derived from plans and sections created by the company, namely John Cuthill, Sr. Geologist. The author believes that the geology as represented is an accurate characterization of the geologic units and is a realistic depiction of the main units controlling mineralization.

Figure 17.12: Geologic Section from which geologic solids are derived looking north. Topo is blue, vein is red, quartz breccia is orange and andesite (hangingwall and footwall) are green.

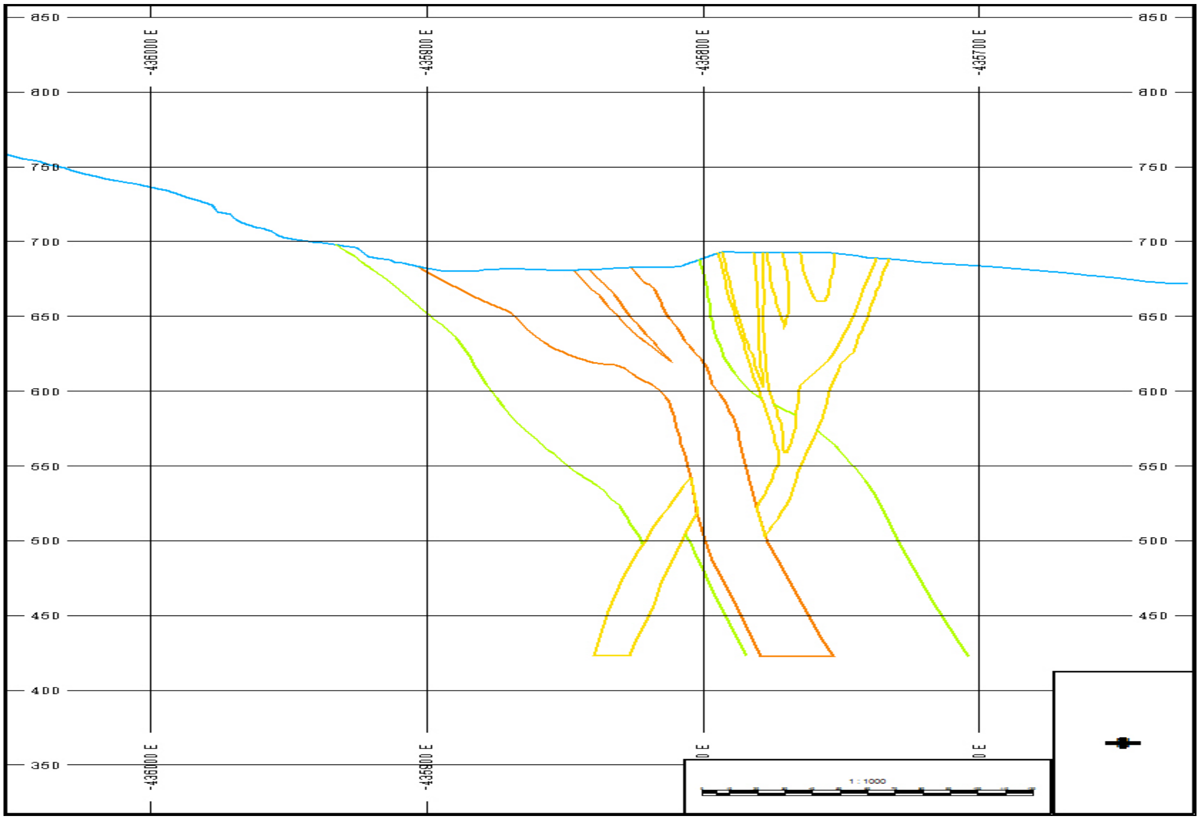


Figure 17.13: Plan section of geology. Vein is red, quartz breccia is orange, stopes and voids are light blue and andesite (hangingwall and footwall) are green.

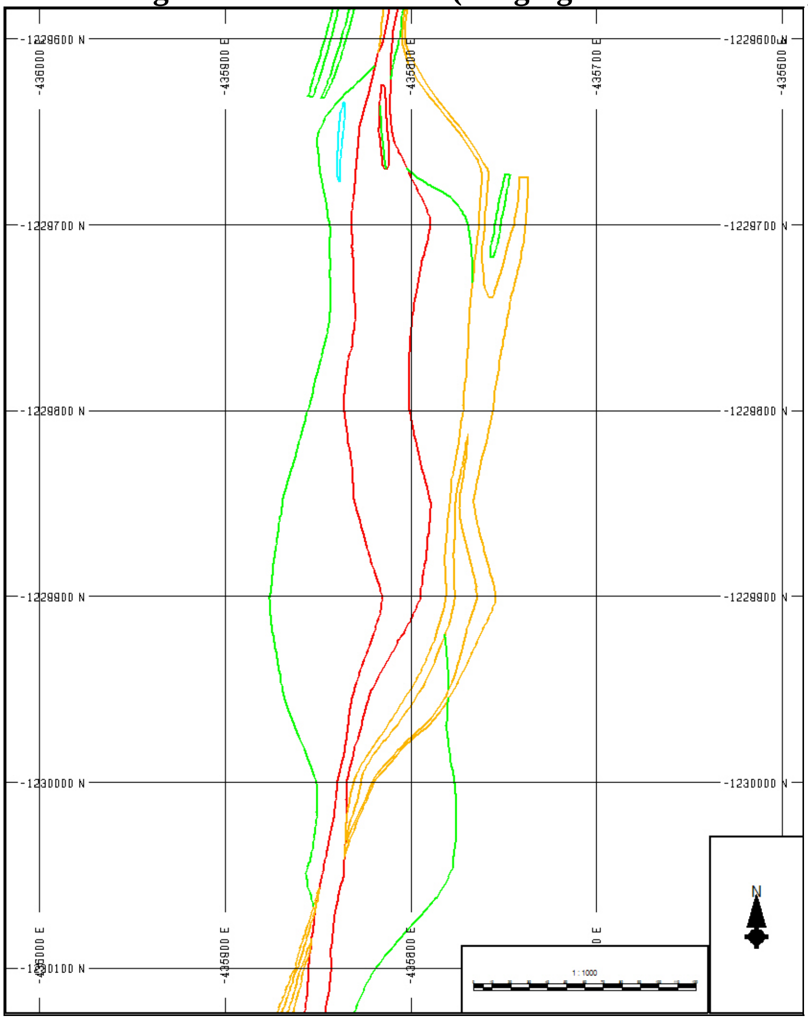


Figure 17.14: Solids model of vein material.

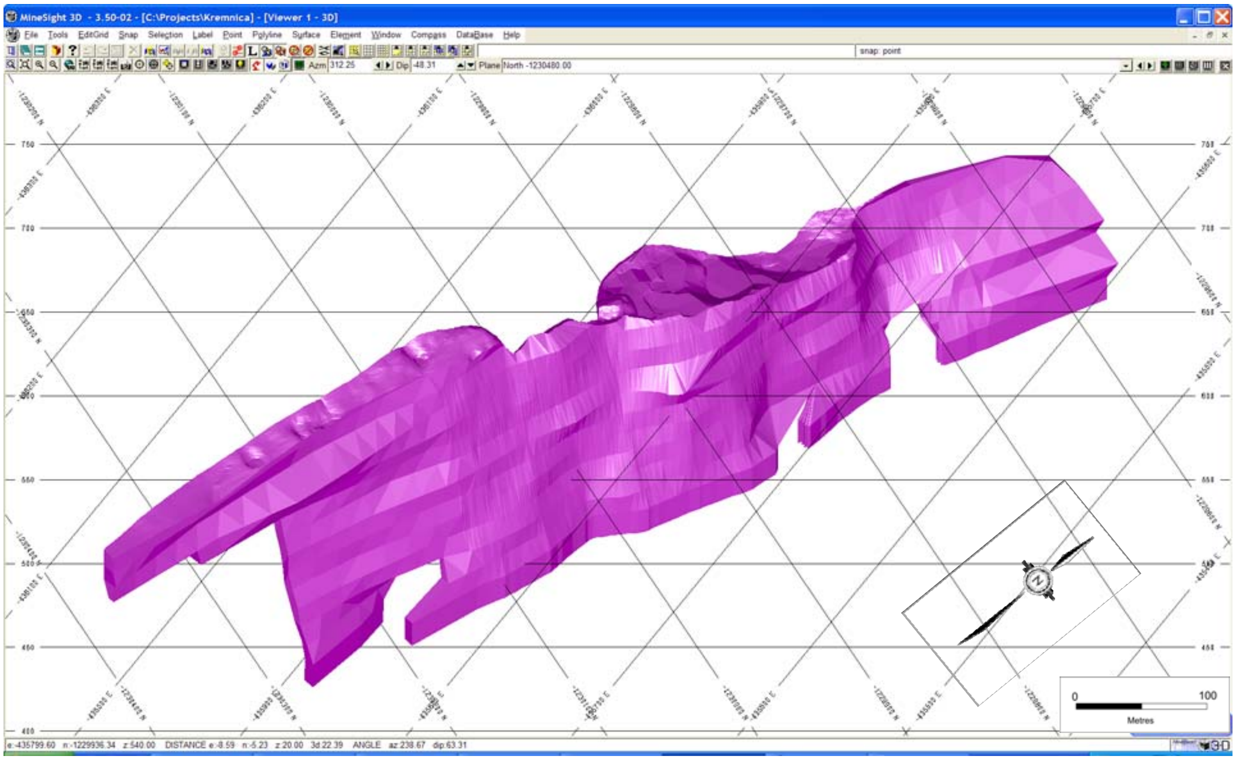
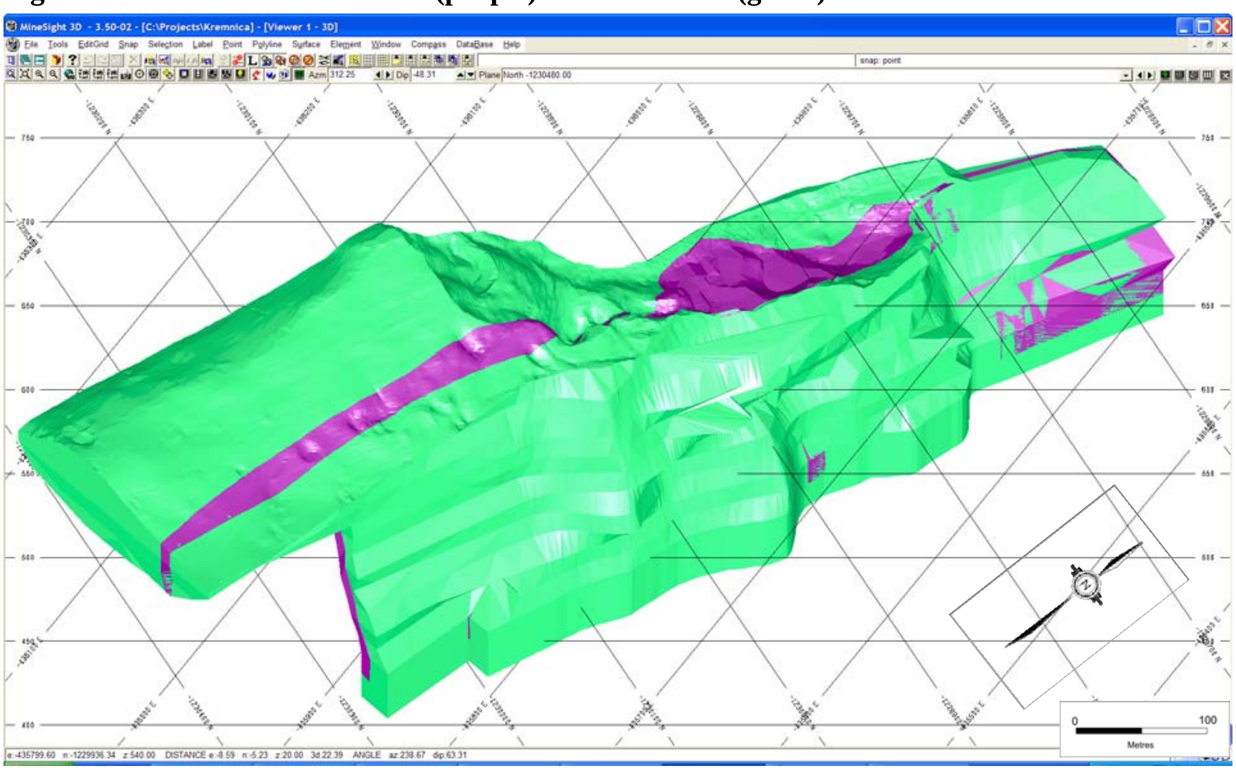


Figure 17.15: Solids model of vein (purple) and stockwork (green).

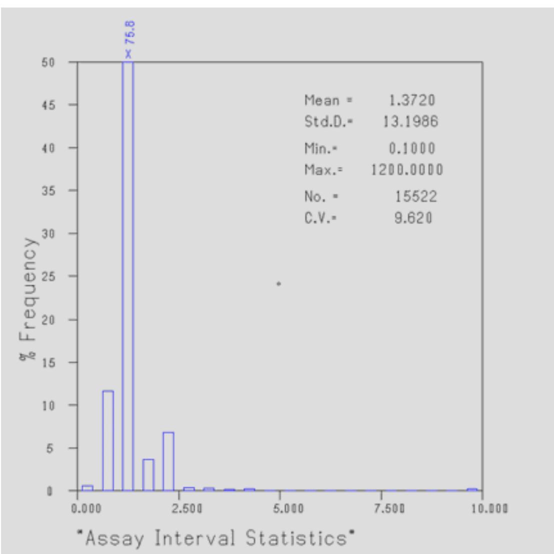


17.7 COMPOSITING

Diamond drillhole, RC drillhole, underground drift chip, underground drillhole and bench channel assay data was composited to 3 meter intervals. Composites tails at the end of holes were retained if greater than 1.5 meters and combined with the preceding interval if less than 1.5 meters. All other composite tails such as those at the transition between geologic zones or directly in contact with a void were retained if greater than 1.5 meters and discarded if less than 1.5 meters in length.

A three meter composite length was chosen due to the fact that greater than 80% of the data has an the assay interval length of one meter or less with the overall mean length being 1.37 meters as shown in Figure 17.16. This was slightly skewed due to the fact that there were some very large intervals. In addition, with the treatment of the recovery issues within the data base, as described above, and the existence of the voids, it was decided by the author that larger composite lengths would greatly dilute that assay data and not give a realistic result. Therefore, in terms of selectivity and estimation quality, it was decided that a 3 meter composite provided the best compromise between number of composites available for estimation, and a reasonable degree of dilution and regularization.

Figure 17.16: Histogram and Basic Statistics Assay Intervals



Composites are needed to smooth out the noisy assay data and to regularize the intervals used for estimation. Another important factor is that they incorporate some sub-vertical dilution into the model, which aids in estimating a fully diluted model.

Tables 17.6 and 17.7 summarizes statistics for the complete Au and Ag assay (i.e. after selection criteria has been applied as described above) database (i.e. between 1,230,500 South and 1,229,450 South) used for the resource evaluation. The composite database has 4,932 Au and 4,879 Ag values with minimum value 2 of 0.001 and 0.000, respectively. The average overall Au grade is 0.8478 g/t, with a standard deviation of 1.582, resulting in a fairly high coefficient of variation[4] (CV) of 1.842. As for Ag grades, the average overall grade is 7.2671 g/t with a standard deviation of 11.1223, resulting in a fairly high coefficient of variation (CV) of 1.531.

Figures 17.17 through 17.20 show the histogram and statistics along with probability plots for the 3 meter Au and Ag composites, respectively.

Table 17.6: Statistics of all Au composites by Cut-off Grade.

Cutoff	Weight	%	AU	SD	CV
0.0	4932	100.0%	0.85	1.56	1.84
0.5	1989	40.3%	1.88	2.06	1.09
1.0	1308	26.5%	2.49	2.31	0.93
1.5	880	17.8%	3.10	2.61	0.84
2.0	595	12.1%	3.75	2.96	0.79
2.5	409	8.3%	4.44	3.35	0.75
3.0	293	5.9%	5.11	3.75	0.73
3.5	198	4.0%	6.01	4.28	0.71
4.0	142	2.9%	6.91	4.76	0.69
4.5	100	2.0%	8.03	5.30	0.66
5.0	75	1.5%	9.12	5.71	0.63
5.5	60	1.2%	10.08	6.02	0.60
6.0	53	1.1%	10.67	6.18	0.58
6.5	45	0.9%	11.45	6.39	0.56
7.0	36	0.7%	12.64	6.64	0.53
7.5	31	0.6%	13.51	6.77	0.50
8.0	28	0.6%	14.12	6.86	0.49
8.5	26	0.5%	14.56	6.92	0.48
9.0	23	0.5%	15.32	7.01	0.46
9.5	21	0.4%	15.90	7.07	0.44
10.0	20	0.4%	16.21	7.11	0.44
10.5	19	0.4%	16.53	7.16	0.43
11.0	17	0.3%	17.21	7.28	0.42
11.5	14	0.3%	18.53	7.39	0.40
12.0	12	0.2%	19.63	7.43	0.38
12.5	10	0.2%	21.09	7.30	0.35
13.0	10	0.2%	21.09	7.30	0.35
13.5	9	0.2%	21.94	7.20	0.33
14.0	7	0.1%	24.27	6.37	0.26
14.5	7	0.1%	24.27	6.37	0.26
15.0	6	0.1%	25.87	5.22	0.20

[4] The coefficient of variation is defined as CV=σ/m (standard deviation/mean), and represents a measure of variability that is unit-independent. This is a variability index that can be used to compare different and unrelated distributions.

Table 17.7: Statistics of all Ag composites by Cut-off Grade.

Cutoff	Weight	%	AG	SD	CV
0	4879	100.0%	7.27	11.12	1.53
1	3735	76.6%	9.38	11.94	1.27
2	3060	62.7%	11.13	12.53	1.13
3	2620	53.7%	12.59	12.99	1.03
4	2280	46.7%	13.94	13.40	0.96
5	2018	41.4%	15.17	13.78	0.91
6	1766	36.2%	16.56	14.19	0.86
7	1572	32.2%	17.80	14.57	0.82
8	1421	29.1%	18.90	14.91	0.79
9	1267	26.0%	20.16	15.32	0.76
10	1147	23.5%	21.27	15.69	0.74
11	1026	21.0%	22.54	16.12	0.72
12	925	19.0%	23.75	16.54	0.70
13	839	17.2%	24.90	16.95	0.68
14	745	15.3%	26.35	17.46	0.66
15	673	13.8%	27.61	17.91	0.65
16	607	12.4%	28.93	18.38	0.64
17	562	11.5%	29.93	18.75	0.63
18	522	10.7%	30.89	19.12	0.62
19	466	9.6%	32.37	19.72	0.61
20	429	8.8%	33.49	20.17	0.60
21	382	7.8%	35.10	20.82	0.59
22	345	7.1%	36.56	21.40	0.59
23	314	6.4%	37.95	21.95	0.58
24	289	5.9%	39.21	22.44	0.57
25	268	5.5%	40.36	22.91	0.57
26	248	5.1%	41.56	23.41	0.56
27	225	4.6%	43.10	24.05	0.56
28	209	4.3%	44.30	24.55	0.55
29	193	4.0%	45.61	25.10	0.55
30	176	3.6%	47.16	25.77	0.55

Figure 17.17: Histogram and Basic Statistics of 3 meter Au Composites

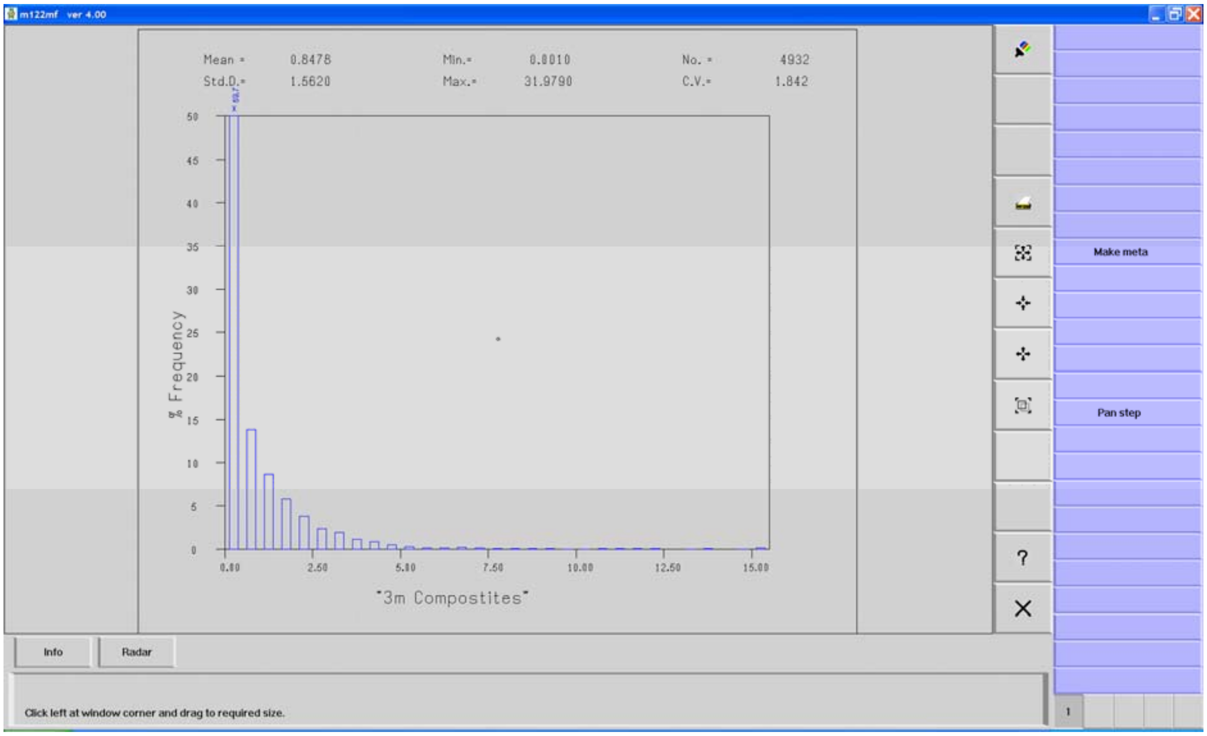


Figure 17.18: Probability Plot and Statistics for 3 meter Au Composites.

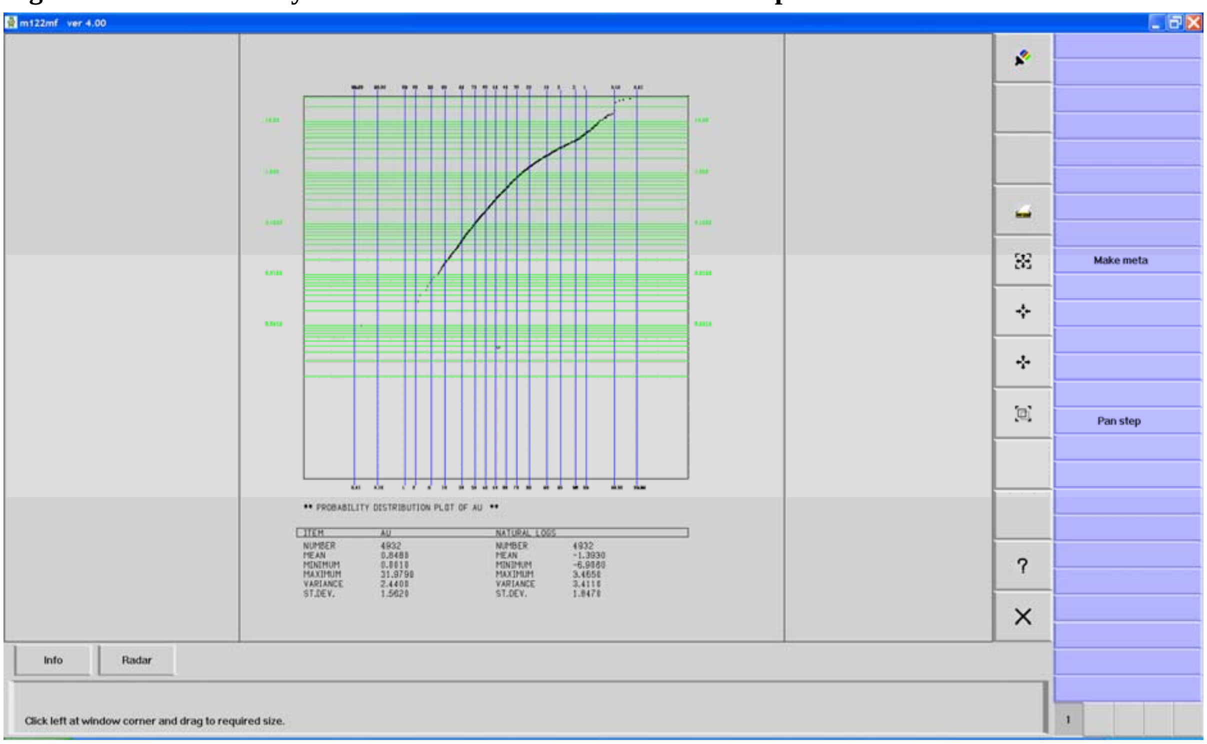


Figure 17.19: Histogram and Basic Statistics of 3 meter Ag Composites

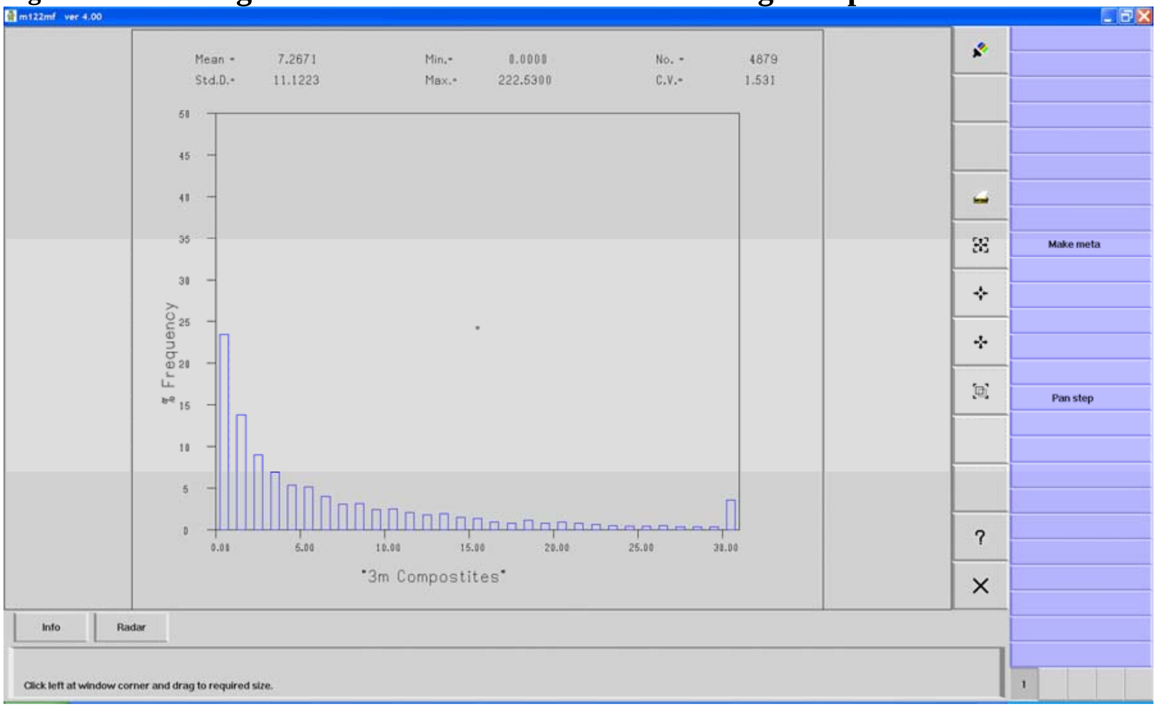
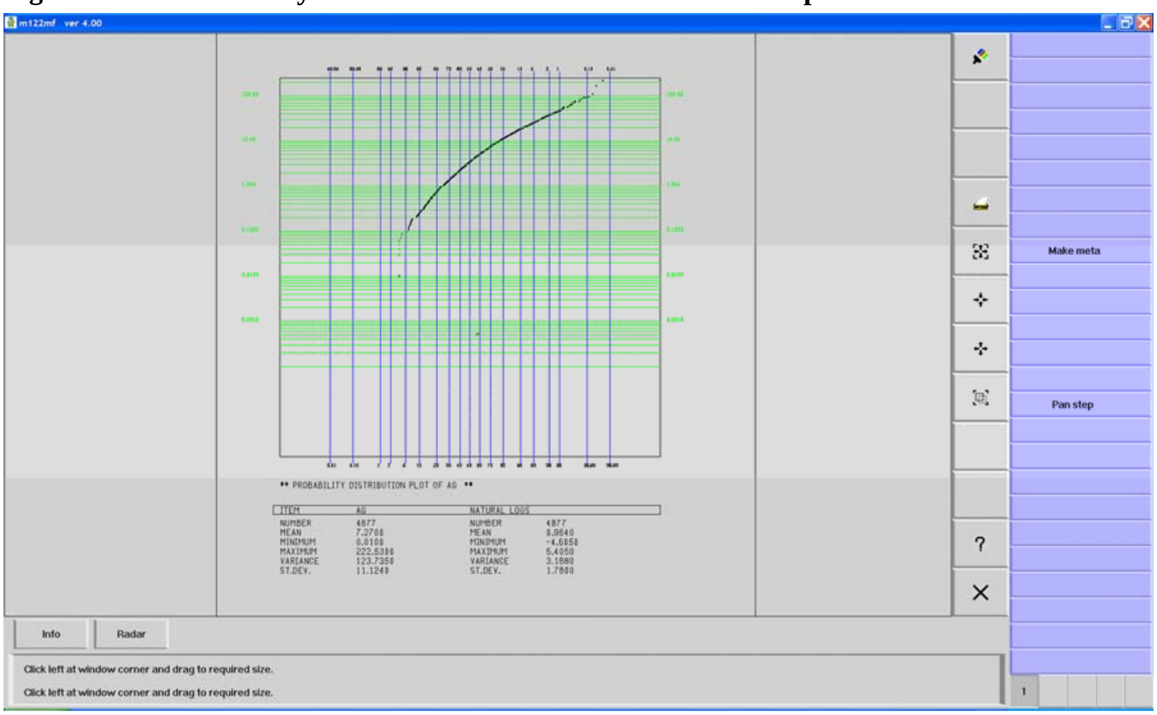


Figure 17.20: Probability Plot and Statistics for 3 meter Au Composites.



17.8 AU AND AG HIGH GRADE OUTLIERS

As previously mentioned, there are a number of outliers within the gold and silver populations that require addressing. To better understand the effect of a few high grade Au and Ag composites on the estimation of grades into the block model, a small study was performed to decide whether it was necessary to cut (cap) grades, and to what extent. It should be emphasized that, the impact of high grades in the Kremnica Deposit is not expected to be significant, since the high grade population is not large relative to the overall database.

The analysis was based on looking at the cumulative probability curve by Au and Ag cutoff grades as shown in Figures 17.21 and 17.22, respectively. Controlling the overestimation of Au and Ag grades due to a few high-grade outliers is, in effect, an arbitrary decision as to how much Au and Ag should be removed from the database to avoid such overestimation of grades in the resource model.

It can be seen that, the cumulative distribution curve begins to loose continuity at some point above 10 g/t for gold and 90 g/t silver. Therefore, 12 g/t for Au and 90 g/t for Ag was chosen as the most reasonable threshold at which to limit grades from the composited database. This represents 0.5% and 0.2 % of the total number of Au and Ag composites, respectively.

It is important to note the method employed for this study is not to cut the high grade outliers but to limit their influence. The range chosen at which to limit grades greater than 12 g/t Au composites and 90 g/t Ag composites was the size of one block or 10 meters. In other words, composite grades greater than 12 g/t Au and 90 g/t Ag, would not be used in the estimation of blocks if those high grade composites are outside a 10 meter radius from that block.

Figure 17.21: Zoom of Au Probability Plot Illustrating Break between 10 and 20 gpt.

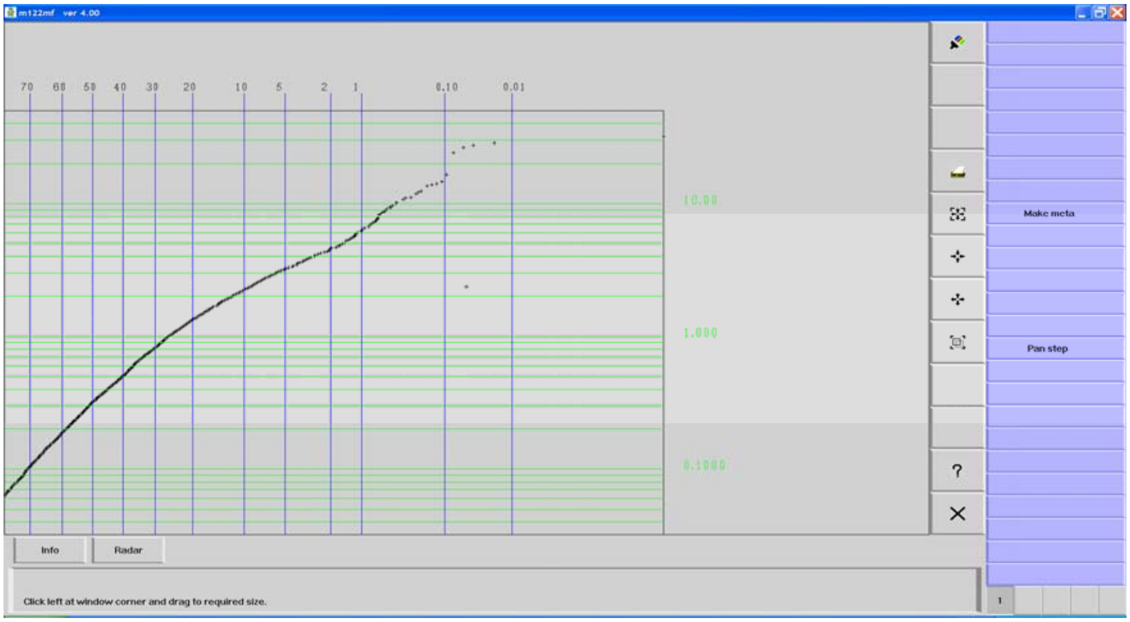
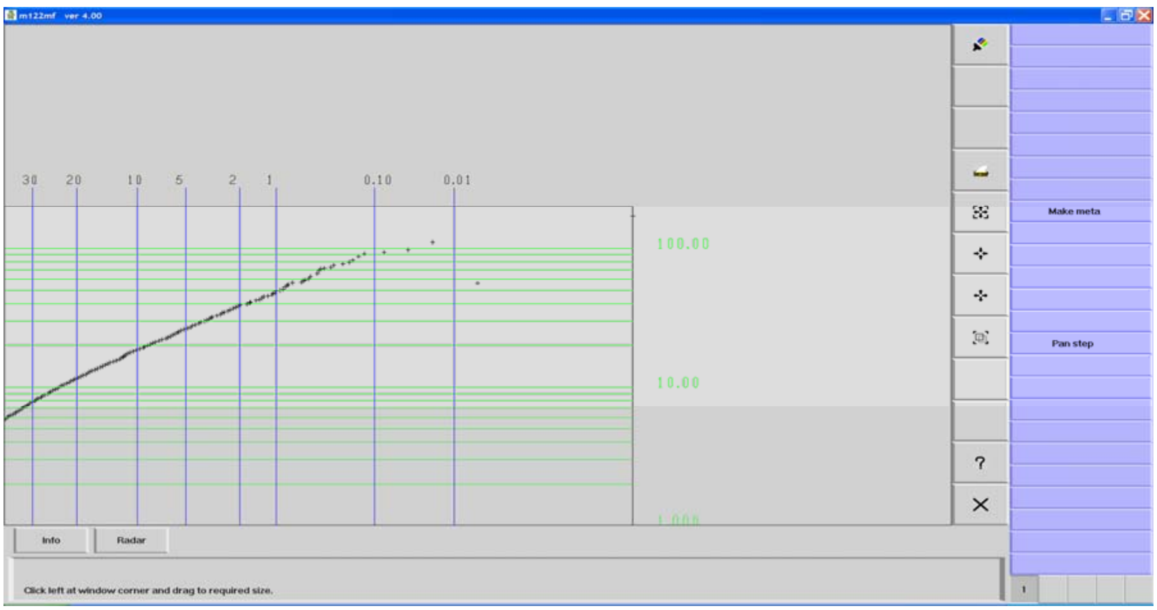


Figure 17.22: Zoom of Ag Probability Plot Illustrating Break between 90 and 100 gpt.



17.9 VARIOGRAPHY

The grade estimation methodology used did not involve kriging so the geostatistics were of limited relevance. The author carried out a preliminary geostatistical analysis on the composites, however, to evaluate the search parameters used in the grade estimate

Downhole correlogram was generated in order to make an estimate of the nugget effect as that is the direction in which there is most abundant data. The downhole correlogram indicates that the nugget effect is in the order of 65% and 45% of the sill for Au and Ag as shown in Figures 17.23 and 17.24, respectively. This is somewhat high but within an acceptable range for Au and Ag deposits. Note the downhole range is 75 meters and 100 meters for Au and Ag, respectively

Figure 17.23: Downhole correlogram for Au.

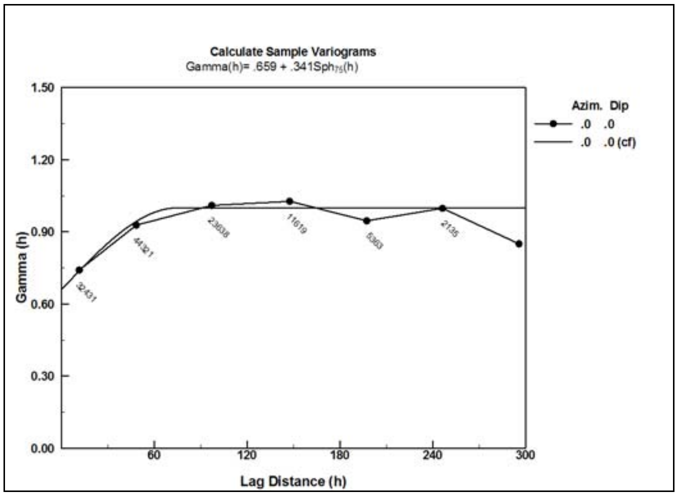
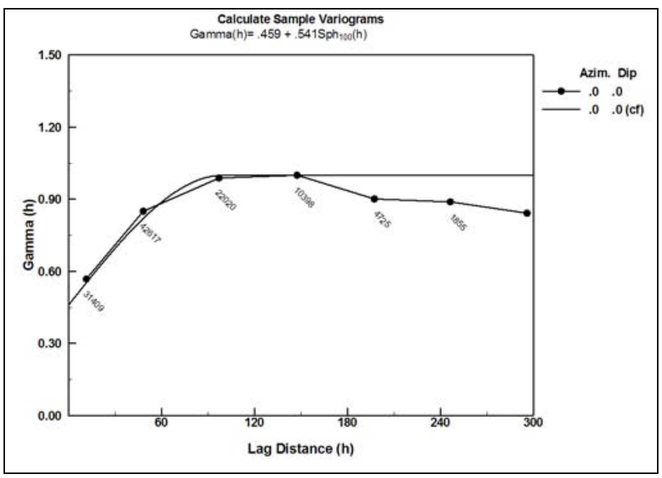


Figure 17.24: Downhole correlogram for Ag.



Geostatistical analyses were performed on the assays and composites using no constraints in addition to the coded intervals within the zone solids (i.e. vein, stockwork and andesite) and the oxide and sulphide solids.

The ellipsoid direction chosen for the estimation process was chosen to be 100 degrees azimuth and 67 degrees dip for the major axis, 10 degrees and 0 degrees for the minor axis and 100 degrees and 23 degrees for the vertical axis. This direction follows the orientation of the vein, which is the major mineralized structure in addition to the stockwork envelope along strike and down dip. Variography illustrated that the shortest range at which composites have a demonstrated relationship was 154m, 82m and 19m for Au and 217m, 90m and 19m for Ag.

In preparation for the implementation of the grade estimation method as described in Section 17.12, variograms for Au grades were run using the 3 meter composites. The spatial continuity estimator chosen for this study was the correlogram, which has been shown in previous work to be more robust with respect to drift and data variability, allowing therefore for a better estimation of the observed continuity (Parker and Srivastava, 1988).

It should be noted that these correlograms resulted in a relatively high nugget effect. This was the main reason why the author decided that an ID^3 method was more appropriate for the resource estimate. Note that the sill of the variograms has been standardized to one, and therefore they are in fact relative variograms.

Figures 17.25 and 17.26 show the summary of the correlogram model used to guide the estimate of the Kremnica resources. The rotation of the angles are given according to the convention used by GSLIB in MineSight Compass.

Figure 17.25: Summary Description of the Au Variogram Model

Calculate Sample Variograms

GSLIB Rotation Conventions

Nugget ==> 0.725
C1 ==> 0.275

First Structure -- Exponential with Traditional Range

LH Rotation about the Z axis ==> -65
RH Rotation about the X' axis ==> 69
RH Rotation about the Y' axis ==> -69
Range along the Z' axis ==> 81.9 Azimuth ==> 185 Dip ==> 7
Range along the Y' axis ==> 153.8 Azimuth ==> 295 Dip ==> 69
Range along the X' axis ==> 18.8 Azimuth ==> 93 Dip ==> 19

Modeling Criteria

Minimum number pairs req'd ==> 300
Sample variogram points weighted by # pairs

Figure 17.26: Summary Description of the Ag Variogram Model

Calculate Sample Variograms

GSLIB Rotation Conventions

Nugget ==> 0.526
C1 ==> 0.474

First Structure -- Exponential with Traditional Range

LH Rotation about the Z axis ==> -72
RH Rotation about the X' axis ==> 67
RH Rotation about the Y' axis ==> -71
Range along the Z' axis ==> 89.1 Azimuth ==> 180 Dip ==> 7
Range along the Y' axis ==> 217.0 Azimuth ==> 288 Dip ==> 67
Range along the X' axis ==> 19.3 Azimuth ==> 87 Dip ==> 21

Modeling Criteria

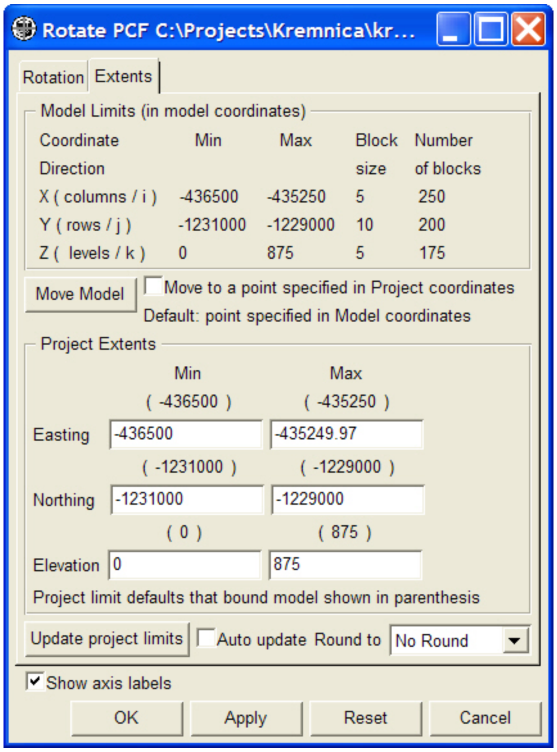
Minimum number pairs req'd ==> 300
Sample variogram points weighted by # pairs

17.10 THE KREMNICA BLOCK MODEL DEFINITION

The block size chosen was 5m x 10m x 5m oriented at an un-rotated east, north, elevation respectively, in an effort to adequately decretize the mineralized zones so as not to inject an inordinate amount of internal dilution and to somewhat reflect drill hole spacing available. In addition, it is planned that the pit bench height is to be 10 meters therefore retaining a block height that is a multiple of the bench height is desirable for pit optimization and pit design studies that are to be performed as a part of the pre-feasibility study.

The Kremnica Resource Block Model used for calculating the bulk tonnage resource was defined according to the limits shown in Figure 17.27

Figure 17.27: Block Model Limits. Note that coordinates are listed in negative northings and eastings (i.e. westings and southings).



- Minimum Westing: 436500 W;
- Maximum Westing: 435250 W;
- Minimum Southing: 1231000 S;
- Maximum Southing: 1229000 S;
- Minimum Elevation: 0;
- Maximum Elevation: 875.

Of the potential 8,750,000 blocks to be estimated (200 rows, 250 columns, 175 levels), less than 141,000 blocks or approximately 2% have estimated values in them (weighted against topography and ore zone). This is primarily due the constraints applied to the estimation process namely the limited search distances applied, search ellipsoid direction and the use of inverse distance to the third power as the modeling method.

17.11 ID3 MODEL

The choice interpolator was that of inverse distance to the 3rd power. For an open pit, bulk tonnage scenario, inverse distance to the second or third power is preferred. In this case where there is a large vein being evaluated from an open pit mining perspective, inverse distance to a higher power is more appropriate as it localizes the grade to a certain extent.

Correlograms and other variogram estimators were used to attempt to obtain a spatial variability model that could be used in the estimation of the resources. Since the models obtained had, in all cases, relative nugget effects higher than 65% and the fact that the coefficient of variance for the dataset is relatively high, it was decided that any form of kriging would be ineffective, and that an inverse distance method would be just as appropriate.

From a simplistic point of view, inverse distance weighting applies more weight to closer samples and less to those farther away depending upon the power utilized. The weight of each sample is inversely proportional to its distance from the point or block being estimated. From one end of the spectrum, inverse distance to the 0 power is just a straight arithmetic average and on the other end of the spectrum is inverse distance to the 10^{th} power for example which will heavily weight the point closest to the virtual exclusion of all others.

The implication of the high relative nugget effects is that the spatial continuity for the material is poor and that any resource estimate obtained within this will have poor local accuracy although the global resource is expected to be reasonably well estimated.

17.12 ESTIMATION PLANS

The three estimation passes were used to estimate the Resource Model because a better block-by-block estimation can be achieved by using more restrictions on those blocks that are closer to drill holes, and thus better informed. The three passes are based on increasing levels of information used to estimate blocks where each pass is less constrained than the previous pass. Knowing which block was estimated with what level of information (on which pass) provides an indicator for resource classification, (See Table 17.8)

<u>Pass 1</u> - A search ellipse with dimensions of 18 m at azimuth 100^{o} and dip of 23^{o}, 36 m at azimuth 10^{o} dip 0^{o} and 51 m at azimuth 100^{o} dip -67^{o} was used to find a minimum of 5 and maximum of 10 composites to estimate a block. In addition, a maximum of 2 composites per drillhole were allowed. The blocks estimated within this pass are then classified as measured resource blocks.

<u>Pass 2</u> - For blocks not estimated during Pass 1 the search ellipse was expanded to 2 times that in pass 1 the ranges and a minimum of 4 and maximum of 12 composites was required to estimate the block. In addition, a maximum of 2 composites per drillhole were allowed. The blocks estimated within this pass are then classified as indicated resource blocks.

Pass 3 - For blocks not estimated by Pass 2 the search ellipse was expanded to three times the ranges of the 1st pass in each direction. A minimum of 3 and maximum of 14 composites were required to estimate the block. In addition, a maximum of 2 composites per drillhole were allowed. The blocks estimated within this pass are then classified as inferred resource blocks. In addition, the maximum distance that the closest a composite within the footwall and hangingwall andesite could be informed was 50 meters in an effort not to smear outlying composites or overestimate the inferred blocks on the outer regions of the deposit.

Table 17.8: Estimation Plan, ID^3 Grade Estimation.

Pass	*Search in X, Y, and Z*	*Min No. of Comps*	*Max No. of Comps*	*Min No. of DDHs*	*Outlier Cut-off Au*	*Outlier Cut-off Ag*	*Dist. To Limit*
1	51mx36mx18m	5	10	2	12	90	10m
2	102mx72mx36m	4	12	2	12	90	10m
3	153mx108mx54m	3	14	2	12	90	10m

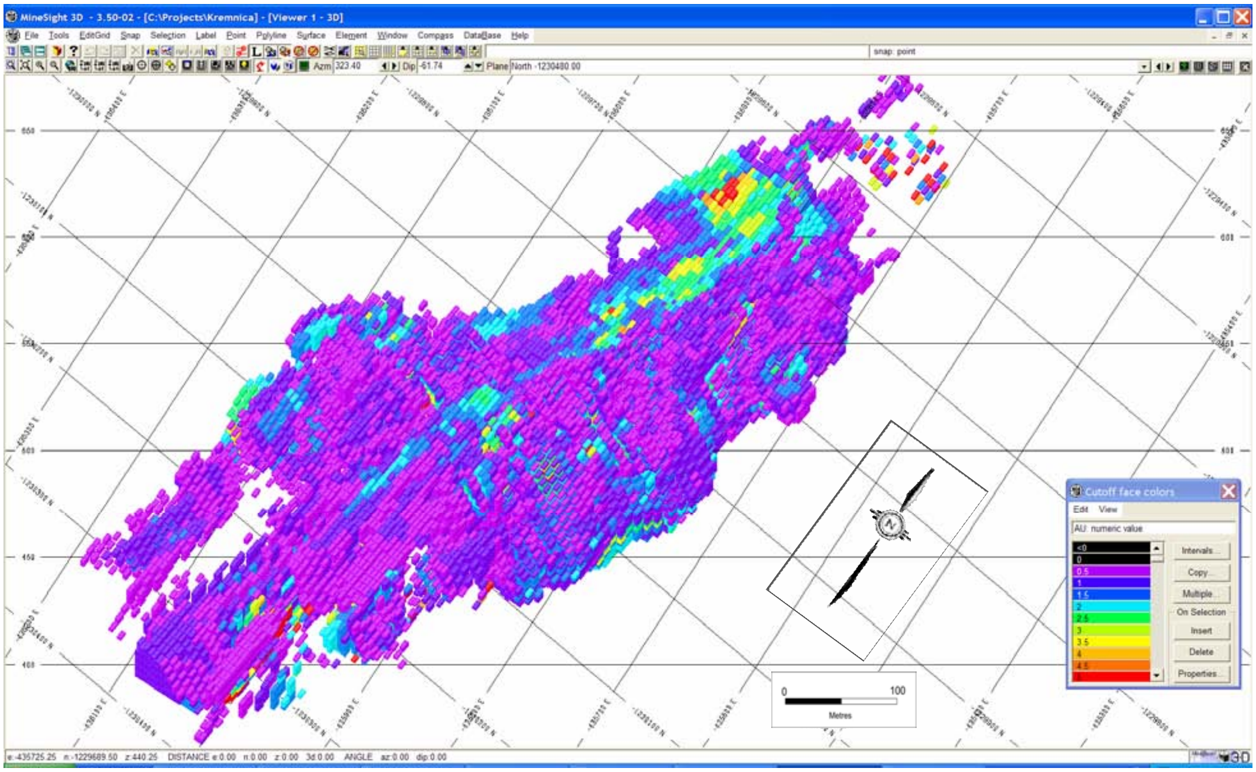
Geologic matching is applied in all cases utilizing the geologic solids previously created to constrain the interpolation process. Therefore, in the first 3 passes all blocks lying within the vein solid are estimated as described above using the composites that were correspondingly tagged within those same solids. This process was then repeated for the solids created for the stockwork.

The same process and estimation parameters are repeated for those blocks lying within the hangingwall and footwall andesites. One exception is that the farthest distance that the closest composite may be informed is 50 meters as noted above. This feature eliminates the smearing of a small number of composites with relative elevated grade throughout a large volume, that being all blocks outside the vein and stockwork. All blocks estimated within the andesite are deemed to be categorized as inferred resource blocks.

As discussed previously, 3m composites were used in the estimation. The model reported in this section uses all drill hole data available and may be therefore considered diluted.

Also, an octant search with a maximum of 2 composites being informed per octant, was used in all passes as it aids in declustering the estimate. This means that it helps to avoid over-influence of individual drill holes or sectors being overly informed, avoiding the use of samples that clustered together and thereby redundant. For inverse distance techniques, a search strategy that accounts for clustering will yield improvements over taking all samples.

Figure 17.28: Three-dimensional view of block model, Kremnica Deposit with estimated Au grade ≥ 0.75 g/t.



17.13 RESOURCE CLASSIFICATION

The resources are classified on a block-by-block basis, using the defined estimation passes by assigning a flag which is stored at each pass indicating whether the block was estimated on the first, second, or third pass. This indicates the quantity and quality of information used to estimate each block.

The classification of resources derived for the Kremnica resources is complaint with NI 43-101 and CIM classification systems. As stated elsewhere in this report, the resource model proposed is the ID^3 model, and the resource classification discussed here refers exclusively to this model.

Figure 17.28 illustrates a three dimensional view of the Kremnica block model. Vein material is a solid purple line while the andesite / stockwork boundary is in green solid line. Also, voids are in dashed grey lines..

Figure 17.29 and 17.30 are sample plan and cross – sections, respectively, through the block model and the geology solids units.

Figure 17.29: Plan View of Block Model, Kremnica Deposit with Au grade ≥ 0.75 g/t and geologic units with the vein in solid purple, andesite / stockwork boundary in green and voids in dashed grey.

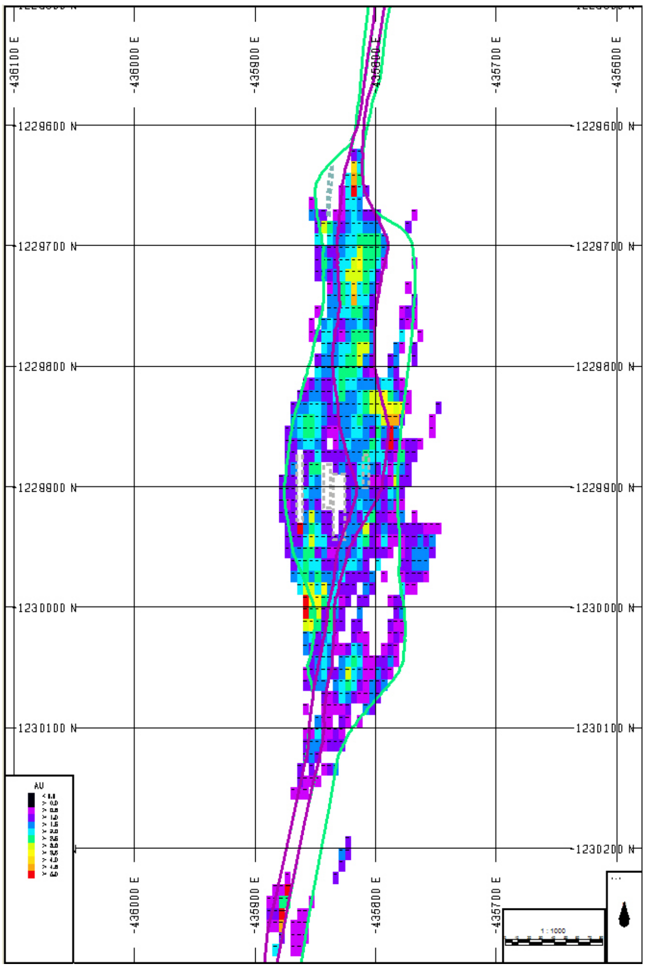
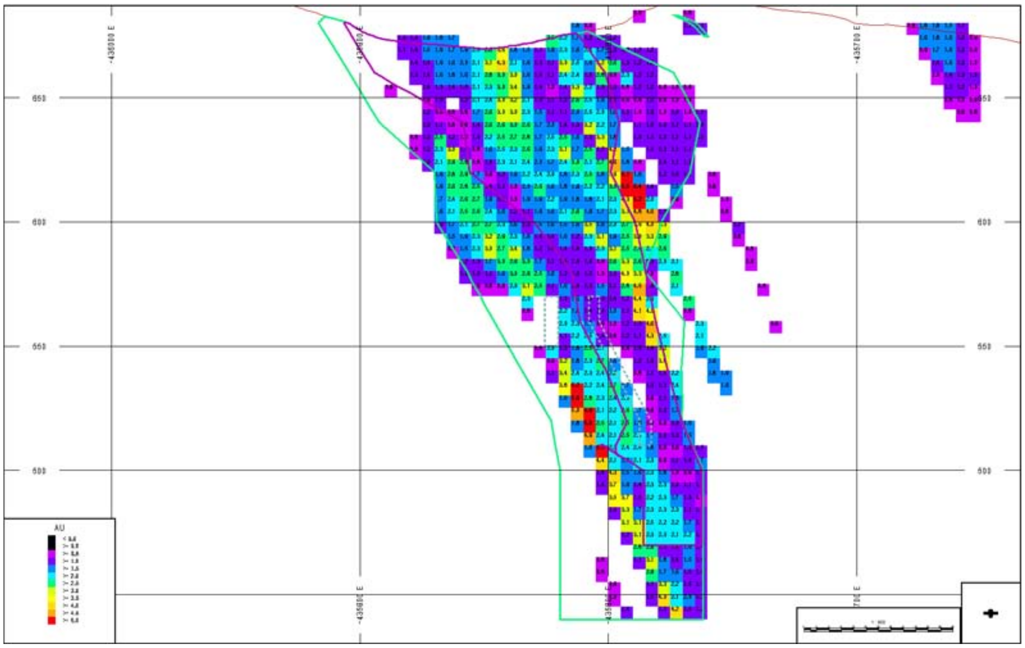


Figure 17.30: Cross-sectional view of block model with estimated Au grade ≥ 0.75 g/t and geologic units with the vein in solid purple, andesite / stockwork boundary in green and voids in dashed grey.



17.14 STATISTICAL AND GRAPHICAL VALIDATION OF THE BLOCK MODEL

Several validation tasks were performed on the resource model. The author carried out a statistical comparison of the block grades and the composites. The results of this analysis are listed below in Tables 17.10 and 17.11 for minimum grades of 0.01 g/t for both Au and Ag. The mean block grade is lower (by approximately 30%) than the mean composite grade. In the author's opinion, this indicates that there is a negative bias in the ID^3 grade estimation and this suggests that the model incorporates internal dilution and that the model may be considered diluted.

Table 17.9: Au Composite vs. Block Model Statistics by Cut-off Grade.

	COMPOSITES					BLOCK MODEL				
Cutoff	**Weight**	**%**	**AU**	**SD**	**CV**	**Weight**	**%**	**AU**	**SD**	**CV**
0.0	4649	100.00%	0.90	1.59	1.77	81,195,288	100.00%	0.62	0.80	1.29
0.5	1989	42.78%	1.88	2.06	1.09	31,206,626	38.43%	1.34	0.88	0.66
1.0	1308	28.14%	2.49	2.31	0.93	17,355,882	21.38%	1.84	0.91	0.49
1.5	880	18.93%	3.10	2.61	0.84	9,805,797	12.08%	2.32	0.95	0.41
2.0	595	12.80%	3.75	2.96	0.79	5,277,711	6.50%	2.83	1.05	0.37
2.5	409	8.80%	4.44	3.35	0.75	2,742,522	3.38%	3.39	1.21	0.36
3	293	6.30%	5.11	3.75	0.73	1,445,064	1.78%	3.99	1.42	0.36
3.5	198	4.26%	6.01	4.28	0.71	757,750	0.93%	4.68	1.67	0.36
4.0	142	3.05%	6.91	4.76	0.69	444,110	0.55%	5.37	1.90	0.35
4.5	100	2.15%	8.03	5.30	0.66	296,696	0.37%	5.94	2.10	0.35
5.0	75	1.61%	9.12	5.71	0.63	204,231	0.25%	6.49	2.33	0.36
5.5	60	1.29%	10.08	6.02	0.60	144,664	0.18%	7.01	2.59	0.37
6.0	53	1.14%	10.67	6.18	0.58	88,211	0.11%	7.83	3.04	0.39
6.5	45	0.97%	11.45	6.39	0.56	55,518	0.07%	8.76	3.51	0.40
7.0	36	0.77%	12.64	6.64	0.53	39,306	0.05%	9.63	3.85	0.40
7.5	31	0.67%	13.51	6.77	0.50	26,221	0.03%	10.83	4.23	0.39
8.0	28	0.60%	14.12	6.86	0.49	18,278	0.02%	12.19	4.43	0.36
8.5	26	0.56%	14.56	6.92	0.48	13,813	0.02%	13.49	4.36	0.32
9.0	23	0.49%	15.32	7.01	0.46	11,553	0.01%	14.41	4.18	0.29
9.5	21	0.45%	15.90	7.07	0.44	9,837	0.01%	15.33	3.85	0.25
10.0	20	0.43%	16.21	7.11	0.44	8,702	0.01%	16.04	3.52	0.22
10.5	19	0.41%	16.53	7.16	0.43	8,127	0.01%	16.45	3.28	0.20
11.0	17	0.37%	17.21	7.28	0.42	6,950	0.01%	17.43	2.42	0.14
11.5	14	0.30%	18.53	7.39	0.40	6,950	0.01%	17.43	2.42	0.14
12.0	12	0.26%	19.63	7.43	0.38	6,950	0.01%	17.43	2.42	0.14
12.5	10	0.22%	21.09	7.30	0.35	6,950	0.01%	17.43	2.42	0.14
13.0	10	0.22%	21.09	7.30	0.35	6,365	0.01%	17.88	2.02	0.11
13.5	9	0.19%	21.94	7.20	0.33	6,365	0.01%	17.88	2.02	0.11
14.0	7	0.15%	24.27	6.37	0.26	5,780	0.01%	18.28	1.66	0.09
14.5	7	0.15%	24.27	6.37	0.26	5,780	0.01%	18.28	1.66	0.09
15.0	6	0.13%	25.87	5.22	0.20	5,187	0.01%	18.68	1.23	0.07

Table 17.10: Ag Composite vs. Block Model Statistics by Cut-off Grade.

	COMPOSITES					BLOCK MODEL				
Cutoff	Weight	%	AG	SD	CV	Weight	%	AG	SD	CV
0	4877	100.00%	7.27	11.12	1.53	88,565,976	100.00%	4.97	6.25	1.26
1	3735	76.58%	9.38	11.94	1.27	66,477,620	75.06%	6.45	6.58	1.02
2	3060	62.74%	11.13	12.53	1.13	51,304,096	57.93%	7.93	6.82	0.86
3	2620	53.72%	12.59	12.99	1.03	41,789,628	47.18%	9.17	6.98	0.76
4	2280	46.75%	13.94	13.40	0.96	34,277,008	38.70%	10.42	7.13	0.68
5	2018	41.38%	15.17	13.78	0.91	28,198,468	31.84%	11.71	7.24	0.62
6	1766	36.21%	16.56	14.19	0.86	23,752,266	26.82%	12.87	7.32	0.57
7	1572	32.23%	17.80	14.57	0.82	20,397,208	23.03%	13.92	7.39	0.53
8	1421	29.14%	18.90	14.91	0.79	17,599,034	19.87%	14.95	7.46	0.50
9	1267	25.98%	20.16	15.32	0.76	14,993,025	16.93%	16.07	7.53	0.47
10	1147	23.52%	21.27	15.69	0.74	12,810,620	14.46%	17.19	7.60	0.44
11	1026	21.04%	22.54	16.12	0.72	11,106,158	12.54%	18.22	7.65	0.42
12	925	18.97%	23.75	16.54	0.70	9,657,375	10.90%	19.23	7.71	0.40
13	839	17.20%	24.90	16.95	0.68	8,414,200	9.50%	20.23	7.78	0.38
14	745	15.28%	26.35	17.46	0.66	7,369,866	8.32%	21.19	7.86	0.37
15	673	13.80%	27.61	17.91	0.65	6,492,216	7.33%	22.09	7.95	0.36
16	607	12.45%	28.93	18.38	0.64	5,680,560	6.41%	23.04	8.07	0.35
17	562	11.52%	29.93	18.75	0.63	4,986,946	5.63%	23.95	8.21	0.34
18	522	10.70%	30.89	19.12	0.62	4,285,088	4.84%	25.00	8.39	0.34
19	466	9.56%	32.37	19.72	0.61	3,692,989	4.17%	26.05	8.59	0.33
20	429	8.80%	33.49	20.17	0.60	3,175,734	3.59%	27.12	8.81	0.32
21	382	7.83%	35.10	20.82	0.59	2,747,706	3.10%	28.16	9.04	0.32
22	345	7.07%	36.56	21.40	0.59	2,384,750	2.69%	29.17	9.29	0.32
23	314	6.44%	37.95	21.95	0.58	2,049,107	2.31%	30.27	9.59	0.32
24	289	5.93%	39.21	22.44	0.57	1,762,057	1.99%	31.37	9.91	0.32
25	268	5.50%	40.36	22.91	0.57	1,476,722	1.67%	32.71	10.31	0.32
26	248	5.09%	41.56	23.41	0.56	1,236,052	1.40%	34.11	10.71	0.31
27	225	4.61%	43.10	24.05	0.56	1,069,614	1.21%	35.30	11.05	0.31
28	209	4.29%	44.30	24.55	0.55	933,474	1.05%	36.44	11.39	0.31
29	193	3.96%	45.61	25.10	0.55	808,600	0.91%	37.68	11.76	0.31
30	176	3.61%	47.16	25.77	0.55	711,865	0.80%	38.79	12.11	0.31

Figure 17.31: Block Model Histogram and Basic Statistics for Au.

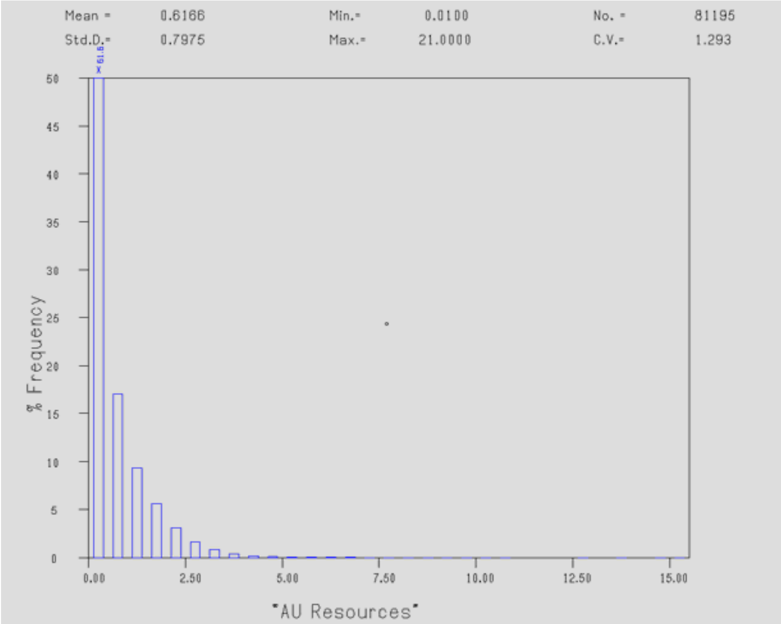
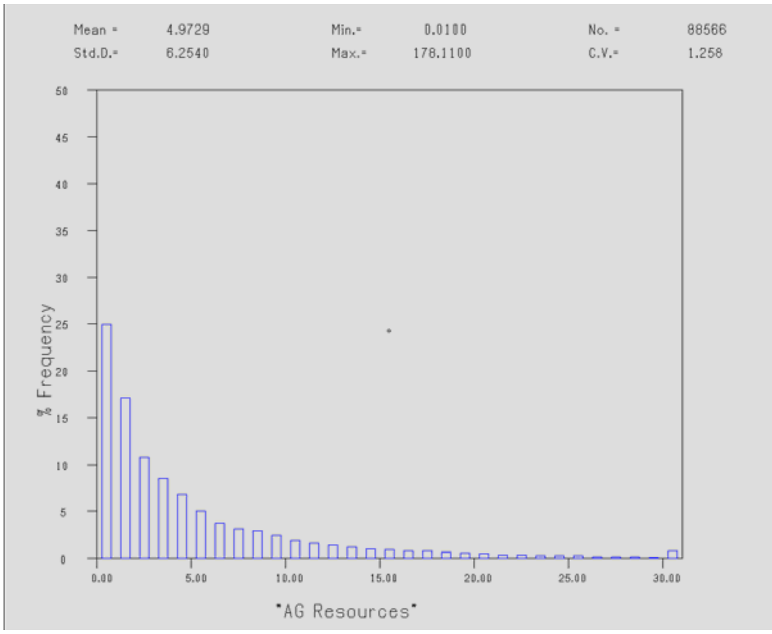


Figure 17.32: Block Model Histogram and Basic Statistics for Ag.



In addition to the above, a graphical validation was done, Figures 17.31 and 17.32, on the block model where cross sections and plans were used to check the block model on the computer screen, showing the block grades, the composite data and the topographic surface. No evidence of any block being wrongly estimated was found: it appears that every block grade can be explained as a function of the surrounding composites, the search strategy employed for modeling and the estimation plan applied.

17.15 RESOURCE ESTIMATES

The estimated measured and indicated resource for the Šturec deposit, using a 0.5 g/t AuEq cut-off, is 23.642 million tonnes at an average grade of 1.37 g/t gold and 11.36 g/t silver. There is an additional inferred resource of 10.592 million tonnes at an average grade of 1.01 g/t gold and 6.27 g/t silver.

Table 17.12 through 17.14 shows the measured and indicated resources at grades ranging from 0.5 g Au/t to 5.0 g Au/t. Table 17.15 shows the inferred resources.

Table 17.11: Measured Resources[5]

Cut-off Grade g/t AuEq	Tonnes	Au g/t	Ag g/t	AuEq g/t	Au Ounces	Ag Ounces	Equivalent
0.50*	**8,228,885**	**1.61**	**13.32**	**1.81**	**425,157**	**3,525,070**	**478,070**
0.75	7,293,229	1.75	14.24	1.96	409,408	3,339,277	459,353
1.0	6,445,921	1.88	15.11	2.10	388,785	3,131,629	435,829
1.5	4,536,192	2.21	17.16	2.46	321,582	2,501,925	359,210
2.0	2,814,223	2.62	19.61	2.91	236,785	1,774,124	263,386
2.5	1,631,124	3.08	22.10	3.41	161,364	1,159,020	178,722
3.0	896,814	3.60	24.64	3.97	103,800	710,568	114,468
3.5	495,901	4.17	26.83	4.57	66,421	427,815	72,830
4.0	273,662	4.81	29.57	5.26	42,338	260,197	46,245
4.5	157,944	5.55	31.67	6.02	28,173	160,821	30,585
5.0	97,307	6.33	33.99	6.84	19,813	106,338	21,405

Notes: **1.** *Base Case. **2.** AuEq is based upon 66.7:1 Au:Ag. . **3.** The above resources have been prepared in accordance with CIM standards.

[5] All the resources within this section are not rounded whereas in the Section 1, Table 1 of the technical report (May 2006) are rounded.

Table 17.12: Indicated Resources

Cut-off Grade g/t AuEq	Tonnes	Au g/t	Ag g/t	AuEq g/t	Au Ounces	Ag Ounces	Equivalent
0.50*	**15,413,258**	**1.24**	**10.31**	**1.40**	**615,472**	**5,109,110**	**691,786**
0.75	11,514,240	1.48	11.86	1.66	549,365	4,388,630	614,889
1.0	8,708,008	1.72	13.37	1.92	481,268	3,741,794	537,262
1.5	5,029,930	2.19	16.16	2.43	353,835	2,612,852	392,971
2.0	2,988,578	2.63	18.72	2.91	252,704	1,799,004	279,608
2.5	1,619,599	3.19	20.40	3.49	165,848	1,062,310	181,781
3.0	902,639	3.78	21.18	4.10	109,756	614,511	118,955
3.5	514,398	4.44	21.43	4.77	73,480	354,416	78,805
4.0	313,903	5.10	22.10	5.43	51,440	223,079	54,791
4.5	196,869	5.79	23.66	6.15	36,660	149,737	38,907
5.0	146,616	6.25	25.17	6.63	29,471	118,628	31,248

Notes: **1.** *Base Case. **2.** AuEq is based upon 66.7:1 Au:Ag. **3.** The above resources have been prepared in accordance with CIM standards.

Table 17.13: Measured and Indicated Resources

Cut-off Grade g/t AuEq	Tonnes	Au g/t	Ag g/t	AuEq g/t	Au Ounces	Ag Ounces	Equivalent
0.50*	**23,642,143**	**1.37**	**11.36**	**1.54**	**1,040,629**	**8,634,180**	**1,169,856**
0.75	18,807,469	1.59	12.78	1.78	959,711	7,727,907	1,074,242
1.0	15,153,929	1.79	14.11	2.00	870,053	6,873,423	973,091
1.5	9,566,122	2.20	16.63	2.45	675,418	5,114,777	752,180
2.0	5,802,801	2.62	19.15	2.91	489,489	3,573,128	542,994
2.5	3,250,723	3.13	21.25	3.45	327,212	2,221,330	360,504
3.0	1,799,453	3.69	22.90	4.03	213,556	1,325,079	233,424
3.5	1,010,299	4.31	24.08	4.67	139,901	782,232	151,636
4.0	587,565	4.96	25.58	5.35	93,778	483,276	101,035
4.5	354,813	5.68	27.22	6.09	64,833	310,558	69,492
5.0	243,923	6.28	28.69	6.71	49,284	224,966	52,653

Notes: **1.** *Base Case. **2.** AuEq is based upon 66.7:1 Au:Ag. **3**. The above resources have been prepared in accordance with CIM standards.

Table 17.14: Inferred Resources[6]

Cut-off Grade g/t Au	Tonnes	Au g/t	Ag g/t	AuEq g/t	Au Ounces	Ag Ounces	Equivalent
0.50*	**10,591,781**	**1.01**	**6.27**	**1.11**	**344,961**	**2,135,151**	**376,972**
0.75	6,397,808	1.32	7.42	1.44	272,134	1,525,227	295,172
1.0	4,360,442	1.58	8.20	1.71	221,783	1,149,993	239,027
1.5	2,133,902	2.07	9.90	2.22	141,810	679,344	152,032
2.0	1,072,772	2.54	10.62	2.70	87,744	366,324	93,228
2.5	542,745	3.02	10.93	3.18	52,663	190,795	55,525
3.0	261,474	3.51	12.06	3.69	29,499	101,400	31,020
3.5	85,017	4.44	13.88	4.64	12,122	37,936	12,691
4.0	39,601	5.55	15.69	5.79	7,068	19,977	7,365
4.5	31,453	5.96	15.92	6.20	6,024	16,099	6,265
5.0	27,332	6.16	17.17	6.42	5,417	15,084	5,642

Notes: **1.** *Base Case. **2.** AuEq is based upon 66.7:1 Au:Ag. **3**. The above resources have been prepared in accordance with CIM standards.

17.16 OPEN PIT

A variety of scenarios were run based on a number of production cases that evaluated different revenue and cost scenarios, depth of pit, pit wall slope parameters, and grade cut-off thresholds in an effort to determine the optimum pit along with options for staged interim pits. Apart from the cost data, the source data for the pit optimization process are the calculated grades for gold and silver along with the subsequent calculation for gold equivalent (AuEq), and the specific gravity (SG) values for each block.

Once the optimal pits were chosen, they were used as templates to design the pit layout and haul road. The design parameters for the pit were to utilize a double bench, 10-meter safety berm, 70° inter-ramp wall slopes, and 10% grade for the haul road. In addition, to accommodate geotechnical constraints, an overall pit slope of 50° was used with the exception of those sections of the wall that transect the vein (shown as a purple solid in Figures 17.33 through 17.35). The pit sectors transected by the vein required a reduced pit slope of 45° from azimuth 0° through 20° in the north sector and from 180° through 200° in the south sector. The green lines show the reduced slope sectors.

For scheduling, the approach was to create a Phase 1 starter pit (Figure 17.33) to extract the relatively higher grade resources in the initial years, followed by a Phase 2 ultimate pit in Figure 17.34. Figure 17.35 shows the Phase 1 pit solid inside the final combined Phase 1 and Phase 2 pit shell to illustrate the material to be extracted in the initial years. Figures 17.36 and 17.37 illustrate the Phase 1 pit solid (light blue) to be extracted followed by the development of the Phase 2 pit, which is represented by the remainder within the pit shell (dark blue) in Figures 17.38 and 17.39.

[6] Due to the uncertainty that may be attached to an inferred mineral resource, it cannot be assumed that all or any part of an inferred mineral resource will be upgraded to an indicated or measured resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred resources must be excluded from estimates from feasibility or other economic studies.

Figure 17.33: Phase 1 Starter Pit with Haul Road Intersected by Vein (purple solid) with Slope Sectors in Green

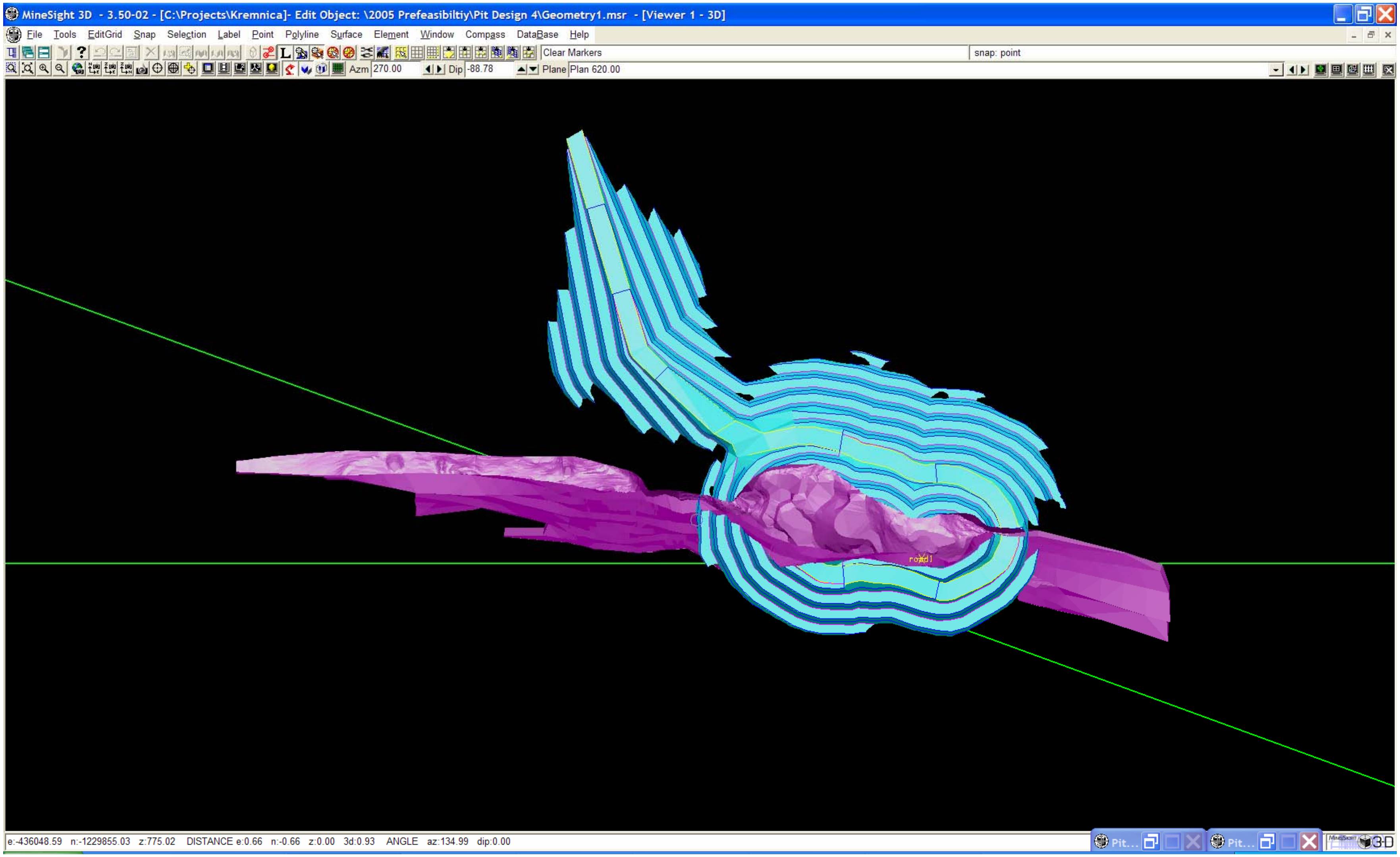


Figure 17.34: Phase 2 Ultimate Pit with Haul Road Intersected by Vein (purple solid) with Slope Sectors in Green

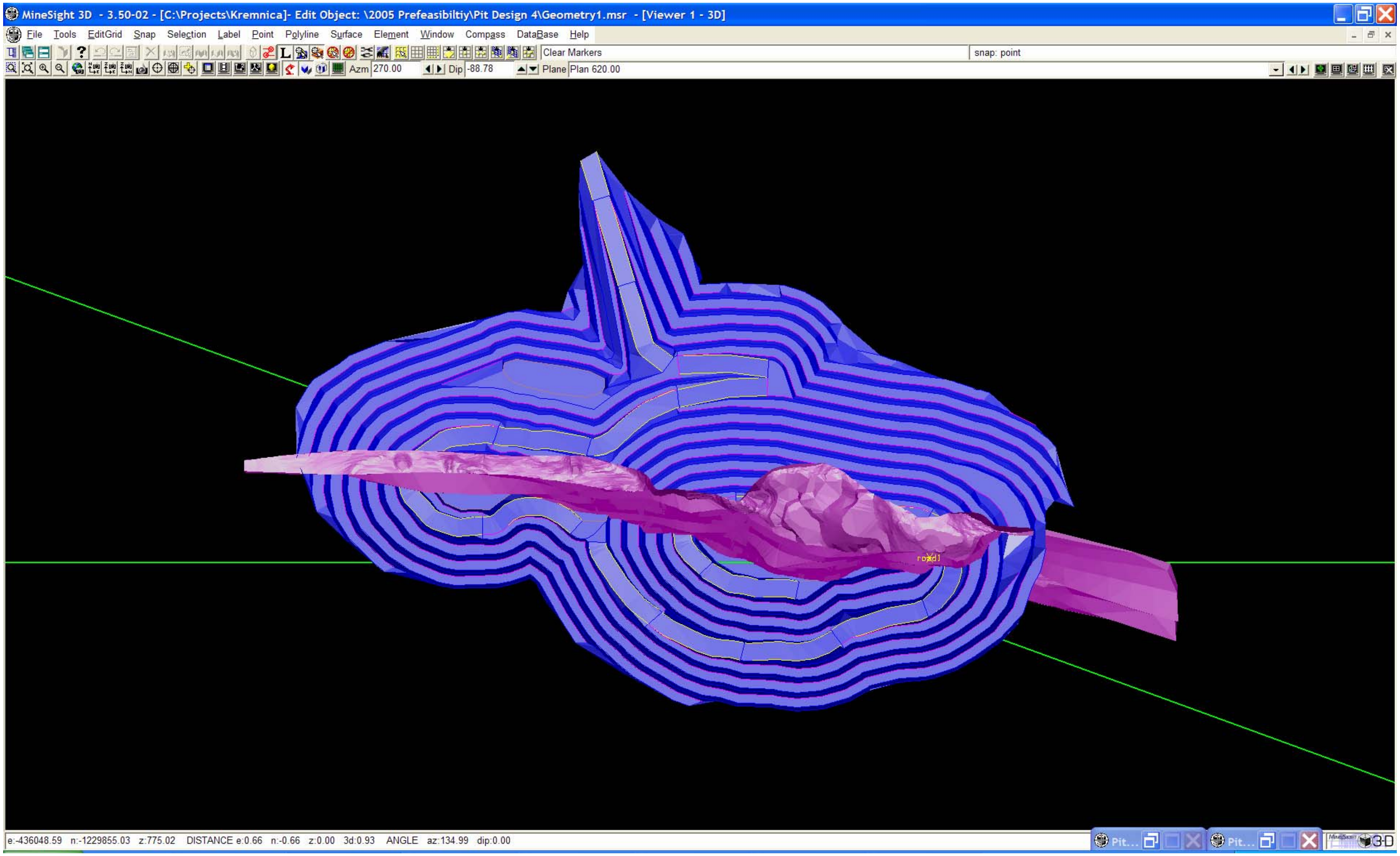


Figure 17.35: Combined Phase 1 and Phase 2 Pit

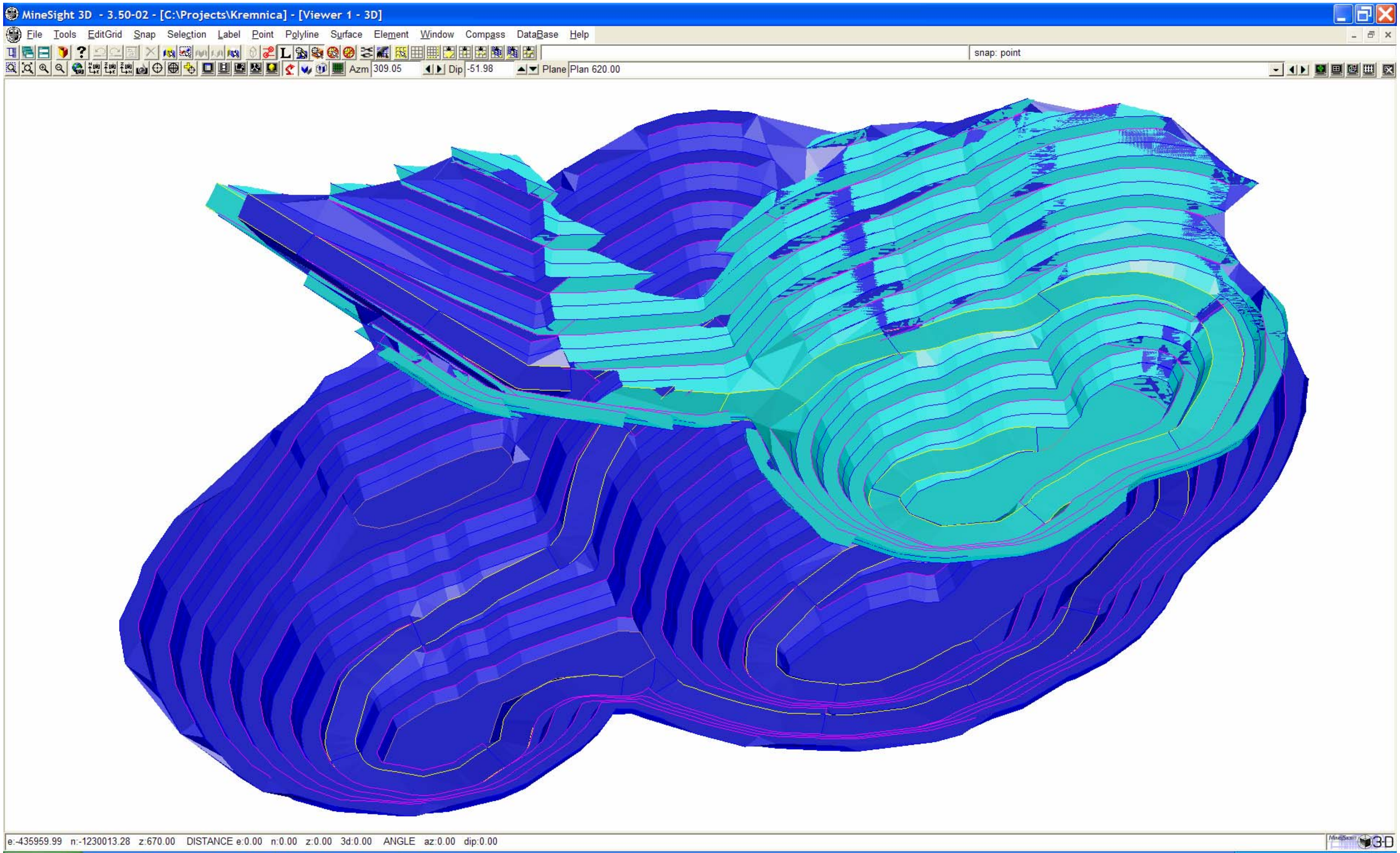


Figure 17.36: Plan View of Phase 1 Pit Intersected with Topographic Solid Surface

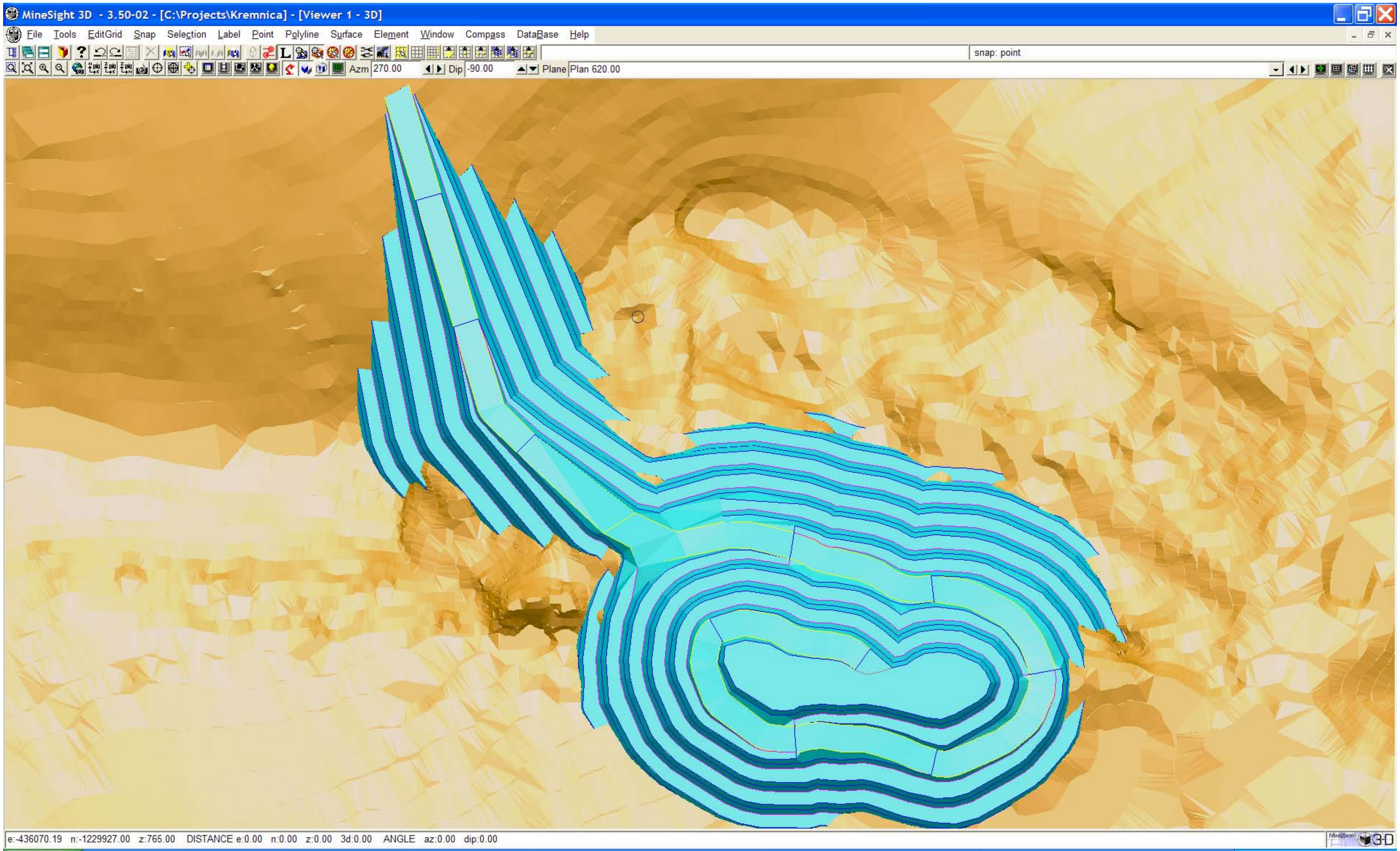


Figure 17.37: Perspective View of Phase 1 Pit Intersected with Topographic Solid Surface

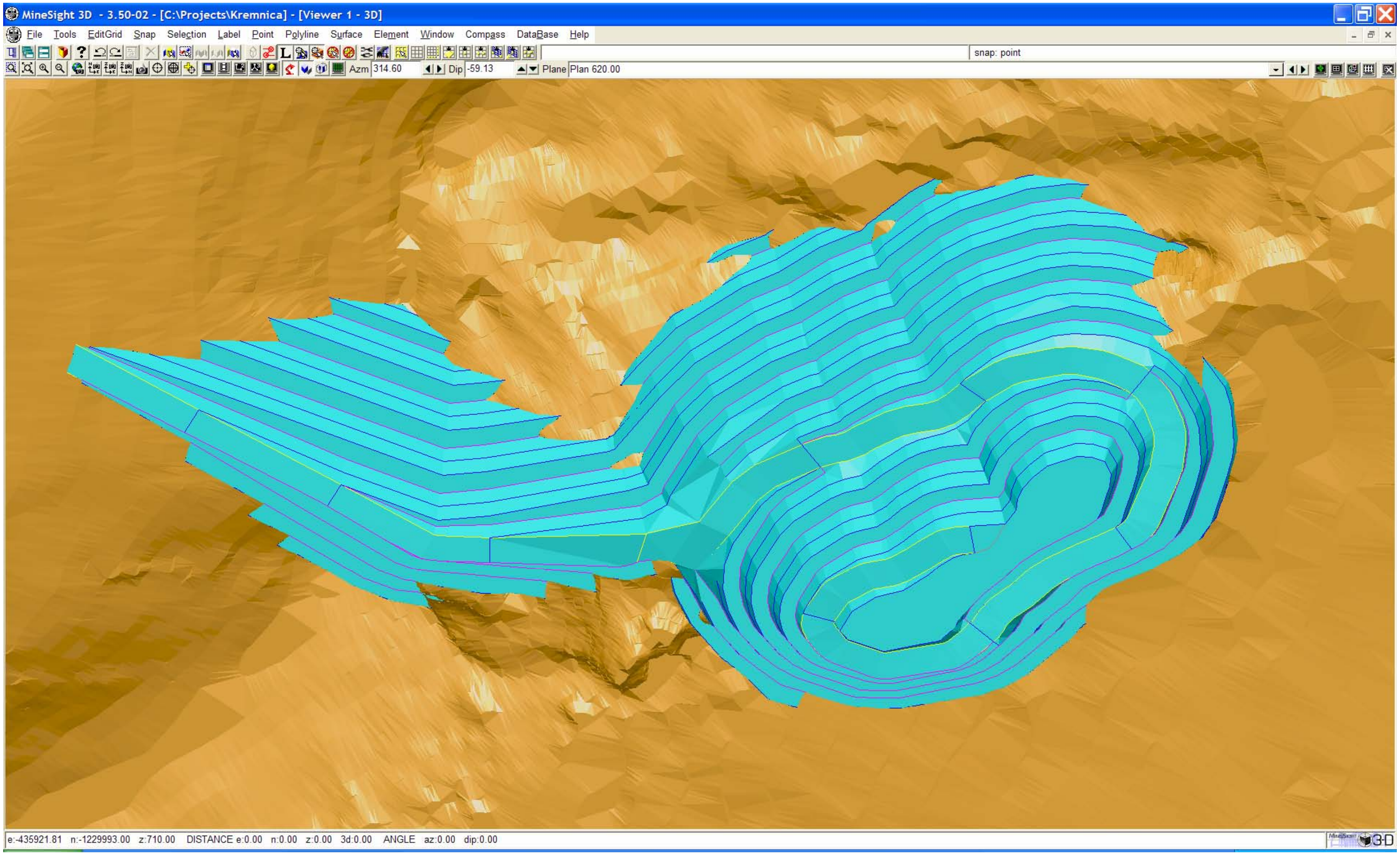


Figure 17.38: Plan View of Phase 2 Pit Intersected with Topographic Solid Surface

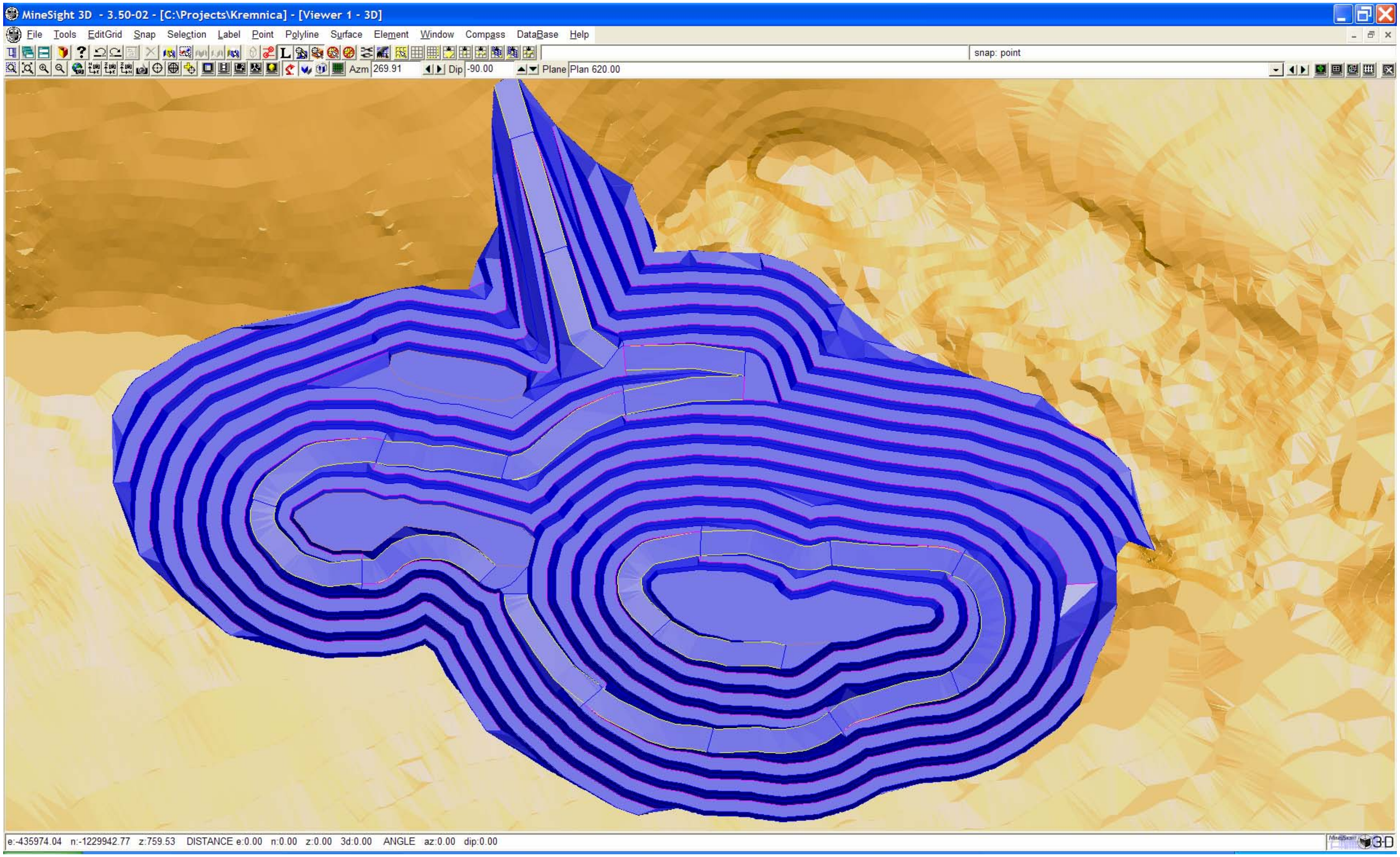
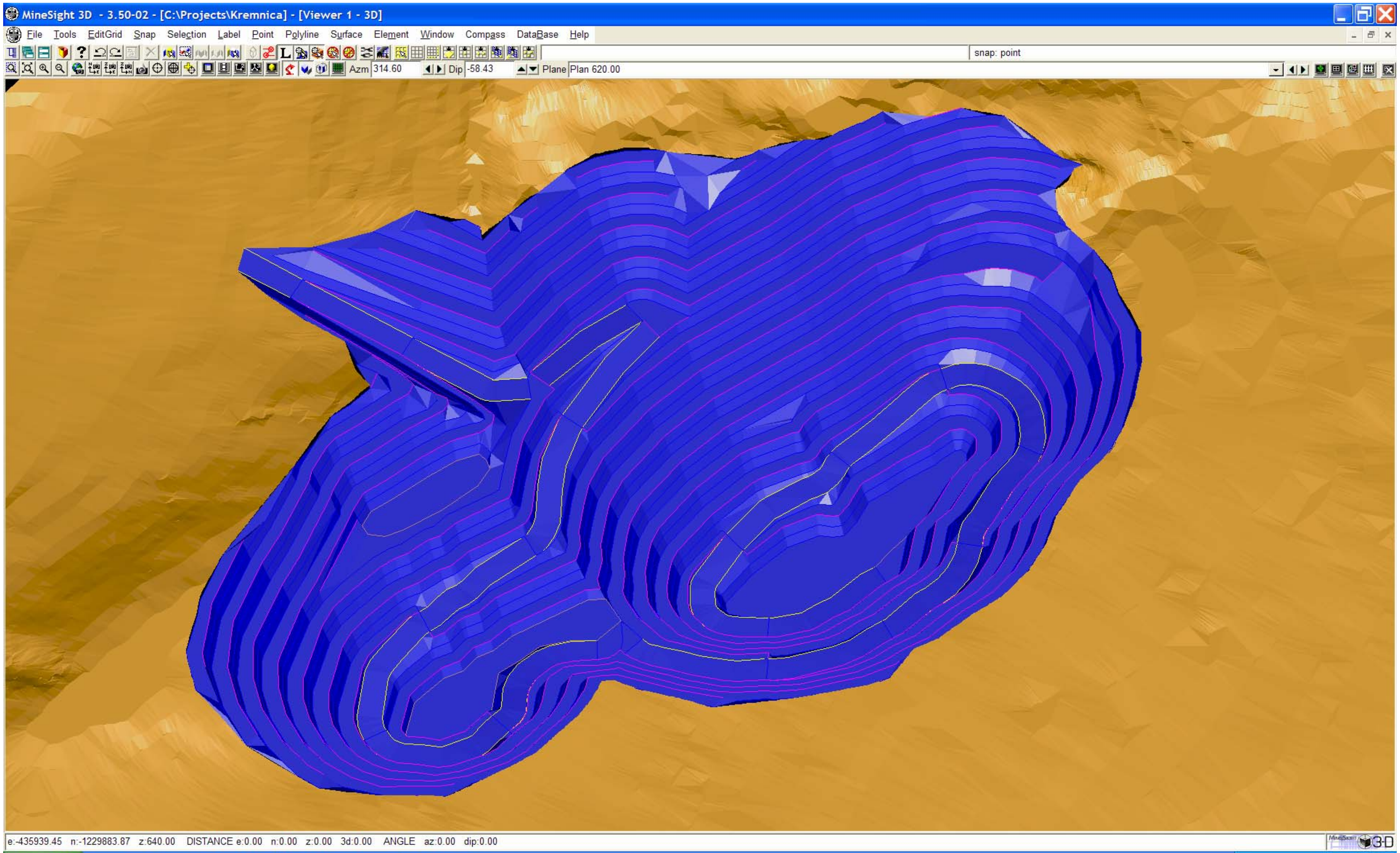


Figure 17.39: Perspective View of Phase 2 Pit Intersected with Topographic Solid Surface

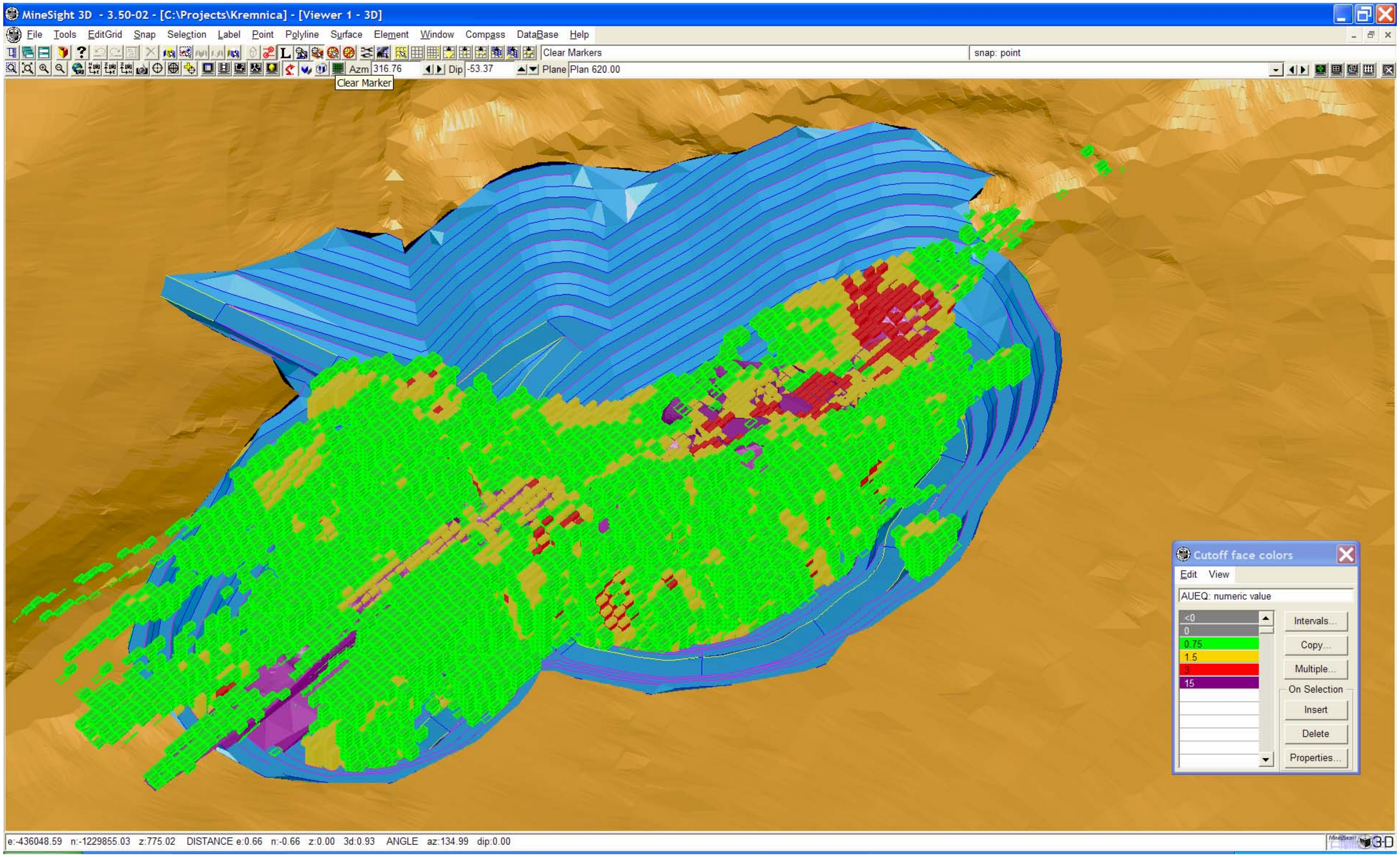


The resulting resources are listed for the Phase 1, Phase 2, and combined pits in Table 1.1, Section 1.0. Note that these resources are based on using a cut-off grade of 0.75 gAuEq/t.

In addition, the option of extracting material into a low-grade stockpile was included in the analysis. This entailed calculating the differential between the material that is below 0.75 gAuEq/t but greater than 0.5 gAuEq/t. Table 1.1 lists the amount and grade of this differential, with the premise being that although this material is not profitable under the current revenue-cost scenario, it has already been extracted. Therefore, the cost of mining may be removed from the costs incurred, and only the cost of milling may now be considered for determining the profitability of this marginal material.

Figure 17.40 illustrates the combined Phase 1 and Phase 2 in-pit resources, displaying only those blocks that exceed the 0.75 gAuEq/t cut-off.

Figure 17.40: Perspective View of Phase 1 and Phase 2 Combined Pit with AuEq Blocks > 0.75 g/t with Topography



17.17 MINING PLAN

17.17.1 Introduction

The revised mineral estimate for the Šturec deposit has formed the basis for the mining plan described in this section. The overall plan has taken into account a number of key aspects to ensure that mine development has a minimal effect on the landscape and the people living in Kremnica and the surrounding villages. The plan has been based upon the following principles and design:

- minimize disturbance in the area of the mine, the town of Kremnica, and the surrounding villages
- use visual barriers and vegetation to provide a screen and minimize the visual effect of the project in the area
- eliminate a waste rock disposal area and instead use that rock as material for building the tailings storage facility dam
- maximize the recycling of water used during ore processing and minimize the use of fresh water obtained from the local watershed
- ensure that all mine roads are placed in locations where there is minimal visual effect on the landscape in the area
- construct a mine access road so that trucks hauling equipment and supplies do not pass through the town of Kremnica enroute to the mine; minimize disturbance along the highway to the mine access road
- locate the tailing storage facility on a site that minimizes the visual effect on the landscape; where feasible, reclaim the area as mining proceeds
- conduct drilling, blasting, and ore and waste haulage 6 d/wk, 16 h/day (except statutory holidays); the process plant will continue to operate 7 d/wk, 24 h/d, fed by ore from a stockpile located adjacent to the plant

17.17.2 Open Pit Optimization

17.17.2.1 Pit Optimization Study

A pit optimization study was completed by running a variety of scenarios, shown on Table 17.16, on a number of production cases to evaluate varying revenue and cost scenarios, pit depths, pit wall slope parameters, grade cut-off thresholds and strip ratios to determine the optimum pit along with options for staged interim pits. The methodology chosen was to produce the basis for an initial high grade starter pit (if possible), an intermediate pit and a final ultimate pit. These pit scenarios were to be produced whilst minimizing strip ratio. In order to accomplish this, a variety of pits were produced at ever decreasing gold prices in order to derive the smaller tonnage, high grade scenarios as shown. Tables 17.17, 17.18, and 17.19 list three pit scenarios that resulted in the optimal grade and lowest strip ratio for the desired tonnage extraction amounts. The models were based on increasing prices for gold and silver.

Table 17.15: Optimization Scenarios

	Mill Recovery	Market Price Au per ounce	Marketing+Freight per gram	Op cost Ore	Op. Cost Waste	Op. Cost Processing	G&A Ore
Scenario 1	90.00%	*$385.00*	*$0.57*	1.34	1.28	$7.39	$0.82
Scenario 2	90.00%	*$385.00*	*$0.57*	1.34	1.28	$7.39	$0.82
Scenario 3	90.00%	*$350.00*	*$0.57*	1.7	1.48	$3.50	$0.82

Note: Scenario 1 and 2 use the same criteria for evaluating two different depths of the open pit.

Apart from the cost data, the source data for the pit optimization process are the calculated grades for gold and silver, along with the subsequent calculation for gold equivalent (AuEq), and the specific gravity (SG) values for each block.

Table 17.16: Pit Optimization Results – Scenario 1

Scenario 1	Cut-off	Tonnes	AuEq	Au	Ag	Tonnes Waste	Strip Ratio
	0.00	1,483,749	2.46	2.17	18.88		
	0.50	1,468,091	2.48	2.19	19.02		
Measured	0.75	1,446,189	2.51	2.22	19.19		
	1.00	1,407,677	2.55	2.26	19.49		
	2.00	974,650	2.99	2.66	22.32		
	0.00	1,081,451	1.83	1.62	13.85		
	0.50	1,009,558	1.94	1.72	14.46		
Indicated	0.75	931,523	2.05	1.82	15.12		
	1.00	792,845	2.25	2.00	16.49		
	2.00	420,450	3.02	2.70	21.11		
	0.00	307,975	1.06	0.96	6.86		
	0.50	179,350	1.64	1.50	9.21		
Inferred	0.75	132,067	2.01	1.85	10.26		
	1.00	80,977	2.73	2.55	11.68		
	2.00	50,375	3.48	3.28	13.47		
	0.00	**2,873,175**	**2.07**	**1.84**	**15.70**	**248,903**	**0.09**
	0.50	**2,657,000**	**2.22**	**1.97**	**16.62**	**465,078**	**0.18**
TOTALS	**0.75**	**2,509,780**	**2.31**	**2.05**	**17.21**	**612,298**	**0.24**
	1.00	**2,281,499**	**2.45**	**2.18**	**18.17**	**840,579**	**0.37**
	2.00	**1,445,475**	**3.02**	**2.69**	**21.66**	**1,676,603**	**1.16**

Table 17.17: Pit Optimization Results – Scenario 2

Scenario 2	Cut-off	Tonnes	AuEq	Au	Ag	Tonnes Waste	Strip Ratio
Measured	0.00	6,096,727	1.90	1.69	14.21		
	0.50	5,744,667	2.00	1.78	14.76		
	0.75	5,395,119	2.09	1.86	15.28		
	1.00	4,983,185	2.19	1.95	15.85		
	2.00	2,419,702	2.92	2.62	19.97		
Indicated	0.00	4,804,140	1.38	1.23	10.12		
	0.50	4,036,887	1.57	1.40	11.27		
	0.75	3,364,127	1.76	1.58	12.37		
	1.00	2,671,113	1.99	1.79	13.72		
	2.00	992,566	3.01	2.73	18.52		
Inferred	0.00	3,329,991	0.50	0.44	4.09		
	0.50	1,112,726	1.13	1.02	7.39		
	0.75	627,355	1.53	1.40	8.89		
	1.00	402,085	1.90	1.75	10.17		
	2.00	112,131	3.18	2.99	12.92		
TOTALS	**0.00**	**14,230,858**	**1.40**	**1.24**	**10.46**	**1,536,875**	**0.11**
	0.50	**10,894,280**	**1.75**	**1.56**	**12.71**	**4,873,453**	**0.45**
	0.75	**9,386,602**	**1.94**	**1.73**	**13.81**	**6,381,131**	**0.68**
	1.00	**8,056,383**	**2.11**	**1.89**	**14.86**	**7,711,350**	**0.96**
	2.00	**3,524,400**	**2.95**	**2.66**	**19.34**	**12,243,333**	**3.47**

Table 17.18: Pit Optimization Results – Scenario 3

Scenario 3	Cut-off	Tonnes	AuEq	Au	Ag	Tonnes Waste	Strip Ratio
Measured	0.00	8,650,539	1.70	1.51	12.52		
	0.50	7,825,900	1.84	1.64	13.38		
	0.75	7,032,876	1.98	1.77	14.20		
	1.00	6,265,511	2.12	1.89	15.02		
	2.00	2,770,596	2.92	2.62	19.50		
Indicated	0.00	11,730,818	1.26	1.12	8.89		
	0.50	9,366,930	1.49	1.33	10.23		
	0.75	7,398,317	1.72	1.55	11.53		
	1.00	5,679,249	1.98	1.78	12.96		
	2.00	2,016,371	3.04	2.77	18.40		
Inferred	0.00	10,146,766	0.46	0.41	3.41		
	0.50	3,084,658	1.10	1.01	6.37		
	0.75	1,867,744	1.42	1.32	7.24		
	1.00	1,243,777	1.71	1.59	7.61		
	2.00	272,384	2.99	2.84	9.95		
TOTALS	**0.00**	**30,528,124**	**1.12**	**0.99**	**8.10**	**4,723,536**	**0.15**
	0.50	**20,277,488**	**1.57**	**1.40**	**10.86**	**14,974,172**	**0.74**
	0.75	**16,298,938**	**1.80**	**1.62**	**12.19**	**18,952,722**	**1.16**
	1.00	**13,188,536**	**2.02**	**1.82**	**13.43**	**22,063,124**	**1.67**
	2.00	**5,059,350**	**2.97**	**2.69**	**18.55**	**30,192,310**	**5.97**

For both ore and waste tonnage calculations, specific gravity was derived from the block model on a block-by-block basis, as described in the Technical Report (May 2006) posted on SEDAR on behalf of Tournigan. Initial overall variable pit slopes are determined to be 48° in the north and south pit walls where the vein material is predominant. A 53° pit slope was used in the east and west pit walls, where it was determined the material is more competent (andesite) from the footwall and hangingwall.

The 48° slopes, which include a 20° sweep along with a 5° transition zone to 53°, were necessary to accommodate geotechnical considerations associated with the vein running approximately 10° azimuth north through the deposit. The block size is equal to one half the bench height of 10 meters.

The block model published in the Technical Report (May 2006) formed the basis of the pit optimization study, along with the gridded topographic model defining the upper surface.

With regard to optimization algorithms, there are two options that may be used: the floating cone option and the more widely used and accepted Lerchs-Grossman option. For this exercise, the Lerchs-Grossman algorithm was chosen, because: (1) it performs many more passes or iterations than the floating cone option, (2) it has the capability of reassigning some marginal waste blocks in the pit where they may be profitable once extracted, and (3) it is more commonly used throughout the mining industry. In addition, the floating cone technique is known for both overestimating and underestimating results.

Based on the above criteria, the pit optimization process utilizes the block model to derive the gold equivalent grade and specific gravity. Other pertinent data relevant to calculating the ultimate pit are:

- metric tonne = 2204.6 pounds per tonne
- troy ounce = 31.1034 grams
- metal price of $385/oz
- recovery equals 90%.

Figure 17.41: Elevations by Colors for Figures 17.42 through 17.45

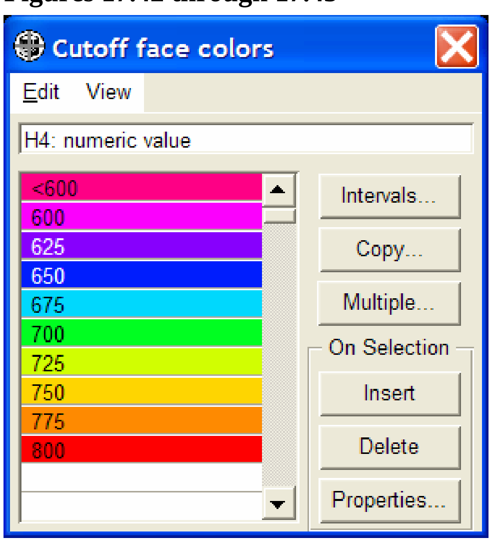


Figure 17.42: Scenario 1 Optimized Pit (Starter Pit)

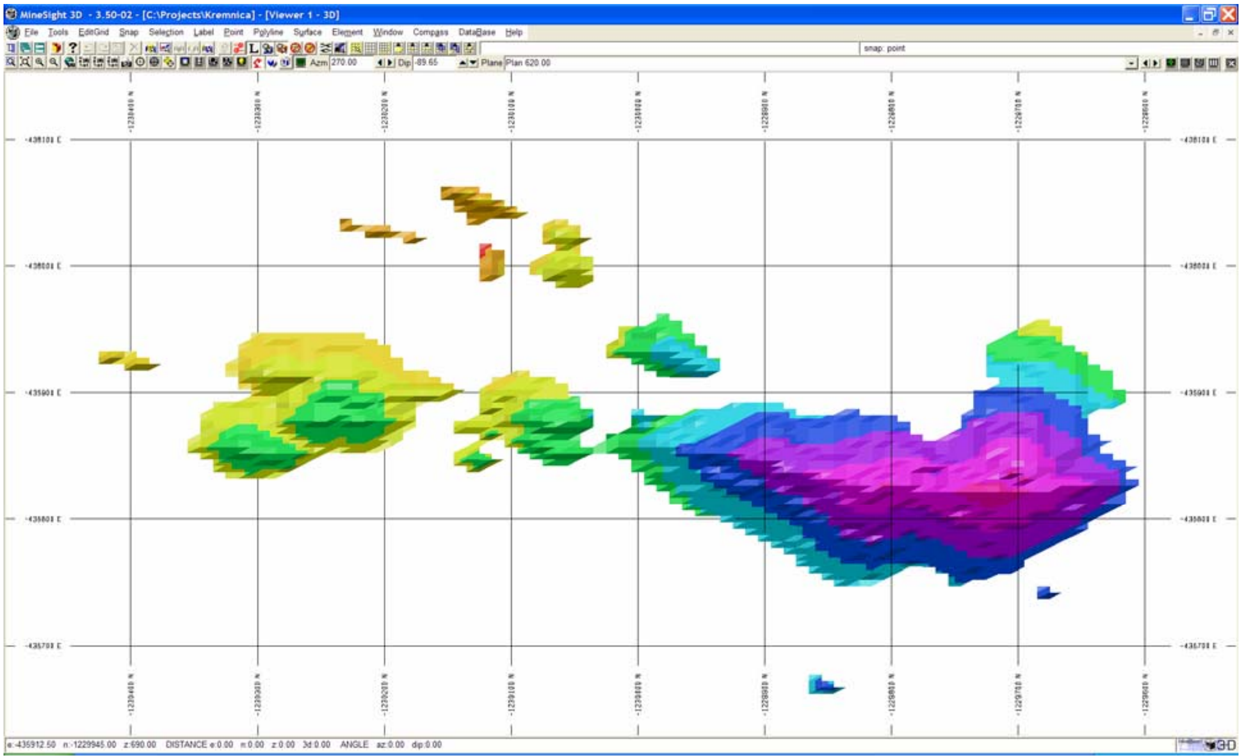


Figure 17.43: Scenario 2 Optimized Pit (Intermediate Pit)

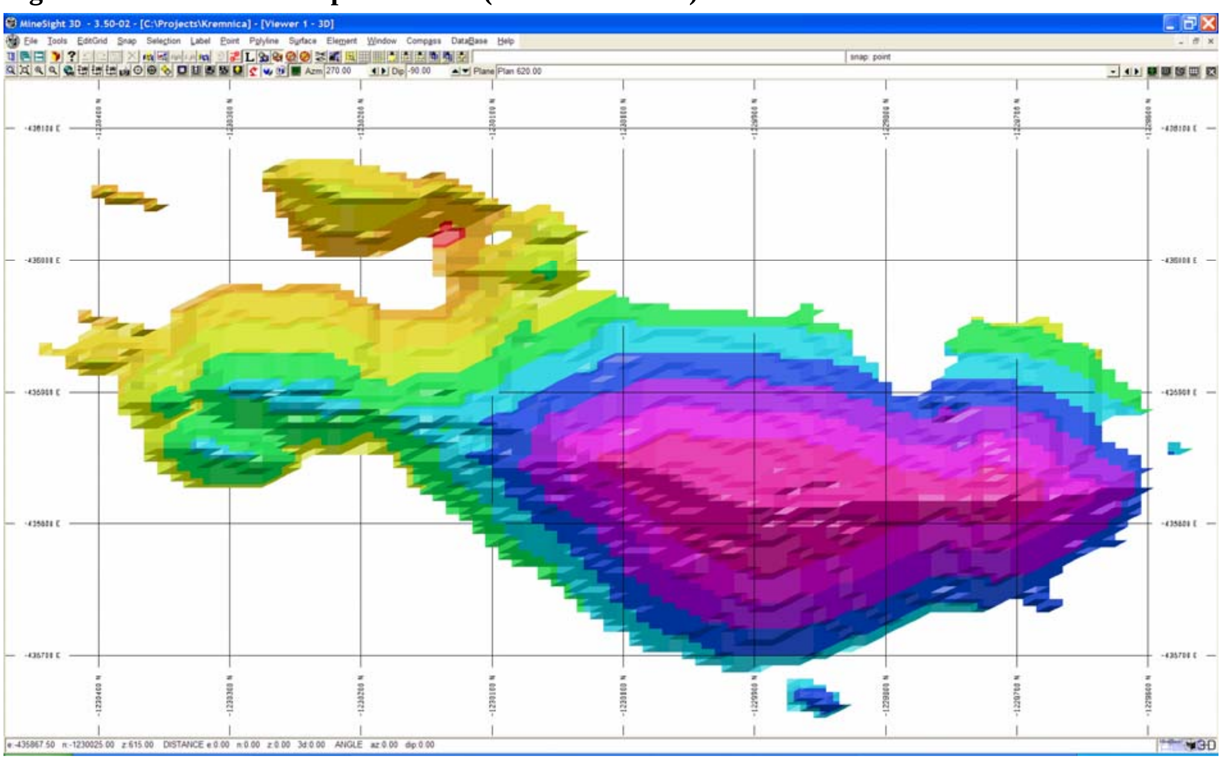


Figure 17.44: Scenario 3 Optimized Pit (Ultimate Pit)

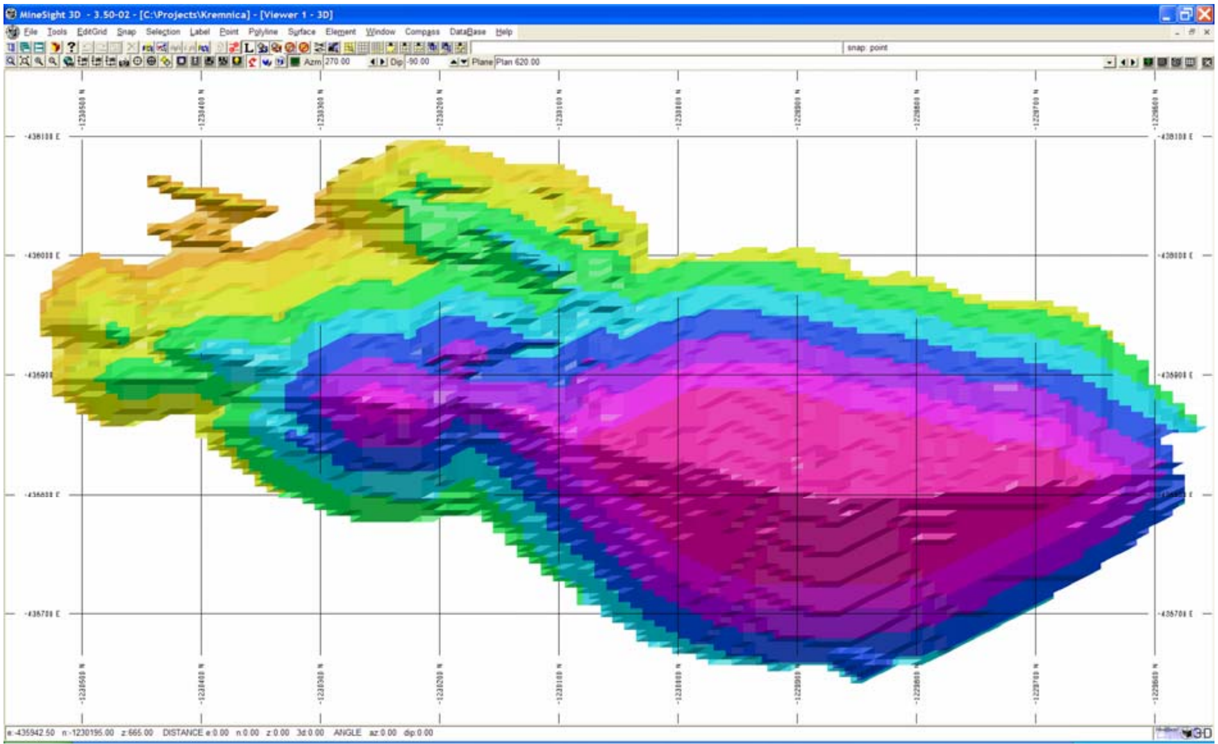
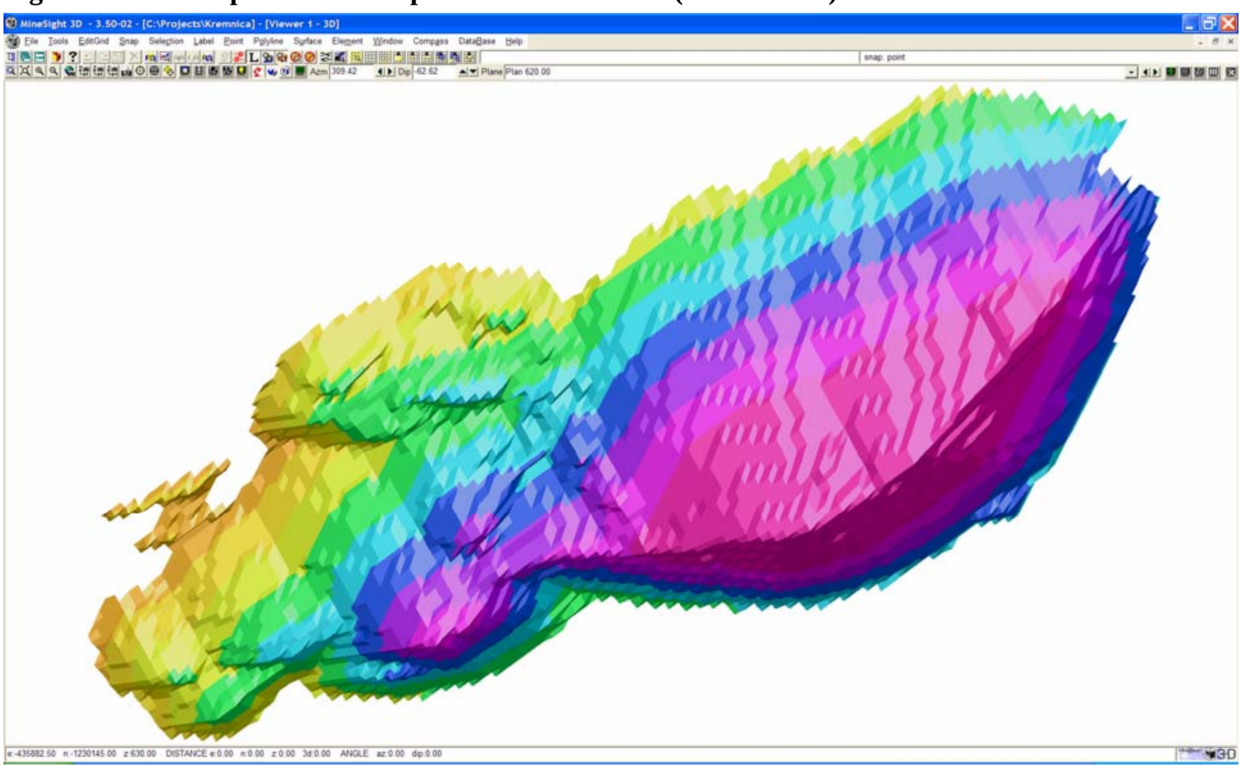


Figure 17.45: Perspective View Optimized Pit Scenario (Ultimate Pit)



17.17.3 Cut-off Values

The cut-off calculation for the resource estimate was based upon a gold price of $475/oz and an operating estimated cost of $10.50/tonne mined. The NSR for 1 gAuEq was calculated using a process plant recovery for gold of 93%, silver 66% and the following smelter recoveries and charges;

- a smelter recovery of 99.5% for gold, 98% for silver;
- smelter charge of $0.50 for gold, $0.25 for silver;
- a 99.99% smelter payment for gold and 99.50% for silver.

These smelter charges and criteria were those as quoted by Metalor Technologies SA.of Switzerland.

The NSR/gAuEq was calculated as $13.89 and thus the cut-off grade was determined as $10.50 divided by $13.89, which equates to 0.75 gAuEq/t.

A cut-off grade was established for low grade mineralization, initially mined as waste but could be stockpiled and processed after the high grade material. The basis for this calculation was the NSR/gAuEq of $13.89 and the estimated cost of $7.38/t for rehandling, processing and associated onsite costs. The low grade cut-off, $7.38/13.89, equates to 0.53 gAuEq/t and was rounded off to 0.50 gAuEq/t. Thus, the material stockpile grade ranged from 0.50 to 0.75 gAuEq/t.

As noted above, these scenarios were chosen based on optimal grade vs. tonnage and strip ratio. However, the amount of waste within the andesite hangingwall and footwall was also maximized, as it is necessary for the creation of the tailings impoundment area.

17.17.4 Open Pit Geotechnical Assessment

17.17.4.1 Geotechnical Characterization

Knight Piésold completed a technical review of the existing geotechnical database of the Šturec deposit. Basic geotechnical information including rock quality designation (RQD), rock hardness and joint characteristics were collected in ten of 79 existing exploration drillholes. However, no oriented core data has yet been collected. The geotechnical data was collected from core that was about 10 years old and had been disturbed by sampling for mineral testing. These factors mean that the quality of the core and geotechnical data is reduced.

Two simplified geological domains were delineated as follows:

- **Tertiary Andesite**: dominantly propylized lava flows with minor inter-bedded tuffs and breccias.
- **Quartz Veins**: dominantly white, vughy (containing voids), banded or brecciated massive quartz vein, often includes a mix of quartz and andesite material, broken.

Preliminary geotechnical parameters were estimated for the two rock mass types. Geomechanical properties of rock masses were characterized using Bieniawski's Rock Mass Rating (RMR) classification system. In general, the intact rock strengths were found to be medium strong with a typical value of 50 MPa for both units. The rock mass quality for the Tertiary Andesite unit is FAIR with a typical RMR value of 55, and the relatively broken Quartz Vein unit has been defined as a FAIR to POOR rock with a typical RMR value of 40. Sub-vertical north to northeast striking faults and veins cut across the pit area but small-scale structural features are unknown, and a relatively close joint spacing has been assumed. The groundwater table is expected to be low due to drainage of the pit by the underlying heritage drainage adits.

17.17.4.2 Results

The recommended pit slope angles for the prefeasibility level study are included in Table 17.20. It is noted that the current geological model is based on the extrapolation from a published regional geological map and the limited geotechnical database. Detailed geological model and additional geotechnical data will be required to complete a detailed slope stability assessment for the full feasibility pit slope design. Further detailed kinematic and rock mass stability analyses should be conducted to refine the pit design sectors and slope geometries once additional geotechnical data is available.

It is suggested that three oriented core holes and geological mapping be implemented in the Sturec pit area to collect confident geotechnical data for the feasibility level pit slope study.

Table 17.19: Recommended Pit Slope Angles

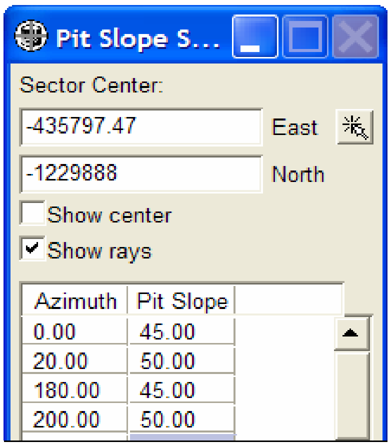
Pit Design Sector	Total Slope Height(1)	Major Geology on Final Wall	Kinematic Stability Analyses (2)			Rock Mass Stability Analyses (3)		**Recommended Slope Design (4), (5)**				Comments
			Max. Bench Face Angle	Max. Inter-ramp Slope Angle	Potential Instability Mechanism	Max. Overall Slope Angle	Disturbance Factor, D	**Inter-ramp Angle**	**Bench Face Angle**	**Bench Height**	**Bench Width**	
	m		degrees	degrees	-	degrees	-	**degrees**	**degrees**	**m**	**m**	
Northwest	180	Quartz Vein	70	45	Planar/Wedge	48	0.85	**45**	**70**	**20**	**12.5**	GOOD controlled production blasting (D=0.85) are required.
East	140	Tertiary Andesite	70	50	N/A	53	0.85	**50**	**70**	**20**	**9.5**	GOOD controlled production blasting (D=0.85) are required.
South	140	Quartz Vein	70	45	N/A	48	0.85	**45**	**70**	**20**	**12.5**	GOOD controlled production blasting (D=0.85) are required.
West	240	Tertiary Andesite	70	50	N/A	53	0.85	**50**	**70**	**20**	**9.5**	GOOD controlled production blasting (D=0.85) are required.

Notes:
(1) Represent the highest wall in each design sector.
(2) Typical values were apply when data is absent. A minimum bench width of 8 m is assumed.
(3) A minimum Factor of Safety (FOS) of 1.3 is targeted.
(4) The recommended slope angles were determined by the lowest value of the kinematic and rock mass stability analyses.
(5) A single bench height of 10 m is assumed and a double-bench configuration is recommended to achieve steeper slope angles.

17.17.5 Pit Design

Once the optimal pits were chosen, they were utilized as templates to design the pit and haul road. The design parameters for the pit were to utilize a double bench, 12.5 meter safety berm on the northeast and south walls and 9.5 meter on the east and west, 70° bench face angle, and 10% grade for the haul road. In addition, to accommodate geotechnical constraints, an inter-ramp angle of 50° was used with the exception of those sections of the wall that transect the vein (shown as a purple solid in Figure 17.47 and 17.48 (overleaf). The pit sectors transected by the vein required a reduced inter-ramp angle of 45° from azimuth 0° through 20° in the north sector and from 180° through 200° in the south sector (see Figure 17.46 below). The green lines in Figures 17.47 and 17.48 show the reduced slope sectors.

Figure 17.46: Azimuth and Pit Slope Angles



It should be noted that the initial location of the plant site was to the northwest of the pit. The open pit access was to the northeast. This open pit access was abandoned for the more desirable location west of the pit, which required the haul road exit to take a large notch out of the prominent ridge as shown in figures 17.47 and 17.48.

For scheduling, the approach was to create a Phase 1 Starter pit that would extract the relatively higher grade resources in the initial years, followed by a Phase 2 Ultimate.

The resulting resources are listed for the Phase 1, Phase 2 and combined pits in Tables 17.21 through 17.22, respectively. Note that these resources are based on using a cut-off grade of 0.75 g/t AuEq (i.e., gold equivalent).

In addition, the option of extracting material into a low grade stockpile was included in the analysis. This entailed calculating the differential between that material that is below 0.75 g/t AuEq, but greater than 0.5 g/t. The amount and grade of this differential material is listed in Table 17.22 denoted as "Stockpile" (in bold at bottom of table) which amounts to 3,064,123 at gold equivalent grade of 0.61g/t, gold 0.53g/t and silver 5.39g/t. with the premise that this material is profitable under the current revenue-cost scenario once it has already been extracted. That is, the cost of mining may be removed from the costs incurred, and only the cost of milling may now be considered for determining the profitability of this marginal material.

Figure 17.47: Phase 1 Starter Pit with Haul Road Intersected by Vein (purple solid) with Slope Sectors in Green

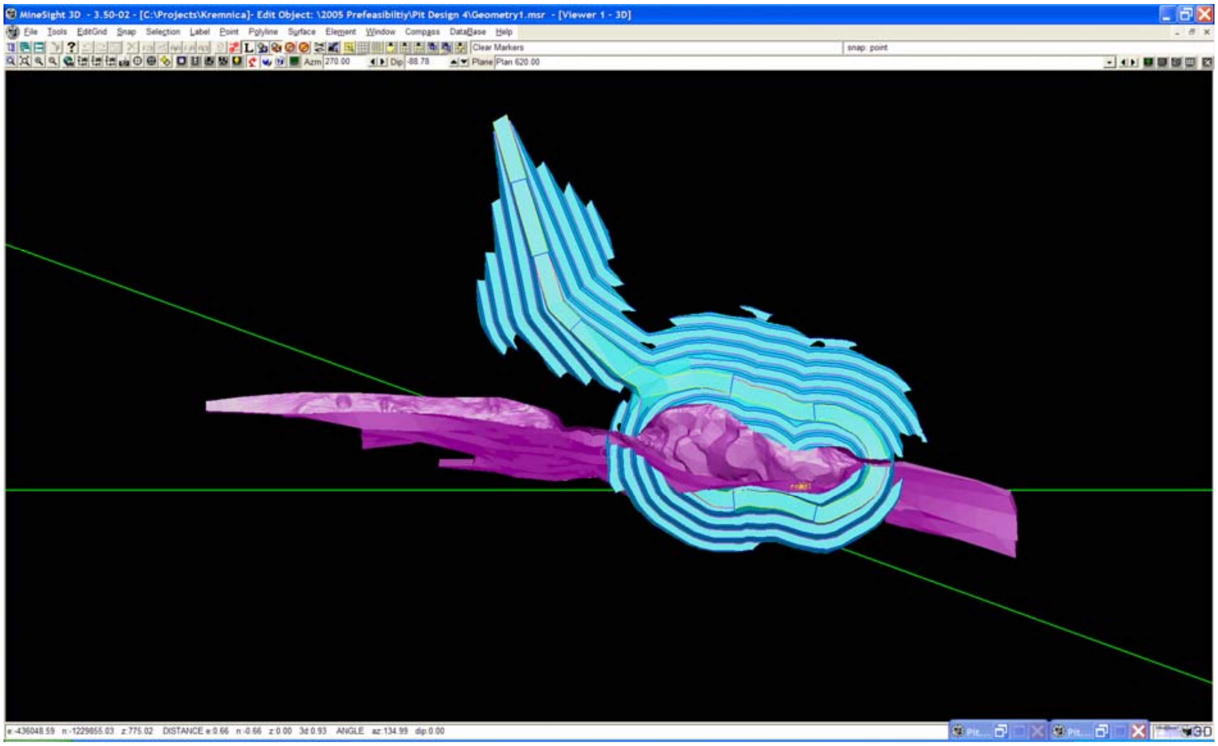


Figure 17.48: Phase 2 Ultimate Pit with Haul Road Intersected by Vein (purple solid) with Slope Sectors in Green

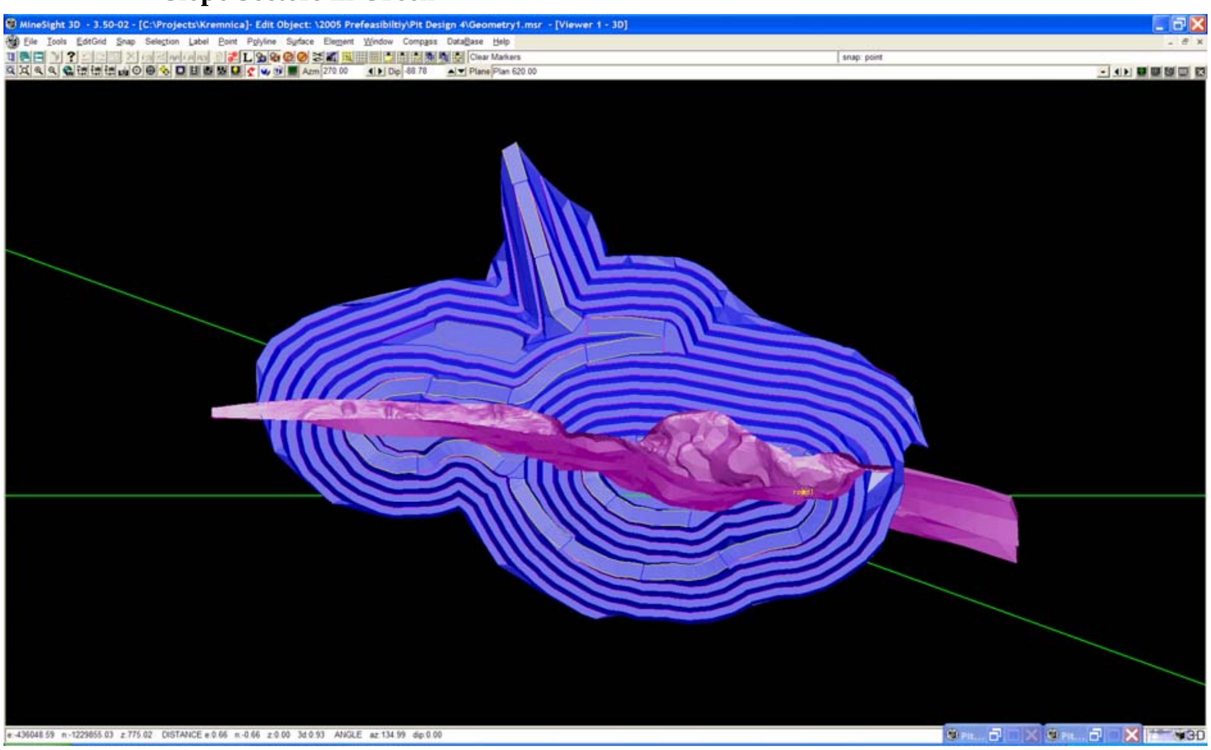


Table 17.20: Benched Resources for Phase 1 Pit

Bench	Resource	Waste	Strip Ratio	Grade		
Toe	Tonnes	Tonnes		gAuEq/t	gAu/t	gAg/t
795	-	5,765		0.98	0.90	5.63
790	1,781	13,983	7.85	0.94	0.86	4.99
785	5,006	22,772	4.55	1.02	0.93	5.85
780	5,208	38,627	7.42	1.02	0.93	6.15
775	1,511	40,752	26.97	1.02	0.92	6.34
770	1,088	66,023	60.68	1.02	0.92	6.54
765	866	95,449	110.22	1.03	0.93	6.48
760	2,633	126,095	47.89	0.95	0.89	4.45
755	4,867	119,962	24.65	1.13	1.06	4.69
750	8,049	153,730	19.10	1.23	1.16	4.62
745	12,608	192,040	15.23	1.46	1.38	5.36
740	20,521	230,453	11.23	1.64	1.56	5.62
735	15,738	191,721	12.18	1.70	1.61	6.12
730	13,733	234,423	17.07	1.53	1.45	5.80
725	13,222	274,071	20.73	1.54	1.45	5.84
720	13,491	304,806	22.59	1.50	1.42	5.56
715	6,393	206,477	32.30	1.16	1.11	3.62
710	7,695	213,154	27.70	1.25	1.18	4.14
705	11,246	220,429	19.60	1.45	1.37	5.25
700	15,358	235,068	15.31	1.59	1.48	6.99
695	8,831	179,568	20.33	1.52	1.39	8.65
690	19,635	192,064	9.78	1.65	1.50	9.70
685	38,836	222,568	5.73	1.72	1.55	10.81
680	53,606	256,082	4.78	1.72	1.55	10.83
675	58,643	230,475	3.93	1.86	1.68	11.81
670	84,429	258,902	3.07	1.83	1.64	12.87
665	100,929	270,415	2.68	1.83	1.63	13.66
660	114,939	279,111	2.43	1.91	1.68	15.05
655	129,054	193,159	1.50	1.95	1.71	16.09
650	166,147	203,130	1.22	2.01	1.76	16.72
645	205,275	200,517	0.98	2.15	1.88	17.96
640	234,397	177,078	0.76	2.10	1.83	18.09
635	240,377	94,451	0.39	2.23	1.95	18.78
630	237,520	78,065	0.33	2.29	2.00	19.03
625	227,181	59,920	0.26	2.33	2.05	18.65
620	210,302	50,363	0.24	2.43	2.14	19.06
615	156,833	14,874	0.09	2.52	2.24	19.04
610	133,115	13,217	0.10	2.42	2.15	18.24
605	106,909	11,814	0.11	2.42	2.14	18.68
600	89,871	5,993	0.07	2.35	2.06	18.74
595		0				
590		0				
585		0				
580		0				
575		0				
570		0				
565		0				
560		0				
555		0				
550		0				
545		0				
540		0				
TOTAL	**2,777,843**	**5,977,566**	**2.15**	**1.79**	**1.89**	**16.64**

Table 17.21: Benched Resources for Phase 2 Pit

Bench	Resource	Waste	Strip Ratio	Grade		
Toe	Tonnes	Tonnes		gAuEq/t	gAu/t	gAg/t
795	9	252	28.00	0.88	0.84	2.67
790	889	8,255	9.29	1.16	1.10	4.05
785	7,116	34,265	4.82	1.09	1.01	5.50
780	17,886	125,256	7.00	1.13	1.06	4.89
775	24,676	203,935	8.26	1.12	1.05	5.10
770	45,244	263,387	5.82	1.09	1.01	5.26
765	67,982	306,969	4.52	1.05	0.96	5.64
760	94,090	332,601	3.53	1.05	0.96	5.68
755	114,834	313,608	2.73	1.07	0.99	5.81
750	145,105	330,147	2.28	1.15	1.06	6.22
745	177,835	348,751	1.96	1.21	1.11	6.81
740	211,609	378,274	1.79	1.27	1.16	7.29
735	238,793	386,801	1.62	1.30	1.18	7.68
730	269,981	404,531	1.50	1.34	1.22	7.86
725	283,131	441,461	1.56	1.38	1.26	8.06
720	290,628	501,240	1.72	1.43	1.31	8.12
715	295,137	578,308	1.96	1.43	1.31	8.18
710	287,641	652,427	2.27	1.50	1.37	8.29
705	277,675	714,372	2.57	1.46	1.34	8.40
700	291,164	759,751	2.61	1.48	1.35	8.73
695	295,115	688,827	2.33	1.51	1.37	9.07
690	301,828	736,654	2.44	1.50	1.37	8.95
685	292,983	784,961	2.68	1.51	1.37	9.02
680	261,808	853,222	3.26	1.46	1.32	8.97
675	246,787	840,359	3.41	1.42	1.28	9.32
670	250,153	920,269	3.68	1.50	1.34	10.25
665	222,116	952,278	4.29	1.49	1.33	10.76
660	226,135	969,449	4.29	1.55	1.39	11.07
655	212,472	868,093	4.09	1.67	1.49	11.98
650	213,618	848,117	3.97	1.71	1.53	11.65
645	211,458	832,304	3.94	1.76	1.58	11.91
640	199,776	827,312	4.14	1.77	1.59	12.23
635	225,519	683,989	3.03	1.82	1.63	12.61
630	225,439	654,043	2.90	1.84	1.65	12.30
625	191,507	588,936	3.08	1.89	1.71	12.42
620	204,250	568,630	2.78	1.90	1.71	12.42
615	236,090	469,587	1.99	1.94	1.74	13.17
610	240,346	457,849	1.90	1.97	1.77	13.21
605	243,630	445,132	1.83	1.94	1.75	13.17
600	220,181	437,915	1.99	1.92	1.72	12.93
595	273,001	332,562	1.22	2.05	1.83	14.67
590	279,146	303,903	1.09	2.02	1.80	14.53
585	273,563	273,379	1.00	2.02	1.80	14.46
580	264,782	255,901	0.97	2.03	1.81	14.76
575	241,175	156,757	0.65	2.07	1.84	15.20
570	233,900	134,244	0.57	2.07	1.84	15.75
565	212,166	115,682	0.55	2.15	1.90	16.55
560	203,847	94,488	0.46	2.19	1.93	16.95
555	163,550	31,281	0.19	2.27	2.01	17.48
550	145,787	21,626	0.15	2.28	2.02	17.42
545	125,262	16,038	0.13	2.37	2.09	18.51
540	108,377	12,057	0.11	2.48	2.19	19.49
TOTAL	**10,387,192**	**23,260,435**	**1.73**	**1.79**	**1.60**	**12.45**
Stockpile	**3,064,123**			**0.61**	**0.53**	**5.39**

Note: "Stockpile" is material that grades from 0.5 g/t AuEq to 0.75 g/t AuEq, which will be stockpiled and processed subject to a return being achieved based upon rehandling and processing costs.

17.18 RESERVES ESTIMATE AND PRODUCTION RATE

The estimated open pit resources for the Šturec deposit were established by the optimization studies described in Section 17. These resources were used to derive the yearly mineral production rate based upon mining 6,000 tpd (2.1 million tonnes per year). This production rate was chosen based on the configuration of the mineralized zones and a mine life that balances capital expenditures against operating costs to provide the optimum return on investment. The open pit design incorporated a starter pit to enhance the grade during the initial years of production. Due to grade distribution, the starter pit was limited to 1.25 years. The overall development schedule was based on balancing the waste to reduce any peak periods and minimize equipment requirements.

Normally the stripping ratios are kept to a minimum during the early years of production and the waste pile is located as close to the open pit as possible to reduce waste haul costs and maximize the return on investment. It was decided by Tournigan, to eliminate the waste pile and instead haul it to the TSF and use the non-acid generating waste (NAG) for dam construction. As a result, the majority of dam construction material was available at a lower cost than if borrowed locally. To accommodate the requirements of dam construction, waste development prior to and during early production was maximized.

The access ramp to the open pit was relocated to remove it from view of the Kremnica townspeople, and berms were added at certain locations along the pit rim so that the view from town would be similar to the surrounding countryside. The process plant and associated facilities were located to reduce the visual impact to the people living at the town of Kremnica and surrounding villages. In addition, a rock berm around the site was designed into the site plan with suitable vegetation, such as trees, bushes and grass to further mitigate the visual effect of the facilities and allow these facilities to be assimilated into the surrounding landscape.

The optimization study indicated that the pit contained 13,165,000 tonnes of high-grade ore at 1.60 g/t gold and 12.40 g/t silver, and 3,064,000 tonnes of low grade ore, grading greater than 0.5 g/t AuEq. The low grade material will be stockpiled and milled after the higher grade material has been processed. This approach reduces the waste to 26,078,000 tonnes and the strip ratio to 1.6. The overall production parameters are shown on Table 18.1.

The additional low grade material, based upon the parameters as described in this study, can be mined and if they are processed after the higher grade material show a positive cash flow. Thus the additional low grade material can be included with the higher grade mineralization that based upon the findings of this report can be determined as reserves. **Thus, the estimated reserve of the Sturec deposit is 16,229,000 tonnes grading 1.40 gAu/t and 11.08 gAg/t.**

These mineral reserve estimates are based upon a 6,000 tpd production rate for a mine life of approximately 8 years. Included within the waste is material designated as inferred[7]. Further work may upgrade this material to a higher level of confidence, thus reducing the waste and increasing the mine life of the property.

[7] Due to the uncertainty that may be attached to an inferred mineral resource, it cannot be assumed that all or any part of an inferred mineral resource will be upgraded to an indicated or measured resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred resources must be excluded from estimates for feasibility or other economic studies.

SECTION 18.0 – OTHER RELEVANT DATA AND INFORMATION

18.1 MINING

18.1.1 Pit Design

Development of the Šturec open pit is planned as a conventional truck-and-shovel operation, utilizing diesel-powered equipment to maintain flexibility and minimize capital costs. Several combinations of truck/shovel size ranges were evaluated, with the most cost effective being a 10 m^3 excavator in combination with 90 tonne haul trucks. Overall pit design was based on an overall slope angle of 48° for the northeast and south walls, and 53° for the east and west walls Ore and waste will be mined on 10 meter benches, except in those areas where the ore configuration demands a more selective mining procedure to minimize dilution. In such cases, 5 meter benches will be utilized. Bench width on the northwest and south walls will be 12.7 m; on the east and west it will be 9.5 m. The overall height between benches will be 20 m, with a bench face angle of 70°.

Selective operations will be used at the pit perimeter using airtrack equipment 76mm holes to reduce the effect of blasting at surface that may cause anxiety for the local residents and to establish well contoured and defined walls.

The plant site and associated infrastructure has been located close to the open pit entrance to minimize haul distances for waste and rock (see Figure 18.1). The haul road within the pit has a maximum grade of 10%. The process plant is oriented so that trucks can dump their loads immediately after leaving the pit, thereby reducing haul distances to a minimum.

A system of five conveyors of varying lengths has been included in the overall design to transport the waste rock from the plant site to the TSF. The conveyors have been designed to accommodate <305 mm rock. Non acid generating (NAG) and potentially acid generating (PAG) waste will be hauled separately to a screen. Waste greater than 305 mm will be sent to the crusher and then returned to the screen. Underflow from the screen will be fed onto the conveyor where it will be transported to the TSF. Two stockpiles will be maintained at the TSF, one for NAG and one for PAG material. Utilizing a front-end loader and truck, the NAG waste will be used for dam building, while the PAG material will be placed in the tailings pond.

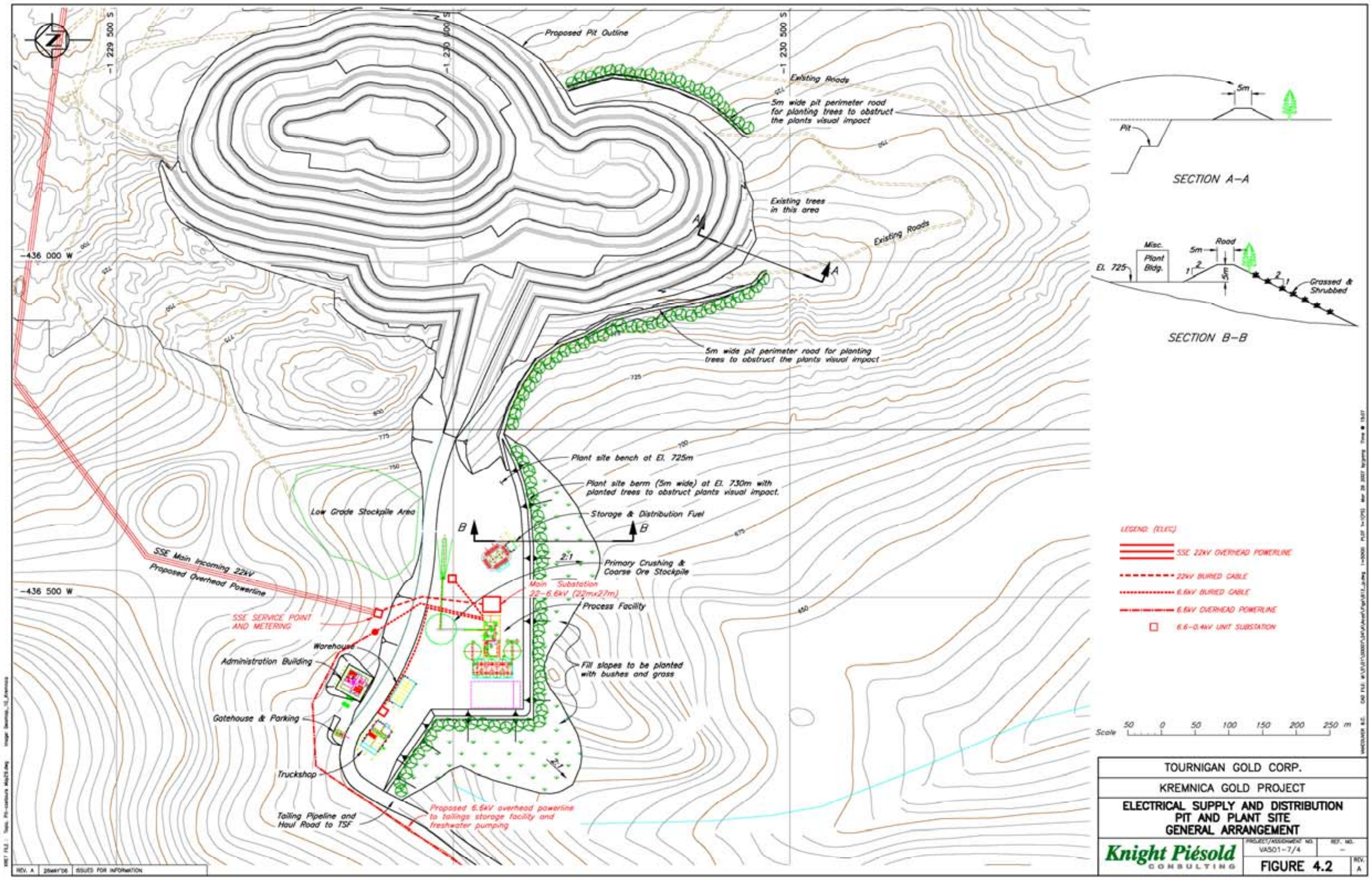
A road will be constructed adjacent to the conveyor to provide maintenance access, and the power supply system will parallel the conveyor road. Construction of the conveyor will follow topographical contours to minimize costs. A second roadway will be established to provide access to the property by vehicle. The elevation of the process plant will provide an effective slope for gravity flow of tailings to the TSF.

Table 18.1: Production Schedule 6,000 tpd Conveyor

Kremnica Gold Corp
Šturec Deposit
Production Schedule
6000tpd Conveyor

	Pre-strip	1	2	3	4	5	6	7	8	Total
Ore Mined tonnes	203,844	2,296,024	3,019,971	2,909,891	2,731,668	2,373,270	2,247,535	446,955		**16,229,158**
High Grade Stockpile tonnes	145,656									**145,656**
Low Grade Stockpile tonnes	58,188	196,024	919,971	809,891	631,668	273,270	147,535	27,576		**3,064,123**
HG Stockpile Processed tonnes								145,656		**145,656**
LG Stockpile Processed tonnes								1,534,965	1,529,158	**3,064,123**
Ore Processed tonnes		**2,100,000**	**2,100,000**	**2,100,000**	**2,100,000**	**2,100,000**	**2,100,000**	**2,100,000**	**1,529,158**	**16,229,158**
Waste tonnes	2,472,115	3,154,978	2,613,322	3,526,495	7,141,734	5,692,427	1,519,381	53,426		**26,173,878**
Total Tonnes Mined	**2,675,959**	**5,451,002**	**5,633,293**	**6,436,386**	**9,873,402**	**8,065,697**	**3,766,916**	**500,381**		**42,403,036**
Waste: Ore Strip Ratio	12.13	1.37	0.87	1.21	2.61	2.40	0.68	0.12		**1.61**
Gold Grade Mined gAu/t	1.11	1.74	1.13	1.12	1.19	1.56	1.76	1.99		**1.40**
Silver Grade Mined gAg/t	5.06	16.00	8.27	7.45	9.27	12.08	14.90	17.69		**11.08**
Gold Grade Stockpile gAu/t	1.11	0.52	0.55	0.54	0.53	0.52	0.50	0.48		**0.53**
Silver Grade Stockpile gAg/t	5.06	6.47	4.45	4.75	5.82	7.25	7.95	8.68		**5.39**
Gold Grade Processed gAu/t		**1.85**	**1.39**	**1.34**	**1.39**	**1.69**	**1.85**	**0.90**	**0.53**	**1.40**
Silver Grade Processed gAg/t		**16.89**	**9.94**	**8.49**	**10.31**	**12.70**	**15.39**	**7.97**	**5.39**	**11.08**

Figure 18.1: Electrical, Supply and Distribution with Pit and Plant Site General Arrangement



18.1.2 Mining Operations

Mining activities at the Šturec pit will be conducted on 6 days per week, two 8-hour shifts per day, 302 days per year (based upon 52 weeks, 6 days per week gives 312 days less 10 statutory holidays. This schedule is being used to eliminate noise from the open pit operations during the night shift, Sundays and statutory holidays). To achieve a throughput for the process plant of 6000 tpd, 2.1 million tonnes per year, the open pit will produce some 6950 tpd. The study has been based upon owner-purchased and owner-operated equipment. Budget costs for a contractor operated open pit operation were obtained from a Canadian contractor who has experience in Slovakia. The costs were substantially higher than those calculated for an owner operated approach. Thus, contractors will only be used for specialized services.

Ore and waste will be drilled and blasted using 165 mm diameter holes on a 4.75 x 4.75 m pattern. Drilling and blasting of all ore has been included in the cost estimate even though a portion of the rubbly material in the existing pit may be rippable. Some of this collapsed, blocky material will be difficult to drill, however, and any cost savings seen as a result of being able to rip some of the rubbly ore will likely be offset by increased drilling and blasting costs in the remaining portion.

Blasting operations will use a combination of ammonium nitrate and fuel oil (ANFO) and water-resistant emulsion explosives. Estimated powder factors range between 0.31 and 0.33 kg/t material. The pit should be relatively well drained due to the existence of underground workings below the pit, so it has been assumed that only about 20% of the holes will be "wet."

The equipment list is based upon the foregoing production rates. As a result of the limited hours of mine operations per year, some 57% of the total hours available, it is considered that approximately 40% of the available hours will be available for preventative maintenance.

In addition to the 10 m^3 hydraulic loading shovel and 90 tonne rigid body haul trucks, several pieces of auxiliary and service equipment will be required to maintain haul roads and waste dumps, and for general pit operations. A list of all major equipment is shown in Table 18.2.

Table 18.2: List of Major Equipment

Description	# Req'd
Atlas Copco Viper Drill	2
Komatsu PC1800 Shovel	2
Komatsu WA600 Loader	2
Komatsu HD785 Truck	7
Komatsu D155 Dozer	2
Komatsu GD825 Grader	1
Komatsu WA320 Loader	1
Air Trac/compressor	1
Utility/Water Truck	2
Service truck	2
Crane truck	1
Blaster's truck	2
Light plant	4
Pick-ups	5
Pump truck	1
Misc. Shop Equipment (lot)	1
Spares @ 5% (lot)	1
Conveyor	5

18.2 WASTE MANAGEMENT

18.2.1 Introduction

18.2.1.1 General

The Kremnica Gold project will generate mine waste in the form of waste rock, overburden, and tailings from ore processing. This waste needs to be disposed of in a safe, environmentally and socially acceptable manner both during operation and after closure and rehabilitation. This section of the report summarizes the pre-feasibility design of the mine waste management facilities.

18.2.1.2 Design Basis

Preliminary design and operating criteria for the mine waste management facilities have been developed to allow preparation of the pre-feasibility study. Key aspects of the design basis are listed below.

- It is intended that the mill will operate at 6,000 tpd (2.1 million tonnes per year) for an eight-year mine life. During this time, the open pit will yield approximately 16.2 million tonnes of ore and 26 million tonnes of waste rock.
- Laboratory testing results for potential acid generation or metal leaching of waste rock are not yet available. For the pre-feasibility study it was assumed that all altered andesite and quartz vein material will be potentially reactive (PR) and all non-altered andesite will be non-reactive (NR). On this basis, 8.6 million tonnes or approximately 30% of the waste rock is PR. This waste must

be disposed of sub-aqueously to prevent acid generation or metal leaching. The PR waste rock will be disposed within the tailings storage facility (TSF) below the final tailings surface elevation.

- The TSF has been designed to permanently store 16.3 million cubic meters of waste consisting of 12.0 million cubic meters (16 million dry tonnes) of tailings and 4.3 million cubic meters (8.6 million dry tonnes) of PR waste rock.
- In compliance with Slovakian Waste Management legislation, the TSF will be fully lined with a composite liner consisting of 2.5 mm thick high-density polyethylene (HDPE) over 500 mm of compacted clay.
- A reclaim barge located within the TSF will supply water (precipitation, runoff, and recovered water from tailings consolidation) to the mill for use in the milling process.
- An additional freshwater supply of at least 1350 m^3/day will be required for the milling process, plus requirements for potable water of 100 liters/person/day.

18.2.2 Proposed Layout of Waste Management Facilities

A site selection study was carried out for the TSF, plant site, and waste rock disposal area. This work included producing a desktop study of potential sites, visiting the site to examine potential and preferred locations, and conducting preliminary geotechnical investigations to confirm the suitability of these sites.

The arrangement of mine and waste management facilities adopted for the pre-feasibility design is shown on Figure 18.2; a detailed view of the open pit and plant site is shown on Figure 18.3. Key aspects of the arrangement are summarized below.

- The open pit has been designed such that the pit haul road passes through a cut in the ridge to the west and leads directly to the plant site. This design minimizes the required haul distance and associated costs. It also minimizes the visual impact of the road by avoiding the route around the ridge to the south of the pit.
- A berm will be constructed around the eastern and southern edges of the pit. This berm will be vegetated to provide a visual screen.
- The plant site is constructed on a fill platform at an elevation of 725 m above Lúĉky village. The majority of the plant facilities are located on a ridge that has good foundation conditions and will provide a visual barrier. Combined with additional tree planting around the perimeter, the plant site should be mostly hidden from Lúĉky village.
- All waste rock will be transported to the TSF by conveyor. An access road and the tailings and reclaim pipelines will follow a similar route.
- The TSF is located about 5 km south of the proposed plant site in the upper portion of a valley above Kopernica village.
- An access road will be constructed from south of Kremnica to the plant site. All mine construction equipment will use this new access road instead of passing through Kremnica or other towns.
- An access road will also be constructed to the existing bentonite plant south of Kopernica village. This road will be provided as benefit to the community, because trucks will be able to access the plant directly rather than passing through Kopernica and Lúĉky .

- Two potential supply options have been identified for fresh water: (1) pumping from the nearby Heritage adit, or (2) pumping from groundwater wells above the plant site.

The TSF site above Kopernica was selected because of the following advantages:

- Its proximity to the open pit results in lower costs for tailings delivery and hauling of waste rock.
- The site is downhill from the proposed plant, allowing gravity flow of the tailings.
- The site is within a valley and will generally be hidden from view.
- The catchment area above the impoundment is relatively small, and water management will have lower costs than other options.
- The site is generally grassed with only a small portion of forest. The environmental impacts are expected to be relatively low.

Figure 18.2: General Arrangement

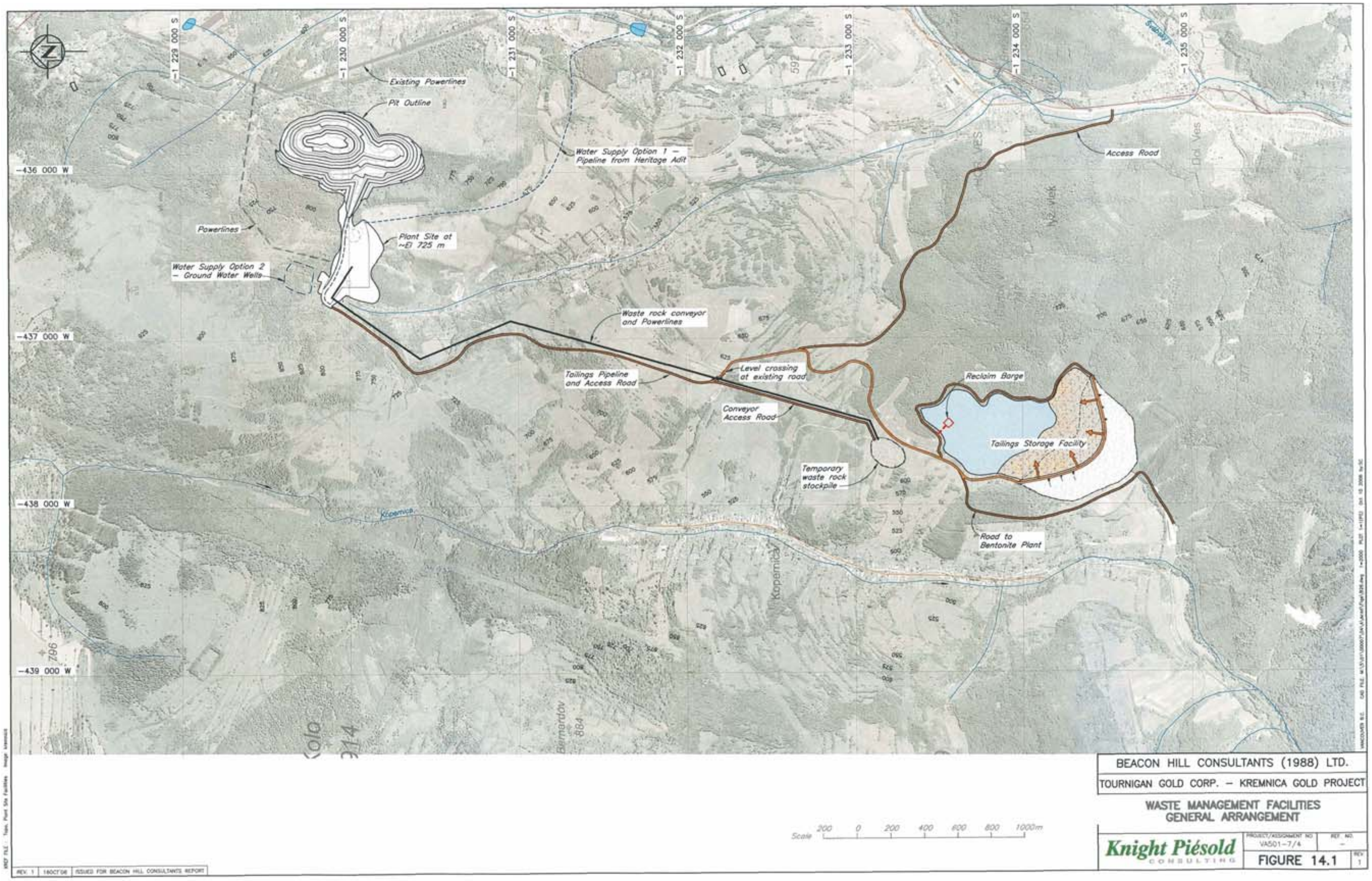
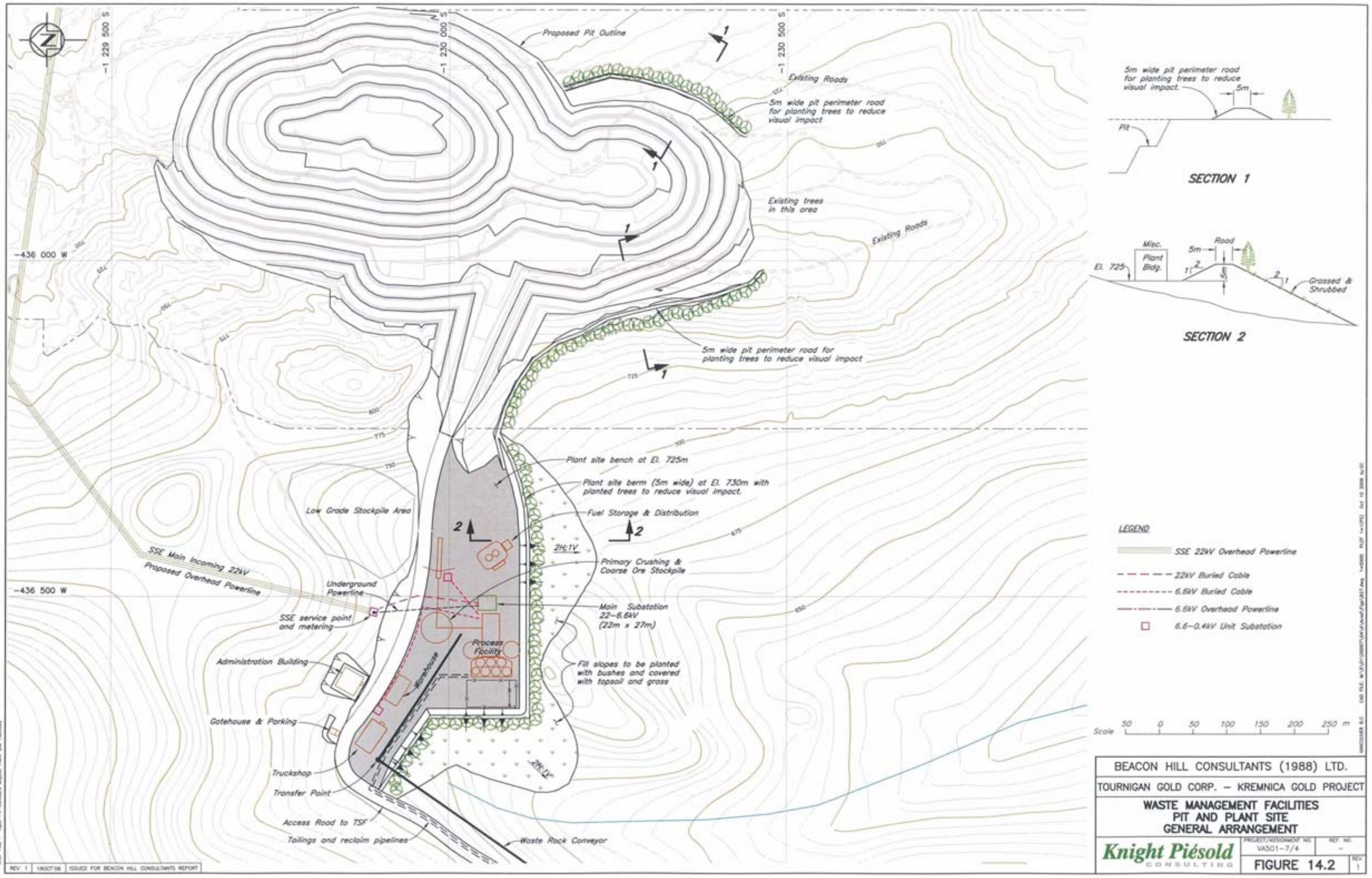


Figure 18.3: Pit and Plant Site General Arrangement



18.2.3 Tailings Storage Facility Design

18.2.3.1 General

The principal objectives of the pre-feasibility design for the tailings storage facility (TSF) are to provide storage for all tailings, PR rock, and site water, while ensuring the protection of the regional groundwater and surface waters both during operations and in the long-term (after closure), and to achieve effective reclamation at mine closure.

18.2.3.2 Construction Stages

The TSF will be constructed in a staged manner, which offers a number of advantages including:

- the ability to reduce upfront capital costs
- better coordination with the open pit excavation for reuse of NR waste rock for embankment construction
- the ability to refine design and construction methodologies as experience is gained with local conditions and constraints
- the ability to change plans at a future date to remain current with state-of-the-art engineering and environmental practices.

A filling schedule for the TSF has been developed, as shown on Figure 18.4. The proposed construction stages are also shown on Figure 18.4. The first (start-up) stage provides approximately two years' operating capacity before raising is required. Three additional stages of raising are proposed over the eight-year mine life. The final crest elevation will be 583 m, resulting in an embankment that is approximately 85 m at maximum height.

18.2.3.3 TSF Seepage Control

Slovakian legislation currently classifies the TSF as a landfill containing hazardous waste under the *Waste Act*. This legislation requires the TSF embankment and impoundment to have an artificially completed composite liner consisting of 0.5 m thick clay (with permeability of $kf \leq 1.0 \times 10^{-10}$ m/s) and at least one layer of 2.5 mm thick HDPE.

To facilitate the installation of the composite liner, the foundation upstream of the embankment will be contoured by cutting or filling steep areas to provide final slopes that are flatter than 3H:1V. Drains will be constructed in the foundation to assist with initial basin preparation and liner installation.

18.2.3.4 Embankment Arrangement

The dam will be constructed as a staged, downstream embankment, as required by the European Commission's "Best Available Technology" recommendations. The overall arrangement of the TSF at start-up is shown on Figure 18.5; the final arrangement plan is shown on Figure 18.6. A cross-section of the TSF embankment is shown on Figure 18.7.

Figure 18.4: Tailings Storage Facility Filling and Construction Schedule

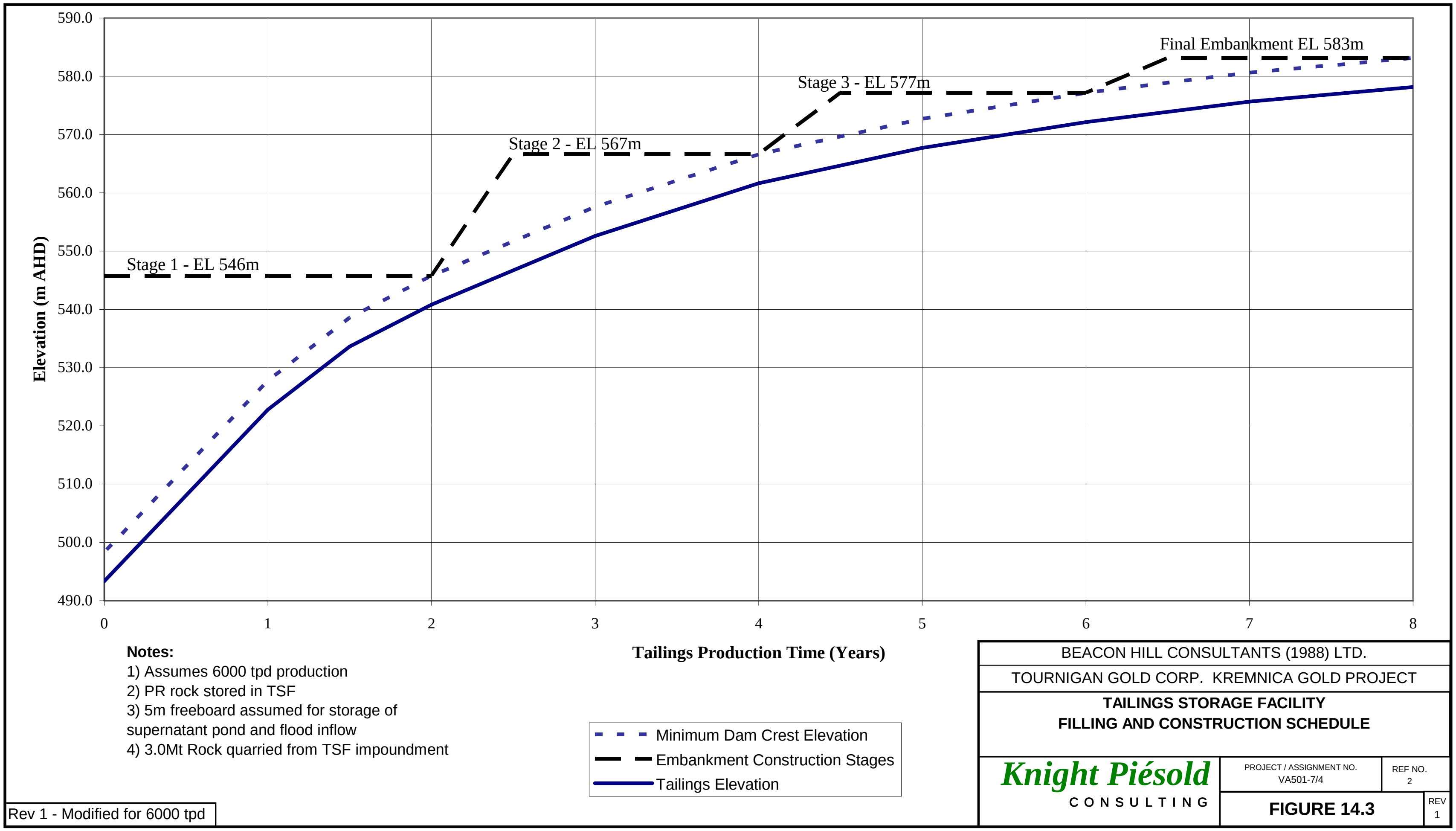


Figure 18.5: Tailings Storage Facility - Start-up Plan

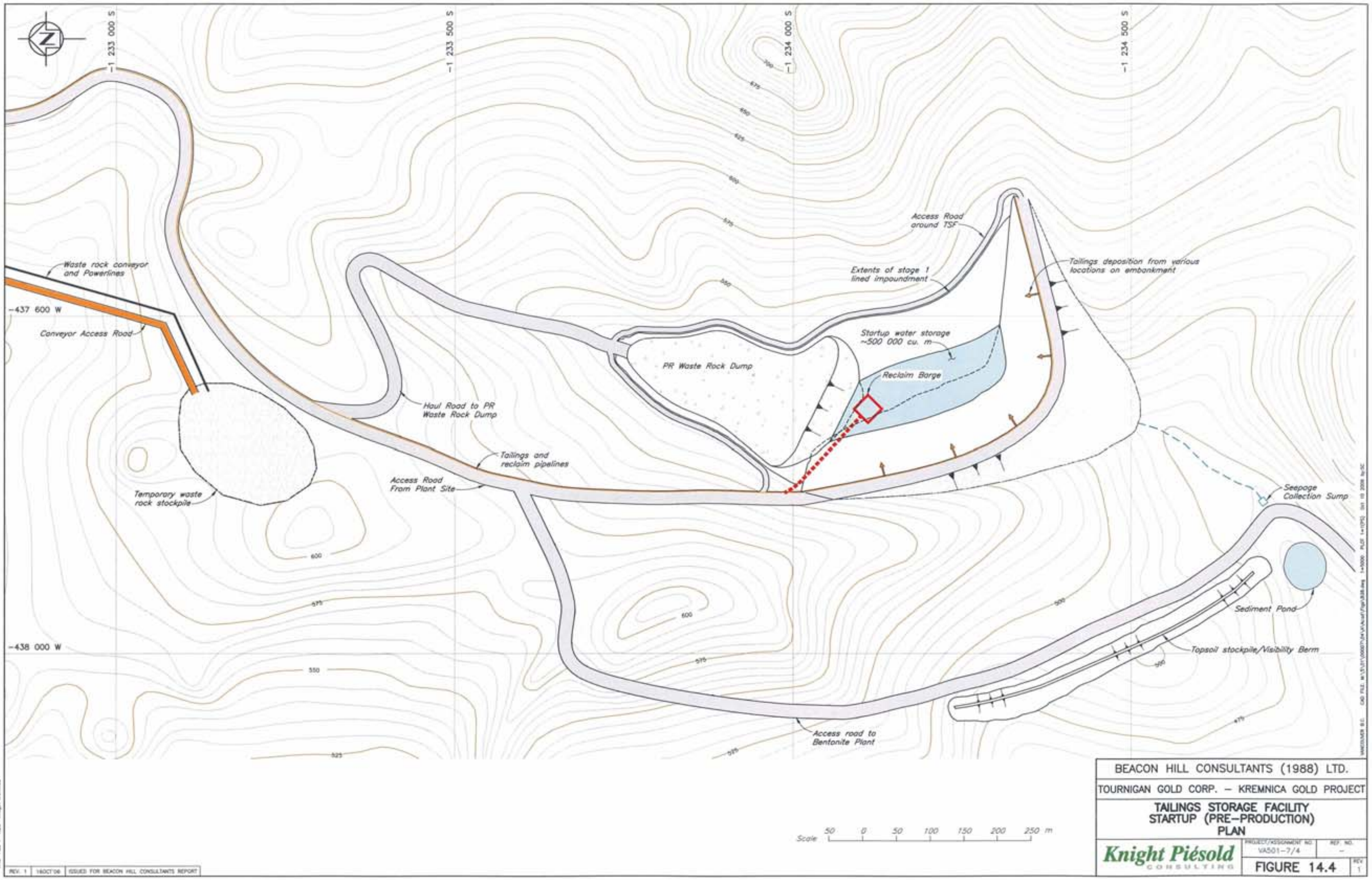
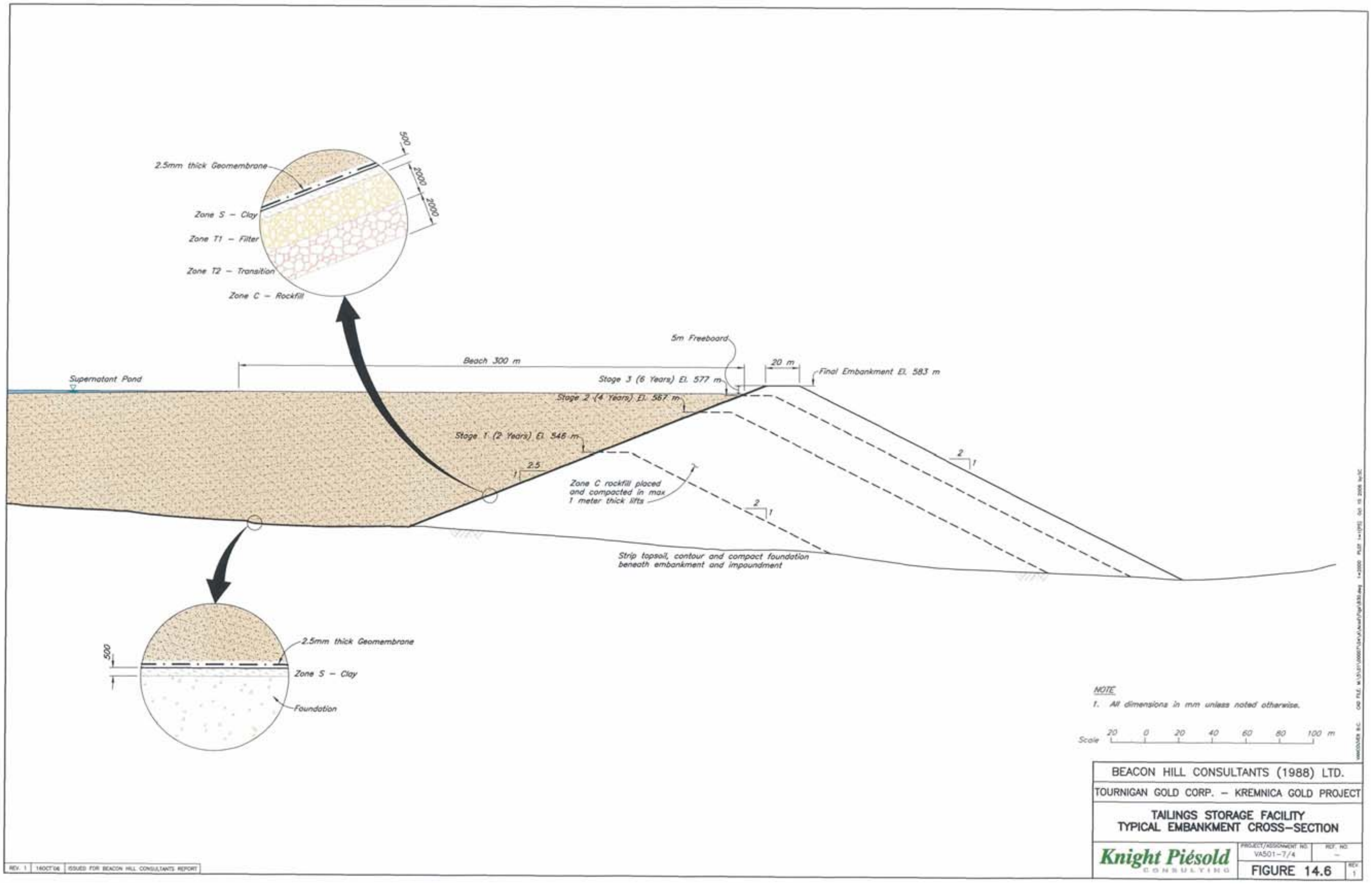


Figure 18.6: Tailings Storage Facility - Final Arrangement Plan

Figure 18.7: Tailings Storage Facility - Typical Embankment Cross-section



Key features of the embankment arrangement are summarized below.

- The upstream face will have a composite liner consisting of a 2.5 mm thick geomembrane liner over 500 mm of clay. This will be continuous, with the impoundment liner providing a completely sealed impoundment.
- A filter/transition zone will be provided downstream of the liner to avoid potential piping and ensure self-healing if the liner were to fail.
- The remainder of the embankment will be constructed with general rockfill, with a maximum upstream slope of 2.5H:1V. A maximum downstream slope of 2H:1V will permit topsoil to be spread for reclamation purposes when the embankment is completed.

18.2.3.5 Embankment Materials

The following embankment materials will be sourced from local areas:

- Clay Liner (Zone S) – The upstream clay liner will be constructed with clay material sourced from nearby deposits. The area surrounding the TSF has numerous existing and proposed clay quarries where bentonite and illite are mined in commercial operations. It is anticipated that suitable clay materials will be available from these sources.
- Transition Zone (Zone T) – No sources of free draining sand and gravel suitable for transition zone materials have been identified to date. Further investigations are required to search for such materials. At this stage it is assumed that crushing and screening of NR waste rock from the pit will produce the transition zone materials.
- General Rockfill (Zone C) – The main embankment will be constructed with random fill obtained from local quarries or NR rock from open pit. Pit pre-stripping has been scheduled such that approximately 2 million tonnes of NR waste rock will be produced prior to mine operation. An additional 3 million tonnes of rockfill will be required from quarries immediately upstream of the embankment, within the TSF impoundment area.

  The stages of embankment raising during mine operation will be constructed using NR waste rock from the open pit operations. The advantages of hauling this waste material for TSF construction rather than placing it in waste rock piles are:

  - no external dump of waste rock is required
  - the excavated material is reused
  - the cost of hauling the waste rock 5 km to the embankment is less than the combined cost of quarrying for Zone C and placing the waste rock in an external dump.

18.2.3.6 Disposal of Potentially Reactive Rock

Potentially reactive (PR) waste rock will be co-disposed with the tailings in the TSF. The PR rock will be transported by conveyor to a temporary stockpile, adjacent to the TSF. It will then be loaded into trucks and end-dumped in the impoundment. A haul road will be constructed from the temporary stockpile to the base of the impoundment, where an initial PR rock dump will be placed during the pre-operations phase. As tailings are deposited in the impoundment, the haul road will be progressively raised using PR rock to maintain a dry causeway above the tailings and supernatant pond elevation.

The PR rock will ultimately be encapsulated by tailings and stored sub-aqueously to prevent acid generation or metal leaching. Very little PR rock will be generated in the final two years of mining, so encapsulation can be achieved.

18.2.3.7 Disposal of Non-Reactive Rock

Non-reactive (NR) mine waste rock will be used to construct the TSF embankment, plant site, roads, and other fills throughout the project site. NR rock will be transported by conveyor from the plant site to a temporary stockpile adjacent to the TSF. It will then be loaded into trucks and transported to the required location on the TSF embankment.

Sufficient quantities of rockfill to construct the ultimate embankment will be available after about five years of operation. After the embankment has been constructed, excess NR waste rock will be disposed of in a confined valley to the south of the TSF embankment or placed on the downstream face of the TSF dam to increase stability. The approximate quantity of excess waste rock is 2 to 3 Mm^3, which can easily be accommodated within this valley. Most of this NR rock can be reused for reclamation of the TSF impoundment surface at mine closure. It is likely that the final stockpile of excess NR rock will be quite small.

18.2.3.8 Seepage Analysis

A preliminary steady-state seepage analysis for the TSF was carried out to estimate the amount of seepage from the lined impoundment.

At start-up, the TSF will store water for plant start-up. The results of the seepage analyses indicate that the total leakage through the liner for the start-up impoundment will be approximately 0.01 L/min, which is negligible.

The seepage flow rate is expected to vary over the life of the TSF, as it is gradually filled with tailings and the tailings permeability decreases due to ongoing consolidation. The estimated seepage from the ultimate impoundment is approximately 0.1 L/min, which again is negligible.

18.2.3.9 Stability Analyses

Embankment stability analyses were carried out to verify the stability of the final downstream embankment arrangement for each of the following cases:

1) The minimum acceptable factor of safety for the embankment for static conditions is 1.3 for short-term operating conditions and 1.5 after closure of the TSF. The predicted factor of safety is approximately 1.8.
2) Earthquake loading for the assumed Maximum Design Earthquake (MDE) was a peak bedrock acceleration of 0.2 g. A factor of safety of greater than 1.0 for earthquake loading conditions indicates that the embankment is seismically stable. The predicted factor of safety is approximately 1.1.

The stability analysis was based on assumed foundation material properties, which should be confirmed by geotechnical investigations.

18.2.4 Closure and Reclamation

The primary objective of the closure and reclamation initiatives will be to transform the plant and waste facility sites into an integrated component of the surrounding ecosystem, mimicking the pre-mining usage of this area. There is potential to add value to the pre-mining use of the area during closure; for example, a sports field could be built on the tailings impoundment or tourist facilities could be erected at the mine site, etc. Ongoing public consultation on this issue will be considered when discussing mine closure. The TSF closure design will be required to maintain long-term physical and geochemical stability, protect the downstream environment, and manage surface water.

Upon mine closure, surface facilities will be removed in stages and full reclamation of the TSF will be initiated. General aspects of the closure plan include:

- revegetating the embankment face using topsoil stockpiled from stripping activities immediately after construction of the final embankment
- selectively discharging tailings around the TSF during the final years of operations to establish a final tailings beach that will facilitate surface water management and reclamation
- dismantling and removing the tailings and reclaim delivery systems and all pipelines, structures, and equipment not required beyond mine closure
- removing and possibly treating excess ponded water within the impoundment
- constructing a dry engineered cover over the tailings surface (the excess NR waste rock will be used to cover the tailings surface followed by a layer of topsoil; at this stage, it is assumed that a geomembrane liner will not be required for capping the TSF)
- removing the seepage monitoring system when water quality is shown to be acceptable
- removing and regrading all access roads, ponds, ditches, and borrow areas
- ensuring the long-term stabilization of all exposed erodible materials.

18.2.5 Site Water Management

18.2.5.1 Water Management Plan

The Water Management Plan describes methods that will be adopted to control all water that originates within, or is brought into, the project area in an environmentally responsible manner, and to maintain a flow of freshwater from local runoff within the project boundaries, where possible.

18.2.5.2 Water Management and Sediment Control – Pre-production

The pre-production water management and sediment control plan consists of the following:

- Sediment control measures will be established prior to construction of the TSF embankment, plant site, and initial pit development.
- Approximately nine months of runoff from the TSF catchment will need to be collected and stored to provide sufficient water for plant start-up. If the TSF liner cannot be installed at least nine months prior to start-up, a supplemental water supply for the TSF, such as pumping from the Heritage adit or a nearby creek, may be required.

18.2.5.3 Water Management and Sediment Control – During Operations

The water management and sediment control plan during operations consists of the following:

- Runoff within the TSF catchment area will generally be diverted around the TSF, although some may be collected for the mill process in the event of any shortfall of available water.
- The sediment control measures established during pre-production will be maintained and managed during operations.
- Groundwater and undiverted runoff from the pit will be collected and used in the mill process.
- Process water will be discharged into the TSF with the tailings slurry. Tailings supernatant water will be pumped back to the mill for reuse. There will be no untreated discharge from the TSF.
- The TSF will be designed with sufficient freeboard to retain the probable maximum flood event due to either extreme rainfall or snowmelt. The design will assume that the diversion ditches around the TSF are unserviceable and there will be no release of surface water from the TSF.

18.2.5.4 Fresh Water Supply

Fresh water will be required for certain inputs to the mill process and for potable water. The ongoing fresh water requirements at the mine site will be approximately 480,000 m^3/year for the mill process and potable water supply. This water may be obtained from three potential sources:

- pumping the required fresh water from the Heritage adit
- installing groundwater wells above the plant site
- Collecting water diverted from the TSF catchment in a freshwater pond and pumping to the plant for supply of makeup water.

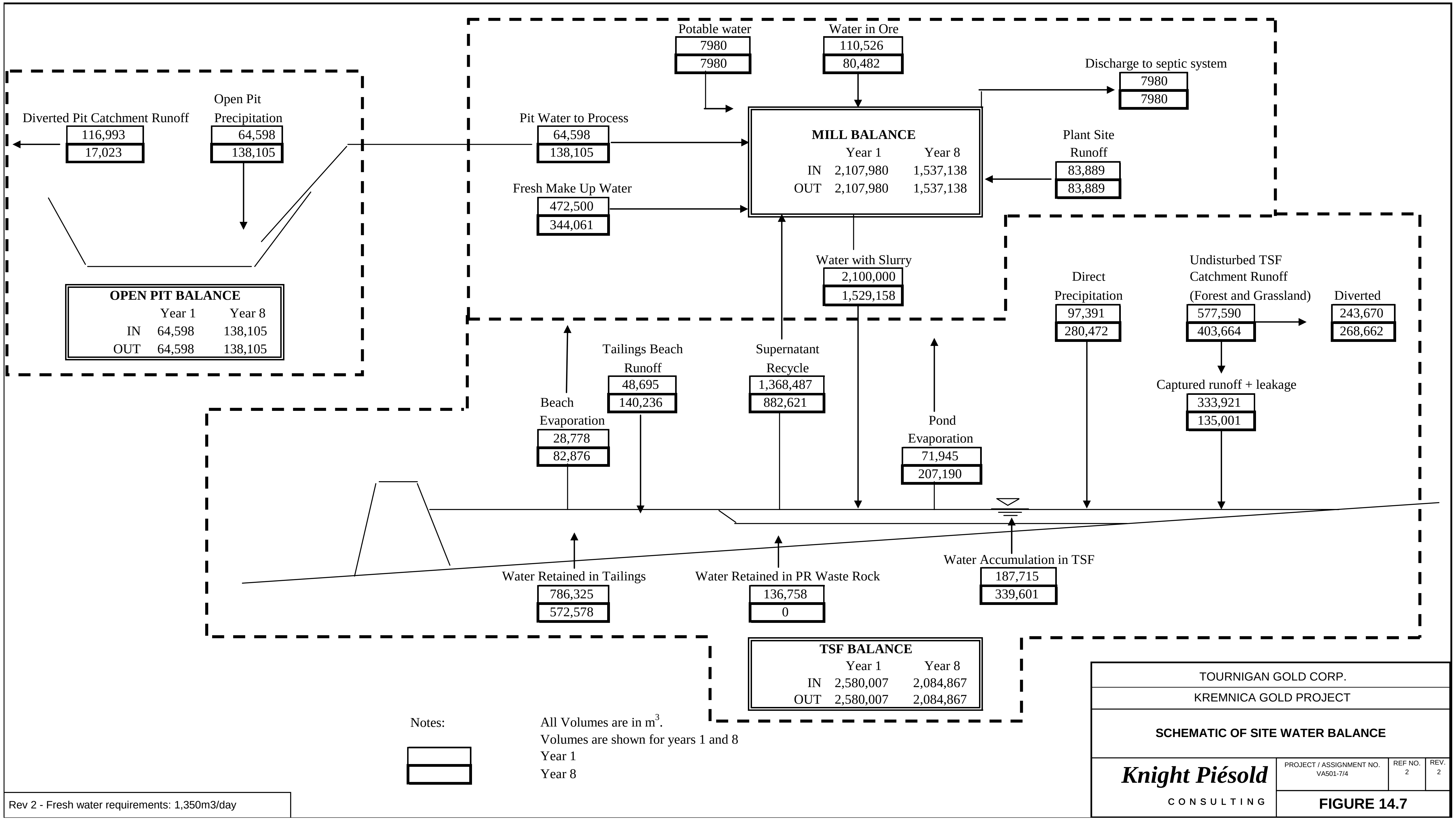
The water supply options (shown on Figure 18.2) have not been considered in detail in the pre-feasibility study. Additional monitoring is being carried out to examine the feasibility of these two potential sources.

18.2.5.5 Site Water Balance

A site-wide annual water balance was prepared to aid in the development of the water management plan and to estimate the supernatant pond volume in the TSF during operations. The results of the water balance are shown schematically on Figure 18.8. It should be noted that the water balance is very sensitive to hydrometeorological inputs and may change significantly when better site-specific information is obtained.

Prior to start-up, in Year -1 of the water balance, a pond volume of at least 525,000 m^3 will be stored in the TSF to provide three months' supply for mill processing.

Figure 18.8: Schematic of Site Water Balance



The water balance shows a slight water surplus in the system, from 30,000 m^3 in year 2 increasing to become an excess of 340,000 m^3 by Year 8. This results in an estimated final pond volume of approximately 2 million m^3. The implications of this excess water include requiring a greater freeboard for containment of the pond plus PMF flood inflow and a greater cost at closure for the treatment of water prior to release. Future studies should consider potential ways to reduce this water excess and the associated costs and risks.

18.2.5.6 Tailings and Reclaim Pipeline Systems

Bulk tailings will flow by gravity to the TSF in a single 300 mm HDPE pipeline and will be discharged through valved outlets in a pipeline laid along the inside crest of the TSF embankment. Discharging tailings will fan out over previously deposited material and form a beach that will slope gently down to the supernatant pond at the lowest point in the facility. Water bleeding from the tailings will flow to the supernatant pond. The planned sequential rotation of the point of tailings discharge will allow the elevation of the beach to be managed and the surface pond location to be controlled, and will ensure maximum and efficient material storage in the TSF.

Tailings will be thickened at the mill to minimize the requirement for water reclaim from the TSF. Thickening will be such that gravity discharge of tailings to the TSF, with no pumping, is possible for the life of the project.

Water will be reclaimed from the TSF and returned to the mill using a floating pump station. The reclaim pump station will be mounted on a barge platform that floats on the TSF supernatant pond. It will be moored to anchor blocks by adjustable cables and linked to shore by its discharge pipeline and a walkway. The pump station will be located in a channel or deep section of the surface water pond, where clean water recovery can be maximized.

The reclaim pipeline will consist of two sections: a permanent section located outside the footprint of the TSF, and a section within the TSF that will require periodic relocation in conjunction with the pump station. The upper section will consist of an HDPE pipe; steel pipe will be required for the lower section due to the high pressure. The line may be buried to provide resistance to movement and minimize any potential for freezing.

18.2.6 Pit and Plant Site Geotechnical Assessment

18.2.6.1 Open Pit

The proposed pit is approximately 800 m x 400 m in plan and extends to a depth of approximately 200 m below ground surface. A preliminary pit slope design has been carried out based on the limited existing geotechnical data.

18.2.6.2 Plant Site

The proposed plant site area lies on a ridge at an elevation of approximately 725 m (Figure 18.3). The plant site is part of a moderately sloping, grassed slope above Lúĉky village. The ridge is wide, relatively flat, and is well drained on both sides. This will minimize potential problems with groundwater and surface water.

Geotechnical investigations at the proposed plant site generally encountered shallow topsoil overlying sandy clay soils. Weathered bedrock was encountered at 1.5 to 2.5 m depth. Good foundation conditions should be achievable for the plant. Feasible foundation types include spread footings, raft or mat foundations, and end-bearing piles.

The site is snow-covered in winter, which means that the ground will freeze, and without adequate site preparation, frost heave in the foundations may be problematic. Because of the significantly higher foundation strength and non-frost susceptibility of the rock foundations, it is recommended that heavy structures be founded on rock. This will be done by excavating and pouring blinding concrete directly onto the rock for spread footings.

18.2.7 Risks and Opportunities

The development of the pre-feasibility design has highlighted a number of risks and opportunities related to the waste management facilities and geotechnical assessment. Some opportunities that should be investigated during the feasibility study include:

- Approximately 30% of the initial capital costs for the TSF are in quarrying Zone C rockfill for the starter embankment. Additional rock is NR rock that is mined as part of the open pit excavation and transported to the TSF site by conveyor. After geochemical testing of the waste rock materials is completed, it may be possible to focus the pre-stripping to produce more NR waste and reduce the quantity of quarried material. Other ways to reduce this quarry volume should be investigated.
- Reclaiming water from the TSF is costly because of the significant distance and head. The cost advantages of transporting a thicker tailings slurry or dry tailings, and the associated reduction of reclaim water pumping, should be taken into consideration.
- The full composite liner in the impoundment has been provided to comply with Slovakian Waste Management legislation. It is understood that the legislation may be changing and a more economic alternative may become possible.

Some risks and uncertainties associated with waste management and the geotechnical design that should be studied further during the feasibility study include:

- A geotechnical investigation and condemnation drilling for the preferred TSF site 4 was underway at the time of this report. The results of the fieldwork and associated analysis will help mitigate the potential risk of TSF 4.
- The costs associated with land purchase at TSF site 4 and the plant site may be significant due to the large number of landholders.
- Geochemical testing of waste rock for characterization as PR or NR waste has not been completed. There is a high cost associated with storing PR waste in the TSF, so changes in the ratio of PR to NR waste will have significant cost consequences.
- Laboratory results of condemnation drilling at the proposed plant site are not yet available. If the plant site is not feasible, an alternative must be found.
- Preliminary recommended slope angles for the pit were based on limited geotechnical data and some variation may occur.
- A suitable source of potable water has not yet been identified. Two options were highlighted but the suitability of these options has not been confirmed.

- The storage of sufficient water for the mill at start-up may be problematic. It has been estimated that nine months of TSF catchment runoff are required to supply the first three months of process water requirements. If mine operations start immediately before winter, then even more water will be required to provide sufficient supply during a time when there is limited runoff and some water is locked up as ice. Alternative sources of start-up water should be investigated.

18.3 ELECTRICAL SUPPLY AND DISTRIBUTION

18.3.1 Summary

This section provides a preliminary assessment of the electrical power, control, and communications needs of the project based on a design plant processing rate of 6,000 tpd. Both capital and operating costs are included in the discussion.

Electrical loads allow for a 6,000 tpd processing plant, ancillary buildings, and reclaim and fresh water pumping. It has been assumed pit equipment will be diesel powered.

Power supply to the project will originate at an existing SSE (Stredo Slovenska Energetika) 22 kV power line approximately 1 km from the process plant site. From this line tap, a new 22 kV three-phase overhead line will extend west and south to the proposed main Kremnica Gold plant substation (see the layout drawing in Figure 18.9).

The Kremnica substation will consist of two similar primary transformers, each with the capacity to serve the load alone on a standby rating. The site distribution voltage will be 6.6 kV, which will be utilized to transmit power to the various load centers around the site.

Electrical services generally will be to current IEC standards and will include grounding, lighting, security, small power, welding, heating, process power, process control, instrumentation, fire protection, and on-site and off-site communications systems.

Small electrical loads will be supplied at 230 V, 50 Hz, single-phase, while larger process loads will be supplied at 400 V, 50 Hz, three-phase. Only the SAG and ball mill motors will be supplied directly at 6.6 kV, 50 Hz, three-phase.

18.3.2 Options for Power Supply

Given the relatively short project life and the relatively small load (7.5 MVA), it has been assumed that the most practical and economical source of electrical power supply will be from the local electrical utility. Despite the local existence of geothermal energy sources and small hydroelectric installations, it is unlikely that sufficient power for the Kremnica load could be developed quickly or economically.

The Slovakian power market liberalization was completed in January 2005 in accordance with requests from the EU who encouraged an end to the former State monopoly. Currently Slovenske Elektrarne (SE), the primary national generator of electricity, has been privatized with a majority interest being taken by ENEL of Italy. SE generates about 85% of Slovakia's power needs, with about 57% of capacity coming from nuclear sources, 22% from hydro, 11% from thermal, and 10% from renewable sources. The remaining power comes from the regional distribution companies and IPPs serving large industrial customers.

Power transmission throughout Slovakia is operated by Slovenska Elektrizaena Prenosova Sustava (SEPS), formerly part of SE. SEPS is responsible for operating and maintaining the main 400 and 220 kV electrical grids

Three regional distribution utilities are responsible for transmission and distribution to customers. In the central region of Slovakia, the distribution utility is Stredo Slovenska Energetika (SSE), the company that would supply the Kremnica project. SSE was partially privatized in 2002 with Electricité de France (EdF) taking a 49% ownership interest.

Although, in theory, the Slovakian power market is open to customer selection of supplier, it is recommended at this stage to proceed with SSE as the most appropriate power contractor.

Tournigan may wish to carry out further studies on alternatives as the project advances. The three possible alternatives are, in order of likely importance: (1) choice of a generator other than SE, although SEPS and SSE would still have to transmit as required over their wires, (2) hybrid supply using small hydro, and (3) hybrid supply using small geothermal power.

18.3.3 Electrical Power Supply

Electrical power supply will be provided via a 22 kV connection to the SSE distribution grid at Ludvika approximately 1 km northeast of the proposed process plant. Note: SSE has been requested to confirm the adequacy of this service to supply the Kremnica load (peak demand about 7.5 MW). The alternative would be a 110 kV service from the transmission line that supplies the town of Kremnica and runs approximately east-west about 3 km south of the proposed process plant. No response has been received at the time of writing this report.

The Kremnica electrical service point will be located at the Kremnica substation, and the 22 kV line to Kremnica will be built, maintained, and owned by SSE. SSE revenue meters will be located at the service point. Alternatively, Kremnica Gold may build this line to SSE standards and turn it over to SSE upon completion. This option is not favoured unless SSE agrees to take responsibility for right-of-way acquisition and related legal and social issues.

Power line costs have been estimated by SSE based on the cost of similar lines in the region; however, no site inspection or survey has been carried out to determine land ownership, timber lot licenses, access, or clearing costs.

It should be noted that the tariff to be charged by SSE is composed of two parts: the first part is fully regulated by Urad Pre Regulaciu Sietovych Odvetvi (URSO, the national government regulator) and charges are fixed; the second covers SSE's distribution and service costs. For this type and size of load (Key Account), the second part is fully negotiable. Further, on the SSE side, there are myriad billing options, including the length of contracted energy and capacity amounts, and time-of-use agreements. Also of note are cost penalties (under varying circumstances) of under- and over-consumption for both energy and capacity calculated against contracted amounts.

Capital costs have been included in the cost estimate. Operating costs are based on likely consumption and load factor, and the assumption that SSE's negotiated Key Account rate will be in effect.

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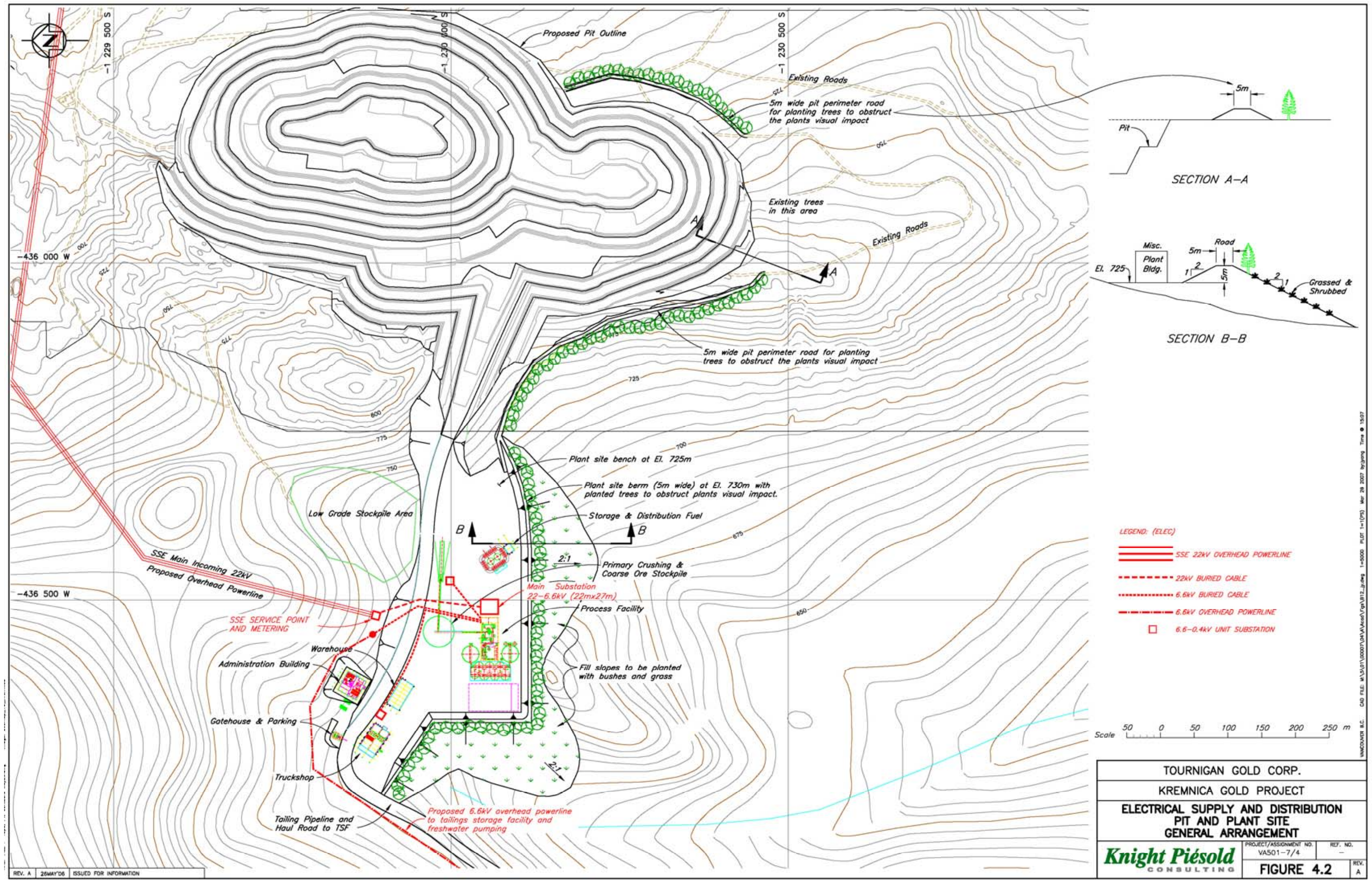
Figure 18.9: Electrical Supply and Distribution – Pit and Plant Site General Arrangement



18.3.4 Kremnica Gold (Kg) Substation

At the Kremnica plant site, a substation will be established consisting of two identical three-phase outdoor power transformers complete with primary isolation and protective devices. Check metering may be installed (by Kremnica Gold) if desired. Each of the main transformers will have sufficient capacity to carry the plant load at its forced cooling rating. The main secondary outdoor bus will transfer power to the main 6.6 kV switchgear line-up located indoors in an adjacent electrical room in the mill building. This switchgear will also distribute power at 6.6 kV to the various plant load centers. Refer to the plant single-line diagram in Figure 18.10.

18.3.5 Electrical Distribution

Power will be distributed to main load centers from the KG substation at 6.6 kV, three-phase, 50 Hz via cable feeders mounted on ladder trays buried underground, or on overhead lines as appropriate. Between the mill building and the shop complex, a 6.6 kV feeder will run in a buried duct; and from the shop complex, 400 V will be distributed to the warehouse and fuel storage and distribution area. A second feeder will supply the administration building and gatehouse. Main load centers in the process plant will be in the grinding and reagent areas. In addition, 6.6 kV overhead lines will service the crushing building, as well as the fresh and reclaim water pumping systems. The main 6.6 kV load centers will each consist of an outdoor transformer and secondary 400 V distribution switchgear. The mill will have dual transformers feeding into one electrical room. Transformers (6600 to 400 V) will be outdoor-rated epoxy cast type. Switchgear at 6600 and 400 V, motor control centers at 400 V, low-voltage lighting and power panels, and AC and DC power supplies will be located in electrical rooms near their respective loads.

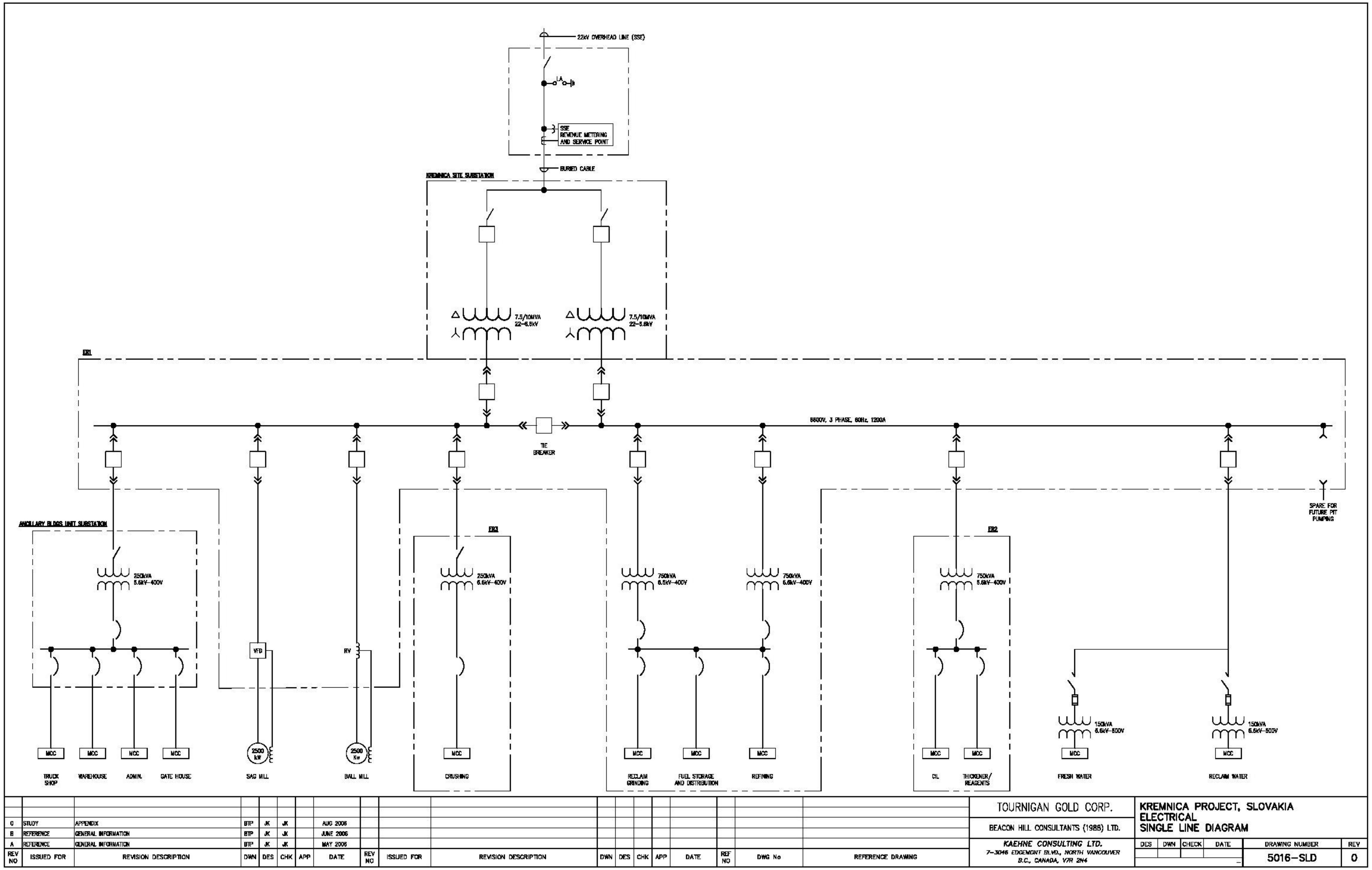
18.3.6 Process Drive Motors

High-efficiency process drive motors will be totally enclosed and fan cooled (TEFC). These motors have been priced as part of the mechanical equipment and are not included in this portion of the cost estimate. SAG and ball mill motors (large, special motors) have been quoted by GE, complete with start/run equipment, and are included in this estimate.

18.3.7 Grounding

The main substation will be provided with an overall ground grid, gradient control mats where required, and structure and fence grounding systems. Overall ground resistivity will be limited to 1 ohm to ensure acceptable touch-and-step potentials. If necessary, a remote ground electrode will be used. All buildings and major outdoor structures will be connected to a buried grounding system with a maximum resistance to ground of 5 ohms. This may be achieved with grounding rods or with one or more deep wells with copper piping installed.

Figure 18.10: Kremnica Electrical Single Line Diagram



18.3.8 Lighting

Lighting will conform to local statutory requirements. Highbay discharge lighting will be used in process and service buildings with a mounting height of 6 m or more. Lowbay vapour-tight discharge lighting will be provided in other process and service areas, fluorescent lighting in offices, control rooms, and similar. Exit lights will be provided where required. Emergency lighting will provide minimal lighting for safe egress in the event of a power failure. Outdoor lighting will be provided where required from fixtures mounted on building exteriors.

18.3.9 Fire Detection and Suppression

POC (products of combustion) and ROR (rate of rise) detector heads will be installed in transformer rooms, electrical rooms, and control rooms. No sprinkling is permitted in these areas. Suppression systems will be either CO_2 or halon replacement, manually operated.

18.3.10 Heat Tracing

The tailings line will be insulated and since it is a gravity line which will drain should there be a power outage, no heat tracing has been included.

18.3.11 Security

Surveillance/security systems have not been included in this report. It is assumed that all site security systems will be coordinated and monitored at a central location such as the guardhouse or mill control room.

18.3.12 Process Control / Instrumentation

The main elements of the process control equipment will be controlled/monitored through a 100% redundant PC-based HMI (human-machine interface) similar to Wonderware's FactorySuite, and will be centralized at a small control room in the crushing building. Another control room will be located in the mill building. Control will be PLC-based and will not have manual override.

Conveyors will have standard control devices: emergency pull cord switches, belt misalignment switches, and speed and plugged chute switches where applicable.

The SAG mill feed conveyor will have a weightometer. Process water tanks will have continuous level monitoring with ultrasonic level transmitters.

The thickener will have a sludge level detector. Monitors for pH and automated lime addition will be installed at various points in the process.

18.3.13 Space Heating

Space heating will be provided by a combination of propane-fired hot water boilers, electric fan heaters, and, in offices and small rooms, electric baseboard heaters. No capital cost has been allowed in this report for any space heating equipment; however, electrical capacity and supply costs have been included.

18.3.14 Plant Site Communications

An allowance has been made for an in-plant intercom system. This will be a self-contained system with multi-channel selective calling.

Allowance has been made for telephone communications to the Kremnica project site.

18.4 INFRASTRUCTURE

18.4.1 Property Access

The property can be accessed from the town of Kremnica via a road network that also provides access to villages within the area. An access road from the main highway south of Kremnica to the mine will be constructed, as shown on Figure 1.2, Section 1. This will prevent all mine-related vehicle traffic, with the exception of cars and small, half-ton trucks, from travelling the roads close to the town of Kremnica and surrounding villages. In addition, it is planned that Kremnica Gold a.s. will construct an access road to the bentonite plant, thereby preventing plant-related traffic from travelling through Lúĉky village. These roads are shown on Figure 1.2.

18.4.2 Transportation of Doré Bars

The transportation of the silver/gold doré bars does not pose a significant concern. Doré bars will be transported to Bratislava by a security truck specially designed for carrying small value loads. Security personnel trained for this purpose will travel with the truck.

18.4.3 Water

The initial water supply for the process water will be obtained from the Heritage adit. Once in the system, this water will be recirculated from the process plant to the TSF and back to the process plant. Approximately 1350 m^3 per day of makeup water will be required from either the Heritage adit or from wells located next to the mining operation and specifically drilled for that purpose.

18.4.4 Fuel Storage

Fuel storage requirements at the site have been estimated at 500,000 liters based upon an average one-month consumption. Access to the site is available at all times, thus delivery of fuel would be on a regular schedule.

18.4.5 Explosive Storage

Minimal explosives will be stored on site since access to explosives in the area on a regular basis is feasible.

18.4.6 Warehouse/Shops

The warehouse will be 20 m wide x 42 m, and will include offices. The maintenance facilities will have two service bays, a wash bay, a tire repair bay, small truck repair bays, offices, and a shop area.

18.4.7 Administration Building / Security and Gatehouse

The administration building consists of a two-storey building designed to provide sufficient area for the projected staff and associated persons.

The security building will have sufficient facilities to store emergency vehicles and serve as a gatehouse.

Figures 18.11 through 18.17 outline the proposed surface ancillary facilities.

Figure 18.11: Administration / Mine Dry – General Arrangement Plan (Sheet 1 of 2)

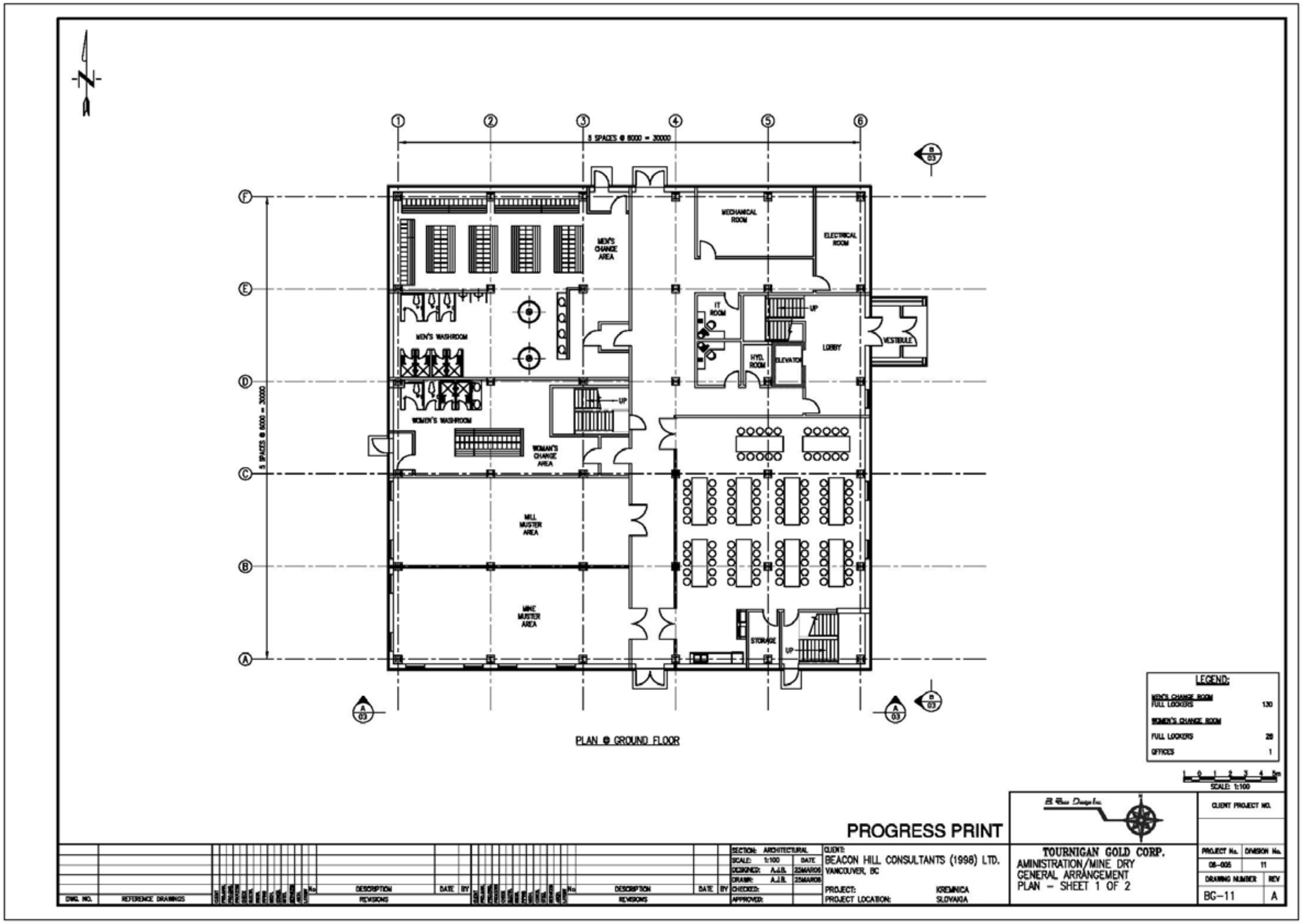


Figure 18.12: Administration / Mine Dry – General Arrangement Plan (Sheet 2 of 2)

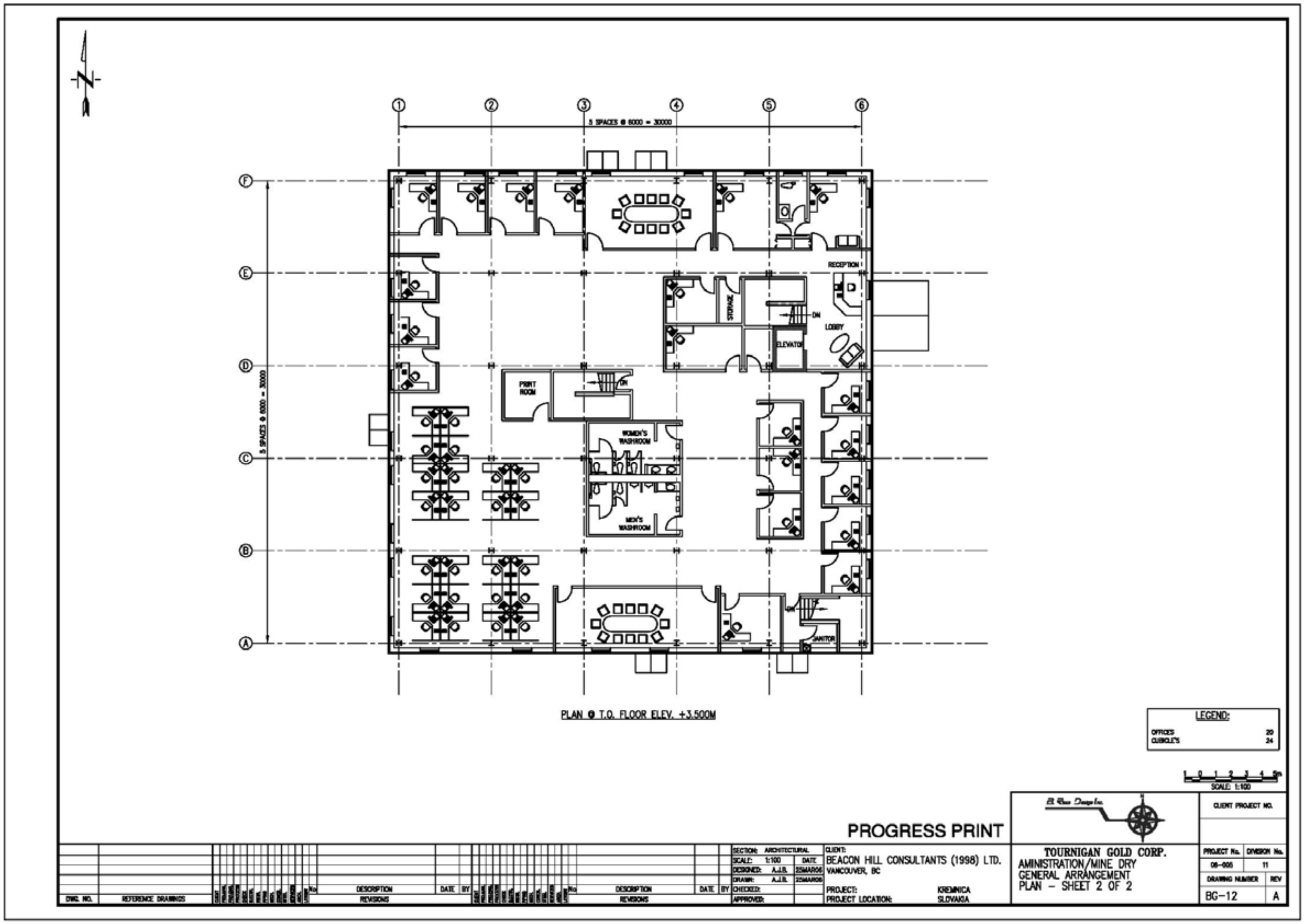


Figure 18.13: Administration / Mine Dry – General Arrangement Elevations

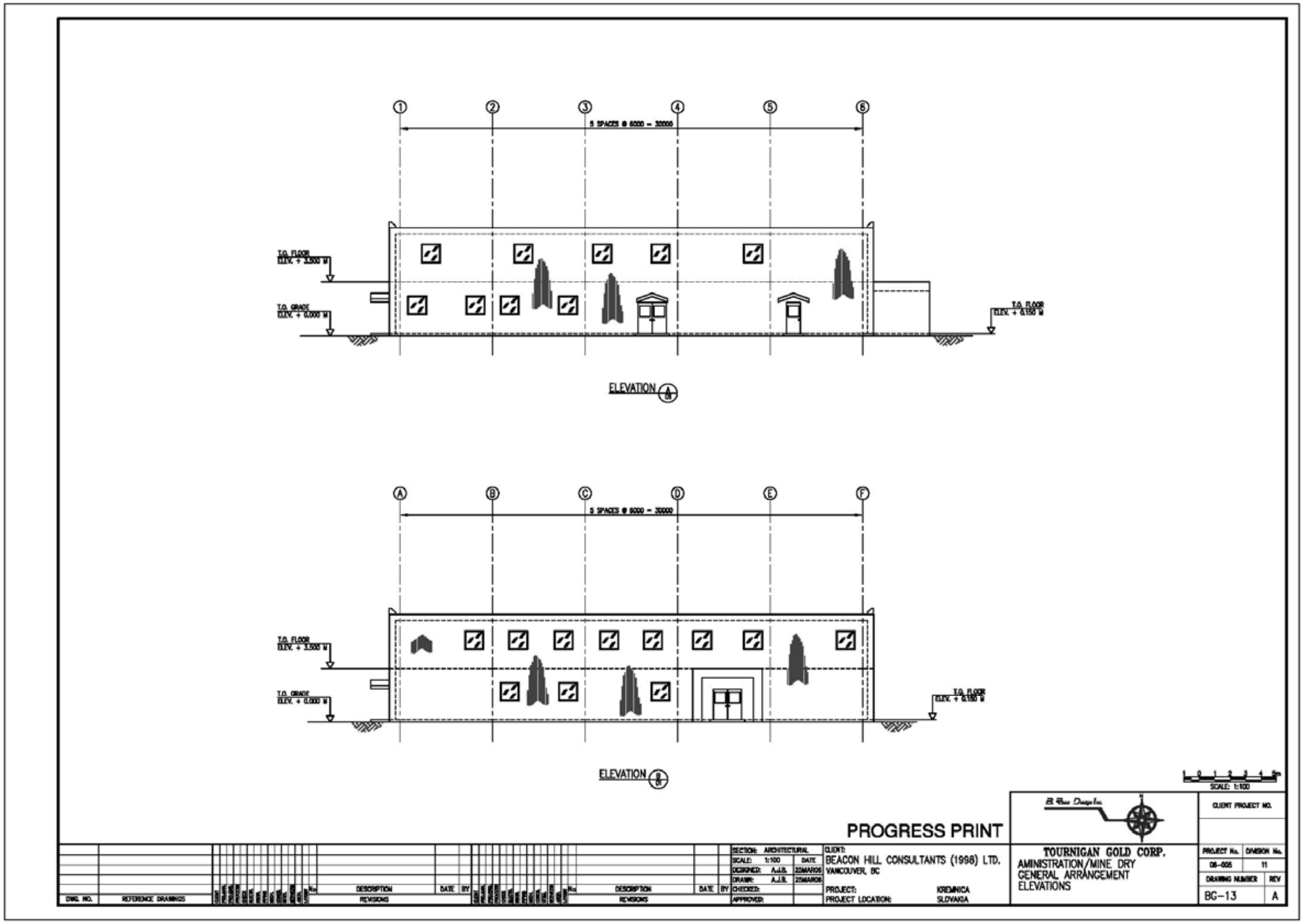


Figure 18.14: Truckshop Maintenance Complex – General Arrangement Plan and End View

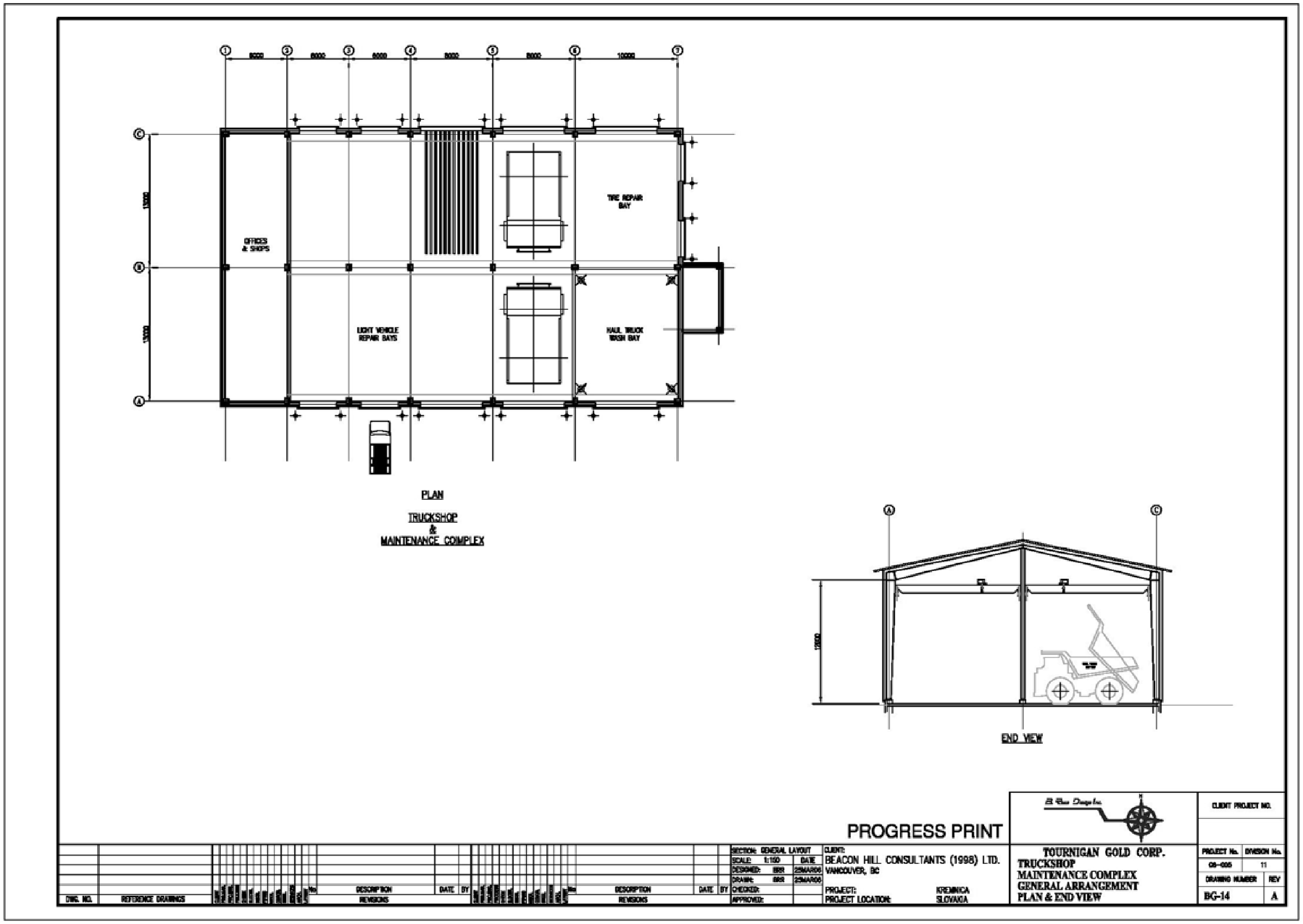


Figure 18.15: Warehouse – General Arrangement Plan and End View

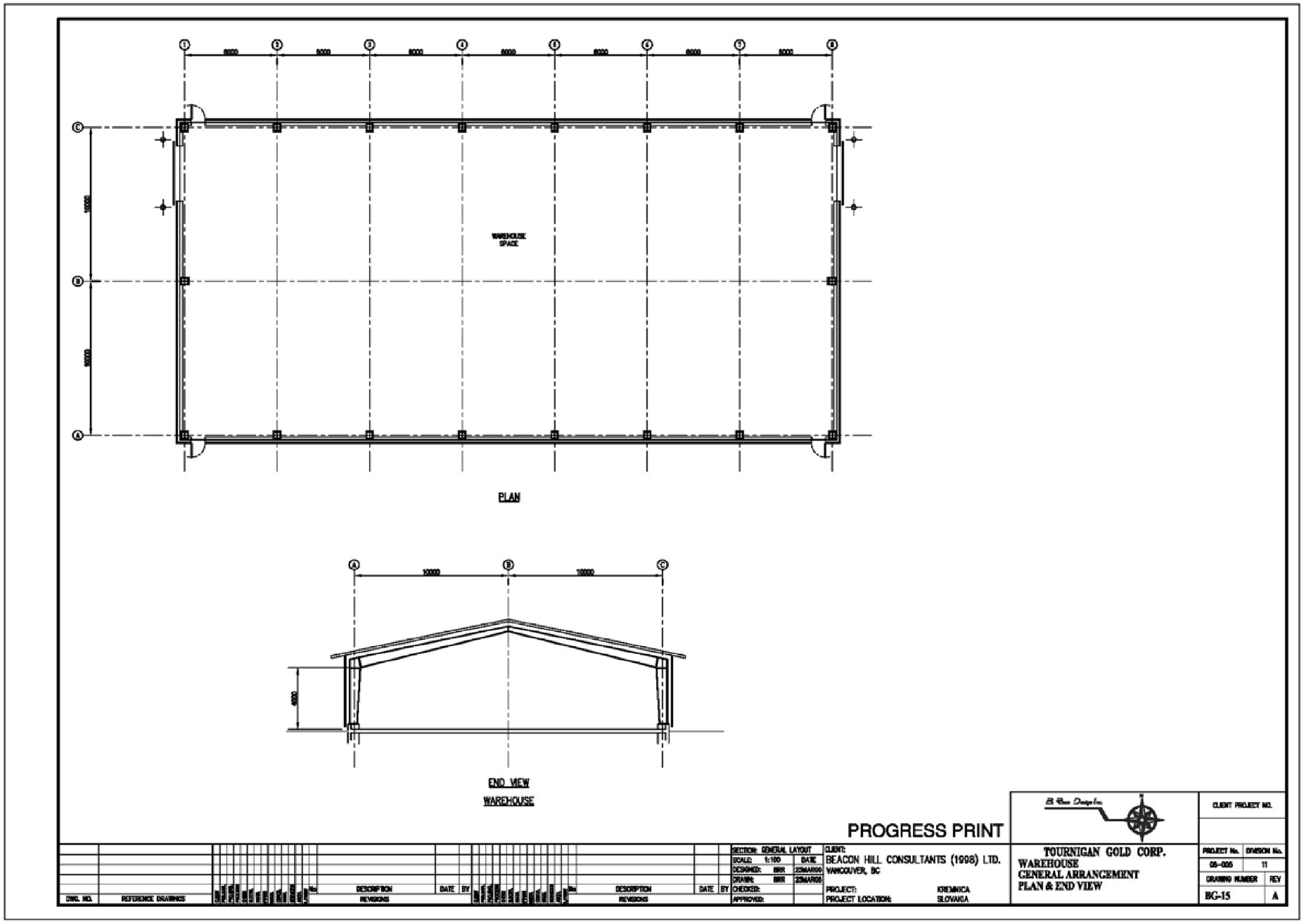


Figure 18.16: Security Gatehouse – General Arrangement Plan and Elevation

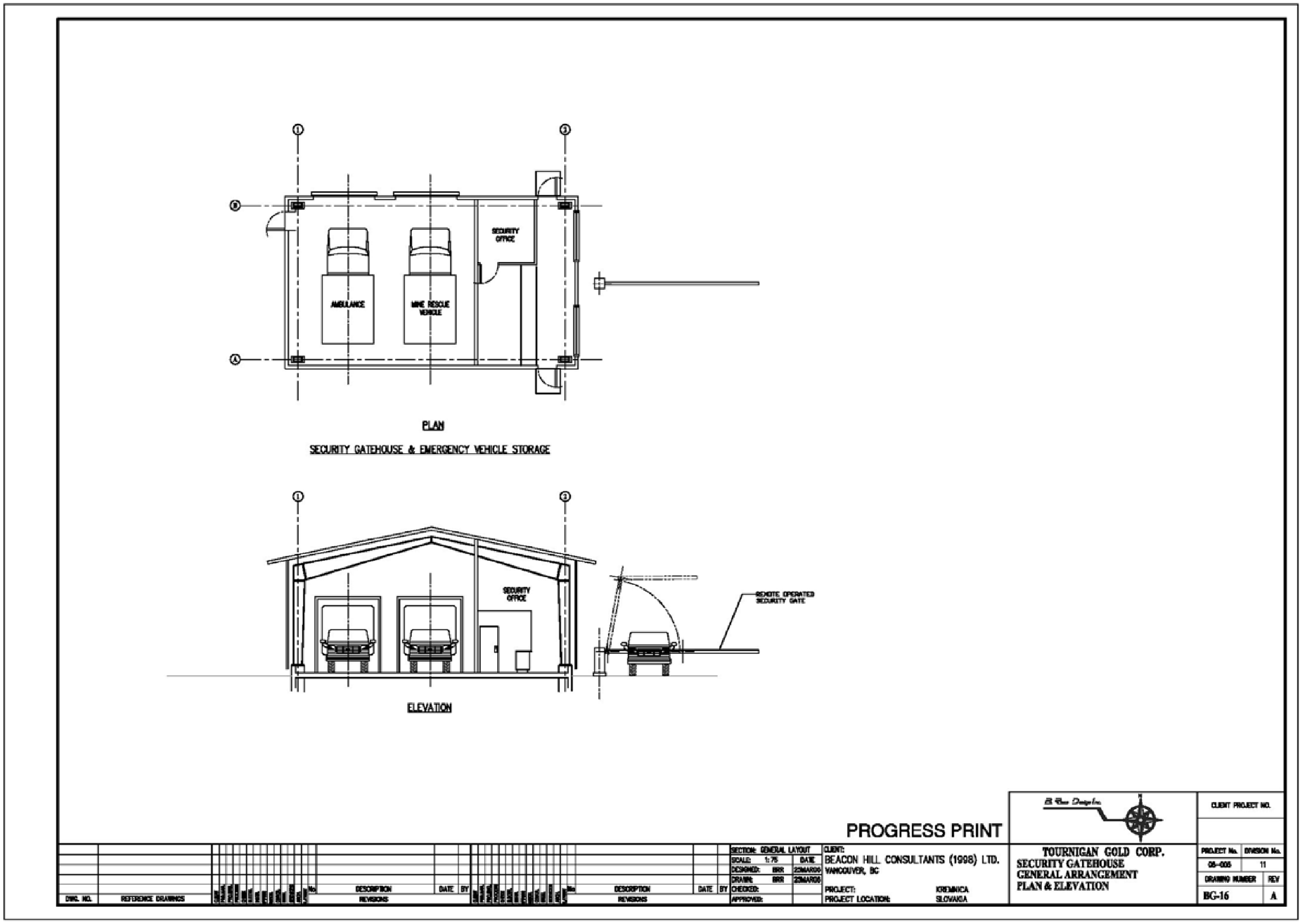
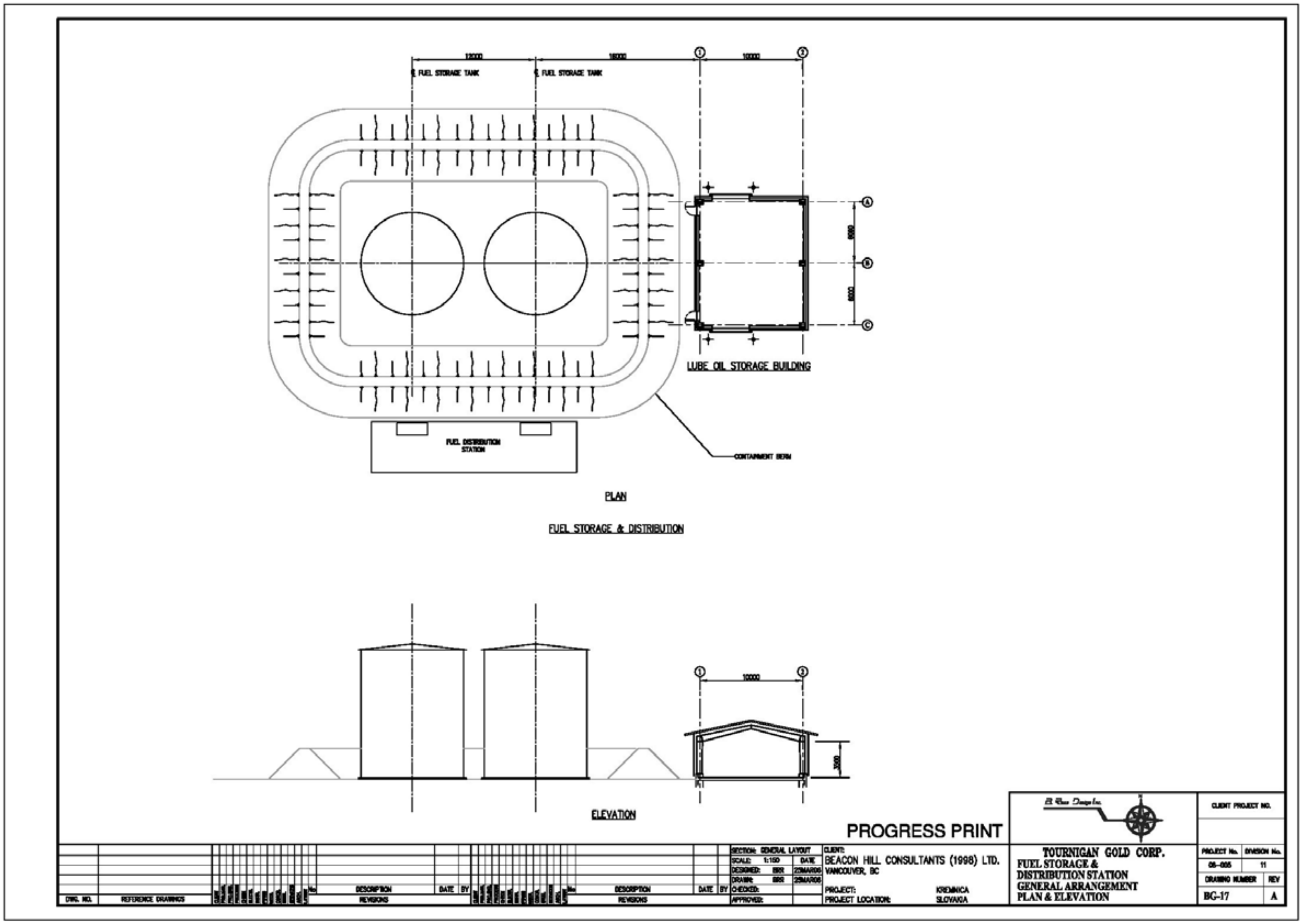


Figure 18.17: Fuel Storage and Distribution Station – General Arrangement Plan and Elevation



18.5 ENVIRONMENTAL OVERVIEW

18.5.1 Introduction

The Kremnica region of Slovakia is characterized by a mountainous topography with narrow valleys and steep hillsides. The climate is generally cool with an average high temperature of 25°C in July and August and an average low temperature of –8°C in January. Precipitation is moderate, with an average annual precipitation of 686 mm. June is the wettest month, receiving an average of 85 mm of rain.

The project is situated within the Hron River watershed, which has been described by the World Wildlife Foundation as ecologically stable, but moderately affected by fragmentation. The Hron River flows into the Danube River at the border of Slovakia and Hungary.

The region suffers from high unemployment, which reached 25% in 2002.

18.5.2 Permitting Process

Environmental permitting in Slovakia is guided by the *Environmental Impact Assessment Act* of the Slovak Republic, published in 1994. There are five primary steps to the permitting process:

1. Notification, whereby the proponent summarizes the proposed project and available information in a submission to the Ministry of Environment. This information is circulated to affected municipalities and relevant central, regional, and local authorities.
2. Scoping, whereby the authorities and municipalities outline the extent of the environmental impact study required.
3. The Environmental Impact Statement is then prepared, including the collection of baseline environmental information, the qualitative and quantitative identification of possible impacts for each proposed activity, and potential mitigation measures.
4. This document is then circulated to the relevant authorities and municipalities, and public hearings are held in affected municipalities.
5. The Ministry of the Environment then issues the Final Statement, with the final recommendation.

The purpose of the *Environmental Impact Assessment Act* is to "ensure the procedure for the complete expert and public assessment of planned constructions, facilities and other activities…before the decision on their permission." Under Annex 1 of the Act, a Preliminary Environmental Report must be prepared, the purpose of which is:

1. to identify, describe and evaluate direct and indirect impacts of the activity on environment
2. to determine measures to prevent or mitigate pollution and damage to the environment
3. to explain and compare the advantages and disadvantages of the submitted preliminary environmental study including its alternatives, in comparison also with the State if the activity would not be provided.

Kremnica Gold submitted a Preliminary Environmental Report to the Slovakian Ministry of Environment in December 2005, thus triggering the permitting process described above.

18.5.3 Environmental Observations

The current environmental and social-economic conditions have been assessed in the project assessment area to identify key issues of potential environmental or social significance as identified through public consultation and to determine the studies that will be undertaken in preparation of an Environmental Impact Statement.

The findings were based on desktop and preliminary field studies that were undertaken in preparation of a Preliminary Environmental Report (PER) which was submitted to the Ministry of Environment in December 2005 in conformance with the requirements of the Slovakian Environmental Assessment Act (No. 127/1994)

The project assessment area referred to in the PER section includes the biophysical and human environment that will be directly affected by mine operations (Area of Mine Operation) and a broader regional area that may be directly or indirectly affected through environmental transfer pathways or social impacts (Project Assessment Area).

18.5.4 Rock Characterization

18.5.4.1 Introduction

This report contains the first phase of drainage chemistry predictions for the Kremnica project. It is based primarily on existing information, geochemical static tests known as acid-base accounting (ABA), and total element contents. Because Tournigan Gold and its Slovak subsidiary are not responsible for past environmental issues, it is critical that the background chemistries of rock, tailings, soils, and water are well documented. Anything overlooked could become Tournigan's responsibility. Potential human-health issues, like mercury used in the past, should carefully be characterized. This report contributes to this task by including samples that are currently oxidizing and leaching.

For this study, 168 small-scale rock samples were collected from (1) reverse-circulation chips from drilling in 2005, (2) older, damp core from Argosy drilling up to ten years old, (3) waste dumps around the existing Šturec pit, and (4) outcrops in and around the existing Šturec pit. The samples were grouped according to low, medium, and high contents of pyrite and ferric-iron oxyhydroxides ("pyrite/feox"), with eight of the samples having no identification or pyrite/feox group. Other groupings were made according to source, such as older core and hangingwall / footwall / ore. All 168 samples were analyzed for expanded standard Sobek acid-base accounting (ABA), and for total element contents using X-ray fluorescence and ICP after four-acid digestion.

18.5.4.2 Results of Acid-Base Accounting

The "paste pH" values, measured in a mixture of pulverized sample and water, ranged from 3.0 to 8.2 in the 168 samples, with a mean and median of 5.7 and 5.8, respectively. Thus, many samples were already acidic at the time of analysis.

Most of the measured total sulphur in the Kremnica rock samples above 1 wt-%S was acid-generating sulphide, whereas below 1%S, leachable sulphate and other sulphur species became dominant. Mass-balance calculations for the sulphur species typically showed good agreement within the general analytical accuracy of 10%. Nevertheless, any "missing" sulphur species was conservatively assumed to be acid-generating sulphide. Ranges and means of sulphide and total iron showed that the three general rock categories (low, medium, and high pyrite/feox) cannot be reliably distinguished on the basis of pyrite or iron.

Bulk neutralization potential (NP) was variable, with no trends among the low, medium, and high pyrite/feox groups. Below a measured NP of 10 kg $CaCO_3$ equivalent/tonne, acidic paste pH values were encountered, indicating at this preliminary stage that 10 kg/t were typically "unavailable" for neutralization and thus had to be subtracted from measured values. However, a few samples showed their unavailable NP values were up to 75 kg/t, so this issue of unreactive NP is not yet resolved. For most samples, correlations suggested that NP values above 10 kg/t were generally explained by impure calcite ($CaCO_3$), with the highest NP values explained by dolomite ($CaMg(CO)_3$).

Based on net balances of acid-generating and acid-neutralizing capacities, more than 80% of the samples were expected to become net acid generating at some point, and several were already acidic. Even when separated as low, medium, and high pyrite/feox groups, at least 80% of all groups were predicted to be net acid generating. When grouped by spatial and geologic location, roughly 80% of the groups were still net acid generating. The exceptions were existing benches and outcrops around the Šturec pit, where one-third of the samples had already apparently exhausted most of their acid-generating capacity. Based on these initial small scale ABA accounting results, pervasive ARD would be expected from mine walls and rock piles at Kremnica. However, large scale ARD has not been observed on the site. Therefore, a program of large samples (~1 tonne each) on leach pads has been initiated to test the acid generating potential of seven rock types.

18.5.4.3 Results of Total Element Analyses

Total element analyses showed that Kremnica rock frequently contained elevated levels of silver, arsenic, bismuth, cesium, mercury, sulphur, antimony, selenium, thallium, and tungsten. Correlations with sulphide and NP were not helpful in inferring the potential extent of metal leaching. Kinetic leaching tests were recommended.

18.5.4.4 Recommendations

This first phase of rock drainage prediction for the Kremnica project has shown that acid rock drainage (ARD) is likely, and that accelerated leaching of some elements even at near-neutral pH is possible. However, this information is not sufficient to estimate the intensity of ARD or metal leaching, which may affect waste- and water-management options and would influence the choice and economics of any water treatment. If intensities, options, and costs are concerns, then the following work is recommended:

1. Six laboratory-based kinetic humidity cells should be operated for approximately one year. These cells contain approximately 1 kg of sample and are repetitively leached under controlled conditions on a weekly basis. Samples with ranges of sulphide, NP, and metals should be tested. The results will include bulk rates of acid generation, acid neutralization, and elemental leaching, and will assist in resolving uncertainties such as unavailable NP.

2. Four on-site kinetic leach pads should be built, each containing approximately one tonne of rock. These larger tests are leached naturally under on-site conditions, providing critical scale-up data for the 1 kg laboratory cells.
3. Periodic, detailed water chemistry surveys of puddles and seeps should be conducted around the mine site and underground to highlight the focal points of water quality issues. This would assist Tournigan in delineating the existing water quality problems for which they are not responsible. The results would also assist in predicting future water chemistry by representing full-scale geochemical kinetic tests. These surveys should include field pH, field conductivity, field temperature, and flow rate with selected locations sampled for full laboratory analyses of cations, anions, and general parameters.
4. Detailed drainage chemistry predictions for sections of mine walls and any rock piles require estimates of their contents of sulphide, NP, and metal levels. If such predictions are needed, additional static tests, like those in this report, will be required, and the results should be block-modelled using the Kremnica assay database. The resulting maps of net acid-generating and net acid-neutralizing zones may assist in cost-benefit assessments of options for waste and water management.
5. This report does not address the drainage chemistry of tailings. Parallel work on tailings should be conducted so that all water chemistry impacts of future mining can be estimated.

18.6 MANPOWER

Although the country has a rich mining history, some personnel will need to be trained to fill the jobs created by the Kremnica project; however, because it will be an open pit operation, and there are many quarries in the local area and throughout the country, experienced labour requiring minimal training should be available. It is assumed that process plant personnel will require training.

The manpower estimates for operations are based upon similar mining operations, the type and number of pieces of equipment, and the general operations of the facilities.

The estimate of persons employed directly by the project is 160 persons. A breakdown of personnel is shown in Tables 18.3, 18.4, 18.5 and 18.6.

In addition to direct employment, the project will create approximately 3 to 4 times more indirect jobs through associated service providers and mine suppliers.

Table 18.3: Breakdown of Mine Operations Staff

Mine Operations – Staff	
Category	**# Required**
Foreman	2
Drill & Blast foreman	2
Mechanical foreman	2
Senior Engineer	1
Engineer	2
Surveyor	2
Senior Geologist	1
Geologist	2
Sampler	2
Clerical	1
Labourers	4
Total	**21**

Table 18.4: Mine Operations Labour Breakdown

Mine Operations – Labour		
Activity	**Category**	**# Required**
Drilling	Drill Operator	2
	Helper	2
Blasting	Blaster	1
	Helper	2
Loading	Operator	2
	Helper	2
Haulage	Truck Drivers	8
Roads / Dumps	Dozer Operator	2
	Grader Operator	2
Maintenance	Lead Mechanic	2
	Mechanics	2
	Serviceman	2
	Labourers	2
Total		**33**

Table 18.5: Breakdown of Administration Staff

Administration Staff	
Description	**# Required**
Staff	
General Manager	1
Mine Superintendent	1
Mill Superintendent	1
Maintenance Superintendent	1
GM Secretary	1
Chief Accountant	1
Purchasing Agent	1
Personnel / Office Manager	1
Environmental Coordinator	1
Safety Director	1
Safety Trainers	2
Accountant/Purchasing/Administration Assistants	4
Subtotal	**16**
Surface Operations	
Equipment Operations	2
Utility Personnel	2
Maintenance	4
Misc. Labourers	4
Subtotal	**12**
Total	**28**

Table 18.6: Process Plant Labour Breakdown

Process Plant	
Category	**# Required**
Supervisory & Technical	
Senior Metallurgist	1
Metallurgist	1
Operation General Foreman	1
Maintenance Foreman	1
Chief Assayer/Chemist	1
Assayers	1
Metallurgical Technician	2
Operations Shift Foreman	4
Mill Clerk / Metallurgical Accountant	1
Subtotal	**13**
Operations	
Crusher Operator	2
Grinding Operator	4
Leach/CIL Operator	4
Carbon Operators	4
Refiners	2
Reagent Mixer	4
Mill Assistants	8
Day Crew	4
Laboratory Labour	6
Subtotal	**38**
Maintenance	
Leadhand Millwright	2
Millwrights	6
Welders	3
Pipe Fitter	3
Electrician	3
Instrument Technician	2
Maintenance Labour	9
Subtotal	**28**
Total	**78**

18.7 CAPITAL COSTS

18.7.1 Introduction

The capital costs for the surface facilities have been estimated by Merit Consultants International Inc. with input from all consultants working on the project. Open pit mobile equipment costs have been estimated by Beacon Hill.

18.7.2 Basis Of Estimate

Mine

The cost of mobile equipment is based upon budget vendor pricing of new equipment for a recent North American project and from in-house files. These costs were then adjusted for the major pieces of equipment to reflect the re-sale price in the market at the time of writing. Although there is no certainty, present investigations indicate this equipment will be available. All other equipment prices have been based on the cost of new equipment.

The purchase of trucks has been timed to reflect the requirements throughout the mine life. Consequently, a number of trucks are purchased as ongoing capital because they are not required until Year 4 of operations when waste tonnages increase. To minimize the number of trucks required, every effort was made to balance waste production over the mine life.

The equipment requirements were derived from the proposed mine plan and projected equipment performance criteria. Costs for replacement equipment were included as ongoing capital throughout the mine life.

Surface Facilities

The capital costs for the surface facilities were estimated by Merit Consultants International Inc. based upon the following information:

- project general layouts
- project general arrangements
- project flowsheets
- miscellaneous cost information from Slovakia
- mechanical equipment list
- plant site, access roads and tailings costs by Knight Piésold
- power supply and electrical distribution costs by Kaehne & Associates Consulting
- waste to tailings conveyor design specifications by L.M. Scovell & Associates, Inc.

All costs are expressed in second quarter 2006 US dollars, with no allowance for escalation, interest during construction, taxes, or duties. Rates of exchange are stated as CDN$0.89 to US$1.00 and SKK29.0 to US$1.00.

Quantities are derived from a combination of historical data (process and infrastructure) and contributors such as Knight Piésold Consulting Ltd. (site preparation, tailings and water system), Beacon Hill (mechanical equipment), L.M. Scovell & Associates Inc. (waste conveyors), and Kaehne Consulting (electrical and instrumentation).

An in-country labour rate of $7.50/hr was used, as indicated by Beacon Hill. The productivity of the labour force is estimated to be similar to that experienced in North America, provided experienced supervision is included.

The all-in rates are based on the following criteria:

- base labour wage rate
- overtime premiums (but not casual overtime allowance)
- benefits and burdens
- worker's compensation premiums
- travel allowances
- appropriate crew mixes
- small tools and consumables allowance
- field office overheads
- home office overheads
- contractors' profit

Waste Disposal

The capital cost estimate for the waste rock facility includes the work required to prepare the ground for waste dumping and installation of the necessary safeguards for the safety and stability of the waste pile and any facilities located below it. The costs include allowances for a haul road down the valley slope to the top of the first lift and for a groundwater drainage system.

Initial capital costs for the tailings impoundment have been estimated based on the quantities of material required for the starter embankment, using typical Slovak and North American unit costs for excavation, hauling, placing, and compacting in the dam. It is expected that the majority of the material requirements will be obtained from the open pit with local borrow areas used where open pit rock is not suitable or not available due to scheduling requirements. Additional costs for the basin liner, filter material, drainage, pipelines, and site preparation have been based on typical costs from other similar projects.

Ongoing capital costs include raising the embankment of the dam in three stages over the remaining life of the project, and purchasing and replacing equipment. Closure costs for both the waste rock facility and the tailings impoundment area have also been included.

A 10% contingency has been added to the estimate to allow for unforeseen capital requirements and overruns.

18.7.3 Cost Summaries

A summary of initial capital expenditures and ongoing capital costs for a 6,000 tpd operation is provided in Tables 18.7 and 18.8, respectively.

Table 18.7: Initial Capital Expenditures

KREMNICA PROJECT
ŠTUREC DEPOSIT
CAPITAL COSTS $(000)s
Base Case Open Pit 6000tpd

Description	Year	2007	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	Total
Full Feasibility Study		2,108												**2,108**
Pre-production Development			3,708											**3,708**
Direct Costs														
Atlas Copco Viper Drill			1,683											**1,683**
Komatsu PC1800 Shovel			3,754											**3,754**
Komatsu WA600 Loader			1,341											**1,341**
Komatsu HD785 Truck			4,638											**4,638**
Komatsu D155 Dozer			1,360											**1,360**
Komatsu GD825 Grader			415											**415**
Komatsu WA320 Loader			238											**238**
Air trac/compressor			255											**255**
Utility/Water Truck			722											**722**
Service truck			306											**306**
Crane truck			204											**204**
Blaster's truck			714											**714**
Light plant			82											**82**
Pick ups			128											**128**
Pump truck			31											**31**
Misc. Shop Equipt (lot)			204											**204**
Spares @ 5% (lot)			1,034											**1,034**
Sub-total			**17,107**											**17,107**
Plant and Surface Facilities														
Crushing			1,792	1,792										**3,585**
Conveying & Coarse Ore storage			1,608	1,608										**3,217**
Conveying Process to Waste			2,542	2,542										**5,084**
Grinding Gravity Concentration and Thickening			3,363	3,363										**6,727**
Pre- Areation and leaching			2,267	2,267										**4,535**
Carbon Treatment and Gold Recovery			902	902										**1,804**
Cyanide Destruction and Tailings Thickener			1,273	1,273										**2,545**
Reagents Handling and Storage			845	845										**1,691**
Process Control			174	174										**347**
Process Building			1,618	1,618										**3,236**
Tailings			6,610	6,610										**13,221**
Service Complex			1,827	1,827										**3,654**
Assay Lab			250	250										**500**
Plant Site Preparation and Site Road			458	458										**917**
Primary Substation and Distribution			3,248	3,248										**6,496**
Water System			250	250										**500**
Communication			39	39										**78**
Fuel storage and Distribution				418										**418**
Sub-total			**29,068**	**29,485**										**58,553**
Indirect Costs														
Engineering and Procurement			3,513	878										**4,391**
Construction Management			1,025	1,025										**2,049**
Construction Indirects			1,714	1,714										**3,428**
Freight			1,123	1,123										**2,245**
Start-up and Commisioning				420										**420**
First Fills and Capital spares				2,554										**2,554**
Sub-total			**7,374**	**7,714**										**15,088**
Total without Contigency		**2,108**	**57,257**	**37,199**										**96,564**
Contingency	10%	$211	$5,726	$3,720										**9,656**
Total Initial Capital		**$2,319**	**$62,983**	**$40,919**										**106,220**

Table 18.8: Ongoing Capital Costs

Description	Year	2007	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	Total
On-going capital														
Tailings Dam						$3,571		$3,653		$6,337				**13,561**
Replacement vehicles 5% of Initial Cap. Cost						$941	$941	$1,972	$941	$941	$941			**6,676**
Reclamation													$4,000	**4,000**
Contingency 10%														
Total On-going Capital						**$4,512**	**$941**	**$5,625**	**$941**	**$7,278**	**$941**		**$4,000**	**24,237**

18.8 OPERATING COSTS

18.8.1 Basis of Estimate

Mining

Project operating costs were estimated based on a production rate of 6,000 tpd and an average stripping ratio of 1.61 (i.e., a total daily production of approximately 15,700 tonnes of material) which includes the low grade (0.5 to 0.75 gAuEq/t) stockpiled ore processed after the higher grade ore. An operating schedule of 2 x 8 hr shifts per day for 302 days per year was incorporated into the mine plan based upon a six-day week and no mine operations on statutory holidays (52 weeks @ 6 days per week gives 312 days, less 10 statutory holidays gives 302 days). This work schedule was determined as a result of public consultation with local peoples who were concerned about mine-related noise at night, Sundays and Statutory holidays.

The overall plan called for cut-off of 0.75 gAuEq/t, above which ore will be sent directly to the process plant. Material grading 0.5 to 0.75 gAuEq/t will be placed on a low grade stockpile to await processing after the higher grade material has been exhausted. An allowance has been made to re-handle this material. Non-acid-generating waste rock will be conveyed to the TSF for use in building the TSF dam, while potentially acid-generating waste will be deposited into the tailings pond. This material will be screened prior to conveying; rock larger than +305 mm will be crushed to -305 mm, in accordance with the conveyor design criteria. An allowance has been made for the cost of crushing, conveying, and depositing the waste at the TSF.

The operating cost estimate is based on local labour rates and productivity factors, as well as local material and supply costs where applicable. Where these were unavailable, North American costs were used. The labour rates obtained covered a broad range of job categories for both hourly rated and staff personnel and included all allowances for shift premiums, statutory social benefit premiums, vacations, medical benefits, etc. The costs for some bulk supply and consumable items, such as fuel and explosives, were obtained from local sources, but items such as drilling supplies, tires, and equipment maintenance costs were estimated from typical North American cost criteria.

Process Plant

The process plant operating costs reflect the metallurgical work completed for this study and are based on Slovakian labour rates. Slovakian costs for consumables available within the country were incorporated into the cost estimate; otherwise, data derived from the international mining cost base (e.g., cyanide and carbon costs) were used. Consumption was derived from the metallurgical work completed as part of this study and from general metallurgical experience with CIL plants of the type and size proposed.

A significant component of the process plant operating cost estimate is related to the hardness of the ore. Most of the power in the plant will be consumed by grinding, and the cost of power becomes a very critical part of the plant operating cost. The estimate derived for the power cost is $0.093/kilowatt-hour. A rate has been assumed of $0.083 per kilowatt-hour based upon an analysis of the rate structure and anticipated consumption and the opinion of Tournigan that a saving can be achieved in negotiations with the power supply company, SSE.

Overheads (G&A) and Owner's Costs

The overhead operations costs were estimated based on the administrative labour expected for this size of operation, miscellaneous surface operations personnel, and expenses such as communication, insurance, and supplies that are general and not included in the mine or process plant costs.

Owner's costs were estimated by Tournigan at $500,000 per year for off-site costs.

18.8.2 Other Costs

Other costs that are deducted from revenue are: (1) fees for waste disposal and tenure fees, and (2) royalties collected by the Slovakian Government. These are described below.

Fees

Under Slovakian regulations, there are a number of fees that could be charged, including fees for inert waste, "other" waste, hazard waste, a mining license, and for road and crusher dust. Investigations have concluded that the only applicable fees to the Kremnica Gold project are the mining license fees, which are based on $172.41/ha for 1,170 ha, and road dust at $172.41/tonne for 100 tonnes per year. These costs have been included in the estimate.

Royalties

The Slovakian Government levies a royalty for mining silver and gold. The royalties for the minerals mined are calculated as follows:

Royalty for the Minerals Mined = (MC/TC) x R x rate (in %)

Where:

MC (costs for mining the minerals) = the expenses (direct and indirect) related to opening, preparing, mining, and liquidating the mining area, as well as the cost of transporting the mineral to the processing plant

TC (total costs for preparing the product from the mined minerals) = MC and other expenses related to processing the minerals

R (revenue for the product prepared from mined minerals) = the revenues for the products sold as well as for products retained for own use

The rate (from 0.1% to 10%) depends on the type of mineral that is mined. For silver and gold, the rate is 5%.

Tournigan has indicated that, through negotiation, the royalty rate can likely be reduced to 3%, and has instructed that this rate be used for this evaluation.

18.8.3 Cost Summaries

The operating cost summary is shown below in Table 18.9.

Table 18.9: Base Case Operating Costs

ŠTUREC DEPOSIT Base Case Open Pit 6000tpd OPERATING COSTS $										
		2008	2009	2010	2011	2012	2013	2014	2015	Total
Description	Cost/tonne	1	2	3	4	5	6	7	8	
Processed Ore		2,100,000	2,100,000	2,100,000	2,100,000	2,100,000	2,100,000	2,100,000	1,529,158	**16,229,158**
Ore	$1.35	$3,128,462	$4,114,880	$3,964,890	$3,722,051	$3,233,714	$3,062,393	$609,001		**$21,835,391**
Waste	$2.03	$4,377,759	$3,626,172	$4,893,265	$9,909,670	$7,898,652	$2,108,250	$74,132		**$32,887,901**
Rehandle Stockpile	$0.10							$840,311	$764,579	**$1,604,890**
Plant	$5.81	$12,209,400	$12,209,400	$12,209,400	$12,209,400	$12,209,400	$12,209,400	$12,209,400	$8,890,525	**$94,356,325**
Tailings & Conveyor	$0.20	$420,000	$420,000	$420,000	$420,000	$420,000	$420,000	$420,000	$305,832	**$3,245,832**
G/A	$0.62	$1,311,422	$1,311,422	$1,311,422	$1,311,422	$1,311,422	$1,311,422	$1,311,422	$954,939	**$10,134,891**
Owners Cost	$0.25	$500,000	$500,000	$500,000	$500,000	$500,000	$500,000	$500,000	$500,000	**$4,000,000**
Total	$10.36	**$21,947,042**	**$22,181,874**	**$23,298,977**	**$28,072,542**	**$25,573,187**	**$19,611,465**	**$15,964,266**	**$11,415,874**	**$168,065,228**
Average cost/tonne ore		**$10.45**	**$10.56**	**$11.09**	**$13.37**	**$12.18**	**$9.34**	**$7.60**	**$7.47**	**$10.36**

18.9 Financial Analysis

The financial analysis Base Case has been prepared using standard discounted cash flow methods to determine the NPV and IRR of the project based upon 100% equity financing and metal prices of US$525/oz Au and US$9.25/oz Ag. The analysis was performed in constant 2006 US dollars, excluding inflation. The results of the analysis are summarized in Section 1.2. Various sensitivities have been completed to determine the effect of metal prices, capital costs, operating costs and exchange rates.

The economic evaluation of a 6,000 tpd open pit operation at the Šturec deposit estimates that, based on an initial capital expenditure of US$106.2 million and the metal prices stated above, the project will generate an after-tax IRR of 13.08% and an NPV of US$75 million undiscounted and US$20.1 million discounted at 8%. Payback of initial capital can be achieved in 4.37 years. The reserves established in this study are 16.3 million tonnes grading 1.40 g/t of gold and 11.08 g/t of silver.

SECTION 19.0 – INTERPRETATION AND CONCLUSIONS

19.1 CONCLUSIONS

The results of this study indicate that the Šturec deposit can be developed and mined with a return of investment on capital expenditures of approximately 13.08%. Certain studies should be completed to evaluate alternatives prior to full feasibility work. Suggested alternate studies are listed in Project Improvements shown below.

The results of this study show lower returns from previous studies for the following reasons:

- The mineable gold and silver grades estimated in this study are approximately 32% lower than the grades estimated in the Preliminary Assessment completed in 2004. The tonnage processed in the 2006 study is some 5% greater than the 2004 study. In addition, the process plant recovery, which was based on metallurgical work completed for the 2006 study, increased by 1% for gold and decreased by 6% for silver.
- The capital costs have increased approximately 71% compared to the 2004 study. The 2004 study reflected an overall compact plant for the mine, plant and waste disposal facilities and was commissioned to ascertain if there was any potential for the Šturec deposit to be viable. The 2006 study, also commissioned to determine if the project is viable, has incorporated local concerns with respect to location of those facilities to ensure acceptance of the project with the local communities. Measures have been incorporated in the 2006 study to mitigate these concerns, which has resulted in increasing the estimated capital costs. In addition, the world economy has significantly improved and placed a demand on all equipment supply and services thus increasing equipment and labour costs.
- Operating costs are very similar to those estimated in the 2004 study.
- Metal prices have risen considerably since the completion of the 2004 study. Metal prices used for the 2006 study are 36% to 54% greater.

19.2 PROJECT IMPROVEMENTS

Below are a number of improvements that could increase the viability of the project:

- Further in-fill drill to ascertain if, to what extent the inferred resources could be upgraded to indicated, and measured categories.
- A number of incentives are available through the European Union and Slovakia that could reduce taxes and provide grants.
- Capital costs are based in part on North American costs. While every effort has been made to use Slovakian costs where applicable, Slovakian construction companies have been reluctant to provide budget costs. It is recommended that the drawings and data in subsequent studies be developed to a level where Slovakian companies will be willing to provide bids on the proposed work. It is expected that the quotes from Slovakian companies will be lower than North American costs, resulting in a reduction of initial capital costs.

- A review of the working week to determine if a 7 day 24 hour per day schedule can be used.
- A review of waste disposal options to determine if there is potential to placing waste close to the open pit and or design of the open pit for waste disposal within the pit.
- Determine if TSF5a has more potential for cost saving than the alternative used within this study site TSF4.
- Relocation of the surface facilities to eliminate the conveyor or any long haulage of waste rock.
- A preliminary study, commissioned by the company, indicated that there may be regions of higher grade within the deposit that may offer opportunities for increased grade in localized regions. This study utilized indicator models and resulted in a number of grade shells based on probability of occurrence. This approach should be considered for future grade modeling in an effort to accurately identify and locate these areas of elevated grade and to possibly increase grade overall.
- Analyze assay results for screen metallics as there may be opportunities for grade increases due to the nature of the coarse gold within the vein.
- A model of the cross-cutting vein structure within the footwall and hangingwall andesites which are currently inferred should result in opportunities for localized high grade areas during mining

The results of the pre-feasibility study show that the Šturec deposit at the Kremnica Gold property has the potential to be mined in a viable manner. Thus, it is recommended that investigations into a number of possible improvements to the project as shown above are completed and a positive result from those investigations prior to a full feasibility be completed on the project. Although a definitive estimate has not been made for these studies, they cab be expected to cost some $500,000 to complete. The total estimated cost, inclusive of a 10% contingency; to complete a full feasibility study is estimated at $2,383,000. A description of the work and breakdown of the estimate is provided in Section 20. It should be noted that some of the condemnation work and drilling has been completed by Tournigan.

SECTION 20.0– RECOMMENDATIONS

20.1 RECOMMENDED WORK PROGRAM

20.1.1 Drill Program[8]

The proposed drilling requirements for the pit geotechnical, TSF geotechnical, metallurgical testwork and geological data for upgrade of the resources from inferred to measured and or indicated for the initial three years of production are shown in Tables 24.1 to 24.4. Some comments and restrictions are provided below:

- The metallurgical holes in Table 20.1 are required for that purpose only and should be drilled PQ.
- The two geotechnical holes in Table 20.3 are required for that purpose only.
- The geological holes in Table 20.2 may upgrade the inferred resources for the first three full years of production and provide samples for metallurgical testwork; hole 14 will also be used to provide geotechnical information for assessing pit slopes, etc. It should be noted while the placement and location of the geological holes are suggested within this report it is the responsibility of on-site geologic staff, who have extensive experience in the area, for the selection, placement, location and orientation of the holes for this program.
- All holes are to be geologically and geotechnically logged.
- The boreholes and test pits required for TSF4 are listed in Table 20.4.
- It is recommended that all holes, with the exception of the metallurgical holes, be HQ.

Table 20.1: Metallurgical Holes

HOLE	EASTING	NORTHING	LEVEL	AZIM	DIP	DEPTH	Vein	Altered Andesite	HW Andesite	FW Andesite	Oxide	Sulphide
A	-435859.49	-1230155.45	734	270	-55	100	y	y			y	y
B	-435815.723	-1229955.492	700	260	-55	150	y		y	y	y	y

Table 20.2: Geological Holes

HOLE	EASTING	NORTHING	LEVEL	AZIM	DIP	DEPTH	Vein	Altered Andesite	HW Andesite	FW Andesite	Oxide	Sulphide
11	-436059.75	-1230130.5	784.75	0	-90	50				y	y	?
12	-436044.25	-1230090.5	779.75	0	-90	75				y	y	?
13	-436024.75	-1230059.5	754.75	0	-90	75				y	y	?
14	-435769.75	-1229969.5	696.75	0	-90	120		y	y		y	?
15	-435769.75	-1229930.5	701.75	0	-90	100		y	y		y	?
16	-435759.75	-1229859.5	691.75	0	-90	100		y	y		y	?
17	-435764.75	-1229819.5	686.75	0	-90	100		y	y		y	y
18	-435754.25	-1229770.5	674.75	0	-90	100		y	y		y	y

[8] It should be noted that much of this work has been completed as of the release of this report.

Table 20.3: Geotechnical Holes for the Open Pit

Drillhole No.		KP06-1	KP06-2	Total
Location		West Wall	Southwest Wall	-
Coordinates - Northing	m	-1,229,850	-1,230,250	-
Coordinates - Easting	m	-435,850	-435,900	-
Collar Elevation (Approx.)	m	680	760	-
Azimuth	degrees	270	240	-
Inclination	degrees	55	55	-
Drillhole Length	m	150	130	280

Table 20.4: Geotechnical Test Pits and Boreholes for the TSF

TEST PITS		
	Coordinates	
LABEL	West	South
FEASIBILITY INVESTIGATIONS		
TP-S4-01	-437,822	-1,234,200
TP-S4-02	-437,760	-1,234,336
TP-S4-03	-437,696	-1,234,434
TP-S4-05	-437,748	-1,234,590
TP-S4-06	-437,888	-1,234,398
TP-S4-07	-437,510	-1,233,590
TP-S4-08	-437,572	-1,234,612
TP-S4-09	-437,514	-1,234,354
TP-S4-10	-437,694	-1,234,036
TP-S4-11	-437,606	-1,234,082
TP-S4-12	-437,436	-1,234,646
TP-S4-13	-437,396	-1,234,744
TP-S4-23	-437,100	-1,235,144
TP-S4-24	-436,986	-1,235,264
TP-S4-14	-437,874	-1,233,616
TP-S4-15	-437,660	-1,233,210
TP-S4-16	-437,668	-1,233,586
TP-S4-17	-437,612	-1,233,740
TP-S4-18	-437,536	-1,233,902
TP-S4-19	-437,314	-1,234,748
TP-S4-20	-437,442	-1,233,984
TP-S4-21	-437,204	-1,235,292
TP-S4-22	-437,112	-1,235,218
TP-S4-25	-437,724	-1,233,820

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BOREHOLES		
	Coordinates	
LABEL	West	South
FEASIBILITY INVESTIGATIONS		
BH-S4-1	-437,664	-1,234,384
BH-S4-2	-437,790	-1,234,670
BH-S4-3	-437,460	-1,234,472
BH-S4-4	-437,526	-1,234,402
BH-S4-5	-437,742	-1,234,280
BH-S4-6	-437,832	-1,234,228
BH-S4-7	-437,656	-1,234,588
BH-S4-8	-437,884	-1,233,510
BH-S4-9	-437,606	-1,233,888
BH-S4-10	-437,622	-1,233,156
BH-S4-11	-437,514	-1,234,138
BH-S4-12	-438,022	-1,234,290

20.1.2 Geological

The geology should be reviewed based on information from the drill program. The geologic interpretations are to be reviewed and revised by site and company geological staff. The on-site assay preparation facility requires review and possible re-certification. QA and QC procedures require review and further controls put in place. An extensive QA/QC program should be commissioned by the company, documented and implemented.

20.2 RESERVE ESTIMATE

20.2.1 Scope of Work

The work consists of supplying the drill hole data analysis, geological modelling, and resource estimation required for the feasibility study.

This will include data input/import, data validation, QA/QC, statistical and geostatistical analysis, geologic and resource modelling, and open pit design.

20.2.2 Overview

Data input includes transferring the drill hole data, topographic data, underground development and specific gravity data. Quality assurance and quality control is to be applied and checked to ensure data accuracy, followed by statistical and geostatistical analysis. Geologic models will be created using interpreted geology supplied by the client. Upon completion, grade models will be created using a variety of techniques and parameters in order to evaluate the methodology that best characterizes the deposit. Historic mining will be incorporated into the complete integrated model for masking out the resources.

20.2.2.1 Modelling Objectives

The following activities and applications should be performed to achieve the stated objectives:

Data transfer and input – Input / import drill hole and trench data, density (SG) data, and topographic data in addition to historic underground development.

Review and analysis – Check and validate data in addition to performing statistical and geostatistical analysis to confirm grade modelling parameters and evaluate alternatives. Evaluate alternative compositing lengths and methods. Determine what additional data (if any) or analyses are required for completion of the project.

SG Analysis and Reconciliation of Recoveries - A detailed study of SG’s in an effort to, as accurately as possible, calculate SG’s, adjust the SG domains, values and adjustment factor. Input of recovery data, particularly from the 2006 RC drilling program for analysis to determine recoveries more accurately.

Geological modelling – Create geologic and alteration models of the deposit and surrounding strata based on revised sectional and plan interpretations supplied by Tournigan from previous studies.

Review and revise the collapse zone and void models along with oxide, mixed zone, sulphide interface models.

Resource modelling – Resource models created with parameters and methodology to be developed and determined following an evaluation of the statistical and geostatistical analysis results. Utilize varying methods and procedures that assist in the accurate characterization of the deposit. Utilize system capabilities for masking the grade models with the appropriate geology. Grade models and volumetric report produced.

Conclusions and recommendations – Final report with conclusions and recommendations.

20.2.2.2 Deliverables

The following are specific deliverables that will be produced at completion of the project:

1. complete geoscience database
2. statistical and geostatistical analysis
3. 3D geologic models
4. spatial prediction (resource) models
5. auditable procedures and results
6. report.

20.2.3 Metallurgical

To bring the pre-feasibility study to the full feasibility stage will require upgrading the existing metallurgical work in conformance with the expectations of the banks/project financiers. The work done to date indicates Kremnica ore presents no particular problems in treatment.

The amount of additional metallurgical testwork required is to satisfy the banks that all areas are amenable to processing with the developed flowsheet, and that variability in ore hardness will not jeopardize plant capacity. A work index and a gravity/leach test on each of the 24 samples will be required with an allowance for some additional tests.

Due to high power costs, the work will include a review of SAG vs. fine crushing to provide a balance between methods and determine the lowest overall cost. It is also thought that fine crushing may be more acceptable in Slovakia.

The work will include:

- variability testwork
- metallurgical testwork
- upgrade flowsheets
- trade-off studies
- General arrangement and sectional drawings

- plant design details
- equipment costing
- capital cost estimate
- operating cost estimate
- report
- site visit.

20.2.4 Geotechnical

- *Project Management* – This task covers the project management work necessary to develop the project. Under this task, cost and budget control for the feasibility study would be provided and reported on a monthly basis. Knight Piésold has an ISO9001-registered Quality System and a quality plan for this project would be prepared under this task.
- *Site Investigations* – The site investigation program will include the supervision and logging of drilling, test pitting, and mapping of surface features to assess the geotechnical and hydrogeological conditions. Representative samples would be collected and a laboratory program specified for testing within Slovakia. The costs associated with laboratory testing are not included in the cost estimate. It is estimated that the proposed geotechnical site investigation program will require approximately 8 to 10 weeks to complete. One full-time geotechnical engineer would be required during that period. Two flights are allowed in the estimate, because typically the maximum duration for a single site assignment before a break is five weeks. At the end of this task, Knight Piésold will produce a report on the site visit and investigation program.
- *Open Pit Geotechnical* – This task includes a geotechnical assessment of the open pit including pit slope design. The work includes assessing the geotechnical core log data and carrying out kinematic and finite element stability analyses to provide recommended pit slope angles and bench dimensions.
- *Plant Site Geotechnical* – This task includes a geotechnical assessment of the plant site foundations to ensure suitable foundations are designed for each individual plant items, and an assessment of the overall stability of the plant site foundation platform.
- *Waste Rock Disposal* – A geochemical assessment of the waste rock is being carried out by others. Once these data are received, their implications for the waste rock disposal methodology need to be determined. This includes:
  - assessment of quantities and scheduling of potentially reactive and non-reactive waste rock
  - design and scheduling of potentially reactive rock co-disposal within the TSF
  - design of external stockpiles for excess non-reactive rock if required.
- *Tailings Storage Facility Design* – Further tailings physical characterization work is required. This work will be based on laboratory testing of tailings samples obtained from metallurgical process testing, including laboratory tests for consolidation and slurry rheology.

  The TSF concept will be optimized based on updated waste production schedules, tailings physical characterization testwork, revised legislative requirements, and economics. A number of potential opportunities were identified in the pre-feasibility study that can be studied during the feasibility study.

The TSF design will be advanced to the feasibility study level through additional seepage and stability analysis; detailing, such as foundation drainage; and an analysis of embankment and impoundment liners, embankment raising methodology, material zoning and specification, potential material sources, etc. A site visit has been included during the feasibility design stage.

Tailings deposition pipeworks design will include design of the tailings, reclaim water, and pump systems. A study of potential thickening of the tailings and consequent savings in reclaim water pumping and impoundment storage volume will be carried out. Details such as outlets for deposition in the TSF and moving the pipelines during TSF raising will be considered.

Initial and sustaining capital quantities and costs for the waste management works will be estimated with greater detail as required for feasibility study.

- *Water Management Plan* – This task includes revising estimates of precipitation and runoff based on new hydrology data, investigation of water supply options, and updating and refining the water balance model. The work will include a review of meteorological and site hydrology data, streamflow measurements, water supply availability, water inputs of precipitation and runoff from the surface facility, groundwater from the mining operations and make-up water, and water outputs such as consumption associated with evaporation, storage in the tailings facility, and water in concentrates. Options for plant site and start-up water supply will be investigated.
- *Mine Site Civil Infrastructure* – This task includes the various aspects of civil infrastructure around the mine site that are not covered by the other tasks. This may include haul road design (geometric layout and pavement design), the location, layout and design of stockpiles for topsoil and low grade ore, and earthworks design for the site.
- *Reporting* – This task includes preparation of a report on the mine waste management aspects of the overall feasibility report.

20.2.5 Electrical

Electrical work during the feasibility stage will consist of:

- Review power supply options and select preferred option. This work would include preliminary negotiation with SSE, leading to preparation of a draft supply contract with costs and timing defined.
- Prepare plant electrical single line diagrams down to 400 V load centers (no block diagrams).
- Prepare conceptual P&IDs superimposed on flowsheets (flowsheets by others).
- Prepare electrical power supply, main substation, and MV distribution layouts that are superimposed on plant and building layout drawings (prepared by others).
- Prepare electrical load list from the equipment list prepared by others. Estimate peak and average demand load and annual energy consumption.
- Prepare written description of electrical scope and installation (in MSWord) for embedding by others into main report.
- Prepare electrical capital and operating cost estimates covering power supply and plant services (in MSExcel) for embedding into main estimate by others.

- Prepare electrical design, procurement, construction, and commissioning schedule (in MSProject) for embedding into main schedule by others.

The work is expected to take four months to complete.

20.2.6 Environmental

20.2.6.1 Environmental and Social Impact Assessment

In April 2006, Kremnica Gold received notice that a full environmental and social impact assessment was required for the Kremnica Gold project. The Terms of Reference, as issued by the Slovakian Ministry of Environment, described the scope of work to include:

- compliance with Annex 3 of the *Environmental Impact Assessment Act*, which prescribes the content and format of the Environmental Impact Statement submission;
- compliance with project-specific Terms of Reference issued by the Slovakian Ministry of Environment, including questions raised by the Hungarian Ministry of Environment regarding cross-border impacts
- compliance with relevant Slovakian regulations, planning, and permitting requirements, as well as European Union directives.

An Environmental Impact Assessment (EIA) designed to conform to these requirements was initiated in May 2006 and is scheduled for submission during the second quarter of 2007. It should be noted that the Kremnica Gold project is the first mine in Slovakia to be permitted under the terms of the current EIA Act and since the entry of Slovakia into the European Union. Consequently, the detailed permitting and planning process is not entirely clear. Further specific project requirements will be determined as part of the environmental impact assessment process.

The scope of environmental work for the EIA has been developed to explicitly address the regulatory requirements detailed above and international standards of good practice. The key activities to be undertaken include:

- characterize baseline environmental and social conditions (including likely future development without the project) in the project area
- provide a detailed description of the project (including alternatives and all life cycle stages) to identify potential direct and indirect project impacts
- assess the significance of project impacts relative to the base case
- identify measures for mitigation (elimination, avoidance, minimization, management) of adverse impacts and optimization of beneficial effects
- identify a preferred project configuration that provides a technically achievable and cost-effective way to minimize adverse impacts and optimize beneficial effects
- develop appropriate systems and monitoring programs to ensure that project-related activities and impacts are well managed.

20.2.6.2 Public Consultation

The scope of work for public consultation includes the following:

- conduct research and provide strategic advice and technical support to the Kremnica Gold community liaison team; conduct public communications activities on an as-required basis
- prepare material and technical support for public consultation as required by law
- prepare a public consultation and disclosure plan, to be submitted as part of the EIS documentation.

The Public Consultation and Disclosure Plan will identify the management, process, and mechanisms through which the company will conduct ongoing stakeholder engagement, consultation, and dispute resolution during the construction, operations, and post-operational stages of the project. The content of this document will be confirmed through the ongoing public relations activities to be undertaken in upcoming months.

20.2.7 Acid Rock Drainage (ARD)

The ARD work proposed for the feasibility study is described below.

- Six laboratory-based kinetic humidity cells should be operated for approximately one year. These cells contain approximately 1 kg of sample and are repetitively leached under controlled conditions on a weekly basis. Samples with ranges of sulphide, neutralization potential (NP), and metals should be tested. The results will include bulk rates of acid generation, acid neutralization, and elemental leaching, and will assist in resolving uncertainties such as unavailable NP.
- Four on-site kinetic leach pads should be built, each containing approximately one tonne of rock. These larger tests are leached naturally under on-site conditions, providing critical scale-up data for the 1 kg laboratory cells.
- Periodic, detailed water chemistry surveys of puddles and seeps should be conducted around the mine site and underground to highlight the focal points of water quality issues. This would assist Tournigan in delineating the existing water quality problems for which they are not responsible. The results would also assist in predicting future water chemistry by representing full-scale geochemical kinetic tests. These surveys should include field pH, field conductivity, field temperature, and flow rate with selected locations sampled for full laboratory analyses of cations, anions, and general parameters.
- Detailed drainage chemistry predictions for sections of mine walls and any rock piles require estimates of their contents of sulphide, NP, and metal levels. If such predictions are needed, additional static tests, like those in this report, will be required, and the results should be block-modelled using the Kremnica assay database. The resulting maps of net acid-generating and net acid-neutralizing zones may assist in cost-benefit assessments of options for waste and water management.

20.3 COST ESTIMATE – FULL FEASIBLITY STUDY

TASK DESCRIPTION		EST. COST CDN$
Project Management		**$50,000**
Drilling, Geological, Metallurgical and Geotechnical		
(Cost estimated by Tournigan)		**$508,000**
Geology		**$10,000**
Resource Estimate and Open Pit Optimization		
Data input, import, transfer	$5,000	
Data analysis and composting	$5,000	
QA & QC	$6,000	
Geological interpretation and modelling	$6,000	
Geological modelling	$4,000	
Resource estimation and modelling	$6,000	
Pit optimization	$4,000	
Pit design	$4,000	
Report	$5,000	
Digitizer support, plotter & output	$3,000	
Total Resource Estimate and Open Pit Optimization		**$48,000**
Mine Planning		
Develop production schedules	$20,000	
Develop overall mine plan	$10,000	
Estimate mining capital costs	$10,000	
Estimate mining operating costs	$7,000	
Report	$10,000	
Total Mine Planning		**$57,000**

TASK DESCRIPTION	EST. COST CDN$
Metallurgy and Process Plant	
Variability testwork: 24 samples	$75,000
Metallurgical testwork	$5,200
Upgrade flowsheets	$7,400
Tradeoff studies	$5,200
GA/sections	$14,800
Plant design details	$13,200
Equipment costing	$5,200
Operating cost estimate	$2,200
Report	$5,200
Site visit	$5,200
Total Metallurgy and Process Plant	**$138,600**

Environmental and EIA		
Description of proponent and proposed development	$40,000	
Description of environmental baseline conditions	$100,000	
Assessment of environmental impact of proposed development	$80,000	
Description of social baseline conditions	$50,000	
Assessment of social impact of proposed development	$70,000	
Selection of and justification for preferred alternative	$50,000	
Provision of advice	$50,000	
Project management and report preparation	$100,000	
Travel, subsistence, consumables, report copies:	$30,000	
Total Environmental and EIA		**$570,000**
ARD		
Kinetic testwork	$80,000	
On-site leach pads	$10,000	
Tailings solids and supernatant	$3,000	
Review of data and report	$25,000	
Total ARD		**$118,000**
Socio-economic (Allowance)		**$50,000**
Public Consultation (Allowance)		**$50,000**

Geotechnical		
Project management	$17,000	
Site investigations	$94,000	
Open pit geotechnical	$28,000	
Plant site geotechnical	$15,000	
Waste rock disposal	$25,000	
Tailings storage facility design	$112,000	
Water management plan	$32,000	
Mine site civil infrastructure	$14,000	
Reporting	$18,000	
Total Geotechnical		**$355,000**
Surface Buildings & Infrastructure		
Maintenance facility	$5,000	
Warehouse	$4,000	
Offices	$3,000	
Dry	$4000	
Fuel storage	$7,000	
Explosive magazines	$5,000	
Access road layouts	$10,000	
Total Surface Buildings & Infrastructure		**$38,000**

Power Supply/Electrical Distribution/Communication		
Review docs to date, establish electrical design criteria	$5,000	
Prepare load list, finalize power supply requirements	$10,000	
Site visit, assess power options, SSE meetings	$20,000	
Bid inquiries - major equipment, design power supply	$15,000	
Costing - power supply and plant	$10,000	
Draft and final report	$6,000	
Total Power Supply/Electrical Distribution/Communication		**$66,000**
Cost Estimates		**$50,000**
Financial Analysis		**$25,000**
Project Disbursements		**$32,000**
Subtotal	$2,165,600	
Contingency 10%	$217,000	
TOTAL		**$2,382,600**

SECTION 21.0 – REFERENCES

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Western Services Engineering Inc., 1997: Report prepared for Argosy Mining Corporation. Preliminary in situ geological resource estimate. Šturec south deposit. Kremnica gold and silver project.

Golder Associates (UK) Limited, December 2005, Preliminary Environmental Report for the Kremnica Gold Project.

European Commission (EC 2004) "Reference Document on Best Available Techniques for Management of Tailings and Waste Rock in Mining Activities (2004)"

Envigeo (2005) "Legislative and Policy Consideration (Draft SK 01)"

European Union (EU 2005) "Directive on the Management of Waste from the Extractive Industries (PE-CONS 3665-05)"

SECTION 22.0 – DATE AND SIGNATURE PAGE

CERTIFICATE OF QUALIFICATION

I, W. Peter Stokes, of 113, 6505 3rd Ave, Tsawwassen Vancouver, British Columbia, V4L 2N1, do hereby certify that:

1. I am a consulting mining engineer with an office at 1400-750 West Pender Street Vancouver, British Columbia V6C 2T8

2. I am a graduate of the Stoke-on–Trente College of Technology, Stafforshire, UK in 1965 with an HND in Mine Engineering.

3. I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia.

4. This certificate applies to the "Kremnica Gold Project Pre-feasibility Study" dated July 5, 2007 prepared for Tournigan Gold Corp., Vancouver, B.C.

5. I have visited the property on a number of occasions the last being from 22 to the 29 October 2005.

6. In the independent report titled "Kremnica Gold Project Pre-feasibility Study", I am responsible for the overall preparation of the report and Sections 1 to 5, 18.1, 18.3 to18.9, 19 and 20.

7. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in draft National Policy 43-101.

8. I have continuously practiced my profession since 1965 and been involved in the development of many similar properties which include the Ormsby Deposit, NWT, Canada, Canatuan Deposit, Canatuan, Philippines, King King Property, Pantukan, Philippines.

9. I was in charge of the project team that worked on this study. I have been involved with the project on two previous occasions, 1998 and 2003.

10. I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report.

11. I have read National Instrument 43-101, Standards for Disclosure of Mineral Properties and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

12. As of the date of this certificate, to my the best of my qualified knowledge, information and belief, this technical report contains all the scientific and technical information that is required to be disclosed to make the report not misleading.

Dated this 5th day of July, 2007.

PROFESSIONAL
PROVINCE OF
W. P. STOKES
BRITISH COLUMBIA
ENGINEER

CERTIFICATE OF QUALIFICATION

I, Garth David Kirkham, of 3178 Three Cedars Drive, Vancouver, British Columbia, V5S-4K5, do hereby certify that:

1. I am a consulting geoscientist with an office at 3178 Three Cedars Drive, Vancouver, British Columbia, V5S-4K5.

2. I am a graduate of the University of Alberta in 1983 a B.Sc. in Geophysics. I am a member in good standing of the Association of Professional Engineers, Geologists and Geophysicists of the Province of Alberta (#M40899), the Association of Professional Engineers and Geoscientists of British Columbia (#30043) and the Northwest Territories and Nunavut Association of Engineers and Geologists and Geophysicists (#L1606).

3. This certificate applies to the "Kremnica Gold Project Pre-feasibility Study" dated July 5, 2007 prepared for Tournigan Gold Corp., Vancouver, B.C.

4. I have visited the property from 22 to the 29 October 2005.

5. In the independent report titled "Mineral Kremnica Gold Project Pre-feasibility Study", I am responsible for the preparation of sections 6, 7, 8, 9, 10, 11, 12, 13, 14, 15 and 17 of the report.

6. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in draft National Policy 43-101.

7. I have continuously practiced my profession since 1988 and been involved in geostatistical and resource modeling studies, both as an employee of a geostatistical modeling, mine planning software and consulting company and as an independent consultant. I have worked on resource estimates for numerous polymetalic projects including Morrison (Au, Cu), McLymont Creek/Newmont Lake (Au, Au, Ag), Adi Nefas and Debarwa Projects (Au, Ag, Cu, Zn, Pb), Niblack and Tahuehueto Project. I have also implemented a computerized grade control system at Eskay Creek and am currently the Project Manager for Hecla Mining's Coeur d'Alene District (Ag, Zn, Pb, Cu, Au) data integration, compilation and deposit modeling project.

8. I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report.

9. I have read National Instrument 43-101, Standards for Disclosure of Mineral Properties and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.

10. As of the date of this certificate, to my the best of my qualified knowledge, information and belief, this technical report contains all the scientific and technical information that is required to be disclosed to make the report not misleading

Dated this 5th day of July, 2007.

CERTIFICATE OF QUALIFICATION

I, John R.W. Fox, of 1677 Deep Cove Road, North Vancouver, British Columbia, V7G 1S4 do hereby certify that:

1) I am a consulting metallurgical engineer with an office at 302-304 W. Cordova St. Vancouver, British Columbia V6B 1E8

2) I am a graduate of the University of Leeds (UK) in 1971 with a B.Sc. in Applied Minerals Sciences.

3) I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia.

4) I have not visited the property.

5) I have practised my profession continuously since 1971 and have been involved with the development of many similar properties including the Escalante Silver mine, Utah, USA, the Canatuan Mine, Philippines, the Julietta Mine, RFE, Russia and others.

6) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in draft National Policy 43-101.

7) In the Independent report titled "Kremnica Gold Project Pre-Feasibility Study", I am responsible for supervising metallurgical testwork, development of process flowsheets and plant General Arrangement drawings and Section 16. .

8) I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report.

9) I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

10) As of the date of this certificate, to my the best of my qualified knowledge, information and belief, this technical report contains all the scientific and technical information that is required to be disclosed to make the report not misleading.

Dated this 5th day of July 2007

PROFESSIONAL PROVINCE OF J. R. W. FOX #12578 BRITISH COLUMBIA ENGINEER

John Fox, P.Eng

06 Jul 07 10:40 John Fox +1(604) 6814415 p.2

CERTIFICATE OF QUALIFICATION

I, Dr. Mark R Locke, of 2611 / 939 Expo Blvd, Vancouver, British Columbia, V6Z3G8, do hereby certify that:

1. I am a consulting civil engineer with Knight Piesold Ltd., with an office at 1400-750 West Pender Street Vancouver, British Columbia V6C 2T8
2. I am a graduate of the University of Tasmania, Australia in 1994 with a Bachelor of Engineering, and the University of Wollongong, Australia in 2001 with a PhD in Geotechnical Engineering.
3. I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia.
4. I have visited the property on two occasions the last being from 8 to the 12 May 2006.
5. In the independent report titled "Kremnica Gold Project Pre-feasibility Study", I am responsible for the preparation of Section 18.2.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in draft National Policy 43-101.
7. I have continuously practiced my profession since 1995.
8. I was in charge of the pre-feasibility design of waste and water management facilities for the project.
9. I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report.
10. I have read National Instrument 43-101, Standards for Disclosure of Mineral Properties and Form 43-101F1. This technical report has been prepared in compliance with that instrument and form.
11. As of the date of this certificate, to my the best of my qualified knowledge, information and belief, this technical report contains all the scientific and technical information that is required to be disclosed to make the report not misleading.

PROFESSIONAL PROVINCE OF Dr. M. R. LOCKE # 31320 BRITISH COLUMBIA ENGINEER

Dated this 5th day of July, 2007. 5 July 07.

SECTION 23.0ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

The Kremnica project is an exploration property and there are no additional requirements to report

SECTION 24.0 ILLUSTRATIONS

There are no illustrations.