UNITED STATES
SECURITIES AND EXCHANGE COMMISSION
Washington, D.C. 20549

FORM 6-K

REPORT OF FOREIGN ISSUER PURSUANT TO
RULE 13a-16 AND 15d-16 UNDER THE
SECURITIES EXCHANGE ACT OF 1934

For the month of:	December 2006
Commission File Number:	000-50486

TOURNIGAN GOLD CORPORATION

(Translation of registrant's name into English)

24th Floor, 1111 West Georgia Street, Vancouver, British Columbia, Canada, V6E 4M3
(Address of principal executive offices)

Document List, Page 2

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

Indicate by check mark if the registrant is submitting the Form 6-K in paper as permitted by Regulation S-T Rule 101(b)(1): ____

Note: Regulation S-T Rule 101(b)(1) only permits the submission in paper of a Form 6-K if submitted solely to provide an attached annual report to security holders.

Indicate by check mark if the registrant is submitting the Form 6-K in paper as permitted by Regulation S-T Rule 101(b)(7): ____

Note: Regulation S-T Rule 101(b)(7) only permits the submission in paper of a Form 6-K if submitted to furnish a report or other document that the registrant foreign private issuer must furnish and make public under the laws of the jurisdiction in which the registrant is incorporated, domiciled or legally organized (the registrant's "home country"), or under the rules of the home country exchange on which the registrant's securities are traded, as long as the report or other document is not a press release, is not required to be and has not been distributed to the registrant's security holders, and, if discussing a material event, has already been the subject of a Form 6-K submission or other Commission filing on EDGAR.

Indicate by check mark whether by furnishing the information contained in this Form, the registrant is also thereby furnishing the information to the Commission pursuant to Rule 12g3-2(b) under the Securities Exchange Act of 1934. Yes ___ **No XXX**

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

Explanatory Note:

Tournigan Gold Corporation (the "Company") voluntarily filed a registration statement on Form 20-F/R which was declared effective by the Securities and Exchange Commission (the "SEC") in January 2006. The Company is not currently listed on any securities exchange or automated quotation system in the United States. On March 1, 2006, the Company filed an Annual Report on Form 20-F in respect of its fiscal year ended August 31, 2005 ("2005 Annual Report"). The Company mistakenly failed to file any Current Reports on Form 6-K with the SEC since the filing of its 2005 Annual Report. The purpose of this Form 6-K is to furnish to the SEC such information that the Company believes should have been filed on a Form 6-K subsequent to the filing of its 2005 Annual Report. For ease of reference, concurrent with the filing of the report, another 6-K with additional documents is being filed.

Documents included:

TECHNICAL REPORT
OF
JAHODNA URANIUM PROJECT,
SLOVAKIA

for
Tournigan Gold Corporation
Suite 301, 700 West Pender Street,
Vancouver, B.C. V6C 1G8
CANADA

Prepared by
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ACA HOWE INTERNATIONAL LTD

March 28, 2006

Berkhamsted
Herts, UK

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1. SUMMARY

ACA Howe International Ltd, of UK, was commissioned by Tournigan Gold Corporation in October 2005, to undertake a Canadian National Instrument 43-101 report, on the Jahodna uranium project in Slovakia. The author of this report was contracted by ACA Howe International to compile and write this report.

The author visited Slovakia for approximately two weeks during October 2005. During this time, he spent time at the offices of Kremnica Gold Company in Kremnica, and of Koral SRO in Spisska Nova Ves. A one day visit was made to the project area, to make geological ground traverses and examine drill core from the new drill hole named KG-J-1. Since returning to the UK, the author has compiled relevant data, assisted with various aspects of digital mapwork, and written the 43-101 report. Other resource calculation and map drafting work for this report was completed by geologists of ACA Howe International Ltd in Berkhamsted, UK.

Much of the content of this report relies on English translations of previous reports in Slovak. While it is felt that previous work undertaken in the area, in general, probably conformed in detail to the exploration and evaluation regulations in force in Slovakia at that time, there is probably much detail of past work that has not yet been translated into English, so it was difficult to obtain or evaluate for the purposes of this report.

In addition to these, conversations were had, through an interpreter, with a number of Slovak technical personnel who had previous involvement with the Jahodna project. Since the author only spent two weeks in Slovakia, and less than one day on site at Jahodna, heavy reliance has been placed on the various sources of data mentioned above.

The full title of the current exploration licence refers to “Cermel-Jahodna – U-Mo, Cu ores”, and it was granted on March 21st 2005 by the Geology and Natural Resources Department at Ministry of the Environment of the Slovak Republic. The period of validity of the licence is four (4) years, and various conditions were attached to the granting of the exploration licence. To the author’s knowledge, the company has complied with all the required conditions.

The Jahodna property is situated in East-Central Slovakia. Topographically, the region forms part of the Western Carpathian mountain chain. The actual surface is undulating, with little or no outcrop and deep soil cover. The vegetation is a type of mature mixed woodland, being made up mostly of broadleaved types, but also lesser conifers. The forest is in fact part of a forestry reserve, though according to the Tournigan staff in Slovakia, this should not pose a prohibitive problem with regard to planning permission for exploration or mining development activities.

The Jahodna property is situated some 10-15 km NW of the large town of Košice, a regional centre in East-Central Slovakia. The Jahodna property is relatively accessible, and lies quite close to the regional main road between Kosice in the SE and Spisska Nova Ves in the NW. From this main road, a network of minor unsurfaced tracks traverse the forest, and give access to the project area.

The climate is essentially Central European, but is moderated by the 600m altitude. In effect this gives the area cold winters, with snow on the ground between about December and March. The mean January temperature is around -5^0 C, and the mean July temperature is 19^0 C. Total annual precipitation is 700 to 800 mm, with over 30 mm precipitation falling as snow in January.

The Jahodna uranium deposit belongs to a belt of U-Mo deposits, of Lower Permian age, within the western Carpathians of Slovakia. These deposits are largely stratabound bodies within volcanosediments of Permian age. It appears that the U-Mo mineralisation at Jahodna was disseminated within the volcanosedimentary pile, and was subsequently enriched into stratabound zones by post depositional geological activities.

The Jahodna deposit appears to occupy an open-space fill, on a geological contact between an overlying competent meta-andesitic volcanic unit, and an underlying meta-sediments. Tectonic disturbances have also resulted in slaty cleavage and well developed shistosticy in places (giving poor ground conditions in some softer sedimentary units) and fault offsets, some of which disrupt the main deposit. The deposit is partially blind at the surface. Uranium mineralization detectable with a hand-held scintilometer outcrops along the meta-andesite – meta-sediment contact above the deposit.

The deposit area is covered by thick soils, with extensive forest cover at surface. The deposit has a NW-SE strike, and a steep dip to the SW. The overall dimensions of the main deposit established to date are some 500m x 500m in longitudinal section, and an average of about 2.5m in thickness. There are also minor mineralised zones in the hanging wall of the main deposit, though their relationship to the main deposit is still uncertain.

The Jahodna uranium deposit represents a difficult exploration target, since it is partially blind and deep-lying, in hilly terrain with deep soil cover and thick forest at surface. At these elevations, snow lies on the ground during the winter, sometimes for more than 3 months per year. A combination of these factors virtually negates surface exposure, and make useful surface mapping harder to achieve. Furthermore, the geology of the deposit itself is in some ways complex, and is as yet incompletely understood. As detectable mineralization does outcrop in places along the favourable stratigraphic horizon (meta-andesite – meta-sediment contact) it may be possible to explore for additional mineralization along strike by prospecting the meta-andesite – meta-sediment contact with a hand-held scintilometer and conducting ground radiometric and/or soil-gas (radon) surveys.

The complex mineralisation at Jahodna comprises U-Mo-Cu, with lesser Fe-Pb-Ti-P-Ba. The mineralisation at Jahodna has been described as a “Saddle Hills analogue”, after the Saddle Hills uranium deposit in Mongolia. Although there is insufficient detailed information to fully back this up, there would appear to be some broad similarities between Jahodna and the Saddle Hills type deposits. The Jahodna deposit itself is a high grade deposit

In many ways, it is to the credit of Uranovy Prieskum that they discovered the Jahodna deposit at all. Such are the complexities of the setting of this deposit, that it was only after considerable exploration drilling that it was discovered in the 1980’s. Following airborne radiometric surveys, and ground follow up radiometric surveys, the best way forward was found to be to systematically drill the airborne and ground anomalies. Consequently, on the Jahodna project, a total of 53 exploration holes, totalling over 17,000m of drilling, were completed between 1985 and 1991.

Having established that deep drilling was the best way to evaluate the Jahodna deposit, it has to be said that the drilling undertaken by Uranovy Prieskum was unsatisfactory in some key respects. Their pre-wireline, thin-walled, conventional B-size drill string deviated considerably from the targets, often intersecting the mineralized horizon many tens of metres or more from the intended point of intersection. In a deposit where it would appear that post-mineralization normal faulting has broken up the deposit zone into blocks, the positions of which are likely to be of critical importance at the mine planning stage, this lack of directional drilling control was unfortunate. Perhaps largely because of the directional problems with the earlier drilling, the resulting drill cross sections of the original evaluation were overall not normal but somewhat oblique to the plane of the deposit.

This affects the quality and reliability of the samples from this drilling and therefore influences the reliability of calculated resource estimates.

Furthermore, in drilling a strongly foliated, tuffaceous meta-volcanosedimentary sequence, the core recovery of the drilling was generally poor. This was unfortunately particularly the case in the mineralised zone, where shearing on the base of the meta-andesite unit resulted in strongly broken ground, and consequently core recoveries in the region of no more than 50% (though this was improved towards the very end of the drilling after 1990). For this reason, there were apparently no complete mineralised zone intersections from the original 14 holes drilled in the immediate vicinity of the Jahodna deposit.

With a record of persistently poor core recovery in the mineralised zone, the only way to attempt to establish the average widths and grades of the mineralised zone was by down hole radiometric logging. Fortunately this technology appears in itself to have worked well – though it has to be emphasised that there were no complete, assayed borehole intersections at Jahodna against which to calibrate the gamma logger response. However, all aspects of the tonnage and grade calculation at Jahodna appear to have adhered to strict technical guidelines set down by the national mining regulatory bodies at that time, and also drew on former radiometric logging experience of drilling and mining similar deposits within the country.

Within the area of the Jahodna deposit itself, the past drilling has been both irregularly and quite widely spaced. It would seem that by the traditional Czech type reserve classification this stratiform but irregular replacement type body was categorised as a "Z-3 reserve", equating to a Soviet type "C_2 reserve" or an "inferred resource" under JORC classification.

One prominent feature highlighted by the drilling at Jahodna is the poor ground conditions existing in much of the ground near the deposit itself. However, these bad ground conditions appear to be largely restricted to the thick hanging wall volcaniclastic sequence. Although the mineralised zone itself appears to be badly tectonised, it seems to be largely supported by the more competent andesitic body which immediately overlies it.

For reasons including core recovery, limited geological / structural interpretation and poor ground conditions, in terms of systematically establishing the true average grades of the mineralised zone at Jahodna, or important geological aspects like the detailed positions of cross faults, the earlier drilling undertaken by Uranovy Prieskum would in my opinion have to be regarded as somewhat preliminary in nature.

The first major resource calculation was undertaken in 1996 by Jozef Daniel, a geologist experienced in the uranium industry of the former Czechoslovakia. The tonnage and grade calculation was updated in 2005 by the same author, in a large report entitled "Calculation of Reserves Deposit Kosice I, U – Mo Ore", which was translated into English. This update derived the following summarised figures:

- Variant I – (0.015% U Cutoff) – 2.188 Million tonnes at 0.329% U to derive an in situ figure of 15.876 Million lbs of U.
- Variant II – (0.030% U Cutoff) – 1.396 Million tonnes at 0.472% U to derive an in situ figure of 14.528 Million lbs of U.

Molybdenum grade calculations were made as well as uranium (with a calculated average figure of 0.38% Mo). However, with the poor core recovery, and lack of an alternative method for determining Mo grades, the volume of data available to determine Mo grades was reduced, for which reason the Mo grades calculated are regarded as less reliable than the uranium. There is also an unresolved question regarding the detailed distribution of the Mo mineralisation within the Jahodna deposit. Therefore, Mo can currently only be regarded as a possible potential by-product.

Three deep diamond drill holes (totalling 1,364.90m) have recently been drilled by Tournigan into the Jahodna deposit. The purpose of these holes was to provide preliminary confirmation of the thickness and average grade of the deposit. In contrast to the earlier drilling of the 1980s, this recent drilling achieved good directional control and excellent core recovery. Assay results from this drilling substantially confirmed previous drilled widths and uranium grades.

Since the previous reserve calculations at Jahodna were based on a large volume of detailed data, parts of which were not available for examination, it was not possible to conduct a full recheck of all data pertaining to the former (historical) reserve calculations. For this reason it was decided by ACA Howe to undertake an in-house Micromine computer study of the available Jahodna drill data, in order to make an independent assessment of the historical resources calculation.

Accordingly, a Polygonal Wireframe Resource Estimate (PWRE) was calculated for the Jahodna Resource, using data from thirteen (13) historical drill holes over the project. The thirteen deep historical drill holes, amounting to some 6,290 m of drilling, covered the main part of the Jahodna deposit, though the wireframe model was somewhat constrained in that it did not have all the deposit edge data used for the earlier estimates. Nevertheless, the PWRE appears to have broadly confirmed the historical tonnage and grade estimates.

The PWRE work was conducted in two phases – firstly, using only the original Jahodna drill hole data, and secondly, introducing the three new boreholes drilled by Tournigan Gold Corporation. In both cases, the PWRE appears to have broadly confirmed the historical tonnage and grade estimates.

The recent PWRE resource calculation, using the new Tournigan drilling results, can be summarised as follows:

- (0.030% U cutoff) -1,256,088 tonnes @ 0.56% U to give 15,510,172 lbs of contained uranium.

Taken together, it would appear that the recent PWRE work on the original Jahodna data, the new Tournigan drilling and the updated PWRE model have all broadly confirmed the historical tonnage and grade estimates at Jahodna using the same cut-off grade of 0.03% U. Based on this, it is reasonable to state that all this work confirms the tonnage and grade estimate at Jahodna as an “inferred resource”.

Although surface exploration at Jahodna is made difficult by a number of factors (deep soil cover, very little outcrop, extensive forest cover etc), there are possible ways where further work at surface might be feasible. Prospecting along the meta-andesite – meta-sediment lithologic contact with a hand-held scintilometer and conducting surface radiometric and/or soil-gas (radon) geochemical surveys over the favourable horizon may prove effective as surface exploration techniques. The accurate positioning of the large cross faults is a critical factor. One possibility to consider may be to use high resolution aerial photography and / or satellite imagery flown during the ‘Spring window’ – when the snow has melted but before the forest vegetation has grown. In this way a working geological / structural model of the area can be developed, to be modified with the results of the ongoing drilling programme. If the remote sensing methods mentioned above are not feasible, another method to pinpoint fault structures may be to undertake detailed ground magnetic surveys, though a tight grid (eg. 50m x 10m or 50m x 25m) would be recommended.

It is recommended that a conceptual mining study be undertaken at Jahodna. The purpose of such a study would be to give exploration guidance, at a relatively early stage of exploration, to an apparently attractive project in a geographic area where mining operations were not previously costed to the same degree as in the west.

Even though the work to date suggests that the Jahodna deposit can be regarded as an inferred resource, much more exploration work is recommended in order to upgrade the confidence level of the resource to the indicated category. Specifically, a substantial drilling programme is recommended, in order to effectively drill the deposit on 50m centres. This will achieve sufficient drilling density to increase the level of confidence in grade and geological continuity through the deposit, and may facilitate an upgrading of the resource classification to “indicated.” In view of the deep, narrow and steep-dipping nature of the deposit, intensive drilling from surface on 50m centres is not recommended. Drilling from surface to give 50m centre coverage would involve some 85,000m of drilling, resulting in a high drilling cost in the region of USD 7.6 million. Therefore, an initial phase of in-fill and step-out diamond drilling within and immediately along strike of the deposit should provide more even-spaced drill intercepts along the strike of the deposit at two depth-horizons as well as test presently un-tested gaps in the resource block that we believe are possibly mineralized.

Instead it appears appropriate, following a further limited programme of deposit outline drilling, to embark on a programme of underground development, in order to facilitate a drilling programme from underground. Such underground access would enable 50m centre drilling using only some 39,000m of drilling (as opposed to 85,000m of drilling from surface), and equating to a cost of approx. USD 4.9 million (ie. a 35% cost saving compared to the surface drilling programme). It should also be mentioned that, with an average hole length less than half that of the surface drilling (ie. average length of 211m per hole as opposed to 436m per hole), the drill holes of the underground drilling programme would be generally much more precise in intersecting their target areas.

To provide the underground access for such a drilling programme, a preliminary costing has been made of an underground development programme. This scenario allows for an access decline of some 2900m, a 500m long horizontal drive in the deposit's hanging wall, 1200m of crosscuts for drilling access, and a 350m ventilation raise. The preliminary costing for this programme has been calculated at USD 5.5 million. Besides enabling access for underground drilling and bulk sampling, the underground access would be laid out such that it could be used for subsequent mining operations.

Besides the further detailed evaluation of the Jahodna deposit, it is recommended to undertake additional grass roots type exploration within the licence area. This is especially the case in the SE part of the license area, where former systematic exploration did not cover.

2. INTRODUCTION

The author visited Slovakia for approximately two weeks during October 2005. During this time, he spent time at the offices of Kremnica Gold Company in Kremnica, and of Koral SRO in Spisska Nova Ves, examining paper and digital data of various ages, and arranging for scanning, digitising and other map work activities. A one day visit was made to the project area, to make geological ground traverses and examine drill core from the new drill hole named KG-J-1.

Since returning to the UK, the author has compiled relevant data, assisted with various aspects of digital mapwork, and written the 43-101 report. Other work for this report was completed by Mssrs. Mark Butcher and Galen White of ACA Howe International Ltd in Berkhamsted, UK. The former assisted with general digital map work, and the latter constructed a Micromine computer model of the deposit for tonnage and grade estimation purposes.

2.1 Disclaimer

If the author of all or a portion of the technical report has relied on a report, opinion or statement of legal or other experts who are not qualified persons for information concerning legal, environmental, political or other issues and factors relevant to the technical report, the author may include a disclaimer of responsibility in which the author identifies the report, opinion or statement relied upon, the maker of that report, opinion or statement, the extent of reliance and the portions of the technical report to which the disclaimer applies.

Much of the content of this report relies on English translations of previous reports in Slovak, in particular that listed above by Daniel (2005). While it is felt that previous work undertaken in the area, in general, probably conformed in detail to the exploration and evaluation regulations in force in Slovakia at that time (eg. official mining directorates of 1989 and 1992), there is probably much detail of past work that has not yet been translated into English, so was difficult to obtain or evaluate for the purposes of this report. Such data would include details of exactly how equivalent grades of uranium were derived historically from down hole geophysical gamma logs, using a number of coefficients and factors which appear to have been based on past uranium exploration / mining history in the area.

In addition to these, conversations were had, through an interpreter, with a number of Slovak technical personnel who had previous involvement with the Jahodna project. In particular, these included Mssrs. Jozef Daniel and Ladislav Novotny. In addition, much general information of the former Czechoslovakian mining industry was gained from Dr. Boris Bartalsky, currently the General Manager of Tournigan's Kremnica Gold Project in Slovakia.

Since the author only spent two weeks in Slovakia, and less than one day on site at Jahodna, heavy reliance has been placed on the various sources of data mentioned above.

3. RELIANCE ON OTHER EXPERTS

The main sources used in the compilation of this report were the following:

- **Calculation of Reserves, Deposit Kosice I, U-Mo Ore April 2005**, Daniel J., Bartelsky B., unpubl. company report by Kremnica Gold Corp.
- **Resolution on Granting of the Exploration Licence,** March 21 2005, issued by Ministry of the Environment of the Slovak Republic, Geology and Natural Resources Dept.
- **Technological Research of U-Mo ore from Jahodna Site, 1993**, Kopecky J., unpubl. report by MEGA, joint stock company Strazpod Ralskem, Czech Republic.

In addition to these sources, which specifically concerned the Jahodna deposit, a number of other sources and references were used, which are listed at the back of this report.

4. PROPERTY DESCRIPTION AND LOCATION

The full title of the current exploration license refers to "Cermel-Jahodna – U-Mo, Cu ores", and it was granted on March 21st 2005 by the Geology and Natural Resources Department at the Ministry of the Environment of the Slovak Republic. The full area involved is shown on Maps 6 and 7, and this area amounts to 31.75 km^2 in surface area (see Appdix.1 for further details). The period of validity of the licence is four (4) years.

The name and "code" of the region is Kosicky 8, and the name and code of the counties are Kosice I - 802, Kosice III - 803, and Kosice – okolie - 806. The names and numbers of the cadastral areas are shown on the table below:

NAMES AND NUMBERS OF CADASTRAL AREAS

No	The No. of the Cadastral Area	The Name of the Cadastral Area	The Name of the Village	Relative Ratio of the Villages %	Cost SKK
1	827207	Cermel	Košice -mestská cast Sever	51.59	24,763
2	827428	Myslava	Košice -mestská cast Myslava	9.20	4,416
3	802123	Baška	Baška	7.09	3,403
4	827606	Košická Belá	Košická Belá	20.93	10,046
5	841129	Nižný Klátov	Nižný Klátov	6.41	3,077
6	871516	Vyšný Klátov	Vyšný Klátov	4.78	2,295

The license area, which is a single, contiguous area, is shown in some detail on Maps 6 and 7, which shows UTMs using the Krovak – Gaussian equiangular conic projection. The coordinates of the licence area are given as:

COORDINATES OF JAHODNA LICENSE AREA

Point No.	Y	X
1	268 31000	1 241 45000
2	274 59000	1 229 34000
3	273 70000	1 228 60000
4	268 20000	1 234 68000
5	266 81000	1 241 56000

The "conditions" of the exploration licence are shown below:

The holder of the exploration licence:

1. will perform the geologic works in accordance with the project of the geological work that was submitted with the application on granting of the exploration license and the holder will perform the geological works in compliancy with the geological law and other legal regulations.
2. will prepare the final report in compliance with §14 of the geological law and will submit to ministry the calculation of the resources for the approval, in compliancy with § 16 par. 2
3. will send the approved final report to geological survey of Dionýz Štúr Bratislava for archiving in compliance with §17 of the geological law.
4. will submit the annual report of the geological work with the results of special geological works and spent money on exploration up to six weeks after the end of the year.
5. will follow the requirements of nature and land protection pursuant the law,
6. will cut the trees out of the wood territory if necessary and ask the resident village for permission pursuant the law,
7. will secure the places of holes against fuel leakages into the underground or surface water and surrounding,
8. will clean the field and put it into the previous conditions after finishing of geological works,
9. will keep regulations of the law Nr. 364/2004 about waters
10. will require demarcation of protective zone by resident water company if any technical works needed
11. will ask for statement the resident company if any technical works in the area of holiday and sport centre
12. will keep the law about using of agricultural land and control of pollution of the environment,
13. will ask for he statement the resident keeper of Bukovec water tank which provides local villages Košická Belá, Vyšný Klátov with water,
14. will keep the law about forests,
15. will follow the various regulations about protection of the forest land reserves,
16. will announce the geological works in the Protective deposit area Košice VI. to the resident company Uranpres, s.r.o. Spišská Nová Ves pursuant to the regulations set by the Slovak mining bureau,
17. will announce the existence of the mineral water and gas resources to the Ministry of Health up to 15 days since found pursuant to the law,
18. will follow the law if any archaeological findings,
19. will not realize any geological works where any cultural sights,
20. will ask for statement from the local municipality in Košice – landed estate department before any geological works,
21. will ask for statement where any roads of the II. and III. type the local municipality in Košice,
22. will ask for statement the Slovak gas industry before any geological works,
23. will keep various standards and the law about power industry,

24. will ask for statement the Slovak Telecom a.s.,
25. will respect the water managing objects and lines of protective zones of the water resources,
26. will not realize technical works in the protective zones of water resources,
27. will ask for statement East-Slovak water company, Košice before any geological works,
28. will keep valid standards and regulations if dealing with dangerous substances to prevent any pollution of surface and underground waters while geological works,
29. will ask for statement the East-Slovak power company before any geological works.

To the knowledge of the report author, all appropriate obligations have been fulfilled by the licence holder, prior to commencement of exploration works on the licence area.

Within the exploration licence in question, there is one known mineralised zone, which is the Jahodna deposit, which was historically drilled and evaluated by Uranpres. The location of this area is indicated by the drill holes on Maps 1 to 13. Within the licence area, there are no known mine workings, existing tailing ponds, waste deposits or other workings relating to previous exploration or mining.

Other than the above annual licence payment, the report author is not aware of the terms of any royalties, back-in rights, or other agreements and encumbrances to which the property is subject. All the known environmental liabilities, and permits that must be acquired to conduct the work proposed for the property, are listed above. As stated above, to the knowledge of the author, all appropriate contractual obligations have been fulfilled by the licence holder, and all necessary permits have been acquired, prior to commencement of exploration works on the licence area.

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

The Jahodna property is situated in East-Central Slovakia (see Map 1). Topographically, the region forms part of the Western Carpathian mountain chain. Locally the hilly terrain is part of the Volovec Hills, in more detail the Kojsovska Hola. Around Jahodna, the ridges trend NW-SE and the topography is quite incised, with hills up to around 650m amsl, and valley floors typically down to some 500m amsl (in the immediate area of the project).

The actual surface is undulating, with little or no outcrop and deep soil cover of many metres depth. The vegetation is a type of mature mixed woodland, being made up mostly of broadleaved types (eg. beech), but also lesser conifers (see Plate 1). The forest is in fact part of a forestry reserve, though according to Tournigan staff in Slovakia, this should not pose a prohibitive problem with regard to planning permission for exploration or mining development activities.

The Jahodna property lies quite close to (less than 300m south of) the regional main road (No. 547) between Kosice in the SE and Spisska Nova Ves in the NW. From this main road, a network of minor unsealed tracks traverse the forest, and give access to the project area.

The Jahodna property is situated some 10-15 km NW of the large town of Košice, a regional centre in East-Central Slovakia (see Map 1). It is situated outside the town lands of Košice. As mentioned above, the property is situated near the main tarred road between Košice and Spisska Nova Ves. A seasonal ski resort (referred to as “Jahodna Chalet”) occurs further to the NW along the same range of hills.

The climate is essentially Central European, but is moderated by altitude (ie. the project is in hilly terrain at around 600m altitude). In effect this gives the area cold winters, with snow on the ground between about December and March. According to the Slovak Encyclopaedia (Reference 5) the mean January temperature is around -5^0 C, and the mean July temperature is 19^0 C. Total annual precipitation is 700 to 800 mm, with over 30 mm precipitation falling as snow in January. Records indicate that snow lies on the ground for over 80 days per year (generally January to March). With access to the

project area by unsurfaced tracks, the snow cover is expected to cause periodic difficulties with access during the winter months, probably most particularly during the spring thaw.
While the existing surface rights are sufficient for exploration purposes, it is not known whether they are completely sufficient for all aspects of a mining operation. The surface area is undulating and hilly, and it is not yet established whether there is sufficient suitable and reasonably level ground for potential waste rock and tailings storage areas (such aspects would more appropriately be covered in a scoping study concerned with mining aspects).

A small stream of intermittent flow drains NE along the valley traversing the Jahodna deposit, flowing into the Crmel valley which lies to the NE side of the hill range. Another larger river (the Vrbica) occurs approx. 1 km to the west, bounding the hills on the west side. The Vrbica and Crmel rivers are tributaries of the Hornad river, which flows southwards past Košice – see Maps 1 and 6. Apparently, electric grid power occurs in the area, though the exact distance from site is not known at present.

Up to the time of the so-called Velvet Revolution in 1989, Czechoslovakia had a large, state-funded mining industry, most of which closed down when state funding was withdrawn following the demise of the communist system. For this reason, it is likely that Slovakia still contains a significant number of trained mining personnel eg. from the old Novovesta Huta mining operation, which was operated some 40-50 km north west of Jahodna, and close to the town of Spisska Nova Ves.

6. HISTORY

Up to the time of the demise of communism in 1989, all uranium exploration and mining in Czechoslovakia was conducted by the State-owned organisations, such as KORA, CSUP and URANPRES. All early exploration work on the Jahodna property was undertaken by these organisations. Following the country's return to the market economy system, and the subsequent separation of Slovakia and the Czech Republic, very little work has been undertaken on the Jahodna property. Tournigan Gold Corporation acquired the Jahodna property in 2005.

The Jahodna deposit was discovered in 1985, following a regional uranium exploration programme undertaken by the Czechoslovakian KORA and CSUP groups. Regional airborne radiometric surveys had delineated radiometric anomalies on surface, which were followed up by ground radiometric surveys, geological mapping and trenching. Thereafter, systematic diamond drilling was used to investigate ground radiometric anomalies (see Maps 8 and 9). The Jahodna deposit, which is partially blind and therefore does not outcrop at surface except as discontinuous zones of low-level mineralization, was discovered by routine diamond drilling of surface anomalies. It was thereafter drilled by Uranovy Prieskum (URANPRES), though the exploration programme was cut short by the political events following the 1989 Velvet Revolution.

In all, some 17,000 m of drilling were undertaken in 53 holes on and around the property. Most of the drilling was by conventional (ie. pre-wireline) diamond drilling. The thin-walled drill strings deviated considerably during drilling, and core recovery was generally poor (overall around 50%). To compensate for this poor core recovery, down-hole radiometric logging was routinely used on all the drilling in the Jahodna area, and it seems to have worked very well. Conversion formulae were developed, based on different factors and coefficients which were derived from previous uranium exploration and mining experience from nearby uranium projects (eg. Novovesta Huta), in order to convert down hole radiometric measurements into equivalent in-situ uranium grades. This down hole work compensated at least in part for the poor core recovery, which resulted in an incomplete assay database for the project.

To summarise the effectiveness of the historical exploration, it has to be said that it is to the credit of the former Czechoslovakian state exploration companies that the Jahodna deposit was discovered at all. The deposit itself does not outcrop near to surface, though its distal peripheral margins do sub-outcrop at surface, and gave sufficient radiometric response for it to be identified as a radiometric anomaly detected during airborne, and ground radiometric surveys and with a hand-held scintilometer on outcrop. Soil cover at surface was generally too deep for surface mapping, trenching and pitting to be effective other than to generate drill targets, so the best way forward was found to be systematic drilling to investigate the depth extensions of the surface anomalies. In all, several thousand metres of diamond drilling from surface were used as a regional exploration methodology, before the discovery of the Jahodna deposit itself.

However, while the former Czech exploration enterprises are to be commended on the systematic and persistent approach which led to the Jahodna discovery, they also encountered considerable difficulties during the more detailed evaluation stages. Since the depth of the Jahodna deposit meant that drilling was to be the main evaluation method, a heavy reliance was put on deep diamond drilling programmes. The main problems here revolved around not only the depth of the target (necessitating drilling of up to almost 1000m in depth), but the generally poor ground conditions that the drilling had to encounter, and also the unsophisticated drilling equipment used.

The majority of the drill holes had to pass through up to several hundred metres of hanging wall intermediate meta-volcaniclastics/tuffs (Petrovohorske Formation, No. 12 on geological maps), before reaching the mineralised zone. This formation comprised tuffaceous volcaniclastics, which are steeply dipping and strongly cleaved, with cleavage planes more or less paralleling the bedding planes – this combination caused persistent problems with poor ground and consequently poor recovery. In addition to the poor ground conditions in the hanging wall sequences, the mineralised zone itself was generally weak and friable, also resulting in very poor core recovery. On top of this, multi-tubed wireline drilling equipment was only used late in the exploration programme – the majority of the drilling programme was undertaken with thin-walled, single tubed, conventional drilling equipment (this resulted not only in poor core recovery, but in poor directional control of the drill path).

Besides the purely technical aspects of the drilling problems, the main result was the very poor core recovery within the mineralised zone. Obviously, this affected the sampling integrity of the U and Mo, the two main economic minerals. If a drill achieves only 50% recovery in a broken mineralised intersection, it is difficult, based on assays alone, to obtain a clear picture of both the average grade, and the distribution of grade, of a particular mineral within the mineralised zone. Fortunately, in the case of uranium, the difficulties were eased by the down hole radiometric surveys. Such surveys had been used previously on uranium exploration and mining, in similar deposits, so correlation coefficients had been worked out to convert the down hole radiometric response into equivalent uranium grades. These down hole surveys enabled a much more complete picture of the uranium grade and distribution than was possible with the assays alone. Unfortunately for the Mo analyses, there was no alternative way to determine equivalent Mo concentrations, consequently the Mo grades of the drill samples remain less reliable than the U.

See Maps 6 to 13, and Sections 1 to 6.

The first major resource calculation was undertaken in 1996 by Jozef Daniel, a geologist experienced in the uranium industry of the former Czechoslovakia. The method used a block model method, with two variants based on different minimum cutoff grades (0.015% and 0.03% U). The calculation methodologies were constrained by government mining directives, established in 1987 and updated in 1992. The calculation concentrated on the main deposit, which was placed in the category of "Z-3 supposed reserves" – the lesser zones of mineralisation in the hanging wall of the main deposit were assigned to the lower "prognostic" category. The tonnage and grade calculation was updated in 2005 by the same author, in a large report entitled "Calculation of Reserves Deposit Kosice I, U – Mo Ore". This report was translated into English, and since it represented a comprehensive technical history of the Jahodna project, it was consulted in detail by the present author.

Please see attached table for tonnage and grade estimates.

#	Name of Study / Description	Tonnes	Grade % U	Content lbs	Comment
1	1996 Calculation (J-1-Z-3)	1,148,000	0.459	11,618,851	"Economic reserves exploitable"
	(by J. Daniel)				Used in Tournigan website
2	1996 Calculation (J-1-Z-3N)	1,080,000	0.187	4,453,218	"Potentially economic reserves exploitable"
	(by J. Daniel)				Used in Tournigan website
3	1996 Calculation (J-1-Z-3)	1,396,000	0.472	14,529,010	"Total geological reserves"
	(by J. Daniel)				
4	Josef Daniel Study April 2005	2,188,553	0.329	15,876,000	Latest updated figures from 2005 study.
	Variant I (0.015% U cutoff)				
5	Josef Daniel Study April 2005 Variant II (0.030% U cutoff)	1,395,975	0.472	14,528,745	As of April 2005 study

COMPILATION TABLE FOR DIFFERENT URANIUM RESOURCE/ RESERVE

A full description of previous exploration work, and tonnage and grade calculations based on them, are given in Josef (2005).

Molybdenum grade calculations were made as well as uranium (with an average derived of 0.38% Mo), though with the poor core recovery, and lack of an alternative method for determining Mo grades, the volume of data available to determine Mo grades was reduced, for which reason the Mo grades calculated are regarded as less reliable than the uranium. There is also an unresolved question regarding the detailed distribution of the Mo mineralisation within the Jahodna deposit. Some evidence was encountered suggesting that Mo grade variations were not sympathetic with the U grade variations, and even that Mo was enriched on the margins of the deposit. Therefore, with the present data base, Mo can be regarded only as a potential by-product.

The former historical resource calculations are not compliant with NI 43-101 definitions. For this reason, they are not described as other than "Historical Reserves". However, confidence levels have been increased by recent developments (see below).

No production or mining activities have yet been undertaken from the Jahodna property, to the writer's knowledge.

7. GEOLOGICAL SETTING

The Jahodna uranium deposit belongs to a belt of U-Mo deposits within the western Carpathians of Slovakia, which are largely stratabound bodies within volcanosediments of Permian age. It appears that the U-Mo (Cu) mineralisation was disseminated within the volcanosedimentary pile, and was subsequently enriched into stratabound zones by post depositional (tectonic deformation) geological activities. See Maps 2 to 5.

The Jahodna deposit is contained within a Lower Permian volcanosedimentary sequence, designated as the Petrovohorske Formation. Its main units at Jahodna are briefly described below:

- Overlying the immediate hangingwall are the intermediate volcaniclastics of the Hutniansky Complex (designated No. 4 on the geological maps). They are a few hundred metres thick in the

Jahodna area, and are generally incompetent (on account of their parallel, steeply dipping bedding and cleavage planes).

- The rock type which forms the immediate hangingwall to the Jahodna deposit is the meta-andesite of the Hutniansky Complex (designated No. 43 on the geological maps). It forms a semi-competent zone, varying in thickness from 20m to 50m, immediately above the deposit. In addition to the main zone of mineralisation at its base, this unit also contains lesser “stringers” of U-Mo-Cu mineralisation within it.
- The main deposit – is hosted along the faulted, disturbed contact of the hangingwall meta-andesite and the footwall meta-sediments within the basal part of the meta-andesite unit. It averages some 2.5m in thickness, and basically comprises a uranium / polymetallic mineral assemblage, which has been deposited into a tectonically disturbed zone, on the contact of an overlying competent rock and a footwall sequence of less competence.
- The meta-sediments (slates, quartzites) of the Knolske Formation form the immediate footwall to the mineralised zone. This unit is designated No. 12 on the geological maps. They are up to hundreds of metres thick in the Jahodna area, and are of varying competence.

The upper 2 units, described above, belong to the Hutniansky Volcanic Complex (part of the Petrovohorske Formation), while the footwall to the deposit is contained within the Knolske Formation. The entire sequence is contained within the Lower Permian Krompasska Group.

The Jahodna deposit occupies dilational zones along the geologic contact between the overlying competent andesitic meta-volcanic unit and the underlying meta-sediments. Shearing along this contact has resulted in tectonic disturbance and poor ground conditions. Tectonic disturbances have also resulted in schistose foliation and slaty cleavage (giving poor ground conditions in some softer sedimentary units) and fault offsets, some of which disrupt the main deposit.

The deposit is partially blind (ie. limited surface expression), and is covered by thick soils, with extensive forest cover at surface. The deposit has a NW-SE strike, and a steep dip to the SW (60^0 in the upper part, 47^0 in the lower part). The overall dimensions of the main deposit established to date are some 500m x 500m, and about 2.5m in average thickness. As mentioned, there are also minor mineralised zones in the hanging wall of the main deposit, though their relationship to the main deposit is still uncertain.

8. DEPOSIT TYPES

The Jahodna deposit has been described as a “Saddle Hills” analogue, after the Saddle Hills / Dornod uranium deposits in eastern Mongolia. However, while the Saddle Hills deposits have been relatively well explored and documented, insufficient information is known about the Jahodna deposit to place it firmly in this category. However, there are broad similarities between the two - like Saddle Hills, Jahodna appears to be a replacement type deposit (both stratabound and cross-cutting), hosted in a strongly deformed Mesozoic volcaniclastic sequence. Also like Saddle Hills, Jahodna is enriched in a number of minerals besides uranium.

Besides the Jahodna deposit itself, which is an advanced exploration project, and will therefore require largely further drilling and detailed geological / sampling studies, there are known to be a number of mineralised lenses along strike of Jahodna within the intermediate (andesite / dacite) “Hutniansky” meta-volcaniclastic/tuffs of the Petrovohorske Formation. Several of these have been investigated in the past by the CSUP and KORA groups, though so far, Jahodna was the only mineralised lens discovered in the area with clear economic potential. The majority of these mineralised occurrences showed as radiometric anomalies of some sort at surface, though many were very subtle anomalies, on account of the depth of soil cover and the depth of some mineralised bodies. This depth of soil cover meant that pitting and trenching were less than successful as exploration methods, and in fact routine diamond drilling proved to be the most successful exploration tool to investigate ground radiometric anomalies at depth. This was how the Jahodna deposit itself was discovered.

Besides the further detailed evaluation of the Jahodna deposit (described elsewhere in this report), it is recommended to undertake additional grass roots type exploration within the licence area. Apparently the previous exploration by the CSUP and KORA groups started from the NW and worked towards the SE (since the regional exploration was spreading along strike from known deposits like Novovesta Huta in the NW), and following the Jahodna discovery, little further work was undertaken in the SE part of the concession (this cessation in exploration activities also coincided with the political developments following the Velvet Revolution of 1989, after which time virtually all exploration and mining activities in the former Czechoslovakia ceased). For these reasons, the writer understands from local geologists that the SE half of the concession is less well explored than the NW part. Consequently, it is recommended that grass roots type exploration activities be concentrated in this area.

9. MINERALISATION

The main mineralised body at Jahodna, based on past work, is like a large but thin, sheet like form – typically 500m x 500m in surface area, but only in the order of 2.5m thick. The deposit is partially blind, rarely outcropping at surface, with the top of the main zone of mineralization occurring about 200m below surface (though this figure is relative since the surface in this area undulates from some 500m to 630m amsl), extending for some 500m in a down dip direction. The upper half of the deposit has a dip of about 60^0, and the lower half a dip of about 45^0.

Basically, it would appear that the uranium mineralisation represents secondary type mineralisation localized along foliation and within ptygmatically folded quartz-carbonate veins. The main reason for this observation is that the majority of the mineralisation previously described from Jahodna occurs as veins, veinlets, or other open space fill. Mineralised zones have a clear lithologic and structural control. For example, mineralization is stratabound along the contact of the hanging wall meta-andesite unit and the foot wall meta-sediment unit and is localized in folded fracture-fill veins and along foliation planes.

Another interesting point, not yet quantified, is that the spatial position of the main deposit seems to indicate the importance of big cross faults in the area. At least 2 big cross faults (with ENE orientation and apparent dextral throws of up to 20m) occur in the vicinity of the deposit. What factors control and delimit the margins or the deposit are not yet known, but one possible one is distance from mineralising cross faults. With the tendency of the main Jahodna deposit to be spatially associated with cross faults, this may suggest that the cross faults were the original conduits for the U-Mo-Cu mineralisation to be transported into the vicinity. Again, considerably more detailed exploration work would be needed to confirm this point.

Regarding the detailed mineralogy of the deposit, the following data is taken largely from the report by Josef (2005). Based on historic work at Jahodna, the main mineralised minerals are molybdenite, uraninite, brannerite, U-Ti oxides and subordinate coffinite, with main accessory minerals being abundant pyrite and subsidiary chalcopyrite. Based on former petrographic studies of mineralized drill samples, the following minerals were shown to be associated with the Mo-U-Cu mineralization: molybdenite, uraninite, U-Ti oxides, brannerite, coffinite, chalcopyrite, tennantite, pyrite, marcasite, galena, chalcocite, bornite, covellite, hematite, rutile, leucoxene, apatite, barite, malachite, goethite, iron-dolomite, calcite, quartz, sericite and chlorite (see Plate 13).

Molybdenite is the dominant mineral. It occurs as veinlets and aggregates in association with chlorite, quartz and sericite. It also commonly occurs together with uraninite, brannerite and pyrite at the contact with altered andesite and crosscutting carbonate veinlets. Molybdenite is also found associated with the uranium minerals and pyrite. U-Mo mineralization cuts Fe-dolomite veinlets, calcite and quartz, latter with younger sulphides (chalcopyrite, pyrite and tennantite).

Metal concentrations are variable and high. From the lithogeochemical studies (drill holes 1247 and 1248), the following contents were detected: 660-4500 ppm Mo, 750-18700 ppm U, 23-765 ppm Cu, 48-393 ppm Pb, 2669-4070 ppm Ti, 24-248 ppm Ni, 99-256 ppm Zr and 114-214 ppm As. The REE content does not exceed 300 ppm.

10. EXPLORATION

Since officially acquiring the exploration licence in question in March 2005, the issuer (Tournigan Gold Corporation) have undertaken no new exploration work other than drilling three diamond drill holes on the property (see description below).

11. DRILLING

Since acquiring the exploration licence, the issuer has drilled 3 relatively deep diamond drill holes on the Jahodna property. The purpose of these holes was to provide preliminary confirmation of the thickness and average grade of the deposit. (see Map 13 for hole locations).

The drilling was undertaken by Geo Technical Consulting of Bratislava. They used a wireline type Prospector II drill for the shallow drilling from surface on all three holes. This track-mounted drill used PQ size equipment, to drill in the region of 100m in each hole. Thereafter, a Longyear 38 drill was used, drilling HQ sized core as deep as possible, and thereafter reducing to NQ.

In view of the difficult drilling conditions (ie. caused by steeply dipping bedding and cleavage planes), the drilling speed was reduced in order to improve the core recovery (average daily metreage achieved was 23m / day). In addition to this, an organic polymer (Premix type, made in France) was mixed with water and used throughout the drilling programme. These precautions helped to maintain a high standard of core recovery throughout the 3 hole programme (ie. 96-98% recovery overall, or almost 100% in the fresh rock).

These holes are:

KG-J-1 - situated within 20 m of two old holes which gave high grade intersections, being Nos. 1218 and 1222. This hole was drilled as follows:

- 0.0-17.5 m - 137mm
- 17.5-75.0 m PQ
- 75.0-347.0 m HQ
- 347.0-440.4 m NQ

KG-J-2 – collared some 80m SE of KG-J-1, within the broad outline of the Jahodna deposit. This hole was drilled as follows:

- 0.0-15.0 m 137 mm
- 15.0-93.0 m PQ
- 93.0-343.5 m HQ
- 343.5-480.4 m NQ

KG-J-1a – situated adjacent to hole no. KG-J-1. This hole was drilled as follows:

- 0.0-18.6 m 156 mm
- 18.6-97.0 m PQ
- 97.0-351.0 m HQ
- 351.0-444.1 m NQ

Please see Map 13 for drill collar positions, and see Plates 2 to 16.

In the part of the deposit where it was intersected by the drill holes, the dip of the deposit would be in the region of 50 to 60^0 to the SE. The holes were drilled at steep inclinations, starting off near vertical at surface, and shallowing progressively at depth. This would mean that the intersection with the mineralised zone would have been quite close to normal (90^0). For this reason, true width corrections have not been applied to the mineralised intersections from the latest drilling. In addition to this, the Micromine PWRE model incorporates the drill data "as drilled", and effectively turns these into true dimensions when calculating the block model volume.

12. SAMPLING METHOD AND APPROACH

As mentioned above, a wireline coring system was used, with a Prospector rig drilling from surface (PQ core size) and a Longyear 38 rig completing the holes (HQ and NQ size). In view of the known zones of poor ground conditions, the wireline drilling equipment used a double-tube core barrel, in order to maximise core recovery.

The core was geologically logged on site at Jahodna. In addition to this, the mineralised zones were identified with a ZRUP Gamma Logger. Once logging was complete, the core was removed to the company's exploration facility in Kremnica, where the mineralised zones were halved, using a diamond saw. The sample intervals were defined geologically, and on the basis of the gamma logger.

Core recovery was generally very high (always well over 90% average in the mineralised zones, and frequently 100%), and with the gamma logger being used first to define the mineralised zones, it would appear highly likely that all the good zones of uranium mineralisation were identified for chemical analysis.

13. SAMPLE PREPARATION, ANALYSES AND SECURITY

The samples from the first 2 drill holes (KG-J-1 and KG-J-2), totalling 26 core samples, were airfreighted to the OMAC lab in Ireland for analysis. The samples were dried at 85^0C, jaw crushed to -2 mm and the total amount of crushed material was milled using LM2 mill to -100 µm.

Prepared samples were analysed for 45 element suite using MA/ES procedure, which involves digestion of 0.2 g of sample in the mixture of nitric, hydrofluoric, hydrochloric and perchloric acids, bringing solution to dryness and re-dissolving salts in 10 ml of 10% aqua regia solution followed by reading using ICP-OES spectrometer. The samples were also analysed for gold using Au4 procedure that involves fusion of 50 g of sample with lead collection, cupellation, dissolving resulting prill in aqua regia and AA analysis.

Standard QC procedures were applied. 10 % of samples were analysed in duplicate, blanks and reference materials were analysed along with the samples. Certified reference materials of uranium mineralisation BL-1 and BL-2 manufactured by Canmet were used in multi-element analysis. All QC data were included in test reports.

Because the mineralised interval from the 3rd hole (KG-J-1a) was so rich (over 6% U for the whole interval), it was too high grade to be assayed at the OMAC laboratory. Accordingly, it was sent to the Ecochem laboratory in the Czech Republic (owned by ALS Chemex). There they undertook a spectrophotometric determination of uranium (with an ICP determination of other elements). The final determination of uranium grade was by the David-Gray-Eberle titrimetric method.

RECENT INTERSECTIONS

Drill Hole No.	From (m)	To (m)	Interval (m)	U %
0.030%U cut-off				
KG-J-1	406.90	408.10	1.20	0.387
KG-J-2	450.90	452.00	1.10	1.140
KG-J-1a	424.00	424.90	0.90	8.829

14. DATA VERIFICATION

With regard to the original drilling undertaken at Jahodna, it has not been possible for the author to verify this, since no core or other samples of any type remained from this drilling. While in Slovakia in October 2005, the author was able to see the recent Tournigan drilling in action, and to use a ZRUP Gamma Logger to confirm the radioactivity in the dark-coloured mineralised core (this was a minor mineralised zone, not the main ore zone). However, the author was not in Slovakia to see any of the main mineralised intersections from the recent Tournigan drilling, or personally verify them in any other way.

Similarly, the original Jahodna borehole data, when used for the Micromine computer model, could not be verified by ACA Howe. This was because none of these drill samples remained now to verify them.

15. ADJACENT PROPERTIES

Not applicable.

16. MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

Since the previous reserve calculations at Jahodna were based on a large volume of detailed data, parts of which were not available for examination, it was not possible to conduct a full recheck of all data pertaining to the former (historical) reserve calculations. For this reason it was decided to conduct a resource estimate based on limited historical, and current drill data.

Accordingly, a Polygonal Wireframe Resource Estimate (PWRE) was calculated for the Jahodna Resource, using data from thirteen (13) historical drill holes over the project area. The thirteen deep historical drill holes, amounting to some 6,290 m of drilling, covered the main part of the Jahodna deposit, though the wireframe model was somewhat constrained in that it did not have all the deposit edge data used for the earlier estimates. Nevertheless, the PWRE appears to have substantially confirmed the historical tonnage and grade estimates.

The Micromine study is written up in more detail in a separate report by G. White of ACA Howe, to which the reader is referred for further detail.

The PWRE work was conducted in two phases – firstly, using only the original Jahodna drill hole data, and secondly, introducing the three new boreholes drilled by Tournigan Gold Corporation. In both cases, the PWRE appears to have broadly confirmed the historical tonnage and grade estimates.

The recent PWRE resource calculation, incorporating the new Tournigan drilling results, can be summarised as follows:

- (0.030% U cutoff) - 1,256,088 tonnes @ 0.56% U to give 15,510,172 lbs of contained uranium.

Please see attached report on Micromine PWRE study (Appendix 3).

Taken together, it would appear that the recent PWRE work on the original Jahodna data, the new Tournigan drilling and the updated PWRE model have all broadly confirmed the historical tonnage and grade estimates at Jahodna. This resource estimation work classifies the tonnage and grade estimate at Jahodna as an "inferred resource".

Name of Study/Description	Tonnes	U%	Contained Lbs	Comment
Micromine 2006 Study	1,256,088	0.56	15,510,172	0.030% cut-off grade used by ACA Howe

NEW MINERAL RESOURCE ESTIMATES

17. MINERAL PROCESSING AND METALLURGICAL TESTING

Not applicable.

18. INTERPRETATION AND CONCLUSIONS

Geological Complexity – The Jahodna uranium deposit represents a difficult exploration target, for a number of reasons:

- It is a partially blind deposit, with limited surface expression other than a subdued radiometric anomaly, limiting the exploration methods available to explore it.
- It is deep – lying, in the range of 250 to 650m below surface – this means that it will be an expensive target to evaluate, since it will be done mostly by deep drilling from surface, or by shorter holes from underground access.
- The surface in the area of the deposit is hilly, with deep soil cover and thick forest, and the forest appears to be old and well established. At these elevations, snow lies on the ground during the winter, sometimes for more than 3 months per year. A combination of these factors virtually negates surface exposure, and make useful surface mapping harder to achieve.
- The geology of the deposit itself is in some ways complex, and is as yet incompletely understood. For example:
  - Although the main deposit lies at the base of the meta-andesitic unit, its relationship with the other mineralised bodies within the andesite unit are not well understood.
  - There are significant cross faults which displace the mineralised zone by up to several tens of metres, thus effectively breaking up the mineralised body into faulted blocks – it is important that the relative positions of these mineralised blocks be determined more accurately before evaluation proceeds too much further.
  - The same cross faults may have been conduits for the U-Mo-Cu mineralisation, yet the spatial relationship between the cross faults and the uranium deposit has not been properly established.

- o With a replacement – type uranium deposit, which appears to have exploited zones of deformation related permeability within the Permian meta-volcanosedimentary sequence, the detailed grade distribution within the mineralised zone is likely to be complex – but the drilling to date appears to have been too irregularly and widely spaced to clearly quantify such variability.

- o Regarding geological factors which might affect the development of the Jahodna resource, an obvious one is the existence of significant cross faults – these faults run across the main deposit, and locally may offset it by up to several tens of metres. At the least, the positions of these faults are likely to delimit sections of a future underground mine, so detailed knowledge of their positions will likely become vital in the ongoing evaluation of the deposit. At worst, the cross faulting may render parts of the deposit inaccessible for mining purposes.

- o With regard to the ways in which the Jahodna resources might be affected By non-geological factors (eg. any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues), the only one of which the author is aware is that the deposit occurs within a forest reserve. However, Tournigan staff in Slovakia claim that this is not a critical or prohibitive issue with regard to the future of the project.

Other Geological Features of the Jahodna Deposit

- It is possible that the likely source of the uranium mineralisation at Jahodna was the Permian acid-intermediate volcanic sequence, with the U-Mo-Cu mineralisation infiltrating out from the volcanic centres into suitable structural / lithological traps during deformation and metamorphism. In the case of Jahodna itself, the most obvious depositional loci would appear to be zones of permeability, both within the meta-andesitic volcanic unit and in the foliation planes and localized shearing along the basal contact of the meta-andesitic unit.

- The complex mineralisation at Jahodna comprises U-Mo-Cu, with lesser Fe-Pb-Ti-P-Ba. The mineralisation at Jahodna has been described as a "Saddle Hills analogue", after the Saddle Hills uranium deposit in Mongolia. Saddle Hills contains uranium-rutile-leucoxene mineralisation in a variety of Mesozoic volcanosedimentary lithologies. Although there is insufficient detailed information to back this up, there would appear to be some broad similarities between Jahodna and the Saddle Hills type deposits.

- It appears that the good mineralisation at Jahodna is generally clearly visible, with dark brown and grey / black mineralisation resulting from hematite, graphite, chlorite, the uranium mineral brannerite, as well as pyrite, chalcopyrite, molybdenite etc. Obviously, this is helpful for core logging and sampling purposes.

- The majority of the rocks at Jahodna are meta-volcaniclastics/tuffs of acid-intermediate type, being predominantly andesitic in nature. Most of the rocks are strongly foliated, and therefore schistose, with schistosity generally appearing to parallel the now steeply dipping original bedding planes of the volcanosediments. This combined bedding / schistosity gives rise to poor ground conditions in some of the rock types encountered in the vicinity of the Jahodna deposit, which would impact on the underground exploration and possible future mining stages.

Positive factors on the Jahodna Project

- After over 20 years of being depressed, the international uranium market is riding high at present, and shows every indication of continuing to do so. Conventional major sources of power (coal, hydrocarbons etc) will not be able to meet the worldwide energy requirements, making the case for a return to nuclear power very strong. Furthermore, uranium enjoys some significant advantages over coal and hydrocarbons – ie. no atmospheric pollution, significant ability to recycle nuclear fuels, relatively abundant known undeveloped resources etc.

- Slovakia is a traditional mining country, whose state-subsidised mining industry suffered devastating cutbacks following the cessation of state mining subsidies after the Velvet Revolution in 1989. The general impression is that the Slovak Government are keen to promote a return to mining, provided it can be financed by free market sources.

- Following the switch to communism in 1948, and the consequent effective economic embargo with the West, the Slovakian authorities encouraged and financed vigorous exploration programmes for a wide range of mineral commodities. This means that a large amount of regional scale grass roots exploration has already been done over much of the country, and a large number of mineral prospects (such as Jahodna) have already been discovered.

- Since the deposit occurs within a forest reserve, its relatively deep nature is probably an advantage in that an underground mine in this situation would more likely receive planning permission from the regulatory authorities than an open pit mine.

Past Exploration Work

- In many ways, it is to the credit of Uranovy Prieskum that they discovered the Jahodna deposit at all. Such are the complexities of the setting of this deposit that it was only after considerable exploration drilling that it was discovered in the 1980's. Following airborne radiometric surveys and ground follow up radiometric surveys, the best way forward was found to be to systematically drill the airborne and ground anomalies. However, it must be borne in mind that at that time, there was a high level of government support for mineral exploration, especially for a commodity like uranium, which would have been identified as a commodity of strategic importance during the Cold War. For this reason, considerable sums must have been expended on the exploration for strategic commodities like uranium. Apparently on the Jahodna project, a total of 53 exploration holes, totalling over 17,000m of drilling, were completed between 1985 and 1991.

- During the Soviet-dominated communist era, many mining operations were commenced and operated more for strategic than for solely economic reasons. As such, it is likely that less economic parameters were considered during the exploration and development of most mineral deposits in the USSR, than for equivalent mining operations in the West. For this reason, the detailed economics of mining an underground deposit like Jahodna would probably never have been studied previously.

- Having established that deep drilling was perhaps the only way to evaluate the Jahodna deposit, it has to be said that the drilling undertaken by Uranovy Prieskum was unsatisfactory in some key respects. In the days of pre-wireline drilling equipment, the thin-walled, conventional B-size drill string deviated considerably from the targets, often intersecting the mineralised horizon many tens of metres or more from the intended point of intersection. In a deposit where it would appear that post-mineralisation normal faulting has broken up the mineralised zone into blocks, the positions of which are likely to be of critical importance at the mine planning stage, this lack of directional drilling control was unfortunate.

- Perhaps largely because of the directional problems with the earlier drilling, the resulting drill cross sections of the original evaluation were overall not normal but somewhat oblique to the plane of the deposit. Although such problems frequently arise in early exploratory drilling on a new prospect, they should generally be rectified in the more advanced stages of exploration.

- Furthermore, in drilling a strongly foliated, tuffaceous meta-volcanosedimentary sequence, the core recovery of the drilling was generally poor. This was unfortunately particularly the case in the mineralized zone, where shearing on the base of the meta-andesite unit resulted in strongly broken ground, and consequently core recoveries in the region of no more than 50% (though this was improved towards the very end of the drilling after 1990). For this reason, there were apparently no complete mineralized zone intersections from the original 14 holes drilled in the immediate vicinity of the Jahodna deposit. In any surviving core samples from the mineralised zone, variable amounts of available material was taken for chemical assays, resulting apparently in no drill samples of any description remaining to be available for inspection today. In a potentially significant, high grade uranium prospect, with a total of 17,000m drilled on the entire exploration licence area (equating to well over USD 1 million in today's costs), this was an unfortunate shortcoming.

- With a record of persistently poor core recovery in the mineralised zone, the only way to attempt to establish the average widths and grades of the mineralised zone was by down hole radiometric logging. Fortunately this technology appears in itself to have worked well – though it has to be emphasised that there were no complete, assayed borehole intersections at Jahodna against which to calibrate the gamma logger response. However, all aspects of the tonnage and grade calculation at Jahodna appear to have adhered to strict technical guidelines set down by the national mining regulatory bodies at that time, and also draw on former radiometric logging experience of drilling and mining similar deposits within the country.

- In terms of systematically establishing the true average grades of the mineralised zone at Jahodna, or important geological aspects like the detailed positions of cross faults, the earlier drilling undertaken by Uranovy Prieskum would in my opinion have to be regarded as somewhat preliminary in nature.

- Because of the generally poor outcrop at surface, the reconstruction of the late, cross cutting faults has been very much on a 'best fit' basis, with uncertainties regarding their exact positions and throws. Since these cross cutting faults effectively break up the deposit into different segments, the positions and offsets of these faults will be of critical importance in a potential future mining scenario. Since it would appear that the former tonnage and grade calculations have not taken into account the positions and offsets of these faults, it is recommended that the tonnage and grade be recalculated bearing this in mind.

- Within the area of the Jahodna deposit itself, the past drilling has been both irregularly and quite widely spaced. Within the best area of mineralisation, which is the deeper, NW part, drill density works out at approx. 8 intersections within an area pf 100,000m^2 – equating to drilling on a 110m x 110m grid. Within the wider deposit, the equivalent figures are 14 intersections within an area of approx 250,000 m^2, equating to drilling on a 130m x 130m grid. It would seem that by the traditional Soviet type reserve classification for a stratiform but irregular replacement type body, these would have equated to a "C_2 reserve" or an "inferred resource" in current Western terminology (eg. JORC type).

- One prominent feature highlighted by the drilling at Jahodna is the poor ground conditions existing in much of the ground near the deposit itself. However, these bad ground conditions appear to be largely restricted to the thick hanging wall volcaniclastic sequence. Although the mineralised zone itself appears to be badly tectonised, it seems to be largely supported by the more competent andesitic body which immediately overlies it.

Former Resource Calculations

- The first major resource calculation was undertaken in 1996 by Jozef Daniel, a geologist experienced in the uranium industry of the former Czechoslovakia. The method used a block model method, with two variants based on different minimum cutoff grades (0.015% and 0.03% U). A total of 14 drill holes were used in the model, with average uranium grades established largely by downhole geophysical logging. The calculation methodologies were constrained by government mining directives, established in 1987 and updated in 1992. The calculation concentrated on the main deposit, which was placed in the category of "Z-3 supposed reserves" the lesser zones of mineralisation in the hanging wall of the main deposit were assigned to the lower "prognostic" category.

- The tonnage and grade calculation was updated in 2005 by the same author, in a large report entitled "Calculation of Reserves Deposit Kosice I, U – Mo Ore", which was translated into English. This update derived the following summarised figures:
    - Variant I – (0.015% U Cutoff) – 2.188 Million tonnes at 0.329% U to derive an in situ figure of 15.876 Million lbs of U.
    - Variant II – (0.030% U Cutoff) – 1.396 Million tonnes at 0.472% U to derive an in situ figure of 14.528 Million lbs of U.

- Molybdenum grade calculations were made as well as uranium (with a calculated average figure of 0.38% Mo). However, with the poor core recovery, and lack of an alternative method for determining Mo grades, the volume of data available to determine Mo grades was reduced, for which reason the Mo grades calculated are regarded as less reliable than the uranium. There is also an unresolved question regarding the detailed distribution of the Mo mineralisation within the Jahodna deposit.

Recent Exploration Drilling by Tournigan Gold

- Three deep diamond drill holes (totalling 1,364.90m) have recently been drilled by Tournigan into the Jahodna deposit. The purpose of these holes was to provide preliminary confirmation of the thickness and average grade of the deposit. The drilling was undertaken by Geo Technical Consulting of Bratislava, using a tracked Prospector rig to start the holes (with PQ coring and casing), and a Longyear 38 to complete the holes to final depth.

- In contrast to the earlier drilling of the 1980s, this recent drilling achieved good directional control and excellent core recovery. Mineralised core was delineated with a gamma logger, and subsequently cut with a diamond saw. Half of the mineralised samples were sent for assay, to the OMAC laboratory in Ireland, and the Ecochem laboratory in Czech Republic. Results from this drilling substantially confirmed previous drilled widths and uranium grades.

Recent Resource Calculation

- Since the previous reserve calculations at Jahodna were based on a large volume of detailed data, parts of which were not available for examination, it was not possible to conduct a full recheck of all data pertaining to the former (historical) reserve calculations. For this reason it was decided by ACA Howe to undertake an in-house Micromine computer study of the available Jahodna drill data, in order to make an independent assessment of the historical resources calculation.

- Accordingly, a Polygonal Wireframe Resource Estimate (PWRE) was calculated for the Jahodna Resource, using data from thirteen (13) historical drill holes over the project area… The thirteen deep historical drill holes, amounting to some 6,290 m of drilling, covered the main part of the Jahodna deposit, though the wireframe model was somewhat constrained in that it did not have all the deposit edge data used for the earlier estimates. Nevertheless, the PWRE appears to have broadly confirmed the historical tonnage and grade estimates.

- The PWRE work was conducted in two phases – firstly, using only the original Jahodna drill hole data, and secondly, introducing the three new boreholes drilled by Tournigan Gold Corporation. In both cases, the PWRE appears to have broadly confirmed the historical tonnage and grade estimates.

- The recent PWRE resource calculation, using the new Tournigan drilling results, can be summarised as follows:

  - (0.030% U lower cut-off) -1,256,088 tonnes @ 0.56% U to give 7,034 tonnes or 15,510,172 lbs of contained uranium.

19. RECOMMENDATIONS

- Although surface exploration at Jahodna is made difficult by a number of factors (deep soil cover, very little outcrop, extensive forest cover etc), there are possible ways where further work at surface might be feasible. With the importance of the large cross faults, both as feeder zones and in breaking up the mineralised zone into structural blocks, the positioning of these cross faults with maximum precision becomes critical. One possibility to consider may be to fly high resolution aerial photography during the 'Spring window' – when the snow has melted but before the forest vegetation has grown. Such photography would hopefully be able to see down to the ground surface with sufficient precision so that a stereoscopic study of such photography might position the cross faults with greater accuracy. Another possibility may be to acquire high precision satellite imagery (eg. IKONOS, Aster) at this spring window period, and process the imagery in software such as ER Mapper's Hill Shade function, to maximise surface structural contrasts. Once the cross faults can be positioned at surface with maximum precision, their dips should be projected to depth based on structural knowledge of the area. In this way a working structural model of the area can be developed, to be modified with the results of the ongoing drilling programme. Hopefully in this way, the exploration programme can proceed taking due regard of the detailed positions of the important cross faults.

- If the remote sensing methods mentioned above are not feasible, another method to pinpoint fault structures may be to undertake detailed ground magnetic surveys, though a tight grid (eg. 50m x 10m or 50m x 25m) would be recommended.

- It is recommended that a conceptual mining study be undertaken at Jahodna. The purpose of such a study would be to give exploration guidance, at a relatively early stage of exploration, to an apparently attractive project in a geographic area where mining operations were not previously costed to the same degree as in the west. Such a study would use existing tonnage and grade assumptions, and could examine the likely economic impact of features such as underground mining costs typical for Central Europe; relatively high cost backfill mining scenarios; to what extent depth of mining is a critical cost factor; likely economic viability impact in variations in mineralisation and uranium price trends etc.

- Even though the work to date suggests that the Jahodna deposit can be regarded as an inferred resource, much more exploration work is recommended in order to upgrade the confidence level of the resource to the indicated category. Specifically, a substantial drilling programme is recommended, in order to effectively drill the deposit on 50m centres. Such drilling would add a substantial volume of data, of geological, geotechnical and assay type, to the developing computerised geological model of the deposit. Most of all, it would confirm or deny geological and grade continuity over the approx. 500m x 500m surface area of the deposit.

- In view of the deep, narrow and steep-dipping nature of the deposit, intensive drilling from surface on 50m centres is not recommended. Drilling from surface to give 50m centre coverage would involve some 85,000m of drilling, resulting in a high drilling cost in the region of USD 7.6 million.

- Instead it appears appropriate, following a further limited programme of deposit outline drilling, to embark on a programme of underground development, in order to facilitate a drilling programme from underground. Such underground access would enable 50m centre drilling using only some 39,000m of drilling (as opposed to 85,000m of drilling from surface), and equating to a cost of approx. USD 4.9 million (ie. a 35% cost saving compared to the surface drilling programme). It should also be mentioned that, with an average hole length less than half that of the surface drilling (ie. average length of 211m per hole as opposed to 436m per hole), the drill holes of the underground drilling programme would be generally much more precise in intersecting their target areas.

- To provide the underground access for such a drilling programme, a preliminary costing has been made of an underground development programme. This scenario allows for an access decline of some 2900m, a 500m long horizontal drive in the deposit's hanging wall, 1200m of crosscuts for drilling access, and a 350m ventilation raise. The preliminary costing for this programme has been calculated at USD 5.5 million. Besides enabling access for underground drilling and bulk sampling, the underground access would be laid out such that it could be used for subsequent mining operations.

- Besides the further detailed evaluation of the Jahodna deposit, it is recommended to undertake additional grass roots type exploration within the licence area. This is especially the case in the SE part of the license area, where former systematic exploration did not cover.

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CERTIFICATE OF QUALIFICATIONS

David Pelham
Geologist

28 York Road
Colwyn Bay
Conwy LL29 7EN
United Kingdom

I, David Pelham, do hereby certify that:

1. I am an Associate Consulting Geologist with A.C.A. Howe International Limited, whose office address is 254 High Street, Berkhamstead, Herts HP4 1AQ, United Kingdom.

2. I graduated with a BSc Honours degree in Geology/Geography in 1974 from Derby College of Technology (London University), and an MSc degree in Mineral Exploration in 1982 from Rhodes University (South Africa) and have practiced my profession continuously since 1976.

3. I hold membership in the following mineral industry technical societies:
   Professional Member Institution of Materials, Minerals and Mining
   Gemmological Association
   Small Mining International
   Welsh Mines Society

4. I have practiced my profession as a geologist continually for over 29 years.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the overall preparation of the technical report titled, "Technical Report of Jahodna Uranium Project, Slovakia", dated March 28, 2006.

7. I visited Slovakia for approximately two weeks during October 2005 to review project data, and this included a visit to the Jahodna property.

8. I have not had prior involvement with the Jahodna property that is the subject of the Technical Report. I have had prior involvement with other uranium properties in Botswana, Namibia, South Africa and Niger. The nature of my prior involvement was in the exploration for calcrete type, granitic type, placer type and sandstone-hosted uranium deposits.

9. As of March 29, 2006, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

10. I am independent of Tournigan Gold Corporation, applying all of the tests in section 1.4 of National Instrument 43-101.

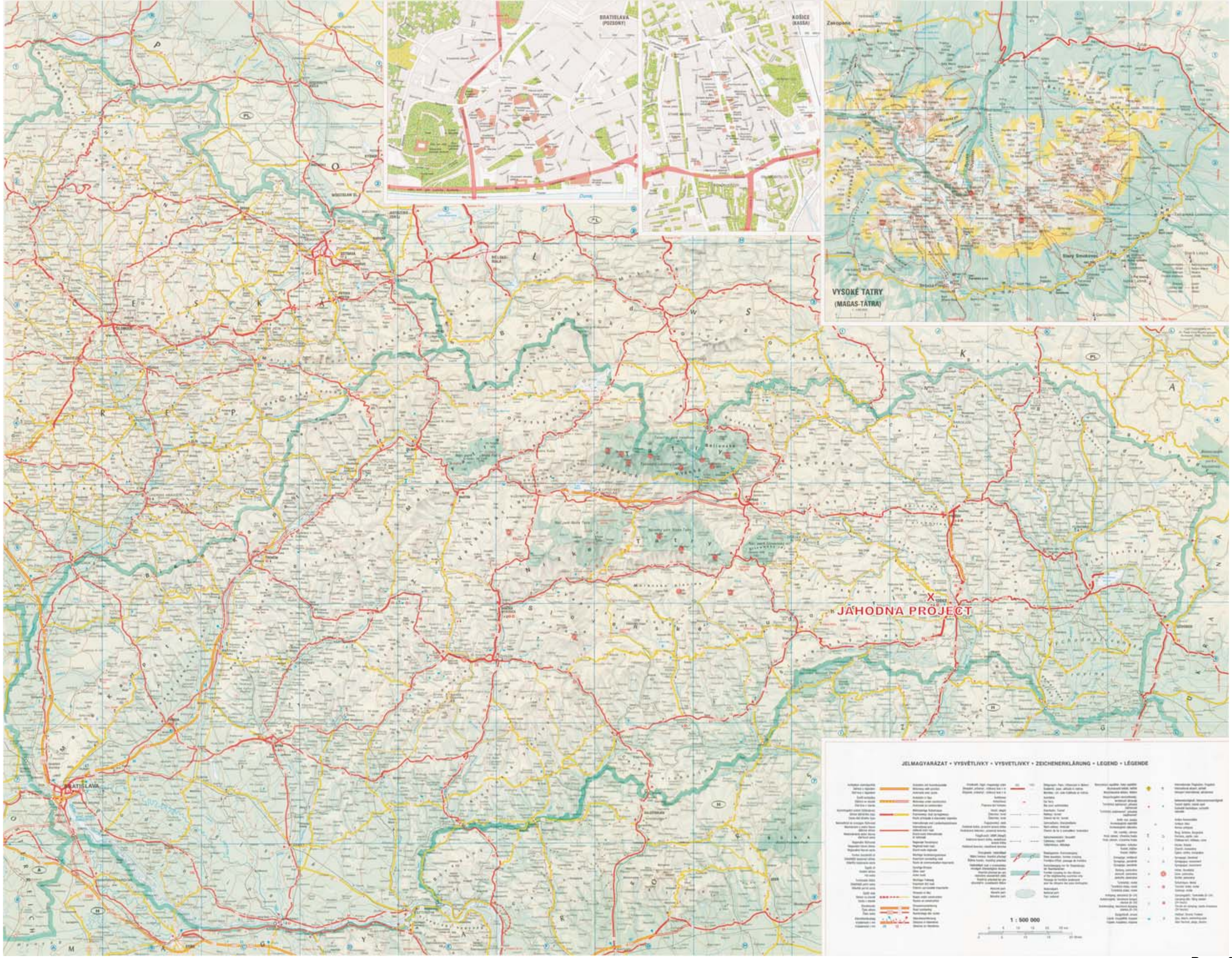
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

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

Signed and dated this 29th day of March, 2006

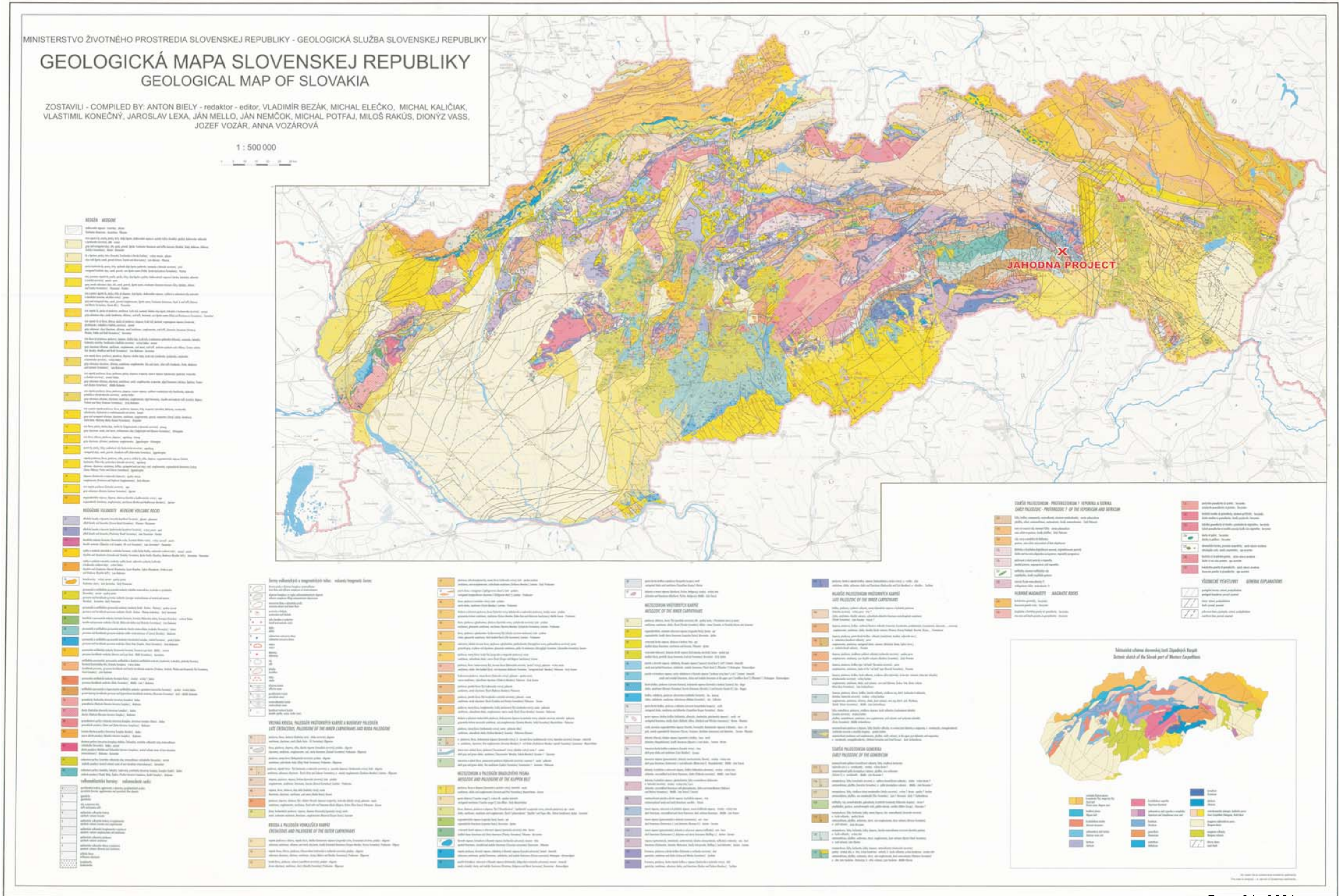
David Pelham, Geologist

APPENDIX 1.



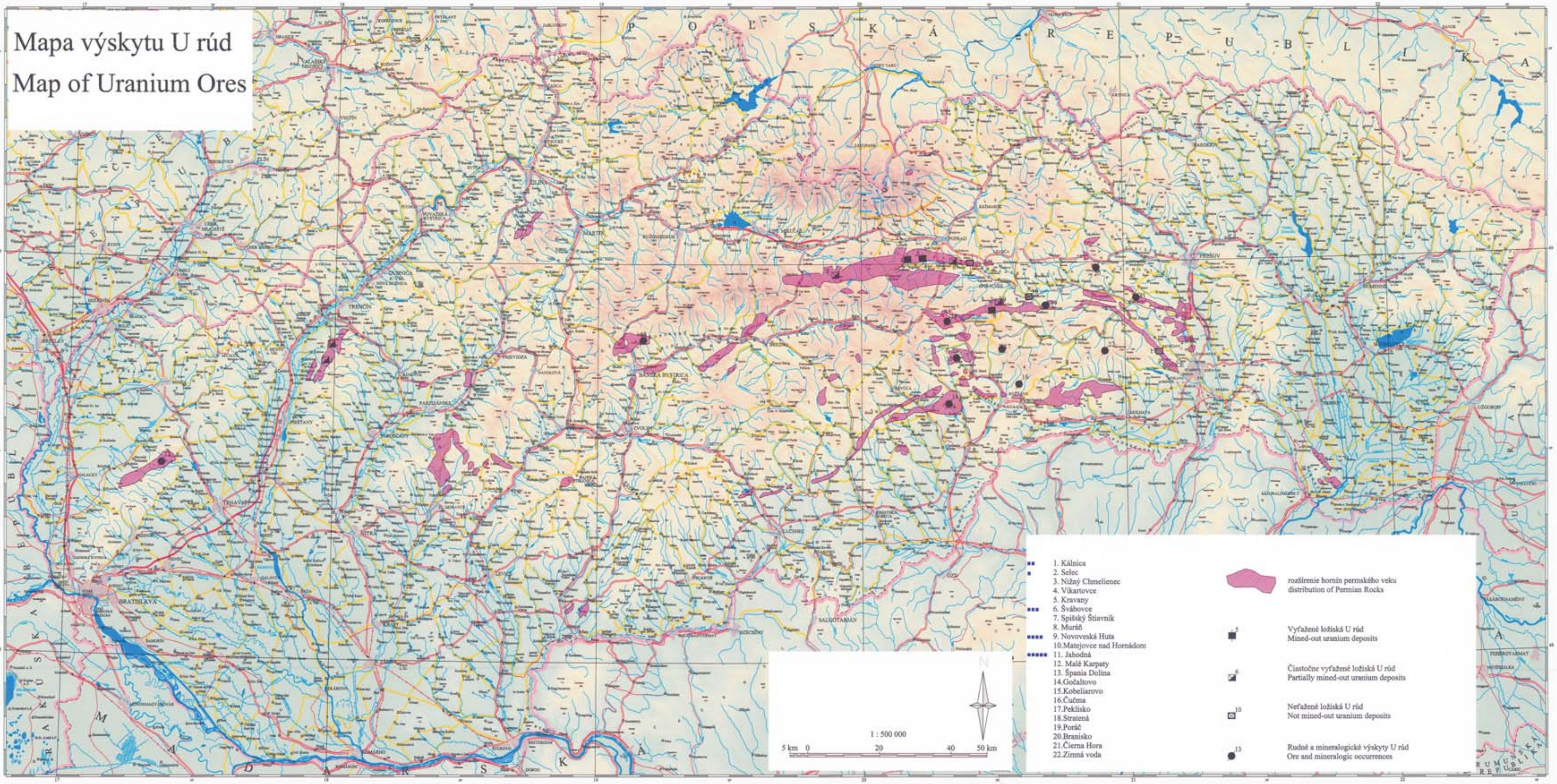
MAP 1- SLOVAKIA ROAD MAP- scale 1:500,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia.

A C A Howe International Limited



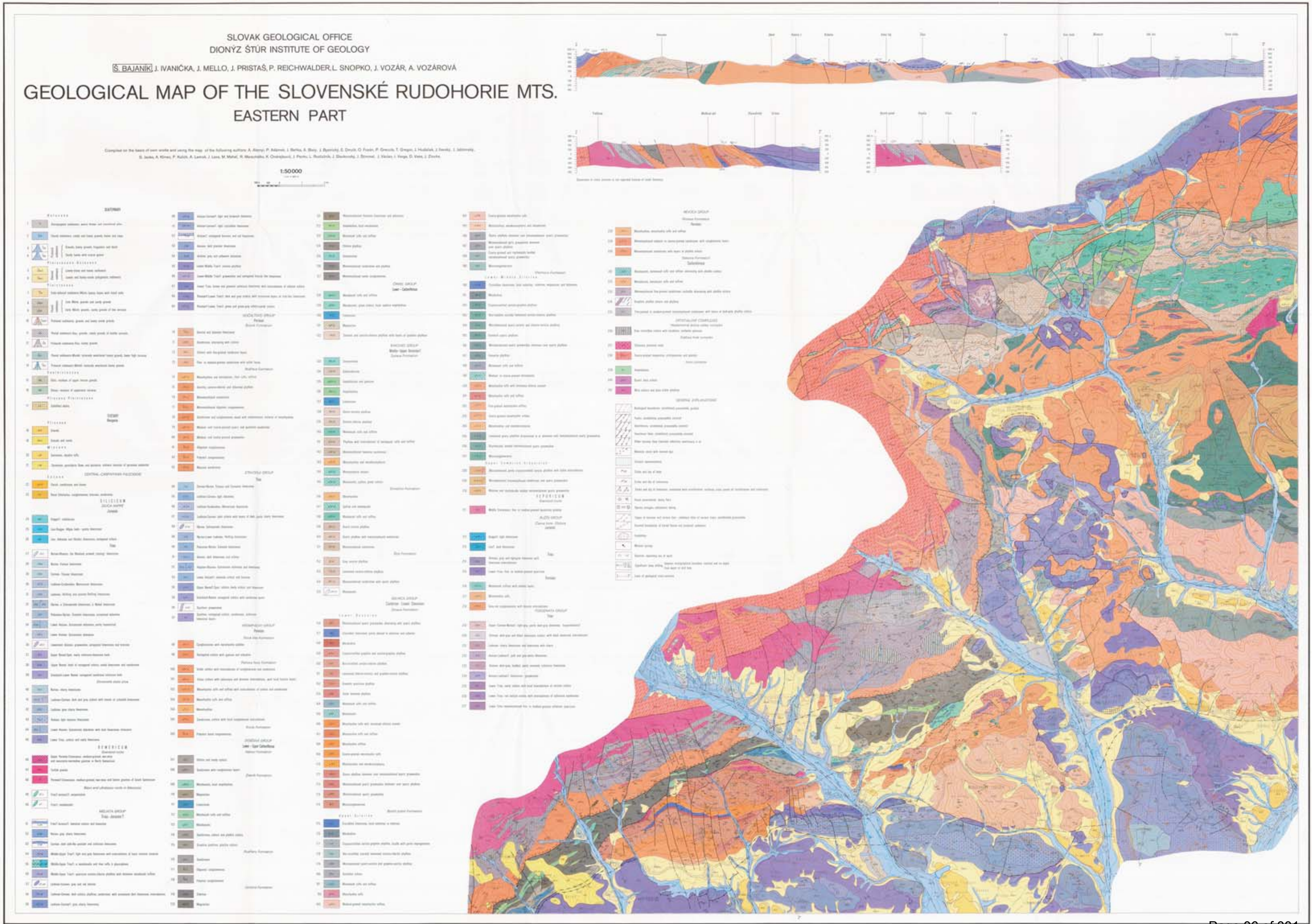
MAP 2- SLOVAKIA GEOLOGY MAP- scale 1: 500,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia.

A C A Howe International Limited



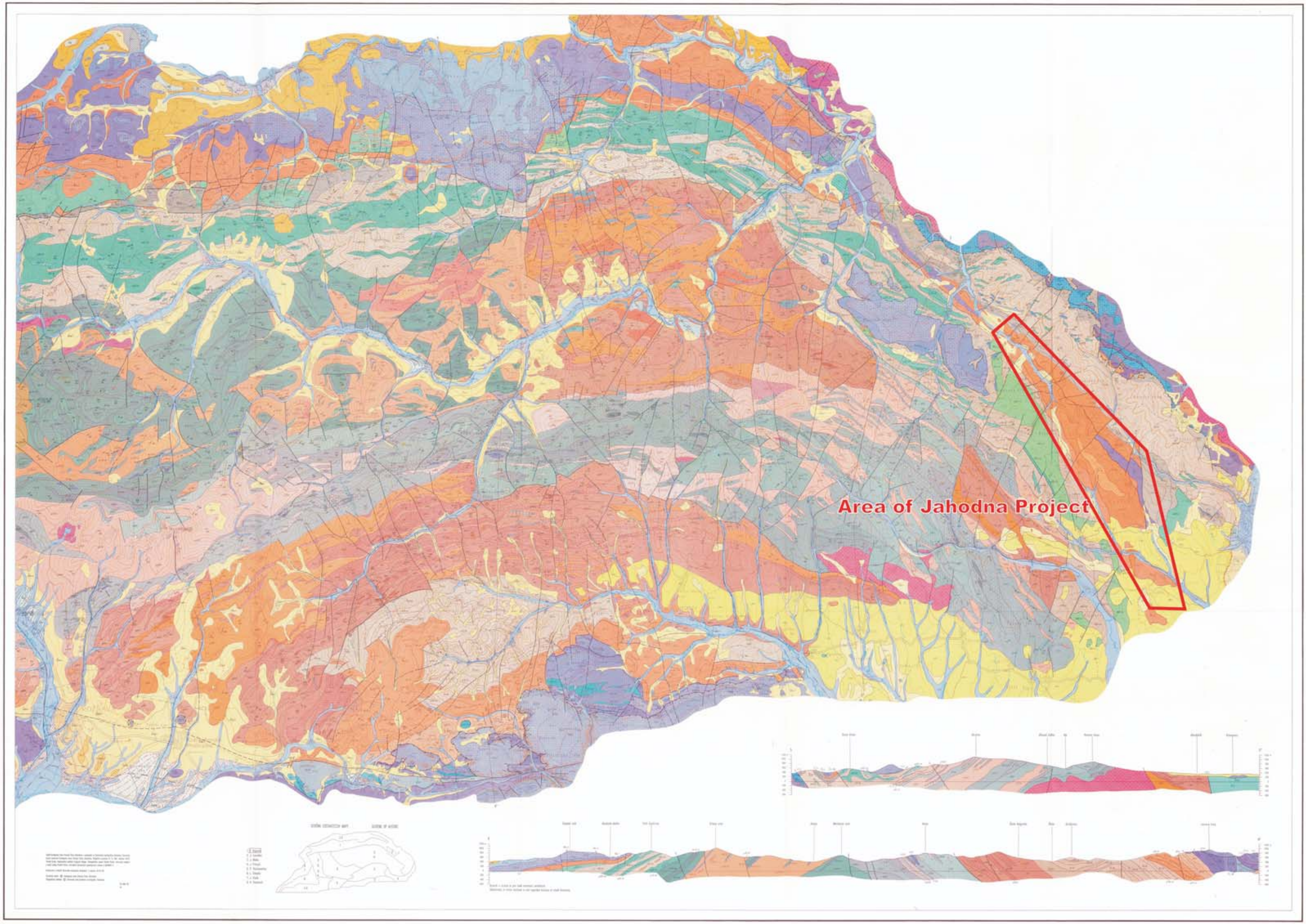
MAP 3- SLOVAKIAN URANIUM DEPOSITS- scale 1:500,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited



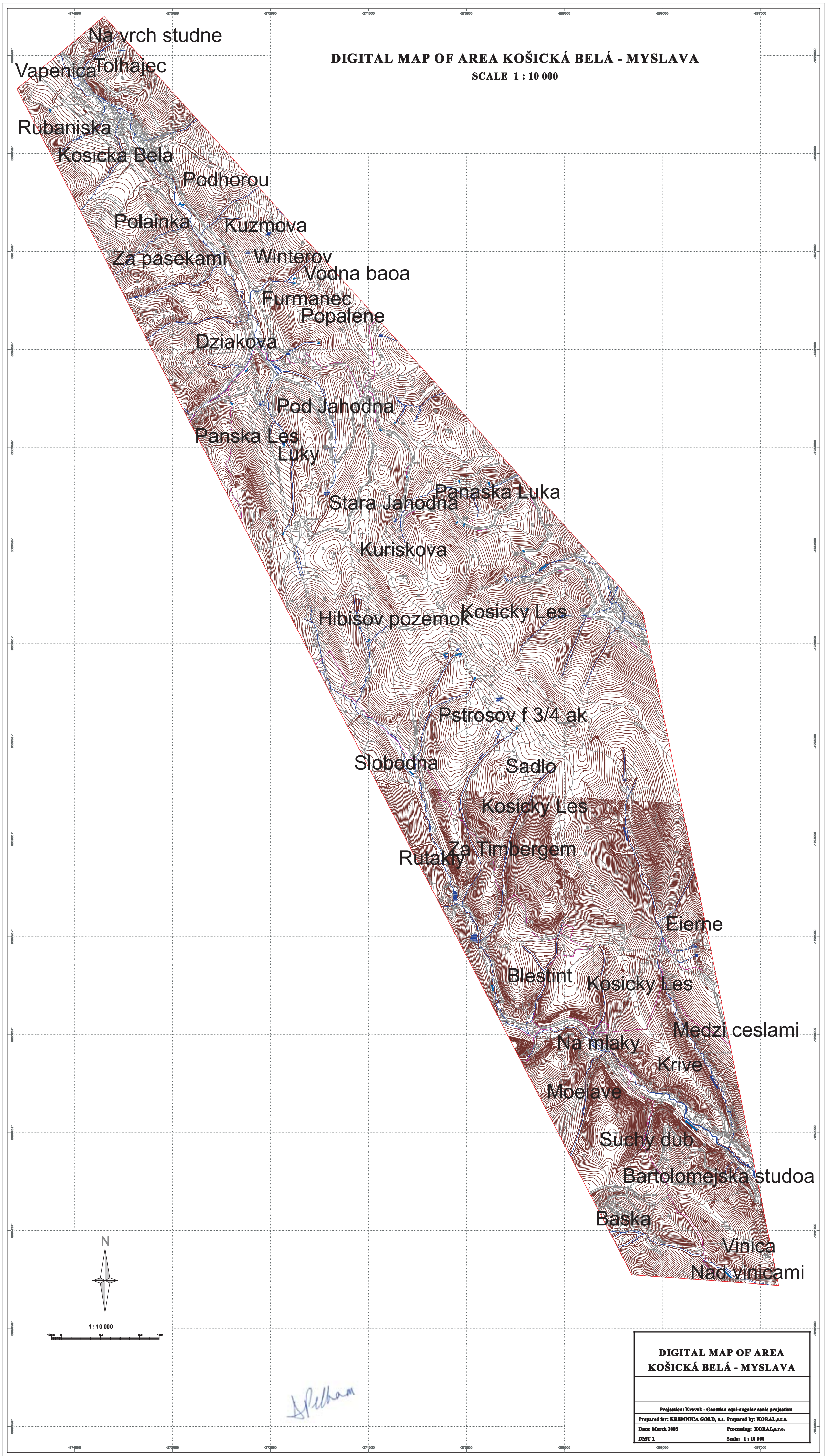
MAP 4- SLOVENSKE RUDOHORIE, WEST SHEET- scale 1:50,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited



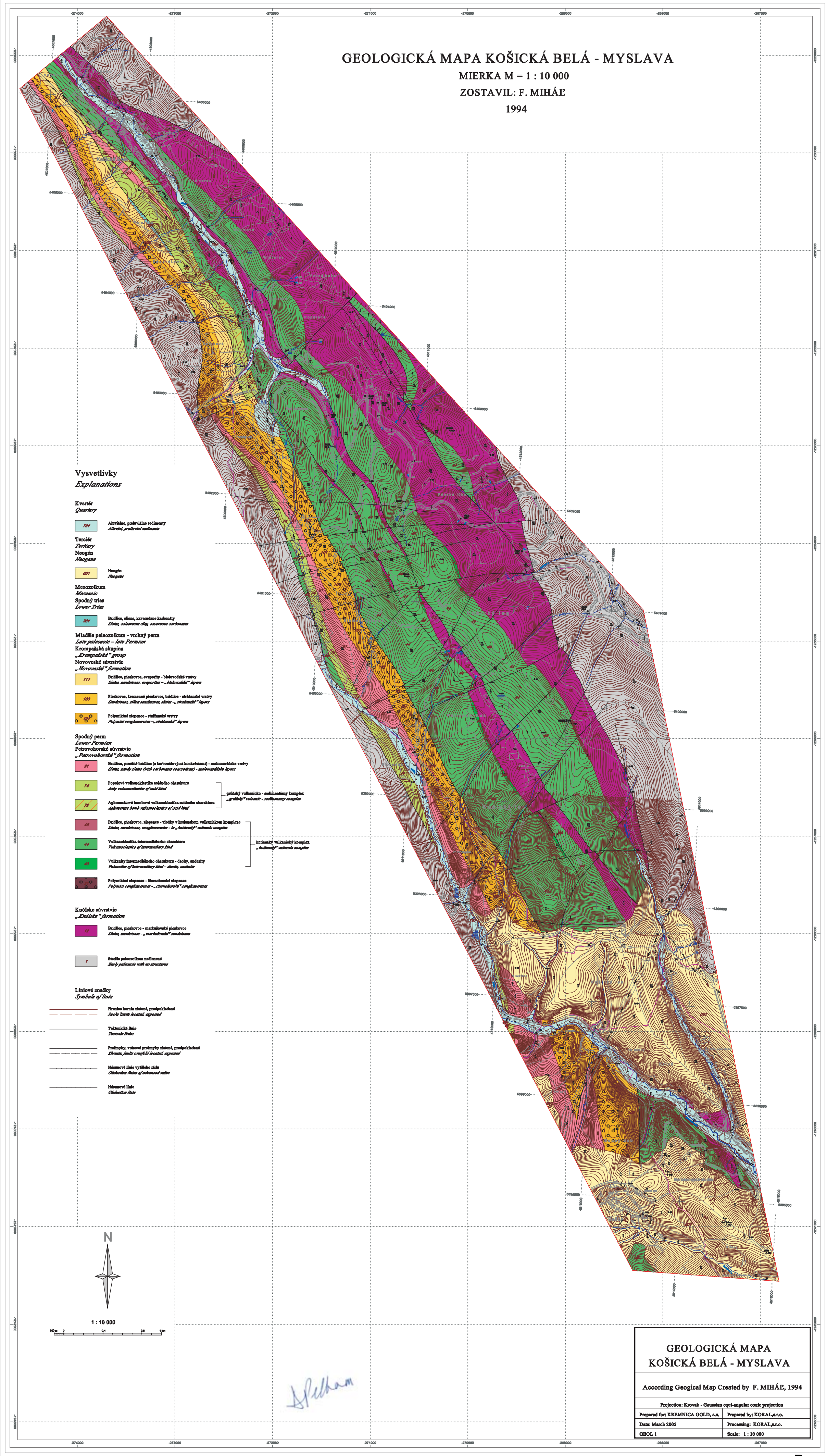
MAP 5- SLOVENSKE RUDOHORIE, EAST SHEET- scale 1:50,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited

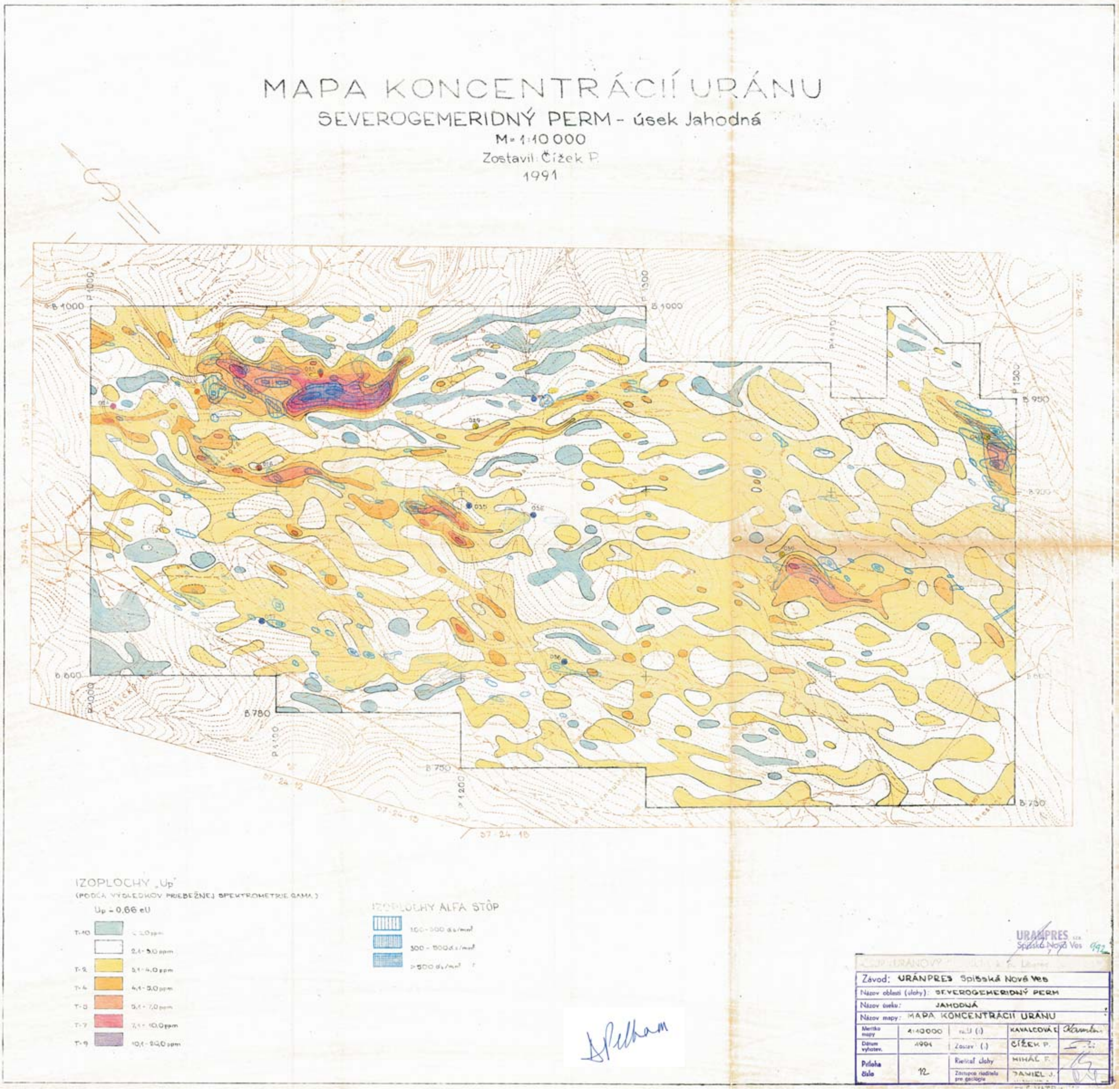


MAP 6- JAHODNA TOPOGRAPHIC MAP- scale 1:10,000
For Tournigan Gold Corporation, Jahodna
Uranium Project, Slovakia

A C A Howe International Limited

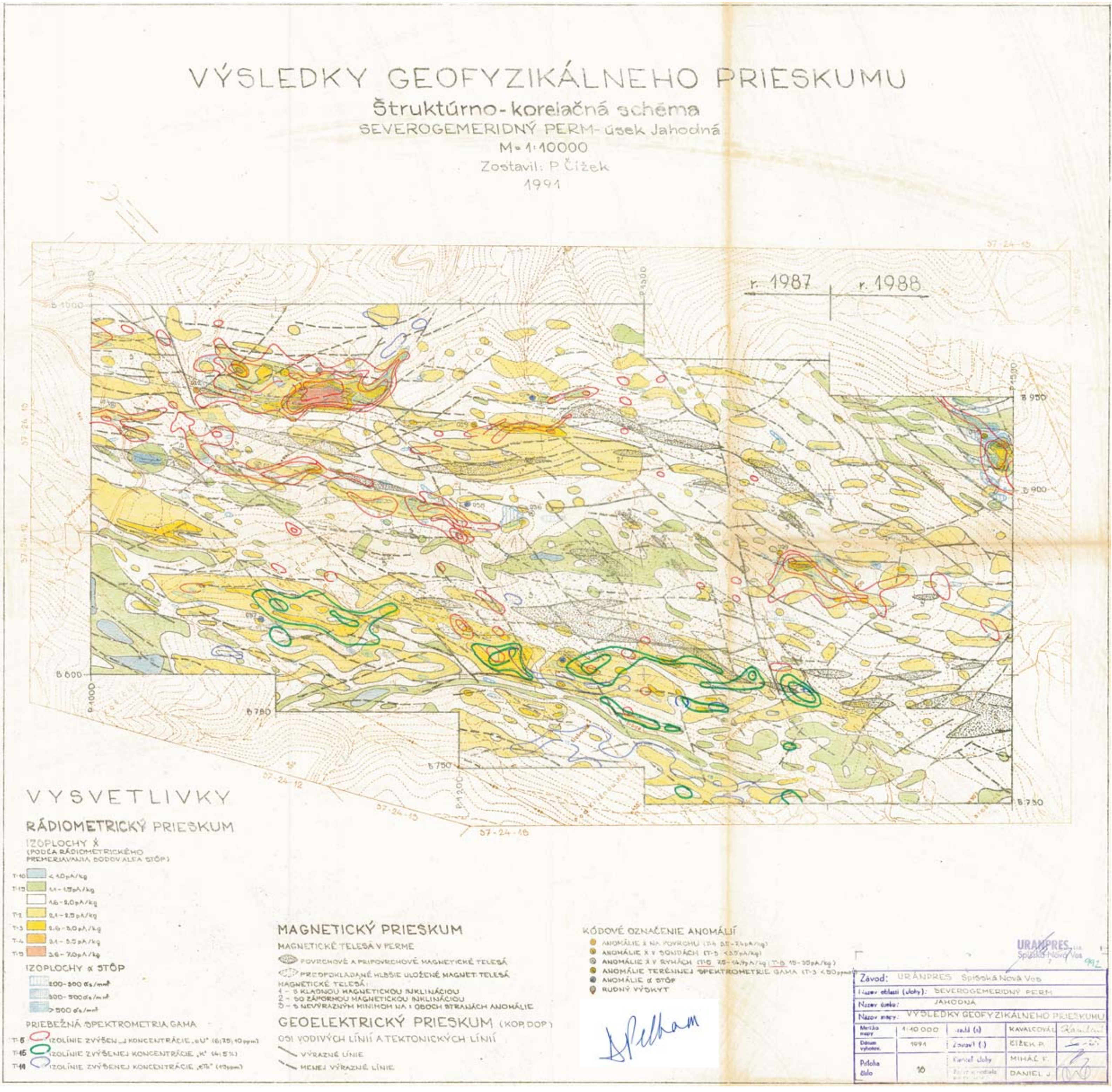


MAP 7- JAHODNA GEOLOGICAL MAP - scale 1:10,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia



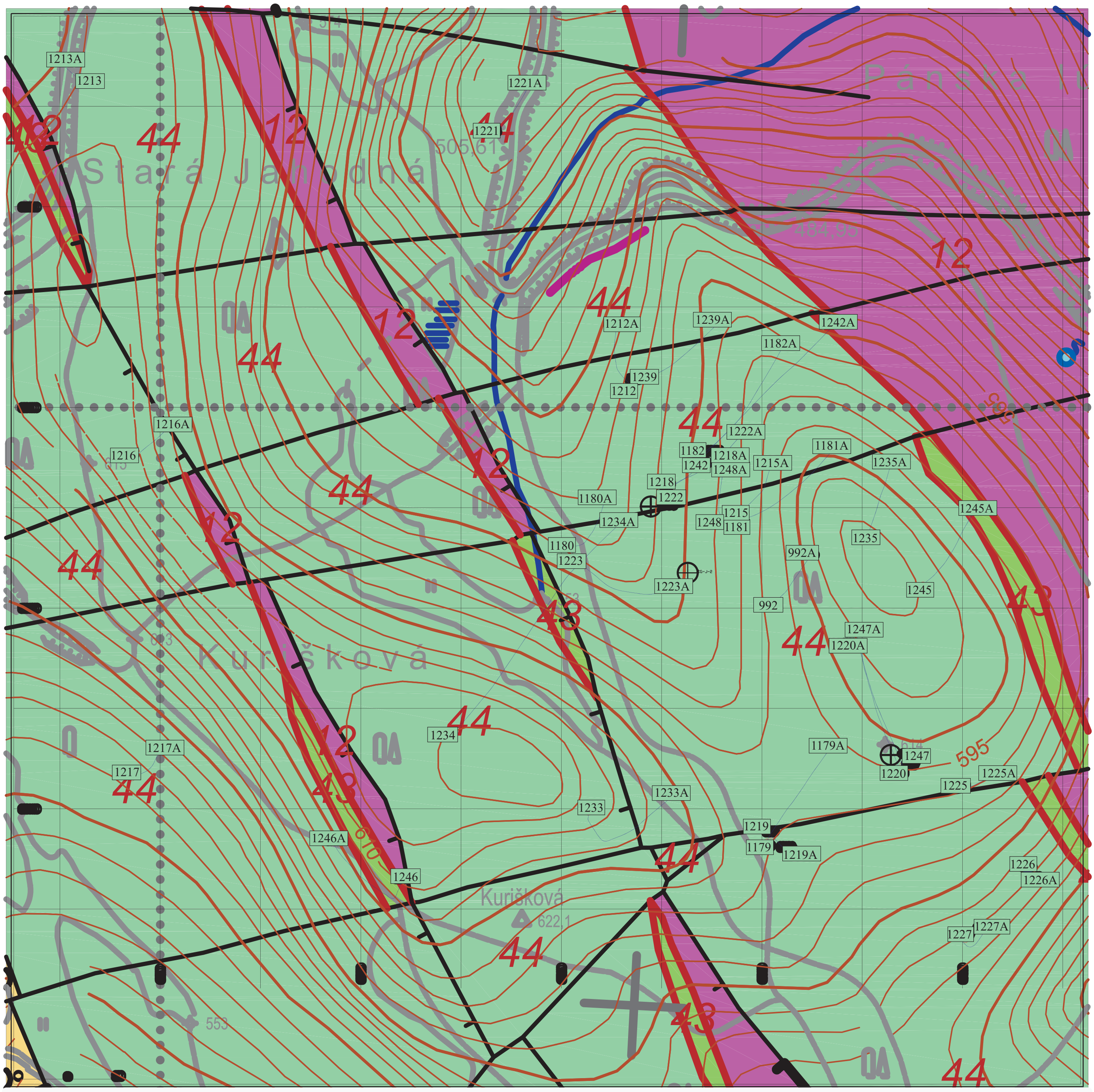
MAP 8- JAHODNA- GAMMA & ALPHA READINGS
- scale 1:10,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited

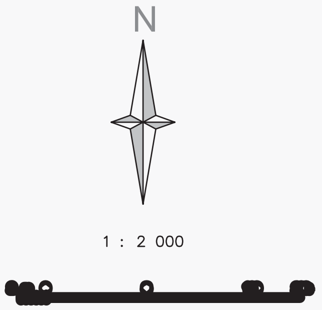


MAP 9- JAHODNA- GEOPHYSICAL COMPILATION PLAN- scale 1:10,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited



12	Bridlice, pieskovce - markušovské pieskovce *Slates, sandstones - „markušovské" sandstones*
44	Vulkanoklastika intermediálneho charakteru *Vulcanoclastics of intermediary kind*
43	Vulkanity intermediálneho charakteru - dacity, andezity *Vulcanites of intermediary kind – dacite, andesite*



GEOLOGICKÁ MAPA
KOŠICKÁ BELÁ - MYSLAVA

According Geogical Map Created by F. MIHÁĽ, 1994

Projection: Krovak - Gaussian equi-angular conic projection

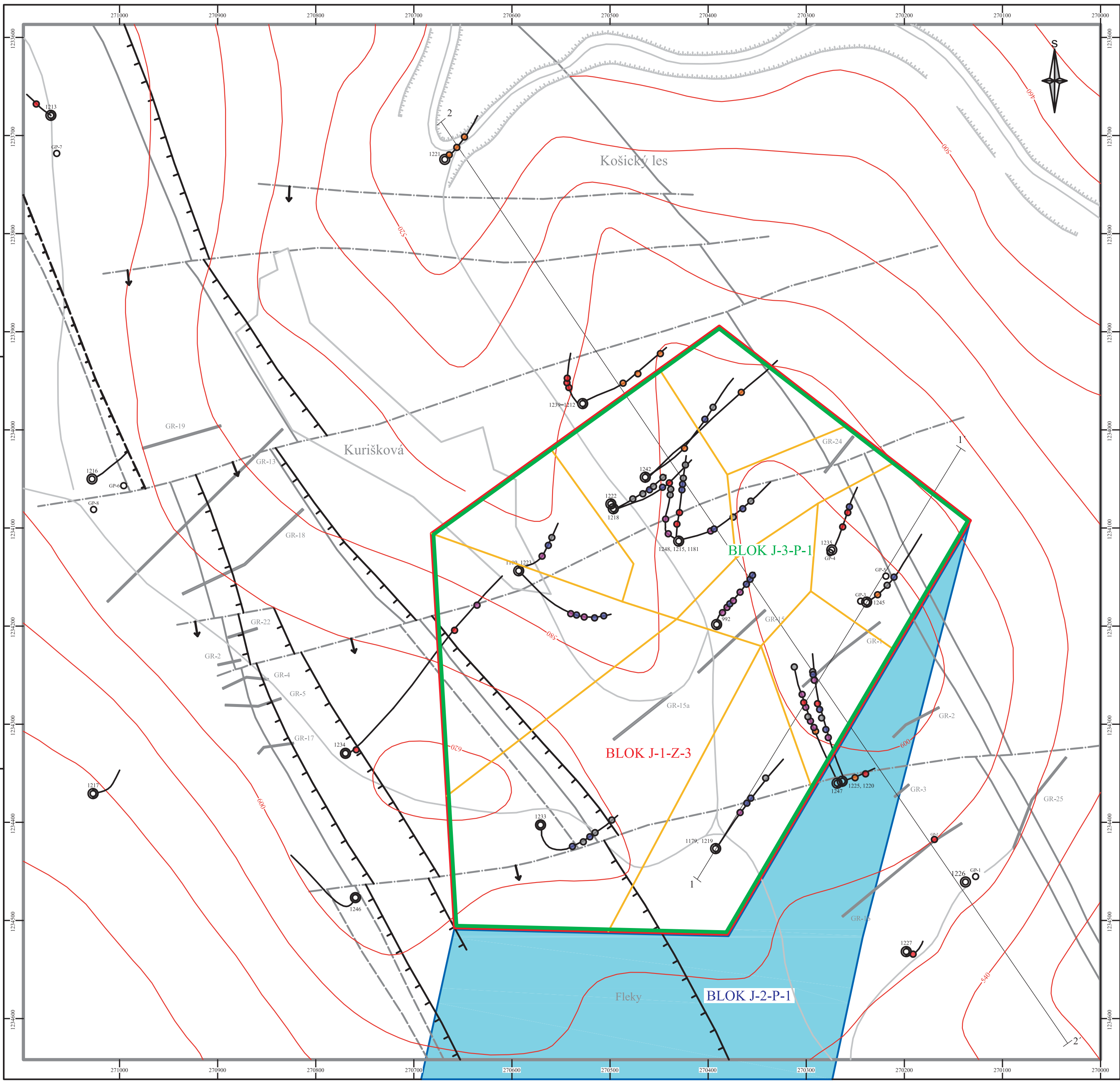
Prepared for: KREMNICA GOLD, a.s.	Prepared by: KORAL,s.r.o.
Date: March 2005	Processing: KORAL,s.r.o.
GEOL 2	Scale: 1 : 2 000

MAP 10- JAHODNA - GEOLOGICAL MAP- scale 1:2,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

MAPA BLOKU ZÁSOB, VARIANT I. A BLOKOV PROGNÓZNYCH ZDROJOV, LOŽISKO KOŠICE I.

M = 1 : 2 000

Zostavil: F. Miháľ, 1991 a J. Daniel, 2005



Vysvetlivky:

- pôsobnosť vrtov
- hranice bloku J - 1 - Z - 3
- hranice bloku J - 2 - P - 1
- hranice bloku J - 3 - P - 1

Poznámka: iné vysvetlivky sú na prílohe č. 1

Blok J-1-Z-3	
Variant I.	S = 339 500 m^2
	m = 2.38
	c = 0.304%
	V = 808 010 m^3
	Q = 2 197 787t
	K = 6 681t U

Príloha č. 2

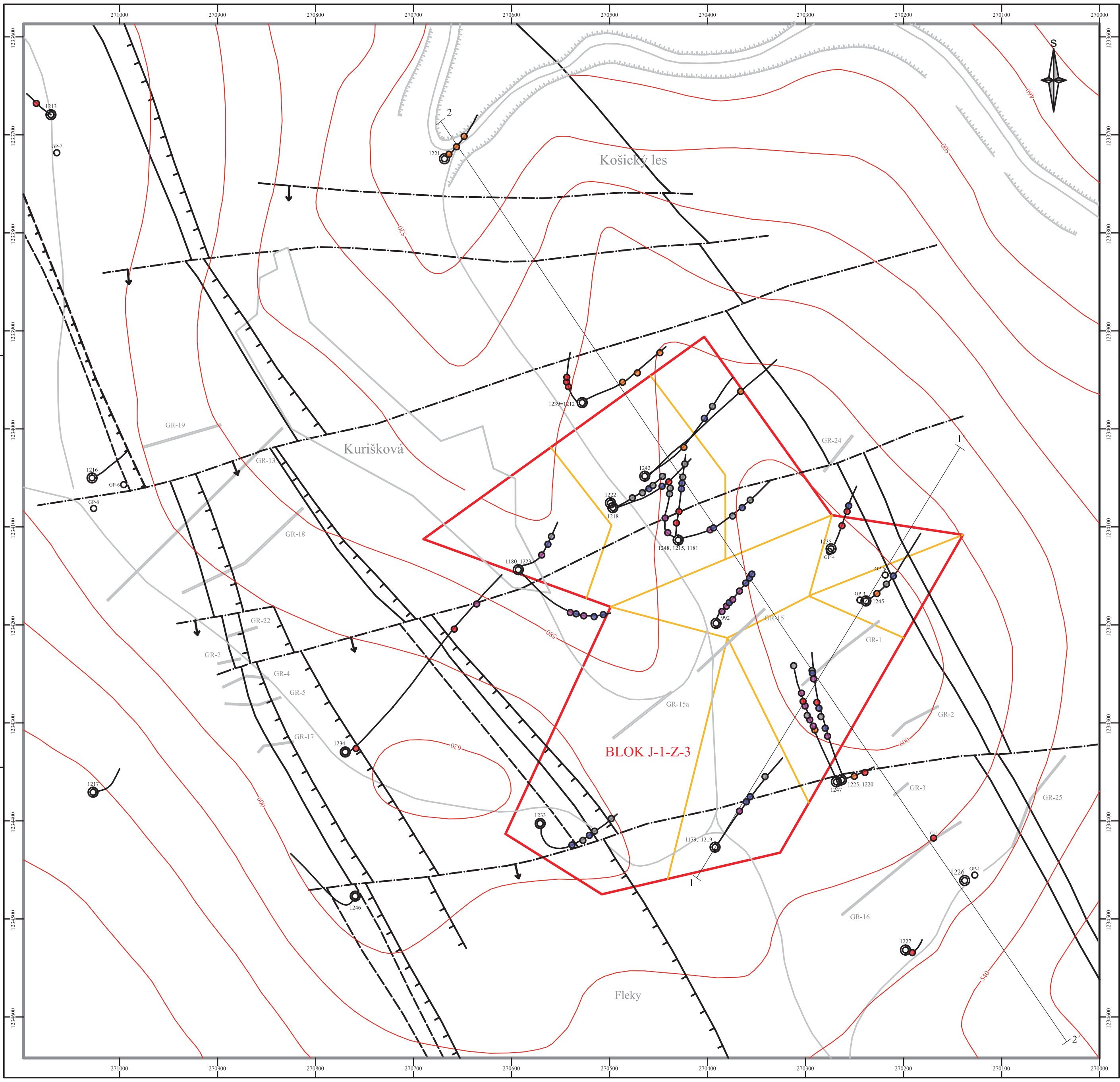
MAP 11- JAHODNA RESERVES- RESERVE BLOCKS
- VARIANT I - scale 1:2,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited

MAPA BLOKU ZÁSOB, VARIANT II., LOŽISKO KOŠICE I.

M = 1 : 2 000

Zostavil: F. Miháľ, 1991 a J. Daniel, 2005



Vysvetlivky:

hranice bloku J-1-Z-3

pôsobnosť vrtov

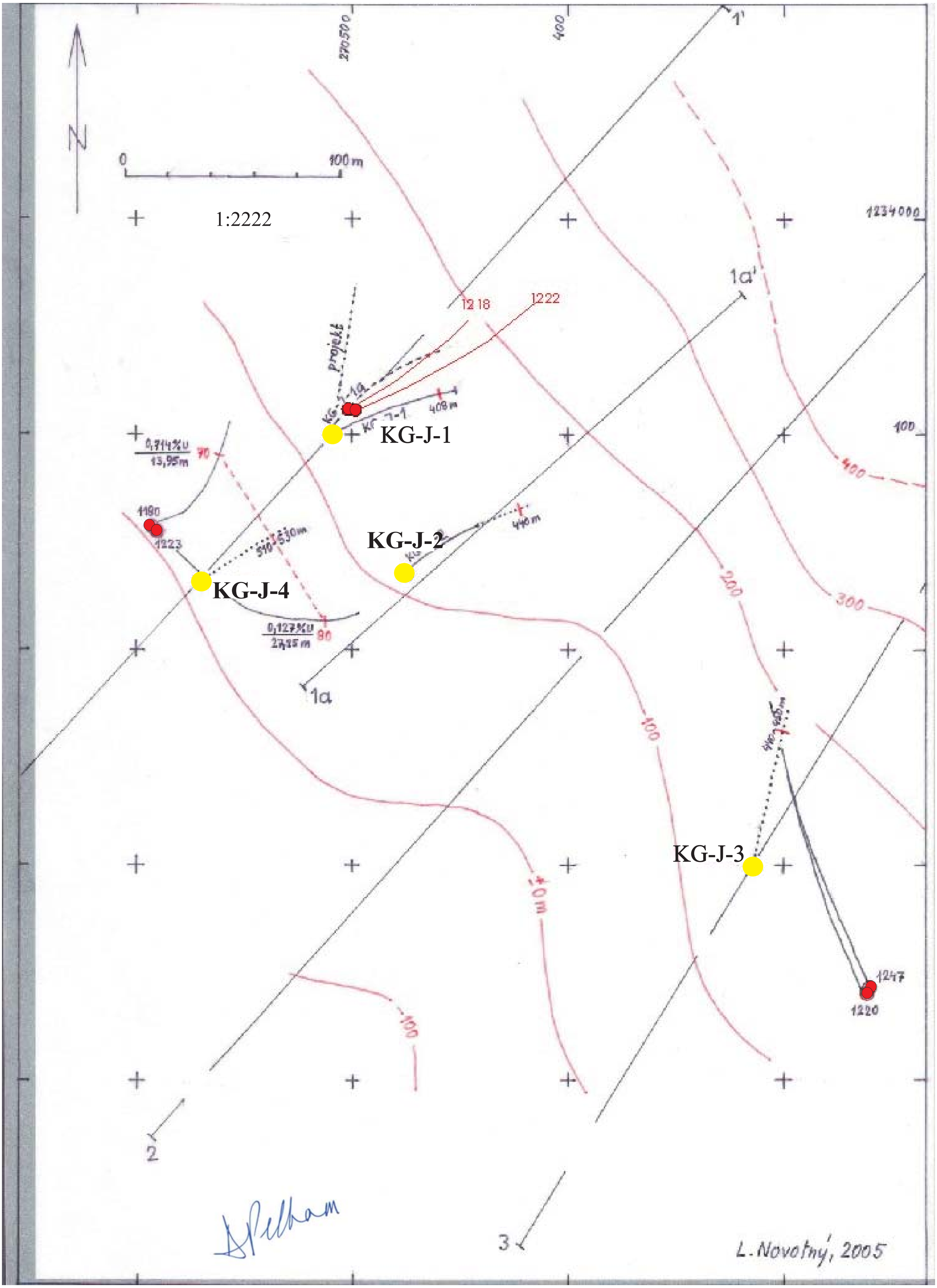
Poznámka: iné vysvetlivky sú na prílohe č. 1

Blok J-1-Z-3	S = 234 350 m^2
Variant II.	m = 2.19
	c = 0.472%
	V = 513 226 m^3
	Q = 1 395 975t
	K = 6 589t U

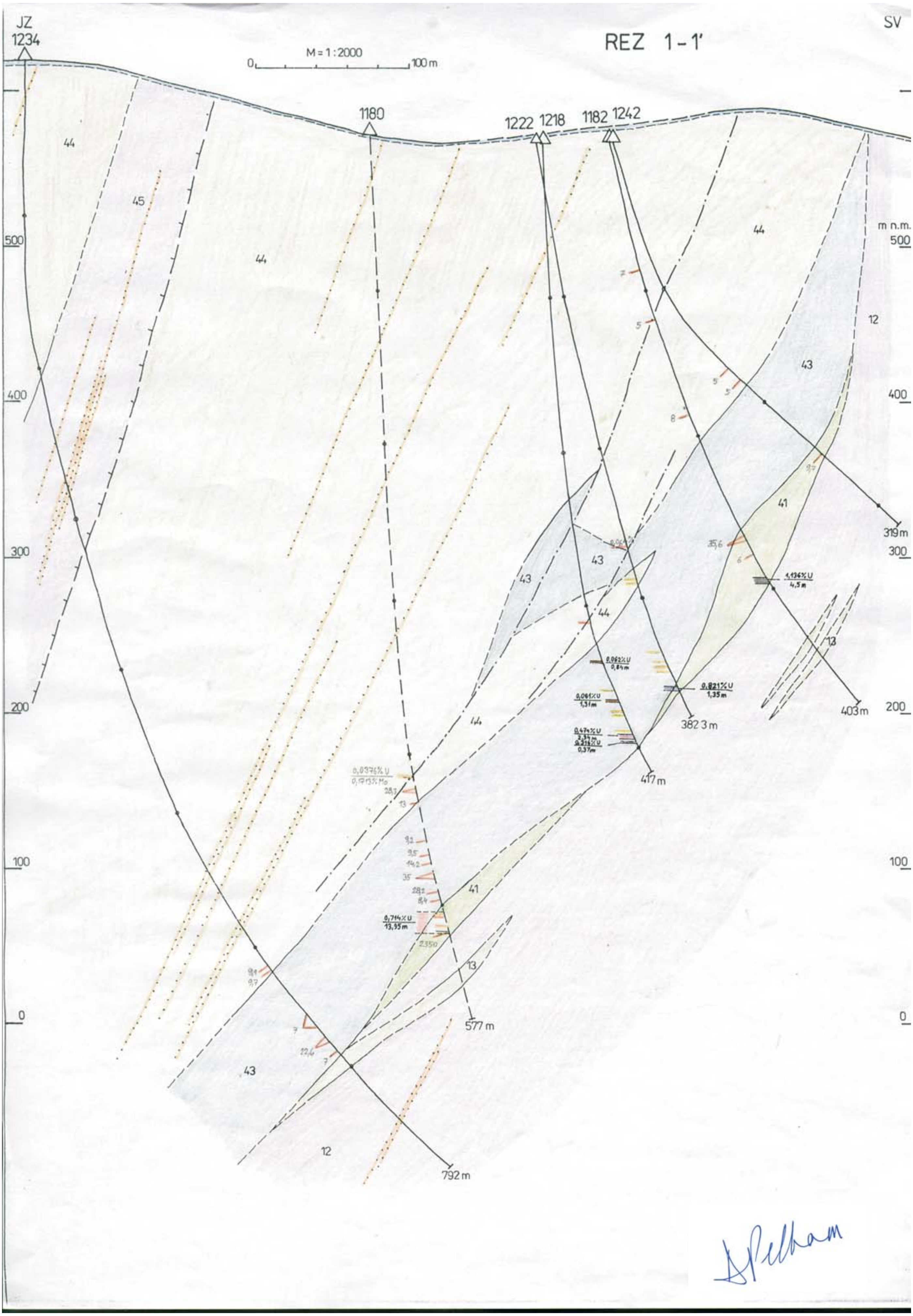
Príloha č. 3

MAP 12- JAHODNA RESERVES- RESERVE BLOCKS
- VARIANT II - scale 1:2,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited

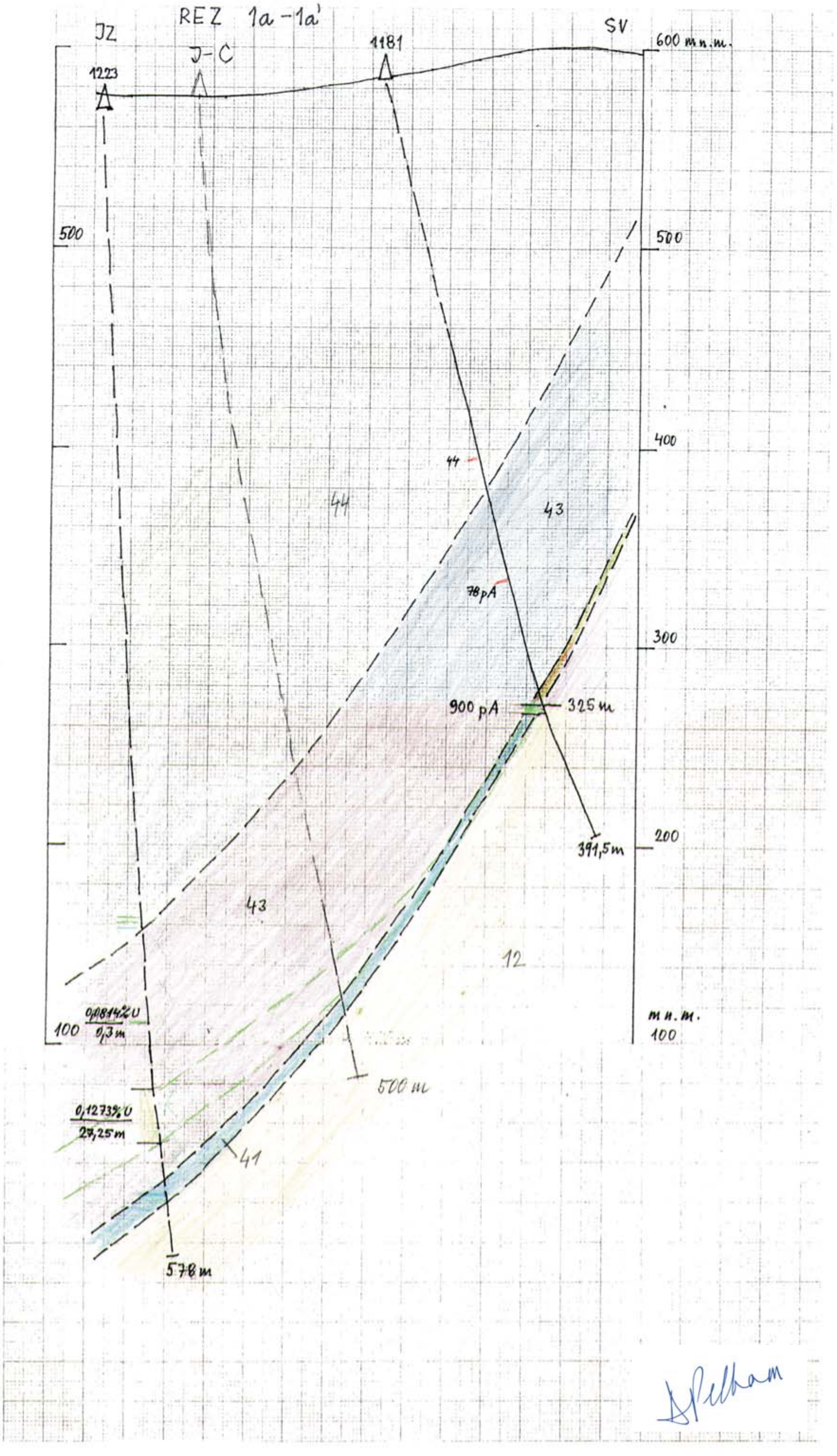


MAP 13- JAHODNA- TOURNIGAN 2005/06 PHASE 1 DRILLING
For Tournigan Gold Corp, Jahodna Uranium Project, Slovakia



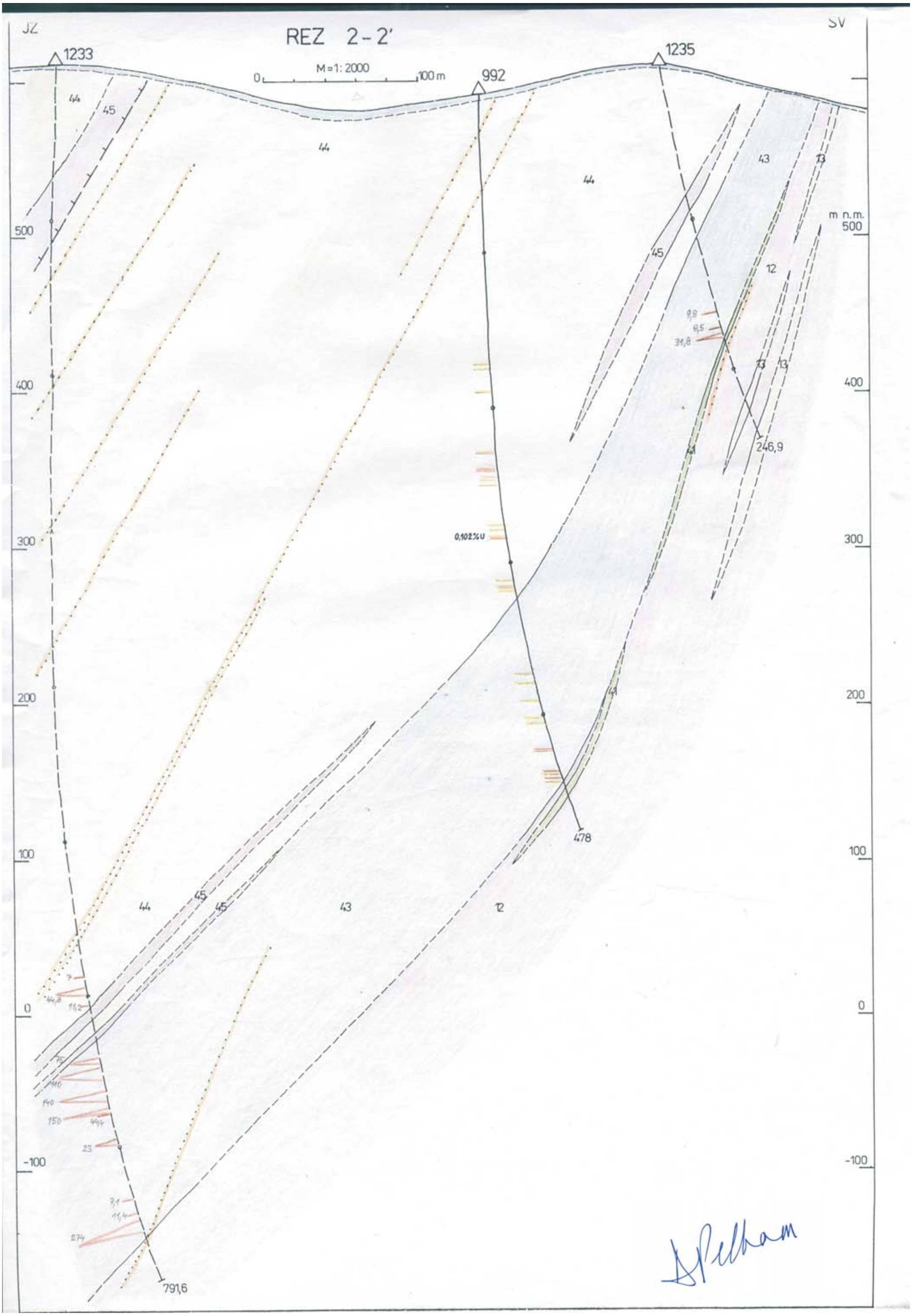
SECTION 1- JAHODNA- GEOLOGICAL SECTION 1-1'
- scale 1:2,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited



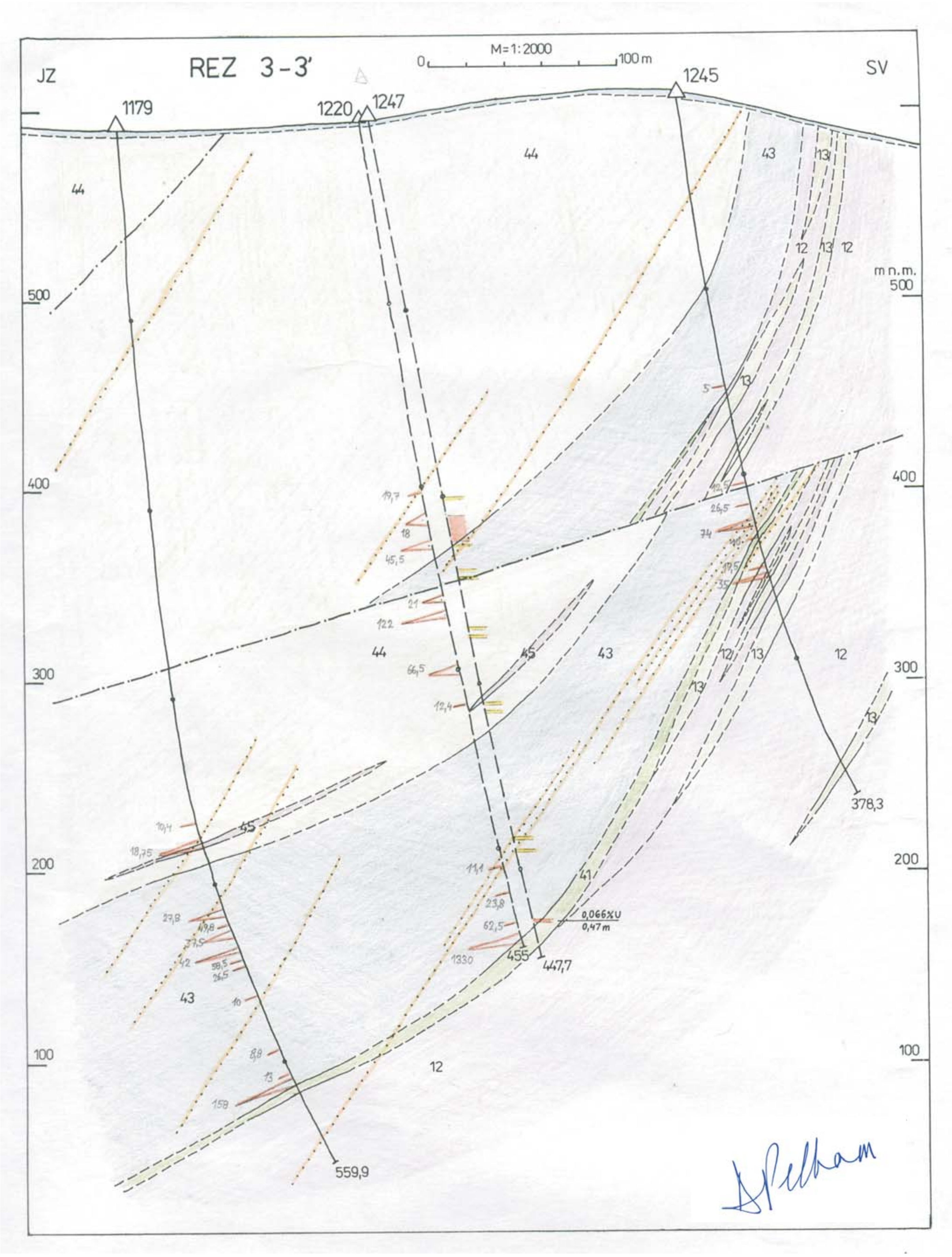
SECTION 2- JAHODNA GEOLOGICAL SECTION 1a-1a'
- scale 1:2,000
For Tourigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited



SECTION 3- JAHODNA- GEOLOGICAL SECTION 2-2'
- scale 1:2,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited



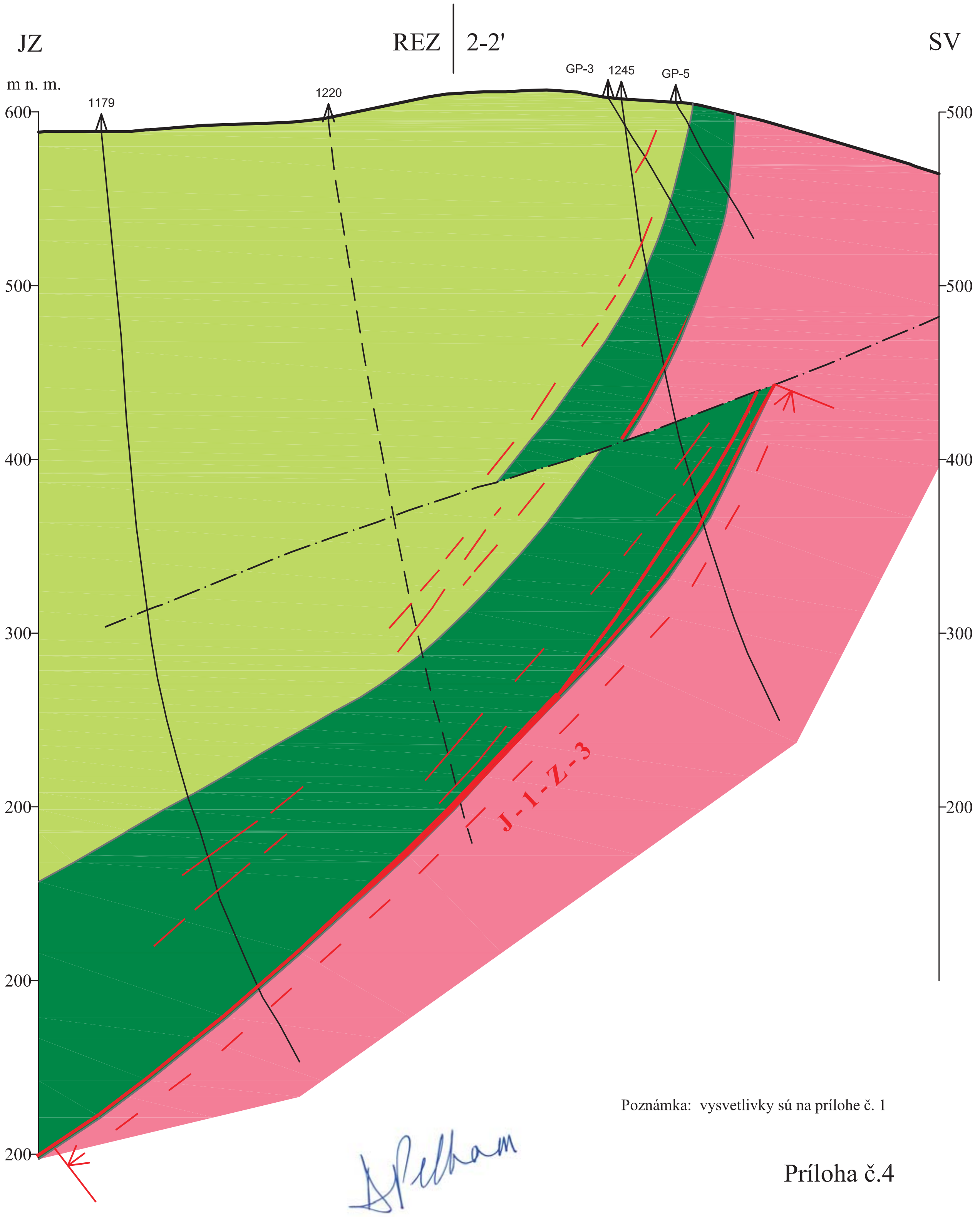
SECTION 4- JAHODNA- GEOLOGICAL SECTION 3-3'
- scale 1:2,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited

PRIEČNY GEOLOGICKÝ REZ 1 - 1'

M=1 : 2 000

Zostavil: F. Miháľ, 1991



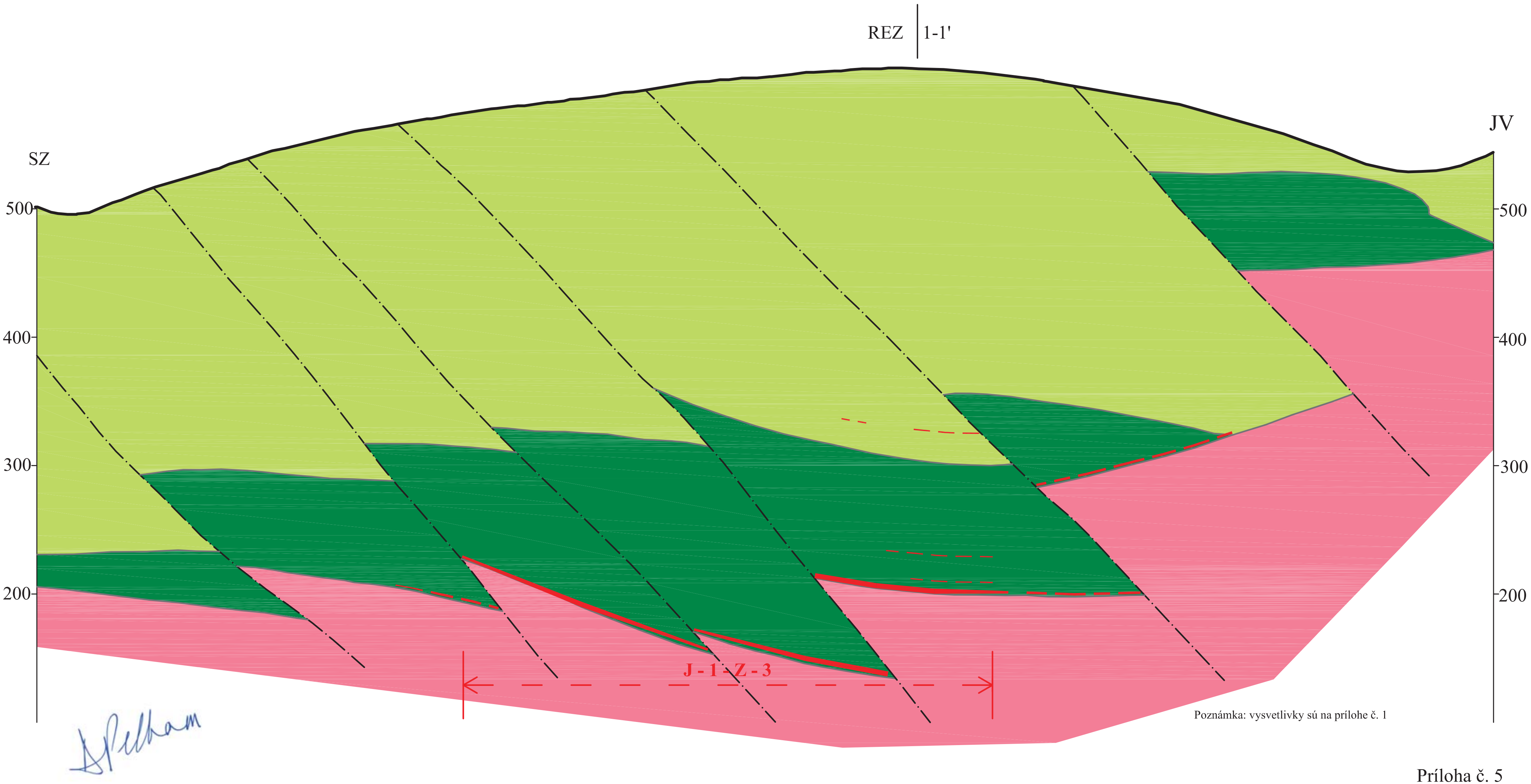
Poznámka: vysvetlivky sú na prílohe č. 1

Príloha č.4

SECTION 5- JAHODNA DEPOSIT- CROSS SECTION 1-1'
- scale 1:2,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia.

A C A Howe International Limited

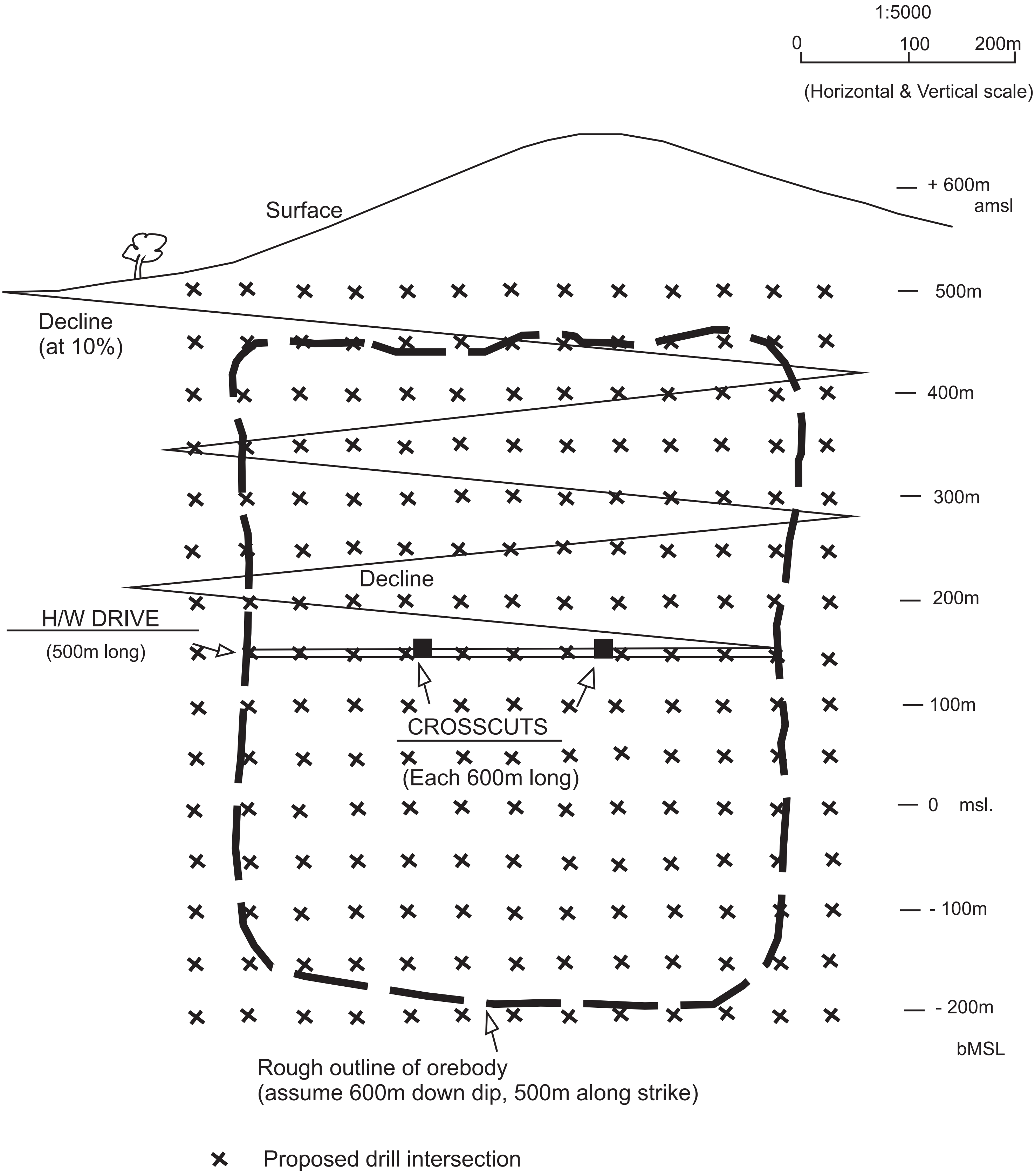
POZDĹŽNÝ GEOLOGICKÝ REZ 2 - 2'
M=1 : 2 000
Zostavil: F. Miháľ, 1991



Príloha č. 5

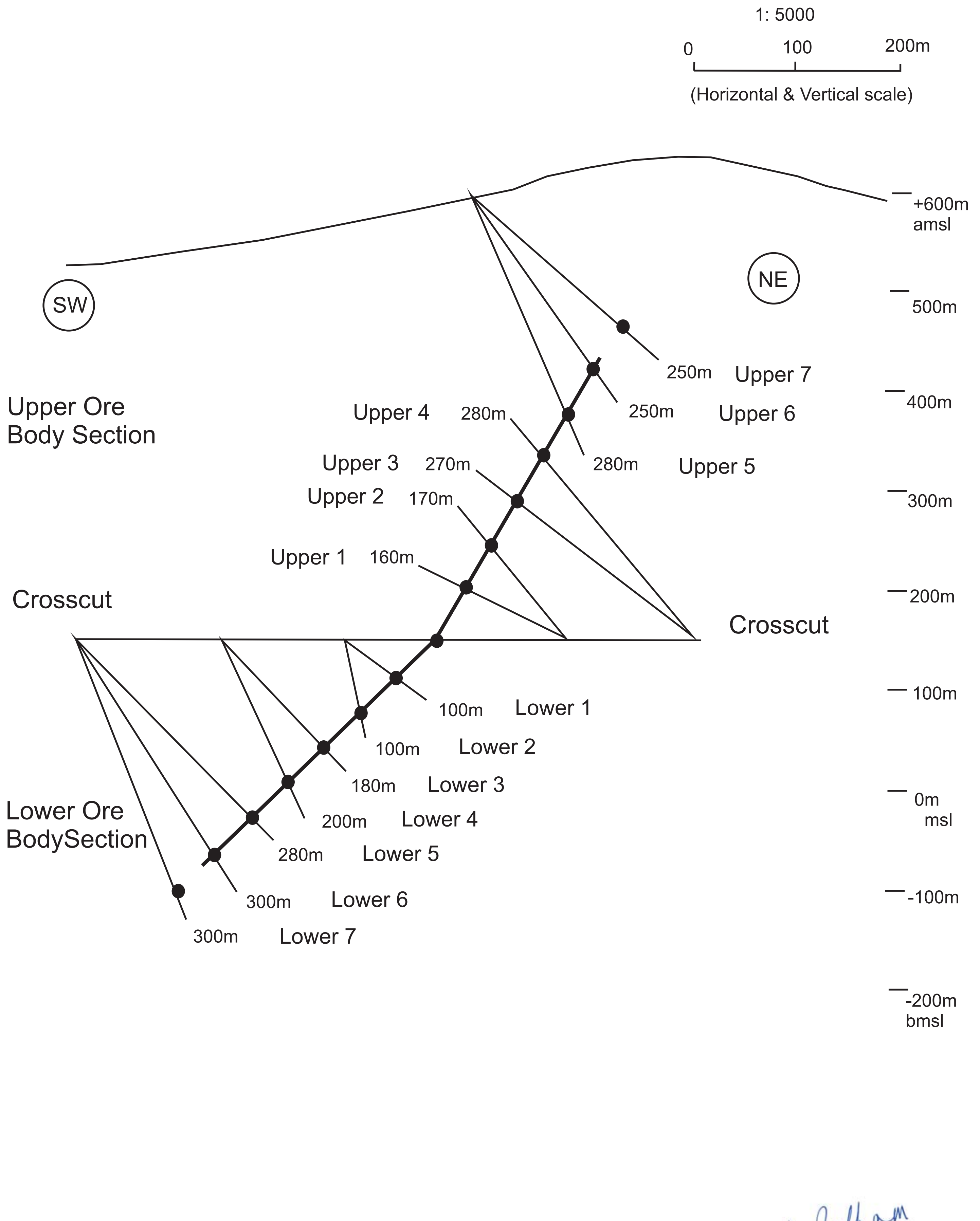
SECTION 6- JAHODNA DEPOSIT- CROSS SECTION 2-2'- scale 1:2,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia.

A C A Howe International Limited



SECTION 7 - SIMPLIFIED LONGITUDINAL SECTION
To show proposed decline position. 1:5000
For Tournigan Gold Corporation Jahodna Uranium Project, Slovakia

A C A Howe International Limited



SECTION 8 - SIMPLIFIED CROSS SECTION (SW - NE)
To show proposed underground drilling layout. 1:5000
For Tournigan Gold Corporation Jahodna Uranium Project, Slovakia

A C A Howe International Limited

APPENDIX 2.

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Plate 1 - Jahodna Area - deciduous woodland

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Plate 3 - Jahodna - Drill Hole KGJ1 - drill and Uaz logging van

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Plate 10 - Jahodna - Drill Hole KGJ1- Tectonised reformed tuff (75m)

Plate 11 - Jahodna - Drill Hole KGJ1- Competent andesitic tuff, foliation at 30 deg to Core Axis (198m)

Plate 12 - Jahodna - Drill Hole KGJ1- Pink Quartz Dolomite Vein (285m)

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Plate 13 - Jahodna - Drill Hole KGJ1- hematitic zone with gamma kick (312m)

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Plate 18 - Jahodna - F. Pomorsky checking radioactivity in Knolske Formation sediments

APPENDIX 3.

MICROMINE STUDY OF THE JAHODNA URANIUM RESOURCE, SLOVAK REPUBLIC

for
TOURNIGAN GOLD CORPORATION

by
ACA HOWE INTERNATIONAL LTD

March 28, 2006

Berkhamsted
Herts, UK

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EXECUTIVE SUMMARY

A polygonal wireframe estimate was calculated for the Jahodna Resource (December 2005) using data from thirteen (13) historical drillholes over the project area. The results are contained in Table 1 and are given for a cut-off of 0.030%U.

Cut-off grade	Tonnage	Grade (%U)	SG	U (lbs)
0.030%U	1,100,363	0.55	2.72	13,344,645

TABLE 1. GLOBAL POLYGONAL WIREFRAME ESTIMATE (DEC05)

Following re-evaluation of the Jahodna deposit and the drilling of three (3) new drill holes at Jahodna (KG-J-1, KG-J-1a and KG-J-2) an updated polygonal wireframe estimate was calculated in February 2006 that included the additional drilling data. The updated resource estimate is contained in Table 2 below:

Cut-off grade	Tonnage	Grade (%U)	SG	U (lbs)
0.030%U	1,1256,088	0.56	2.72	15,510,172

TABLE 2. GLOBAL POLYGONAL WIREFRAME ESTIMATE (FEB06)

The February resource estimate predicts a 14% increase in tonnes, 2% increase in grade and an overall 16% increased in contained U pounds (lbs) compared with the December 2005 estimate. Given the narrow and often high-grade nature of the deposit, a cut-off of 0.030%U is sensible and should be used in future estimations and refined as new drilling is undertaken, if necessary.

1. INTRODUCTION

Following acquisition of historical drill hole data over the Jahodna uranium project, drill hole data was imported in to Micromine software for the purposes of producing a Polygonal Wireframe Resource Estimate (PWRE) for comparison with historical resource estimates reported for the project. This resource calculation was completed in December 2005. In addition, following the drilling of three new holes at the Jahodna Project, the PWRE was calculated again (February 2006) to include the new drill hole data.

2. DATA USED IN MODELLING

Limited historical raw data was presented for the study and consisted of collar, survey and assay data for thirteen (13) drillholes. This historical data was assumed to be correct and no checks were made to confirm its validity. Assay data consisted of two tables, the first contained downhole mineralised intervals at a 0.015% U lower cut-off and the second, mineralised intervals at a 0.030% U lower cut-off , the latter being used for this resource estimate. Following the drilling of new holes at Jahodna (KG-J-1, KG-J-1a and KG-J-2), geochemical assay data for the new holes was added to the assay tables in Micromine (shown as shaded in the tables 3-4 below).

Table 4 also contains true thickness values for each mineralised interval A review of drill hole traces through the mineralised envelope indicates that drilling is largely sub-perpendicular to the mineralised zone and true thickness values are, on average 15% less than drill hole interval thicknesses.

Full down hole survey data, geophysical data and geochemical assay data was received for the new drilling and reviewed prior to incorporating the data into the new model. Downhole survey data was directly imported in to Micromine from excel spreadsheet data provided by Tournigan. All drill hole positions are shown on the accompanied Jahodna Drillhole Plan.

TABLE 3. COLLAR DATA

Hole ID	Depth (m)	Dip	Azimuth	Northing	Easting	RL
VRT_992	470.00	-89.3	25	-1234199.06	-270390.03	590.50
VRT_1179	558.60	-85.5	25	-1234432.75	-270395.30	589.97
VRT_1180	573.00	-90.0	29	-1234142.74	-270593.21	571.38
VRT_1181	390.20	-79.0	75	-1234113.37	-270426.76	576.91
VRT_1182	403.00	-76.6	53	-1234049.28	-270463.45	568.07
VRT_1215	448.00	-86.5	45	-1234114.44	-270430.49	576.51
VRT_1218	405.00	-90.0	68	-1234081.17	-270494.99	566.28
VRT_1220	452.00	-75.0	336	-1234360.28	-270263.07	594.20
VRT_1222	381.00	-78.9	50	-1234084.32	-270496.03	566.95
VRT_1223	580.00	-90.0	165	-1234144.29	-270590.99	571.57
VRT_1233	780.00	-90.0	0	-1234404.37	-270573.31	610.87
VRT_1247	439.00	-74.1	338	-1234356.92	-270260.29	594.79
VRT_1248	412.50	-85.9	296	-1234114.82	-270429.25	576.77
KG-J-1	439.00	-85.0	40	-1234093.73	-270513.97	565.57
KG-J-1a	442.00	-88.5	5	-1234092.02	-270512.46	565.67
KG-J-2	480.00	-88.1	40	-1234165.13	-270473.30	575.41

Hole ID	From (m)	To (m)	Interval (m)	True Thickness (m)	U%
VRT_992	440.03	440.70	0.67	0.62	0.055
VRT_1179	512.01	513.00	0.99	0.97	0.116
VRT_1180	510.51	518.50	7.99	7.55	0.494
VRT_1181	318.39	322.00	3.61	3.49	0.164
VRT_1182	301.23	305.30	4.07	4.01	1.092
VRT_1215	402.58	404.20	1.62	1.48	1.111
VRT_1218	395.21	398.10	2.89	2.80	0.413
VRT_1220	404.62	408.10	3.48	3.23	0.475
VRT_1222	361.85	363.20	1.35	1.31	0.821
VRT_1233	708.35	709.20	0.85	0.82	0.175
VRT_1247	429.22	429.60	0.38	0.36	0.075
VRT_1248	400.43	401.50	1.07	1.00	0.676
KG-J-1	406.90	408.10	1.20	0.99	0.387
KG-J-1a	424.00	424.90	0.90	0.77	8.829
KG-J-2	450.90	452.00	1.10	0.93	1.140

TABLE 4 ASSAY DATA

3. DATA VALIDATION

Data files were created in Micromine for collar, assay and survey data and then a validation function performed to check for data errors. No errors were detected.

4. WIREFRAMING AND INTERPRETATION

Drillholes and assay data were plotted on screen and checked against historical plans and hand drawn cross sections to confirm hole paths and the position of mineralised intervals downhole.

A series of cross sections were then produced (1-1’, 1a-1a’, 2-2’ and 3-3’) for variant 1 and 2 replicating historical hand drawn sections (roughly NE-SW). In addition an extra cross section was produced termed 1b-1b’ and isn’t included in historical sections. Mineralised intervals on each section were then joined up, honouring the geometry of the mineralised zone as interpreted in historical cross sections. Cross sections with the interpreted mineralisation outline at a 0.030% U lower cut-off are contained in Sections 1-5 below along with a drill hole plan (Plan 1) showing the recent drilling (February 2006)

The mineralised zones were found to be striking NW-SE and shallowly dipping to the SW with the dip becoming steeper to the NE. This geometry mirrors that of historical interpretations.

String sections were then produced for each cross section and mineralised intervals extended to the NE and SW typically by half the distance to the next drillhole. Mineralised zones were also extended to the NW and SE by 60m (the average section spacing over the project) and closed off. These strings were then joined up to produce 3-D mineralised wireframe envelopes for the February 2006 model, which were then validated and a volumes reported for all.

Figure 1 shows the mineralised wireframe and drill hole positions. Figure 2 shows the mineralised wireframe looking NW and clearly shows the steep to shallowly dip to the SW, of the mineralised envelope.

5. POLYGONAL WIREFRAME ESTIMATION

A polygonal wireframe estimate was then performed on the 3-D wireframe and a volume, tonnage and grade reported.

The Polygonal Estimation method was chosen as data was limited, and as a simple method of determining a resource figure that could be compared to historical calculations and check their validity.

Polygonal estimation is a linear estimation technique, based on the linear weighted average of all drillhole assay intervals, applied to a given volume (in this case the wireframe).

Parameters used in calculating the Polygonal Wireframe Estimate were as follows;

Wireframe (Dec05 model):		variant2_alljan06
Wireframe (Feb06 model):		variant2_allfeb06mod
Imput files:	Assay:	jahodna_lowcutcomp, jahodna_highcutcomp
	Survey:	jahodna_survey
	Collar:	jahodna_collar
S.G		2.7

6. COMMENT

The February 2006 model predicts a 14% increase in tonnes, 2% increase in grade and an overall 16% increased in contained U pounds (lbs) compared to the December 2005 model.

Validation of the February model and associated cross sections and 3-d interpretation was undertaken upon completion of the model to check the validity of the model and also to explain the variability that exists between both models. Cross sections for Variant 1 are attached here (section 1-1', 1a-1a', 1b-1b', 2-2' and 3-3')

The February 2006 model using a 0.030% lower cut-off shows an increase in tonnage, grade and contained U Lbs compared to the December 2005 model. The increase in tonnes is due to mineralised zone thickness increases on sections 1a-1a' and 1b-1b' by the addition of holes KG-J-1 and KG-J-2. The grade and therefore contained U has increased largely because the grades of hole KG-J-1a and KG-J-2 are the two highest of all the holes and have therefore significantly upgraded the resource.

Given the narrow and often high-grade nature of the deposit, the cut-off of 0.030% U should be used in future estimations and refined as new drilling is undertaken, if necessary.

The Polygonal Estimation method of interpolation introduces conditional bias, i.e. the high grades are overstated, and the low grades are understated, and assumes extraordinary grade continuity between sample points. This means that the resource estimate may overstate the grade (i.e. the higher grade values will have more influence on the overall grade than lower values). When considering the dimensions of the deposit (500m × 500m × 2.5m) which is only informed by 16 grade points, the majority clustered in the NW of the deposit, then high grades will have a large influence on the overall grade.

It was decided to run a polygonal model because of the lack of data with which to calculate a resource (only 16 grade intervals) but with added drill data, a more refined interpolation method (IDW or Kriging) may be undertaken and will provide a more refined grade-tonnage scenario. The polygonal estimates have succeeded in confirming the likely tonnage of the deposit (which compares favourably to historic calculations) but the grade of the deposit may be overstated.

Though simplistic, this method is adequate for the purposes of comparison with historical calculations and for use at the current stage of advancement of the Jahodna deposit.

REVISED MINERAL RESOURCE ESTIMATE

KREMNICA GOLD PROJECT

KREMNICA,
SLOVAK REPUBLIC,
EUROPE

Prepared for

Tournigan Gold Corporation
Suite 301-700 West Pender Street
Vancouver, B.C.
V6C 1G8

Prepared by

Mr. Carl von Einsiedel, P. Geo.
Mr. Garth D. Kirkham, P.Geoph.
Mr. W. Peter Stokes, P.Eng.

Beacon Hill Consultants (1988) Ltd.

May 11, 2006

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SECTION 1.0

SUMMARY

1.1 Purpose of Report

Beacon Hill Consultants (1988) Ltd. has been commissioned by Tournigan Gold Corporation to complete a prefeasibility study on the Sturec deposit, Kremnica Gold Project. As part of this study a revised mineral resource estimate has been completed. This report is the subject of the revised mineral resource estimate. The prefeasibility study is continuing and is scheduled to be completed by the end of the second quarter 2006.

1.2 Results

The results of this revised estimate of the mineral resources for the Sturec deposit, Kremnica Gold project are shown below.

Table 1
Summary Resources

Kremnica Gold Project
Sturec Deposit
Resource Estimate
8-Apr-06

Category	Cut-off gAu/t	Quantity Tonnes	Grade Au Equ.	Grade gAu/t	Grade gAg/t	Gold Equi. Ounces	Gold Ounces	Silver Ounces
Measured	0.75	7,293,000	1.96	1.75	14.24	459,600	410,300	3,338,900
Indicated	0.75	11,514,000	1.66	1.48	11.86	614,500	547,900	4,390,400
Measured and Indicated	**0.75**	**18,807,000**	**1.78**	**1.58**	**12.78**	**1,074,100**	**958,200**	**7,729,300**
Inferred	0.75	6,398,000	1.44	1.32	7.42	296,200	271,500	1,526,300

Note Tonnes rounded to nearest 1000 tonnes, ozs to the nearest 100ozs

Gold equivalent calculated from silver gold ratio of 66.7:1

This resource is compliant with National Instrument 43-101 regulations.

1.3 Data Used for Estimate

The resource estimate includes the results of 41 infill reverse circulation drill holes completed by Tournigan in 2005 as well as 79 historical diamond drill holes, 3,148 historic underground samples and 21 historic underground drillholes. In addition, 9 bench channels were sampled which yielded a total of 317 individual samples. Drill hole spacing was reduced from approximately 100 meters by 50 meters to 50 meters by 50 meters during the 2005 infill drilling program at Sturec.

1.4 Estimate Method

The Sturec Deposit resource was estimated using MineSight™ software to produce a block model, with block sizes of 5 meters by 10 meters by 5 meters. Search ellipsoid dimensions were reduced from earlier resource estimates, providing an even more tightly constrained and robust estimate. The cut-off grade of 0.75 gram per tonne gold is similar to the effective open-pit grade cut-off from the 2004 Sturec preliminary assessment completed by Beacon Hill Consultants (1988) Ltd. Additional quality assurance/quality control work was completed, including the twinning of five historic drill holes, the re-assaying of historical sample pulps, detailed specific gravity measurements and extensive documentation and relogging of historical drill core.

1.5 Property Description

Kremnica Gold Property is located 190 km northeast of Vienna, approximately at the geographic centre of Europe. Tournigan owns 100% of the property through its Slovak subsidiary, Kremnica Gold a.s. The Kremnica deposits detailed in this report are located within the Kremnica Mining Lease (MHD-D.P. 12), dated January 21, 1961, which covers 11.79 km^2. Tournigan has also secured the exploration license Lutila, which lies to the south and covers the 86.38 km^2 of all known mineralized areas within the Kremnica district.

Kremnica is a historical gold mining district located in central Slovakia, where gold mineralization is part of a Tertiary-aged epithermal gold/silver system. The Kremnica deposits have been mined intermittently for at least 600 years, ending in 1992, and yielding 1.5 million ounces of gold and 9 million ounces of silver. The known gold-silver occurrences at Kremnica are Tertiary low sulphidation quartz vein stockwork deposits, hosted within a regionally extensive zone of hydrothermally altered andesite and rhyolite volcanics. The principal vein system has a strike length of at least 7 km, with individual vein groups up to 150 meters thickness. Veins are commonly banded and brecciated and contain numerous voids. Vein mineralogy is multiphase and comprised of quartz, calcite, adularia, sericite, illite, and chalcedony. The predominant sulfide is pyrite. Other sulfides include marcasite, galena, sphalerite, arsenopyrite, and chalcopyrite. Gold is finely disseminated and occurs freely, and associated with the sulfides.

Numerous targets have been identified in addition to the Šturec deposit, including the Vratislav and Wolf targets, respectively 1 and 2 km north along the continuation of the Kremnica vein structure and a large area of strongly clay and silica altered rhyolite, referred to as Kremnica South, located south of the deposit which is believed to be prospective for several styles of epithermal gold mineralization. Previous work by Argosy Mining Corporation and consulting engineers Western Services Engineering Inc, and the Slovak State, defined small historical resources at both Vratislav and Wolf that are not NI 43-101 compliant resource estimates.

SECTION 2.0

INTRODUCTION

2.1 Terms of Reference

Beacon Hill Consultants (1988) Ltd. (Beacon Hill) was commissioned by Tournigan Gold Corporation to prepare a revised mineral resource estimate on the Kremnica Gold property in Central Republic of Slovakia. The main authors, Mr. Carl von Einsiedel, a Registered Professional Geoscientist in the Province of British Columbia and independent consulting geologist, and Mr. Garth D. Kirkham, P.Geoph., Kirkham Geosystems Ltd., are both qualified persons by Canadian Securities Administrators definitions of National Instrument 43-101 (NI 43-101). Mr. W. Peter Stokes, P.Eng., has directed the work and assisted with compilation of the report. Mr. Stokes is also a qualified person by Canadian Securities Administrators definitions of NI 43-101.

In general, Mr. von Einsiedel was responsible for preparing all aspects of the technical report, excluding the calculation of the mineral resource. Mr. Kirkham was responsible for the calculation of the mineral resource. Mr. von Einsiedel spent 6 days at the Kremnica site during July 2005 working in the field for 4 days observing the completion of two of the infill drill holes, observing the sample preparation procedures at the on site prep lab operated by Tournigan and reviewing the diamond drill core, sample storage and sample preparation facilities, as well as reviewing all maps and records stored at the Kremnica offices. Mr. Kirkham and Mr. Stokes visited the property October 24 to 28, 2005 for a period of 5 days, reviewed the data at the site, held discussions with site personnel and examined the site.

The purpose of this report is for Tournigan Gold Corporation to re-estimate the mineral resources for the Sturec deposit at the Kremnica Gold Project based upon additional drilling and data gathering that occurred during 2005. This report is written according to standards set by the National Instrument 43-101, effective 12/31/2005.

2.2 Introduction

The Kremnica Gold property includes the Kremnica Mining Lease (MHD-D.P. 12), the Lucky exploration license, and the Lutila exploration license application. The Lucky and Lutila licenses are now merged into a single Lutila license.

There is a large amount of data available on the project with records of mining activity at Kremnica going back over 600 years to the present day. The following information sources were relied upon by the authors to complete this report:

1. Extensive discussions were held with the management of Tournigan Gold Corporation, who detailed their current understanding of the project as well as their plans for the future.

2. Verbal communications with Ing. Oliver Finca, who acted as guide for Mr. Smith during his field excursions. Mr. Finca, a Slovak Mining Engineer, has worked at the Kremnica site for over 30 years

3. Verbal communications from Ing. Boris Bartalsky, PhD. who coordinated and assisted Mr. von Einsiedel in all his activities at the Kremnica site. Mr. Bartalsky, a Slovak Geologist, has worked at the Kremnica site for over 10 years.

4. ARGOSY MINING CORP., 1998: Kremnica Gold Project – Project Summary.

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9. MATO, L., 1997: Mineralogy and Petrology Evaluation of the Šturec Deposit. Argosy Slovakia s.r.o. Report.

10. MATO, L., FINKA, O., BARTALSKY, B., 1999: Kordíky Formation in the Central and Southern Part of the Šturec Area (Kremnica deposit) – Basement and Barrier of Auriferous Vein Structures of the 1st Vein System, Facts and Indications. Mineralica Slovaca, 31 (1999), 275 - 282.

11. WESTERN SERVICES ENGINEERING INC., 1998: Report prepared for Argosy Mining Corporation. In-situ Geological Resource Update, Exploration Results as of January 1, 1998 – Kremnica Gold and Silver Project Slovakia.

12. 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.

13. SMITH AND KIRKHAM, 2004, Independent Technical Report, Kremnica Gold and Silver Project.

14. BEACON HILL CONSULTING (1988) LTD, 2004, Preliminary Assessment, Kremnica Gold Project, Sturec Deposit

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.

As part of the ongoing pre-feasibility study metallurgical test work is underway. The results of this test work will be fully reported within the pre-feasibility report which is scheduled to be finalized before the end of the second quarter, 2006.

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 1. The town of Kremnica is located 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 nar Hronom. The trip takes about four hours. Kremnica has a population of about 7,000 and is accessible by train from Bratislava, the capital and largest city in Slovakia.

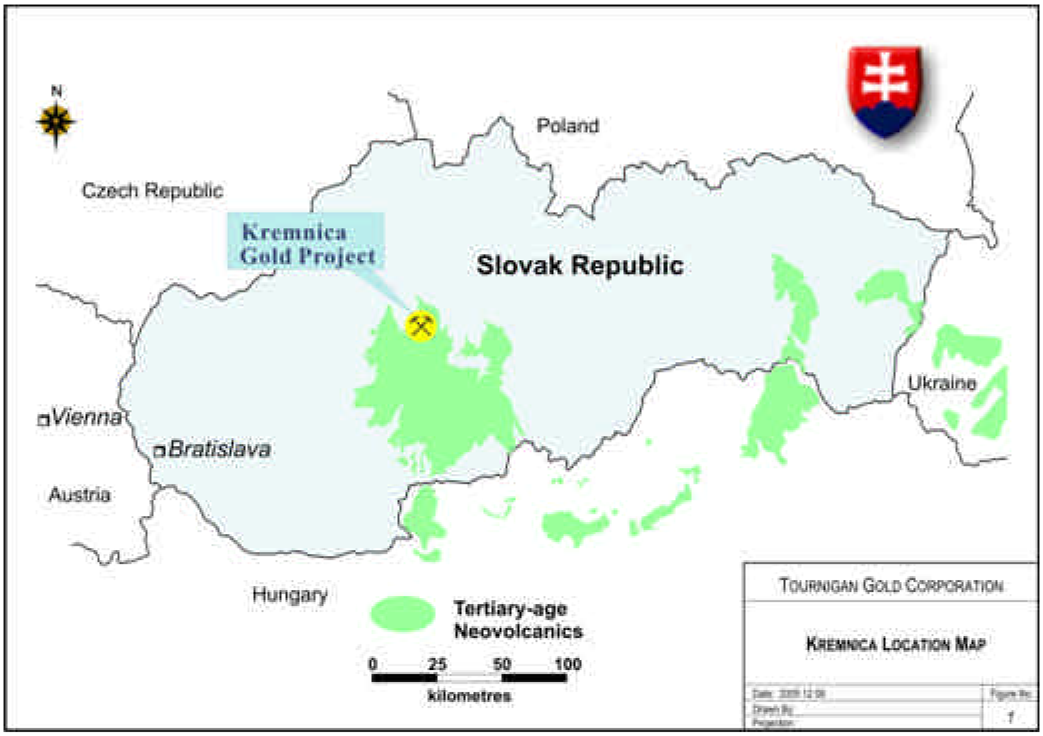

Figure 1: Kremnica Location Map

The Kremnica project comprises two contiguous properties; the Kremnica Mining License and the Lutila exploration license in application. The properties are depicted in Figure 2, and boundary co-ordinates and areas are listed in Tables 2 and 3 below:

Table 2
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

Kremnica Mining License total area: **11.79 km^2** (Slovak coordinates system – JTSK)

Table 3
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

Lutila license total area: **86,38 km^2** (Slovak coordinates system – JTSK)

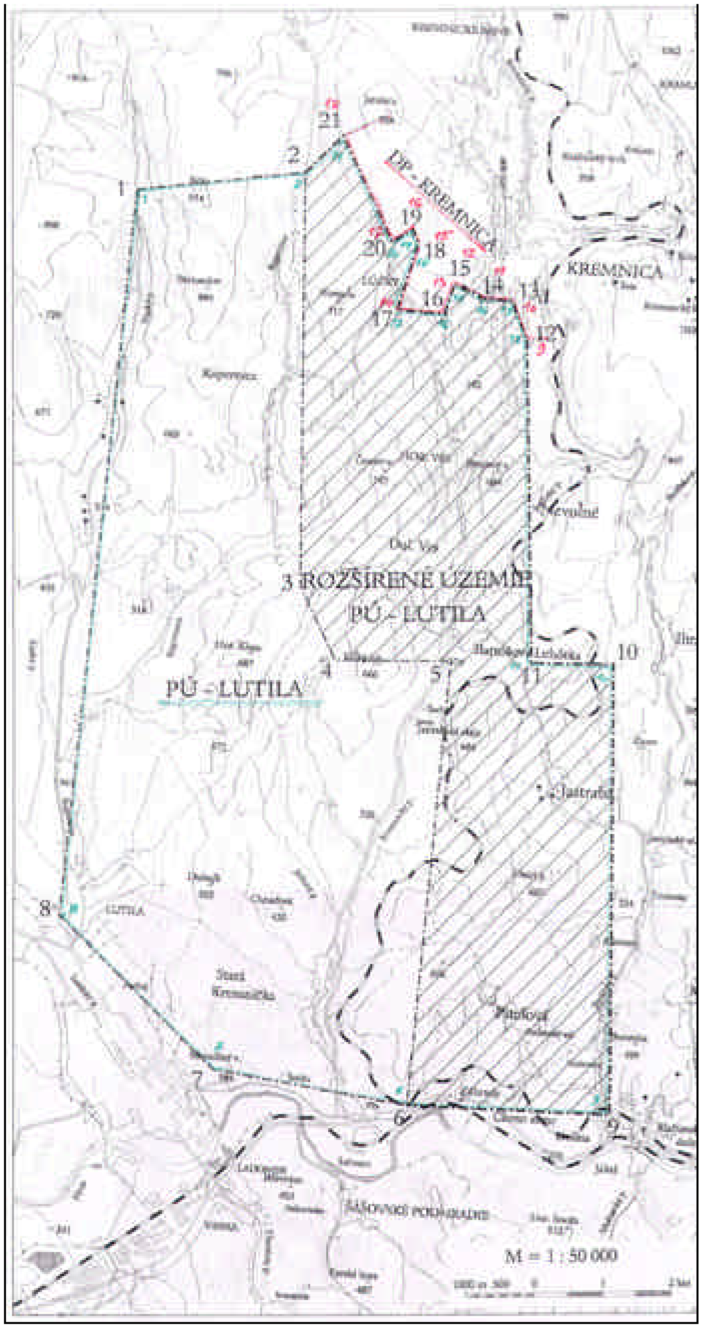

Figure 2: Kremnica Project Licenses

The Kremnica Mining License (MHD-D.P. 12) was issued January 21, 1961, and remains valid in perpetuity. On March 6, 2006 the license was transferred from Kremnica Gold a.s., which is 100% owned by Tournigan Gold Corporation. Currently the license is held by Ludovika Holding, s.r.o. which is also 100% owned by Tournigan Gold Corporation. Within the mining license the Mining Bureau must permit mining projects. The current permit allows only for protection and preservation of the old mine workings and is valid until December, 2010 at which time a new permit for work must be obtained. The mining license fee is currently 60,000 Sk per year (approximately CND $2222.00 per year). Slovak law states that some work should be done within the license, or the license can be withdrawn. The law is not specific on when, what, or how work must be done. All known mineral resources are contained within the mining license as well as areas for future operations, processing waste and tailings.

The Lucky exploration license was registered June 05, 2002, and became effective on 23 September 2002. The license was closed. The license was held by Kremnica Gold a.s.

The original Lutila exploration license was registered on 1 October 2003. The granting process finished on March 05, 2004. The license was registered by Kremnica Gold a.s.. Lutila license was extended and covered the Lucky exploration license on August 19, 2004. The license fees for Lutila is now 261,000 Sk/year (approximately CND $9700.00 per year) upon granting.

As shown in Figure 2, the two licenses are contiguous, and together cover all known prospective and mineralized areas within the Kremnica district. The mining license and exploration license have been legally surveyed. There are currently no other near-by metal claims or other companies working within the district. Within the exploration licenses there are several registered clay deposits. Currently two small clay pits are operating. These clay licenses are not expected to have any material effect on the subject properties.

The Kremnica project is not known to be subject to any royalties, back-in rights, payments, or other agreements and encumbrances, excepting a payment to the Slovak State. 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 between a mine owner and the State. In the case of Kremnica, it is expected that the payment would be close to 5% of net profit.

4.1 Surface Rights

The Kremnica Mining License covers 11.79 km^2, and includes within its boundaries the township of Kremnica (population approximately 7000). There are several hundred different landowners within the boarders of the mining license. 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 viewed maps of land ownership under the mining license and found virtually all of the land expected to be part of a future mining project in the possession of the State Land Fund, or State Department of Forestry. Negotiation with the State for rights to mine on these lands is expected to be straightforward. There is the probability that any future mining operation would likely consider purchase of 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 proceed. Exploration work up through to the definition of a resource is then carried out under the agreement. A new agreement must be reached before mining can begin. All permitting with the government officials, such as the Forest Service and Bureau of Mines, is outlined in the agreement and in straightforward regulations.

It must be noted that, at the present, there is opposition to the project by NGO's (non-governmental organizations) and a number of residents of the town of Kremnica, to the development of a mine in the area. Tournigan Gold Corporation are presently addressing these concerns through public consultation and through the dissemination of information to the resident of Kremnica on the proposed mine development. Tournigan Gold Corp. appears to be committed to acquiring a social license to operate and insuring 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. At the end of the active drilling by Argosy, 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 limiting Kremnica Gold new owners, Tournigan Gold, from any liability for past exploration activities.

Kremnica Gold has, with regard to mining activities, no environmental liabilities for past mining activities. 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 as a result of the past mining activities of the State Company, Rudne Bane, would fall to the State. The old mine works are currently in a stable condition and on inspection the author did not identify any areas that could obviously create any future environmental problems. Kremnica Gold would only assume liability for any environmental problems related to the future operation of a mine.

SECTION 5.0

ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Kremnica project is situated in the Rudnice valley between elevations of 958 and 619 m.s.l. The valley is hilly, but not steep. Mining has been conducted in the Kremnica area for around 1000 years and so the landscape reflects this activity, with old mine workings, mounds, excavations, pits and adits on most parts of the mining license. The majority of the target areas is meadows and mixed regenerating coniferous and broadleaved forest.

The town of Kremnica is situated within the mining license, and there are some scattered residential areas within the Kremnica licenses, but this is not expected to materially affect the future development of a mine within the area. The core areas of the mining license are designated for mining activity as part of the 10-year plan for the city of Kremnica.

Good 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 insure 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^{o}C, with highs around 30^{o} C and lows around -30^{o} C. Precipitation averages around 920mm per year. Snow cover can be expected some 3 to 4 months per year.

Potential sites for waste rock disposal, tailings dam sites and potential processing plant sites have in the past been proposed by Tournigan Gold Corporation and their contracted mining engineers, Beacon Hill, in the report titled "Preliminary Assessment, Kremnica Gold Project, Šturec Deposit, February 2004". These sites together with other sites are presently being evaluated as part of the prefeasibility report. The authors of the current report, has viewed these proposed sites and generally found the space available and no obvious reason why these sites would not suit the purposes mentioned.

With 1000 years of mining history, a well educated population, and high unemployment in the region and future mine can expect to benefit from a highly skilled, professional, and inexpensive labor 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 totaled 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 4km long and 2 km wide. The largest relic of past mining is a surface depression in the Šturec area, measuring 600m long, and up to 200m wide, and 100m deep. The district is pock marked with hundreds of pits, collapsed shafts and adits.

6.2 Modern Slovak Exploration

The Slovak Geological Survey and Rudne Bane (the state mining company) conducted modern exploration in the Kremnica district. The exploration work, which led to discoveries, was initiated in 1962 and conducted intermittently through to 1990. This 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 another kilometer to the north. In the early 1990's, based on the above work, the Czechoslovakian government calculated a resource totaling 7,570,000 tonnes at a gold grade of 1.53 g/t, or 370,000 ounces gold and 2,811,000 ounces silver. This resource is considered historical and is not NI 43-101 compliant. Investors are cautioned not to rely on this estimate.

6.3 1987 to 1992 Mining

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 with in a cyanide mill that operated at about 30 tonnes per day. The operations were not profitable and therefore ceased in 1992.

6.4 Argosy Mining Corporation

Kremnica Banska Spolocnost (KBS) an investment company comprising former mine managers obtained title to the Kremnica Mining Lease (MHD-D.P. 12) from the Slovak government on April 1, 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 October 6, 1995. Argosy Slovakia purchased KBS's share of the joint venture on April 24, 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 drilling program in 1997 for a total of 79 holes (12,306.7m).

Argosy commissioned Western Services Engineering Inc, of California, to produce a preliminary resource estimate for the Šturec South deposit at Kremnica. This resource estimate is considered historical and hence is not NI 43-101 compliant. Investors are cautioned not to rely on this estimate. The resource estimate calculated by Western Services Engineering in June 1997 was based on a block containing mineralized envelopes with 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 Gold Corporation 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 exploration activities north of Sturec at Wolf and Vratislav (test diamond drilling programs) and south of Sturec throughout the Kremnica South area (large soil geochemical survey covering most of Kremnica South area, test diamond drilling at Certov and Bartasova Lehotka areas at Kremnica South, and limited exploration trenching south of the town of Lucky). In 2005 Tournigan conducted an in-fill Reverse Circulation (RC) drilling program at Sturec providing data allowing re-calculation of the Sturec resource.

During 2004 Tournigan commissioned an independent technical report to establish a NI 43-101 compliant resource estimate for the Sturec deposit and commissioned Beacon Hill to complete a Preliminary Assessment based on the resource estimate.

In the summer and autumn of 2005 Tournigan executed a 36-hole program of reverse circulation drilling as infill of Argosy's and Tournigan's earlier core drilling programs. Tournigan drilled a further 5-holes as twins of earlier Argosy 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 by 50 meter centers). The RC program provided a strong correlation between core and RC holes: they were mutually supportive with respect to locations of ore-zones and their grades.

The holes and assay results are 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 also taken and inventoried as geological reference samples. Bulk rejects are stored on the property and are available for any further test work.

The results confirmed the important aspects of solid geology and ore-outlines as previously established by core drilling e.g. rock types and alteration, location of zones of oxidation, locations of ore-bearing veins and stockworks, their hanging walls, footwalls, and their thicknesses, strikes, dips, and grades.

For metallurgical testwork, Tournigan collected and submitted representative suites of different ore types and wallrock types from RC samples.

In 2005 Tournigan commissioned Beacon Hill Consultants (1988) Ltd. of Vancouver, British Columbia to conduct a pre-feasibility study, as the next development stage, to assess the economics of developing a mine at Kremnica. This report will utilize the revised mineral resource estimate as defined with this report as the basis for the resources utilized within the pre-feasibility study.

6.6 Previous Resource Estimates

Three resource estimates have been calculated in the past. The first is based on work prior to 1996 and was performed by the government of Czechoslovakia for the Šturec deposit and the combined Vladimir and Vratislav deposits. This work was based on using a 0.5 g/t cut-off grade and is outlined in Table 4.

Table 4
Historical Resource Estimates; Government of Czechoslovakia

Šturec	**Tonnes**	**Au g/t**	**Ag g/t**	**Au Grammes**	**Au Ounces**	**Ag Grammes**	**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

***Au Eq based on Au=77 Ag**

An additional geologic cross-sectional polygonal resource estimate was independently calculated which results 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 owners of the properties, performed an additional 12,306.7 meters of drilling, of which 9,382.4 meters in the Šturec deposit which significantly expanded the Šturec deposit, extending it at depth, to the south and to the north where it has been connected up to the Vladimir deposit. The resource calculated by Western Services Engineering, Inc. in their March 1998 study, which is the primary reference and source of data for this study, are in Table 5 and 6.

Table 5
1998 Resource Estimates; Indicated Resources, Western Services Engineering, Inc.

Cut –off Grade gAu/t	Total Tonnes	AuEq	Au	Ag	Grammes	Ounces
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	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	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	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	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	86,000	7.02	6.26	37.9	603,720	19,410

Note: AuEq is based upon 50:1 Au:Ag

Table 6
1998 Resource Estimates
All Resource Classes, Western Services Engineering, Inc.

Cut –off Grade gAu/t	Total Tonnes	AuEq	Au	Ag	Grammes	Ounces
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	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	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	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	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	546,000	7.39	6.66	36.6	4,034,940	129,724

Note: AuEq is based upon 50:1 Au:Ag

In 2004, Beacon Hill (1988) Consultants Ltd. performed a Preliminary Assessment of the Kremnica Property and as an integral part of that exercise a NI 43-101 compliant report was created authored by Smith and Kirkham, 2004. The differences in approach and methodology included that the main method for controlling the interpolation and for masking the model in this report was to base it on the geology as defined by the geologic sections, which have been deemed as an accurate representation of the deposit. In addition, there was no cut grade applied to the raw assay data or the composites prior to interpolation. This differs from previous methods that employed a 0.5 g/t gold equivalent mineralization envelope for interpolation along with applying a 0.5 g/t preprocessing cut-off to the composites which will tend to bias the grade higher and the tonnage lower by eliminating low grade or diluted zones.

In addition, the calculation utilized for gold equivalent in this report was based on US$385 gold and US$5.00 silver whereas the WSE report is based upon US$300 gold and US$5 silver.

Due primarily to the direction of continuity (i.e. search ellipse direction) the extent of mineralized blocks extends further to the north and south in addition to depth into the vein. In addition, in previous modeling exercises it appears that the quartz breccia unit had been masked out as waste whereas it has become evident that this unit does contain mineralization and therefore has been included. Table 7 and 8 list the resources for the Sturec Deposit.

Table 7
Resources for the Šturec Deposit, Smith and Kirkham, 2004.

Cut-off Grade gAu/t	Indicated Tonnes	AuEq	SG	Au	Ag	Grammes	Ounces
0.5	9,820,697	1.64	2.31	1.49	11.46	16,124,819	518,416
1	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	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	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	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	361,437	7.75	2.28	7.20	43.00	2,802,439	90,099

Note: AuEq is based upon 77:1 Au:Ag

Table 8
Resources for the Šturec Deposit, Smith and Kirkham, 2004

Cut-off Grade gAu/t	Inferred All Tonnes	AuEq	SG	Au	Ag	Grammes	Ounces
0.5	16,109,256	1.49	2.33	1.35	11.32	24,072,980	773,951
1	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	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	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	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	439,473	8.13	2.29	7.69	33.58	3,572,640	114,861

Note: AuEq is based upon 77:1 Au:Ag

Note; the above resources in Tables 7 and 8 are compatible with the requirements for resources as defined by the CIM and National Instrument 43-101. These resources are considered resources which would encompass indicated and inferred categories. They are relevant from a historical point of view and give an appreciation of the progression of the deposit in the context of modern exploration. The reliability of these resources is very good however a more recent estimate is calculated within this report and is described below.

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 occurs 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 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 2, 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 in the vicinity of 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 as a viable exploration target.

SECTION 8.0

DEPOSITS TYPES

As reported by Smith and Kirkham, 2004, gold-silver mineralization at Kremnica is part of a large low-sulfidation 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 extend 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 sulfides and with silver as electrum. Besides electrum, silver occurs in the minerals polybasite, pyrargyrite, and argentite. Sulfide minerals consist predominately of pyrite and marcasite with much lesser amounts of chalcopyrite, arsenopyrite, stibnite, sphalerite and galena. Sulfide 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 approximately north to northeast striking structural zone that 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 implying normal and/or reverse faulting, but earlier pre-mineralization movement within the structural zone may have been strike-slip. 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 Deposit

The Šturec deposit, depicted in Figure 3, occurs in the southern part of the central First Vein System. The Šturec deposit is continuously mineralized for 1200 m along strike, is typically 100 to 150m wide and extends to a known depth of at least 300 m. The deposit is open to extension both at depth and 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 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 cross cutting 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 back-fill.

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-sulfide bearing silica. They occur predominately adjacent to the veins but breccias can merge into veins vertically and/or along strike.

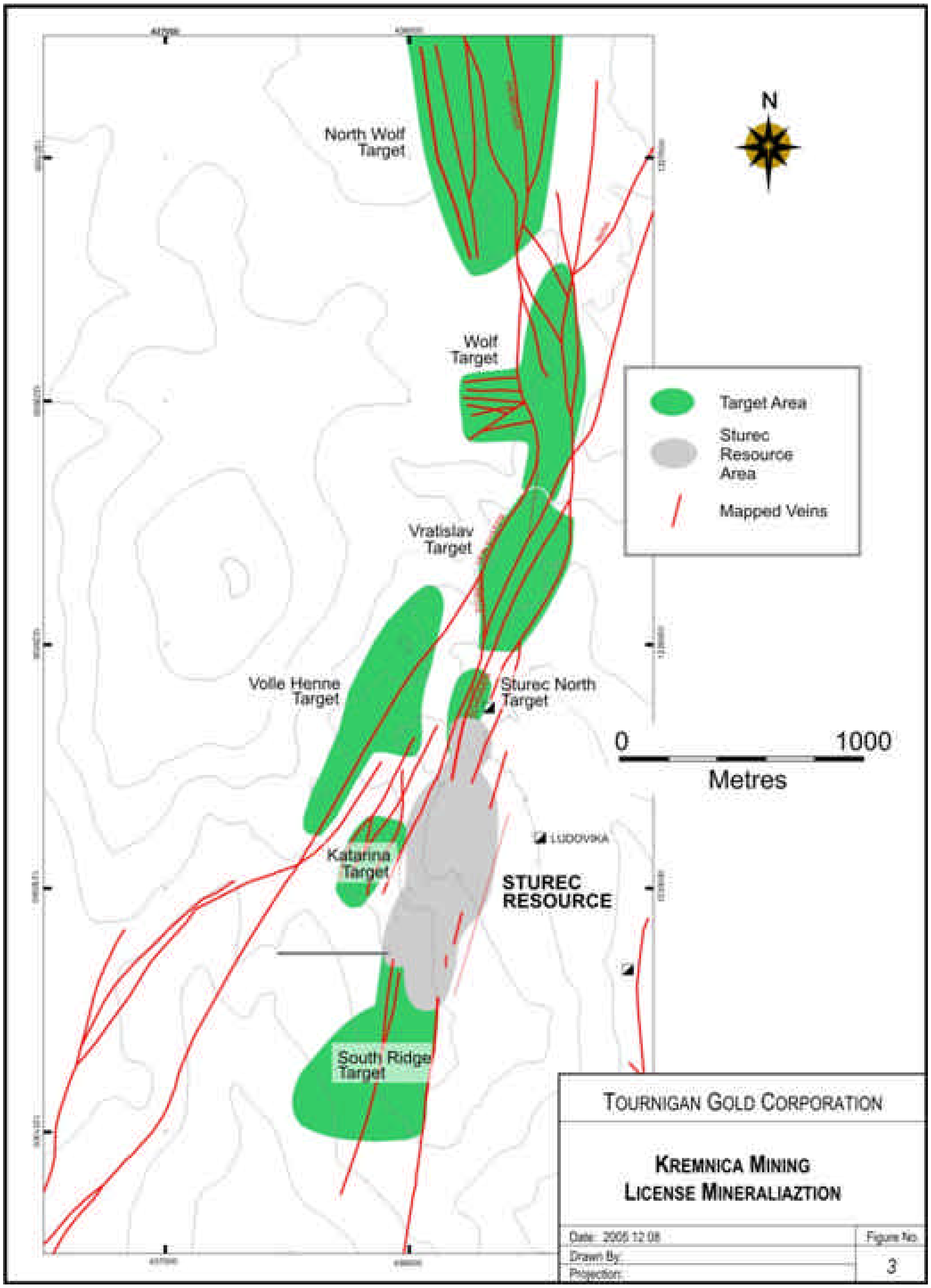

Figure 3: Kremnica Mining License Mineralization

9.2 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. Wolf also contains numerous voids and rubble zones, similar to those encountered at Šturec.

As at Šturec gold mineralization occurs primarily in quartz veins, which 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 Wolf mineralization is defined for 300m strike, and is at least 50m wide and extends to at least 50m depth. The widest vein is the Kirchberger, which is approximately 30-m wide. The mineralogy of the deposit is similar to Šturec, although considerably more silver-rich than the Šturec deposit.

A second sequence of veins at Wolf strike 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 foot wall. 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 down-hole interval 81.5 m to 89.5 m.

9.3 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 of the Schramen vein. The Schramen vein is the eastern-most structure and the Schindler vein the western-most splay, dipping back to the east at 40 ^{o} to 50^{o} 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-m to 10-m thick and grades from 1.5 g/t 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 Slovakian Geologic Survey sampling adjacent to the veins indicates the presence of low-grade stockwork mineralization with gold grades in the 0.25 g/t to 2.1 g/t range.

The Vratislav area was 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 the scarp at Šturec although not as large. An elongated pit approximately 20-m deep 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.

In 2004 Tournigan conducted exploration activities north of Sturec at Wolf and Vratislav (test diamond drilling programs) and south of Sturec throughout the Kremnica South area (large soil geochemical survey covering most of Kremnica South area, test diamond drilling at Certov and Bartasova Lehotka areas at Kremnica South, and limited exploration trenching south of the town of Lucky.

9.4 Other Mineralized Areas within the Kremnica Mining License

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 3. It changes character however 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 1,230,470-South on the mine grid, a major cross structure diverges to the southwest. It branches off of 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 Lucky. The vein system appears to be composed of 3 or more sets of veins striking between N35E to N80E and dips at between 65° to 70° to the NSW. The veins are surrounded by zones of intense stockwork veining and silicification. Between these two major structures is a wedge-shaped block, which 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 g/t 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 1,230,500-South, is about 20-m wide near the ridge crest and appears to fan-out for the next 200 m. Soil survey data indicates that the target may extend 500 m further southwest toward the village of Lucky.

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 3). 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 g/t and 1.05 g/t gold) and silver (42.1 g/t and 24.3 g/t silver).

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 3. 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 by 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 ±3 m-wide, low-grade quartz vein that strikes between N20 to N50E. 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 indicates that 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.

Katarina Target

As reported by Smith and Kirkham, 2004, the Katarina target is located west of the Šturec Resource as illustrated in Figure 3. 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 of 5 holes in the central and northern parts of the Katarina system. There is now an area 150-m by 100-m 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 been also been observed.

A soil-sampling program conducted during 1997 produced a 150m x 400m anomaly. The majority of the soil samples contained gold concentrations 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 5.2, 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 and Underground Sampling".

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

During 2004, Tournigan completed a drilling program in the Kremnica South area. Nine drill holes totaling 2,037 meters were completed at Certov vrch and two holes totaling 421 meters 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.

Tournigan also completed a large soil geochemistry survey over the Kremnica South area along a 40-m X 200-m grid during 2004-2005.

Tournigan is presently focused on completing a pre-feasibility study for the Sturec deposit.

SECTION 11.0

DRILLING

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

1. Underground drilling and channel sampling of various crosscuts by Rudne Bane and Slovak Geological Survey.

2. 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. Figure 4 depicts drillhole locations as well as underground adits with continuous channel sampling.

3. Drilling from the Tournigan 2005 campaign that resulted in 41 infill RC holes including 5 twin holes increasing the assay and lithology database by 3989 sample intervals.

4. Nine Bench channels incorporating a total of 317 sample intervals were also collected during 2005.

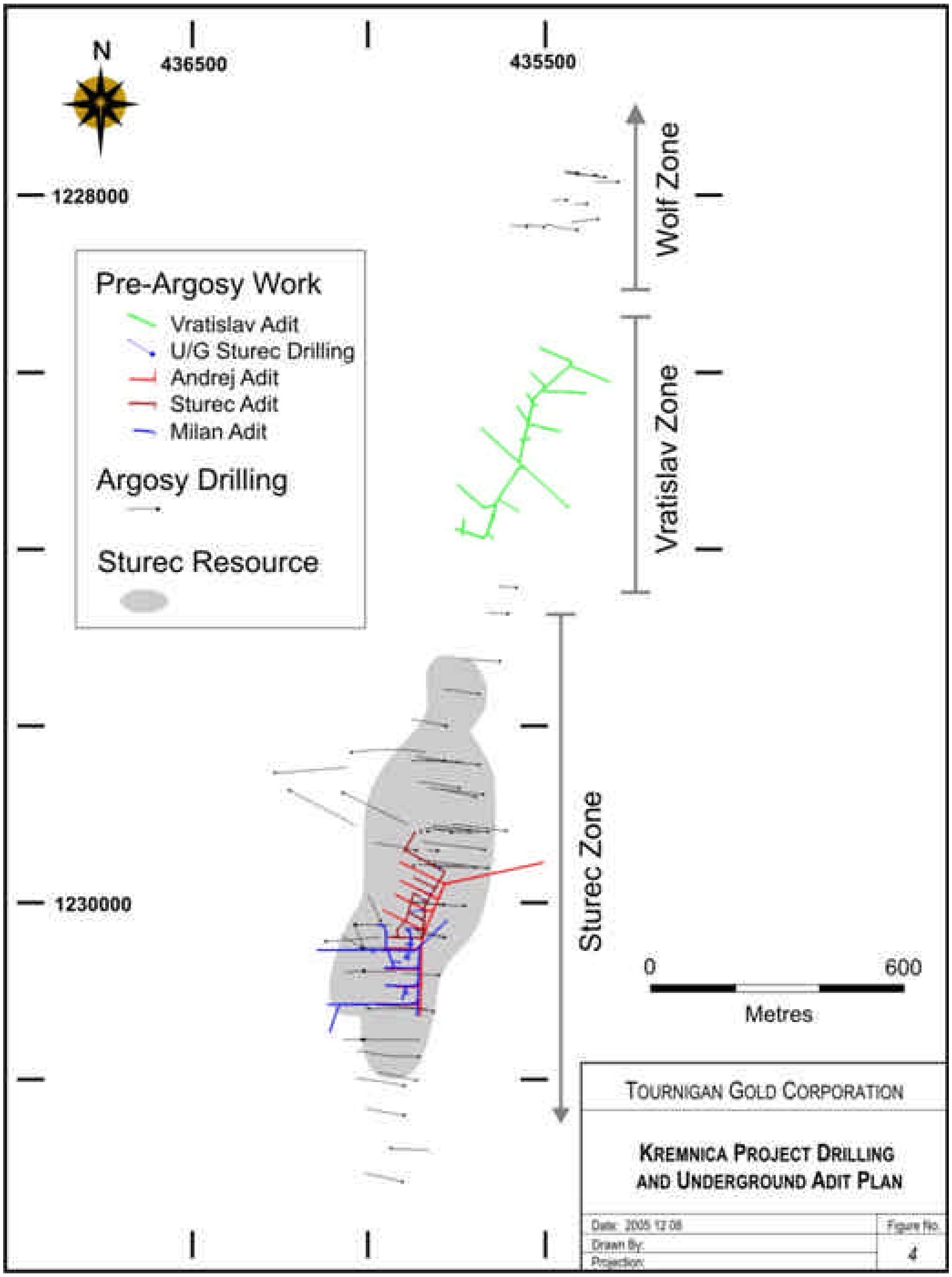

Figure 4: Kremnica Project, Drilling and Underground Adit Plan

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.

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 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.

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 5 through 8. 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.

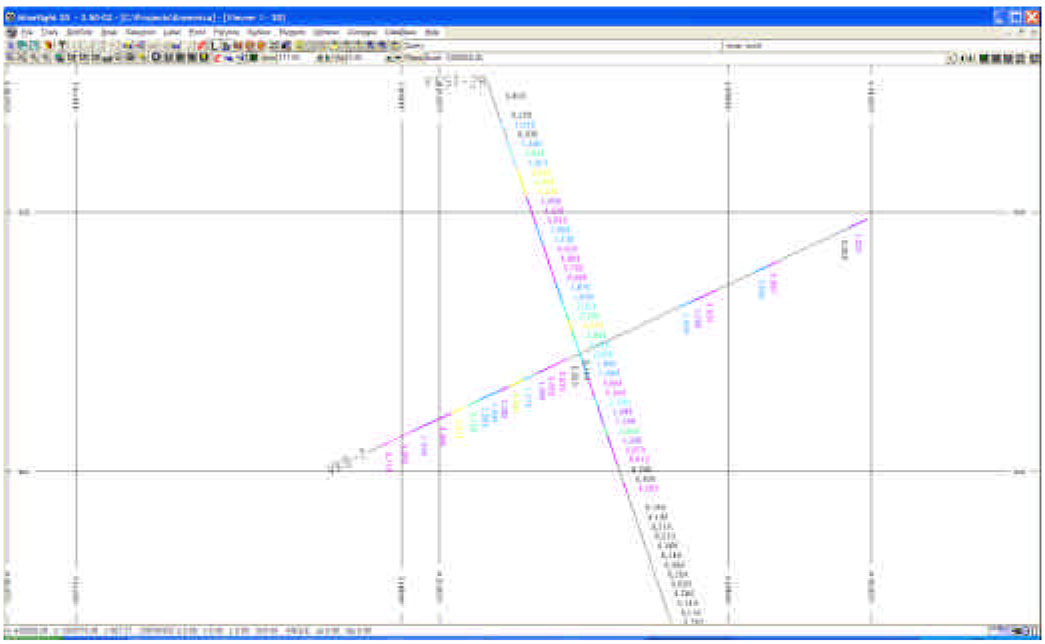

Figure 5: Intersection of KGST-2R and VKB-7 for Assay Comparison and Confirmation

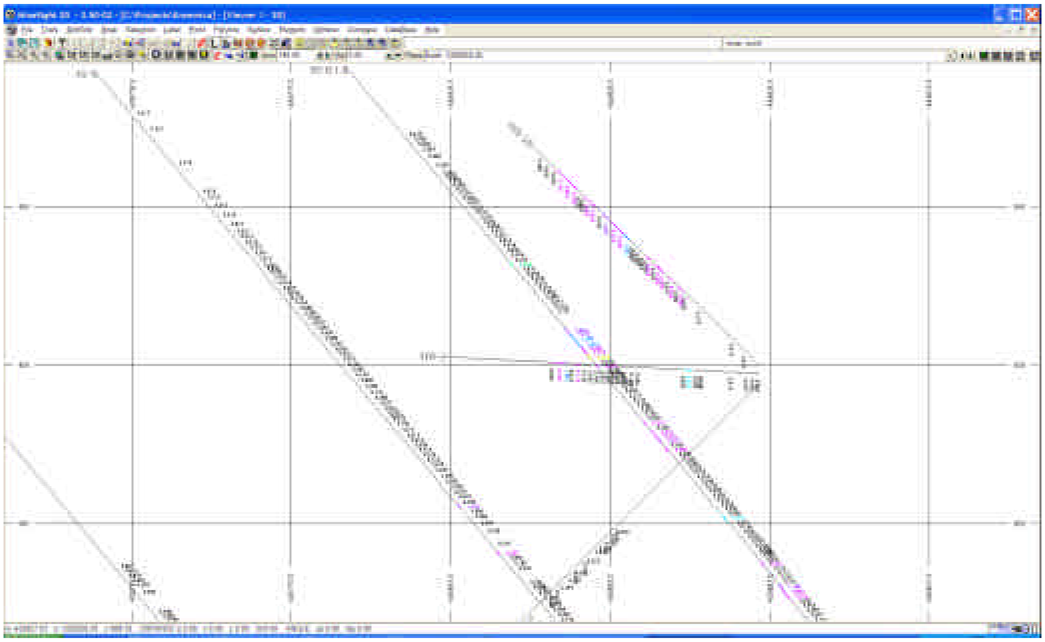

Figure 6: Intersection of AS-8, AS-8.1.B and VKB-2. VKB-2A, VKB-2B for Assay Comparison and Confirmation

Figure 7: Intersection of KGST-3R and VKB-3 and VKB-3R for Assay Comparison and Confirmation

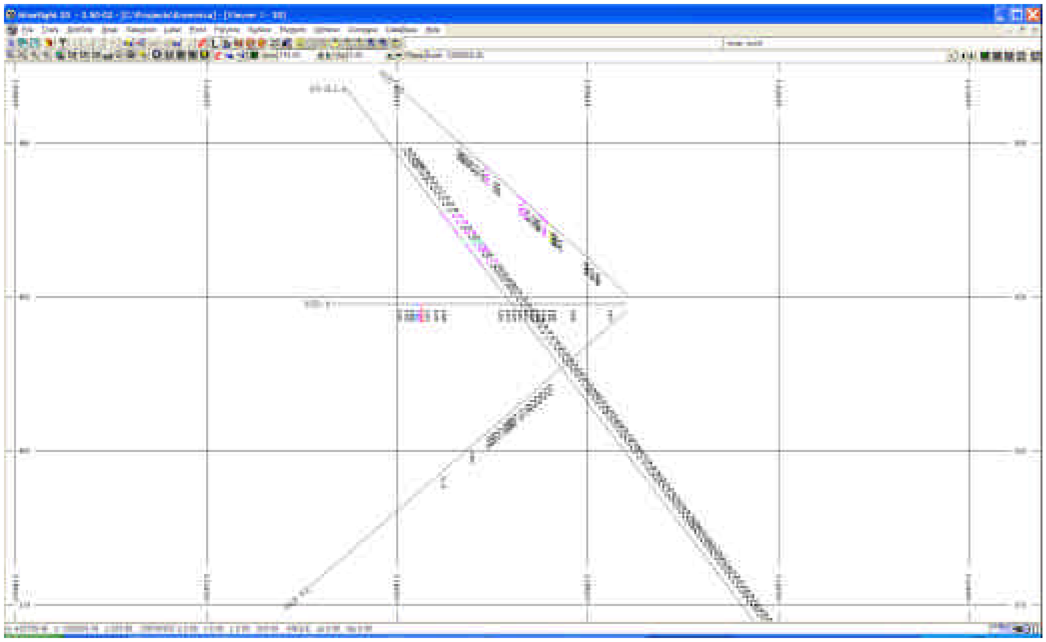

Figure 8: Intersection of AS-9.1.A and VKB-4, VKB-4A, VKB-4B for Assay Comparison and Confirmation

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 9 summarizes the Argosy drill programs.

Table 9
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 resource has been defined in accordance with NI 43-101 regulations.

Drilling at Wolf (12 holes on 3 fences 80m apart) by Argosy lead Western Services Engineering to calculate an indicated mineral resource for Wolf of 74,312 oz gold equivalent, at a 0.5 g/t cut-off. This is a historical resource and is not NI 43-101 compliant.

Exploration by the Slovak Geological Survey in the period 1985 to 1987 led to a resource estimation at Vratislav defined by 12 underground crosscuts and 28 underground diamond drill holes. Vratislav has an indicated resource of 53,152 oz gold equivalent, defined by the Slovak Government in 1992. This is a historical resource and is not NI 43-101 compliant.

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 a 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, ANALYSES 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 (=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 conducted by the author, Mr. von Einsiedel, during his site visit at Kremnica, and a systematic evaluation of the global data set discussed in detail in sections 17.3 Assay Database.

14.1 Control Sampling

During the site visit in July 2005 the author observed the drilling of Drill Hole Number KG-ST-7R and KG-ST-8R. The author personally sealed the 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 9 and 10 show the results for gold and silver, respectively which exhibit an excellent correlation between the two laboratories.

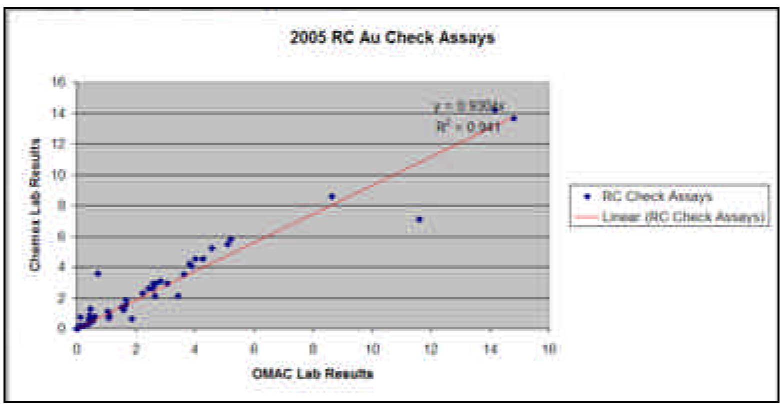

Figure 9: Au Check Assay Results; OMAC vs. Chemex

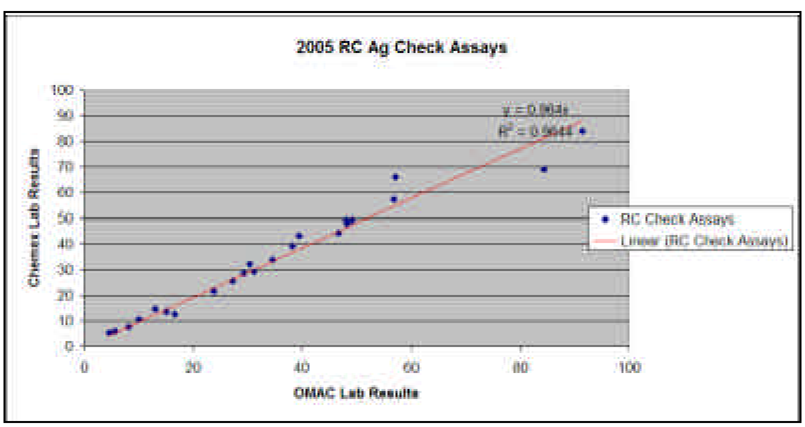

Figure 10: Ag Check Assay Results; OMAC vs. Chemex

In addition to the control sampling program completed to assess the relative accuracy of the OMAC assay facility in Ireland the author also collected a suite of 38 pulps from the Rudne Bane underground crosscut sampling program. Figure 11 and 12 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.

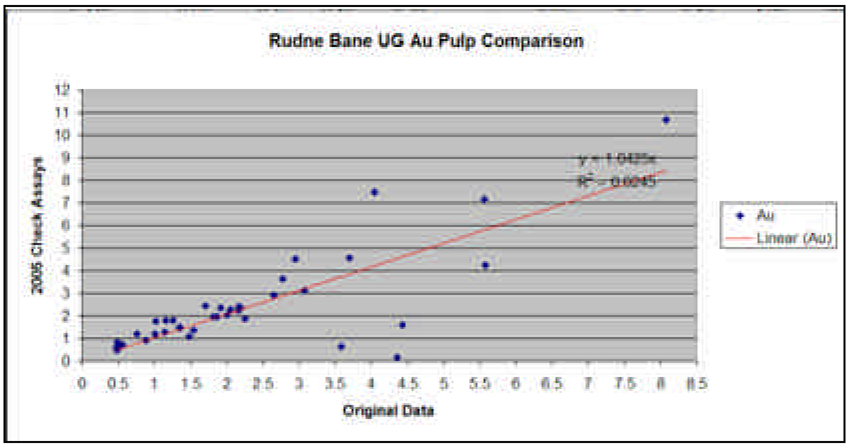

Figure 10: Au Check Assay Results; Rudne Bane vs. Chemex

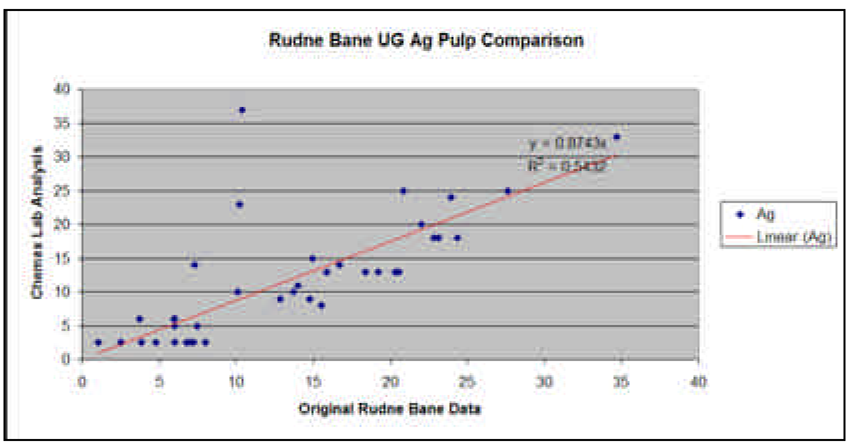

Figure 11: Ag Check Assay Results; Rudne Bane vs. Chemex

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 10. 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 10
Kremnica Project Control Sampling

Bruce Smith Control Samples, November 03

Rock chip field sampling

Location	Sample type	Control No.	Control Assay (OMAC Ireland analysis) Au g/t	Ag g/t
Main Schramen vein, sampled undergr ound 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	Control No.	Au g/t	Ag g/t
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	Control No.	Au g/t	Ag g/t
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	Control No.	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
AS144	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 12 and 13. 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.

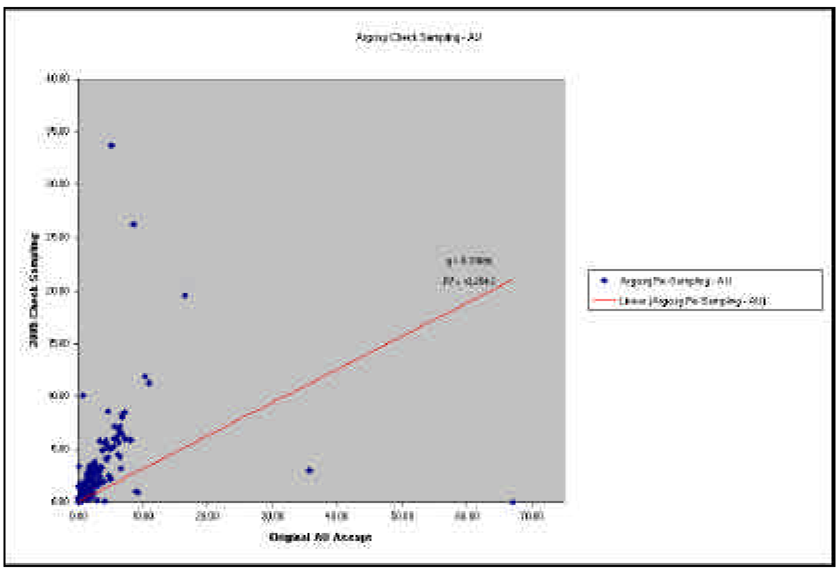

Figure 12: Au Check Assay Results; Argosy Drillhole Data

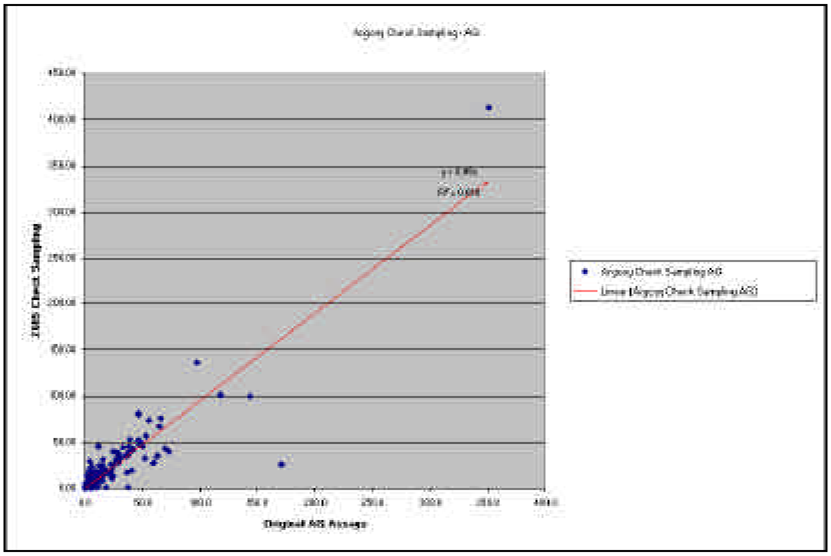

Figure 13: Ag Check Assay Results; Argosy Drillhole Data

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.

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

Currently there are no adjacent properties to the Kremnica property. Tournigan is the only minerals company actively exploring for precious metals in the district, and currently has all of the known prospective ground under license or application.

The nearest other known prospective areas are centered on the historic mining district Banska Stiavnica, approximately 40km to south. At this time small-scale underground Au-Ag production is presently on going at the Hodrusa Mine in the Banska Bystrica district.

SECTION 16.0

MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 Historic Operations

Ore processing has been documented at Kremnica for over 600 years, and records indicate that gold recovery has been relatively straightforward using gravity, flotation and cyanidation processing.

Throughout most of the history of the district, 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 the 70 percent range.

In the mid 1980's, the 1935 plant was upgraded to treat ore by direct cyanidation. Ing. Oliver Finca, the Mine Engineer, reports that subsequent operations from 1987 to 1992 achieved gold recoveries reaching almost 80%. Typical reagent consumptions in kg/t were reported to be 0.88 for cyanide, 0.11 for Zn, 1.96 for Cl and 7.1 for lime.

16.2 Argosy Sponsored Test Work

Argosy utilized Hazen Research, Inc. 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 11 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 have been optimized and that no test work has been done on the Wolf deposit.

Whole ore cyanidation in bottle roll tests yielded gold recoveries of 94.4, 91.0, 89.2, 84.1, and 95.8 percent with cyanide consumption ranging from 1.3 to 2.2 kg/t at minus 170 to 270 mesh grinds. Oxide and sulfide mineralization provided approximately the same results.

Flotation on –200 mesh sulfide mineralized material yielded gold recoveries generally in the 90 to 94 percent range. Similar tests on oxide mineralization gave results in the 65 to82 percent range. Flotation concentrates from quartz vein material total only 1.41 to 6.84 percent of the weight by feed, while concentrates from andesite mineralization total 8.11 to 29.75 percent 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 have been performed on flotation concentrates. Gold recoveries for the two tests on quartz vein mineralized concentrates were 96.4 and 97.7 percent. Gold recovery for the single test on andesite concentrate was 91.0

percent. Cyanide consumption in the three tests ranged from 0.49 to 0.66 kg/t of original ore.

R. Mac Pherson, a joint venture of Hazen Research 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/mt and abrasion indices of 0.3368 to 1.2282 lbs/kWh.

Table 11
Densities and Metallurgical Recoveries

Rock Type	Metallurgical Parameter	Oxide	Mixed	Sulfide
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

( ) Numbers in brackets represent Argosy estimates based on results of other tests.

A metallurgical program, under the guidance of Beacon Hill, is underway as part of the pre-feasibility study.

SECTION 17.0

MINERAL RESOURCE ESTIMATE

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 12 lists some conventions and abbreviations that are encountered throughout the resource estimation section of this report.

Table 12
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	TSX 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 this 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 volume for a 1 meter length of core.
- Specific gravity is assumed to be 2.5 tonne/ m^3 which again are conservative for the calculations.

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, oftentimes 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 13 and 14 summarizes statistics for the 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 (Figure 15 and 17). 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[1] (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 a outliers with 36 Ag values being greater than 100 g/t and the maximum being 501 g/t. Figures 14 and 16 shows the histogram and basic statistics of all Au and Ag assays weighted by assay interval, respectively. In addition, Figures 15 and 17 shows the corresponding probability plot for Au and Ag, respectively. Figure 18 illustrates a plan view of the drill holes for spatial reference.

[1] The coefficient of variation is defined as CV=s/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 13
Statistics of all Au samples by Cut-off Grade.

Cut-off	UnWeighted					Weighted				
Cut-offGrade gAu/t	Weight	%	AU	SD	CV	Weight	%	AU	SD	CV
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	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	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	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	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	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	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	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	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	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	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	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	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	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	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	25	0.2%	29.94	17.55	0.59	29.7	0.2%	29.30	17.33	0.59

Table 14
Statistics of all Ag samples by Cut-off Grade.

Cut-off	UnWeighted					Weighted				
Cut-offGrade gAu/t	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

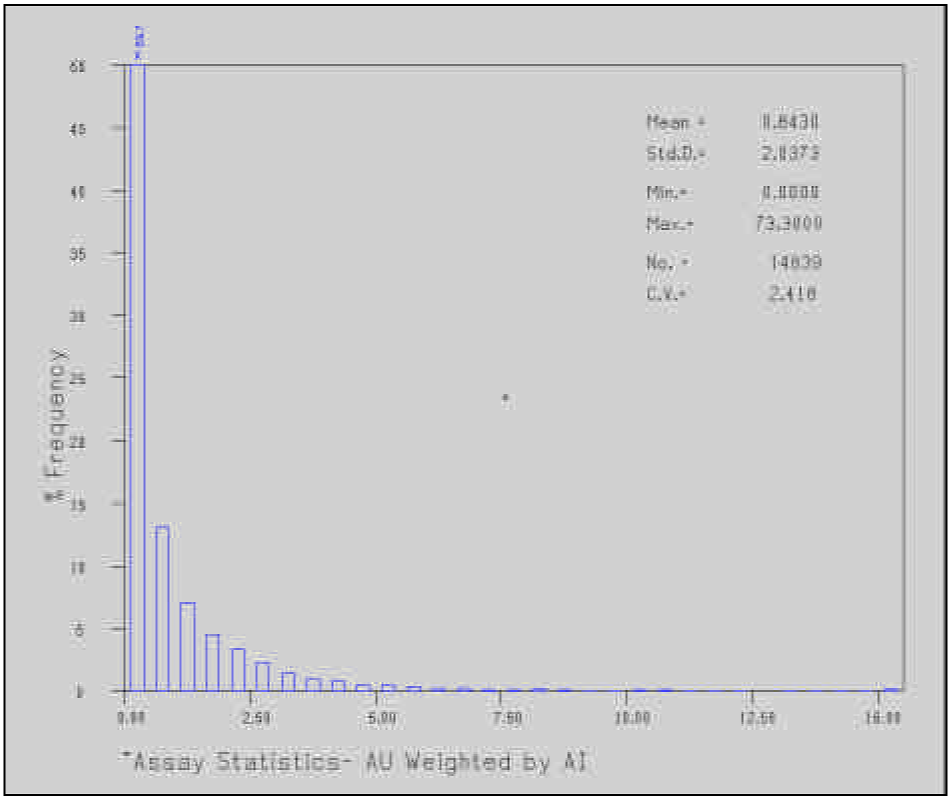

Figure 14: Histogram and Basic Statistics of all Au samples Weighted by Assay Interval Length.

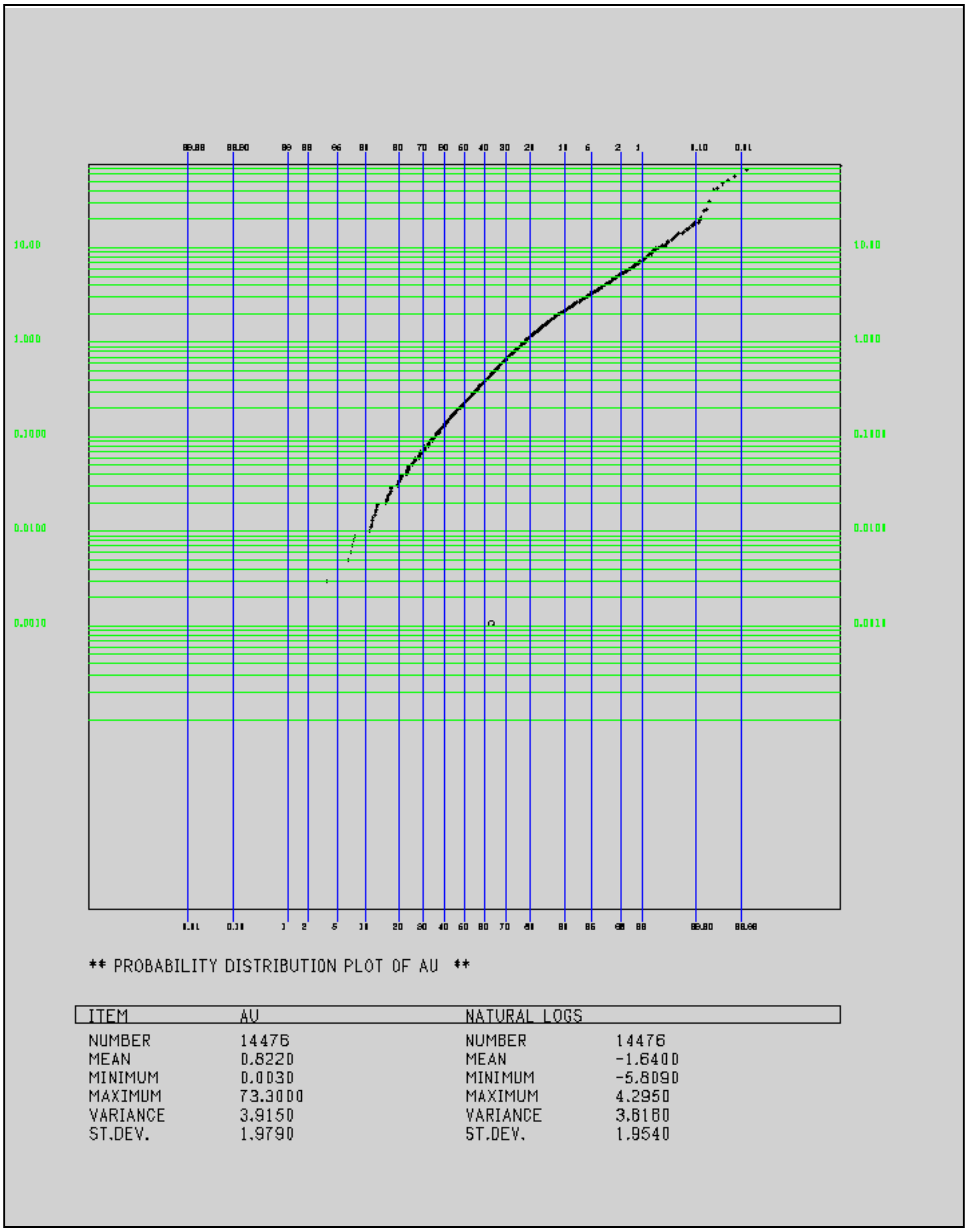

Figure 15: Probability Plot of all Au samples, samesamples as Figure14.

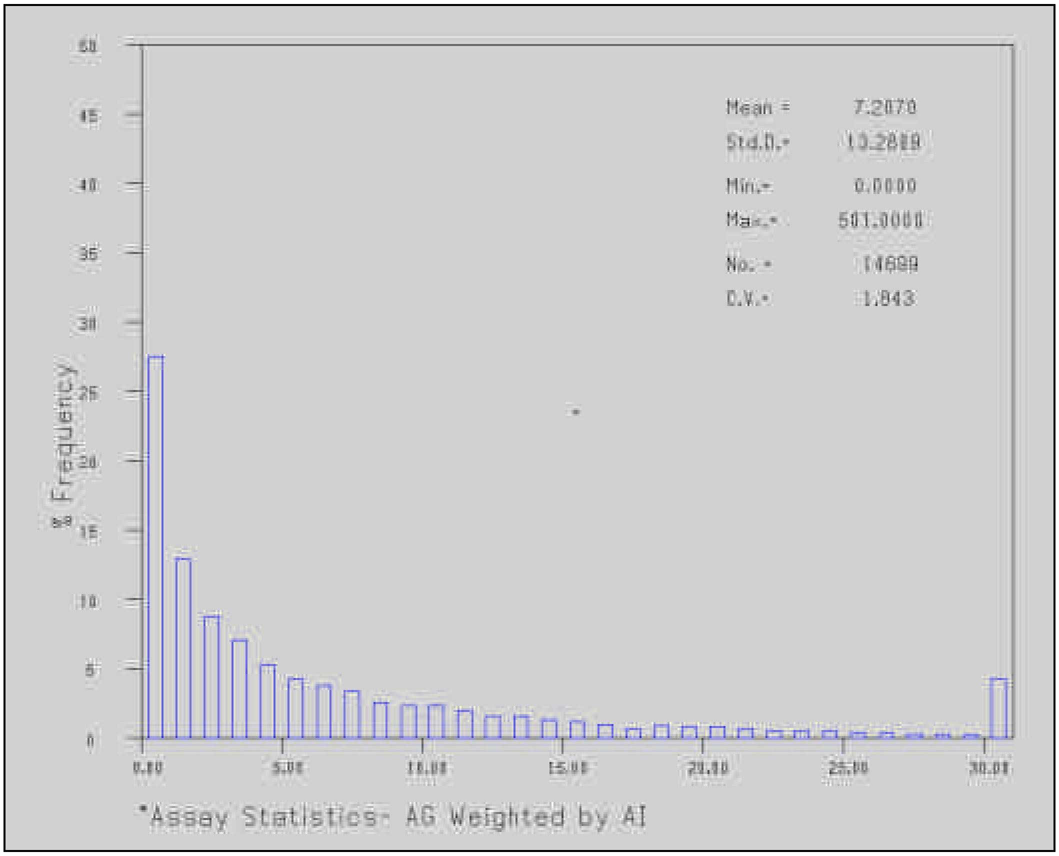

Figure 16: Histogram and Basic Statistics of all Ag samples Weighted by Assay Interval Length.

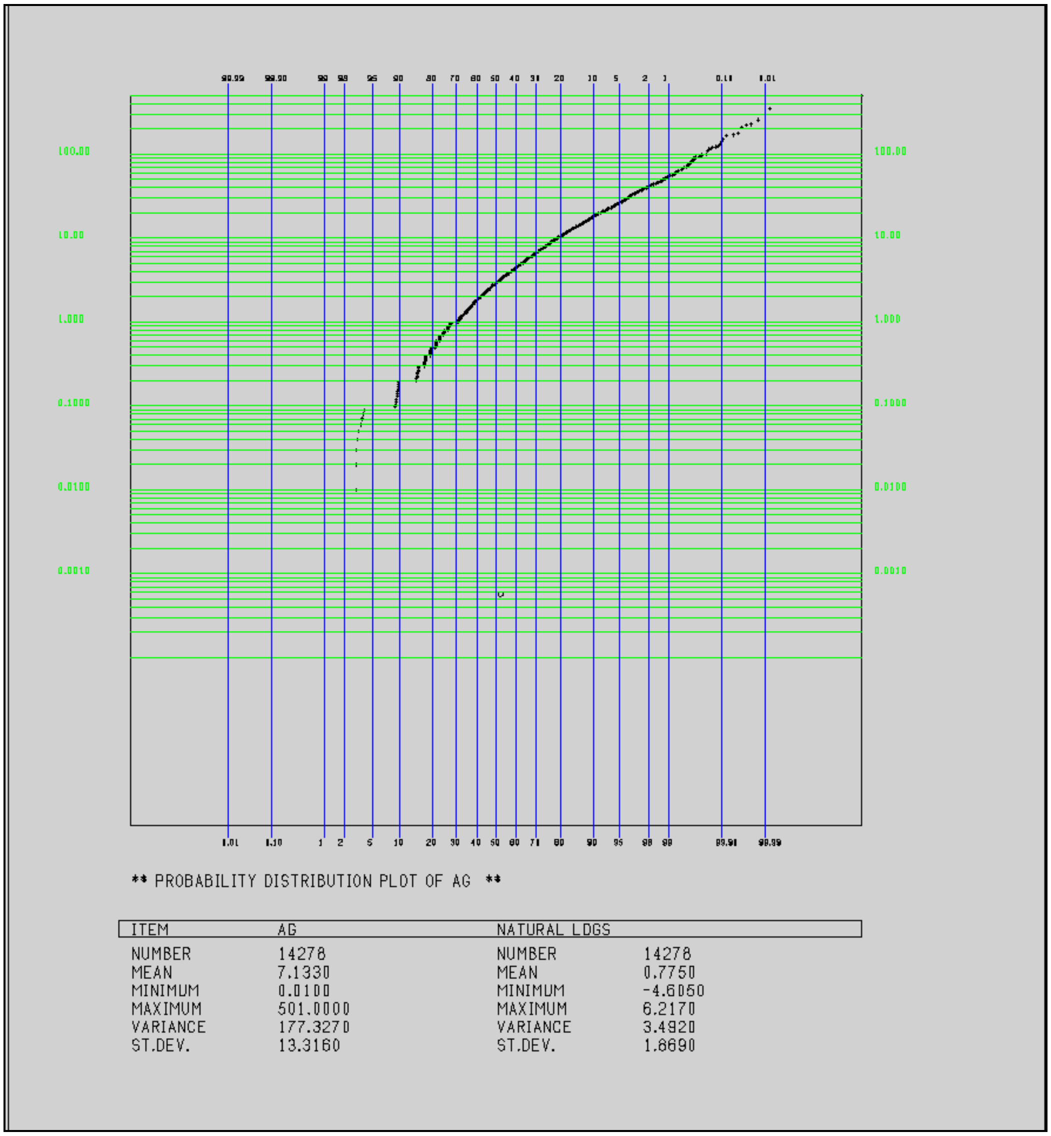

Figure 17: Probability Plot of all Ag samples, same samples as Figure 16.

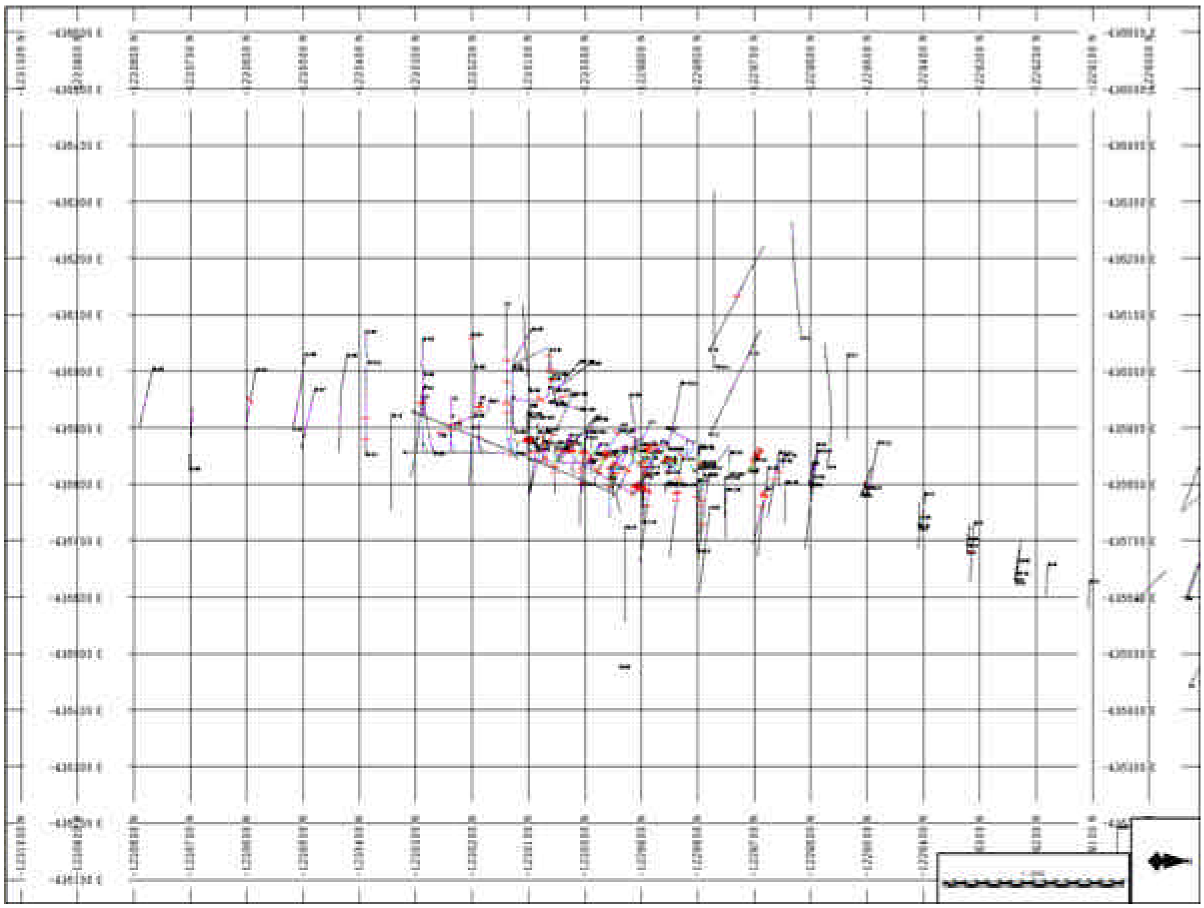

Figure 18: Plan view of the drill holes within the project area displaying the Slovak Grid System.

17.3 Topography

Topography was imported from an AutoCAD topographic map supplied by Tournigan Gold Corp. in DXF format. Figure 10 shows a plan view of the topographic map and Figure 20 illustrates a three-dimensional rendition of the topography used, looking North. Note that the elevations range between 620 and 800 meters (Figure 20) 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.

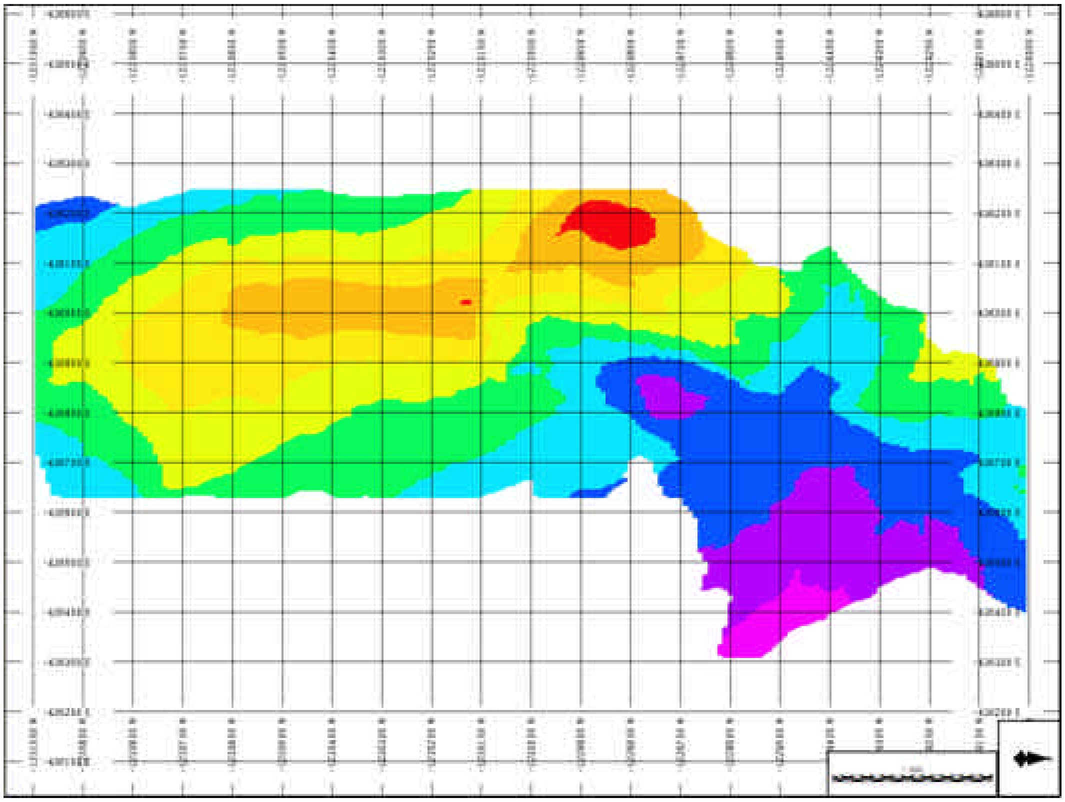

Figure 19: Plan View of Color-coded Topography, Kremnica Deposit with elevations ranging from 620 to 800m, approximately.

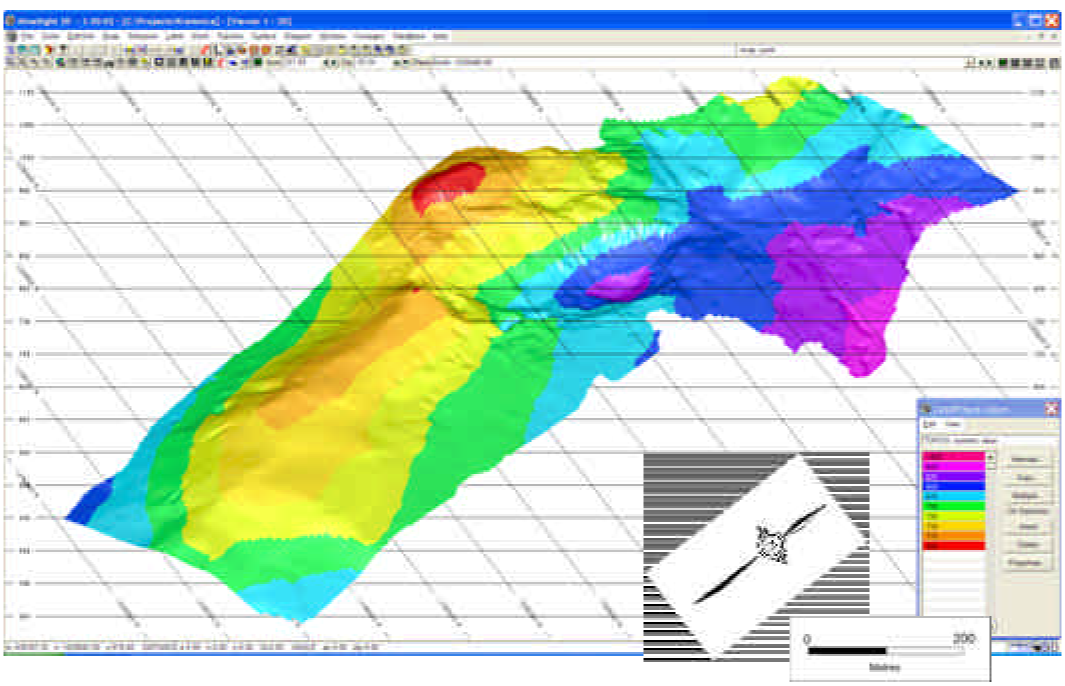

Figure 20: 3D Display of color-coded topography presents elevations ranging from 620m to 800m, approximately. Northwest is into page.

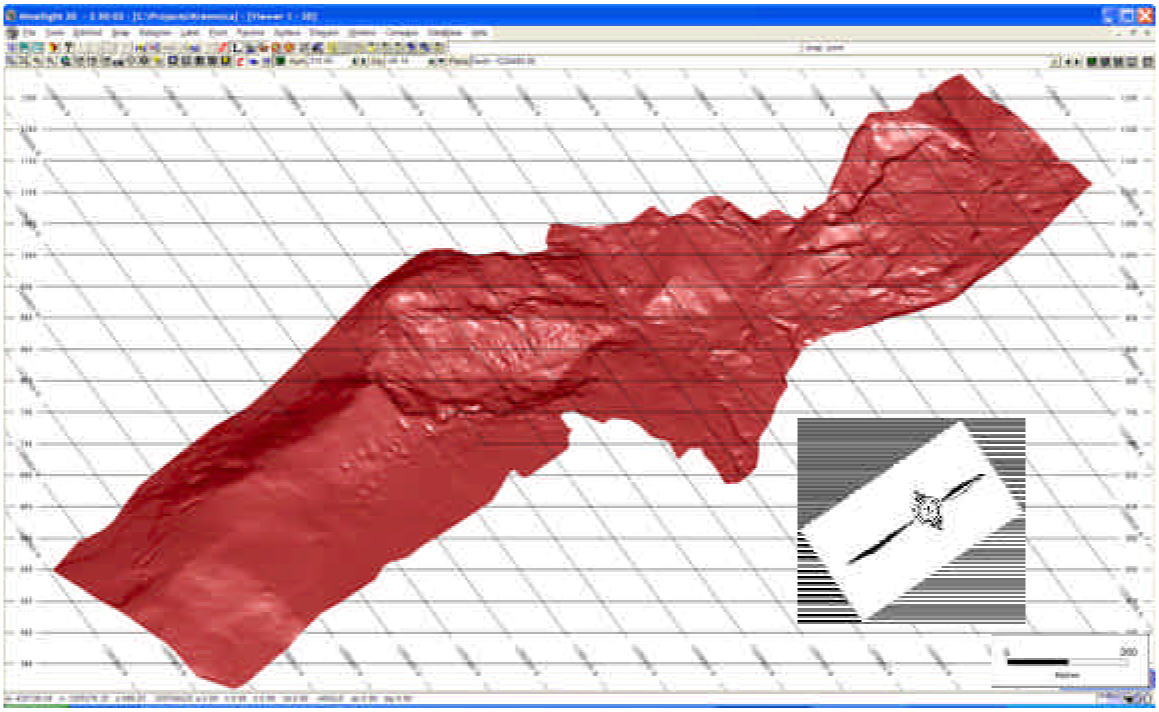

Figure 21: 3D Digital Terrain model. Northwest is into page.

17.4 Density.

A total of 134 specific gravity measurements were taken in order to accurately determine the relative density of the different materials. The individual specific gravity measurements are list in the Appendix however Table 15 lists the averages for each rock type encountered that is used in the specific gravity model as described in the following section.

Table 15
2005 Specific Gravity Determinations for the Kremnica Deposit.

Rock Type	Sulfide /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 16. 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 16.
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 sulfide 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 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.

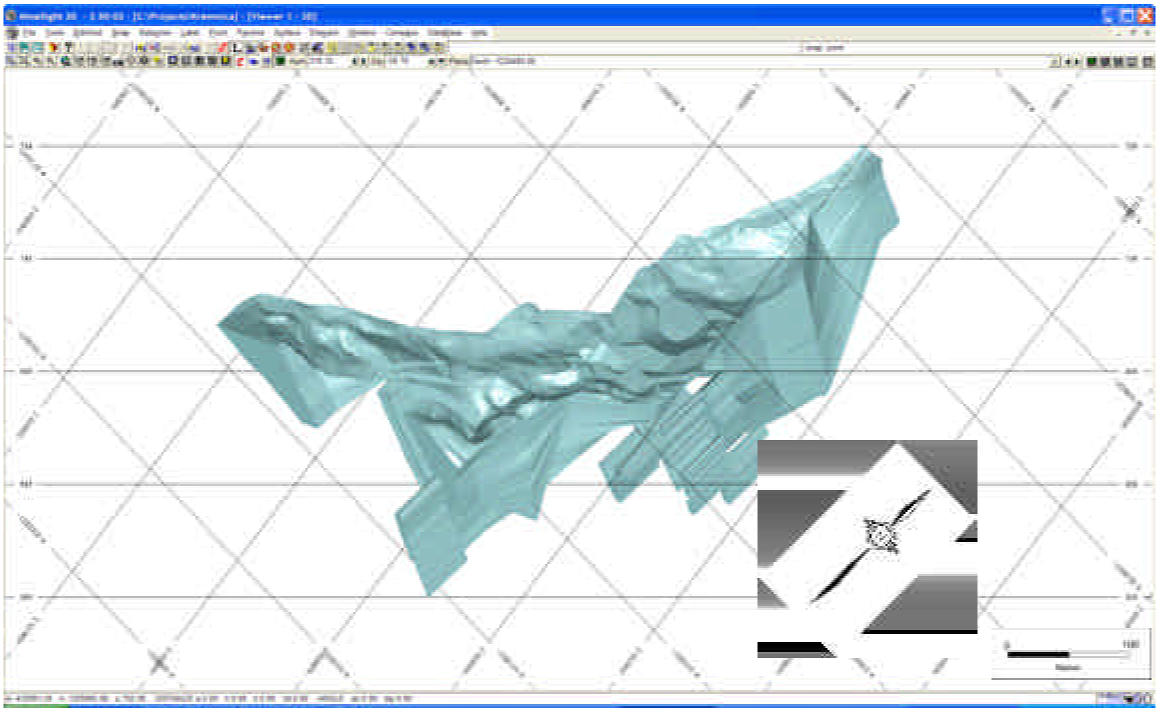

Figure 22: Collapse Zone solids model.

In addition to the collapse zone, a solids model was created for the oxide/sulfide 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 23. As noted in Table 15, the vein and the stockwork have differing SG's depending upon whether they are oxide or sulfide. 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 sulfide zones vary from previous interpretations in that there appears not to be a mixed oxide/sulfide zone as previously thought. It appears that all zones have some oxide and sulfide component to them and are therefore all mixed to some extent however the oxide zone is more strongly oxide than sulfide and the sulfide zone is more sulfide than oxide. In addition, there is no evidence that would indicate that the gold and silver mineralization is controlled by the oxide and/or sulfide 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.

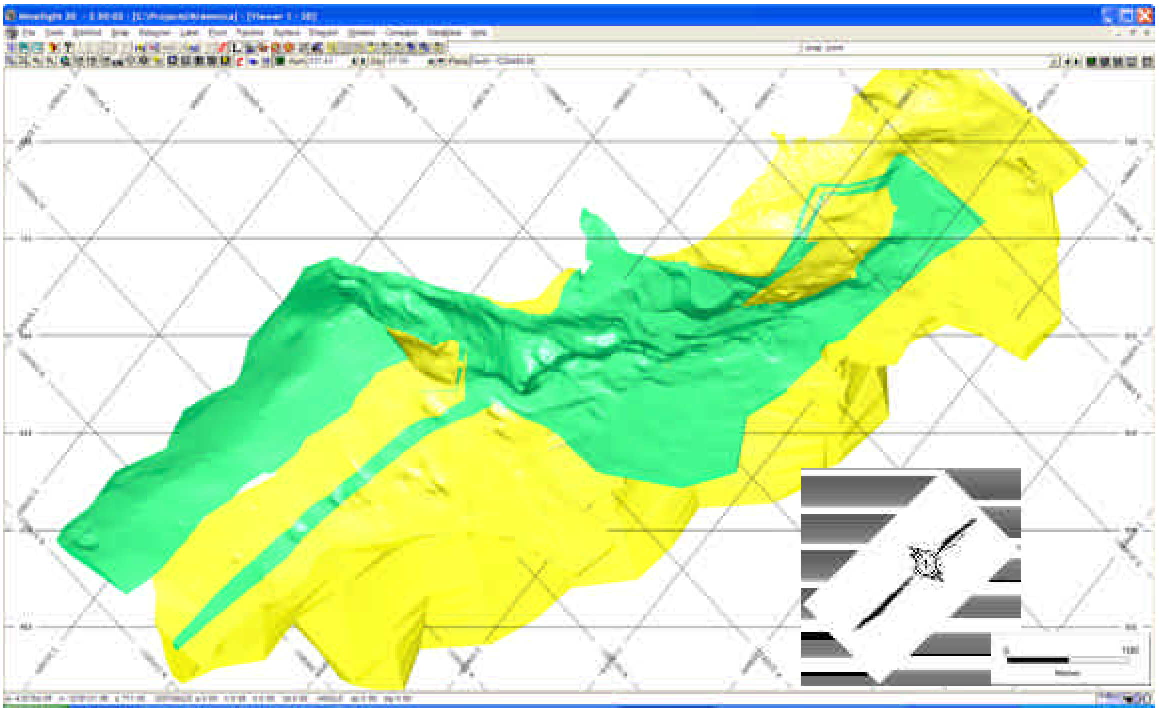

Figure 23: Oxide (green) and sulfide (yellow) solids model

Lastly, a solids model for all known voids was created. These detailed models of the voids were derived from coordinates provided by site personnel; in particular Mr. Finka who is has a long history with the site and is the last person to have visited the underground opening, 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 24, 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 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 black regions.

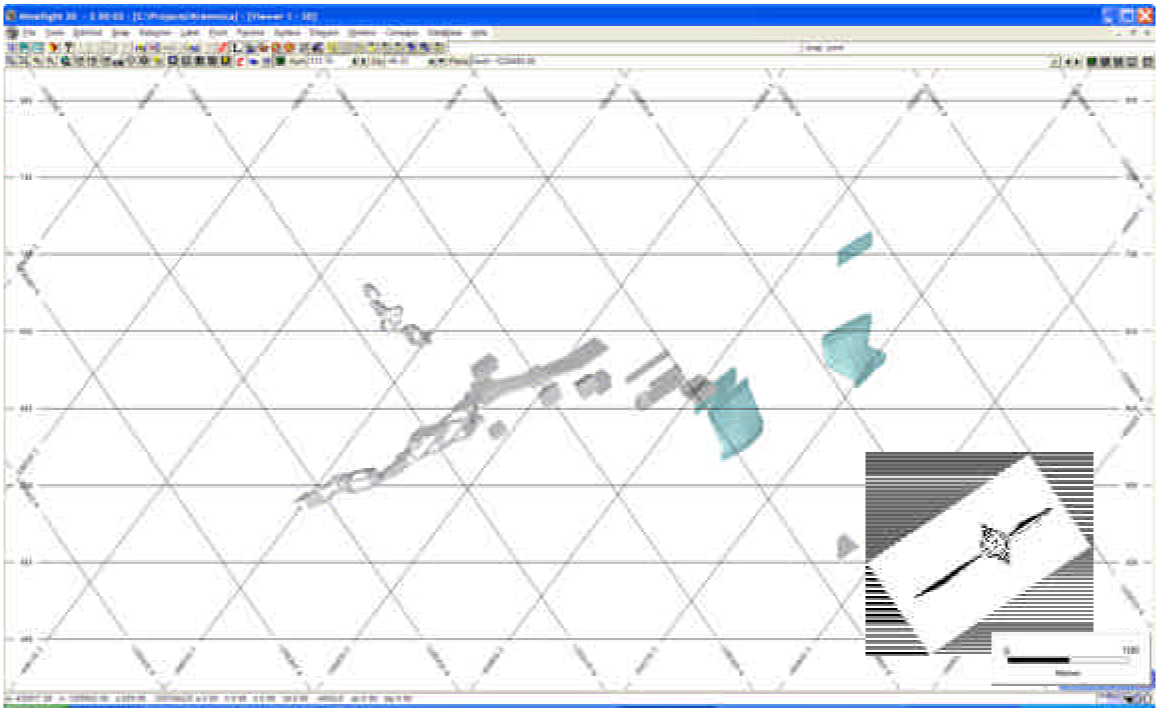

Figure 24: Solids model of voids.

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 Figures 25 and 26. These sections and plans were then digitized into AutoCAD and then imported into MineSight for wireframe modelling. The resultant solids are shown below in Figures 27 and 28. 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 the utilized 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.

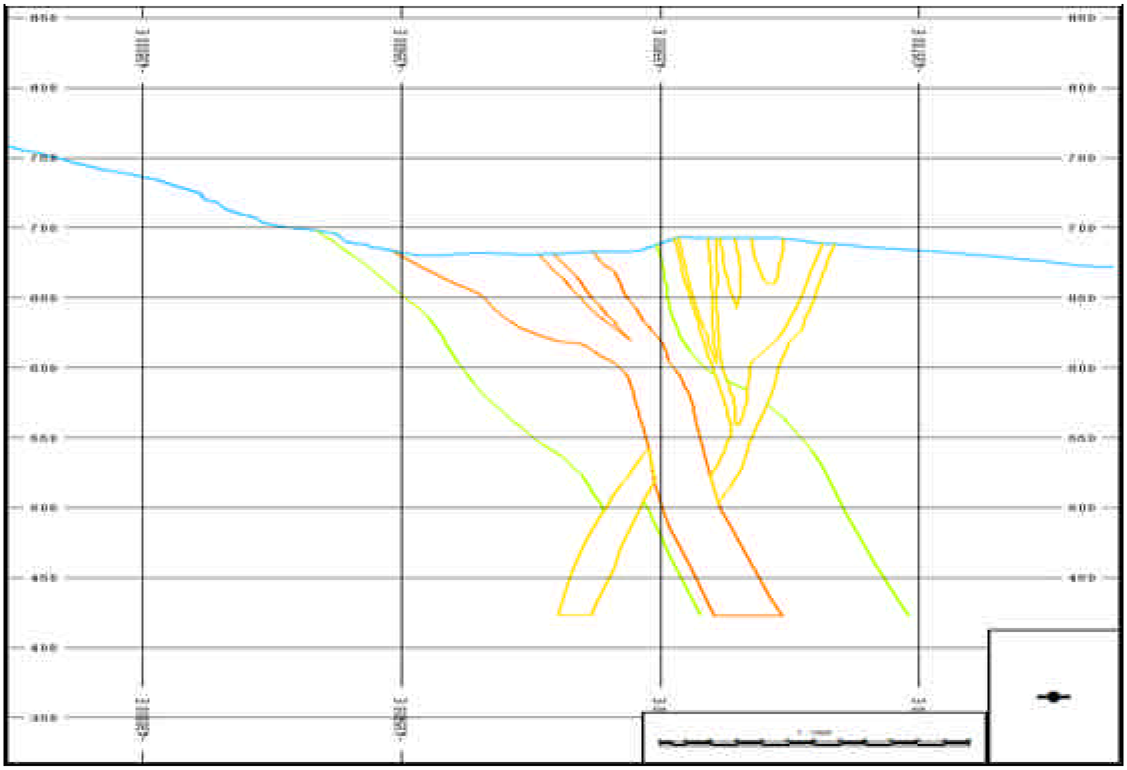

Figure 25: Geologic Section from which geologic solids are derived looking north. Topo is blue, vein is red, quartz breccia is orange and andesite (hangingwall and footwall) are green.

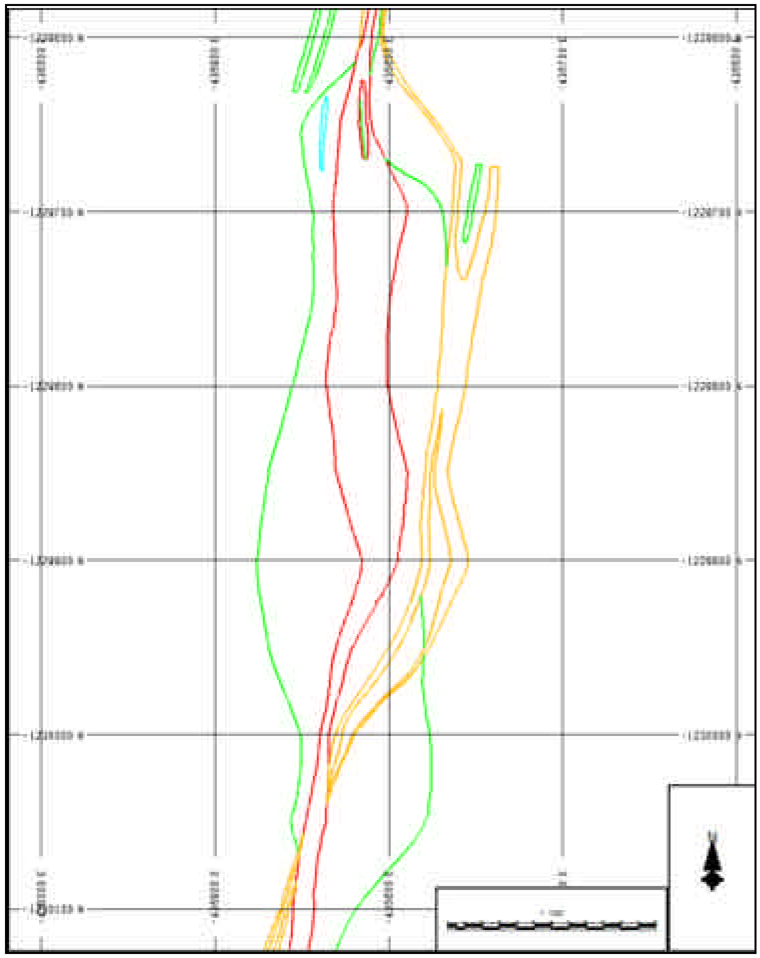

Figure 26: Plan section of geology. Vein is red, quartz breccia is orange, stopes and voids are light blue and andesite (hangingwall and footwall) are green.

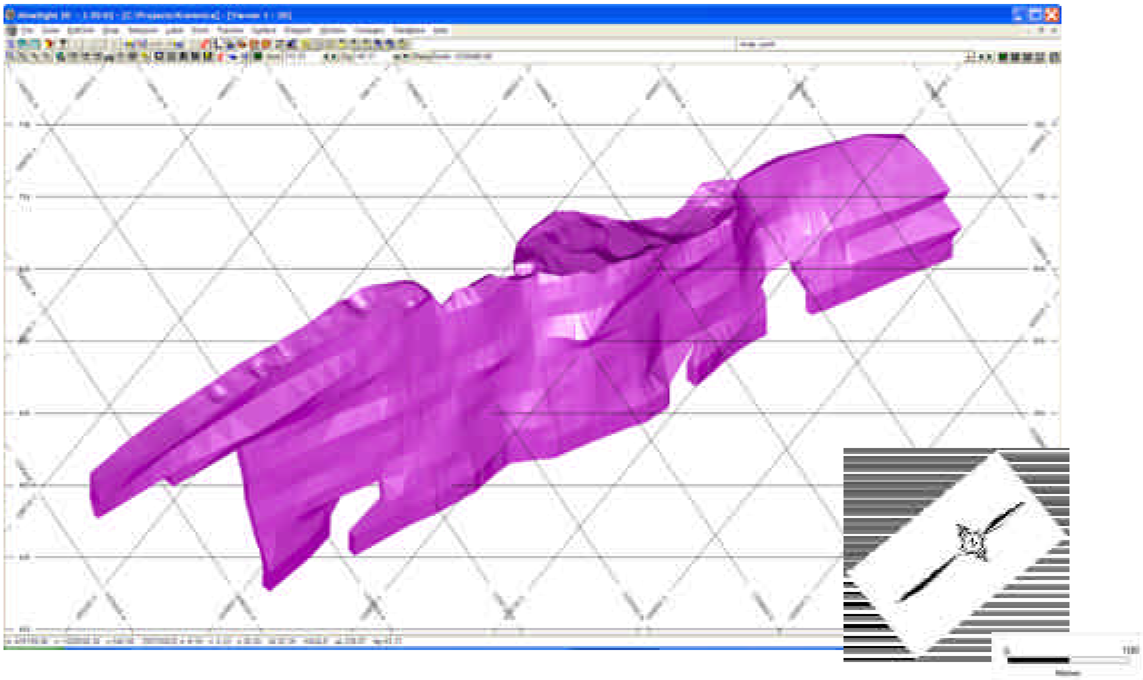

Figure 27: Solids model of vein material.

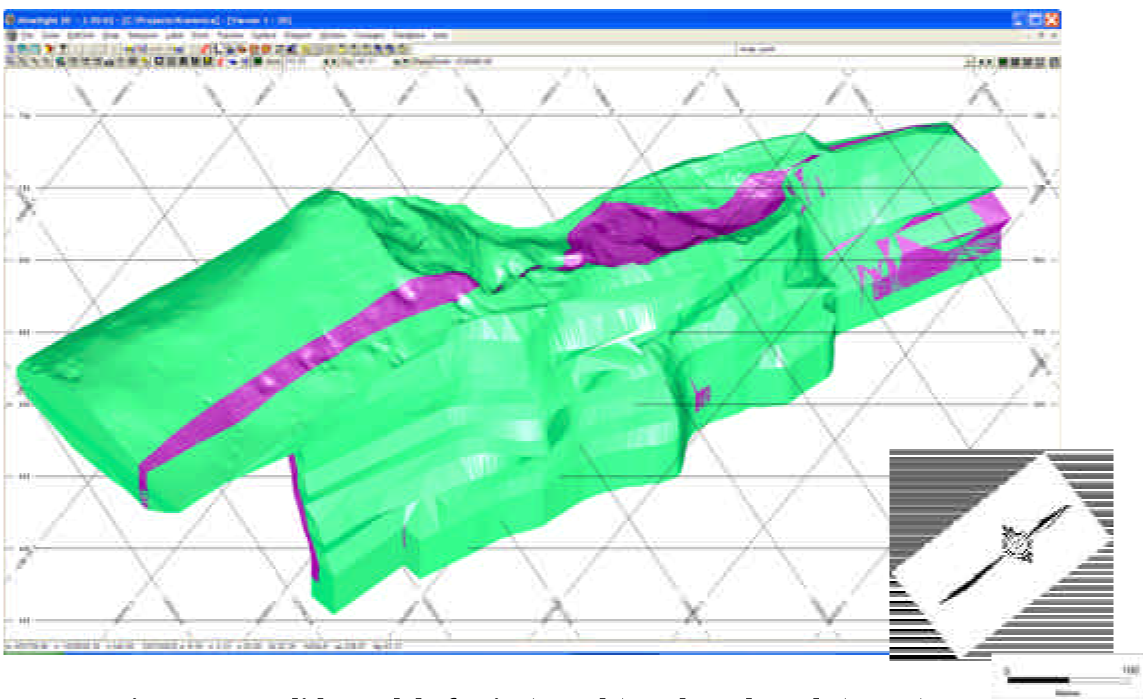

Figure 28: Solids model of vein (purple) and stockwork (green).

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 assay interval length of one meter or less with the overall mean length being 1.37 meters as shown in Figure 29. 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.

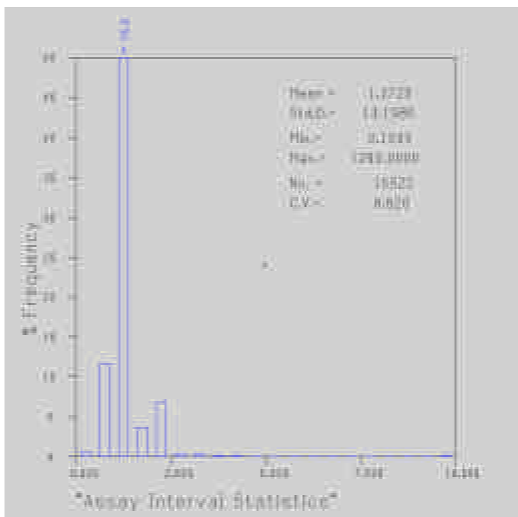

Figure 29: Histogram and Basic Statistics Assay Intervals

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.

Table 17 and 18 summarizes statistics for the 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 a minimum value 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[2] (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.

[2] The coefficient of variation is defined as CV=s/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.

Figure 30 through 33 shows the histogram and statistics along with probability plots for the 3 meter Au and Ag composites, respectively.

Table 17
Statistics of all Au composites by Cut-off Grade.

Cut-off Grade	Weight	%	AU	SD	CV
0	4932	100.0%	0.85	1.56	1.84
0.5	1989	40.3%	1.88	2.06	1.09
1	1308	26.5%	2.49	2.31	0.93
1.5	880	17.8%	3.10	2.61	0.84
2	595	12.1%	3.75	2.96	0.79
2.5	409	8.3%	4.44	3.35	0.75
3	293	5.9%	5.11	3.75	0.73
3.5	198	4.0%	6.01	4.28	0.71
4	142	2.9%	6.91	4.76	0.69
4.5	100	2.0%	8.03	5.30	0.66
5	75	1.5%	9.12	5.71	0.63
5.5	60	1.2%	10.08	6.02	0.60
6	53	1.1%	10.67	6.18	0.58
6.5	45	0.9%	11.45	6.39	0.56
7	36	0.7%	12.64	6.64	0.53
7.5	31	0.6%	13.51	6.77	0.50
8	28	0.6%	14.12	6.86	0.49
8.5	26	0.5%	14.56	6.92	0.48
9	23	0.5%	15.32	7.01	0.46
9.5	21	0.4%	15.90	7.07	0.44
10	20	0.4%	16.21	7.11	0.44
10.5	19	0.4%	16.53	7.16	0.43
11	17	0.3%	17.21	7.28	0.42
11.5	14	0.3%	18.53	7.39	0.40
12	12	0.2%	19.63	7.43	0.38
12.5	10	0.2%	21.09	7.30	0.35
13	10	0.2%	21.09	7.30	0.35
13.5	9	0.2%	21.94	7.20	0.33
14	7	0.1%	24.27	6.37	0.26
14.5	7	0.1%	24.27	6.37	0.26
15	6	0.1%	25.87	5.22	0.20

Table 18
Statistics of all Ag composite s by Cut-off Grade.

Cut-off Grade	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

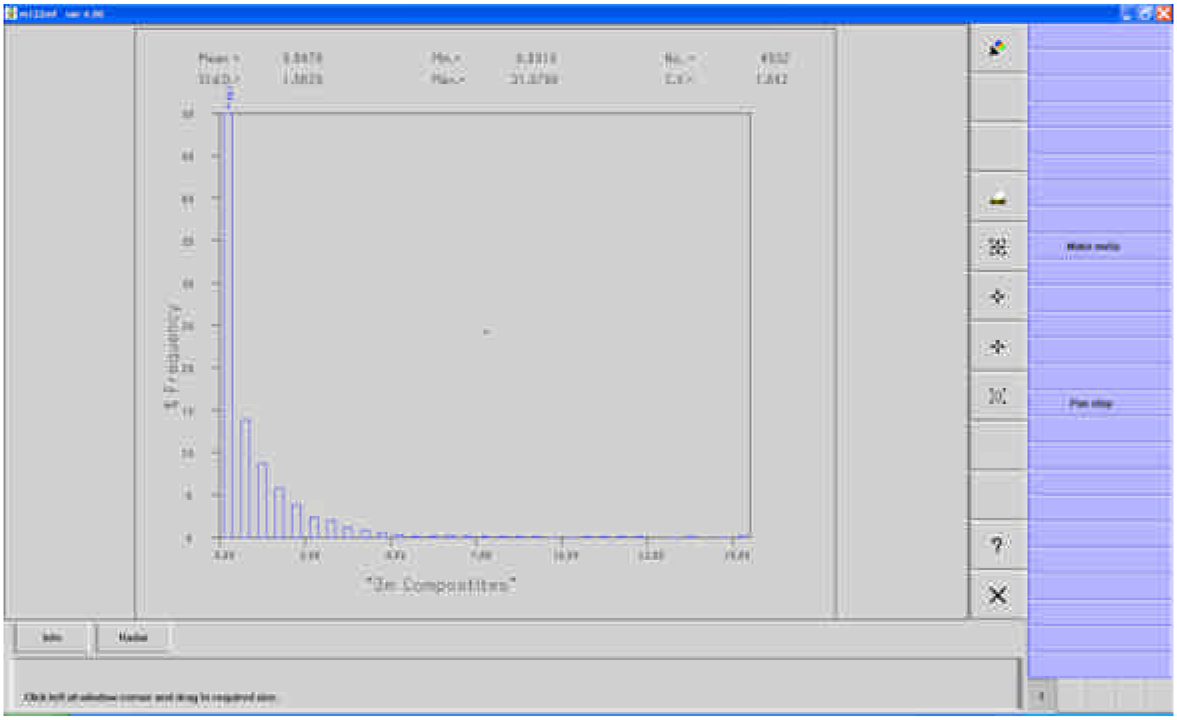

Figure 30: Histogram and Basic Statistics of 3 meter Au Composites

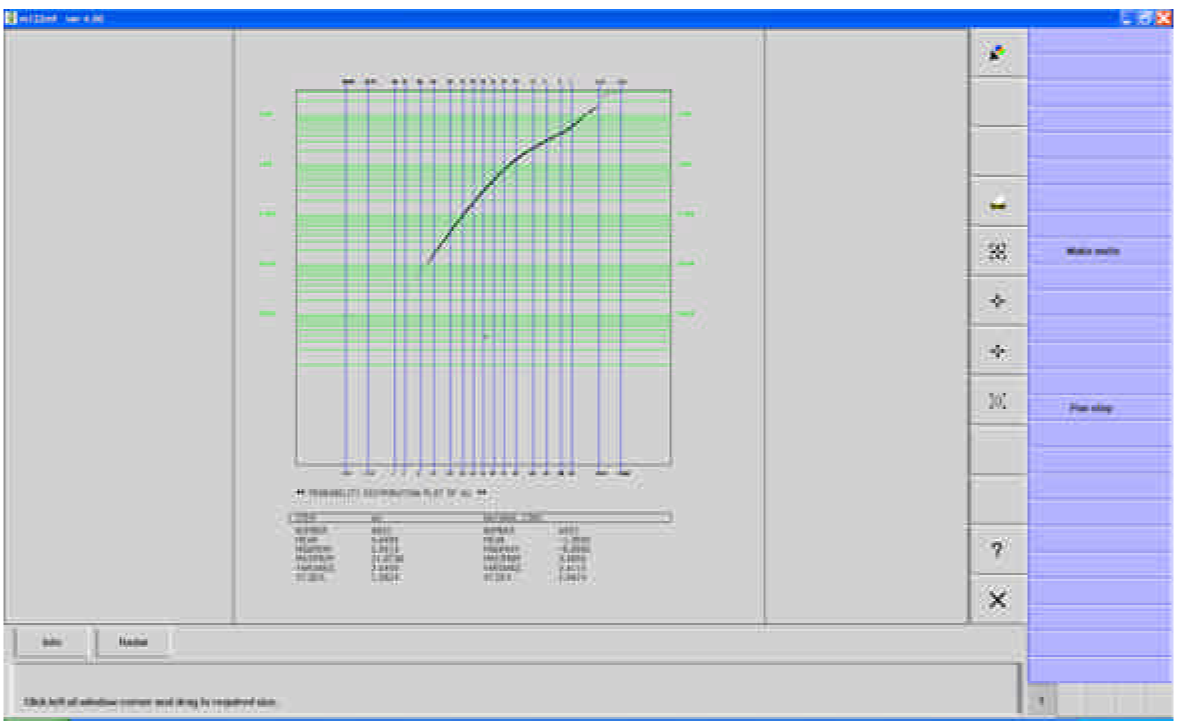

Figure 31: Probability Plot and Statistics for 3 meter Au Composites.

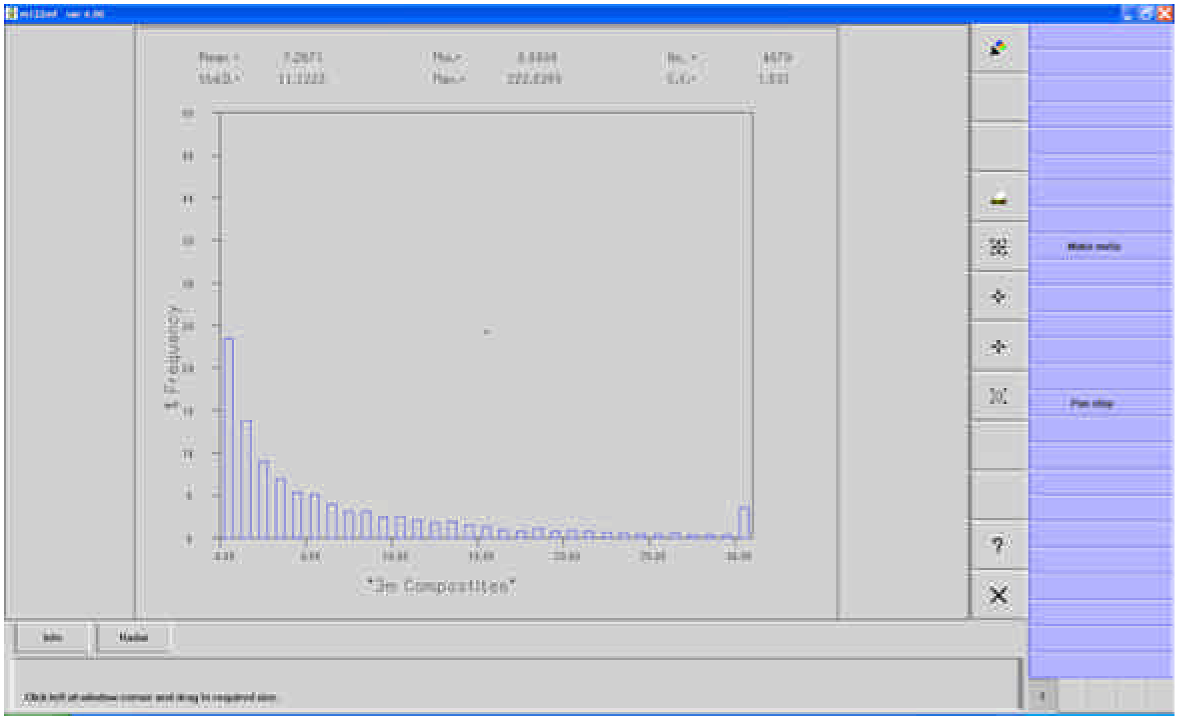

Figure 32: Histogram and Basic Statistics of 3 meter Ag Composites

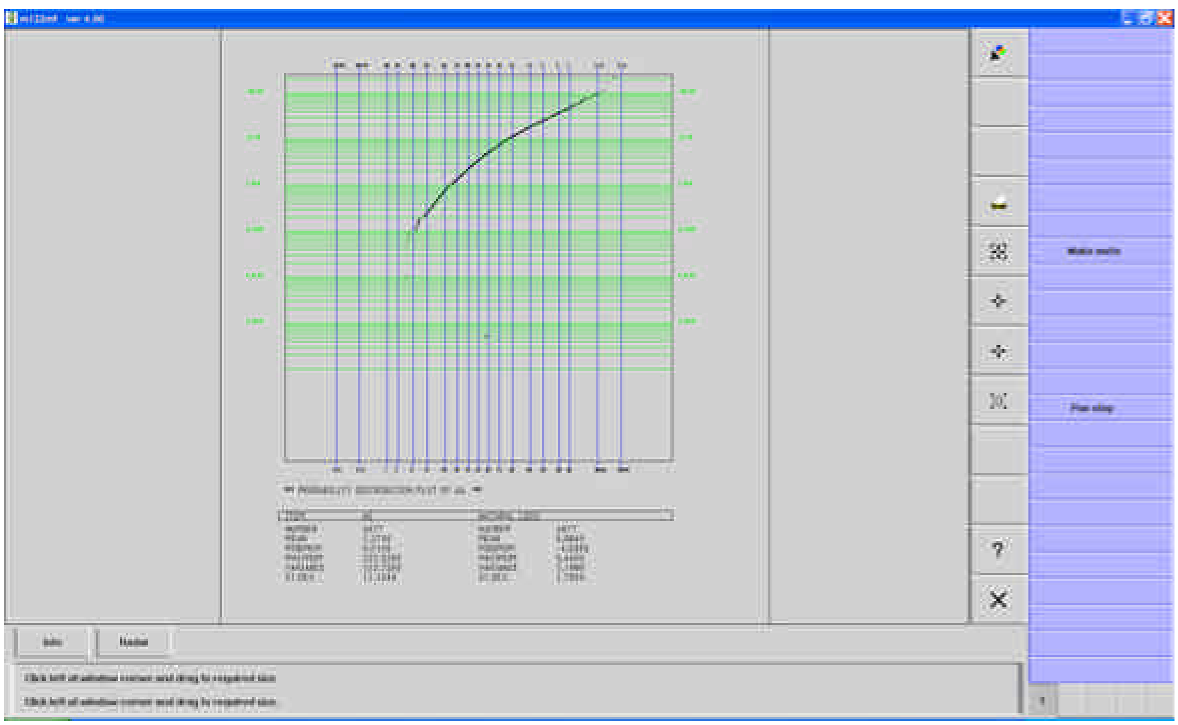

Figure 33: Probability Plot and Statistics for 3 meter Au Composites.

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 cut-off grades as shown in Figures 34 and 35, 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 lose 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.

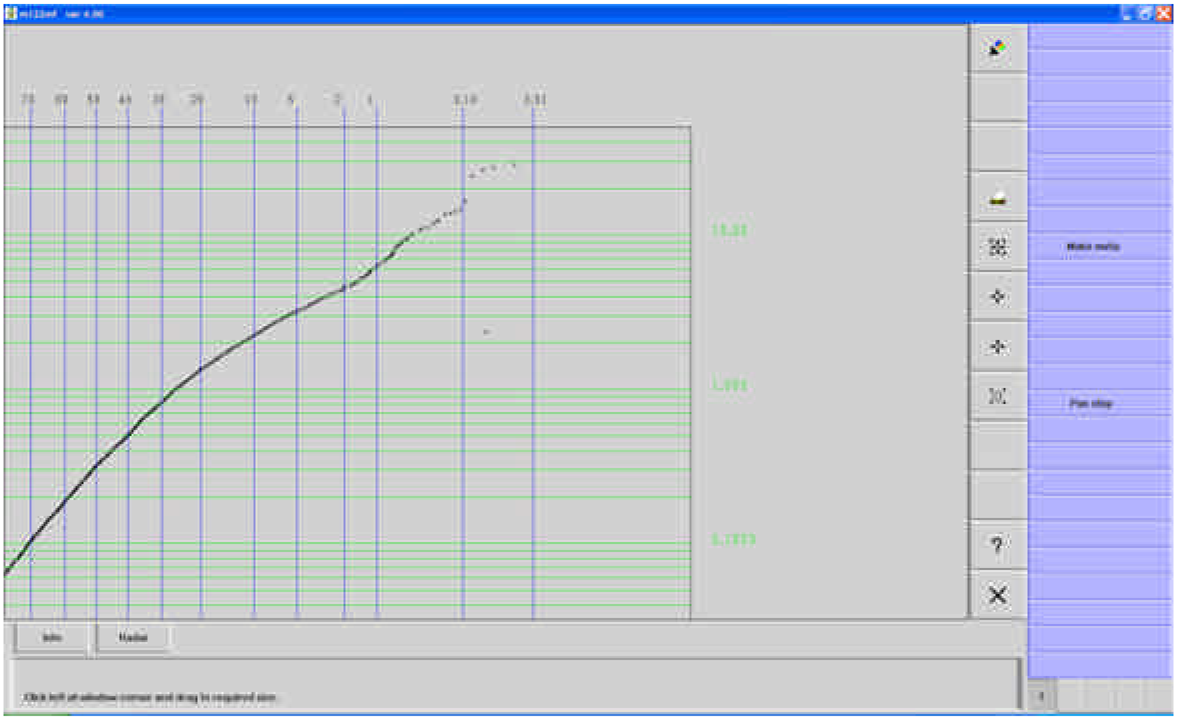

Figure 34: Zoom of Au Probability Plot Illustrating Break between 10 and 20 gpt.

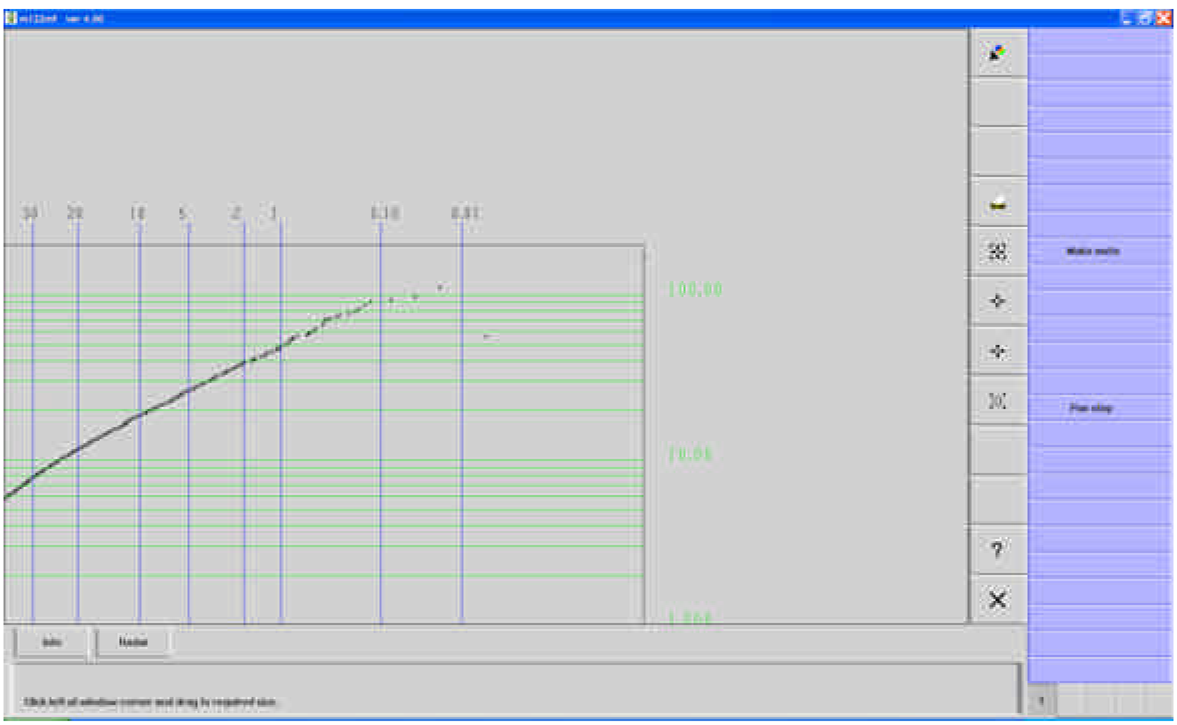

Figure 35: Zoom of Ag Probability Plot Illustrating Break between 90 and 100 gpt.

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. The semi-variograms generated from this analysis are attached to this report in Appendix 4.

An 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 36 and 37, 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

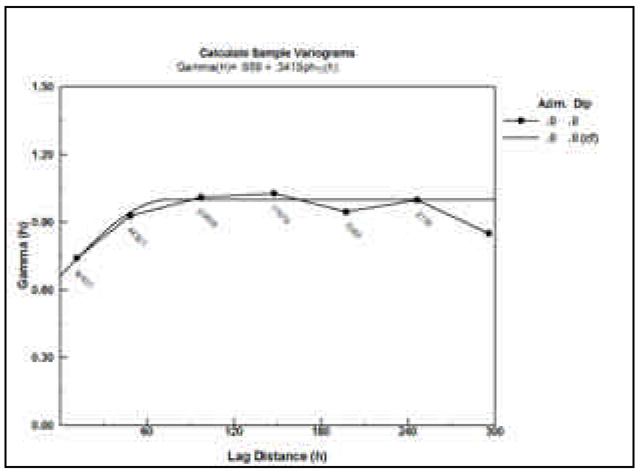

Figure 36: Downhole correlogram for Au.

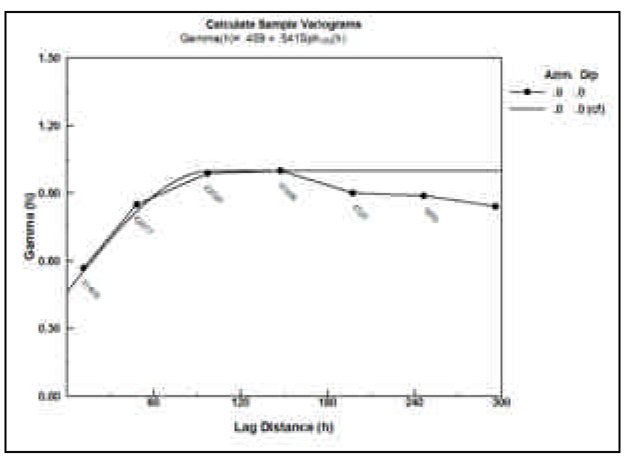

Figure 37: Downhole correlogram for Ag.

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) in addition to the oxide and sulfide 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 51m, 36m and 18m.

In preparation for the implementation of the grade estimation method as described in Section 17.11, variograms (see Appendix 4: Explanation of Semivariograms) 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. The correlograms are shown in Appendix, with their corresponding model. Note that the sill of the variograms has been standardized to one, and therefore they are in fact relative variograms.

Figures 38 shows the summary of the correlogram model used to guide the estimate the Kremnica resources. In this Figure, the rotation of the angles are given according to the convention used by GSLIB in MineSight Compass, see the SAGE2001 documentation for more details.

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 38: Summary Description of the Au 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

Figure 39: Summary Description of the Ag Variogram Model

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 decretitize 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 following limits:

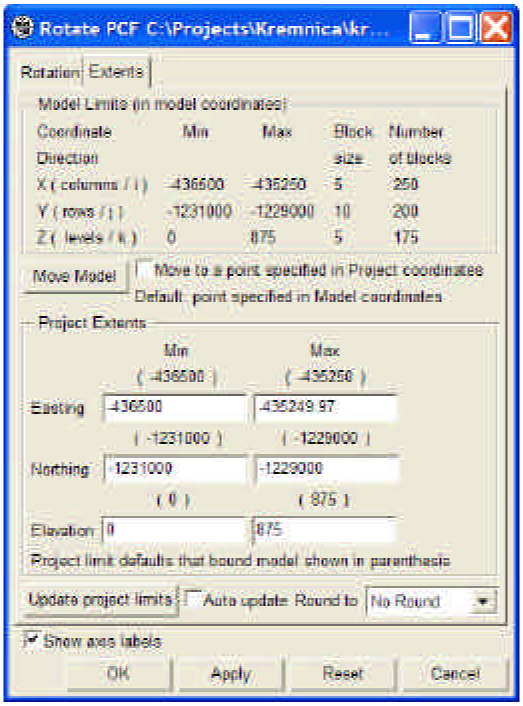

Figure 38: Block Model Limits. Note that coordinates are listed in negative northings and eastings (i.e. westings and southings).

- 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 ID^3 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 10th 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 Resour ce 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 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 done using being 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.

Pass 1 - 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.

Pass 2 - 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 3 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 19
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	51x36mx18m	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.

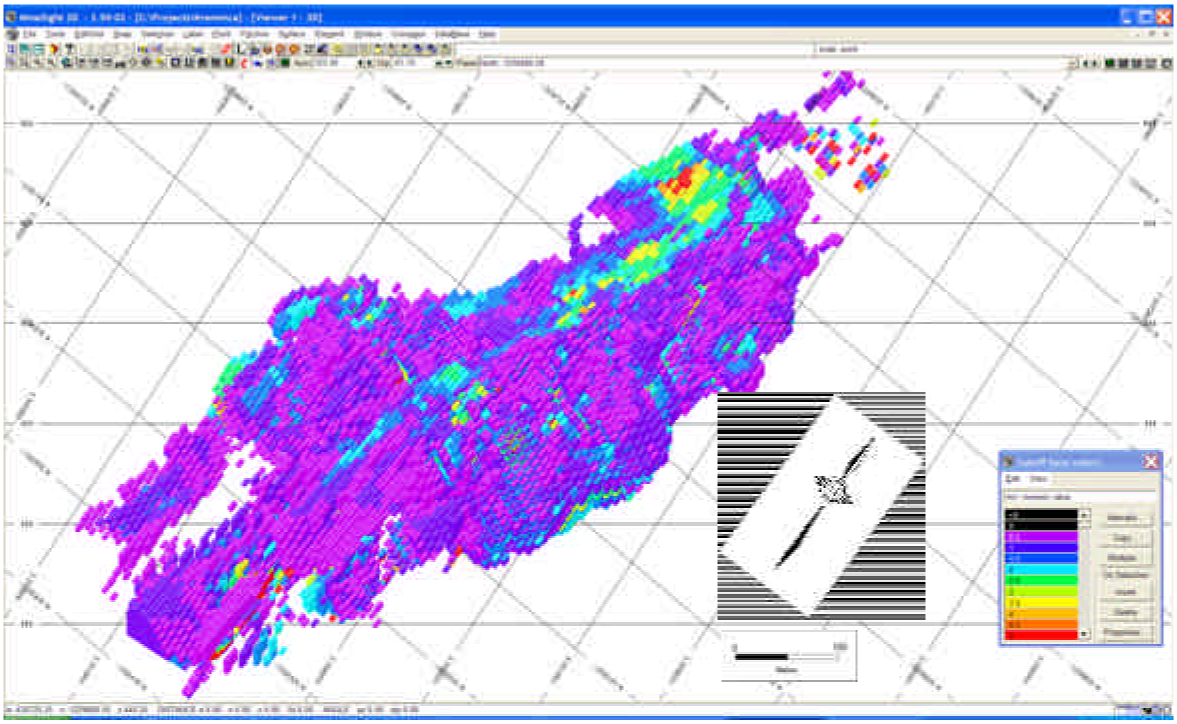

Figure 40: Three-dimensional view of block model, Kremnica Deposit with estimated Au grade = 0.75 g/t.

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.

The classification of resources derived for the Kremnica resources is compliant with NI 43-101 classification systems (see Appendix). 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.

The resources are classified on a block-by-block basis, using the defined estimation passes as shown in Section 17.13. This indicates the quantity and quality of information used to estimate each block.

Figure 40 illustrates a plan view of the block model displaying blocks at a cut-off grade of 0.75 g/t with a cut through the solids geology units. Vein material is a solid purple line while the andesite / stockwork boundary is in green solid line. Also, voids are in dashed grey lines. A sample cross section is shown in Figure 41. In addition, Figures 42 through 44 shows three-dimensional views of the Measured, Indicated, and Inferred resources, respectively.

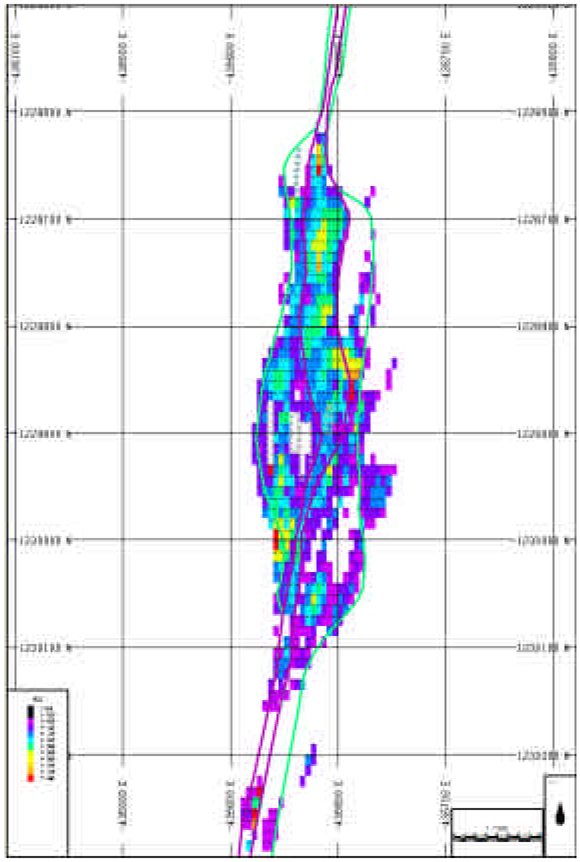

Figure 40: Plan View of Block Model, Kremnica Deposit with Au grade = 0.75 g/t and geologic units with the vein in solid purple, andesite / stockwork boundary in green and voids in dashed grey.

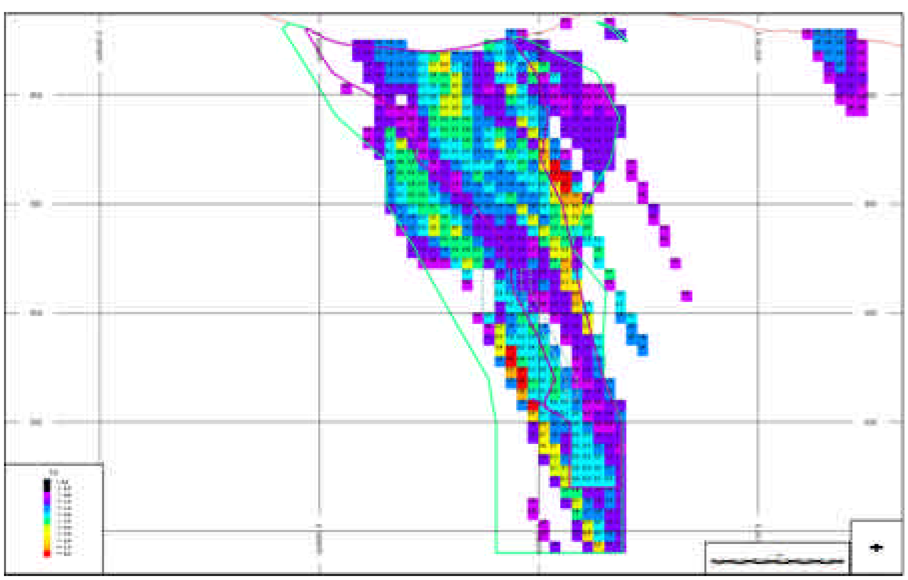

Figure 41: Cross-sectional view of block model with estimated Au grade = 0.75 g/t and geologic units with the vein in solid purple, andesite / stockwork boundary in green and voids in dashed grey.

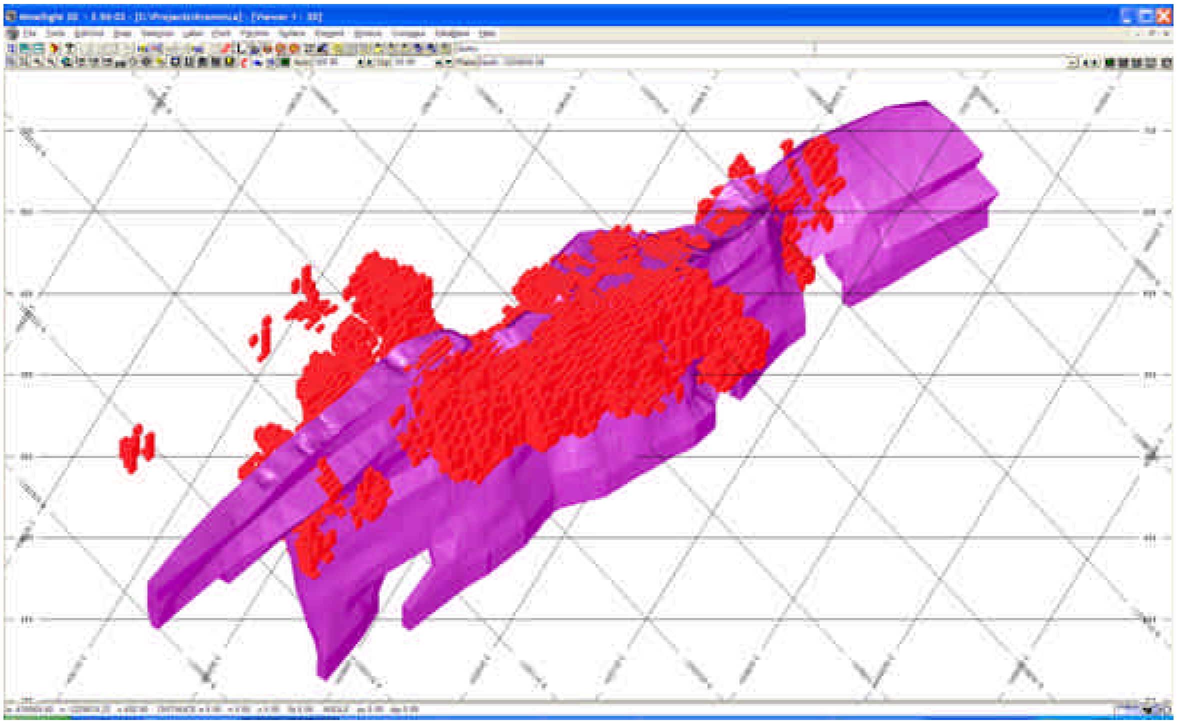

Figure 42: Three-Dimensional View, Measured Resources.

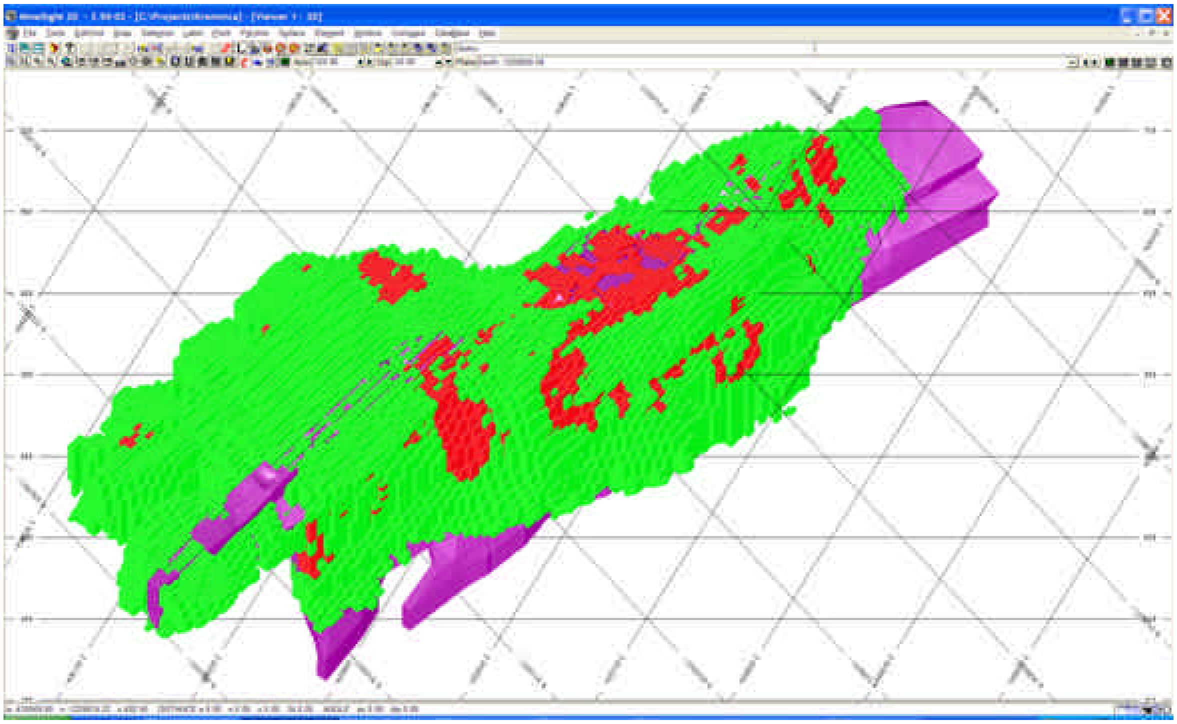

Figure 43: Three-Dimensional View, Measured and Indicated Resources.

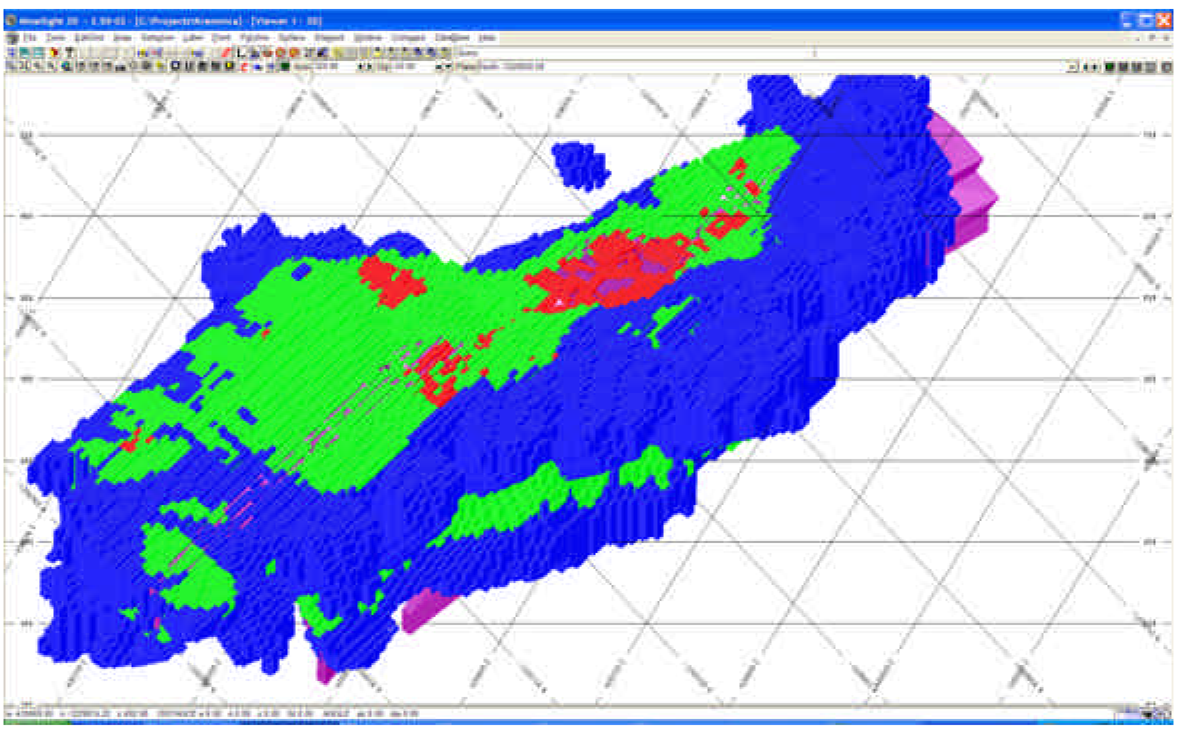

Figure 44: Three-Dimensional View, Measured, Indicated and Inferred Resources.

17.14 Resource Reporting

Based on the above classification and using a 0.75 g/t AuEq cut-off (the AuEq is based upon 66.7:1 Au:Ag. The 0.75 g/t AuEq is based upon the work completed in the Preliminary Assessment, dated February, 2004 which indicated that this cut-off grade is a sufficient threshold for reasonable expectation of extraction) the estimated resources for the Kremnica Deposit contains a measured and indicated resource of 958,000 ounces of gold and 7,729,300 ounces of silver from 18,807,000 tonnes at an average grade of 1.58 g/t gold and 12.78 g/t silver, respectively. An additional inferred resource of 271,500 ounces of gold and 1,526,300 ounces of silver from 6,398,000 tonnes at an average grade of 1.32 g/t gold and 7.42 g/t silver, respectively.

Table 20: Measured Resources

	MEASURED						
Cut-off Grade gAu/t	Tonnes	Au g/t	Ag g/t	AuEq g/t	Au Ounces	Ag Ounces	Equivalent
0.5	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	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	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	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	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	97,307	6.33	33.99	6.84	19,813	106,338	21,405

Note: * Base Case
AuEq is based upon 66.7:1 Au:Ag

Table 21: Indicated Resources

	INDICATED						
Cut-off Grade gAu/t	Tonnes	Au g/t	Ag g/t	AuEq g/t	Au Ounces	Ag Ounces	Equivalent
0.5	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	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	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	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	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	146,616	6.25	25.17	6.63	29,471	118,628	31,248

Note: * Base Case
AuEq is based upon 66.7:1 Au:Ag

Note: The above resources have been prepared in accordance with CIM standards.

Table 22: Measured and Indicated Resources

MEASURED + INDICATED							
Cut-off Grade g/t Au	Tonnes	Au g/t	Ag g/t	AuEq g/t	Au Ounces	Ag Ounces	Au Equivalent Ounces
0.5	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**	**958,773**	**7,727,907**	**1,074,242**
1	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	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	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	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	243,923	6.28	28.69	6.71	49,284	224,966	52,653

Note: * Base Case.
AuEq is based upon 66.7:1 Au:Ag

Table 23: Inferred Resources

INFERRED							
Cut-off Grade g/t Au	Tonnes	Au g/t	Ag g/t	AuEq g/t	Au Ounces	Ag Ounces	Equivalent
0.5	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	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	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	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	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	27,332	6.16	17.17	6.42	5,417	15,084	5,642

Note: * Base Case.
AuEq is based upon 66.7:1 Au:Ag

17.15 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 22 and 23 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 estimated could be somewhat conservative. In addition, volumes as a percentage of the total in relation to composites as a function of the total (i.e. by approximately 15%) also indicate a conservative estimation approach. This would also suggest that internal dilution has been included and that the model may be considered diluted.

Table 24
Au Composite vs. Block Model Statistics by Cut-off Grade.

Cut-off Grade gAu/t	COMPOSITES					BLOCK MODEL				
	Weight	%	AU	SD	CV	Weight	%	AU	SD	CV
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	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	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	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	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	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	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	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	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	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	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	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	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	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	6	0.13%	25.87	5.22	0.20	5,187	0.01%	18.68	1.23	0.07

Table 25
Ag Composite vs. Block Model Statistics by Cut-off Grade.

Cut-off	COMPOSITES					BLOCK MODEL				
Grade gAu/t	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

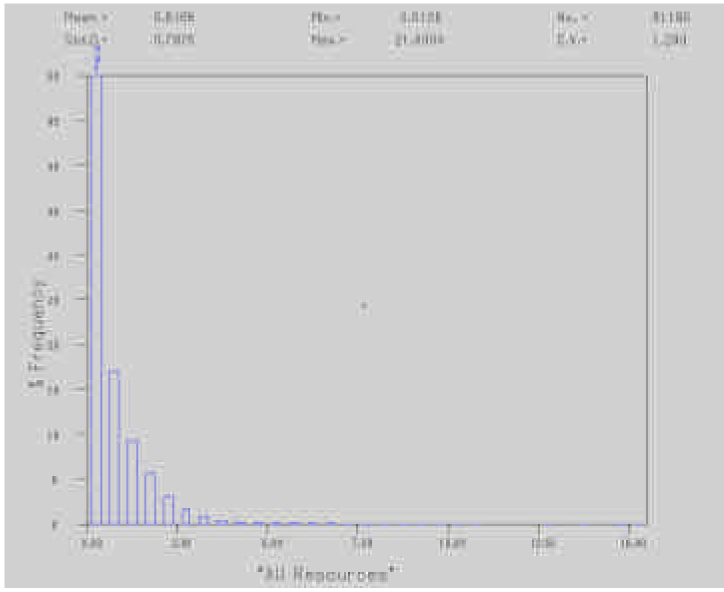

Figure 45 : Block Model Histogram and Basic Statistics for Au.

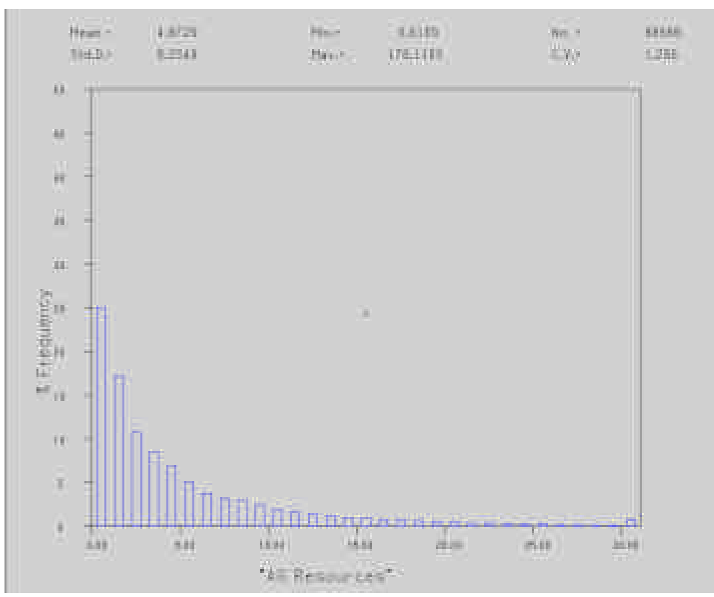

Figure 46 : Block Model Histogram and Basic Statistics for Ag.

In addition to the above, a graphical validation was done 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.

SECTION 18.0

OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information that the authors of this report are aware of that would make this report more understandable and not misleading.

SECTION 19.0

INTERPRETATION AND CONCLUSIONS

Gold-silver mineralization at Kremnica is part of a large low-sulfidation quartz-sericite-adularia epithermal-hydrothermal system hosted in Tertiary andesite volcanic flows and tuffs and lesser diorites and rhyolite dikes. Mineralization occurs in mainly in large banded to massive quartz veins, smaller quartz veins and sheeted veins, quartz stockwork veining, and silicified hydrothermal breccias.

Prior to completion of the infill drilling carried out in 2005 the Sturec Deposit within the Kremnica project had an Indicated Resource of 518,416 oz gold equivalent and an Inferred Resource of 773,951 oz gold equivalent, (assuming 0.5gAuEq/t cut-off, base upon 77:1 silver to gold ratio) contained within the Šturec deposit.

Based on the results of the infill drilling program carried out in 2005 and the associated programs the estimated revised mineral resources are shown in Table 24.

Table 26
Summary of Resources

Kremnica Gold Project
Sturec Deposit
Resource Estimate
8-Apr-06

Category	Cut-off gAu/t	Quantity Tonnes	Grade Au Equ.	Grade gAu/t	Grade gAg/t	Gold Equi. Ounces	Gold Ounces	Silver Ounces
Measured	0.75	7,293,000	1.96	1.75	14.24	459,600	410,300	3,338,900
Indicated	0.75	11,514,000	1.66	1.48	11.86	614,500	547,900	4,390,400
Measured and Indicated	**0.75**	**18,807,000**	**1.78**	**1.58**	**12.78**	**1,074,100**	**958,200**	**7,729,300**
Inferred	0.75	6,398,000	1.44	1.32	7.42	296,200	271,500	1,526,300

Note Tonnes rounded to nearest 1000 tonnes, ozs to the nearest 100ozs

Gold equivalent - silver gold ratio is 66.7:1

This resource is based upon a cut-off of 0.75 gAuEq/t (silver gold ratio is 66.7:1) which reflects the economic cut-off grade derived form the Preliminary Assessment report prepared by Beacon Hill dated February 15, 2004.

This resource estimate is compliant with National Instrument 43-101 regulations.

The Šturec deposit lies along a 5km long mineralized trend that includes several other prospective targets. North, along strike from the Šturec deposit, the Wolf and Vratislav (and Vladimir) targets have had small historical resources defined by previous operators, Wolf has an indicated resource 74,321 oz gold equivalent at a 0.5 g/t cut-off defined by Argosy Mining Corporation in 1998. Vratislav has an indicated resource of 53,152 oz defined by the Slovak Government in 1992. The Wolf and Vratislav (and Vladimir) resources are historical and are not NI 43-101 compliant.

The Šturec deposit, and the Wolf and Vratislav targets are all open to extension at depth and along strike, and it is reasonable to assume that further drill programs have the possibility to increase the size of the known mineralization. There is also good potential for discovery of new economic resources within other target areas, both within the Kremnica mining Lease and Lucky and Lutila exploration licenses.

Results of exploration work to date on the Kremnica Gold Project are indicative of a property of merit that is thus worthy of further development.

The 2005 exploration program and subsequent evaluation confirmed and enhanced the resource estimate for the Sturec deposit. Tournigan have commissioned a pre-feasibility Study to further evaluate the Kremnica Gold Project. This work is presently underway and a full report is scheduled to be completed by the end of the second quarter 2006.

SECTION 20.0

RECOMMENDATIONS

The results of the revised mineral resource estimate for the Sturec deposit at the Kremnica Gold Property clearly shows, based upon the infill drilling and associated exploration work completed by Tournigan, that the previously estimated resources are confirmed and enhanced.

Tournigan has already commissioned a pre-feasibility study which will incorporate the results of this report. It is recommended that this pre-feasibility study be completed forthwith and that the full feasibility study commence immediately and in conjunction with the ongoing pre-feasibility study such that various study activities run concurrently in order for the full feasibility study to be completed at the earliest opportunity.

The total estimated cost to complete these studies is $2,218,500. The breakdown of the estimate is shown below.

COMPLETION OF THE PRE-FEASIBILITY STUDY

COST ESTIMATE

Task Description		Estimated Cost
Environmental		**$75,000**
Socio Economic Study		**$50,000**
Public Consultation		**$50,000**
Geotechnical Site Work		**$25,000**
Mine Plan		
Project Management	$15,000	
Geology	$5,000	
Resource Estimate and Open Pit Optimization	$15,000	
Mine Planning	$34,500	
Metallurgy and Process Plant	$19,500	
Geotechnical	$79,500	
Surface Buildings	$25,000	
Power supply/Electrical distribution/Communication	$25,000	
Cost Estimates	$35,000	
Financial Analysis	$10,500	
Project Disbursements	$4,500	
Total Mine Plan		**268,500**
Sub-total		**468,500**
Contingency 15%		**$70,000**
Total		**$538,500**

KREMNICA GOLD PROJECT
FULL FEASIBILITY
COST ESTIMATE

SUMMARY

Task Description	Estimated Cost
Project Management	$50,000
Drilling (in-fill) allowance	$100,000
Geology	$10,000
Resource Estimate and Open Pit Optimization	$50,000
Mine Planning	$75,000
Metallurgy and Process Plant	$50,000
Environmental and EIA	$250,000
Socio Economic	$200,000
Public Consultation	$50,000
Geotechnical	$350,000
Surface Buildings & Infrastructure	$100,000
Power supply/Electrical distribution/Communication	$50,000
Cost Estimates	$75,000
Financial Analysis	$25,000
Project Disbursements	$25,000
Sub-total	$1,460,000
Contingency 15%	220,000
Total	$1,680,000

SECTION 21.0

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SECTION 22.0

ADDITIONAL 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.

QUALIFICATIONS

CERTIFICATE OF QUALIFICATION

I, W. Peter Stokes, of 206, 1248 Hunter Road, Tsawwassen Vancouver, British Columbia, V4L 1Y8, 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) This certificate applies to the "Revised Mineral Resource Estimate" on the Kremnica Gold Project, Kremnica, Slovak Republic , Europe dated May 11, 2006 prepared for Tournigan Gold Corporation, Vancouver, B.C.

3) I am a graduate of the Stoke-on–Trente College of Technology, Stafforshire, UK in 1965 with an HND in Mine Engineering. I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia. I have practiced my profession as a mining engineer throughout the world continuously since 1965.

4) I have visited the property on two occasions. The first was October 21 and 22, 2003 for a total of two days, the second visit was from October 24 to 28, 2005 for a total of 5 days.

5) In the Independent "Revised Mineral Resource Estimate" report on the Kremnica Gold Project, I am responsible for Sections 1.0, 2.0, 19.0 and 20.0 and the overall study management and study compilation.

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 National Policy 43-101.

7) I have reviewed the property on two previous occasions. In 1998, I completed a conceptual review of the property for Argosy Mining Corp., a previous owner of the property. I also completed the report "Preliminary Assessment", Kremnica Gold Project in February, 2004 and an addendum to that report "Higher Grade Pit" dated March 30, 2004. I am presently working on the pre-feasibility report for the 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 "Revised Mineral Resource Estimate" report.

9) I have read National Instrument 43-101, Standards for Disclosure of Mineral Properties. This technical report has been prepared in compliance with 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.

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

Dated this 11th day of May , 2006

"W. Peter Stokes"

Original Signed by W. Peter Stokes, P.Eng.

CERTIFICATE OF QUALIFICATION

I, Carl von Einsiedel, 8888 Shook Road, Mission, British Columbia, V2V-7N1, do hereby certify that:

12) I am a consulting geologist with an office at 1124-470 Granville Street, Vancouver, British Columbia, V6C 1V5

13) This certificate applies to the "Revised Mineral Resource Estimate" on The Kremnica Gold Project, Kremnica, Slovak Republic, Europe dated May 11, 2006 prepared for Tournigan Gold Corporation, Vancouver, B.C.

14) I am a graduate of Carleton University in Ottawa, Ontario, Canada in 1987 with a BSc. in Geology. I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia. I have practiced my profession as a geologist throughout the world continuously since 1987.

15) I visited the Kremnica Property from July 5 to July 11, 2005 for a total of six days.

16) In the Independent "Revised Mineral Resource Estimate" report on the Kremnica Gold Project", I am responsible for sections 3.0, 4.0, 5.0, 6.0, 7.0, 8.0, 9.0, 10.0, 11.0, 12.0, 13.0, 14.0, 15.0, 16.0, 21.0 and 22.0 of the report.

17) 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 National Policy 43-101.

18) I have had no prior involvement with the Property that is the subject of this report.

19) 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.

20) I have read National Instrument 43-101, Standards for Disclosure of Mineral Properties. This Technical Report has been prepared in compliance with National Instrument 43-101.

21) 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.

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

Dated this 11th day of May, 2006

"Carl von Einsiedel"

Original signed by Carl von Einsiedel, P.Geo.

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) This certificate applies to the "Revised Mineral Resource Estimate on The Kremnica Gold Project, Kremnica, Slovak Republic, Europe dated May 11, 2006 prepared for Tournigan Gold Corporation, Vancouver, B.C.

3) 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 and Geoscientists of the Province of Alberta. I have continuously practiced my profession performing computer modelling since 1988, both as an employee of a geostatistical modelling and mine planning software and consulting company and as an independent consultant.

4) I have visited the property on October 24 to 28, 2005 for a total of 5 days.

5) In the independent report titled "Revised Mineral Resource Estimate on The Kremnica Gold Project", I am responsible for the "Resource Estimation" which is Section 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 reviewed the property on one previous occasion. I was a member of the project team that completed the NI 43-101 resource estimate along with the Preliminary Assessment, Kremnica Gold Project in February, 2004. I am presently a member of the project team that is working on the preparation of the pre-feasibility report on the 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.

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

Dated this 11th day of May, 2006.

"Garth D. Kirkham"

Original signed by Garth D. Kirkham, P.Geoph.

APPENDICES

APPENDIX 1

DRILLHOLE, UNDERGROUND DRIFT AND BENCH SAMPLE LISTING

HOLE	EAST	NORTH	ELEVATION	AZIMUTH	DIP	LENGTH
AS-1	-435800.3	-1230205.1	720.1	273	-60	200.0
AS-10	-435686.8	-1229409.8	649.7	275	-46	139.0
AS-101	-435814.6	-1230308.3	724.5	280	-46	233.0
AS-103	-435857.5	-1230436.3	745.1	272	-46	249.0
AS-106	-435896.8	-1230518.6	760.9	281	-45	194.0
AS-107	-435862.9	-1230503.3	754.8	283	-44	152.0
AS-11	-435626.8	-1229317.9	650.1	274	-46	150.7
AS-110	-435897.3	-1230602.4	761.5	280	-45	152.0
AS-112	-435935.3	-1230696.0	767.9	93	-44	150.0
AS-115	-435902.1	-1230790.0	756.2	283	-48	152.0
AS-118	-436118.6	-1230109.8	753.1	87	-48	227.0
AS-12	-435603.8	-1229182.6	656.1	273	-66	135.5
AS-122	-436035.4	-1230062.8	758.7	90	-45	63.5
AS-122A	-436035.4	-1230062.8	758.7	90	-45	138.6
AS-123	-435965.6	-1230052.3	746.5	320	-45	113.8
AS-124	-435891.3	-1229909.6	682.0	260	-45	100.0
AS-125	-436016.0	-1230128.3	800.5	90	-45	184.2
AS-126	-436018.4	-1230127.8	800.5	90	-85	144.0
AS-127	-436010.0	-1230195.3	790.4	90	-45	130.4
AS-128	-436012.0	-1230195.1	790.5	101	-89	141.0
AS-129	-436007.6	-1230286.6	784.4	90	-45	236.5
AS-13	-435580.3	-1229108.1	656.3	273	-65	113.6
AS-130	-436009.8	-1230286.5	784.3	0	-90	150.4
AS-134	-435502.4	-1228088.1	792.9	277	-60	112.1
AS-135	-435408.5	-1228096.8	781.9	277	-45	204.1
AS-136	-435329.4	-1227948.0	769.0	277	-60	250.0
AS-137	-435409.1	-1227935.6	784.7	277	-60	58.2
AS-141	-436009.3	-1230388.4	785.5	90	-45	219.1
AS-141A	-436016.2	-1230387.4	785.8	0	-90	168.8
AS-142	-435329.3	-1227948.0	770.2	277	-45	50.0
AS-143	-435406.0	-1227935.6	783.8	277	-70	100.0
AS-144	-435550.0	-1228088.1	809.0	277	-60	100.0
AS-145	-436018.4	-1230126.9	800.7	300	-55	120.0
AS-146	-436038.1	-1230062.6	758.9	0	-90	104.8
AS-147	-436041.7	-1230064.9	758.8	155	-45	101.0
AS-148	-436021.2	-1230387.4	784.2	270	-60	99.2
AS-149	-436009.0	-1230288.8	784.2	270	-60	102.7
AS-150	-435352.3	-1228065.8	759.2	270	-60	153.9

HOLE	EAST	NORTH	ELEVATION	AZIMUTH	DIP	LENGTH
AS-151	-436011.5	-1230195.1	790.1	263	-60	106.2
AS-2	-435783.4	-1230099.1	711.4	279	-49	120.0
AS-2.1.A	-435783.3	-1230097.9	711.4	282	-53	200.0
AS-3.1.A	-435786.9	-1230006.1	701.4	274	-53	165.0
AS-3.1.B	-435785.7	-1230006.1	701.4	278	-76	66.0
AS-3.2	-435727.5	-1230008.8	693.5	270	-48	237.0
AS-4	-435803.7	-1229853.5	675.4	273	-50	43.0
AS-4.1.1	-435870.6	-1229895.6	682.4	0	-90	51.0
AS-4.1.A	-435813.3	-1229894.4	682.8	273	-50	68.0
AS-4.1.B	-435814.8	-1229894.4	682.8	282	-79	86.0
AS-4.1.C	-435812.3	-1229893.5	682.8	93	-49	120.0
AS-4.2	-435662.2	-1229902.3	678.6	277	-45	277.8
AS-4.5.1.A	-435864.1	-1229852.3	671.3	282	-49	182.0
AS-4.5.1.B	-435862.7	-1229852.3	671.3	0	-90	46.0
AS-4.5.2	-435670.0	-1229850.0	677.0	277	-45	250.1
AS-4.D	-435802.2	-1229853.5	675.4	0	-90	116.0
AS-5	-435712.3	-1229796.3	680.4	273	-60	250.0
AS-5.1.1.A	-435830.0	-1229798.8	653.1	0	-90	56.0
AS-5.1.1.B	-435849.9	-1229799.1	653.0	273	-50	22.5
AS-5.1.A	-435764.5	-1229800.6	681.8	281	-54	92.0
AS-5.1.B	-435763.6	-1229800.8	681.9	91	-50	126.5
AS-5.2	-435609.7	-1229796.8	668.4	277	-45	213.0
AS-5.3	-435661.0	-1229800.0	674.0	277	-55	281.4
AS-6	-435696.0	-1229700.9	674.3	283	-55	173.0
AS-6.1.A	-435742.3	-1229676.4	671.1	281	-49	179.0
AS-6.2	-435673.7	-1229694.1	668.1	277	-55	279.9
AS-7	-435695.3	-1229900.9	683.2	271	-50	182.0
AS-8	-435744.7	-1229602.1	670.9	273	-50	144.5
AS-8.1.B	-435784.2	-1229598.8	671.6	279	-50	120.5
AS-8.2	-435686.2	-1229609.9	666.7	277	-50	288.4
AS-9.1.A	-435779.7	-1229499.8	658.8	282	-50	150.0
AS-9.1.B	-435778.5	-1229499.8	658.8	275	-84	140.5
KAT-1	-436264.4	-1229633.3	777.9	87	-46	305.0
KAT-2	-436221.8	-1229682.4	779.5	119	-45	302.0
KAT-7	-436070.1	-1229690.9	763.2	116	-50	309.8
KAT-8	-436070.9	-1229689.8	763.4	116	-45	60.0
KAT-9	-436048.3	-1229575.5	744.0	83	-45	333.7
STV-3	-435798.4	-1229903.1	654.5	298	-28	20.0
STV-1	-435837.4	-1229995.3	656.1	301	-30	8.0
STV-11	-435843.9	-1230048.8	656.2	297	35	45.0
STV-12	-435845.8	-1230100.1	657.9	270	35	45.0
STV-2	-435838.3	-1229994.8	658.1	301	29	44.5
STV2A	-435827.0	-1229977.1	656.6	300	-12	35.0
STV-2B	-435826.5	-1229977.4	658.0	300	30	33.0
STV-3A	-435846.1	-1230021.3	657.3	261	25	29.3
STV-3B	-435845.3	-1230021.3	658.4	261	-12	43.0
STV-4	-435846.9	-1230048.3	656.9	297	-12	34.0
STV-5	-435848.8	-1230099.9	656.9	270	-12	72.0

HOLE	EAST	NORTH	ELEVATION	AZIMUTH	DIP	LENGTH
STV-6	-435861.5	-1230074.9	656.6	301	30	15.5
V-1	-435603.8	-1228819.3	682.7	118	0	40.0
V-10	-435523.1	-1228590.1	683.9	104	-3	100.0
V-10A	-435523.6	-1228590.0	683.2	102	-25	96.8
V-11	-435542.5	-1228570.3	683.8	328	-1	60.0
V-11A	-435542.3	-1228570.5	682.8	326	-37	16.0
V-12	-435729.4	-1228832.9	683.0	313	0	50.0
V-13	-435453.6	-1228450.0	685.9	337	0	110.0
V-14	-435753.0	-1228941.9	684.5	291	0	80.0
V-15	-435471.3	-1228512.0	684.7	313	0	71.0
V-15A	-435470.2	-1228512.8	683.3	315	-45	45.0
V-16	-435752.1	-1228940.5	684.7	316	0	120.0
V-17	-435377.6	-1228502.4	685.8	156	0	54.0
V-1A	-435603.7	-1228819.1	681.7	116	-39	135.0
V-1B	-435603.7	-1228819.3	681.4	117	-26	55.0
V-2	-435609.7	-1228816.1	682.7	312	0	120.0
V-2A	-435609.4	-1228816.3	681.5	312	-50	70.0
V-3	-435509.8	-1228890.0	681.9	120	-15	80.0
V-4	-435646.1	-1228915.3	683.0	109	0	50.0
V-4A	-435656.2	-1228915.3	682.0	109	-30	70.0
V-5	-435546.0	-1228685.0	682.7	93	0	60.0
V-5A	-435545.8	-1228685.1	681.7	93	-35	120.5
V-6	-435568.7	-1228685.3	682.7	323	0	64.0
V-6A	-435568.3	-1228685.5	681.5	324	-45	100.0
V-7	-435646.8	-1228970.4	682.1	135	-25	81.0
V-8	-435681.2	-1228655.3	683.2	317	1	130.0
V-9	-435652.7	-1228912.1	683.1	291	0	100.0
V-9A	-435652.2	-1228913.4	681.9	288	-35	90.0
VKB-2	-435848.4	-1229590.4	623.7	105	3	50.0
VKB-2A	-435848.7	-1229590.3	622.5	99	-45	70.0
VKB-2B	-435848.5	-1229590.3	624.9	98	44	50.0
VKB-3	-435862.8	-1229688.3	624.6	120	3	40.5
VKB-3R	-435862.8	-1229688.3	624.6	120	0	46.0
VKB-4	-435830.4	-1229492.3	623.9	110	0	50.0
VKB-4A	-435830.2	-1229492.0	622.7	110	-40	70.0
VKB-4B	-435830.4	-1229492.1	625.5	110	41	50.0
VKB-5	-435768.4	-1229406.8	625.5	92	2	47.0
VKB-5B	-435768.3	-1229406.8	626.8	90	38	33.0
VKB-7	-435868.3	-1229743.4	624.3	127	-25	53.0
AS-152	-435356.7	-1227941.4	775.7	270	-60	97.0
AS-153	-435293.8	-1227960.4	764.8	270	-60	150.0
AS-154	-435439.3	-1228011.6	782.5	277	-45	117.0
AS-155	-435382.2	-1228022.9	778.7	270	-45	217.0
KV-1	-434953.8	-1227509.3	733.7	183	-86	785.0
KV-1-S	-434953.0	-1227516.0	733.7	263	-61	650.0
KV-13	-436794.6	-1232527.3	681.2	90	0	1.0
KV-14	-434942.2	-1226732.6	826.5	90	0	1200.0
KV-15	-434781.0	-1228426.0	683.1	90	0	918.0

HOLE	EAST	NORTH	ELEVATION	AZIMUTH	DIP	LENGTH
KV-18	-435898.2	-1230517.9	759.5	0	-90	650.0
KV-19	-435715.4	-1226492.4	772.0	90	0	650.0
KV-2	-434625.3	-1227108.9	808.1	250	-58	1200.0
KV-2-S	-434620.7	-1227111.9	808.7	337	-85	850.0
KV-3	-434345.8	-1227624.4	844.0	272	-60	1200.0
KV-3-S	-434345.8	-1227624.4	844.5	90	0	1200.0
KV-4	-434679.5	-1228009.0	783.3	90	0	1200.0
KV-5	-435672.2	-1226921.8	795.4	90	0	800.0
KV-6	-436697.1	-1229408.9	856.4	0	-90	806.0
KVS-10-A	-437101.4	-1230919.8	698.6	90	-60	378.0
KVS-10-B	-437101.9	-1230919.8	698.6	270	-60	548.0
KVS-11-A	-436318.1	-1229770.5	763.9	90	-60	620.0
KVS-12	-435234.4	-1226838.5	791.7	270	-60	650.0
KVS-16	-435556.2	-1229929.0	663.7	270	-75	650.0
KVS-17	-435753.1	-1230343.5	716.2	270	-75	650.0
KVS-20	-434886.9	-1226875.6	796.9	264	-62	650.0
KVS-21	-434799.3	-1227255.9	764.9	279	-59	650.0
KVS-22	-434902.7	-1227708.6	729.4	279	-59	650.0
KVS-23	-435276.2	-1226622.4	791.2	281	-62	650.0
KVS-24	-435024.3	-1229056.8	706.5	270	-75	650.0
KVS-25	-434592.4	-1228885.4	658.9	293	-65	484.0
KVS-26	-435242.7	-1226381.6	796.4	286	-64	650.0
KVS-27	-435642.3	-1227181.0	794.0	276	-62	632.0
KVS-28	-435477.7	-1229939.0	653.0	0	-90	1200.0
KVS-4	-434675.1	-1228004.0	784.0	270	-60	1201.0
KVS-7-A	-437488.1	-1231636.3	654.5	310	-80	390.0
KVS-7-B	-437488.9	-1231635.5	654.6	310	-60	496.0
KVS-8-A	-437236.1	-1231431.8	676.6	310	-80	597.0
KVS-9-A	-437168.1	-1230382.3	690.3	270	-66	548.0
KVS-9-B	-437169.0	-1230386.5	690.0	90	-60	550.0
V-18	-435384.3	-1228557.8	686.5	121	0	60.0
VKB-1	-435879.4	-1229534.6	623.3	270	3	150.0
VKB-5A	-435767.8	-1229406.8	624.3	92	-47	60.0
VKB-6	-435728.8	-1229318.3	626.5	96	0	50.0
VKB-6A	-435728.8	-1229318.3	625.8	97	-50	58.0
VKB-6B	-435729.8	-1229318.3	627.8	97	40	35.0
VKB-8	-435684.6	-1229228.9	627.7	97	0	60.0
VKB-8A	-435684.7	-1229228.9	626.7	96	-45	60.0
VKB-8B	-435684.8	-1229229.0	629.7	97	50	30.0
VKB-9	-435700.0	-1229226.6	627.3	100	0	70.0
F1	-435854.9	-1229820.6	623.0	79	0	32.6
F-2	-435777.0	-1229913.0	623.0	261	0	112.5
F-3	-435865.4	-1229922.3	623.0	117	0	80.2
M	-435844.4	-1230118.9	708.0	225	0	207.7
O	-435751.5	-1229937.4	656.0	260	0	48.7
P-1	-435800.0	-1229935.4	652.0	288	0	122.2
P10	-435861.2	-1230181.1	708.0	221	0	95.6
P11	-435862.6	-1230230.0	708.0	229	0	92.2

HOLE	EAST	NORTH	ELEVATION	AZIMUTH	DIP	LENGTH
P11S	-435894.0	-1230254.4	708.0	124	0	8.9
P12	-435864.6	-1230280.3	708.0	236	0	93.7
P-2	-435803.0	-1229984.0	652.0	276	0	115.9
P-3	-435820.1	-1230018.8	656.0	225	0	112.8
P-4	-435844.2	-1230071.3	656.0	241	0	126.1
P5	-435852.1	-1230120.4	657.5	212	0	107.7
P6	-435852.2	-1230179.0	657.7	212	0	97.3
P7	-435855.9	-1230233.0	658.1	229	0	66.2
P8	-435856.5	-1230281.0	658.1	229	0	119.4
P-9	-435855.1	-1230124.4	707.8	227	0	269.0
PP2N	-435868.8	-1229928.1	656.0	116	0	65.5
PP2S	-435869.7	-1229929.8	656.0	118	0	65.3
PP3CN	-435869.7	-1229972.3	656.0	119	0	18.1
PP3CS	-435870.7	-1229974.1	656.0	106	0	18.0
PP3N	-435832.6	-1229995.4	656.0	261	0	7.4
PP3S	-435834.1	-1229998.3	656.0	289	0	8.0
PP4A	-435845.0	-1230073.5	656.0	242	0	17.0
PP4CN	-435884.8	-1230025.3	656.0	118	0	21.8
PP4N	-435842.8	-1230048.3	656.0	286	0	18.1
PP4NS	-435885.3	-1230027.3	656.0	87	0	21.0
PP4S	-435843.8	-1230049.8	656.0	300	0	17.9
PP5N	-435862.9	-1230098.0	656.0	274	0	12.1
PP5S	-435847.7	-1230100.9	656.0	282	0	92.7
S	-435780.9	-1229947.6	656.0	229	0	388.1
SP10	-435925.4	-1230185.1	708.0	135	0	50.5
SP10V	-435928.7	-1230171.4	708.0	270	0	19.0
SP9	-435948.8	-1230123.0	708.0	0	0	74.3
SP9A	-435878.4	-1230136.9	708.0	308	0	44.8
SP9A2	-435874.3	-1230120.5	708.0	126	0	21.0
SP9A3	-435887.1	-1230077.6	708.0	76	0	12.0
STPORT	-435874.1	-1229807.8	656.0	182	0	54.7
PP-1	-435881.3	-1229858.8	656.0	135	0	315.9
A	-435823.6	-1229828.6	656.0	228	0	70.0
B	-435823.0	-1229839.8	662.5	246	0	73.0
C	-435832.3	-1229852.4	672.0	281	0	50.0
D	-435829.1	-1229884.5	678.0	205	0	61.0
E	-435830.6	-1229907.4	683.0	257	0	37.0
F	-435864.3	-1229906.5	683.5	320	0	12.0
G	-435876.2	-1229910.5	685.0	231	0	2.0
H	-435859.7	-1229914.6	687.5	303	0	7.0
I	-435870.5	-1229909.3	688.5	213	0	5.0
KGST-10R	-435839.6	-1229864.0	674.1	0	-90	136.0
KGST-11R	-435741.2	-1229751.6	681.0	270	-60	142.0
KGST-12R	-435756.4	-1229897.3	692.5	270	-60	180.0
KGST-13R	-435777.2	-1229950.4	699.1	270	-60	162.0
KGST-14R	-435784.0	-1230053.0	707.1	270	-60	162.0
KGST-15R	-435705.1	-1229750.6	677.6	270	-60	173.0
KGST-16R	-435821.9	-1229857.5	674.7	0	-90	137.0

HOLE	EAST	NORTH	ELEVATION	AZIMUTH	DIP	LENGTH
KGST-17R	-435841.7	-1230048.9	717.6	270	-57	60.0
KGST-18R	-435841.2	-1230046.4	717.1	310	-60	194.0
KGST-19R	-435857.0	-1230103.0	725.0	270	-60	138.0
KGST-1R	-435830.0	-1229648.4	648.7	270	-60	45.0
KGST-20R	-435852.5	-1230102.3	724.8	310	-60	42.0
KGST-21R	-435741.3	-1229956.8	691.8	270	-59	170.0
KGST-22R	-435839.0	-1229793.8	652.6	0	-90	24.0
KGST-23R	-435832.4	-1229792.6	652.6	0	-90	129.0
KGST-24R	-435818.4	-1229743.1	637.2	0	-90	104.0
KGST-25R	-435818.2	-1229988.6	706.6	270	-60	150.0
KGST-26R	-435966.9	-1230051.9	748.1	310	-56	120.0
KGST-27R	-435966.3	-1230052.4	748.0	0	-90	72.0
KGST-28R	-435959.0	-1230056.1	747.0	270	-60	72.0
KGST-29R	-435946.2	-1230062.3	746.1	340	-59	84.0
KGST-2R	-435823.2	-1229744.3	637.6	270	-67	84.0
KGST-30R	-435946.0	-1230063.4	746.2	0	-90	90.0
KGST-31R	-435944.6	-1230061.6	746.2	12	-60	108.0
KGST-32R	-435907.8	-1230084.0	746.1	95	-60	30.0
KGST-33R	-435919.3	-1230081.0	745.9	0	-90	126.0
KGST-34R	-435870.3	-1229897.8	681.1	150	-58	42.0
KGST-35R	-435878.7	-1229902.5	681.0	205	-60	78.0
KGST-36R	-435833.0	-1229901.6	681.4	225	-57	84.0
KGST-37R	-435798.9	-1229830.5	653.1	0	-90	48.0
KGST-38R	-435871.6	-1229896.3	682.0	0	-90	47.0
KGST-39R	-435814.8	-1229893.9	682.6	270	-80	30.0
KGST-3R	-435821.8	-1229701.3	637.1	270	-65	54.0
KGST-40R	-435863.2	-1229852.1	671.5	0	-90	45.0
KGST-41R	-435802.1	-1229853.0	675.8	0	-90	65.0
KGST-4R	-435831.9	-1229796.6	652.4	270	-67	90.0
KGST-5R	-435796.6	-1229655.5	652.6	270	-60	90.0
KGST-6R	-435732.8	-1229645.1	670.1	270	-60	144.0
KGST-7R	-435753.2	-1229801.9	685.2	270	-60	108.0
KGST-8R	-435829.5	-1229859.4	674.5	270	-60	34.0

APPENDIX 2

SPECIFIC GRAVITY DETERMINATION

Protokol o skúške c. 05-01233
Test report No 05-01233

GEL s.r.o. **Tel.:** 043/490
1562
Laboratóriá spolocnosti **Fax:** 043/492
2203
Akreditované skúšobné laboratóriá **E-mail:**gel@bb.telecom.sk
Robotnícka 841/20
039 01 Turcianske Teplice

Objednávatel skúšok

Objednávatel: Kremnica Gold a.s , Horná ul. 51
97401 Banská Bystrica
Odosielatel: Kremnica Gold Kremnica
Zodp.osoba: Ing. Boris Bartalský
Tel.: 045/6743144 **Fax:** 045/6744273
Objednávka: z 15.11.2005 **Dátum prevzatia vzoriek:** 15.11.2005

SECTION 23.0 **ZÁKAZKA:** **05-00787**

DÁTUM VYKONANIA SKÚŠOK: 15.11.2005 - 28.11.2005

Pocet vzoriek: 133 **Dátum vystavenia protokolu:** 28.11.2005

Poznámky ku vzorkám:
Typ vzorky : horniny(rocks)

Výsledky skúšok

Por. císlo	Laborat. císlo	Oznacenie vzorky	**bulk weight in paraffín (Objemová hmotnost v parafíne)** [kg/m3]
1	05-003797	KSG 1/AS-6.2/10.5m	1922
2	05-003798	KSG 1/AS-6.2/11.5m	2666
3	05-003799	KSG 1/AS-6.2/12.5m	2655
4	05-003800	KSG 1/AS-6.2/13.5m	1959
5	05-003801	KSG 1/AS-6.2/14.5m	1978
6	05-003802	KSG 2/AS-6.2/70.5m	2371
7	05-003803	KSG 2/AS-6.2/71.5m	2569
8	05-003804	KSG 2/AS-6.2/72.6m	2576
9	05-003805	KSG 2/AS-6.2/73.3m	2709
10	05-003806	KSG 2/AS-6.2/74.2m	2516
11	05-003807	KSG 3/AS-6.2/75.2m	2339
12	05-003808	KSG 3/AS-6.2/76.0m	2268
13	05-003809	KSG 3/AS-6.2/77.7m	2582
14	05-003810	KSG 3/AS-6.2/80.0m	2504
15	05-003811	KSG 3/A S-6.2/81.85m	2480
16	05-003812	KSG 3/AS-6.2/83.60m	2361
17	05-003813	KSG 3/AS-6.2/84.70m	2221
18	05-003814	KSG 4/AS-6.2/112.0-113.0m	2184
19	05-003815	KSG 4/AS-6.2/114.0m	2219
20	05-003816	KSG 4/AS-6.2/116.0-117.0m	2069
21	05-003817	KSG 4/AS-6.2/118.5m	2251

Por.	Laborat.	Oznacenie	bulk weight in paraffín (Objemová hmotnost v parafíne)
císlo	císlo	vzorky	[kg/m3]
22	05-003818	KSG 4/AS-6.2/119.5-120.5m	1990
23	05-003819	KSG 5/AS-6.2/172.0-173.0m	2333
24	05-003820	KSG 5/AS-6.2/174.0-175.0m	2316
25	05-003821	KSG 5/AS-6.2/177.0-178.0m	2433
26	05-003822	KSG 5/AS-6.2/179.0-180.0m	2465
27	05-003823	KSG 5/AS-6.2/185.0-186.0m	2173
28	05-003824	KSG 5/AS-6.2/187.0-188.0m	2078
29	05-003825	KSG 5/AS-6.2/190.0-191.0m	2202
30	05-003826	KSG 5/AS-6.2/193.5-194.5m	2417
31	05-003827	KSG 5/AS-6.2/196.0-197.0m	2290
32	05-003828	KSG 6/AS-6.2/221.0-222.0m	2616
33	05-003829	KSG 6/AS-6.2/224.0-225.0m	2413
34	05-003830	KSG 6/AS-6.2/226.0-227.0m	2471
35	05-003831	KSG 6/AS-6.2/228.0-229.0m	2546
36	05-003832	KSG 7/AS-6.1.A/6.8m	2353
37	05-003833	KSG 7/AS-6.1.A/8.0m	2054
38	05-003834	KSG 7/AS-6.1.A/8.4m	2046
39	05-003835	KSG 7/AS-6.1.A/9.2m	1949
40	05-003836	KSG 8/AS-6.1.A/70.5m	2335
41	05-003837	KSG 8/AS-6.1.A/71.4m	2316
42	05-003838	KSG 8/AS-6.1.A/72.7m	2402
43	05-003839	KSG 8/AS-6.1.A/73.3m	2206
44	05-003840	KSG 8/AS-6.1.A/74.5m	2144
45	05-003841	KSG 10/AS-6.1.A/131.9m	2437
46	05-003842	KSG 10/AS-6.1.A/133.0-133.5m	2387
47	05-003843	KSG 10/AS-6.1.A/134.5m	2345
48	05-003844	KSG 10/AS-6.1.A/135.5m	2377
49	05-003845	KSG 10/AS-6.1.A/138.2m	2357
50	05-003846	KSG 11/AS-4.1.B/5.0-5.5m	2313
51	05-003847	KSG 11/AS-4.1.B/7.0-8.0m	2340
52	05-003848	KSG 11/AS-4.1.B/8.5-9.0m	2232
53	05-003849	KSG 12/AS-4.1.B/58.0m	2508
54	05-003850	KSG 12/AS-4.1.B/60.0m	2448
55	05-003851	KSG 12/AS-4.1.B/62.0m	2501
56	05-003852	KSG 13/AS-4.2/3.2m	1839
57	05-003853	KSG 13/AS-4.2/4.6m	1946
58	05-003854	KSG 13/AS-4.2/7.5m	2051
59	05-003855	KSG 14/AS-4.2/40.60m	2293
60	05-003856	KSG 14/AS-4.2/41.5m	2454
61	05-003857	KSG 14/AS-4.2/42.1m	2321
62	05-003858	KSG 14/AS-4.2/43.1m	2415
63	05-003859	KSG 14/AS-4.2/44.5m	2493
64	05-003860	KSG 15/AS-4.2/133.70m	2183
65	05-003861	KSG 15/AS-4.2/136.3m	2457
66	05-003862	KSG 15/AS-4.2/137.0m	2402
67	05-003863	KSG 15/AS-4.2/139.0m	2321
68	05-003864	KSG 15/AS-4.2/140.0m	2032
69	05-003865	KSG 15/AS-4.2/141.3m	2367
70	05-003866	KSG 15/AS-4.2/142.8m	2279
71	05-003867	KSG 16/AS-4.2/152.5m	2235
72	05-003868	KSG 16/AS-4.2/154.0m	2562
73	05-003869	KSG 16/AS-4.2/154.8m	2324
74	05-003870	KSG 16/AS-4.2/155.4m	2384
75	05-003871	KSG 17/AS-4.2/175.0m	2327
76	05-003872	KSG 17/AS-4.2/177.0m	2440
77	05-003873	KSG 17/AS-4.2/179.0m	2250
78	05-003874	KSG 17/AS-4.2/181.0m	2297
79	05-003875	KSG 17/AS-4.2/185.3m	2417

Por. císlo	Laborat. císlo	Oznacenie vzorky	bulk weight in paraffín (Objemová hmotnost v parafíne) [kg/m3]
80	05-003876	KSG 17/AS-4.2/190.2m	2433
81	05-003877	KSG 17/AS-4.2/195.0m	2502
82	05-003878	KSG 17/AS-4.2/195.4m	2346
83	05-003879	KSG 17/AS-4.2/197.6m	2572
84	05-003880	KSG 17/AS-4.2/199.2m	2348
85	05-003881	KSG 18/AS-4.2/209.5m	2599
86	05-003882	KSG 18/AS-4.2/212.5m	2439
87	05-003883	KSG 18/AS-4.2/214.0m	2511
88	05-003884	KSG 18/AS-4.2/215.5m	2582
89	05-003885	KSG 18/AS-4.2/216.9m	2420
90	05-003886	KSG 18/AS-4.2/218.5m	2296
91	05-003887	KSG 18/AS-4.2/219.9m	2333
92	05-003888	KSG 19/AS-4.2/225.0m	2549
93	05-003889	KSG 19/AS-4.2/226.3m	2218
94	05-003890	KSG 19/AS-4.2/227.75m	2279
95	05-003891	KSG 19/AS-4.2/228.7m	2301

Por. císlo	Laborat. císlo	Oznacenie vzorky	bulk weight in paraffín (Objemová hmotnost v parafíne) [kg/m3]
96	05-003892	KSG 19/AS-4.2/229.7m	2278
97	05-003893	KSG 20/AS-4.2/256.9m	2387
98	05-003894	KSG 20/AS-4.2/257.4m	2344
99	05-003895	KSG 20/AS-4.2/258.3m	2262
100	05-003896	KSG 20/AS-4.2/259.7m	2302
101	05-003897	KSG 21/AS-5.1.1/23.0m	2301
102	05-003898	KSG 21/AS-5.1.1/24.0m	2289
103	05-003899	KSG 21/AS-5.1.1/25.0m	2484
104	05-003900	KSG 21/AS-5.1.1/28.0m	2531
105	05-003901	KSG 21/AS-5.1.1/30.0m	2446
106	05-003902	KSG 22/AS-5/18.3m	2159
107	05-003903	KSG 22/AS-5/20.5m	2071
108	05-003904	KSG 22/AS-5/19.4m	2050
109	05-003905	KSG 22/AS-5/22.5m	2239
110	05-003906	KSG 22/AS-5/21.7m	2110
111	05-003907	KSG 23/AS-5/30.5m	2271
112	05-003908	KSG 23/AS-5/32.0m	2058
113	05-003909	KSG 23/AS-5/34.3m	2011
114	05-003910	KSG 23/AS-5/35.4m	2206
115	05-003911	KSG 23/AS-5/36.4m	2214
116	05-003912	KSG 23/AS-5/38.0m	2214
117	05-003913	KSG 23/AS-5/39.7m	2224
118	05-003914	KSG 24/AS-5/43.5m	2283
119	05-003915	KSG 24/AS-5/45.2m	2244
120	05-003916	KSG 24/AS-5/47.7m	2127
121	05-003917	KSG 24/AS-5/49.1m	1944
122	05-003918	KSG 24/AS-5/50.5m	2243
123	05-003919	KSG 25/AS-5/136.9m	2496
124	05-003920	KSG 25/AS-5/138.9m	2390
125	05-003921	KSG 25/AS-5/140.1m	2105
126	05-003922	KSG 25/AS-5/142.7m	2298
127	05-003923	KSG 25/AS-5/144.6m	1889
128	05-003924	KSG 25/AS-5/147.1m	2163

129	05-003925	KSG 25/AS-5/149.4m	1898
130	05-003926	KSG 25/AS-5/150.4m	2049
131	05-003927	KSG 26/AS-5/168.0-177.0a	1941
132	05-003928	KSG 26/AS-5/168.0-177.0b	2188
133	05-003929	KSG 26/AS-5/181.3m	2415

Poznámky ku skúškam:
Zoznam príloh: -
Reklamácia: Reklamovat výsledky laboratórnych skúšok možno do 30 dní od dátumu odoslania výsledkov zákazníkovi. Akceptované a vybavované sú len písomne podané reklamácie.
Uchovanie vzoriek:
a, Uchovávané sú iba vzorky, u ktorých sa pôvodné vlastnosti nemenia.
b, Vzorky sú uchovávané do definitívneho prevzatia výsledkov skúšok zákazníkom t.j. do doby uplynutia podmienok reklamácie.
c, Vrátenie zvyšku vzoriek - vzorky sa vracajú zákazníkovi na základe jeho písomnej žiadosti a na jeho náklady. Vostatných prípadoch sú zvyšky po uplynutí doby uchovania likvidované.

Protokol o skúške vyhotovil
Koraušová Iveta
vedúca TL

Protokol o skúške schválil
Ing. Erhardt Tibor
vedúci VLS

Specific Gravity Determinations for the Kremnica Deposit.

SAMPLE ID/HOLE/INTERVAL	kg/m3	gm/cm3	DESCRIPTION
KSG 1/AS-6.2/10.5m	1922	1.92	Hangingwall andesite near surface
KSG 1/AS-6.2/11.5m	2666	2.67	Hangingwall andesite near surface
KSG 1/AS-6.2/12.5m	2655	2.66	Hangingwall andesite near surface
KSG 1/AS-6.2/13.5m	1959	1.96	Hangingwall andesite near surface
KSG 1/AS-6.2/14.5m	1978	1.98	Hangingwall andesite near surface
KSG 2/AS-6.2/70.5m	2371	2.37	Hangingwall andesite at depth
KSG 2/AS-6.2/71.5m	2569	2.57	Hangingwall andesite at depth
KSG 2/AS-6.2/72.6m	2576	2.58	Hangingwall andesite at depth
KSG 2/AS-6.2/73.3m	2709	2.71	Hangingwall andesite at depth
KSG 2/AS-6.2/74.2m	2516	2.52	Hangingwall andesite at depth
KSG 3/AS-6.2/75.2m	2339	2.34	Hydrothermally brecciated andesite
KSG 3/AS-6.2/76.0m	2268	2.27	Hydrothermally brecciated andesite
KSG 3/AS-6.2/77.7m	2582	2.58	Hydrothermally brecciated andesite
KSG 3/AS-6.2/80.0m	2504	2.50	Hydrothermally brecciated andesite
KSG 3/AS-6.2/81.85m	2480	2.48	Hydrothermally brecciated andesite
KSG 3/AS-6.2/83.60m	2361	2.36	Hydrothermally brecciated andesite
KSG 3/AS-6.2/84.70m	2221	2.22	Hydrothermally brecciated andesite
KSG 4/AS-6.2/112.0-113.0m	2184	2.18	Hydrothermally brecciated andesite
KSG 4/AS-6.2/114.0m	2219	2.22	Hydrothermally brecciated andesite
KSG 4/AS-6.2/116.0-117.0m	2069	2.07	Hydrothermally brecciated andesite
KSG 4/AS-6.2/118.5m	2251	2.25	Hydrothermally brecciated andesite
KSG 4/AS-6.2/119.5-120.5m	1990	1.99	Hydrothermally brecciated andesite
KSG 5/AS-6.2/172.0-173.0m	2333	2.33	Rubbly ore zone at depth
KSG 5/AS-6.2/174.0-175.0m	2316	2.32	Rubbly ore zone at depth
KSG 5/AS-6.2/177.0-178.0m	2433	2.43	Rubbly ore zone at depth
KSG 5/AS-6.2/179.0-180.0m	2465	2.47	Rubbly ore zone at depth
KSG 5/AS-6.2/185.0-186.0m	2173	2.17	Rubbly ore zone at depth
KSG 5/AS-6.2/187.0-188.0m	2078	2.08	Rubbly ore zone at depth
KSG 5/AS-6.2/190.0-191.0m	2202	2.20	Rubbly ore zone at depth
KSG 5/AS-6.2/193.5-194.5m	2417	2.42	Rubbly ore zone at depth
KSG 5/AS-6.2/196.0-197.0m	2290	2.29	Rubbly ore zone at depth
KSG 6/AS-6.2/221.0-222.0m	2616	2.62	Footwall Andesite
KSG 6/AS-6.2/224.0-225.0m	2413	2.41	Footwall Andesite
KSG 6/AS-6.2/226.0-227.0m	2471	2.47	Footwall Andesite
KSG 6/AS-6.2/228.0-229.0m	2546	2.55	Footwall Andesite
KSG 7/AS-6.1.A/6.8m	2353	2.35	Hangingwall andesite near surface
KSG 7/AS-6.1.A/8.0m	2054	2.05	Hangingwall andesite near surface
KSG 7/AS-6.1.A/8.4m	2046	2.05	Hangingwall andesite near surface
KSG 7/AS-6.1.A/9.2m	1949	1.95	Hangingwall andesite near surface
KSG 8/AS-6.1.A/70.5m	2335	2.34	Hangingwall andesite at depth
KSG 8/AS-6.1.A/71.4m	2316	2.32	Hangingwall andesite at depth
KSG 8/AS-6.1.A/72.7m	2402	2.40	Hangingwall andesite at depth
KSG 8/AS-6.1.A/73.3m	2206	2.21	Hangingwall andesite at depth
KSG 8/AS-6.1.A/74.5m	2144	2.14	Hangingwall andesite at depth

SAMPLE ID/HOLE/INTERVAL	kg/m3	gm/cm3	DESCRIPTION
			Rubbly ore zone at depth -not sampled ,-too rubbly core
KSG 10/AS-6.1.A/131.9m	2437	2.44	Footwall Andesite
KSG 10/AS-6.1.A/133.0-133.5m	2387	2.39	Footwall Andesite
KSG 10/AS-6.1.A/134.5m	2345	2.35	Footwall Andesite
KSG 10/AS-6.1.A/135.5m	2377	2.38	Footwall Andesite
KSG 10/AS-6.1.A/138.2m	2357	2.36	Footwall Andesite
KSG 11/AS-4.1.B/5.0-5.5m	2313	2.31	Rubbly ore zone at surface
KSG 11/AS-4.1.B/7.0-8.0m	2340	2.34	Rubbly ore zone at surface
KSG 11/AS-4.1.B/8.5-9.0m	2232	2.23	Rubbly ore zone at surface
KSG 12/AS-4.1.B/58.0m	2508	2.51	Rubbly ore zone at depth
KSG 12/AS-4.1.B/60.0m	2448	2.45	Rubbly ore zone at depth
KSG 12/AS-4.1.B/62.0m	2501	2.50	Rubbly ore zone at depth
KSG 13/AS-4.2/3.2m	1839	1.84	Hangingwall andesite near surface
KSG 13/AS-4.2/4.6m	1946	1.95	Hangingwall andesite near surface
KSG 13/AS-4.2/7.5m	2051	2.05	Hangingwall andesite near surface
KSG 14/AS-4.2/40.60m	2293	2.29	Hangingwall andesite at depth
KSG 14/AS-4.2/41.5m	2454	2.45	Hangingwall andesite at depth
KSG 14/AS-4.2/42.1m	2321	2.32	Hangingwall andesite at depth
KSG 14/AS-4.2/43.1m	2415	2.42	Hangingwall andesite at depth
KSG 14/AS-4.2/44.5m	2493	2.49	Hangingwall andesite at depth
KSG 15/AS-4.2/133.70m	2183	2.18	Hydrothermally brecciated andesite
KSG 15/AS-4.2/136.3m	2457	2.46	Hydrothermally brecciated andesite
KSG 15/AS-4.2/137.0m	2402	2.40	Hydrothermally brecciated andesite
KSG 15/AS-4.2/139.0m	2321	2.32	Hydrothermally brecciated andesite
KSG 15/AS-4.2/140.0m	2032	2.03	Hydrothermally brecciated andes ite
KSG 15/AS-4.2/141.3m	2367	2.37	Hydrothermally brecciated andesite
KSG 15/AS-4.2/142.8m	2279	2.28	Hydrothermally brecciated andesite
KSG 16/AS-4.2/152.5m	2235	2.24	Hydrothermally brecciated andesite
KSG 16/AS-4.2/154.0m	2562	2.56	Hydrothermally brecciated andesite
KSG 16/AS-4.2/154.8m	2324	2.32	Hydrothermally brecciated andesite
KSG 16/AS-4.2/155.4m	2384	2.38	Hydrothermally brecciated andesite
KSG 17/AS-4.2/175.0m	2327	2.33	Intact vein quartz
KSG 17/AS-4.2/177.0m	2440	2.44	Intact vein quartz
KSG 17/AS-4.2/179.0m	2250	2.25	Intact vein quartz
KSG 17/AS-4.2/181.0m	2297	2.30	Intact vein quartz
KSG 17/AS-4.2/185.3m	2417	2.42	Intact vein quartz
KSG 17/AS-4.2/190.2m	2433	2.43	Intact vein quartz
KSG 17/AS-4.2/195.0m	2502	2.50	Intact vein quartz
KSG 17/AS-4.2/195.4m	2346	2.35	Intact vein quartz
KSG 17/AS-4.2/197.6m	2572	2.57	Intact vein quartz
KSG 17/AS-4.2/199.2m	2348	2.35	Intact vein quartz
KSG 18/AS-4.2/209.5m	2599	2.60	Footwall Andesite
KSG 18/AS-4.2/212.5m	2439	2.44	Footwall Andesite
KSG 18/AS-4.2/214.0m	2511	2.51	Footwall Andesite
KSG 18/AS-4.2/215.5m	2582	2.58	Footwall Andesite

SAMPLE ID/HOLE/INTERVAL	kg/m3	gm/cm3	DESCRIPTION
KSG 18/AS-4.2/216.9m	2420	2.42	Footwall Andesite
KSG 18/AS-4.2/218.5m	2296	2.30	Footwall Andesite
KSG 18/AS-4.2/219.9m	2333	2.33	Footwall Andesite
KSG 19/AS-4.2/225.0m	2549	2.55	Rhyolite dyke
KSG 19/AS-4.2/226.3m	2218	2.22	Rhyolite dyke
KSG 19/AS-4.2/227.75m	2279	2.28	Rhyolite dyke
KSG 19/AS-4.2/228.7m	2301	2.30	Rhyolite dyke
KSG 19/AS-4.2/229.7m	2278	2.28	Rhyolite dyke
KSG 20/AS-4.2/256.9m	2387	2.39	Rhyolite dyke
KSG 20/AS-4.2/257.4m	2344	2.34	Rhyolite dyke
KSG 20/AS-4.2/258.3m	2262	2.26	Rhyolite dyke
KSG 20/AS-4.2/259.7m	2302	2.30	Rhyolite dyke
KSG 21/AS-5.1.1/23.0m	2301	2.30	Rubbly ore zone near surface
KSG 21/AS-5.1.1/24.0m	2289	2.29	Rubbly ore zone near surface
KSG 21/AS-5.1.1/25.0m	2484	2.48	Rubbly ore zone near surface
KSG 21/AS-5.1.1/28.0m	2531	2.53	Rubbly ore zone near surface
KSG 21/AS-5.1.1/30.0m	2446	2.45	Rubbly ore zone near surface
KSG 22/AS-5/18.3m	2159	2.16	Hydrothermally brecciated andesite near surface
KSG 22/AS-5/20.5m	2071	2.07	Hydrothermally brecciated andesite near surface
KSG 22/AS-5/19.4m	2050	2.05	Hydrothermally brecciated andesite near surface
KSG 22/AS-5/22.5m	2239	2.24	Hydrothermally brecciated andesite near surface
KSG 22/AS-5/21.7m	2110	2.11	Hydrothermally brecciated andesite near surface
KSG 23/AS-5/30.5m	2271	2.27	Hydrothermally brecciated andesite medium depth
KSG 23/AS-5/32.0m	2058	2.06	Hydrothermally brecciated andesite medium depth
KSG 23/AS-5/34.3m	2011	2.01	Hydrothermally brecciated andesite medium depth
KSG 23/AS-5/35.4m	2206	2.21	Hydrothermally brecciated andesite medium depth
KSG 23/AS-5/36.4m	2214	2.21	Hydrothermally brecciated andesite medium depth
KSG 23/AS-5/38.0m	2214	2.21	Hydrothermally brecciated andesite medium depth
KSG 23/AS-5/39.7m	2224	2.22	Hydrothermally brecciated andesite medium depth
KSG 24/AS-5/43.5m	2283	2.28	Hydrothermally brecciated andesite medium depth
KSG 24/AS-5/45.2m	2244	2.24	Hydrothermally brecciated andesite medium depth
KSG 24/AS-5/47.7m	2127	2.13	Hydrothermally brecciated andesite medium depth
KSG 24/AS-5/49.1m	1944	1.94	Hydrothermally brecciated andesite medium depth
KSG 24/AS-5/50.5m	2243	2.24	Hydrothermally brecciated andesite medium depth
KSG 25/AS-5/136.9m	2496	2.50	Collapsed Qtz Vein
KSG 25/AS-5/138.9m	2390	2.39	Collapsed Qtz Vein
KSG 25/AS-5/140.1m	2105	2.11	Collapsed Qtz Vein
KSG 25/AS-5/142.7m	2298	2.30	Collapsed Qtz Vein
KSG 25/AS-5/144.6m	1889	1.89	Collapsed Qtz Vein
KSG 25/AS-5/147.1m	2163	2.16	Collapsed Qtz Vein
KSG 25/AS-5/149.4m	1898	1.90	Collapsed Qtz Vein
KSG 25/AS-5/150.4m	2049	2.05	Collapsed Qtz Vein
KSG 26/AS-5/168.0-177.0a	1941	1.94	Collapsed Qtz Vein
KSG 26/AS-5/168.0-177.0b	2188	2.19	Collapsed Qtz Vein
KSG 26/AS-5/181.3m	2415	2.42	Collapsed Qtz Vein

APPENDIX 3

ASSAY AND COMPOSITE STATISTICS

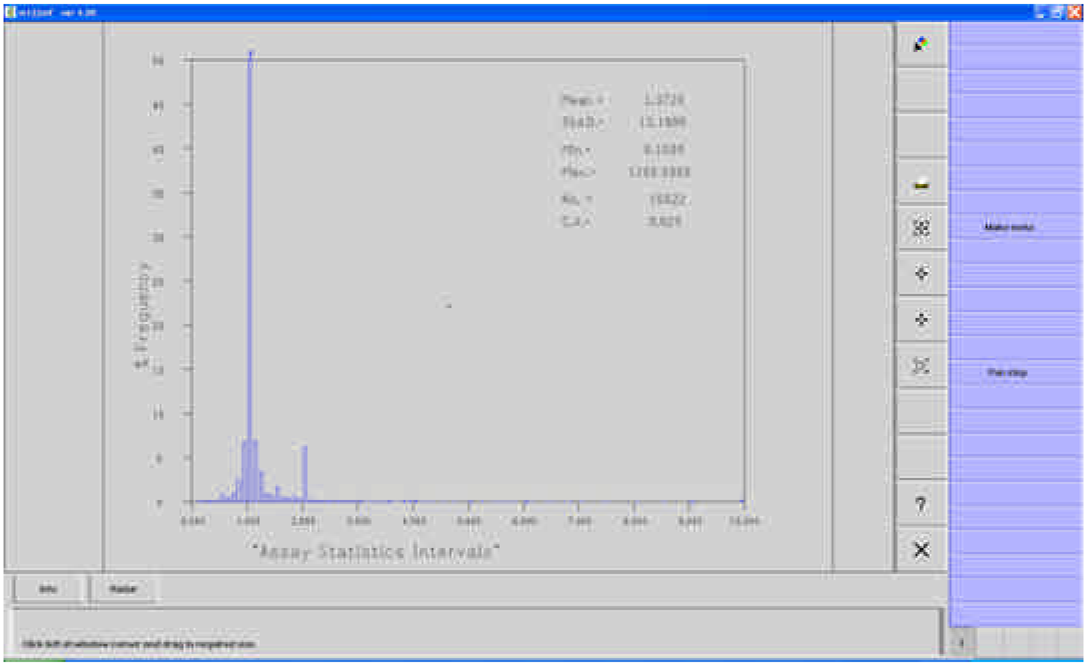


Figure A1: Assay Intervals Statistics.

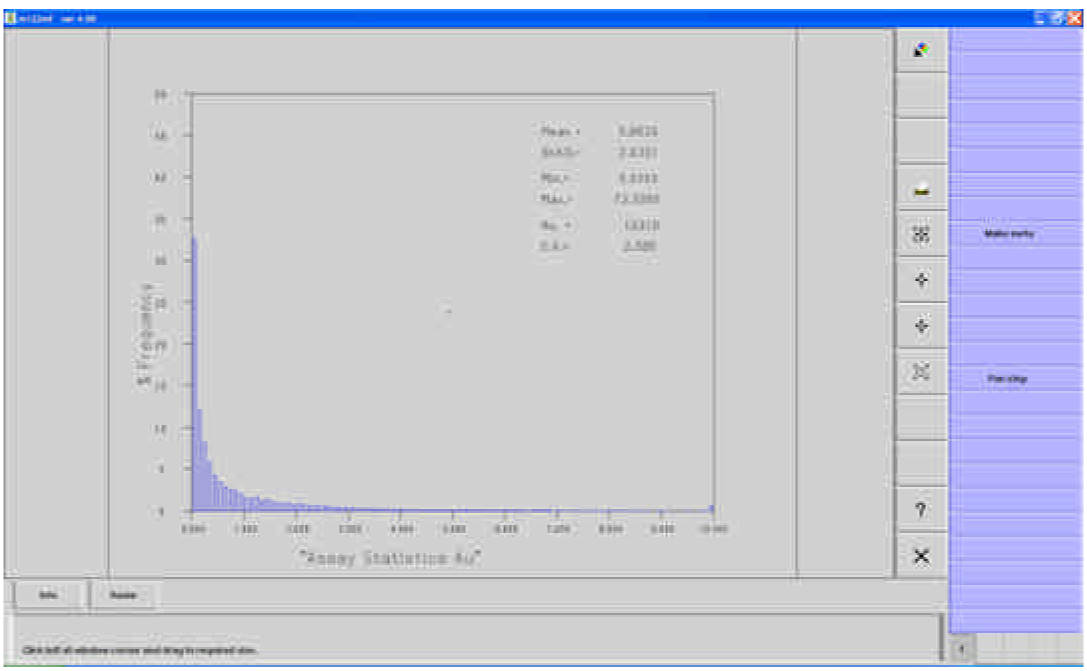


Figure A2: Au Statistics.

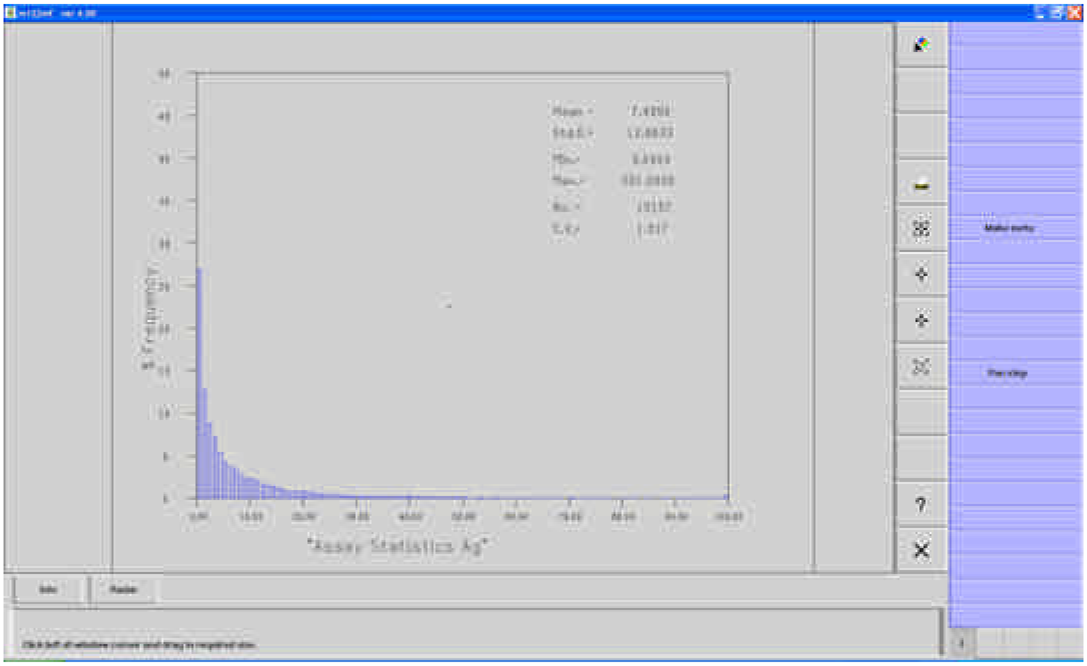


Figure A3: Ag Statistics.

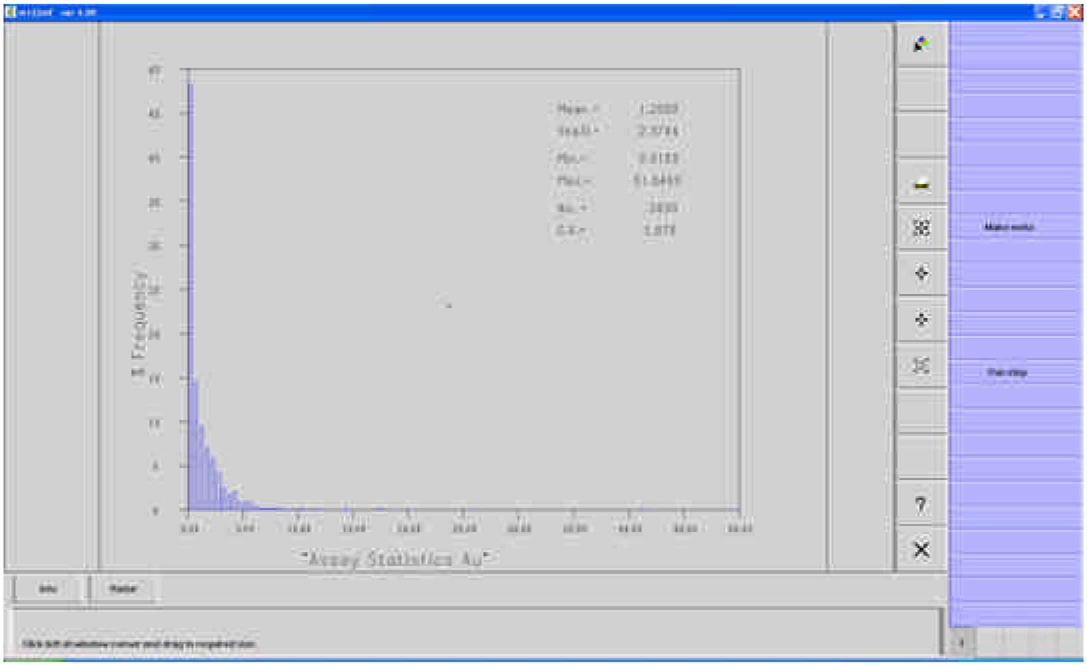


Figure A4: Assay Statistics - RC Only

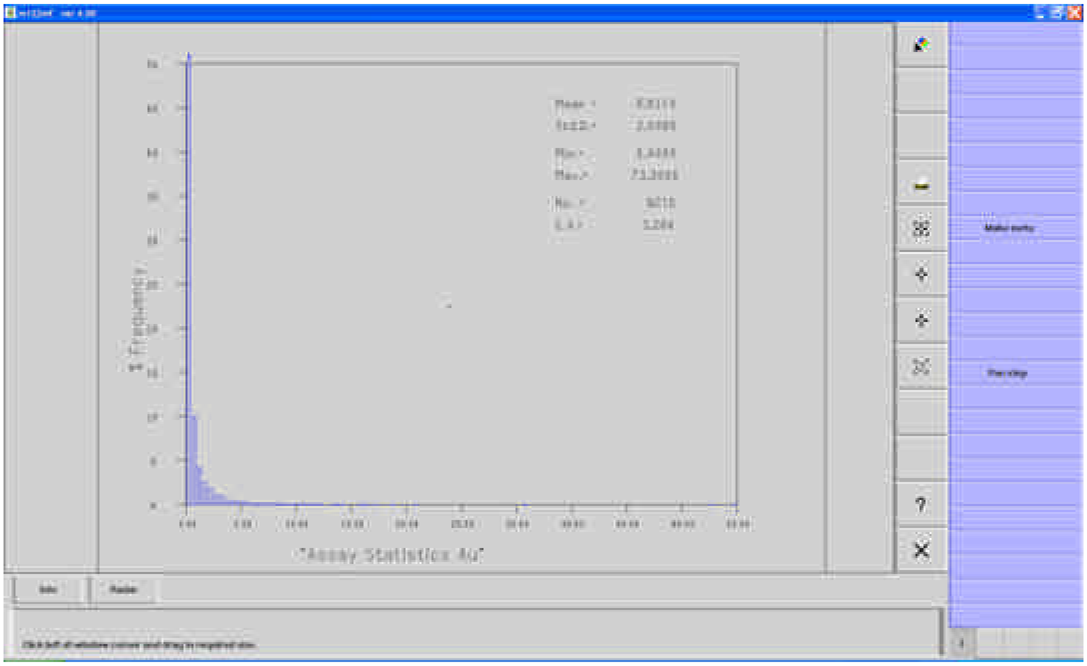


Figure A5: Assay Statistics - Argosy Core Drilling

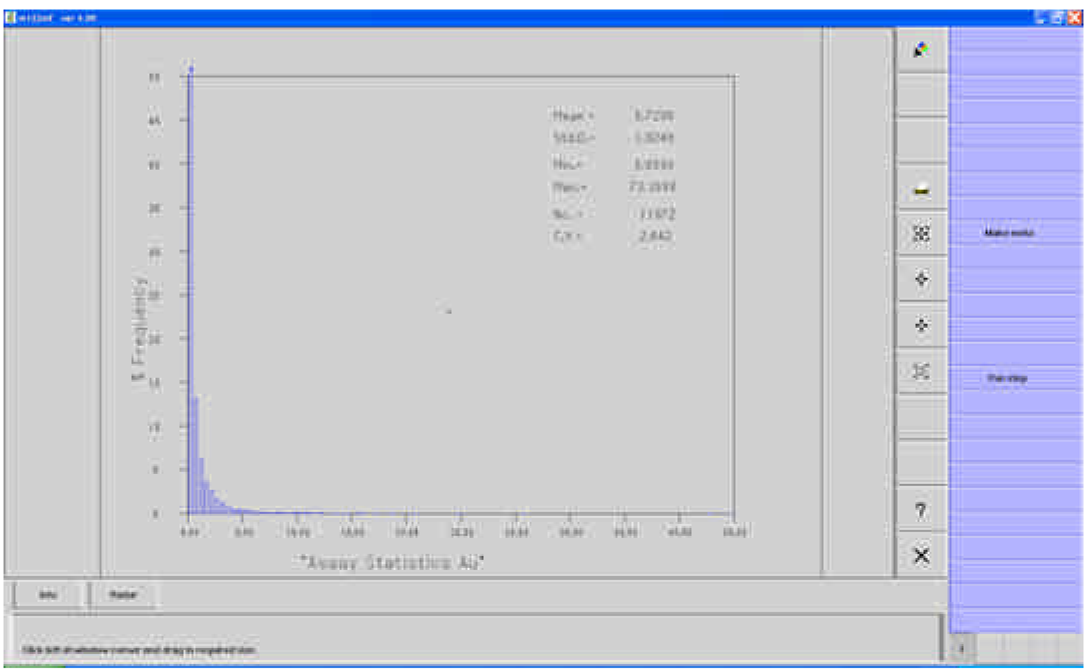


Figure A6: Assay Statistics - Core and Drift Data

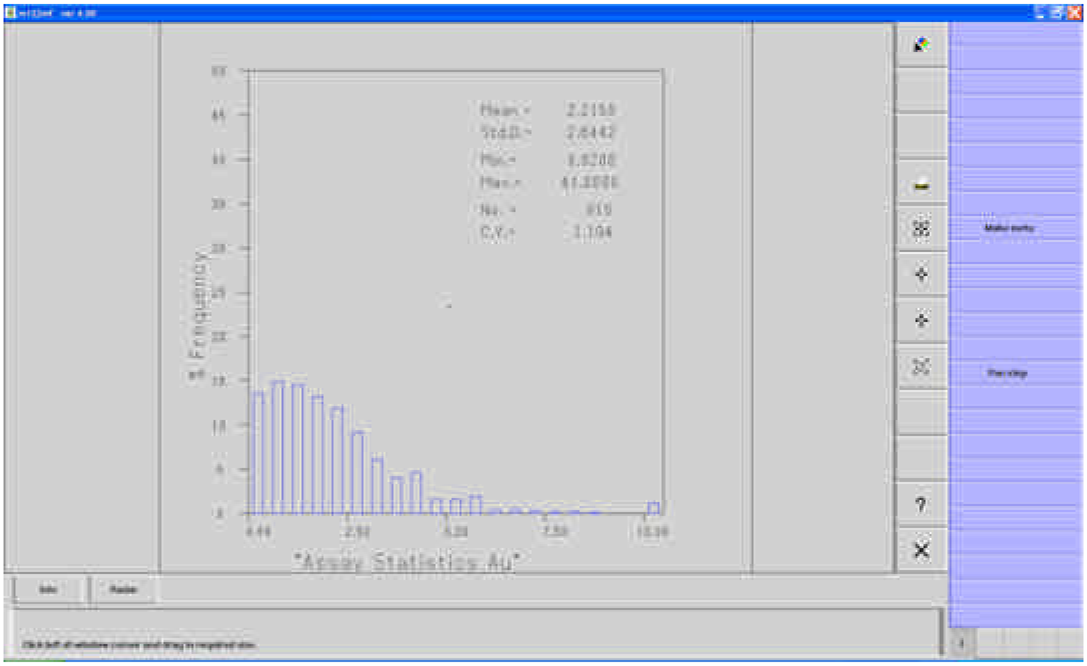


Figure A7: Assay Statistics - RC Vein

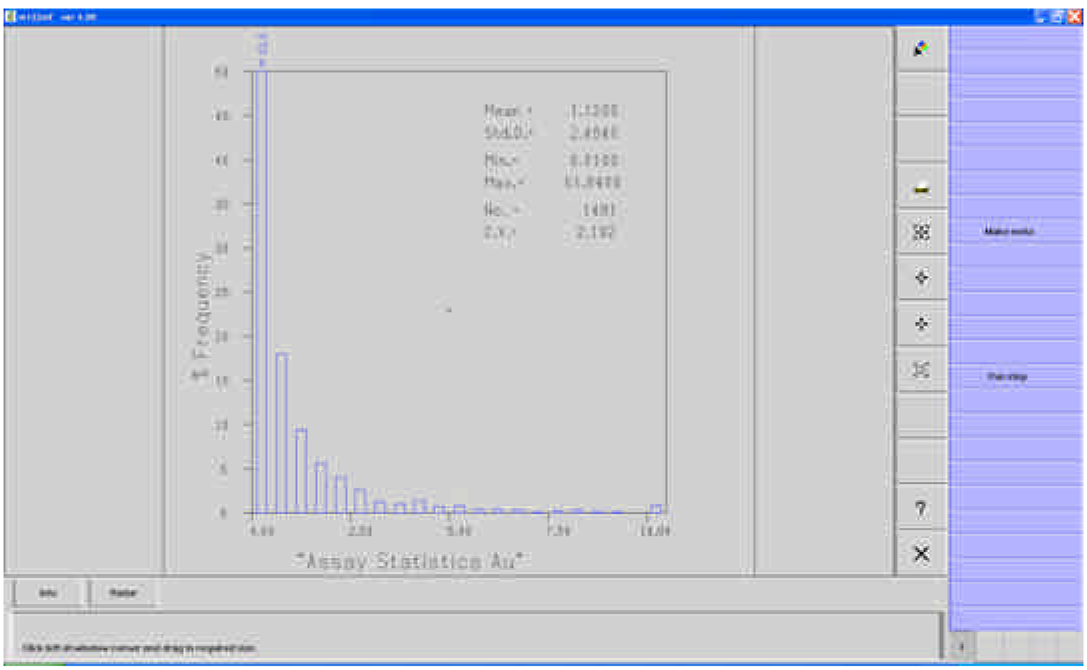


Figure A8: Assay Statistics - RC Stockwork

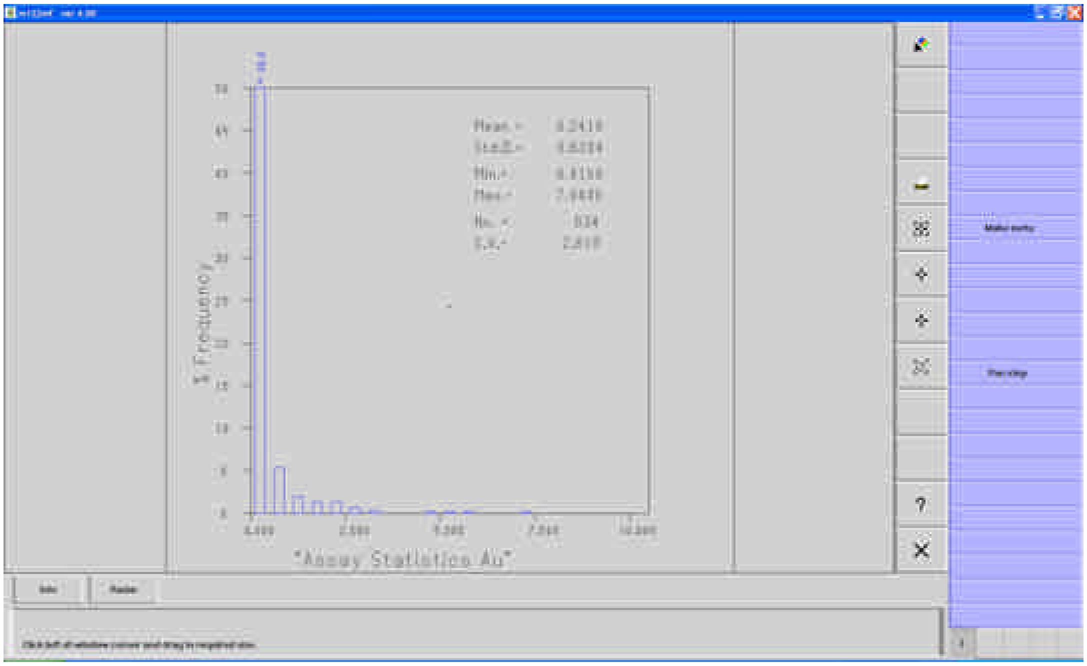


Figure A9: Assay Statistics - RC Andesite

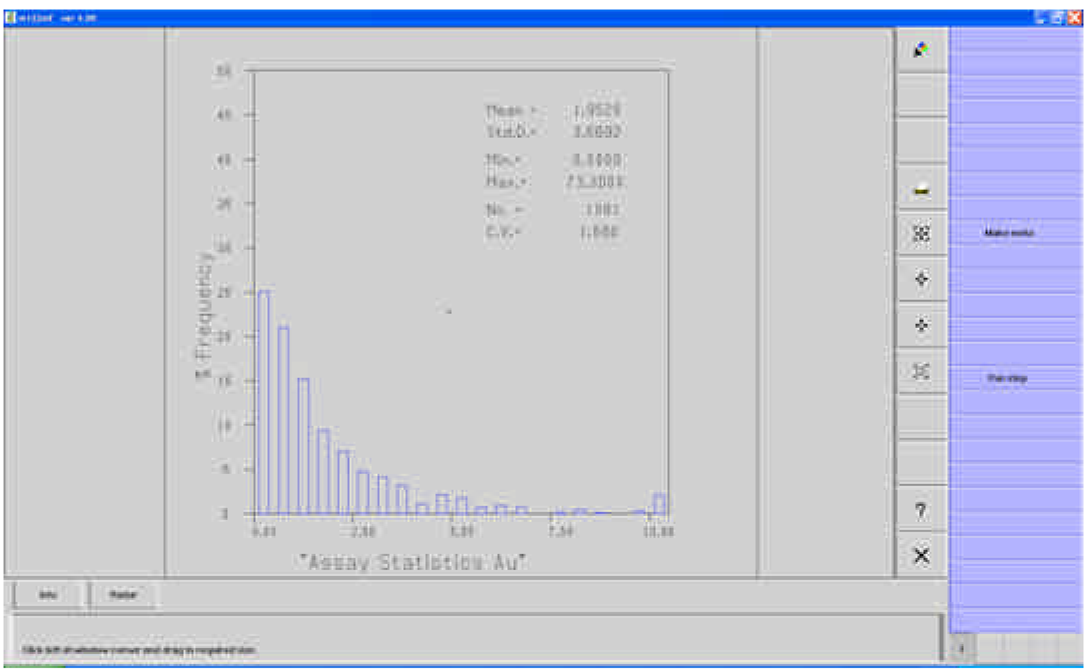


Figure A10: Assay Statistics - DDH Vein

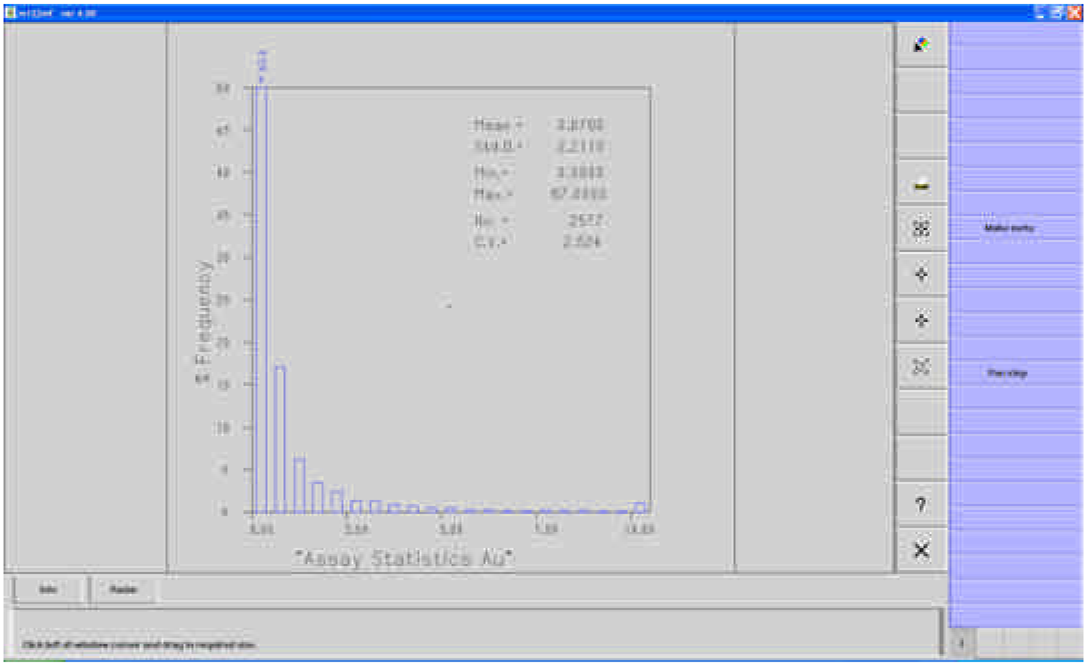


Figure A11: Assay Statistics - DDH SW

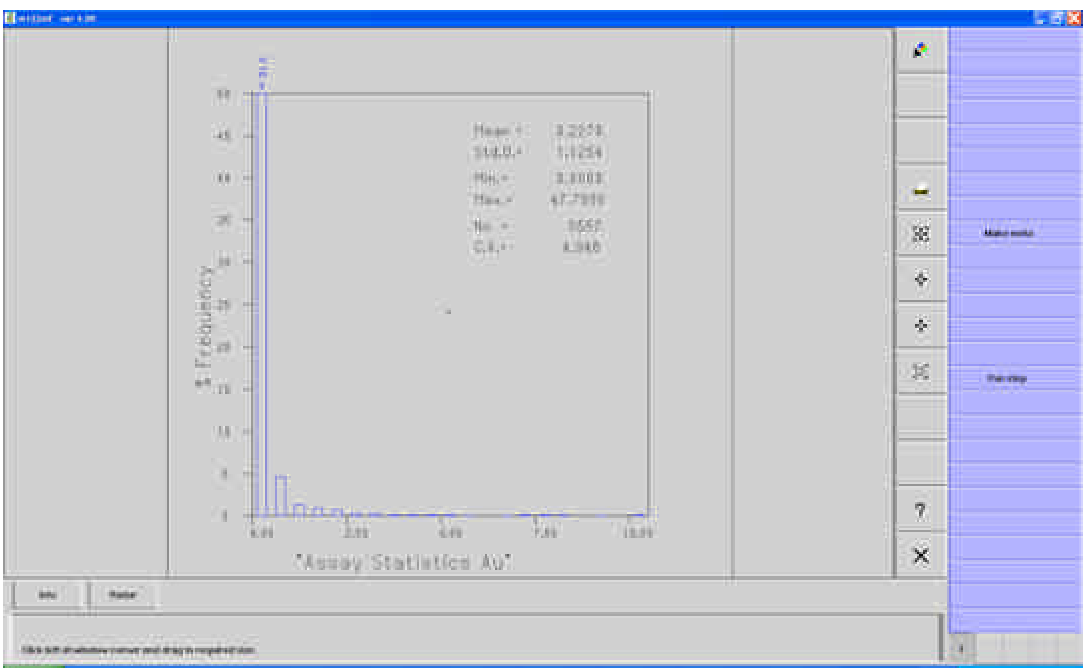


Figure A12: Assay Statistics - DDH Andesite

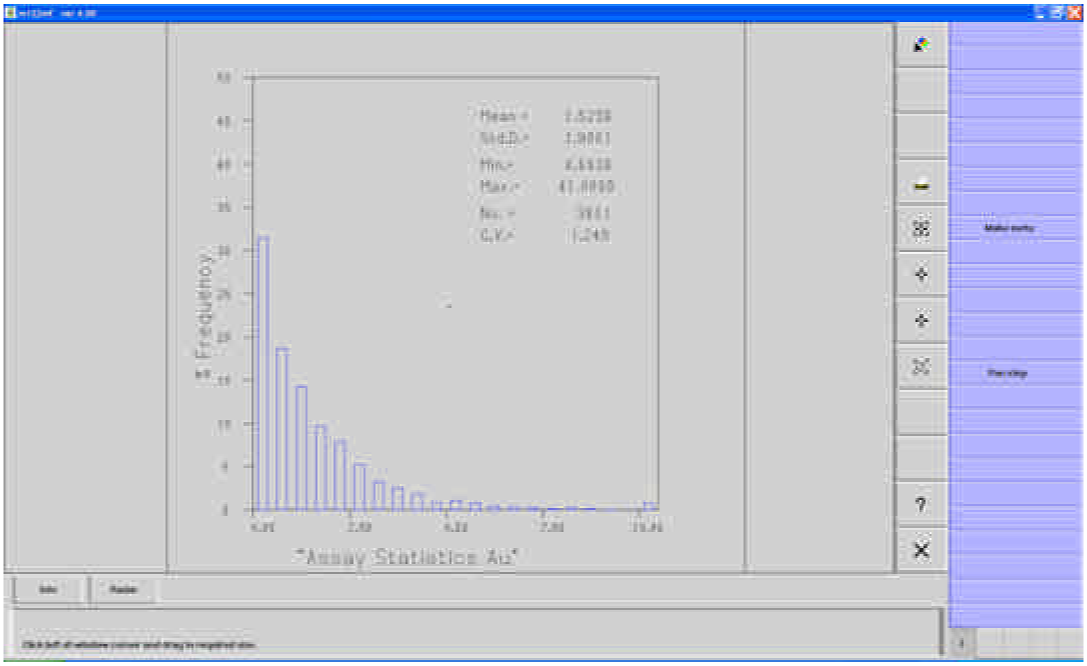


Figure A13: Assay Statistics - Oxide

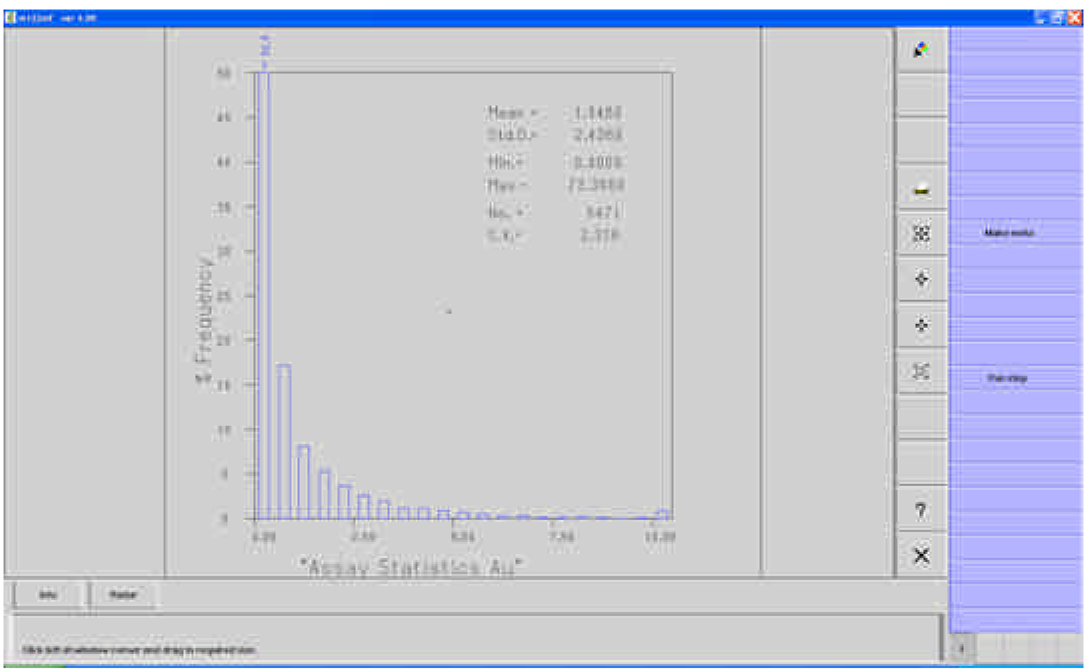


Figure A14: Assay Statistics - Sulfide

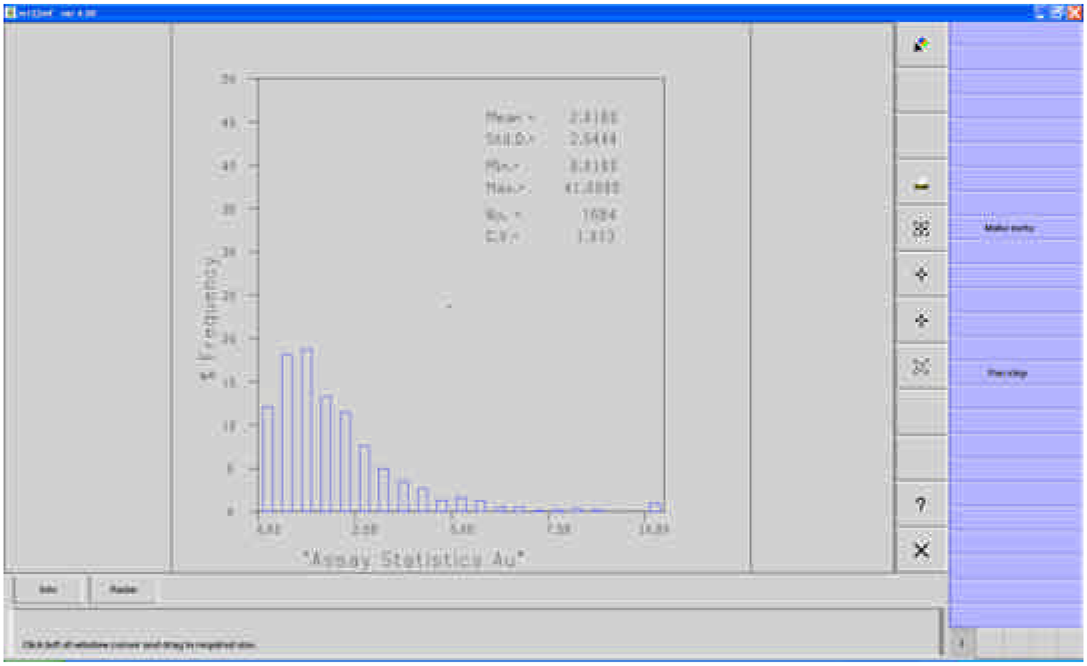


Figure A15: Assay Statistics - Oxide Vein

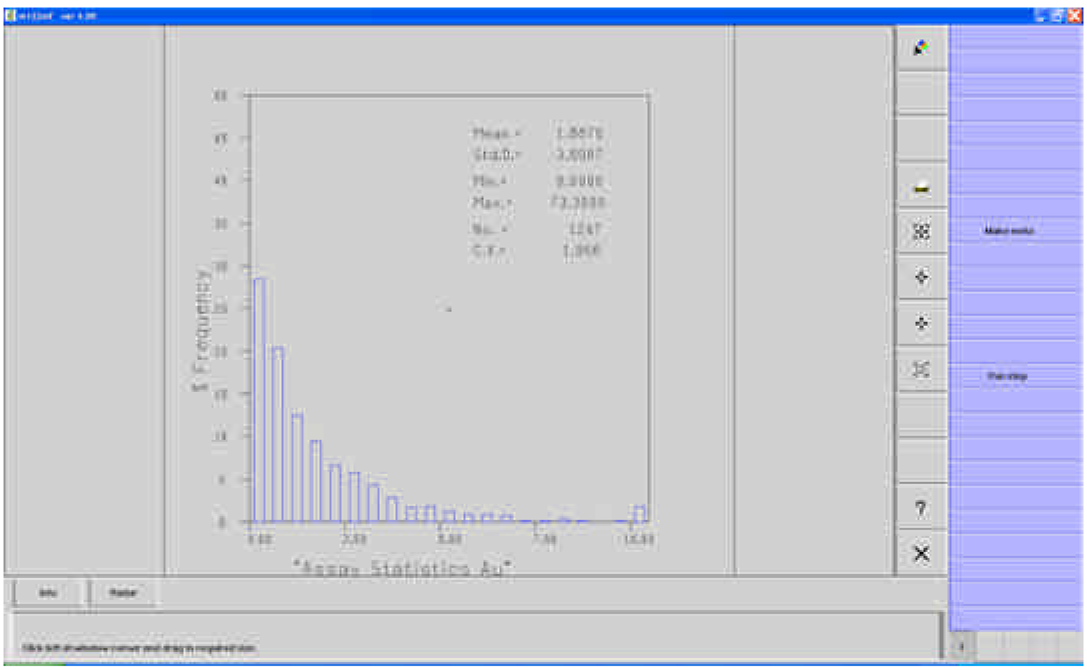


Figure A16: Assay Statistics - Sulfide Vein

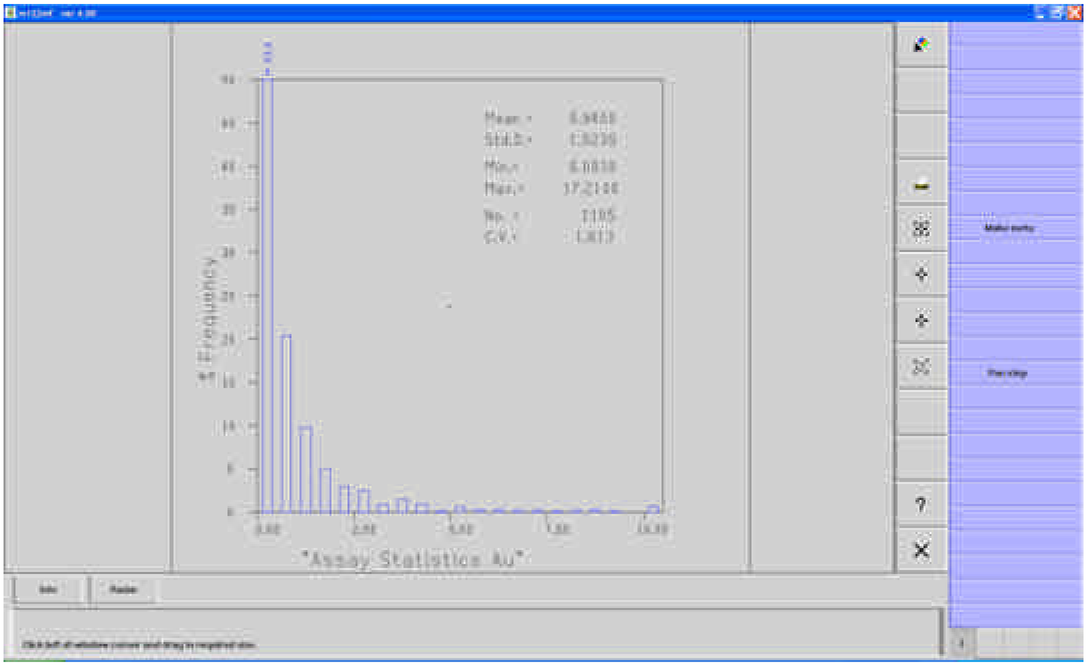


Figure A17: Assay Statistics - Oxide SW

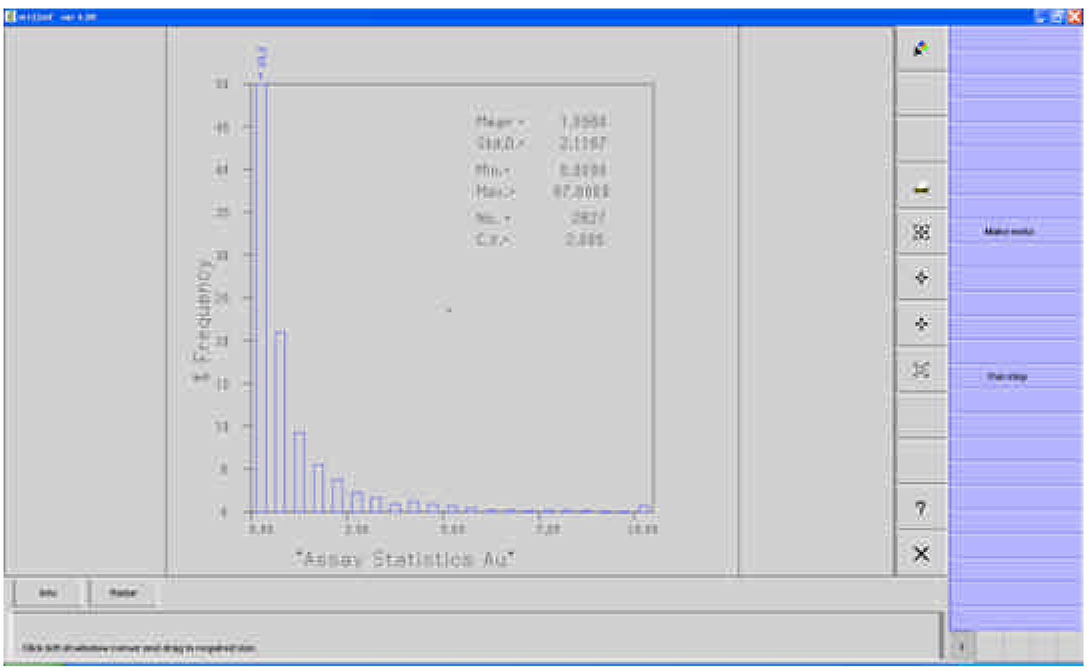


Figure A18: Assay Statistics - Sulfide SW

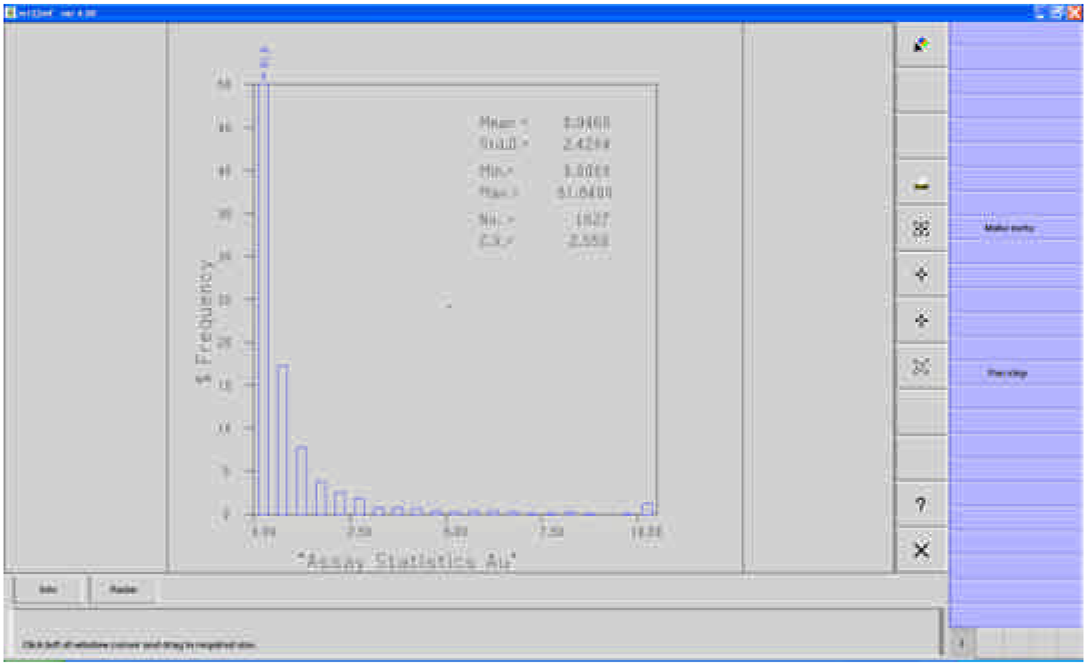


Figure A19: Assay Statistics - Outside OX/SUL in SW

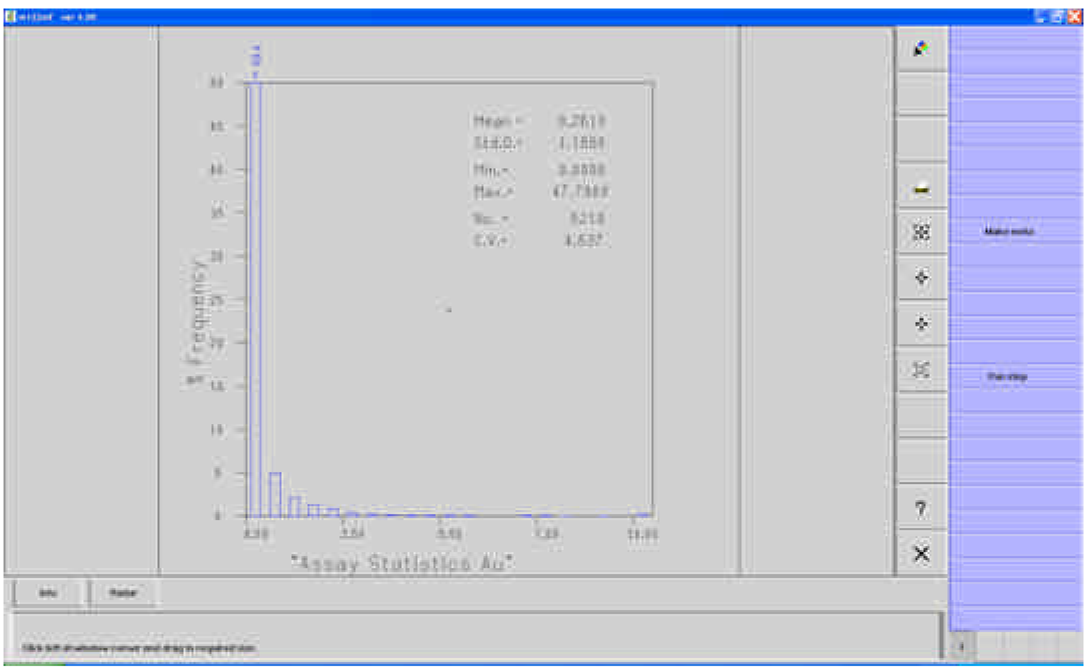


Figure A20: Assay Statistics - Outside OX/SUL in Andesite

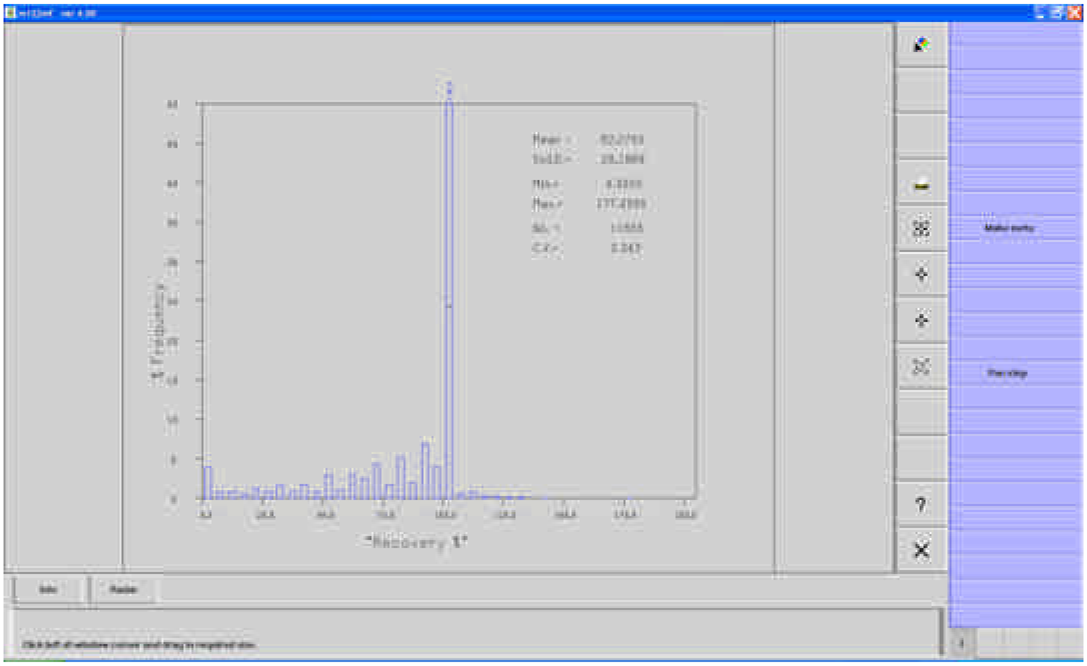


Figure A21: Assay Statistics - Recovery %

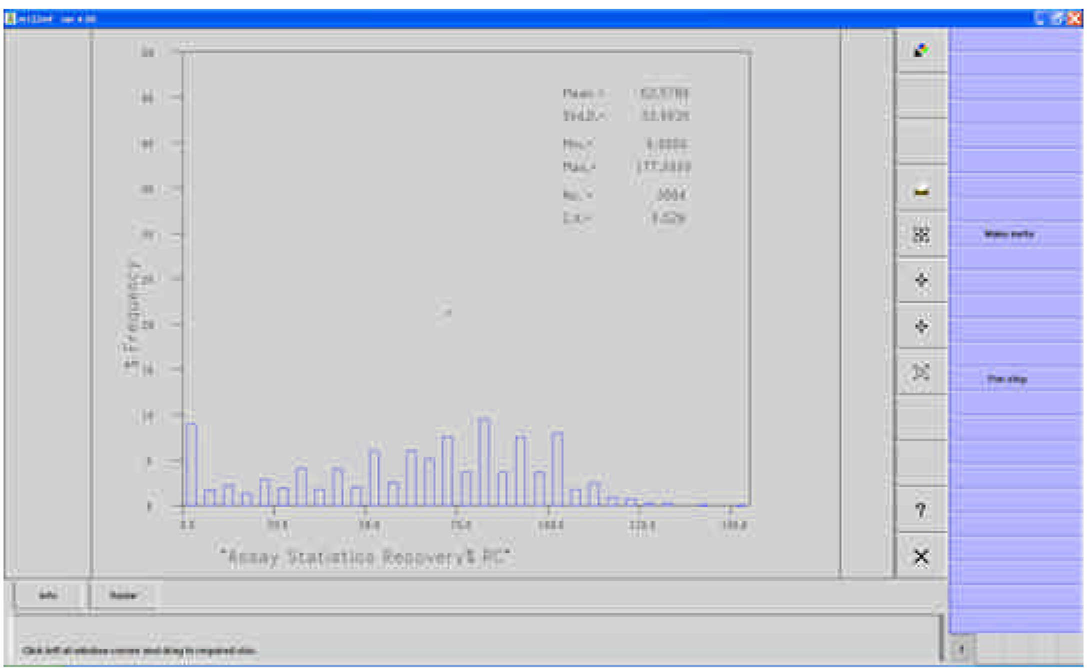


Figure A22: Assay Statistics - RC Recovery %

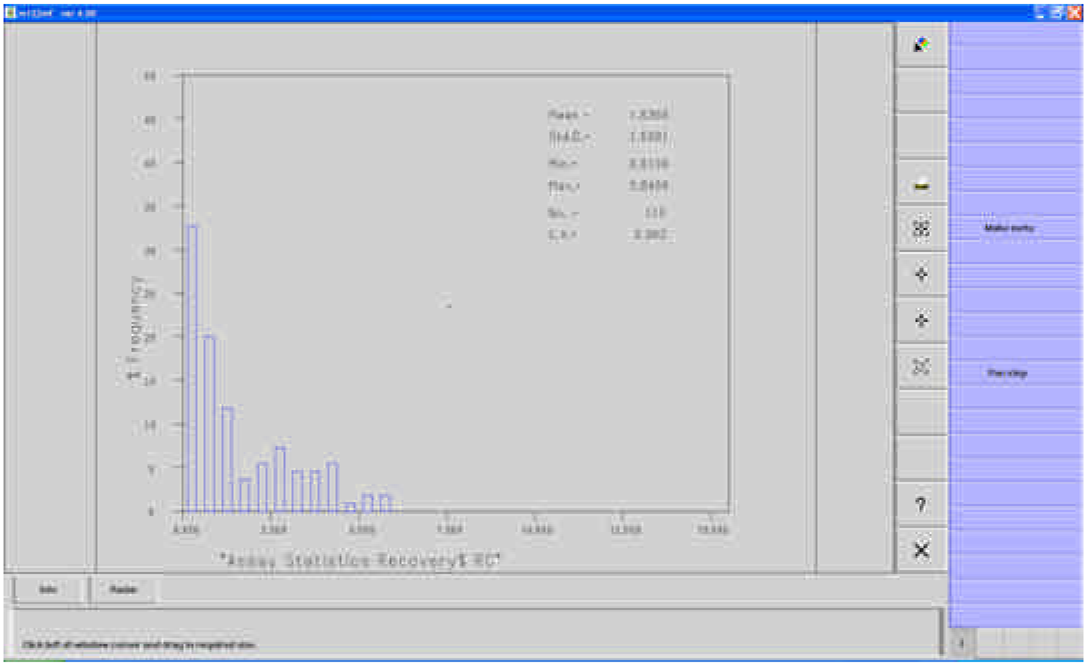


Figure A23: Assay Statistics – RC Intervals with 0-10% Recovery Au Grade

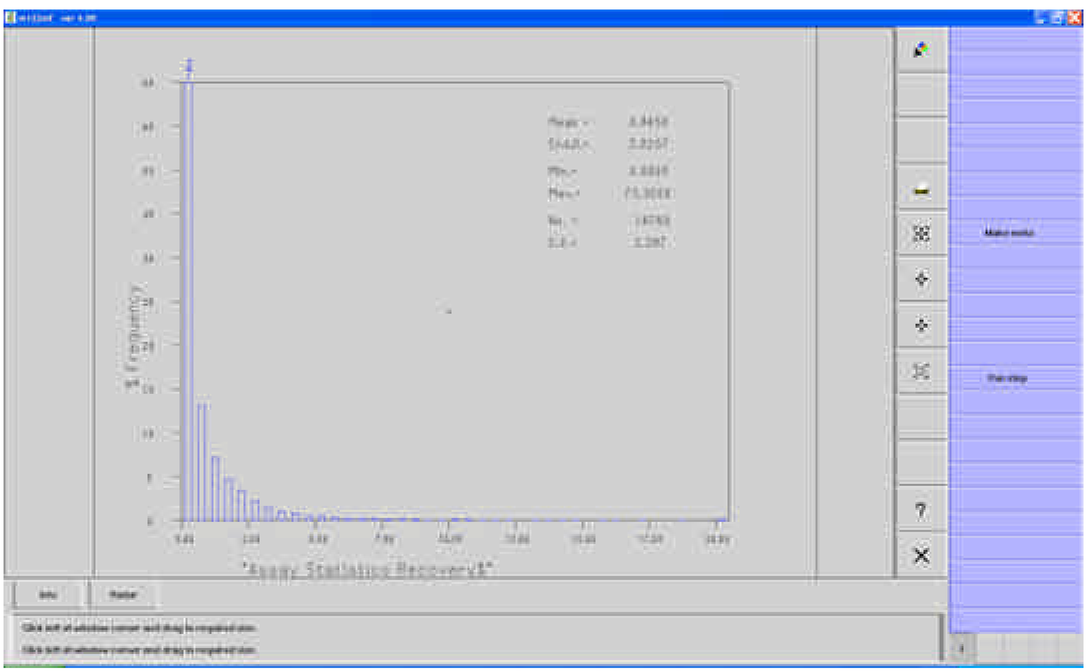


Figure A24: Assay Statistics – Au Grade of DDH Hole with Recovery of 0-29%

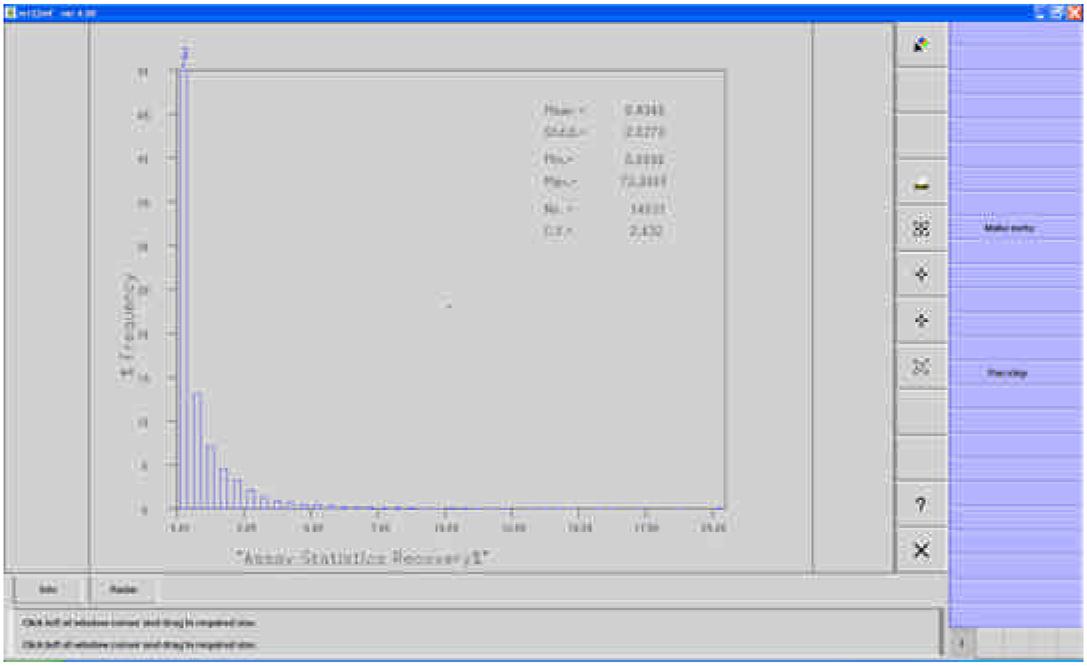


Figure A25: Assay Statistics – Au Grade of DDH Hole with Recovery of 0-39 %

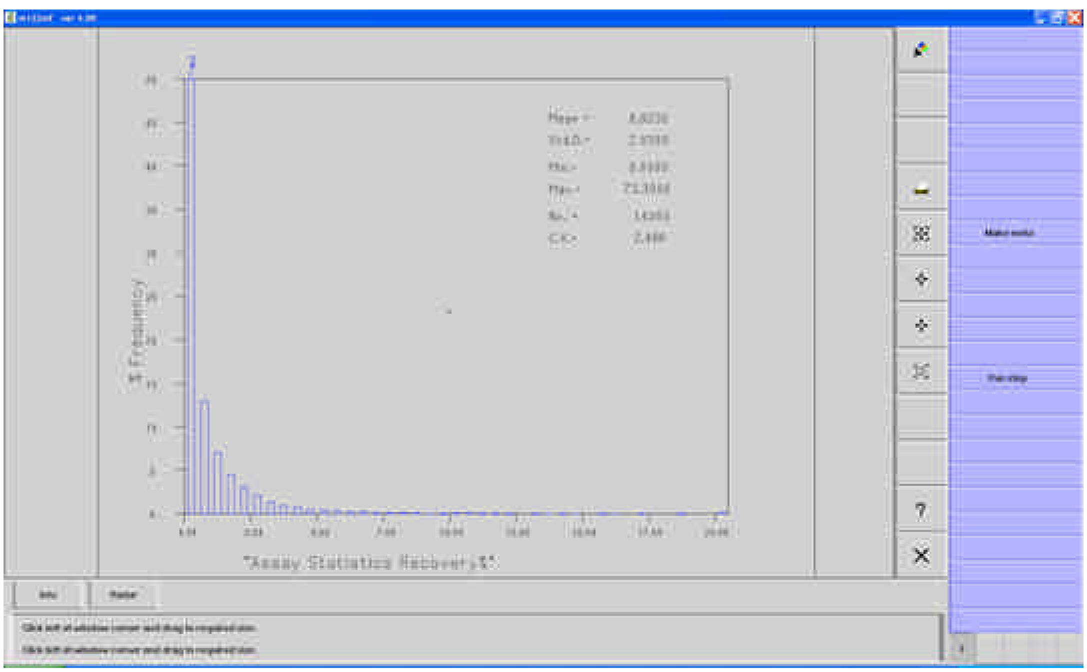


Figure A26: Assay Statistics – Au Grade of DDH Hole with Recovery of 0-49 %

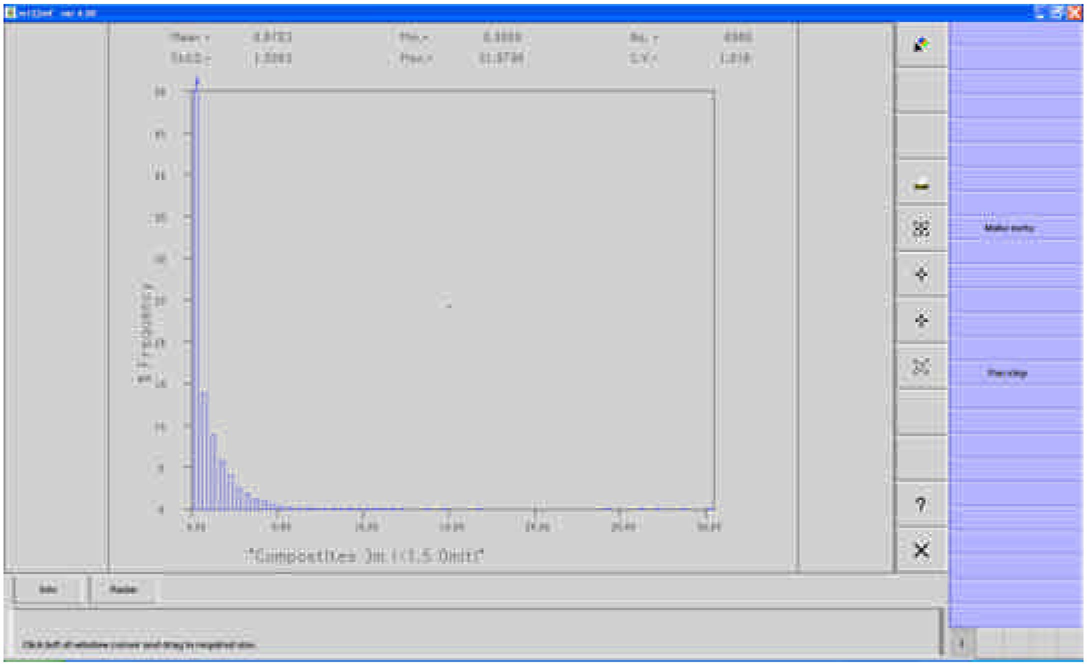


Figure A27: Composite Statistics – Omit Intervals < 1.5m

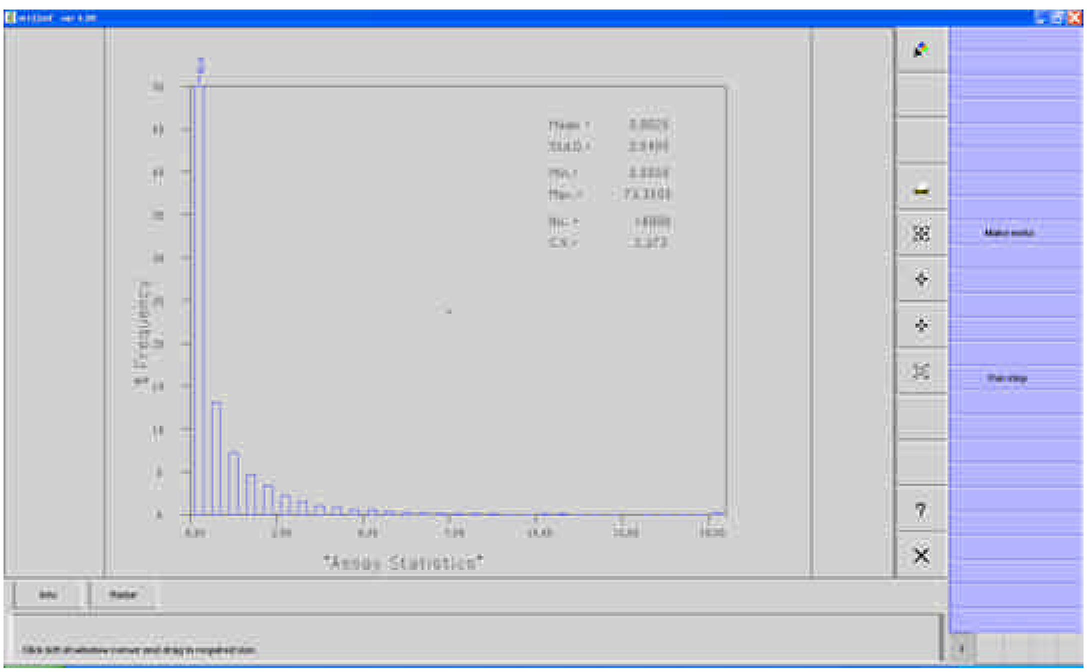


Figure A28: Assay Statistics for Comparison against A29.

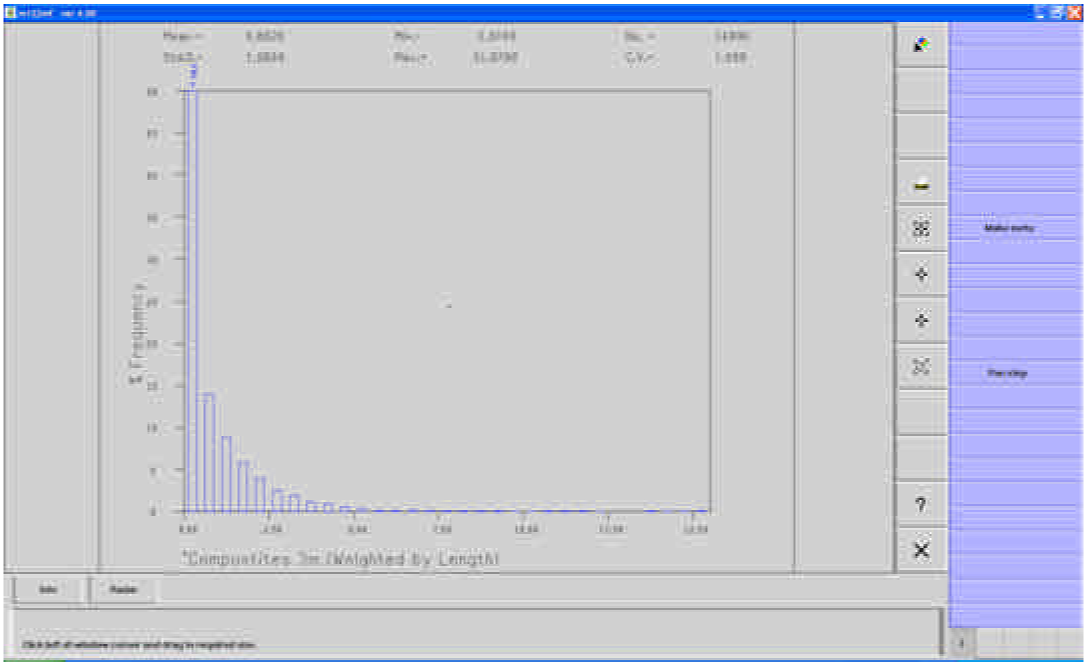


Figure A29: Composite Statistics for Comparison against A28.

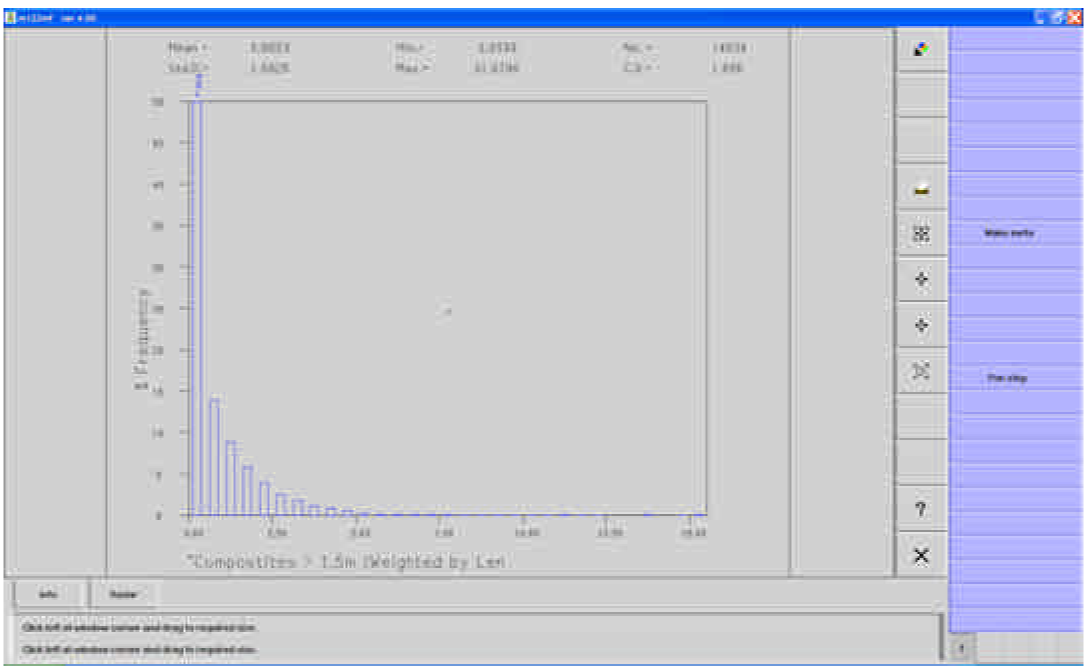


Figure A30: Composite Statistics for Lengths >1.5m for Comparison against A28.

APPENDIX 4

EXPLANATION OF SEMIVARIOGRAMS

In nature, the grade or value of a particular sample in three dimensional space is expected to be affected by its position and its relationship with its neighbours, (i.e. mineralization is not usually random and is influenced by such things as rock porosity, fracturing, distance from source etc.). The fundamental principal behind geostatistics, developed by George Matheron, takes this dependence into account and is known as the theory of regionalized variables. The procedure or tool to quantify both the amount and direction of this dependence is called the semivariogram.

A semivariogram is the fundamental autocorrelation tool of geostatistical procedures. It is defined as half of the mean squared difference of a variable for values separated by a distance h as given by the formula:

$$?(h) = \frac{\overset{n}{\underset{i=1}{?}} (x_i - (x_{i+h}))^2}{2n}$$

where,	$??(h)$	is the semivariogram
	x_i	is the value at location i
	x_{i+h}	is the value at a distance h from i
and	n	is the number of $x_i - (x_{i+h})$ pairs

Gamma (h) is a 3-dimensional function, commonly dependent on direction within a deposit which can also differ from one geological environment to another. An experimental semivariogram is determined from a set of experimental data (e.g. assay values at known locations) and is shown graphically as a plot of gamma (h) versus h (lag or sample spacing). For practical applications a smooth mathematical model is fitted to the normally saw-toothed graph of an experimental semi-variogram. The most common form of mathematical model in general is the spherical or Matheron Model given by the formula:

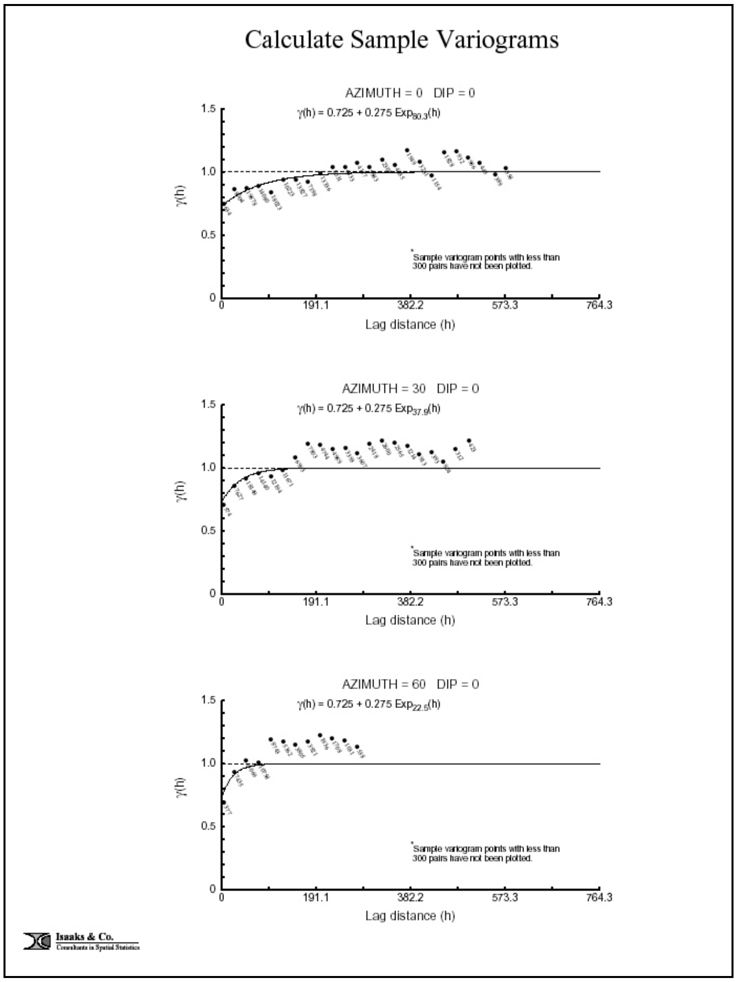
$$??(h) = C_o + C_1 (1.5\, h/a - .5\, h^3 / a^3) \quad \text{for } h\ ?a$$
$$= C_o + C_1 \quad \text{for } h > a$$

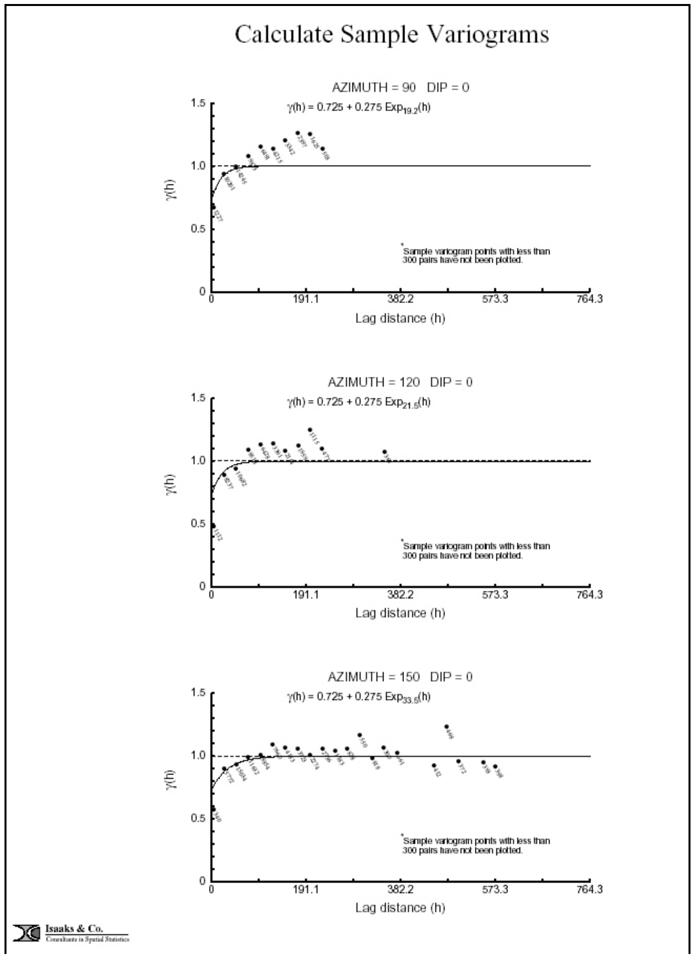
where	C_o	is the nugget effect
	C_1	is the structural component
	$C_o + C_1$	is the sill
	a	is the range (or influence of samples)
and	h	is the lag or sample spacing

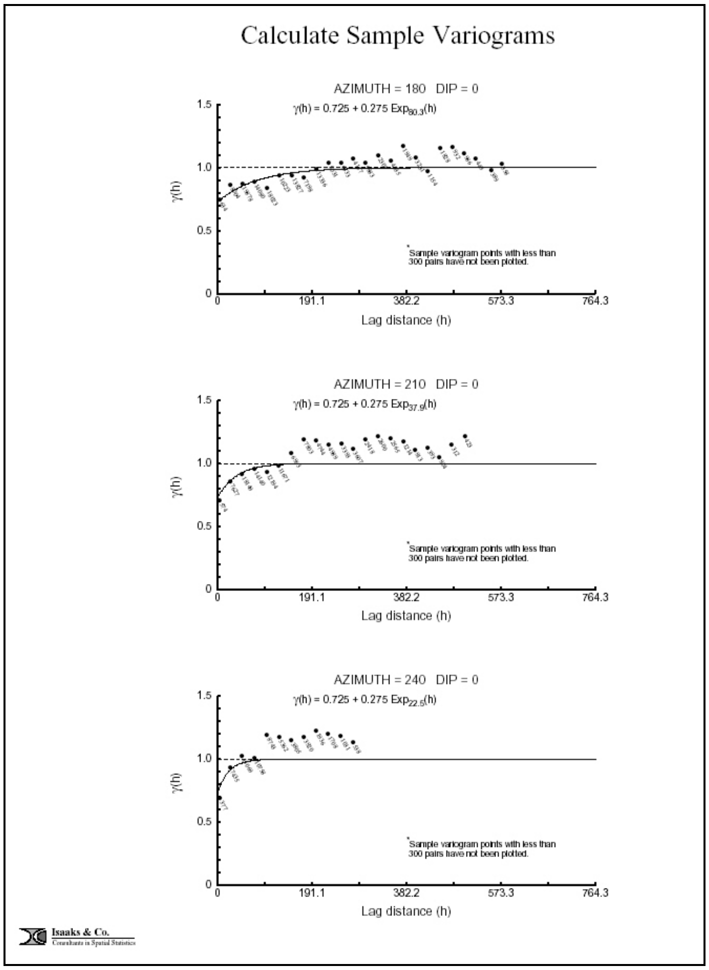
In many cases where the value of $??(h)$ increases systematically with grade it is convenient to determine a relative semivariogram in which $??(h)/m^2$ is plotted versus h, where m is the mean value of all samples used to determine an experimental semivariogram. In this way two (or more) semivariograms determined for different data sets (with different mean values) become more-or-less equivalent.

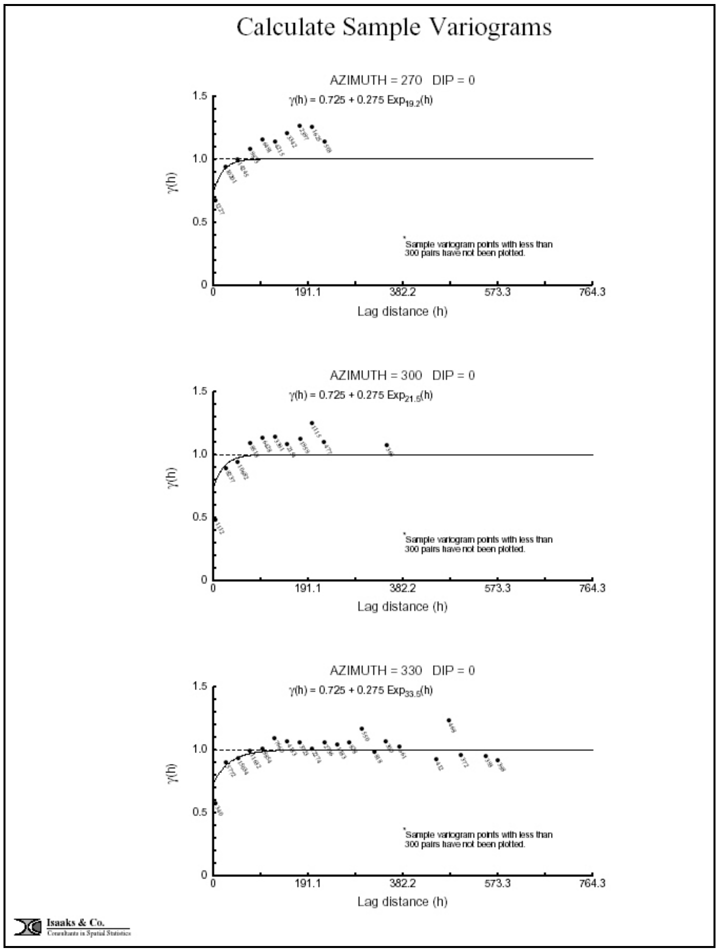
APPENDIX 5

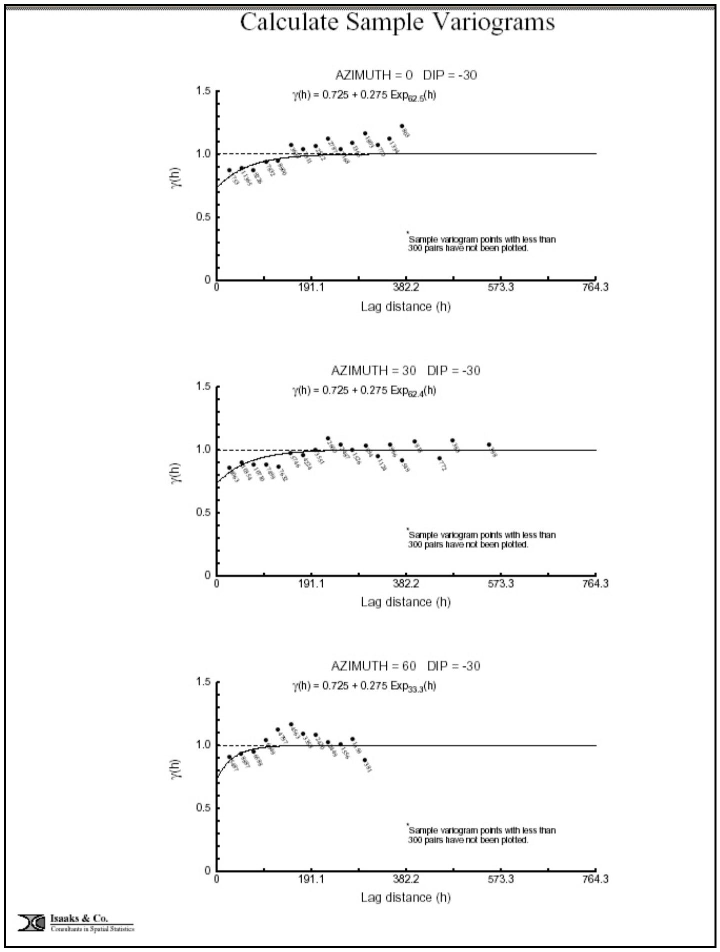
AU EXPERIMENTAL CORRELOGRAMS

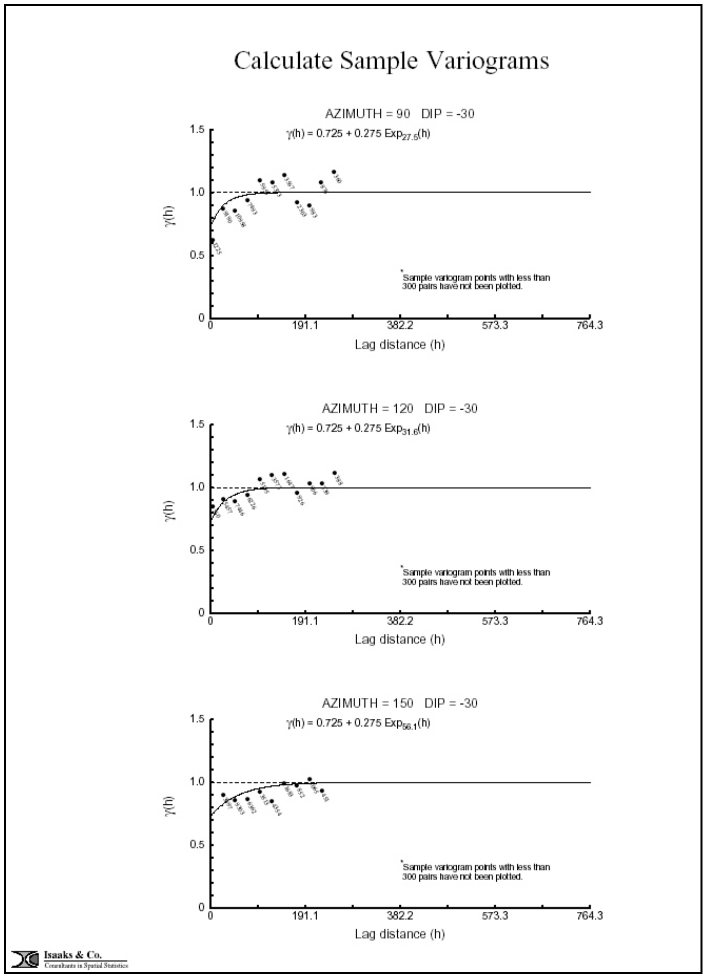


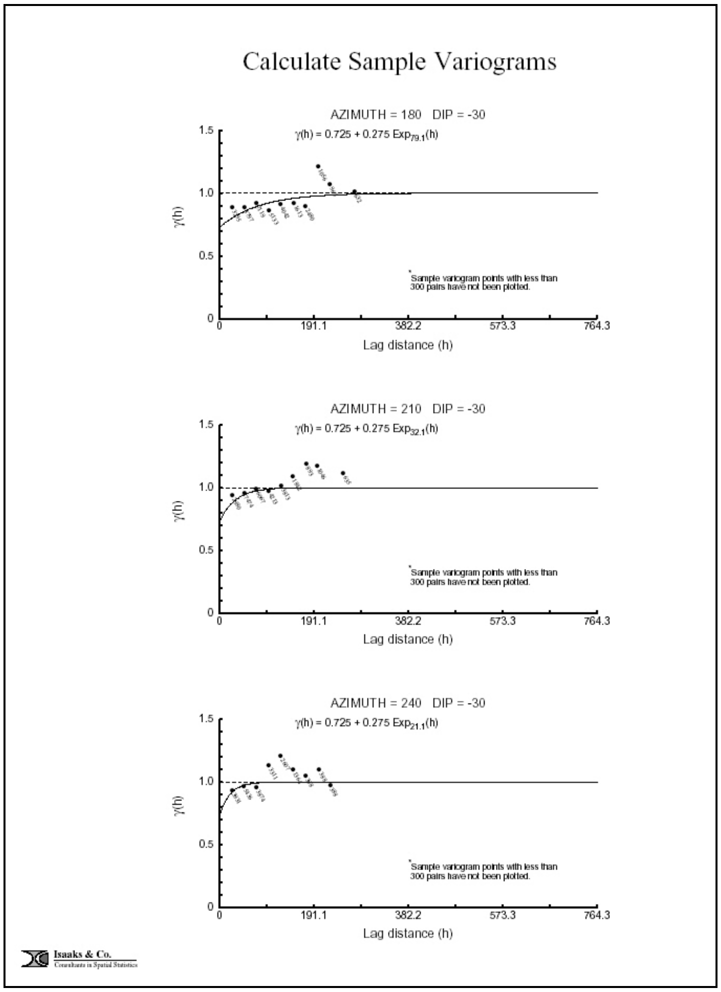

Calculate Sample Variograms
AZIMUTH = 90 DIP = 0
γ(h) = 0.725 + 0.275 Exp19.2(h)
Sample variogram points with less than 300 pairs have not been plotted.
AZIMUTH = 120 DIP = 0
γ(h) = 0.725 + 0.275 Exp21.5(h)
Sample variogram points with less than 300 pairs have not been plotted.
AZIMUTH = 150 DIP = 0
γ(h) = 0.725 + 0.275 Exp33.5(h)
Sample variogram points with less than 300 pairs have not been plotted.
γ(h)
Lag distance (h)
0
191.1
382.2
573.3
764.3
1.5
1.0
0.5
Isaaks & Co.

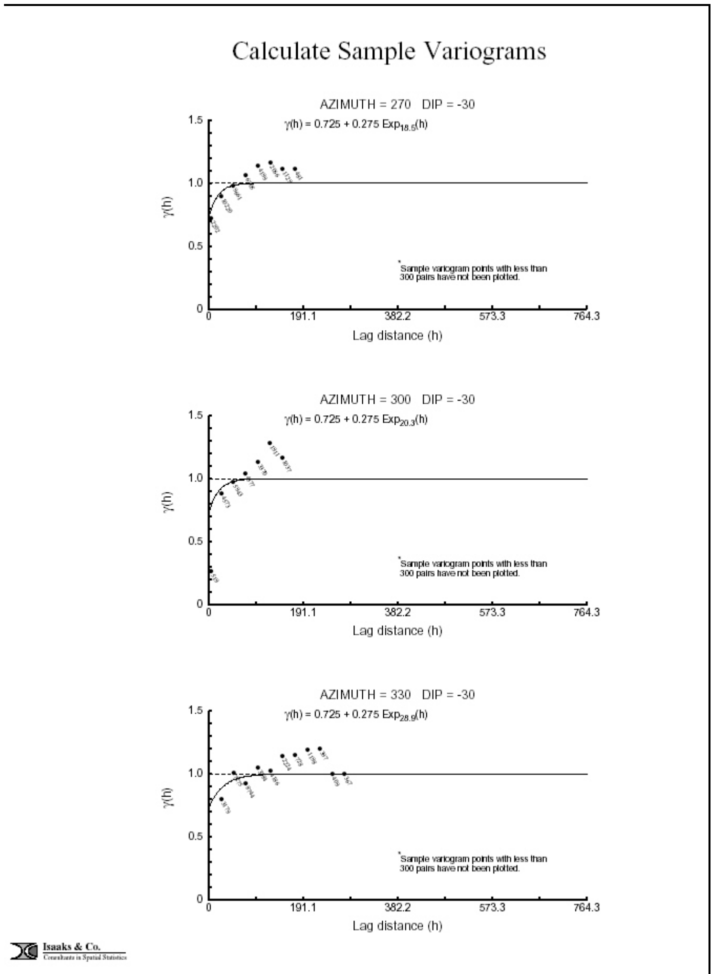

Calculate Sample Variograms
AZIMUTH = 180 DIP = 0
γ(h) = 0.725 + 0.275 Exp80.3(h)
γ(h)
Lag distance (h)
Sample variogram points with less than 300 pairs have not been plotted.
AZIMUTH = 210 DIP = 0
γ(h) = 0.725 + 0.275 Exp37.9(h)
γ(h)
Lag distance (h)
Sample variogram points with less than 300 pairs have not been plotted.
AZIMUTH = 240 DIP = 0
γ(h) = 0.725 + 0.275 Exp22.5(h)
γ(h)
Lag distance (h)
Sample variogram points with less than 300 pairs have not been plotted.
Isaaks & Co.
Consultants in Spatial Statistics

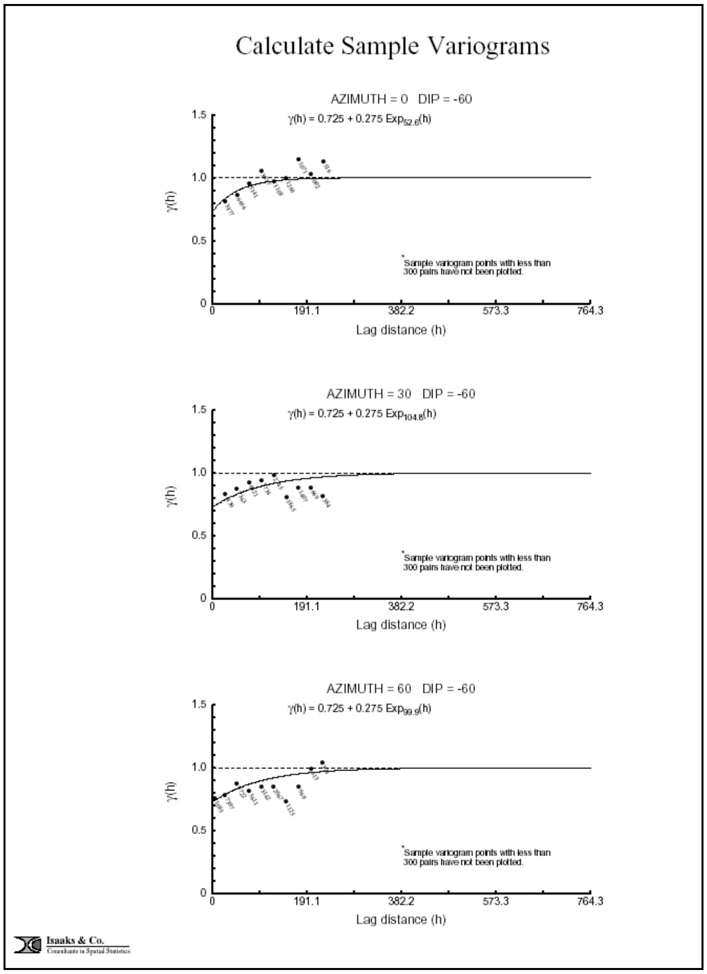

Calculate Sample Variograms
AZIMUTH = 270 DIP = 0
γ(h) = 0.725 + 0.275 Exp19.2(h)
*Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
γ(h)
AZIMUTH = 300 DIP = 0
γ(h) = 0.725 + 0.275 Exp21.5(h)
*Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
AZIMUTH = 330 DIP = 0
γ(h) = 0.725 + 0.275 Exp33.5(h)
*Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
0
191.1
382.2
573.3
764.3
1.5
1.0
0.5
Isaaks & Co.
Consultants in Spatial Statistics

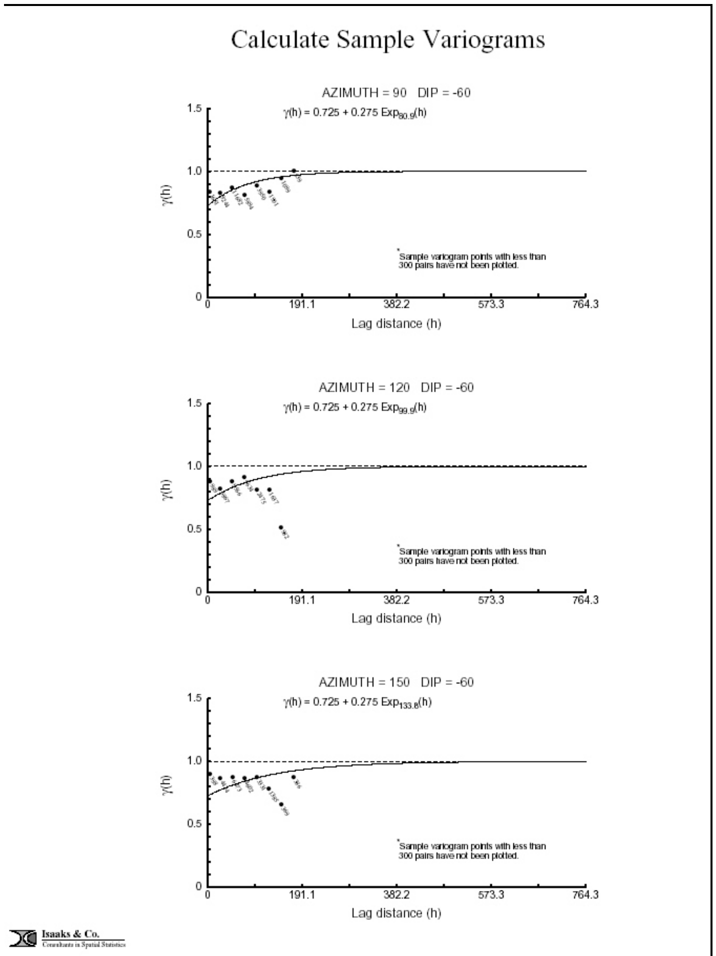

Calculate Sample Variograms
AZIMUTH = 0 DIP = -30
γ(h) = 0.725 + 0.275 Exp82.5(h)
AZIMUTH = 30 DIP = -30
γ(h) = 0.725 + 0.275 Exp82.4(h)
AZIMUTH = 60 DIP = -30
γ(h) = 0.725 + 0.275 Exp33.3(h)
*Sample variogram points with less than 300 pairs have not been plotted.
γ(h)
Lag distance (h)
0
191.1
382.2
573.3
764.3
0.5
1.0
1.5
Isaaks & Co.
Consultants in Spatial Statistics

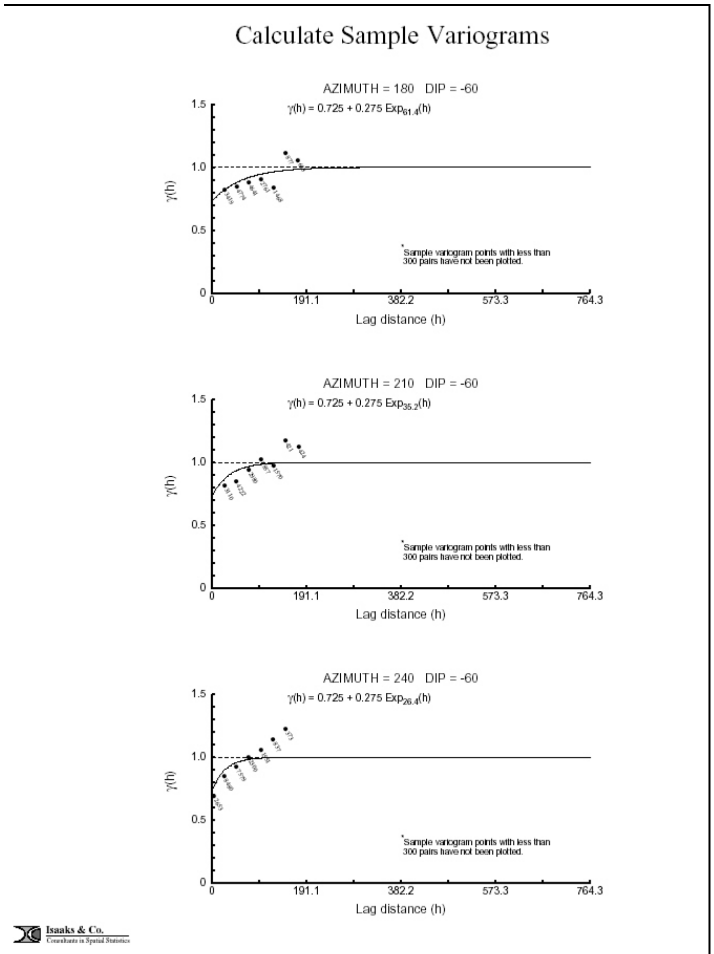

Calculate Sample Variograms
AZIMUTH = 90 DIP = -30
γ(h) = 0.725 + 0.275 Exp27.5(h)
AZIMUTH = 120 DIP = -30
γ(h) = 0.725 + 0.275 Exp31.6(h)
AZIMUTH = 150 DIP = -30
γ(h) = 0.725 + 0.275 Exp56.1(h)
γ(h)
Lag distance (h)
Sample variogram points with less than 300 pairs have not been plotted.
Isaaks & Co.

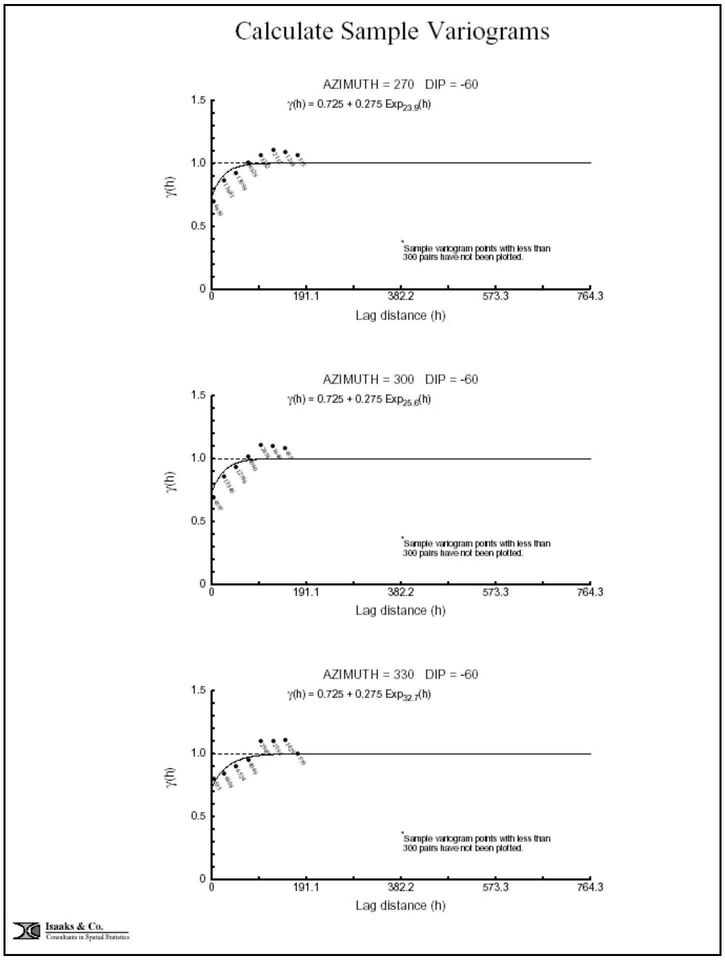

Calculate Sample Variograms
AZIMUTH = 180 DIP = -30
γ(h) = 0.725 + 0.275 Exp79.1(h)
Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
AZIMUTH = 210 DIP = -30
γ(h) = 0.725 + 0.275 Exp32.1(h)
Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
AZIMUTH = 240 DIP = -30
γ(h) = 0.725 + 0.275 Exp21.1(h)
Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
Isaaks & Co.
Consultants in Spatial Statistics


Calculate Sample Variograms
AZIMUTH = 270 DIP = -30
γ(h) = 0.725 + 0.275 Exp18.5(h)
Sample variogram points with less than 300 pairs have not been plotted.
AZIMUTH = 300 DIP = -30
γ(h) = 0.725 + 0.275 Exp20.3(h)
Sample variogram points with less than 300 pairs have not been plotted.
AZIMUTH = 330 DIP = -30
γ(h) = 0.725 + 0.275 Exp28.9(h)
Sample variogram points with less than 300 pairs have not been plotted.
γ(h)
Lag distance (h)
0
191.1
382.2
573.3
764.3
0.5
1.0
1.5
Isaaks & Co.
Consultants in Spatial Statistics

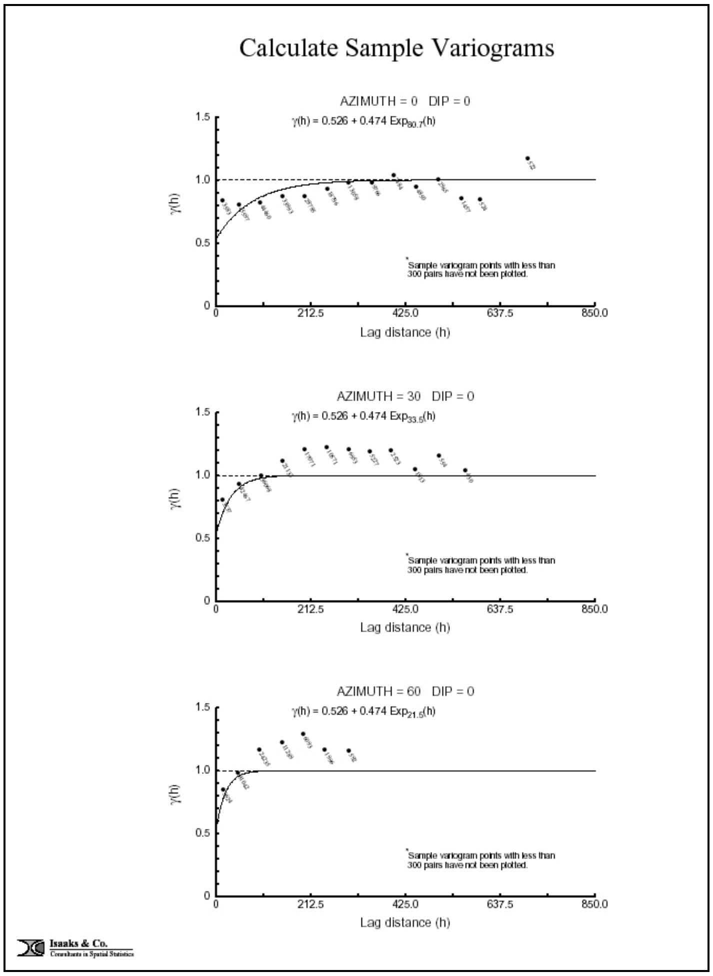

Calculate Sample Variograms
AZIMUTH = 0 DIP = -60
γ(h) = 0.725 + 0.275 Exp52.6(h)
*Sample variogram points with less than 300 pairs have not been plotted.
γ(h)
Lag distance (h)
AZIMUTH = 30 DIP = -60
γ(h) = 0.725 + 0.275 Exp104.8(h)
*Sample variogram points with less than 300 pairs have not been plotted.
γ(h)
Lag distance (h)
AZIMUTH = 60 DIP = -60
γ(h) = 0.725 + 0.275 Exp99.9(h)
*Sample variogram points with less than 300 pairs have not been plotted.
γ(h)
Lag distance (h)
Isaaks & Co.

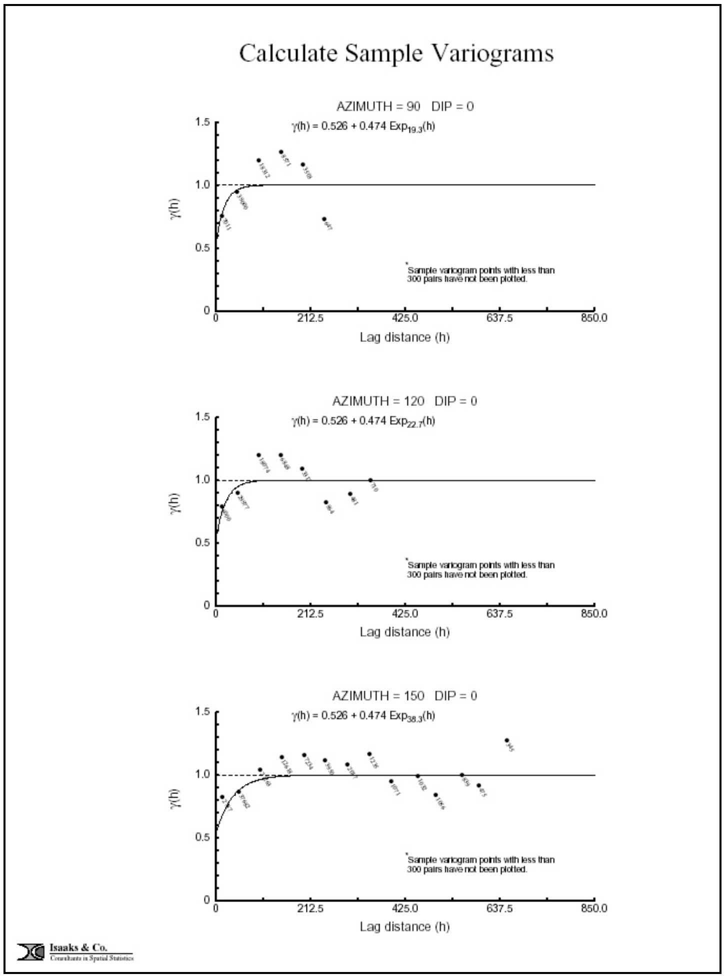

Calculate Sample Variograms
AZIMUTH = 90 DIP = -60
γ(h) = 0.725 + 0.275 Exp80.9(h)
Sample variogram points with less than 300 pairs have not been plotted.
AZIMUTH = 120 DIP = -60
γ(h) = 0.725 + 0.275 Exp99.9(h)
Sample variogram points with less than 300 pairs have not been plotted.
AZIMUTH = 150 DIP = -60
γ(h) = 0.725 + 0.275 Exp133.8(h)
Sample variogram points with less than 300 pairs have not been plotted.
γ(h)
1.5
1.0
0.5
0
0
191.1
382.2
573.3
764.3
Lag distance (h)
Isaaks & Co.
Consultants in Spatial Statistics

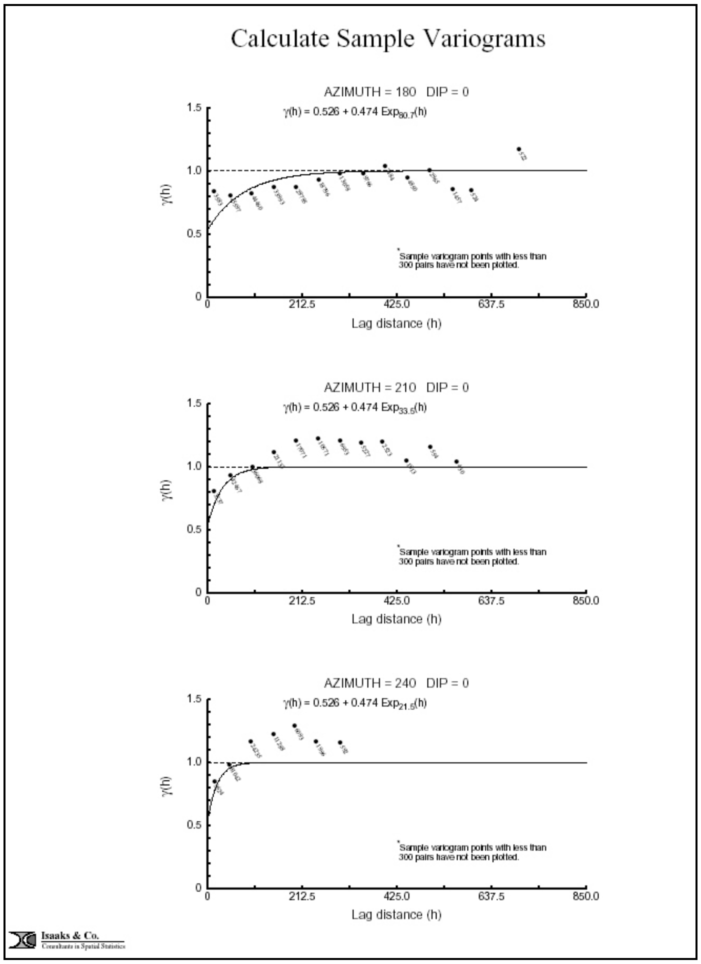

Calculate Sample Variograms
AZIMUTH = 180 DIP = -60
γ(h) = 0.725 + 0.275 Exp61.4(h)
γ(h)
Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
0 191.1 382.2 573.3 764.3
AZIMUTH = 210 DIP = -60
γ(h) = 0.725 + 0.275 Exp35.2(h)
γ(h)
Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
AZIMUTH = 240 DIP = -60
γ(h) = 0.725 + 0.275 Exp26.4(h)
γ(h)
Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
Isaaks & Co.
Consultants in Spatial Statistics

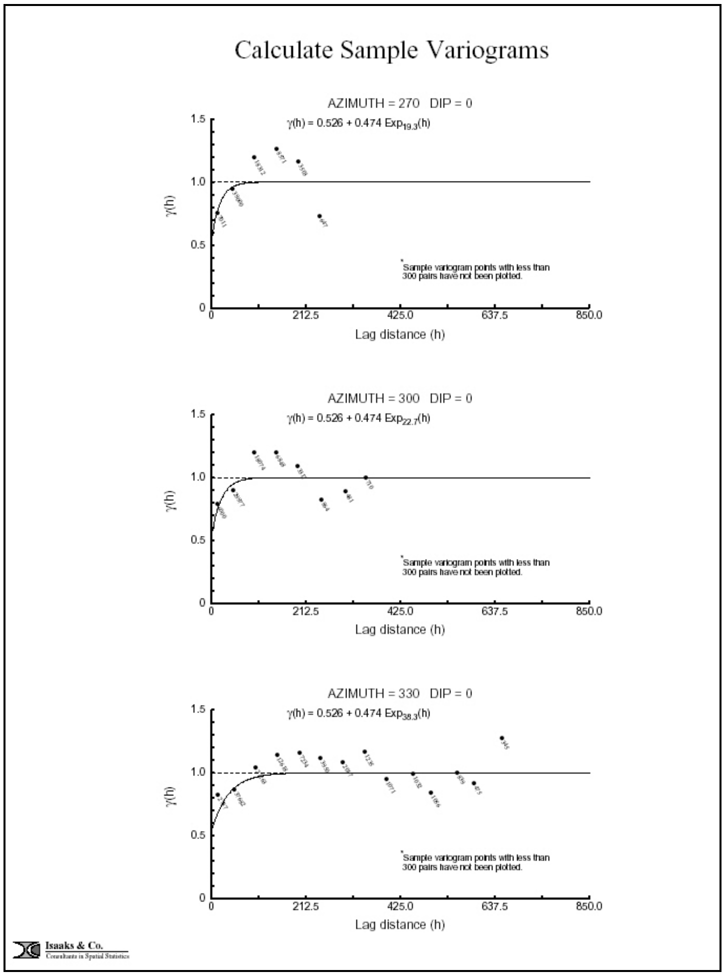

Calculate Sample Variograms
AZIMUTH = 270 DIP = -60
γ(h) = 0.725 + 0.275 Exp23.9(h)
AZIMUTH = 300 DIP = -60
γ(h) = 0.725 + 0.275 Exp25.6(h)
AZIMUTH = 330 DIP = -60
γ(h) = 0.725 + 0.275 Exp32.7(h)
*Sample variogram points with less than 300 pairs have not been plotted.
γ(h)
Lag distance (h)
0
191.1
382.2
573.3
764.3
0.5
1.0
1.5
Isaaks & Co.
Consultants in Spatial Statistics

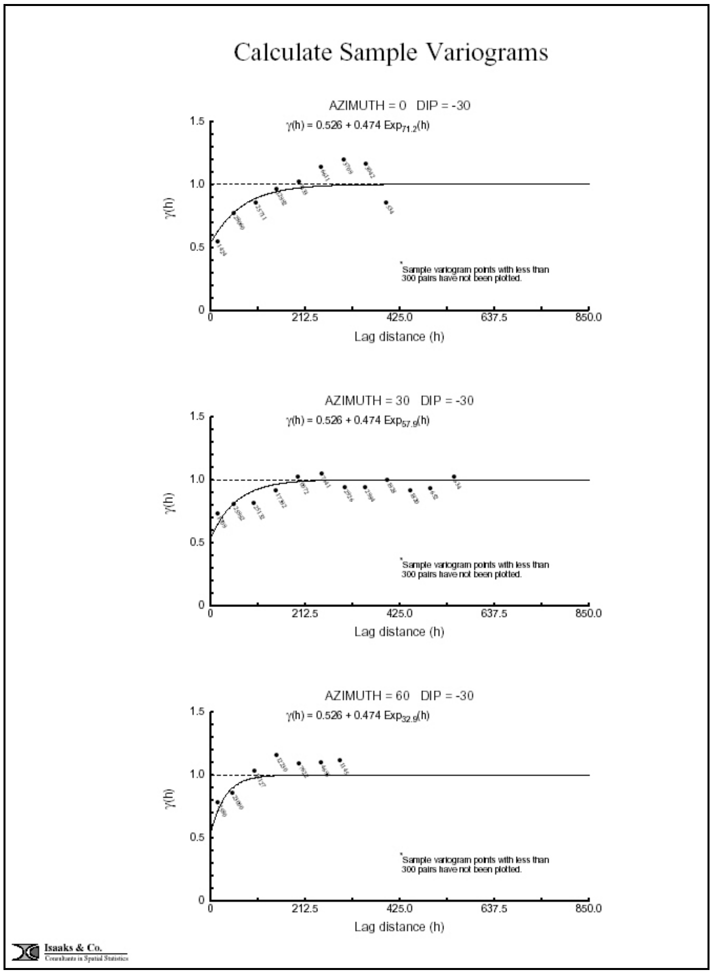
AG EXPERIMENTAL CORRELOGRAMS

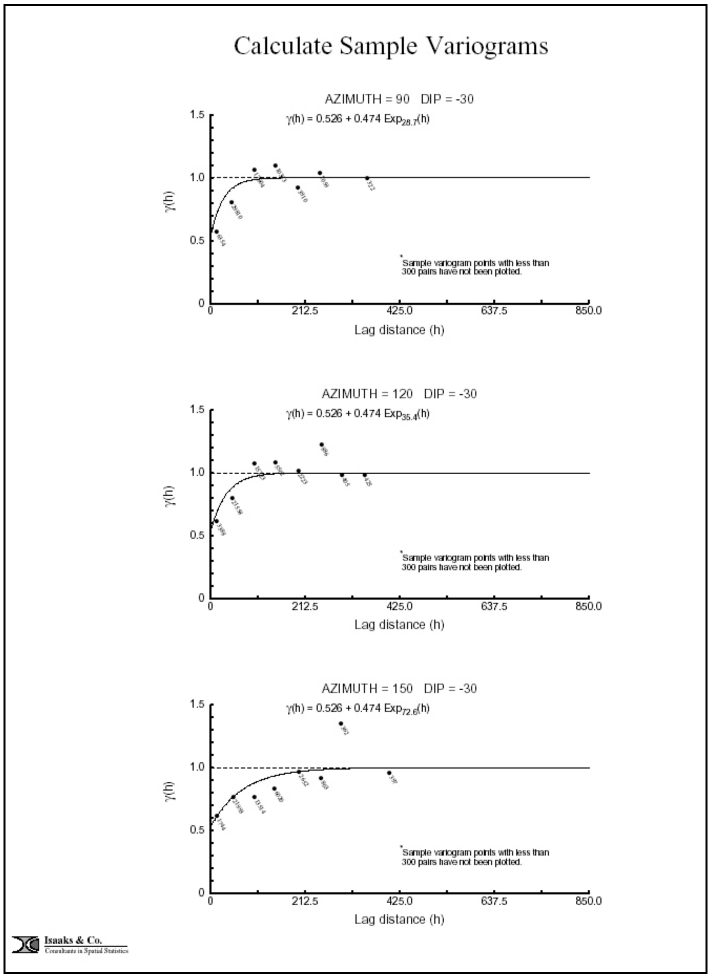


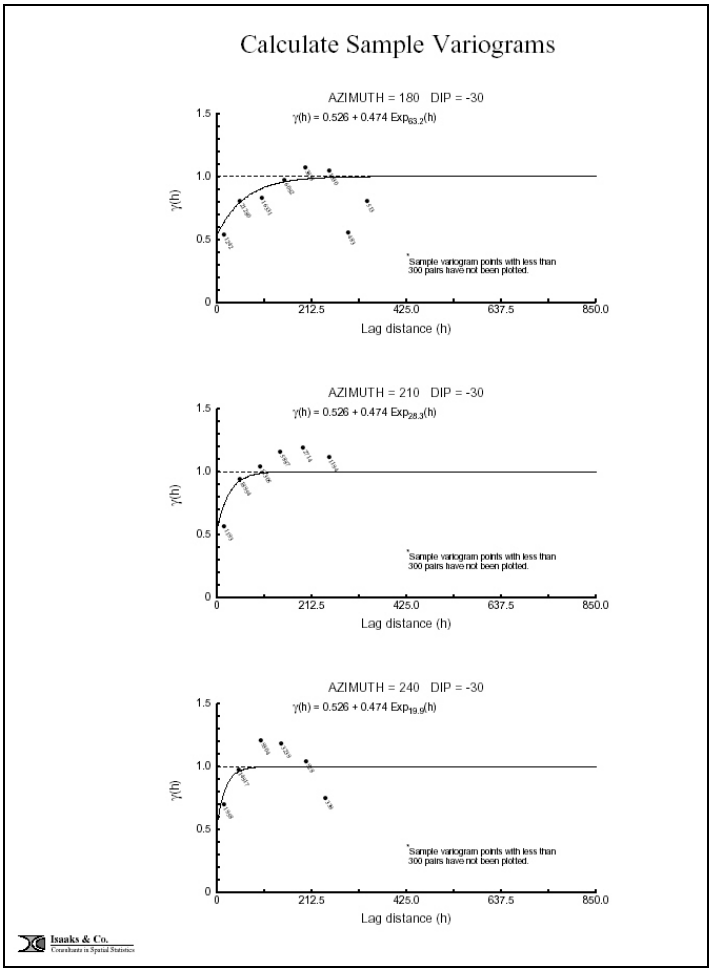

Calculate Sample Variograms
AZIMUTH = 90 DIP = 0
γ(h) = 0.526 + 0.474 Exp19.3(h)
Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
γ(h)
AZIMUTH = 120 DIP = 0
γ(h) = 0.526 + 0.474 Exp22.7(h)
Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
AZIMUTH = 150 DIP = 0
γ(h) = 0.526 + 0.474 Exp38.3(h)
Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
Isaaks & Co.
Consultants in Spatial Statistics

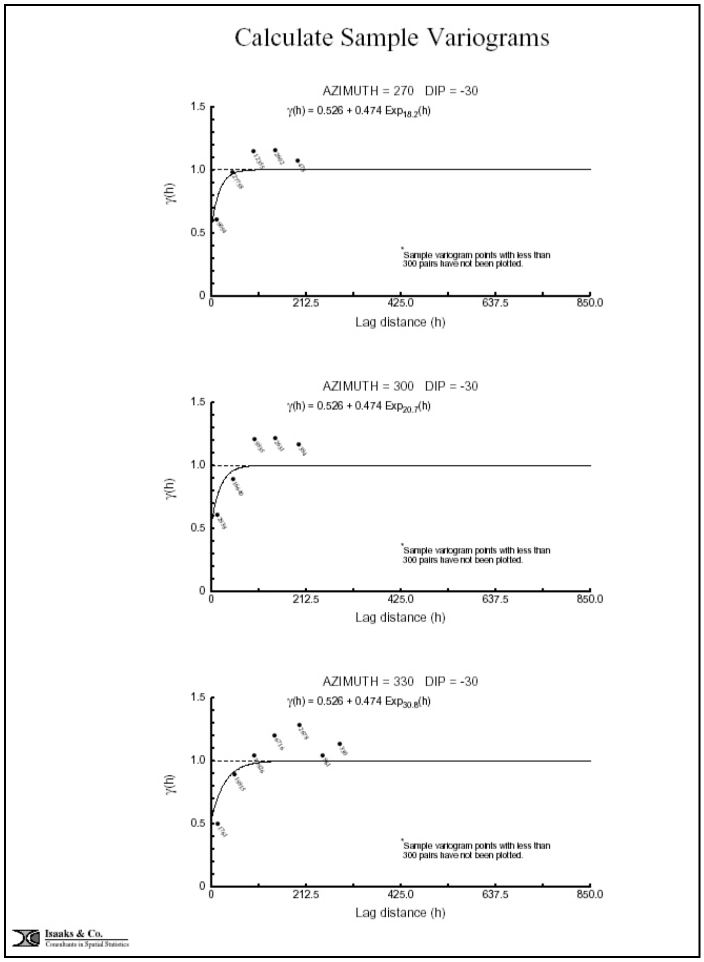

Calculate Sample Variograms
AZIMUTH = 180 DIP = 0
γ(h) = 0.526 + 0.474 Exp80.7(h)
γ(h)
Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
AZIMUTH = 210 DIP = 0
γ(h) = 0.526 + 0.474 Exp33.5(h)
γ(h)
Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
AZIMUTH = 240 DIP = 0
γ(h) = 0.526 + 0.474 Exp21.5(h)
γ(h)
Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
Isaaks & Co.


Calculate Sample Variograms
AZIMUTH = 270 DIP = 0
γ(h) = 0.526 + 0.474 Exp19.3(h)
γ(h)
1.5
1.0
0.5
0
0
212.5
425.0
637.5
850.0
Lag distance (h)
Sample variogram points with less than 300 pairs have not been plotted.
AZIMUTH = 300 DIP = 0
γ(h) = 0.526 + 0.474 Exp22.7(h)
Sample variogram points with less than 300 pairs have not been plotted.
AZIMUTH = 330 DIP = 0
γ(h) = 0.526 + 0.474 Exp38.3(h)
Sample variogram points with less than 300 pairs have not been plotted.
Isaaks & Co.
Consultants in Spatial Statistics

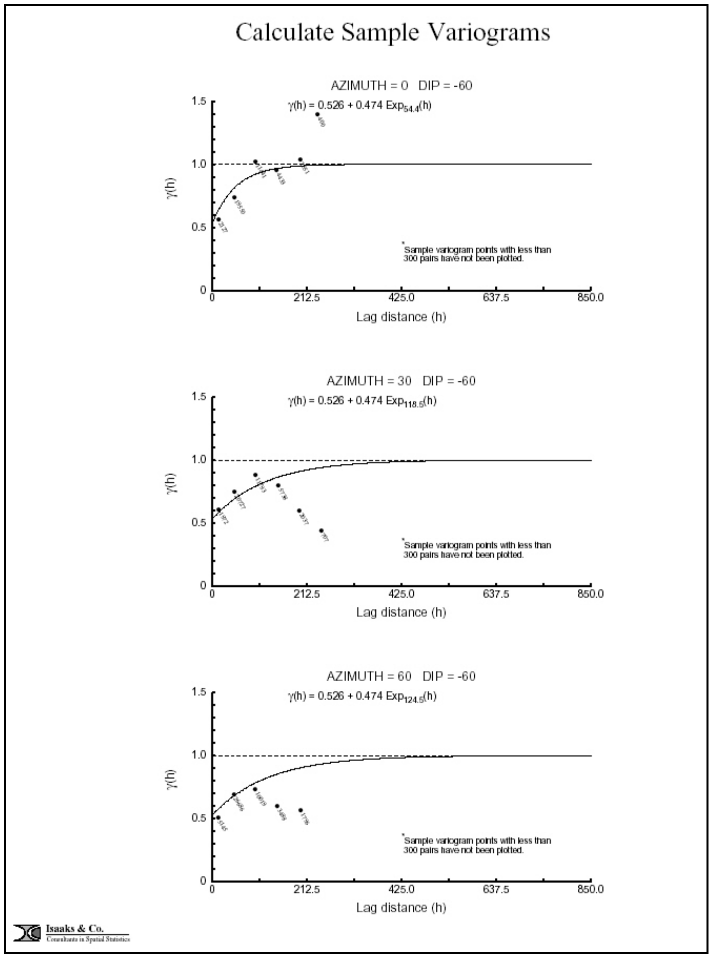

Calculate Sample Variograms
AZIMUTH = 0 DIP = -30
γ(h) = 0.526 + 0.474 Exp71.2(h)
*Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
γ(h)
AZIMUTH = 30 DIP = -30
γ(h) = 0.526 + 0.474 Exp57.9(h)
*Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
γ(h)
AZIMUTH = 60 DIP = -30
γ(h) = 0.526 + 0.474 Exp32.9(h)
*Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
γ(h)
Isaaks & Co.


Calculate Sample Variograms
AZIMUTH = 90 DIP = -30
γ(h) = 0.526 + 0.474 Exp28.7(h)
AZIMUTH = 120 DIP = -30
γ(h) = 0.526 + 0.474 Exp35.4(h)
AZIMUTH = 150 DIP = -30
γ(h) = 0.526 + 0.474 Exp72.6(h)
*Sample variogram points with less than 300 pairs have not been plotted.
γ(h)
Lag distance (h)
0
212.5
425.0
637.5
850.0
0.5
1.0
1.5
Isaaks & Co.


Calculate Sample Variograms
AZIMUTH = 180 DIP = -30
γ(h) = 0.526 + 0.474 Exp63.2(h)
AZIMUTH = 210 DIP = -30
γ(h) = 0.526 + 0.474 Exp28.3(h)
AZIMUTH = 240 DIP = -30
γ(h) = 0.526 + 0.474 Exp19.9(h)
γ(h)
Lag distance (h)
0
212.5
425.0
637.5
850.0
1.5
1.0
0.5
*Sample variogram points with less than 300 pairs have not been plotted.
Isaaks & Co.



Calculate Sample Variograms
AZIMUTH = 270 DIP = -30
γ(h) = 0.526 + 0.474 Exp18.2(h)
Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
AZIMUTH = 300 DIP = -30
γ(h) = 0.526 + 0.474 Exp20.7(h)
Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
AZIMUTH = 330 DIP = -30
γ(h) = 0.526 + 0.474 Exp30.8(h)
Sample variogram points with less than 300 pairs have not been plotted.
Lag distance (h)
Isaaks & Co.


Calculate Sample Variograms
AZIMUTH = 0 DIP = -60
γ(h) = 0.526 + 0.474 Exp54.4(h)
AZIMUTH = 30 DIP = -60
γ(h) = 0.526 + 0.474 Exp118.5(h)
AZIMUTH = 60 DIP = -60
γ(h) = 0.526 + 0.474 Exp124.5(h)
*Sample variogram points with less than 300 pairs have not been plotted.
γ(h)
Lag distance (h)
0
212.5
425.0
637.5
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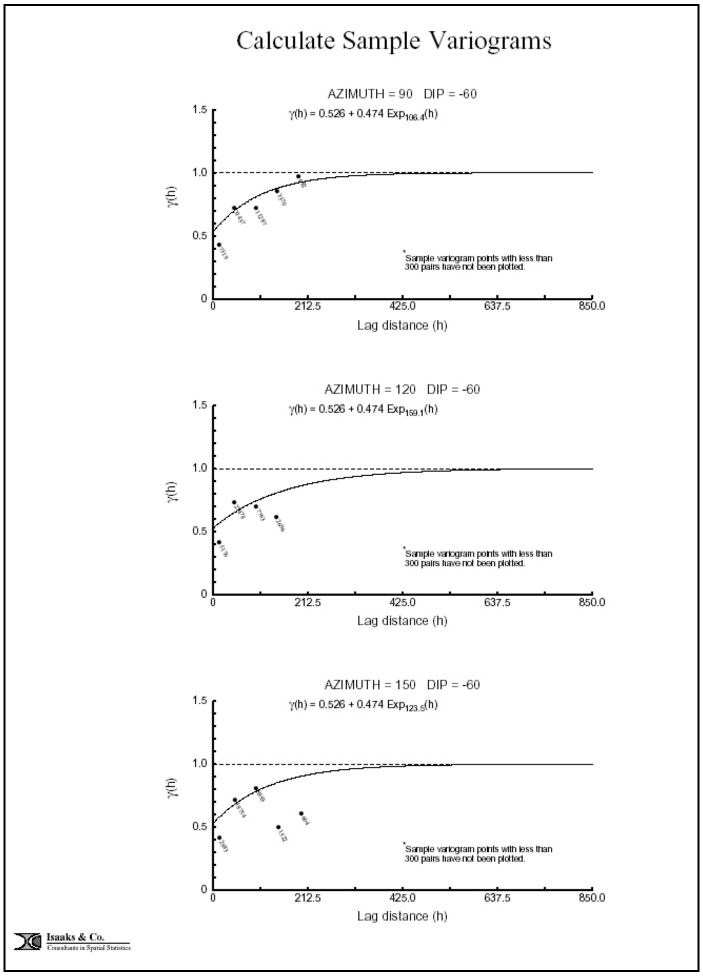
Isaaks & Co.
Consultants in Spatial Statistics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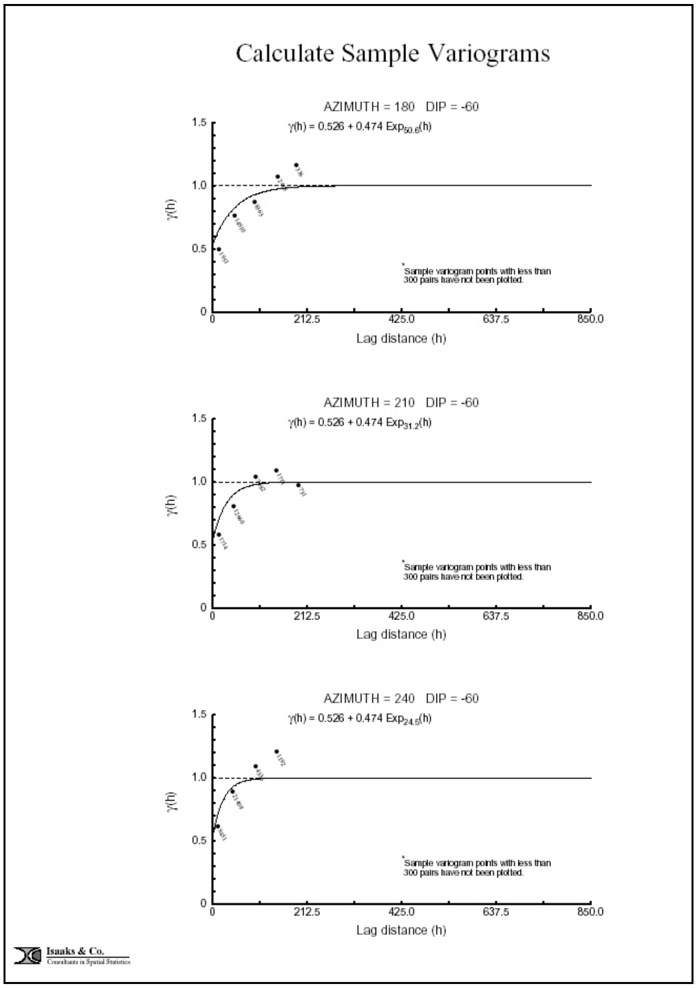
Isaaks & Co.
Consultants in Spatial Statistics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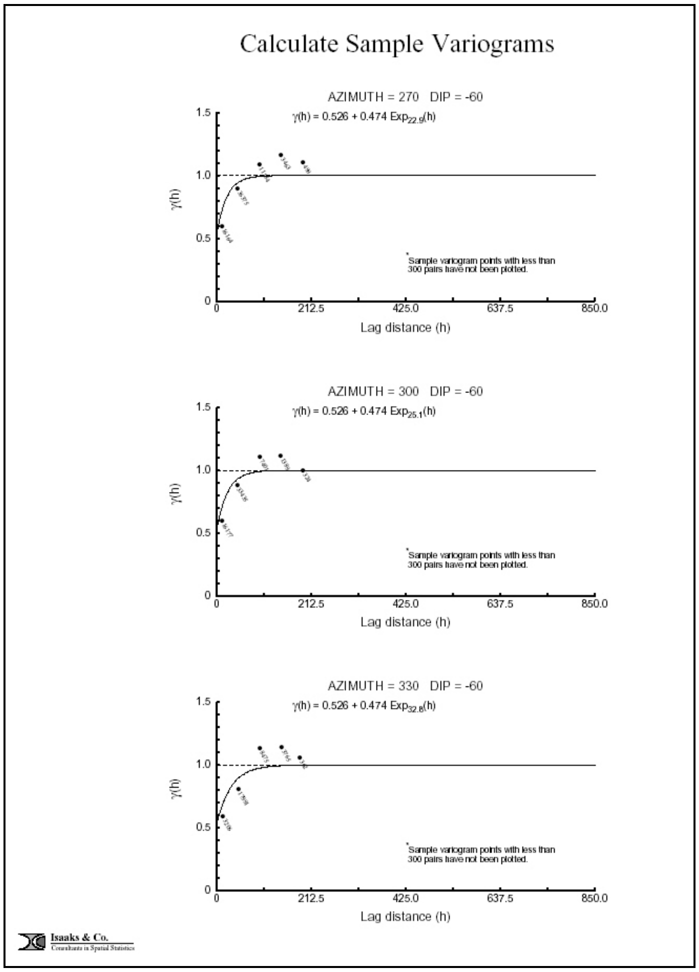
Isaaks & Co.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Lag distance (h)
Isaaks & Co.

APPENDIX 6

CLASSIFICATION OF RESOURCES

Introduction

"*In this Instrument, the terms "mineral resource", "inferred mineral resource", "indicated mineral resource" and "measured mineral resource" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by CIM Council on August 20, 2000, as those definitions may be amended from time to time by the Canadian Institute of Mining, Metallurgy and Petroleum."*

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

The terms Measured, Indicated and Inferred are defined as follows:

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

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

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

APPENDIX 7

RESOURCES

Table A7.1: Measured Resources

MEASURED							
Cut-Off Grade g/t Au	Tonnes	Au g/t	Ag g/t	AuEq g/t	Au Ounces	Ag Ounces	Au Equivalent Ounces
0.5	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	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	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	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	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	97,307	6.33	33.99	6.84	19,813	106,338	21,405

AuEq is based upon 66.7:1 Au:Ag

Table A7.2: Indicated Resources

INDICATED							
Cut-Off Grade g/t Au	Tonnes	Au g/t	Ag g/t	AuEq g/t	Au Ounces	Ag Ounces	Au Equivalent Ounces
0.5	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	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	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	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	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	146,616	6.25	25.17	6.63	29,471	118,628	31,248

AuEq is based upon 66.7:1 Au:Ag

Table A7.3: Measured and Indicated Resources

MEASURED + INDICATED							
Cut-off Grade g/t Au	Tonnes	Au g/t	Ag g/t	AuEq g/t	Au Ounces	Ag Ounces	Au Equivalent Ounces
0.5	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	958,773	7,727,907	1,074,242
1	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	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	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	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	243,923	6.28	28.69	6.71	49,284	224,966	52,653

AuEq is based upon 66.7:1 Au:Ag

Table A7.4: Inferred Resources

INFERRED							
Cut-Off Grade g/t Au	Tonnes	Au g/t	Ag g/t	AuEq g/t	Au Ounces	Ag Ounces	Au Equivalent Ounces
0.5	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	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	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	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	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	27,332	6.16	17.17	6.42	5,417	15,084	5,642

AuEq is based upon 66.7:1 Au:Ag

TECHNICAL REPORT
FOR PRELIMARY ASSESSMENT OF THE JAHODNA URANIUM PROJECT, SLOVAKIA

for
Tournigan Gold Corporation
Suite 301, 700 West Pender Street,
Vancouver, B.C. V6C 1G8
CANADA

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ACA HOWE INTERNATIONAL LTD

May 11, 2006

Berkhamsted
Herts, UK

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1. SUMMARY

ACA Howe International Ltd, of UK, was commissioned by Tournigan Gold Corporation in October 2005, to undertake a resource estimate and prepare a technical report in accordance with Canadian National Instrument 43-101, on the Jahodna uranium project in Slovakia. The author of the geological parts of this report was contracted by ACA Howe International to calculate the estimate, and compile and write this report. Included in this report under item 20 is a further report entitled Jahodna Preliminary Assessment dated 25 April 2006 by Julian Bennett and Dr R Balakrishnan. This covers the operational assessment of a possible future operation.

The first author visited Slovakia for approximately two weeks during October 2005. During this time, he spent time at the offices of Tournigan's 100%-owned subsidiary, Kremnica Gold, a.s. in Kremnica, Slovakia, and of Koral SRO in Spisska Nova Ves, Slovakia. A one day visit was made to the project area, to make geological ground traverses and examine drill core from the new drill hole named KG-J-1. Since returning to the UK, the author has compiled relevant data, assisted with various aspects of digital mapwork, and written the 43-101 report. Other resource calculation and map drafting work for this report was completed by geologists of ACA Howe International Ltd in Berkhamsted, UK.

No other site visits were made by ACA Howe International personnel.

Much of the content of this report relies on English translations of previous reports in Slovak. While it is felt that previous work undertaken in the area, in general, probably conformed in detail to the exploration and evaluation regulations in force in Slovakia at that time, there is probably much detail of past work that has not yet been translated into English, so it was difficult to obtain or evaluate for the purposes of this report.

In addition to these, conversations were had, through an interpreter, with a number of Slovak technical personnel who had previous involvement with the Jahodna project. Since the author only spent two weeks in Slovakia, and less than one day on site at Jahodna, heavy reliance has been placed on the various sources of data mentioned above.

The full title of the current exploration license that hosts the Jahodna project is titled "Cermel-Jahodna – U-Mo, Cu ores", and it was granted on March 21st 2005 by the Geology and Natural Resources Department at Ministry of the Environment of the Slovak Republic. The period of validity of the licence is four (4) years, and various conditions were attached to the granting of the exploration licence. Tournigan has 100% interest in the Cermel-Jahodna exploration licence. To the author's knowledge, the company has complied with all the required conditions.

The Jahodna property is situated in East-Central Slovakia. Topographically, the region forms part of the Western Carpathian mountain chain. The actual surface is undulating, with little or no outcrop and deep soil cover. The vegetation is a type of mature mixed woodland, being made up mostly of broadleaved types, but also lesser conifers. The forest is in fact part of a forestry reserve, though according to the Tournigan staff in Slovakia, this should not pose a prohibitive problem with regard to planning permission for exploration or mining development activities.

The Jahodna property is situated some 10-15 km NW of the large town of Košice, a regional centre in East-Central Slovakia. The Jahodna property is relatively accessible, and lies quite close to the regional main road between Kosice in the SE and Spisska Nova Ves in the NW. From this main road, a network of minor unsurfaced tracks traverse the forest, and give access to the project area.

The climate is essentially Central European, but is moderated by the 600m altitude. In effect this gives the area cold winters, with snow on the ground between about December and March. The mean January temperature is around -5^0 C, and the mean July temperature is 19^0 C. Total annual precipitation is 700 to 800 mm, with over 30 mm precipitation falling as snow in January.

The Jahodna uranium deposit belongs to a belt of U-Mo deposits, of Lower Permian age, within the western Carpathians of Slovakia. These deposits are largely stratabound bodies within volcanosediments of Permian age. It appears that the U-Mo mineralisation at Jahodna was disseminated within the volcanosedimentary pile, and was subsequently enriched into stratabound zones by post depositional geological activities.

The Jahodna deposit appears to occupy an open-space fill, on a geological contact between an overlying competent meta-andesitic volcanic unit, and an underlying meta-sediment unit. Tectonic disturbances have also resulted in slaty cleavage and well developed shistosticy in places and fault offsets, some of which disrupt the main deposit. The deposit is partially blind at the surface. Uranium mineralization detectable with a hand-held scintilometer outcrops along the meta-andesite – meta-sediment contact above the deposit.

The deposit area is covered by thick soils, with extensive forest cover at surface. The deposit has a NW-SE strike, and a steep dip to the SW. The overall dimensions of the main deposit established to date are some 500m x 500m in longitudinal section, and an average of about 2.5m in thickness. There are also minor mineralised zones in the hanging wall of the main deposit, though their relationship to the main deposit is still uncertain.

The Jahodna uranium deposit represents a difficult exploration target, since it is partially blind and deep-lying, in hilly terrain with deep soil cover and thick forest at surface. At these elevations, snow lies on the ground during the winter, sometimes for more than 3 months per year. A combination of these factors virtually negates surface exposure, and make useful surface mapping harder to achieve. Furthermore, the geology of the deposit itself is in some ways complex, and is as yet incompletely understood. As detectable mineralization does outcrop in places along the favourable stratigraphic horizon (meta-andesite – meta-sediment contact) it may be possible to explore for additional mineralization along strike by prospecting the meta-andesite – meta-sediment contact with a hand-held scintilometer and conducting ground radiometric and/or soil-gas (radon) surveys.

The complex mineralisation at Jahodna comprises U-Mo-Cu, with lesser Fe-Pb-Ti-P-Ba. The mineralisation at Jahodna has been described as a "Saddle Hills analogue", after the Saddle Hills uranium deposit in Mongolia. Although there is insufficient detailed information to fully back this up, there would appear to be some broad similarities between Jahodna and the Saddle Hills type deposits. The Jahodna deposit itself is a high grade deposit

In many ways, it is to the credit of Uranovy Prieskum that they discovered the Jahodna deposit at all. Such are the complexities of the setting of this deposit, that it was only after considerable exploration drilling that it was discovered in the 1980's. Following airborne radiometric surveys, and ground follow up radiometric surveys, the best way forward was found to be to systematically drill the airborne and ground anomalies. Consequently, on the Jahodna project, a total of 53 exploration holes, totalling over 17,000m of drilling, were completed between 1985 and 1991.

Having established that deep drilling was the best way to evaluate the Jahodna deposit, it has to be said that the drilling undertaken by Uranovy Prieskum was unsatisfactory in some key respects. Their pre-wireline, thin-walled, conventional B-size drill string deviated considerably from the targets, often intersecting the mineralized horizon many tens of metres or more from the intended point of intersection. In a deposit where it would appear that post-mineralization normal faulting has broken up the deposit zone into blocks, the positions of which are likely to be of critical importance at the mine planning stage, this lack of directional drilling control was unfortunate. Perhaps largely because of the directional problems with the earlier drilling, the resulting drill cross sections of the original evaluation were overall not normal but somewhat oblique to the plane of the deposit.

This affects the quality and reliability of the samples from this drilling and therefore influences the reliability of calculated resource estimates.

Furthermore, in drilling a strongly foliated, tuffaceous meta-volcanosedimentary sequence, the core recovery of the drilling was generally poor. This was unfortunately particularly the case in the mineralised zone, where shearing on the base of the meta-andesite unit resulted in strongly broken ground in places, and consequently core recoveries in the region of no more than 50% (though this was improved towards the very end of the drilling after 1990). For this reason, there were apparently no complete mineralised zone intersections from the original 14 holes drilled in the immediate vicinity of the Jahodna deposit.

With a record of persistently poor core recovery in the mineralised zone, the only way to attempt to establish the average widths and grades of the mineralised zone was by down hole radiometric logging. Fortunately this technology appears in itself to have worked well – though it has to be emphasised that there were no complete, assayed borehole intersections at Jahodna against which to calibrate the gamma logger response. However, all aspects of the tonnage and grade calculation at Jahodna appear to have adhered to strict technical guidelines set down by the national mining regulatory bodies at that time, and also drew on former radiometric logging experience of drilling and mining similar deposits within the country.

Within the area of the Jahodna deposit itself, the past drilling has been both irregularly and quite widely spaced. It would seem that by the traditional Czech type reserve classification this stratiform but irregular replacement type body was categorised as a "Z-3 reserve", equating to a Soviet type "C_2 reserve" or an "inferred resource" under CIM classification. *

One prominent feature highlighted by the drilling at Jahodna is the poor ground conditions existing in much of the ground near the deposit itself. However, these bad ground conditions appear to be largely restricted to the thick hanging wall volcaniclastic sequence. Although the mineralised zone itself appears to be tectonised, it seems to be largely supported by the more competent meta-andesitic body which immediately overlies it.

For reasons including core recovery, limited geological / structural interpretation and poor ground conditions, in terms of systematically establishing the true average grades of the mineralised zone at Jahodna, or important geological aspects like the detailed positions of cross faults, the earlier drilling undertaken by Uranovy Prieskum would in my opinion have to be regarded as somewhat preliminary in nature.

The first major resource calculation was undertaken in 1996 by Jozef Daniel, a geologist experienced in the uranium industry of the former Czechoslovakia. The tonnage and grade calculation was updated in 2005 by the same author, in a report entitled "Calculation of Reserves Deposit Kosice I, U – Mo Ore", which was translated into English. This calculation derived the following summarised figures:

- Variant I – (0.015% U Cutoff) – 2.188 Million tonnes at 0.329% U to derive an in situ figure of 15.876 Million lbs of U.
- Variant II – (0.030% U Cutoff) – 1.396 Million tonnes at 0.472% U to derive an in situ figure of 14.528 Million lbs of U.

*** Note that the Slovak category of Z-3 is roughly analogous to the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") definition of inferred resource. Although this previous work is considered relevant, exploration did not result in a current resource estimate and these are not CIM defined resources, therefore readers are cautioned not to rely upon these estimates.**

Molybdenum grade calculations were made as well as uranium (with a calculated average figure of 0.38% Mo). However, with the poor core recovery, and lack of an alternative method for determining Mo grades, the volume of data available to determine Mo grades was reduced, for which reason the Mo grades calculated are regarded as less reliable than the uranium. There is also an unresolved question regarding the detailed distribution of the Mo mineralisation within the Jahodna deposit.
Three deep diamond drill holes (totalling 1,364.90m) have recently been drilled by Tournigan into the Jahodna deposit. The purpose of these holes was to provide preliminary confirmation of the thickness and average grade of the deposit. In contrast to the earlier drilling of the 1980s, this recent drilling achieved good directional control and excellent core recovery. Assay results from this drilling substantially confirmed previous drilled widths and uranium grades.

Since the previous resource calculations at Jahodna were based on a large volume of detailed data, parts of which were not available for examination, it was not possible to conduct a full recheck of all data pertaining to the former resource calculations. For this reason it was decided to undertake an in-house Micromine computer study of the available Jahodna drill data, in order to make an independent assessment of the previous resources calculation.

Accordingly, a Polygonal Wireframe Resource Estimate (PWRE) was calculated for the Jahodna Resource, using data from thirteen (13) historical drill holes over the project. The thirteen deep historical drill holes, amounting to some 6,290 m of drilling, covered the main part of the Jahodna deposit, though the wireframe model was somewhat constrained in that it did not have all the deposit edge data used for the earlier estimates. Nevertheless, the PWRE appears to have broadly confirmed thetonnage and grade estimates.

The PWRE work was conducted in two phases – firstly, using only the original Jahodna drill hole data, and secondly, introducing the three new boreholes drilled by Tournigan Gold Corporation. In both cases, the PWRE appears to have broadly confirmed the prior tonnage and grade estimates.

The recent PWRE resource calculation, using the new Tournigan drilling results, can be summarised as follows:

- (0.030% U cutoff) -1,256,088 tonnes @ 0.56% U to give 15,510,172 lbs of contained uranium.

Taken together, it would appear that the recent PWRE work on the original Jahodna data, the new Tournigan drilling and the updated PWRE model have all broadly confirmed the prior tonnage and grade estimates at Jahodna using the same cut-off grade of 0.03% U. Based on this, it is reasonable to state that all this work confirms the tonnage and grade estimate at Jahodna as an "inferred resource".

Although surface exploration at Jahodna is made difficult by a number of factors (deep soil cover, very little outcrop, extensive forest cover etc), there are possible ways where further work at surface might be feasible. Prospecting along the meta-andesite – meta-sediment lithologic contact with a hand-held scintilometer and conducting surface radiometric and/or soil-gas (radon) geochemical surveys over the favourable horizon may prove effective as surface exploration techniques. The accurate positioning of the large cross faults is a critical factor. One possibility to consider may be to use high resolution aerial photography and / or satellite imagery flown during the ‘Spring window’ – when the snow has melted but before the forest vegetation has grown. In this way a working geological / structural model of the area can be developed, to be modified with the results of the ongoing drilling programme. If the remote sensing methods mentioned above are not feasible, another method to pinpoint fault structures may be to undertake detailed ground magnetic surveys, though a tight grid (eg. 50m x 10m or 50m x 25m) would be recommended.

It is recommended that a conceptual mining study be undertaken at Jahodna. The purpose of such a study would be to give exploration guidance, at a relatively early stage of exploration, to an apparently attractive project in a geographic area where mining operations were not previously costed to the same degree as in the west.

Even though the work to date suggests that the Jahodna deposit can be regarded as an inferred resource, much more exploration work is recommended in order to upgrade the confidence level of the resource to the indicated category. Specifically, a substantial drilling programme is recommended, in order to effectively drill the deposit on 50m centres. This will achieve sufficient drilling density to increase the level of confidence in grade and geological continuity through the deposit, and may facilitate an upgrading of the resource classification to “indicated.” In view of the deep, narrow and steep-dipping nature of the deposit, intensive drilling from surface on 50m centres is not recommended. Drilling from surface to give 50m centre coverage would involve some 85,000m of drilling, resulting in a high drilling cost in the region of USD 7.6 million. Therefore, an initial phase of in-fill and step-out diamond drilling within and immediately along strike of the deposit should provide more even-spaced drill intercepts along the strike of the deposit at two depth-horizons as well as test presently un-tested gaps in the resource block that we believe are possibly mineralized.

Instead it appears appropriate, following a further limited programme of deposit outline drilling, to embark on a programme of underground development, in order to facilitate a drilling programme from underground. Such underground access would enable 50m centre drilling using only some 39,000m of drilling (as opposed to 85,000m of drilling from surface), and equating to a cost of approx. USD 4.9 million (ie. a 35% cost saving compared to the surface drilling programme). It should also be mentioned that, with an average hole length less than half that of the surface drilling (ie. average length of 211m per hole as opposed to 436m per hole), the drill holes of the underground drilling programme would be generally much more precise in intersecting their target areas.

To provide the underground access for such a drilling programme, a preliminary costing has been made of an underground development programme. This scenario allows for an access decline of some 2900m, a 500m long horizontal drive in the deposit’s hanging wall, 1200m of crosscuts for drilling access, and a 350m ventilation raise. The preliminary costing for this programme has been calculated at USD 5.5 million. Besides enabling access for underground drilling and bulk sampling, the underground access would be laid out such that it could be used for subsequent mining operations.

Besides the further detailed evaluation of the Jahodna deposit, it is recommended to undertake additional grass roots type exploration within the licence area. This is especially the case in the SE part of the license area, where former systematic exploration did not cover.

A summary of the Jahodna Preliminary Assessment is as follows:

The capital cost of building the mine and plant, including the acquisition of all operating equipment is US$29.4M. All costs in this report are in US$. Currency factors used are those pertaining during March 2006. The capital cost is made up of US$10.2M for the mine, US$13.8 for the plant and US$5.4M for indirects and infrastructure. There are no estimates in this report of project acquisition or other past owners costs. There will be sustaining capital

and replacement capital in the mine of around US$1M during year 6 as the mine becomes deeper.
The direct operating cost of mining and process is US$92.50 per tonne, made up of average mining direct costs of US$43 per tonne, process costs of US$45 per tonne and management overhead of US$4.50 per tonne.

The mining and processing costs are relatively high because of the low production rate. This production rate is determined by the nature of the mineralised zone, and is probably close to the optimum figure. Nevertheless, the ore is of high value, and it is probable that these high costs can be absorbed.

There are operating costs attributable to the need to deal with the radioactive ore. Extra precautions are specified to mitigate the affects of gamma radiation particularly, but more detailed work will need to be done on this aspect of the operation. Another contributory factor to high costs is the proposal to use a cemented fill in the stoped out areas. It is possible that cheaper alternatives might be available to reduce the high cost of Portland cement used in the estimate.

ACA Howe has assumed the availability of skilled personnel in all areas of the operation, and the obtaining of all the necessary permits and licences. This is the responsibility of Tournigan and it is assumed that this aspect of the project is being dealt with separately and with positive results.

Notwithstanding all this, there is probably a viable project to be built at this site, especially considering the strong market in uranium yellowcake. The actual value of the project will be revealed during the financial modelling phase, but it is expected that it will be positive.

2. INTRODUCTION

The author visited Slovakia for approximately two weeks during October 2005. During this time, he spent time at the offices of Kremnica Gold, a.s. in Kremnica, and of Koral SRO in Spisska Nova Ves, examining paper and digital data of various ages, and arranging for scanning, digitising and other map work activities. A one day visit was made to the project area, to make geological ground traverses and examine drill core from the new drill hole named KG-J-1.

After returning to the UK, the author compiled relevant data, assisted with various aspects of digital mapwork, estimated an inferred resource and wrote a applicable technical report. Other work for this report was completed by Mssrs. Mark Butcher and Galen White of ACA Howe International Ltd in Berkhamsted, UK. The former assisted with general digital map work, and the latter constructed a Micromine computer model of the deposit for tonnage and grade estimation purposes.

Tourngian Gold Corporation commissions this Preliminary Assessment of the Jahodna project, which was prepared in April 2006 for the purpose of examining the possible scale and cost of operating a mechanised mine at the site. This technical report discussed the possible technologies that could be used, and is included in item 18.

2.1 Disclaimer

If the author of all or a portion of the technical report has relied on a report, opinion or statement of legal or other experts who are not qualified persons for information concerning legal, environmental, political or other issues and factors relevant to the technical report, the author may include a disclaimer of responsibility in which the author identifies the report, opinion or statement relied upon, the maker of that report, opinion or statement, the extent of reliance and the portions of the technical report to which the disclaimer applies.

Much of the content of this report relies on English translations of previous reports in Slovak, in particular that listed above by Daniel (2005). While it is felt that previous work undertaken in the area, in general, probably conformed in detail to the exploration and evaluation regulations in force in Slovakia at that time (eg. official mining directorates of 1989 and 1992), there is probably much detail of past work that has not yet been translated into English, so was difficult to obtain or evaluate for the purposes of this report. Such data would include details of exactly how equivalent grades of uranium were derived historically from down hole geophysical gamma logs, using a number of coefficients and factors which appear to have been based on past uranium exploration / mining history in the area.

In addition to these, conversations were had, through an interpreter, with a number of Slovak technical personnel who had previous involvement with the Jahodna project. In particular, these included Mssrs. Jozef Daniel and Ladislav Novotny. In addition, much general information of the former Czechoslovakian mining industry was gained from Dr. Boris Bartalsky, currently the General Manager of Tournigan's Kremnica Gold Project in Slovakia.

Since the author only spent two weeks in Slovakia, and less than one day on site at Jahodna, heavy reliance has been placed on the various sources of data mentioned above.

3. RELIANCE ON OTHER EXPERTS

The main sources used in the compilation of this report were the following:

- **Calculation of Reserves, Deposit Kosice I, U-Mo Ore April 2005**, Daniel J., Bartelsky B., unpubl. company report by Kremnica Gold Corp.
- **Resolution on Granting of the Exploration Licence,** March 21 2005, issued by Ministry of the Environment of the Slovak Republic, Geology and Natural Resources Dept.
- **Technological Research of U-Mo ore from Jahodna Site, 1993**, Kopecky J., unpubl. report by MEGA, joint stock company Strazpod Ralskem, Czech Republic.

In addition to these sources, which specifically concerned the Jahodna deposit, a number of other sources and references were used, which are listed at the back of this report.

4. PROPERTY DESCRIPTION AND LOCATION

The full title of the current exploration license refers to "Cermel-Jahodna – U-Mo, Cu ores", and it was granted on March 21st 2005 by the Geology and Natural Resources Department at the Ministry of the Environment of the Slovak Republic. The full area involved is shown on Maps 6 and 7, and this area amounts to 31.75 km^2 in surface area (see Appdix.1 for further details). The period of validity of the licence is four (4) years.

The name and "code" of the region is Kosicky 8, and the name and code of the counties are Kosice I - 802, Kosice III - 803, and Kosice – okolie - 806. The names and numbers of the cadastral areas are shown on the table below:

NAMES AND NUMBERS OF CADASTRAL AREAS

No	The No. of the Cadastral Area	The Name of the Cadastral Area	The Name of the Village	Relative Ratio of the Villages %	Cost SKK
1	827207	Cermel	Košice -mestská	51.59	24,763

No	The No. of the Cadastral Area	The Name of the Cadastral Area	The Name of the Village	Relative Ratio of the Villages %	Cost SKK
			cast Sever		
2	827428	Myslava	Košice -mestská cast Myslava	9.20	4,416
3	802123	Baška	Baška	7.09	3,403
4	827606	Košická Belá	Košická Belá	20.93	10,046
5	841129	Nižný Klátov	Nižný Klátov	6.41	3,077
6	871516	Vyšný Klátov	Vyšný Klátov	4.78	2,295

The license area, which is a single, contiguous area, is shown in some detail on Maps 6 and 7, which shows UTMs using the Krovak – Gaussian equiangular conic projection. The coordinates of the licence area are given as:

COORDINATES OF JAHODNA LICENSE AREA

Point No.	Y	X
1	268 31000	1 241 45000
2	274 59000	1 229 34000
3	273 70000	1 228 60000
4	268 20000	1 234 68000
5	266 81000	1 241 56000

The “conditions” of the exploration licence are shown below:

The holder of the exploration licence:

1. will perform the geologic works in accordance with the project of the geological work that was submitted with the application on granting of the exploration license and the holder will perform the geological works in compliancy with the geological law and other legal regulations.
2. will prepare the final report in compliance with §14 of the geological law and will submit to ministry the calculation of the resources for the approval, in compliancy with § 16 par. 2
3. will send the approved final report to geological survey of Dionýz Štúr Bratislava for archiving in compliance with §17 of the geological law.
4. will submit the annual report of the geological work with the results of special geological works and spent money on exploration up to six weeks after the end of the year.
5. will follow the requirements of nature and land protection pursuant the law,
6. will cut the trees out of the wood territory if necessary and ask the resident village for permission pursuant the law,
7. will secure the places of holes against fuel leakages into the underground or surface water and surrounding,
8. will clean the field and put it into the previous conditions after finishing of geological works,
9. will keep regulations of the law Nr. 364/2004 about waters

10. will require demarcation of protective zone by resident water company if any technical works needed
11. will ask for statement the resident company if any technical works in the area of holiday and sport centre
12. will keep the law about using of agricultural land and control of pollution of the environment,
13. will ask for the statement the resident keeper of Bukovec water tank which provides local villages Košická Belá, Vyšný Klátov with water,
14. will keep the law about forests,
15. will follow the various regulations about protection of the forest land reserves,
16. will announce the geological works in the Protective deposit area Košice VI. to the resident company Uranpres, s.r.o. Spišská Nová Ves pursuant to the regulations set by the Slovak mining bureau,
17. will announce the existence of the mineral water and gas resources to the Ministry of Health up to 15 days since found pursuant to the law,
18. will follow the law if any archaeological findings,
19. will not realize any geological works where any cultural sights,
20. will ask for statement from the local municipality in Košice – landed estate department before any geological works,
21. will ask for statement where any roads of the II. and III. type the local municipality in Košice,
22. will ask for statement the Slovak gas industry before any geological works,
23. will keep various standards and the law about power industry,
24. will ask for statement the Slovak Telecom a.s.,
25. will respect the water managing objects and lines of protective zones of the water resources,
26. will not realize technical works in the protective zones of water resources,
27. will ask for statement East-Slovak water company, Košice before any geological works,
28. will keep valid standards and regulations if dealing with dangerous substances to prevent any pollution of surface and underground waters while geological works,
29. will ask for statement the East-Slovak power company before any geological works.

To the knowledge of the report author, all appropriate obligations have been fulfilled by the licence holder, prior to commencement of exploration works on the licence area.

Within the exploration licence in question, there is one known mineralised zone, which is the Jahodna deposit, which was historically drilled and evaluated by Uranpres. The location of this area is indicated by the drill holes on Maps 1 to 13. Within the licence area, there are no known mine workings, existing tailing ponds, waste deposits or other workings relating to previous exploration or mining.

Other than the above annual licence payment, the report author is not aware of the terms of any royalties, back-in rights, or other agreements and encumbrances to which the property is subject. All the known environmental liabilities, and permits that must be acquired to conduct the work proposed for the property, are listed above. As stated above, to the knowledge of the author, all appropriate contractual obligations have been fulfilled by the licence holder, and all necessary permits have been acquired, prior to commencement of exploration works on the licence area.

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

The Jahodna property is situated in East-Central Slovakia (see Map 1). Topographically, the region forms part of the Western Carpathian mountain chain. Locally the hilly terrain is part of the Volovec Hills, in more detail the Kojsovska Hola. Around Jahodna, the ridges trend NW-SE and the topography is quite incised, with hills up to around 650m amsl, and valley floors typically down to some 500m amsl (in the immediate area of the project).

The actual surface is undulating, with little or no outcrop and deep soil cover of many metres depth. The vegetation is a type of mature mixed woodland, being made up mostly of broadleaved types (eg. beech), but also lesser conifers (see Plate 1). The forest is in fact part of a forestry reserve, though according to Tournigan staff in Slovakia, this should not pose a prohibitive problem with regard to planning permission for exploration or mining development activities.

The Jahodna property lies quite close to (less than 300m south of) the regional main road (No. 547) between Kosice in the SE and Spisska Nova Ves in the NW. From this main road, a network of minor unsealed tracks traverse the forest, and give access to the project area.

The Jahodna property is situated some 10-15 km NW of the large town of Košice, a regional centre in East-Central Slovakia (see Map 1). It is situated outside the town lands of Košice. As mentioned above, the property is situated near the main tarred road between Košice and Spisska Nova Ves. A seasonal ski resort (referred to as “Jahodna Chalet”) occurs further to the NW along the same range of hills.

The climate is essentially Central European, but is moderated by altitude (ie. the project is in hilly terrain at around 600m altitude). In effect this gives the area cold winters, with snow on the ground between about December and March. According to the Slovak Encyclopaedia (Reference 5) the mean January temperature is around -5^0 C, and the mean July temperature is 19^0 C. Total annual precipitation is 700 to 800 mm, with over 30 mm precipitation falling as snow in January. Records indicate that snow lies on the ground for over 80 days per year (generally January to March). With access to the project area by unsurfaced tracks, the snow cover is expected to cause periodic difficulties with access during the winter months, probably most particularly during the spring thaw.

While the existing surface rights are sufficient for exploration purposes, it is not known whether they are completely sufficient for all aspects of a mining operation. The surface area is undulating and hilly, and it is not yet established whether there is sufficient suitable and reasonably level ground for potential waste rock and tailings storage areas (such aspects would more appropriately be covered in a scoping study concerned with mining aspects).

A small stream of intermittent flow drains NE along the valley traversing the Jahodna deposit, flowing into the Crmel valley which lies to the NE side of the hill range. Another larger river (the Vrbica) occurs approx. 1 km to the west, bounding the hills on the west side. The Vrbica and Crmel rivers are tributaries of the Hornad river, which flows southwards past Košice – see Maps 1 and 6. Apparently, electric grid power occurs in the area, though the exact distance from site is not known at present.

Up to the time of the so-called Velvet Revolution in 1989, Czechoslovakia had a large, state-funded mining industry, most of which closed down when state funding was withdrawn following the demise of the communist system. For this reason, it is likely that Slovakia still contains a significant number of trained mining personnel eg. from the old Novovesta Huta mining operation, which was operated some 40-50 km north west of Jahodna, and close to the town of Spisska Nova Ves.

6. HISTORY

Up to the time of the demise of communism in 1989, all uranium exploration and mining in Czechoslovakia was conducted by the State-owned organisations, such as KORA, CSUP and URANPRES. All early exploration work on the Jahodna property was undertaken by these organisations. Following the country's return to the market economy system, and the subsequent separation of Slovakia and the Czech Republic, very little work has been undertaken on the Jahodna property. Tournigan Gold Corporation acquired the Jahodna property in 2005.

The Jahodna deposit was discovered in 1985, following a regional uranium exploration programme undertaken by the Czechoslovakian KORA and CSUP groups. Regional airborne radiometric surveys had delineated radiometric anomalies on surface, which were followed up by ground radiometric surveys, geological mapping and trenching. Thereafter, systematic diamond drilling was used to investigate ground radiometric anomalies (see Maps 8 and 9). The Jahodna deposit, which is partially blind and therefore does not outcrop at surface except as discontinuous zones of low-level mineralization, was discovered by routine diamond drilling of surface anomalies. It was thereafter drilled by Uranovy Prieskum (URANPRES), though the exploration programme was cut short by the political events following the 1989 Velvet Revolution.

In all, some 17,000 m of drilling were undertaken in 53 holes on and around the property. Most of the drilling was by conventional (ie. pre-wireline) diamond drilling. The thin-walled drill strings deviated considerably during drilling, and core recovery was generally poor (overall around 50%). To compensate for this poor core recovery, down-hole radiometric logging was routinely used on all the drilling in the Jahodna area, and it seems to have worked very well. Conversion formulae were developed, based on different factors and coefficients which were derived from previous uranium exploration and mining experience from nearby uranium projects (eg. Novovesta Huta), in order to convert down hole radiometric measurements into equivalent in-situ uranium grades. This down hole work compensated at least in part for the poor core recovery, which resulted in an incomplete assay database for the project.

To summarise the effectiveness of the previous exploration, it has to be said that it is to the credit of the former Czechoslovakian state exploration companies that the Jahodna deposit was discovered at all. The deposit itself does not outcrop near to surface, though its distal peripheral margins do sub-outcrop at surface, and gave sufficient radiometric response for it to be identified as a radiometric anomaly detected during airborne, and ground radiometric surveys and with a hand-held scintilometer on outcrop. Soil cover at surface was generally too deep for surface mapping, trenching and pitting to be effective other than to generate drill targets, so the best way forward was found to be systematic drilling to investigate the depth extensions of the surface anomalies. In all, several thousand metres of diamond drilling from surface were used as a regional exploration methodology, before the discovery of the Jahodna deposit itself.

However, while the former Czech exploration enterprises are to be commended on the systematic and persistent approach which led to the Jahodna discovery, they also encountered considerable difficulties during the more detailed evaluation stages. Since the depth of the Jahodna deposit meant that drilling was to be the main evaluation method, a heavy reliance was put on deep diamond drilling programmes. The main problems here revolved around not only the depth of the target (necessitating drilling of up to almost 1000m in depth), but the generally poor ground conditions that the drilling had to encounter, and also the unsophisticated drilling equipment used.

The majority of the drill holes had to pass through up to several hundred metres of hanging wall intermediate meta-volcaniclastics/tuffs (Petrovohorske Formation, No. 12 on geological maps), before reaching the mineralised zone. This formation comprised tuffaceous volcaniclastics, which are steeply dipping and strongly cleaved, with cleavage planes more or less paralleling the bedding planes – this combination caused persistent problems with poor ground and consequently poor recovery. In addition to the poor ground conditions in the hanging wall sequences, the mineralised zone itself was generally weak and friable, also resulting in very poor core recovery. On top of this, multi-tubed wireline drilling equipment was only used late in the exploration programme – the majority of the drilling programme was undertaken with thin-walled, single tubed, conventional drilling equipment (this resulted not only in poor core recovery, but in poor directional control of the drill path).

Besides the purely technical aspects of the drilling problems, the main result was the very poor core recovery within the mineralised zone. Obviously, this affected the sampling integrity of the U and Mo, the two main economic minerals. If a drill achieves only 50% recovery in a broken mineralised intersection, it is difficult, based on assays alone, to obtain a clear picture of both the average grade, and the distribution of grade, of a particular mineral within the mineralised zone. Fortunately, in the case of uranium, the difficulties were eased by the down hole radiometric surveys. Such surveys had been used previously on uranium exploration and mining, in similar deposits, so correlation coefficients had been worked out to convert the down hole radiometric response into equivalent uranium grades. These down hole surveys enabled a much more complete picture of the uranium grade and distribution than was possible with the assays alone. Unfortunately for the Mo analyses, there was no alternative way to determine equivalent Mo concentrations, consequently the Mo grades of the drill samples remain less reliable than the U.

The first major resource calculation was undertaken in 1996 by Jozef Daniel, a geologist experienced in the uranium industry of the former Czechoslovakia. The method used a block model method, with two variants based on different minimum cutoff grades (0.015% and 0.03% U). The calculation methodologies were constrained by government mining directives, established in 1987 and updated in 1992. The calculation concentrated on the main deposit, which was placed in the category of "Z-3 supposed reserves" – the lesser zones of mineralisation in the hanging wall of the main deposit were assigned to the lower "prognostic" category. The tonnage and grade calculation was updated in 2005 by the same author, in a report entitled "Calculation of Reserves Deposit Kosice I, U – Mo Ore". This report was translated into English, and since it represented a comprehensive technical history of the Jahodna project, it was consulted in detail by the present author.

Please see following table for tonnage and grade estimates.

#	Name of Study / Description	Tonnes	Grade % U	Content lbs	Comment
1	1996 Calculation (J-1-Z-3)	1,148,000	0.459	11,618,851	"Economic reserves exploitable"
	(by J. Daniel)				Used in Tournigan website
2	1996 Calculation (J-1-Z-3N) (by J. Daniel)	1,080,000	0.187	4,453,218	"Potentially economic reserves exploitable" Used in Tournigan website
3	1996 Calculation (J-1-Z-3)	1,396,000	0.472	14,529,010	"Total geological reserves"
	(by J. Daniel)				
4	Josef Daniel Study April 2005	2,188,553	0.329	15,876,000	Latest updated figures from 2005 study.
	Variant I (0.015% U cutoff)				
5	Josef Daniel Study April 2005	1,395,975	0.472	14,528,745	As of April 2005 study
	Variant II (0.030% U cutoff)				

COMPILATION TABLE FOR DIFFERENT URANIUM RESOURCE/ RESERVE

*** Note that the Slovak category of Z-3 is roughly analogous to the CIM definition of inferred resource. Although these results are considered relevant for this report, these are not current resources as defined by CIM. As these are not CIM defined resources, readers are cautioned not to rely upon these estimates.**

A full description of previous exploration work, and tonnage and grade calculations based on them, are given in Josef (2005).

Molybdenum grade calculations were made as well as uranium (with an average derived of 0.38% Mo), though with the poor core recovery, and lack of an alternative method for determining Mo grades, the volume of data available to determine Mo grades was reduced, for which reason the Mo grades calculated are regarded as less reliable than the uranium. There is also an unresolved question regarding the detailed distribution of the Mo mineralisation within the Jahodna deposit. Some evidence was encountered suggesting that Mo grade variations were not sympathetic with the U grade variations, and even that Mo was enriched on the margins of the deposit. Therefore, with the present data base, Mo is regarded only as a potential by-product.

The former resource calculations are not compliant with CIM definitions. However, confidence levels have been increased by recent developments (see below).

No production or mining activities have yet been undertaken from the Jahodna property, to the writer's knowledge.

7. GEOLOGICAL SETTING

The Jahodna uranium deposit belongs to a belt of U-Mo deposits within the western Carpathians of Slovakia, which are largely stratabound bodies within volcanosediments of Permian age. It appears that the U-Mo (Cu) mineralisation was disseminated within the volcanosedimentary pile, and was subsequently enriched into stratabound zones by post depositional (tectonic deformation) geological activities. See Maps 2 to 5.

The Jahodna deposit is contained within a Lower Permian volcanosedimentary sequence, designated as the Petrovohorske Formation. Its main units at Jahodna are briefly described below:

- Overlying the immediate hangingwall are the intermediate volcaniclastics of the Hutniansky Complex (designated No. 4 on the geological maps). They are a few hundred metres thick in the Jahodna area, and are generally incompetent (on account of their parallel, steeply dipping bedding and cleavage planes).

- The rock type which forms the immediate hangingwall to the Jahodna deposit is the meta-andesite of the Hutniansky Complex (designated No. 43 on the geological maps). It forms a semi-competent zone, varying in thickness from 20m to 50m, immediately above the deposit. In addition to the main zone of mineralisation at its base, this unit also contains lesser "stringers" of U-Mo-Cu mineralisation within it.
- The main deposit – is hosted along the faulted, disturbed contact of the hangingwall meta-andesite and the footwall meta-sediments within the basal part of the meta-andesite unit. It averages some 2.5m in thickness, and basically comprises a uranium / polymetallic mineral assemblage, which has been deposited into a tectonically disturbed zone, on the contact of an overlying meta-andesite and a footwall meta-sediment unit.
- The meta-sediments (slates, quartzites) of the Knolske Formation form the immediate footwall to the mineralised zone. This unit is designated No. 12 on the geological maps. They are up to hundreds of metres thick in the Jahodna area, and are of varying competence.

The upper 2 units, described above, belong to the Hutniansky Volcanic Complex (part of the Petrovohorske Formation), while the footwall to the deposit is contained within the Knolske Formation. The entire sequence is contained within the Lower Permian Krompasska Group.

The Jahodna deposit occupies dilational zones along the geologic contact between the overlying andesitic meta-volcanic unit and the underlying meta-sediments. Shearing along this contact has resulted in tectonic disturbance and poor ground conditions in places. Tectonic disturbances have also resulted in schistose foliation and slaty cleavage (giving poor ground conditions in some softer sedimentary units) and fault offsets, some of which disrupt the main deposit.

The deposit is partially blind (ie. limited surface expression), and is covered by thick soils, with extensive forest cover at surface. The deposit has a NW-SE strike, and a steep dip to the SW (60^0 in the upper part, 47^0 in the lower part). The overall dimensions of the main deposit established to date are some 500m x 500m, and about 2.5m in average thickness. As mentioned, there are also minor mineralised zones in the hanging wall of the main deposit, though their relationship to the main deposit is still uncertain.

8. DEPOSIT TYPES

The Jahodna deposit has been described as a "Saddle Hills" analogue, after the Saddle Hills / Dornod uranium deposits in eastern Mongolia. However, while the Saddle Hills deposits have been relatively well explored and documented, insufficient information is known about the Jahodna deposit to place it firmly in this category. However, there are broad similarities between the two - like Saddle Hills, Jahodna appears to be a replacement type deposit (both stratabound and cross-cutting), hosted in a strongly deformed Mesozoic volcaniclastic sequence. Also like Saddle Hills, Jahodna is enriched in a number of minerals besides uranium.

Besides the Jahodna deposit itself, which is an advanced exploration project, and will therefore require largely further drilling and detailed geological / sampling studies, there are known to be a number of mineralised lenses along strike of Jahodna within the intermediate (andesite / dacite) "Hutniansky" meta-volcaniclastic/tuffs of the Petrovohorske Formation. Several of these have been investigated in the past by the CSUP and KORA groups, though so far, Jahodna was the only mineralised lens discovered in the area with clear economic potential. The majority of these mineralised occurrences showed as radiometric anomalies of some sort at surface, though many were very subtle anomalies, on account of the depth of soil cover and the depth of some mineralised bodies. This depth of soil cover meant that pitting and trenching were less than successful as exploration methods, and in fact routine diamond drilling proved to be the most successful exploration tool to investigate ground radiometric anomalies at depth. This was how the Jahodna deposit itself was discovered.

Besides the further detailed evaluation of the Jahodna deposit (described elsewhere in this report), it is recommended to undertake additional grass roots type exploration within the licence area. Apparently the previous exploration by the CSUP and KORA groups started from the NW and worked towards the SE (since the regional exploration was spreading along strike from known deposits like Novovesta Huta in the NW), and following the Jahodna discovery, little further work was undertaken in the SE part of the concession (this cessation in exploration activities also coincided with the political developments following the Velvet Revolution of 1989, after which time virtually all exploration and mining activities in the former Czechoslovakia ceased). For these reasons, the writer understands from local geologists that the SE half of the concession is less well explored than the NW part. Consequently, it is recommended that grass roots type exploration activities be concentrated in this area.

9. MINERALISATION

The main mineralised body at Jahodna, based on past work, is like a large but thin, sheet like form – typically 500m x 500m in surface area, but only in the order of 2.5m thick. The deposit is partially blind, rarely outcropping at surface, with the top of the main zone of mineralization occurring about 200m below surface (though this figure is relative since the surface in this area undulates from some 500m to 630m amsl), extending for some 500m in a down dip direction. The upper half of the deposit has a dip of about 60^0, and the lower half a dip of about 45^0.

Basically, it would appear that the uranium mineralisation represents secondary type mineralisation localized along foliation and within ptygmatically folded quartz-carbonate veins. The main reason for this observation is that the majority of the mineralisation previously described from Jahodna occurs as veins, veinlets, or other open space fill. Mineralised zones have a clear lithologic and structural control. For example, mineralization is stratabound along the contact of the hanging wall meta-andesite unit and the foot wall meta-sediment unit and is localized in folded fracture-fill veins and along foliation planes.

Another interesting point, not yet quantified, is that the spatial position of the main deposit seems to indicate the importance of big cross faults in the area. At least 2 big cross faults (with ENE orientation and apparent dextral throws of up to 20m) occur in the vicinity of the deposit. What factors control and delimit the margins or the deposit are not yet known, but one possible one is distance from mineralising cross faults. With the tendency of the main Jahodna deposit to be spatially associated with cross faults, this may suggest that the cross faults were the original conduits for the U-Mo-Cu mineralisation to be transported into the vicinity. Again, considerably more detailed exploration work would be needed to confirm this point.

Regarding the detailed mineralogy of the deposit, the following data is taken largely from the report by Josef (2005). Based on historic work at Jahodna, the main mineralised minerals are molybdenite, uraninite, brannerite, U-Ti oxides and subordinate coffinite, with main accessory minerals being abundant pyrite and subsidiary chalcopyrite. Based on former petrographic studies of mineralized drill samples, the following minerals were shown to be associated with the Mo-U-Cu mineralization: molybdenite, uraninite, U-Ti oxides, brannerite, coffinite, chalcopyrite, tennantite, pyrite, marcasite, galena, chalcocite, bornite, covellite, hematite, rutile, leucoxene, apatite, barite, malachite, goethite, iron-dolomite, calcite, quartz, sericite and chlorite (see Plate 13).

Molybdenite is the dominant mineral. It occurs as veinlets and aggregates in association with chlorite, quartz and sericite. It also commonly occurs together with uraninite, brannerite and pyrite at the contact with altered andesite and crosscutting carbonate veinlets. Molybdenite is also found associated with the uranium minerals and pyrite. U-Mo mineralization cuts Fe-dolomite veinlets, calcite and quartz, latter with younger sulphides (chalcopyrite, pyrite and tennantite).

Metal concentrations are variable and high. From the lithogeochemical studies (drill holes 1247 and 1248), the following contents were detected: 660-4500 ppm Mo, 750-18700 ppm U, 23-765 ppm Cu, 48-393 ppm Pb, 2669-4070 ppm Ti, 24-248 ppm Ni, 99-256 ppm Zr and 114-214 ppm As. The REE content does not exceed 300 ppm.

10. EXPLORATION

Since officially acquiring the exploration licence in question in March 2005, the issuer (Tournigan Gold Corporation) have undertaken no new exploration work other than drilling three diamond drill holes on the property (see description below).

11. DRILLING

Since acquiring the exploration licence, the issuer has drilled 3 relatively deep diamond drill holes on the Jahodna property. The purpose of these holes was to provide preliminary confirmation of the thickness and average grade of the deposit. (see Map 13 for hole locations).

The drilling was undertaken by Geo Technical Consulting of Bratislava. They used a wireline type Prospector II drill for the shallow drilling from surface on all three holes. This track-mounted drill used PQ size equipment, to drill in the region of 100m in each hole. Thereafter, a Longyear 38 drill was used, drilling HQ sized core as deep as possible, and thereafter reducing to NQ.

In view of the difficult drilling conditions (ie. caused by steeply dipping bedding and cleavage planes), the drilling speed was reduced in order to improve the core recovery (average daily metreage achieved was 23m / day). In addition to this, an organic polymer (Premix type, made in France) was mixed with water and used throughout the drilling programme. These precautions helped to maintain a high standard of core recovery throughout the 3 hole programme (ie. 96-98% recovery overall, or almost 100% in the fresh rock).

These holes are:

KG-J-1 - situated within 20 m of two old holes which gave high grade intersections, being Nos. 1218 and 1222. This hole was drilled as follows:

- 0.0-17.5 m - 137mm
- 17.5-75.0 m PQ
- 75.0-347.0 m HQ
- 347.0-440.4 m NQ

KG-J-2 – collared some 80m SE of KG-J-1, within the broad outline of the Jahodna deposit. This hole was drilled as follows:

- 0.0-15.0 m 137 mm
- 15.0-93.0 m PQ
- 93.0-343.5 m HQ
- 343.5-480.4 m NQ

KG-J-1a – situated adjacent to hole no. KG-J-1. This hole was drilled as follows:

- 0.0-18.6 m 156 mm
- 18.6-97.0 m PQ
- 97.0-351.0 m HQ
- 351.0-444.1 m NQ

In the part of the deposit where it was intersected by the drill holes, the dip of the deposit would be in the region of 50 to 60^0 to the SE. The holes were drilled at steep inclinations, starting off near vertical at surface, and shallowing progressively at depth. This would mean that the intersection with the mineralised zone would have been quite close to normal (90^0). For this reason, true width corrections have not been applied to the mineralised intersections from the latest drilling. In addition to this, the Micromine PWRE model incorporates the drill data "as drilled", and effectively turns these into true dimensions when calculating the block model volume.

12. SAMPLING METHOD AND APPROACH

As mentioned above, a wireline coring system was used, with a Prospector rig drilling from surface (PQ core size) and a Longyear 38 rig completing the holes (HQ and NQ size). In view of the known zones of poor ground conditions, the wireline drilling equipment used a double-tube core barrel, in order to maximise core recovery.

The core was geologically logged on site at Jahodna. In addition to this, the mineralised zones were identified with a ZRUP Gamma Logger. Once logging was complete, the core was removed to the company's exploration facility in Kremnica, where the mineralised zones were halved, using a diamond saw. The sample intervals were defined geologically, and on the basis of the gamma logger.

Core recovery was generally very high (always well over 90% average in the mineralised zones, and frequently 100%), and with the gamma logger being used first to define the mineralised zones, it would appear highly likely that all the good zones of uranium mineralisation were identified for chemical analysis.

13. SAMPLE PREPARATION, ANALYSES AND SECURITY

The samples from the first 2 drill holes (KG-J-1 and KG-J-2), totalling 26 core samples, were airfreighted to the OMAC lab in Ireland for analysis. The samples were dried at 85^0 C, jaw crushed to -2 mm and the total amount of crushed material was milled using LM2 mill to -100 µm.

Prepared samples were analysed for 45 element suite using MA/ES procedure, which involves digestion of 0.2 g of sample in the mixture of nitric, hydrofluoric, hydrochloric and perchloric acids, bringing solution to dryness and re-dissolving salts in 10 ml of 10% aqua regia solution followed by reading using ICP-OES spectrometer. The samples were also analysed for gold using Au4 procedure that involves fusion of 50 g of sample with lead collection, cupellation, dissolving resulting prill in aqua regia and AA analysis.

Standard QC procedures were applied. 10 % of samples were analysed in duplicate, blanks and reference materials were analysed along with the samples. Certified reference materials of uranium mineralisation BL-1 and BL-2 manufactured by Canmet were used in multi-element analysis. All QC data were included in test reports.

Because the mineralised interval from the 3rd hole (KG-J-1a) was so rich (over 6% U for the whole interval), it was too high grade to be assayed at the OMAC laboratory. Accordingly, it was sent to the Ecochem laboratory in the Czech Republic (owned by ALS Chemex). There they undertook a spectrophotometric determination of uranium (with an ICP determination of other elements). The final determination of uranium grade was by the David-Gray-Eberle titrimetric method.

RECENT INTERSECTIONS

Drill Hole No.	From	To	Interval	U %
	(m)	(m)	(m)	
0.030%U cut-off				
KG-J-1	406.90	408.10	1.20	0.387
KG-J-2	450.90	452.00	1.10	1.140
KG-J-1a	424.00	424.90	0.90	8.829

14. DATA VERIFICATION

With regard to the original drilling undertaken at Jahodna, it has not been possible for the author to verify this, since no core or other samples of any type remained from this drilling. While in Slovakia in October 2005, the author was able to see the recent Tournigan drilling in action, and to use a ZRUP Gamma Logger to confirm the radioactivity in the dark-coloured mineralised core (this was a minor mineralised zone, not the main ore zone). However, the author was not in Slovakia to see any of the main mineralised intersections from the recent Tournigan drilling, or personally verify them in any other way.

Similarly, the original Jahodna borehole data, when used for the Micromine computer model, could not be verified by ACA Howe. This was because none of these drill samples remained now to verify them.

15. ADJACENT PROPERTIES

There are no known adjacent mineral properties.

16. MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

Since the previous resource calculations at Jahodna were based on a large volume of detailed data, parts of which were not available for examination, it was not possible to conduct a full recheck of all data pertaining to the former resource calculations. For this reason it was decided to conduct a resource estimate based on prior and current drill data.

Accordingly, a Polygonal Wireframe Resource Estimate (PWRE) was calculated for the Jahodna Resource, using data from thirteen (13) historical drill holes over the project area. The thirteen deep historical drill holes, amounting to some 6,290 m of drilling, covered the main part of the Jahodna deposit, though the wireframe model was somewhat constrained in that it did not have all the deposit edge data used for the earlier estimates. Nevertheless, the PWRE appears to have substantially confirmed the previous tonnage and grade estimates.

The Micromine study is written up in more detail in a separate report by G. White of ACA Howe, to which the reader is referred for further detail.

The PWRE work was conducted in two phases – firstly, using only the original Jahodna drill hole data, and secondly, introducing the three new boreholes drilled by Tournigan Gold Corporation. In both cases, the PWRE appears to have broadly confirmed the previously calculated tonnage and grade estimates.

The recent PWRE resource calculation, incorporating the new Tournigan drilling results, can be summarised as follows:

- **(0.030% U cutoff) -1,256,088 tonnes @ 0.56% U to give 15,510,172 lbs of contained uranium.**

Taken together, it would appear that the recent PWRE work on the original Jahodna data, the new Tournigan drilling and the updated PWRE model have all broadly confirmed the previously calculated tonnage and grade estimates at Jahodna. This resource estimation work classifies the tonnage and grade estimate at Jahodna as an "inferred resource".

MINERAL RESOURCE ESTIMATE

Name of Study/Description	Tonnes	U%	Contained Lbs	Comment
Micromine 2006 Study	1,256,088	0.56	15,510,172	0.030% cut-off grade used by ACA Howe

Note that this preliminary assessment is preliminary in nature and it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. The reader is cautioned that there is no certainty that this preliminary assessment will be realized.

17. MINERAL PROCESSING AND METALLURGICAL TESTING

No current metallurgical test work has been undertaken.

18. OTHER RELEVANT DATA AND INFORMATION

18.1. MINING

18.2. PRODUCTION RATE

Based on the estimation of PWRE resource calculation referred to above, an annual mine production of 100,000 tonnes of run-of-mine ore has been used as a base production case.

This has not been optimally determined, and the number has been arrived at to give a mine life of about 10 years, allowing for minor stoping losses. This annual tonnage figure can be refined during the development of the project, as further information becomes available. However, an alternative production rate of 150,000 tonnes per annum (tpa) was tested during the preparation of this report, and it was apparent that it would be difficult to develop the mine at such a rate to enable the establishment of sufficient production faces to sustain the higher figure, without resorting to the use of a highly experienced and thus more costly workforce. This is especially true given the amount of development required for the current ventilation method. As a result, it is considered that the 100,000 tpa case is probably close to optimum.

A grade factor of 95% has been applied to the ore resource grade to allow for dilution losses.

18.3. MINING METHOD

A base-case mining method has been selected - undercut and fill with cemented paste fill, worked in a descending sequence. The choice of this method will ensure that work always takes place under a cemented roof. This is important in view of the potentially poor ground conditions in the mineralized zone. From drill core information, the mineralized zone is highly sheared and incompetent. The undercut and fill method also ensures that there is a high percentage extraction of ore and low dilution.

The mining infrastructure has been sited in the andesite immediately in the hanging wall of the mineralized zone as a conservative measure until the competency of the footwall can be established. The andesite unit is between 20 – 50 m wide, and will enable the establishment of the access ramp, together with the stope access development and the necessary ventilation shafts and drives. It is recommended that further work be carried out to determine the competency of the various rock units.

The mine will be accessed by a main decline, with a gradient of 10% and a size of 5 m by 5 m. Mining levels will be established at 50 m vertical intervals and a hanging wall mining drive will be developed in the andesite at each level. From the mining drive, a steep access ramp will be driven up to the top of the mining section. From this ramp access, the stope drive, in ore will be developed to the stope boundary. There will be 5 accesses along strike and therefore 5 stopes. The stope will be drilled with the drill jumbo, fired and loaded out with a diesel scoop-tram. Ore will be loaded to a tip arrangement for loading into a diesel haul truck. The stope drive will be 4 m high and the width of the ore. Stopes will be drilled to 3.6 m, and each blast is expected to take 3.4 m. In an average stope width of 2.2 m, a stope face will produce about 80 t per blast. Thus, at a production rate of 100,000 tpa, equivalent to 335 tonnes per day (tpd), between 4 and 5 faces must be drilled, blasted and loaded per day.

When the stope reaches the panel boundary, a bulkhead will be built across the stope access. Each stope will be filled with a cemented paste fill. This will be made up of crushed mine waste and cement. The lower half of the stope will have a cement ratio of up to 250 kg per m3 of fill and the upper half a cement ratio of perhaps 170 kg per m3 of fill. The exact cement ratios will be determined in tests, but the higher ratio of cement on the floor of the stope will produce a stronger fill for the roof of the stope below. It is assumed that 0.7 t of fill is used for 1 t of production.

The use of steel reinforcement should also be tested.

After a suitable period of time for curing of the fill, the slice below will be mined in a similar fashion.

Figures 1, 2 and 3 illustrate the conceptual layout of the mine.

A tipping arrangement will be constructed at each mining level for the scooptram to tip into. The tip will have a steel control door to enable the haultruck to be loaded and the ore will be hauled out to surface.

In parallel with the mining drive, and at a different elevation, there will be a ventilation drive constructed. This is to enable the rapid removal of used ventilation air out of the mining section. This is discussed in section 5.6.1 below.

18.4. MINE DEVELOPMENT

The following table summarises the mine development required for mine operations.

MINE DEVELOPMENT REQUIRED FOR MINE DEVELOPMENT

		years											
		-2	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
mining level				400	350	300	200	150	100	0	-50	-100	-200
decline quantity to level	m			1000	1500	2000	3000	3500	4000	5000	5500	6000	7000
main level length	m			500	500	500	500	500	500	500	500	500	500
portal construction	m3	5600											
total ramp development 5*5	m		1,000	500	500	1000	500	500	1000	500	500	1000	
level drive 4*4	m		500	500	500	500	500	500	500	500	500	500	
stope accesses 4*4	m		125	125	125	125	125	125	125	125	125	125	
vent shafts	m		400	100	100	200	100	100	200	100	100	200	
vent crosscust 4*4	m		60	60	60	60	60	60	60	60	60	60	
vent collector drive 4*4	m		500	500	500	500	500	500	500	500	500	500	
tipping arrangement	m3		55	55	55	55	55	55	55	55	55	55	
access to tipping plus extras 4*4	m		191	191	191	191	191	191	191	191	191	191	
linear metres	m		2,376	1,876	1,876	2,376	1,876	1,876	2,376	1,876	1,876	2,376	
shaft metres 4 m2	m		400	100	100	200	100	100	200	100	100	200	
Tips	m3		100	100	100	100	100	100	100	100	100	100	
Development tonnes			141,821	97,572	97,572	134,988	97,572	97,572	134,988	97,572	97,572	134,988	

Notes

1. Development rate (single heading) 6 m per day
2. Development rate (multi heading) 9 m per day
3. Ventilation shaft sinking 2 m per day.

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18.5. MINING EQUIPMENT

MAIN ITEMS OF MINE MOBILE EQUIPMENT

Mobile equipment		**-2**	**-1**	**1**	**6**
Boomer	number		1	1	
	$000		528	528	
Scooptram ST2D	number		1	1	2
	$000		225	225	450
Haultruck MT2000	number		1	1	1
	$000		367	367	367
Boltec MC	number		1		
	$000		564		
Scaler	number		1		
	$000		260		
Wheel loader	number		1		
	$000		250		
Light vehicles	number	3	2	2	4
	$000	60	40	40	80
People carrier	number		1		1
	$000		100		100
Flat bed utility	number		1		
	$000		100		
Subtotal	$000	60	2,433	1,159	997
Delivery and taxes	30%	18	730	348	299
Mobile equipment capex summary	**US$000**	**78**	**3,163**	**1,507**	**1,296**

Note: a third and fourth Haultruck will be required for development and production haulage in year 6 of the project as the mine progressed deeper. The other Year 6 requirements are for normal equipment rebuilds and replacements.

Illustrative prices were obtained from Atlas Copco, and are based on the following Atlas Copco models:

MINING MACHINERY

Single boom jumbo	Rocket Boomer L1
Diesel LHD 2m3	Wagner ST2D
Diesel haul truck 20 tonne capacity	Wagner MT2000
Rockbolter	Boltec MC
Wheel loader	Caterpillar 970

The prices were quoted in sterling and converted to US$ at the rate of £1 = US$1.73.
The choice of equipment reflects the low rate of mine production and the need for flexibility. It is important that equipment is specified with pressurised operators' cabs. This will influence the final equipment selection.

18.6. BACKFILL PLANT

A small paste fill plant will be constructed to supply a cemented paste fill to the underground stopes. It will produce a rich cement mix to provide sufficient strength for the undercut stopes, so that each stope has a secure roof. The plant cost at present is an allowance, pending a better estimation of capital costs currently underway.

18.7. UNDERGROUND ENVIRONMENT

The mined mineralized material will emit radioactive decay particles. This will consist of alpha, beta and gamma radiation. In addition, radon gas will be produced and this has the property of attaching itself to dust particles and thus has the potential to be inhaled by personnel. Radon gas is also soluble in water and will be dissolved in any water drainage.

Mitigation methods will be as follows:

18.7.1. VENTILATION

Ventilation will be force and exhaust to ensure positive airflow in all parts of the mine. Thus 2 surface fans are envisaged. Air will typically be one-use. This means that once air has passed through an area of potential radiation, e.g. a stope or a rock handling point, it will be immediately directed to an exhaust airway. To accommodate this, a parallel exhaust airway will be developed on each level, probably several metres below the main access drive, so that all air has a direct route out of the mine without passing through additional working places.

The exhaust air will be scrubbed with water jets and the water so obtained will be collected and pumped to the tailings management facility.

Auxiliary force fans will be employed in each working area to ensure positive ventilation airflows.

18.7.2. CLOTHING

Alpha particles do not penetrate clothing. Thus, all workers will be required to wear total coveralls. Airstream helmets with visors will be standard issue. Laundry facilities will ensure overalls are regularly and completely cleaned.

18.7.3. MINING EQUIPMENT

All mobile equipment will be specified with pressurised and filtered cabs. Operators will, as far as possible, work in cabs or protected work-places.

18.7.4. DOSIMETERS

Mine personnel will be exposed to radiation during their working day. To ensure that individual workers are not exposed to more than legal levels of radiation, each person will be issued with a personal dosimeter. These will be handed in at the end of each shift and the amount of exposure measured. Once a certain level has been reached in terms of dose and times, the worker will be placed onto other duties, or will be prevented from working in the affected environment until a suitable time lapse. The exact mode of job change or lay-off will have to be determined in conjunction with employees' representative bodies.

18.8. MINE CONSTRUCTION PROGRAMME

It is expected that a two-year pre-production period will be required. The first year (year –2) will enable engineering and procurement, plus the excavation of the mine access portal. The second year (year –1) will be the development of the access decline to the first production level, (+400 m elevation), plus the ventilation raises. All surface construction will be completed in the year –1. The process plant will also have a two-year construction schedule, and therefore production will commence in the third year of the project (year 1).

18.9. MINE PERSONNEL NUMBERS

MINE PERSONNEL REQUIREMENTS

	-2	-1	1	2	3	4	5	6	7	8	9	10
Operations staff	0	17	26	26	26	26	26	29	32	32	32	20
Mine supervision	0	3	4	4	4	4	4	4	4	4	4	4
Backfill crew	0	0	6	6	6	6	6	6	6	6	6	6
Mine maintenance	0	5	6	6	6	6	6	6	6	6	6	6
Technical and engineering	3	7	9	9	9	9	9	9	9	9	9	9
Total mine personnel	**3**	**32**	**51**	**51**	**51**	**51**	**51**	**54**	**57**	**57**	**57**	**45**

Personnel costs have been estimated using data from Tournigan Resources, and using the currency conversion rate of €1=US$1.19. The cost of each employee has been based on the high-end of the range provided, and the costs inflated by 35% to allow for employer's costs. An exchange rate of SK32 = US$1 was used to estimate these costs.

18.9.1. MINE OPERATING COSTS

The basis of the operating costs are shown in the tables below and in the appendices.

MINE OPERATING COSTS

	years								
	-2	**-1**	**1-2**	**3**	**4-5**	**6**	**7-8**	**9**	**10**
	US$000								
Salaries	53	547	837	837	837	887	938	938	736
Explosives	0	166	204	239	204	239	204	239	81
Consumables Diesel	14 42	181 595	1,113 973	1,132 973	1,113 973	1,139 1,169	1,128 1,365	1,147 1,365	1,030 1,024
Electrical power	7	113	669	669	669	669	669	669	641
Mechanical spares and tyres **TOTAL**	24 **140**	221 **1,824**	343 **4,139**	343 **4,193**	343 **4,139**	384 **4,487**	424 **4,728**	424 **4,782**	320 **3,832**
Operating cost per tonne ore			41.39	41.93	41.39	44.87	47.28	47.82	38.32

Note that this preliminary assessment is preliminary in nature and it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. The reader is cautioned that there is no certainty that this preliminary assessment will be realized.

18.9.2. OTHER OPERATING COSTFACTORS

A detailed account of the make up of the mine operating costs are included in reference 14 (see list of references on page 40.)

18.9.3. MINE CAPITAL COSTS

SUMMARY OF MINE CAPITAL COSTS

Capex summary	**year**	**-2**	**-1**	**1**
		US$000		

Mobile equipment		78	3,163	1,507
Site work		356	0	0
Surface infrastructure		617	1,236	375
First fill		22	432	333
Pre-production operating cost		140	1,824	
Total		**1,213**	**6,655**	**2,215**

Note that this preliminary assessment is preliminary in nature and it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. The reader is cautioned that there is no certainty that this preliminary assessment will be realized.

There will be a further additional capital cost in year 6 of $997K. This will be for additional diesel haul-trucks, which will be required as the mine progresses in depth as well as replacements or rebuilds for high-duty mobile equipment

DETAILS OF THE MOBILE EQUIPMENT CAPITAL EXPENDITURE

Mobile equipment		**-2**	**-1**	**1**	**6**
Boomer	number		1	1	
	$000		528	528	
Scooptram ST2D	number		1	1	2
	$000		225	225	450
Haultruck MT2000	number		1	1	1
	$000		367	367	367
Boltec MC	number		1		
	$000		564		
Scaler	number		1		
	$000		260		
Wheel loader	number		1		
	$000		250		
Light vehicles	number	3	2	2	4
	$000	60	40	40	80
People carrier	number		1		1
	$000		100		100
Flat bed utility	number		1		
	$000		100		
Subtotal	$000	60	2,433	1,159	997
Delivery and taxes	30%	18	730	348	299
Mobile equipment capex summary	**US$000**	**78**	**3,163**	**1,507**	**1,296**

PRE-PRODUCTION MINE SITE WORKS, YEAR –2.

Site work	Q	unit cost	total cost	
		US$	US$000	
Site clearance, m2	40,000	2	80	
Roads, m	1,000	50	50	
Temporary buildings	4	10,000	40	
Drainage works	1	10,000	10	Allowance
Fencing, security	1	100,000	100	Allowance
Portal earthmoving, m3	5,600	10	56	
Portal concrete, steelwork	1	20,000	20	
Site work summary			**356**	

SURFACE INFRASTRUCTURE

Surface infrastructure		US$000	US$000	-2	-1	1
Electrical substation	1	297	297	297		
Electrical reticulation underground	1	575	575		575	
Main vent fans including installation	2	50	101		101	
Ventilation scrubber	1	150	150		150	
Rockbreaker	1	80	80		80	
Water storage and reticulation	1	125	125		125	
Cemented paste plant	1	375	375			375
Buildings	3	100	300	200	100	
Explosive magazine	1	75	75		75	
Computers, software	1	100	100	100		
Office equipment, furniture, telecomms	1	50	50	20	30	
Surface infrastructure summary			**2,228**	**617**	**1,236**	**375**

FIRST FILL UNDERGROUND MINE

First fill			**$000**	**-2**	**-1**	**1**
Hoses , fittings etc	1 lot	2	2	2		
First fill pipes	1lot	20	20	20		
Auxiliary fans	6	1	6		3	3
Pumps	4	5	20		20	
Electrical gear	1 lot	100	100		100	
Other equipment	1 lot	100	100		100	
Diesel (3 months)	3		244		149	95
Explosives (3 months)	3		51		42	9
Consumables (3 months)	3		224	.1	8	216
Protective and safety gear	1		20		10	10
Total			**787**	**22**	**432**	**333**

Note that this preliminary assessment is preliminary in nature and it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. The reader is cautioned that there is no certainty that this preliminary assessment will be realized.

18.9.4. PROCESS

18.9.5. PROCESS EQUIPMENT

The choice of the process equipment and their specifications will depend on the quantitative process flow sheet adopted for ore processing. A typical simplified flowsheet (see Table 11) has been provided to illustrate the derivation of operating and capital costs for this study. Further metallurgical work will be required. The data available is limited to a very brief document entitled “Technological Research of U-Mo from Jahodna site” by Jan Kopecky, October 1993. Although the data are related to results from just a few samples from the Jahodna site it is considered that the main conclusions of the report are:

- Sulphuric acid leach of the ore for uranium extraction would result in high acid consumptions with problems associated with the disposal and treatment of dissolved gangue components; the indicated acid consumptions were of the order of 300kg/t.

- Well-proven carbonate pressure leaching would be economically and technically viable for leach extraction of uranium contained.

Standard carbonate leach of the comminuted ore (P80 – 0.15mm) at 140C at about 8 bar pressure with soda solutions and air as the oxidant has been shown to result in the extraction of about 90% of the U and the Mo contained in the ore. The report referred to above had proposed a novel 2 stage leach for the production of U yellow cake and moly sulphide concs. It is believed that the following industrially well proven process route is preferable from economic (lower capex) and technical perspectives:

- Comminution of ore to sizing of P80-0.15 mm with two stage crushing,and one stage of wet ball milling with recycled leach liquors (grading about 3 g/l U and about 30 g/l soda and bicarb of soda)
- Pressure leaching with sodium carbonate liquors (140 C 8 bar pressure and 8 hours residence time)
- Filtration and wash of the leach pulp and management of the solid leach residues (storage in a monitored and managed tails repository- tailings area/ sub- aerial deposition)
- U/Mo separation with the precipitation of uranium by caustic additions to pH11
- Precipitation of MoS_2 with NaSH from the U free liquor from above with pH adjusted to about 3 with sulphuric acid; filtration and wash of the moly sulphide produced followed by drying if necessary (Note that moly is potentially a saleable by product but has not yet been estimated as historical moly data is not verifiable)
- Redissolution of the precipitated U in acid and production of UO_4(yellow cake)with hydrogen peroxide followed by filtration and drying of the yellow cake produced .
- The main liquid effluent from the processing stages will be sodium sulphate bleed liquors (to maintain solution and sodium balance); the sulphate can be concentrated and solid sodium sulphate of marketable quality can be produced.

The attached table lists the major equipment that are likely to be used for the project plant, along with the simplified flowsheet. It must be emphasised that the assumed duty specifications are very preliminary and are intended to be merely indicative.

PROCESS AREAS AND MAJOR EQUIPMENT

Comminution	Crushing	Jaw/impact crusher 70 t/h Secondary cone crusher 20 t/h close circuit;P80 12-15 mm
	Milling in recycled leach liquors	Ball mill to P80 - 0.14 mm 2.2M Ø 2.4 M 150 KW
	Settling u/f to leach below	Thickener 5MØ
Leaching	Carbonate leach 140 C about 35- 40 % solids Air addition 40Nm ^{3}/t.h	Carbon steel autoclaves 30M ^{3} 6 off; turbine impellers;15KW motors
Solid/liquid separation	filtration and wash liquor recycle to mill	40-30 M ^{2} filter area equivalent filter press
Tails disposal	sub aerial deposition of leach tails	Lined retention area managed and monitored
Preconcentration uranium precipitation and filtration	acid and caustic addition 2 stage adjustmentpH to forward flow to 5 and 12; 4 h retention time. stage 1 U/Mo separation	agitated tanks CS or FRP Filter press as per manufacturers tests rated at about0.3 t/h dry solids basis
Molybdenum recovery from deuranised liquors from above	NaSH additions at pH3 (by acid additions; collection and scrubbing of H_2S generated with alkali residence time of 3 h.	agitated reactors of FRP sieve plate scrubber for H_2S collection and reuse.
Moly sulphide filtration	moly sulphide from above settling/ filtration	a lamillae thickener and filter press 250 kg/h solids
spec yellow cake production	dissolution of conc U from above in acid; UO_2 by H_2O_2 at pH3.0 bycaustic additions Filtration and drying of the above yellow cake product.	agitated FRP reactor, filter press as above rotary dryer with indirect heating. Product exportation bag loaders.
Liquid effluent treatment	liquors ex UO_2 and moly circuits sodium sulphate liquors to repulp leach tails for transport ot TRS	

18.9.6. PLANT CAPITAL COST ESTIMATES

The capital cost of a plant with the above-listed equipment and facilities is indicative of the capex requirements at the typical accuracy for a Preliminary Assessment of +/- 30%. Carbonate based U recovery plants with agitation leach as the primary extraction stage are relatively rare, although the technology is well proven historically. A consequence of this is that capital cost estimates need to be derived from first principles requiring detailed definition of process and engineering.

Based on the recent available cost data from similar hydrometallurgical plants the estimated capital cost for the facilities for the treatment of 100,000tpa of ore grading 0.589% U and about 0.5% Mo producing yellow cake and by product molybdenum sulphide is US$9M or $90/annual tonne of ore. Again, it should be noted that costs associated with the moly by-product have been included but revenue/cash-flow for moly has not. A further US$0.5M is included for the initial construction of the tailings management facility. Indirect costs for the plant of US$1.9M include the engineering and vendors costs in the construction period. A first fill equal to 3 months of operating costs is added prior to year 1 and included as a capital cost. This is US$1.12M.

18.9.7. PLANT OPERATING COST ESTIMATE:

The following table summarises the estimated operating costs of the conceptual plant in US$ per tonne of ore.

ESTIMATED OPERATING COSTS OF THE CONCEPTUAL PLANT IN US$ PER TONNE OF ORE

Reagents	kg/t	cost per
	ore	tonne ore
		US$
Caustic	20	6
Soda ash	50	5
Hydrogen peroxide	5	4
Others		4
Total reagents Power kWh/t	 80	19 7.4
Steam GJ/t	0.8	5
Labour		4.6
Tailings management		3
Others		6
Total operating cost per tonne		45

It is important to emphasise that the above estimate is for the directs mentioned and does not include cost items which could be significant eg management overhead costs, product transport costs etc, which ACA Howe estimated in the financial model below.

Mill includes 3 months first fill
Indirects not applied to first fill

Note that this preliminary assessment is preliminary in nature and it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. The reader is cautioned that there is no certainty that this preliminary assessment will be realized.

18.9.8. CASH FLOW MODEL

		year	-2	-1	1	2	3	4	5	6	7	8	9	10
Total ore mined and treated	000 t			0	100	100	100	100	100	100	100	100	100	100
Waste mined	000 t			142	98	98	135	98	98	135	98	98	135	0
Total excavation	000 t			142	198	198	235	198	198	235	198	198	235	100
Mill head ore grade %U					0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45
Mill head ore grade %Mo					0.36	0.36	0.36	0.36	0.36	0.36	0.36	0.36	0.36	0.36
Uranium recovery %					0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85
Molybenum recovery %					0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85
Yellowcake produced	t				383	383	383	383	383	383	383	383	383	383
Molybenum disulphide in con	t				306	306	306	306	306	306	306	306	306	306
Net revenue U	US$M	US$35/lb			22.1	29.5	29.5	29.5	29.5	29.5	29.5	29.5	29.5	29.5
Net revenue Mo	US$M	US$15/lb												
Total net revenue	US$M				22.1	29.5	29.5	29.5	29.5	29.5	29.5	29.5	29.5	29.5
Total operating cost	US$M				9.1	9.1	9.1	9.1	9.1	9.4	9.7	9.7	9.7	8.7
Total capex	US$M		3.1	22.6	3.7	0.0	0.0	0.0	0.0	1.0	0.0	0.0	0.0	0.0
Change in working capital	US$M		0.8	4.9	-4.7	-0.9	0.0	0.0	0.0	0.3	-0.3	0.0	0.0	0.0
Pre-tax Cashflow	US$M		-3.9	-27.5	14.1	21.4	20.4	20.4	20.4	18.8	20.1	19.8	19.8	20.8
Cum. Pre-tax Cashflow	US$M		-3.9	-31.4	-17.2	4.1	24.5	45.0	65.4	84.3	104.3	124.2	143.9	164.7
Operating profit ($M)					13.1	20.4	20.4	20.4	20.4	20.1	19.8	19.8	19.8	20.8
Depreciation ($M)														
Tax rate (%) and tax ($M)	0.19				2.5	3.9	3.9	3.9	3.9	3.8	3.8	3.8	3.8	3.9
Post tax Cashflow ($M)			-3.9	-27.5	11.7	17.5	16.5	16.6	16.6	15.0	16.3	16.1	16.0	16.8
Cum. Post tax Cashflow ($M)			**-3.9**	**-31.4**	**-19.7**	**-2.2**	**14.3**	**30.8**	**47.4**	**62.4**	**78.7**	**94.8**	**110.8**	**127.6**
Pre-tax NPV ($M) at 8% discount rate	**84.59**	**Pre-tax IRR**	**54%**		**Post-tax NPV ($M) at 8% Discount rate**			**63.54**	**Post-tax IRR**	**44%**		**Post – tax pay back 25 months**		

Note that this preliminary assessment is preliminary in nature and it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. The reader is cautioned that there is no certainty that this preliminary assessment will be realized.

Notes for the cash flow model:

1. Salaries, fuel and power costs are provided by Tournigan
2. Owner's management overhead is estimated by Tournigan as 5% of operating cost
3. Income tax is estimated by Tournigan as 19%.
4. Product revenues are $35/lb uranium as estimated by Tournigan. This is the fob price at the mine gate.
5. Revenues are delayed by 3 months but no allowance has been made for the delayed contribution after year 10.
6. Discount factor used for NPV estimation is 8%
7. No site remedial cost has been estimated, since there is every possibility that production will continue after year 10.
8. Post tax calculations do not take into account tax credits or deferred capital.

19. INTERPRETATION AND CONCLUSIONS

Geological Complexity – The Jahodna uranium deposit represents a difficult exploration target, for a number of reasons:

- It is a partially blind deposit, with limited surface expression other than a subdued radiometric anomaly, limiting the exploration methods available to explore it.
- It is deep – lying, in the range of 250 to 650m below surface – this means that it will be an expensive target to evaluate, since it will be done mostly by deep drilling from surface, or by shorter holes from underground access.
- The surface in the area of the deposit is hilly, with deep soil cover and thick forest, and the forest appears to be old and well established. At these elevations, snow lies on the ground during the winter, sometimes for more than 3 months per year. A combination of these factors virtually negates surface exposure, and make useful surface mapping harder to achieve.
- The geology of the deposit itself is in some ways complex, and is as yet incompletely understood. For example:
    - Although the main deposit lies at the base of the meta-andesitic unit, its relationship with the other mineralised bodies within the andesite unit are not well understood.
    - There are significant cross faults which displace the mineralised zone by up to several tens of metres, thus effectively breaking up the mineralised body into faulted blocks – it is important that the relative positions of these mineralised blocks be determined more accurately before evaluation proceeds too much further.
    - The same cross faults may have been conduits for the U-Mo-Cu mineralisation, yet the spatial relationship between the cross faults and the uranium deposit has not been properly established.

- In the case of the Jahodna uranium deposit, which appears to be localized along zones of deformation related permeability within the Permian meta-volcanosedimentary sequence, the detailed grade distribution within the mineralised zone is likely to be complex – but the drilling to date appears to have been too irregularly and widely spaced to clearly quantify such variability.

- Regarding geological factors which might affect the development of the Jahodna resource, an obvious one is the existence of significant cross faults – these faults run across the main deposit, and locally may offset it by up to several tens of metres. At the least, the positions of these faults are likely to delimit sections of a future underground mine, so detailed knowledge of their positions will likely become vital in the ongoing evaluation of the deposit. At worst, the cross faulting may render parts of the deposit inaccessible for mining purposes.

- With regard to the ways in which the Jahodna resources might be affected by non-geological factors (eg. any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues), the only one of which the author is aware is that the deposit occurs within a forest reserve. However, Tournigan staff in Slovakia claim that this is not a critical or prohibitive issue with regard to the future of the project.

Other Geological Features of the Jahodna Deposit

- It is possible that the likely source of the uranium mineralisation at Jahodna was the Permian acid-intermediate volcanic sequence, with the U-Mo-Cu mineralisation infiltrating out from the volcanic centres into suitable structural / lithological traps during deformation and metamorphism. In the case of Jahodna itself, the most obvious depositional loci would appear to be zones of permeability, both within the meta-andesitic volcanic unit and in the foliation planes and localized shearing along the basal contact of the meta-andesitic unit.

- The complex mineralisation at Jahodna comprises U-Mo-Cu, with lesser Fe-Pb-Ti-P-Ba. The mineralisation at Jahodna has been described as a "Saddle Hills analogue", after the Saddle Hills uranium deposit in Mongolia. Saddle Hills contains uranium-rutile-leucoxene mineralisation in a variety of Mesozoic volcanosedimentary lithologies. Although there is insufficient detailed information to back this up, there would appear to be some broad similarities between Jahodna and the Saddle Hills type deposits.

- It appears that the good mineralisation at Jahodna is generally clearly visible, with dark brown and grey / black mineralisation resulting from hematite, graphite, chlorite, the uranium mineral brannerite, as well as pyrite, chalcopyrite, molybdenite etc. Obviously, this is helpful for core logging and sampling purposes.

- The majority of the rocks at Jahodna are meta-volcaniclastics/tuffs of acid-intermediate type, being predominantly andesitic in nature. Most of the rocks are strongly foliated, and therefore schistose, with schistosity generally appearing to parallel the now steeply dipping original bedding planes of the volcanosediments. This combined bedding / schistosity gives rise to poor ground conditions in some of the rock types encountered in the vicinity of the Jahodna deposit, which could impact on the underground exploration and possible future mining stages.

Positive factors on the Jahodna Project

- After over 20 years of being depressed, the international uranium market is riding high at present, and shows every indication of continuing to do so. Conventional major sources of power (coal, hydrocarbons etc) will not be able to meet the worldwide energy requirements, making the case for a return to nuclear power very strong. Furthermore, uranium enjoys some significant advantages over coal and hydrocarbons – ie. no atmospheric pollution, significant ability to recycle nuclear fuels, relatively abundant known undeveloped resources etc.

- Slovakia is a traditional mining country, whose state-subsidised mining industry suffered devastating cutbacks following the cessation of state mining subsidies after the Velvet Revolution in 1989. The general impression is that the Slovak Government are keen to promote a return to mining, provided it can be financed by free market sources.

- Following the switch to communism in 1948, and the consequent effective economic embargo with the West, the Slovakian authorities encouraged and financed vigorous exploration programmes for a wide range of mineral commodities. This means that a large amount of regional scale grass roots exploration has already been done over much of the country, and a large number of mineral prospects (such as Jahodna) have already been discovered.

- Since the deposit occurs within a forest reserve, its relatively deep nature is probably an advantage in that an underground mine in this situation would more likely receive planning permission from the regulatory authorities than an open pit mine.

Past Exploration Work

- In many ways, it is to the credit of Uranovy Prieskum that they discovered the Jahodna deposit at all. Such are the complexities of the setting of this deposit that it was only after considerable exploration drilling that it was discovered in the 1980's. Following airborne radiometric surveys and ground follow up radiometric surveys, the best way forward was found to be to systematically drill the airborne and ground anomalies. However, it must be borne in mind that at that time, there was a high level of government support for mineral exploration, especially for a commodity like uranium, which would have been identified as a commodity of strategic importance during the Cold War. For this reason, considerable sums must have been expended on the exploration for strategic commodities like uranium. Apparently on the Jahodna project, a total of 53 exploration holes, totalling over 17,000m of drilling, were completed between 1985 and 1991.

- During the Soviet-dominated communist era, many mining operations were commenced and operated more for strategic than for solely economic reasons. As such, it is likely that less economic parameters were considered during the exploration and development of most mineral deposits in the USSR, than for equivalent mining operations in the West. For this reason, the detailed economics of mining an underground deposit like Jahodna would probably never have been studied previously.

- Having established that deep drilling was perhaps the only way to evaluate the Jahodna deposit, it has to be said that the drilling undertaken by Uranovy Prieskum was unsatisfactory in some key respects. In the days of pre-wireline drilling equipment, the thin-walled, conventional B-size drill string deviated considerably from the targets, often intersecting the mineralised horizon many tens of metres or more from the intended point of intersection. In a deposit where it would appear that post-mineralisation normal faulting has broken up the mineralised zone into blocks, the positions of which are likely to be of critical importance at the mine planning stage, this lack of directional drilling control was unfortunate.

- Perhaps largely because of the directional problems with the earlier drilling, the resulting drill cross sections of the original evaluation were overall not normal but somewhat oblique to the plane of the deposit. Although such problems frequently arise in early exploratory drilling on a new prospect, they should generally be rectified in the more advanced stages of exploration.

- Furthermore, in drilling a strongly foliated, tuffaceous meta-volcanosedimentary sequence, the core recovery of the drilling was generally poor. This was unfortunately particularly the case in the mineralized zone, where shearing on the base of the meta-andesite unit resulted in strongly broken ground in places, and consequently core recoveries in the region of no more than 50% (though this was improved towards the very end of the drilling after 1990). For this reason, there were apparently no complete mineralized zone intersections from the original 14 holes drilled in the immediate vicinity of the Jahodna deposit. In any surviving core samples from the mineralised zone, variable amounts of available material was taken for chemical assays, resulting apparently in no drill samples of any description remaining to be available for inspection today. In a potentially significant, high grade uranium prospect, with a total of 17,000m drilled on the entire exploration licence area (equating to well over USD 1 million in today's costs), this was an unfortunate shortcoming.

- With a record of persistently poor core recovery in the mineralised zone, the only way to attempt to establish the average widths and grades of the mineralised zone was by down hole radiometric logging. Fortunately this technology appears in itself to have worked well – though it has to be emphasised that there were no complete, assayed borehole intersections at Jahodna against which to calibrate the gamma logger response. However, all aspects of the tonnage and grade calculation at Jahodna appear to have adhered to strict technical guidelines set down by the national mining regulatory bodies at that time, and also draw on former radiometric logging experience of drilling and mining similar deposits within the country.

- In terms of systematically establishing the true average grades of the mineralised zone at Jahodna, or important geological aspects like the detailed positions of cross faults, the earlier drilling undertaken by Uranovy Prieskum would in my opinion have to be regarded as somewhat preliminary in nature.

- Because of the generally poor outcrop at surface, the reconstruction of the late, cross cutting faults has been very much on a 'best fit' basis, with uncertainties regarding their exact positions and throws. Since these cross cutting faults effectively break up the deposit into different segments, the positions and offsets of these faults will be of critical importance in a potential future mining scenario. Since it would appear that the former tonnage and grade calculations have not taken into account the positions and offsets of these faults, it is recommended that the tonnage and grade be recalculated bearing this in mind.

- Within the area of the Jahodna deposit itself, the past drilling has been both irregularly and quite widely spaced. Within the best area of mineralisation, which is the deeper, NW part, drill density works out at approx. 8 intersections within an area pf 100,000m^2 – equating to drilling on a 110m x 110m grid. Within the wider deposit, the equivalent figures are 14 intersections within an area of approx 250,000 m^2, equating to drilling on a 130m x 130m grid. It would seem that by the traditional Soviet type reserve classification for a stratiform but irregular replacement type body, these would have equated to a "C_2 reserve" or an "inferred resource" in current Western terminology (eg. JORC or CIM comparable).

- One prominent feature highlighted by the drilling at Jahodna is the poor ground conditions existing in much of the ground near the deposit itself. However, these bad ground conditions appear to be largely restricted to the thick hanging wall volcaniclastic sequence. Although the mineralised zone itself appears to be tectonised, it seems to be largely supported by the competent meta-andesite body which immediately overlies it and the meta-sediment footwall.

Former Resource Calculations

- The first major resource calculation was undertaken in 1996 by Jozef Daniel, a geologist experienced in the uranium industry of the former Czechoslovakia. The method used a block model method, with two variants based on different minimum cutoff grades (0.015% and 0.03% U). A total of 14 drill holes were used in the model, with average uranium grades established largely by downhole geophysical logging. The calculation methodologies were constrained by government mining directives, established in 1987 and updated in 1992. The calculation concentrated on the main deposit, which was placed in the category of "Z-3 supposed reserves" the lesser zones of mineralisation in the hanging wall of the main deposit were assigned to the lower "prognostic" category. *

- The tonnage and grade calculation was updated in 2005 by the same author, in a report entitled "Calculation of Reserves Deposit Kosice I, U – Mo Ore", which was translated into English. This update derived the following summarised figures:
    - Variant I – (0.015% U Cutoff) – 2.188 Million tonnes at 0.329% U to derive an in situ figure of 15.876 Million lbs of U.
    - Variant II – (0.030% U Cutoff) – 1.396 Million tonnes at 0.472% U to derive an in situ figure of 14.528 Million lbs of U.

*** Note that the Slovak category of Z-3 is roughly analogous to the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") definition of inferred resource. Although this previous work is considered relevant, exploration did not result in a current resource estimate and these are not CIM defined resources, therefore readers are cautioned not to rely upon these estimates.**

- Molybdenum grade calculations were made as well as uranium (with a calculated average figure of 0.38% Mo). However, with the poor core recovery, and lack of an alternative method for determining Mo grades, the volume of data available to determine Mo grades was reduced, for which reason the Mo grades calculated are regarded as less reliable than the uranium. There is also an unresolved question regarding the detailed distribution of the Mo mineralisation within the Jahodna deposit.

Note that this preliminary assessment is preliminary in nature and it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. The reader is cautioned that there is no certainty that this preliminary assessment will be realized.

Recent Exploration Drilling by Tournigan Gold

- Three deep diamond drill holes (totalling 1,364.90m) have recently been drilled by Tournigan into the Jahodna deposit. The purpose of these holes was to provide preliminary confirmation of the thickness and average grade of the deposit. The drilling was undertaken by Geo Technical Consulting of Bratislava, using a tracked Prospector rig to start the holes (with PQ coring and casing), and a Longyear 38 to complete the holes to final depth.

- In contrast to the earlier drilling of the 1980s, this recent drilling achieved good directional control and excellent core recovery. Mineralised core was delineated with a gamma logger, and subsequently cut with a diamond saw. Half of the mineralised samples were sent for assay, to the OMAC laboratory in Ireland, and the Ecochem laboratory in Czech Republic. Results from this drilling substantially confirmed previous drilled widths and uranium grades.

Recent Resource Calculation

- Since the previous resource calculations at Jahodna were based on a large volume of detailed data, parts of which were not available for examination, it was not possible to conduct a full recheck of all data pertaining to the former resource calculations. For this reason it was decided to undertake an in-house Micromine computer study of the available Jahodna drill data, in order to make an independent assessment of the previous resources calculation.

- Accordingly, a Polygonal Wireframe Resource Estimate (PWRE) was calculated for the Jahodna Resource, using data from thirteen (13) historical drill holes over the project area... The thirteen deep historical drill holes, amounting to some 6,290 m of drilling, covered the main part of the Jahodna deposit, though the wireframe model was somewhat constrained in that it did not have all the deposit edge data used for the earlier estimates. Nevertheless, the PWRE appears to have broadly confirmed the former tonnage and grade estimates.

- The PWRE work was conducted in two phases – firstly, using only the original Jahodna drill hole data, and secondly, introducing the three new boreholes drilled by Tournigan Gold Corporation. In both cases, the PWRE appears to have broadly confirmed the previously calculated tonnage and grade estimates.

- The recent PWRE resource calculation, using the new Tournigan drilling results, can be summarised as follows:

  - (0.030% U lower cut-off) -1,256,088 tonnes @ 0.56% U to give 7,034 tonnes or 15,510,172 lbs of contained uranium.

Future Operations

The ACA Howe Preliminary Assessment report concludes that due to the geometry of the deposit the optimum mine production rate should be 100,000 tpa using a horizontal undercut and fill mining method with cemented paste fill; processing of the ore would be by the well proven carbonate leach method which should result in extraction of approximately 90% of both U_3O_8 and MoS_2, though this will be confirmed by testwork. The mine life, based solely on the current resource, would be 10 years.

When in normal production, the mine and plant will produce and treat 100,000 tonnes per year (tpa) of ore at a mill grade of 0.45% U and an as yet underterminable grade of Mo. This allows for a grade factor of 95% for dilution. Production of U in yellowcake will total 383 tpa and Mo as molybdenum sulphide unknown (if any).

The capital cost of building the mine and plant, including the acquisition of all operating equipment is US$29.4M. All costs in this report are in US$. Currency factors used are those pertaining during March 2006. The capital cost is made up of US$10.2M for the mine, US$13.8M for the plant and tailings management facility and US$5.4M for infrastructure and plant indirects. There are no estimates in this report of project acquisition or other sunk owners costs. There will be sustaining capital and replacement capital in the mine of approximately US$1M during year 6 as the mine becomes deeper. The costs are summarised in the table below.

TOTAL CAPITAL COST SUMMARY

Total capex summary	**year**	**-2**	**-1**	**1**	**total**
		US$000			
Mine		596	5,419	1,840	7,855
Mill		1,000	9,125	500	10,625
Infrastructure		617	1,236	375	2,228
Mill indirects (20%)		200	1,600	100	1,900
Sub-total		2,411	17,379	2,816	22,606
Contingency (30%)		723	5,214	845	6,782
Total		**3,134**	**22,593**	**3,661**	**29,388**

The direct operating costs are US$92.5 per tonne, made up of mining direct costs of US$43 per tonne and process costs of US$45 per tonne, with US$4.5 for management overhead. The mining and processing costs are relatively high because of the low production rate. This production rate is determined by the nature of the mineralized zone, and is probably close to the optimum figure. Nevertheless, the ore is of high value, and it is probable that these high costs can be absorbed.

There are operating costs attributable to the need to deal with the radioactive ore. Extra precautions are specified to mitigate the affects of gamma radiation particularly, but more detailed work will need to be done on this aspect of the operation. Another contributing factor to high costs is the proposal to use cemented fill in the stoped out areas. It is possible that cheaper alternatives might be available to reduce the high cost of Portland cement used in the estimate.

ACA Howe has assumed the availability of skilled personnel in all areas of the operation, and the obtaining of all the necessary permits and licences. This is the responsibility of Tournigan and it is assumed that this aspect of the project is being dealt with separately and with positive results.

Notwithstanding all this, there is probably a viable project to be built at this site, especially considering the strong market in uranium yellowcake.

20. RECOMMENDATIONS

- Although surface exploration at Jahodna is made difficult by a number of factors (deep soil cover, very little outcrop, extensive forest cover etc), there are possible ways where further work at surface might be feasible. With the importance of the large cross faults, both as feeder zones and in breaking up the mineralised zone into structural blocks, the positioning of these cross faults with maximum precision becomes critical. One possibility to consider may be to fly high resolution aerial photography during the 'Spring window' – when the snow has melted but before the forest vegetation has grown. Such photography would hopefully be able to see down to the ground surface with sufficient precision so that a stereoscopic study of such photography might position the cross faults with greater accuracy. Another possibility may be to acquire high precision satellite imagery (eg. IKONOS, Aster) at this spring window period, and process the imagery in software such as ER Mapper's Hill Shade function, to maximise surface structural contrasts. Once the cross faults can be positioned at surface with maximum precision, their dips should be projected to depth based on structural knowledge of the area. In this way a working structural model of the area can be developed, to be modified with the results of the ongoing drilling programme. Hopefully in this way, the exploration programme can proceed taking due regard of the detailed positions of the important cross faults.

- If the remote sensing methods mentioned above are not feasible, another method to pinpoint fault structures may be to undertake detailed ground magnetic surveys, though a tight grid (eg. 50m x 10m or 50m x 25m) would be recommended.

- Even though the work to date suggests that the Jahodna deposit can be regarded as an inferred resource, much more exploration work is recommended in order to upgrade the confidence level of the resource to the indicated category. Specifically, a substantial drilling programme is recommended, in order to effectively drill the deposit on 50m centres. Such drilling would add a substantial volume of data, of geological, geotechnical and assay type, to the developing computerised geological model of the deposit. Most of all, it would confirm or deny geological and grade continuity over the approx. 500m x 500m surface area of the deposit.

- In view of the deep, narrow and steep-dipping nature of the deposit, intensive drilling from surface on 50m centres is not recommended. Drilling from surface to give 50m centre coverage would involve some 85,000m of drilling, resulting in a high drilling cost in the region of USD 7.6 million.

- Instead it appears appropriate, following a further limited programme of deposit outline drilling, to embark on a programme of underground development, in order to facilitate a drilling programme from underground. Such underground access would enable 50m centre drilling using only some 39,000m of drilling (as opposed to 85,000m of drilling from surface), and equating to a cost of approx. USD 4.9 million (ie. a 35% cost saving compared to the surface drilling programme). It should also be mentioned that, with an average hole length less than half that of the surface drilling (ie. average length of 211m per hole as opposed to 436m per hole), the drill holes of the underground drilling programme would be generally much more precise in intersecting their target areas.

- To provide the underground access for such a drilling programme, a preliminary costing has been made of an underground development programme. This scenario allows for an access decline of some 2900m, a 500m long horizontal drive in the deposit's hanging wall, 1200m of crosscuts for drilling access, and a 350m ventilation raise. The preliminary costing for this programme has been calculated at USD 5.5 million. Besides enabling access for underground drilling and bulk sampling, the underground access would be laid out such that it could be used for subsequent mining operations.

- Besides the further detailed evaluation of the Jahodna deposit, it is recommended to undertake additional grass roots type exploration within the licence area. This is especially the case in the SE part of the license area, where former systematic exploration did not cover.

- Prior to any mine design, further geotechnical work will be required, based on the geological work discussed above. This will allow the position and methodology of the mine access to be determined, plus it will provide data for the proposed mining method.

- During the further geological work phase, a suite of samples will be acquired for bench scale metallurgical testwork. The samples will have to be positioned to give an optimum range of positions to ensure that the samples are representative of the whole property.

Jahodna Uranium Project
Recommended Work Programs

Summarised Costs for Further Recommended Exploration

Item	**Total USD**
	$
1. First stage of regional exploration work to SE of Jahodna permit	600,000
2. Core drilling at surface and at 50m centres from underground access	4,900,000
3. Underground Development Costs (decline, haulages, crosscuts, raises etc)	5,530,000
4. Underground Development contingency	1,470,000
Total	**$12,500,000**

Jahodna Uranium Project

Estimated Costs for First Stage Regional Surface Exploration Programme

Item	Cost USD
Ground mapping / spectrometer traverses	50,000
Preliminary core drilling	500,000
Sub Total	550,000
Contingency 10%	50,000
Total	**$ 600,000**

Estimated Drilling Costs from Surface and Underground Access

Drilling 7 levels	No. Holes *(Average length 211 metres)*	Total metres	Cost at US $120/metre
Upper Drilling Rows	100	21,000	$ 2,500,000
Lower Drilling Rows	95	20,000	2,400,000
Grand Total Drilling (m)	**195**	**41,000**	**$ 4,900,000**

Underground Development Costs

Item	Cost USD
Decline	3,500,000
Crosscuts	1,000,000
Hanging wall haulage	400,000
Ventilation raise	350,000
Engineering, contract management	130,000
Other equipment and supplies	80,000
Electrical, reticulation	50,000
Site prep	20,000
	5,530,000
Contingency	1,470,000
Total	**$7,000,000**

This estimate is an order of magnitude study to estimate the costs of basic development for an underground drilling programme.

21. REFERENCES

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4. **Košicka – Belá – Myslava – According Planimetric Map,** March 2005, 1:10,000 scale, prepared by Koral s.r.o.

5. **Landscape Atlas of the Slovak Republic,** First Edition, 2002, ISBN 80-88833-27-2, publ. by the Ministry of the Environment of the Slovak Republic, Bratislava, and Slovak Environmental Agency, Banska Bystrica. Produced by Espit, spol s.r.o, Banska Stiavnica.

6. **Uranium Mineralisation in Slovakia, 1997,** Rojkovic I., Acta Geologica Universitatis Comenianae, Monographic Series, Univ. of Komenského, Bratislava.

7. **Mineral Resources of Slovakia,** 1997, publ. by Ministry of the Environment and the Geological Survey of the Slovak Republic.

8. **Geological Map of Slovakia**, 1996, 1:500,000 scale geological map, publ. by the Ministry of the Environment and the Geological Survey of the Slovak Republic.

9. **Explanation to the Geological Map of Slovakia,** 1:500,000 scale, 1996, ed. Anton Biely, publ. by the Ministry of the Environment and the Geological Survey of the Slovak Republic.

10. **Technological Research of U-Mo ore from Jahodna Site, 1993**, Kopecky J., unpubl. report by MEGA, joint stock company Strazpod Ralskem, Czech Republic.

11. **Geological Map with Exploration Works, U-Mo Kosice -Jahodna Deposit**, 1991, prepared by Mihal, F.

12. **Geological Map of the Slovenské Rudhorie Mountains, Eastern Part,** 1984, produced by Dionýz Stur Institute of Geology, Bratislava.

13. **Technical report of Jahodna Uranium Project, Slovakia**, 28th March, 2006, prepared by David Pelham and Galen White, ACA Howe International Ltd.

14. **Prelimary Assesment of Jahodna Uranium Project,** Slovakia, 25th April 2006, prepared by Julian Bennett and Dr. R. Balakrishnan, ACA Howe Int. Ltd.

CERTIFICATE OF QUALIFICATIONS

David Pelham

Geologist

28 York Road
Colwyn Bay
Conwy LL29 7EN
United Kingdom

I, David Pelham, do hereby certify that:

1. I am an Associate Consulting Geologist with A.C.A. Howe International Limited, whose office address is 254 High Street, Berkhamstead, Herts HP4 1AQ, United Kingdom.

2. I graduated with a BSc Honours degree in Geology/Geography in 1974 from Derby College of Technology (London University), and an MSc degree in Mineral Exploration in 1982 from Rhodes University (South Africa) and have practiced my profession continuously since 1976.

3. I hold membership in the following mineral industry technical societies:
   Professional Member Institution of Materials, Minerals and Mining
   Gemmological Association
   Small Mining International
   Welsh Mines Society

4. I have practiced my profession as a geologist continually for over 29 years.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the overall preparation of the technical report titled, "Technical Report of the Preliminary Assessment of the Jahodna Project, Slovakia", dated May 11, 2006.

7. I visited Slovakia for approximately two weeks during October 2005 to review project data, and this included a visit to the Jahodna property.

8. I have not had prior involvement with the Jahodna property that is the subject of the Technical Report. I have had prior involvement with other uranium properties in Botswana, Namibia, South Africa and Niger. The nature of my prior involvement was in the exploration for calcrete type, granitic type, placer type and sandstone-hosted uranium deposits.

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

10. I am independent of Tournigan Gold Corporation, applying all of the tests in section 1.4 of National Instrument 43-101.

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

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

Signed and dated this 11th day of May, 2006

"David Pelham"

David Pelham, Geologist

JULIAN P A BENNETT
43 Spring Way
Sible Hedingham
Essex, United Kingdom CO9 3SB
Telephone: +44 1787 469855
Fax: +44 1787 460016
Email: julian.bennett@btinternet.com

CERTIFICATE OF AUTHOR

I, Julian P A Bennett BSc (Eng), C.Eng, do hereby certify that:

1. I have been employed since 2005 as an associate mining engineer with the firm of A.C.A. Howe International Limited, Mining and Geological Consultants located at 243 High Street, Berkhamsted, United Kingdom.

2. I graduated with a Bachelor of Science (Engineering) in Mining, from the Royal School of Mines, London University, United Kingdom, in 1964, and have practiced the profession of mining engineer since graduation.

3. I am a Chartered Engineer (C Eng) registered with Engineering Council, United Kingdom (registration number 124698). I am a member of the Institute of Materials, Minerals and Mining.

4. I have 41 years of experience in mining projects, both open pit and underground, from exploration, feasibility study, design and construction of new mines, to operations management, in Africa, Australia, Canada, and Central America. Included in this total is 10 years of consultancy throughout the world including the preparation of competent person's reports for listings on the London Stock Exchange and the Johannesburg Stock Exchange, technical due diligence reports and independent engineer's reports for investors and other interested parties. I have a total of 2 years of direct experience with uranium projects located in India and Australia based on technical responsibility for the mining aspects of the evaluation of uranium projects for the owner.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the mining content of the technical report titled: Jahodna Preliminary Assessment for Tournigan Gold Corporation (the "Technical Report") relating to the Jahodna Uranium project in Slovakia. The information

and data used in this report were obtained from the references cited and data collected by Tournigan.

7. I have not had prior involvement with the issuer nor with the property that is the subject of the Technical Report.

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

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

10. I am independent of the issuer applying all the test is section 1.5 of National Instrument 43-101.

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

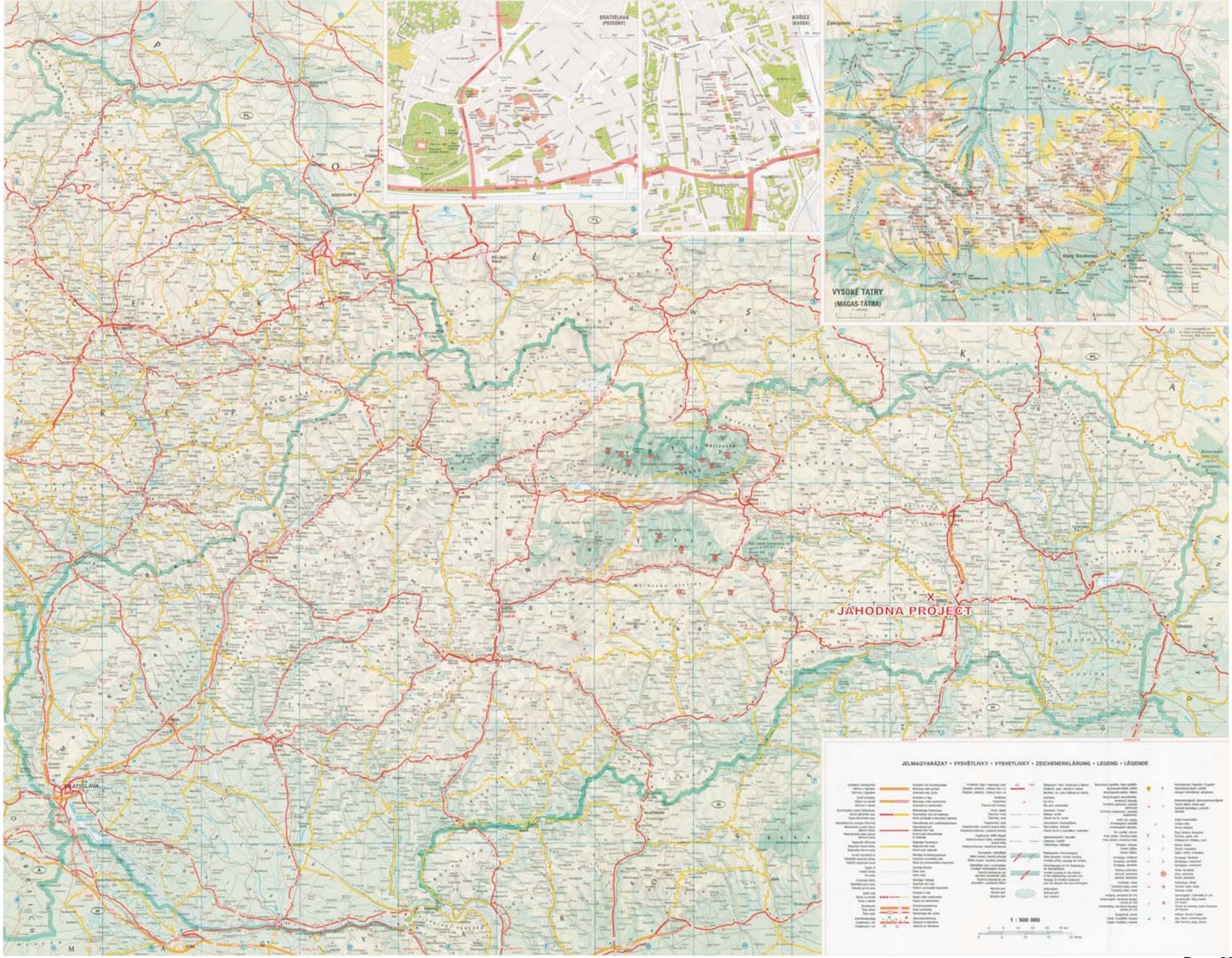
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public.

Signed and dated this 11th Day of May 2006

"Julian P.A. Bennett"

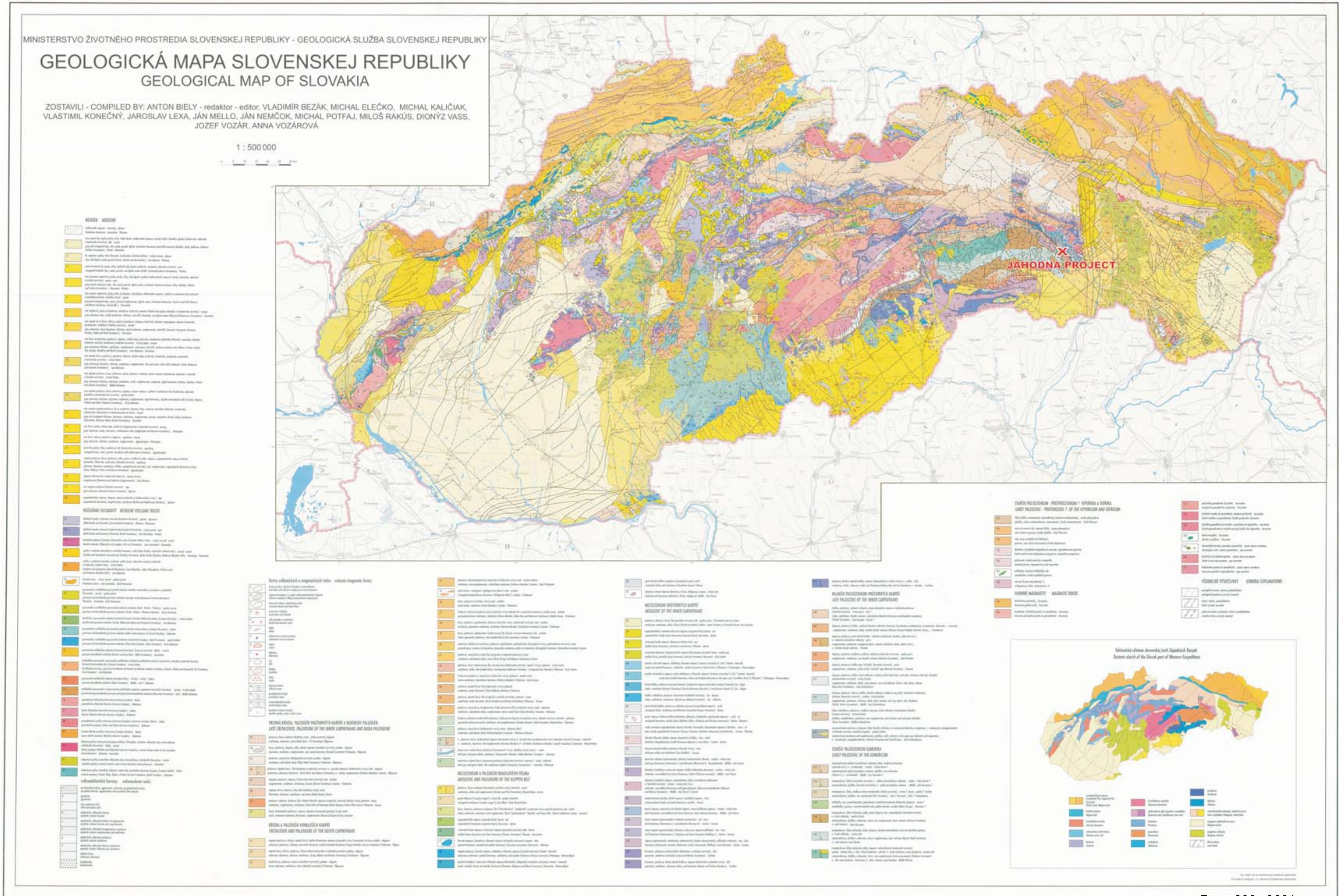
Julian P A Bennett BSc (Eng), C. Eng

APPENDIX 1.



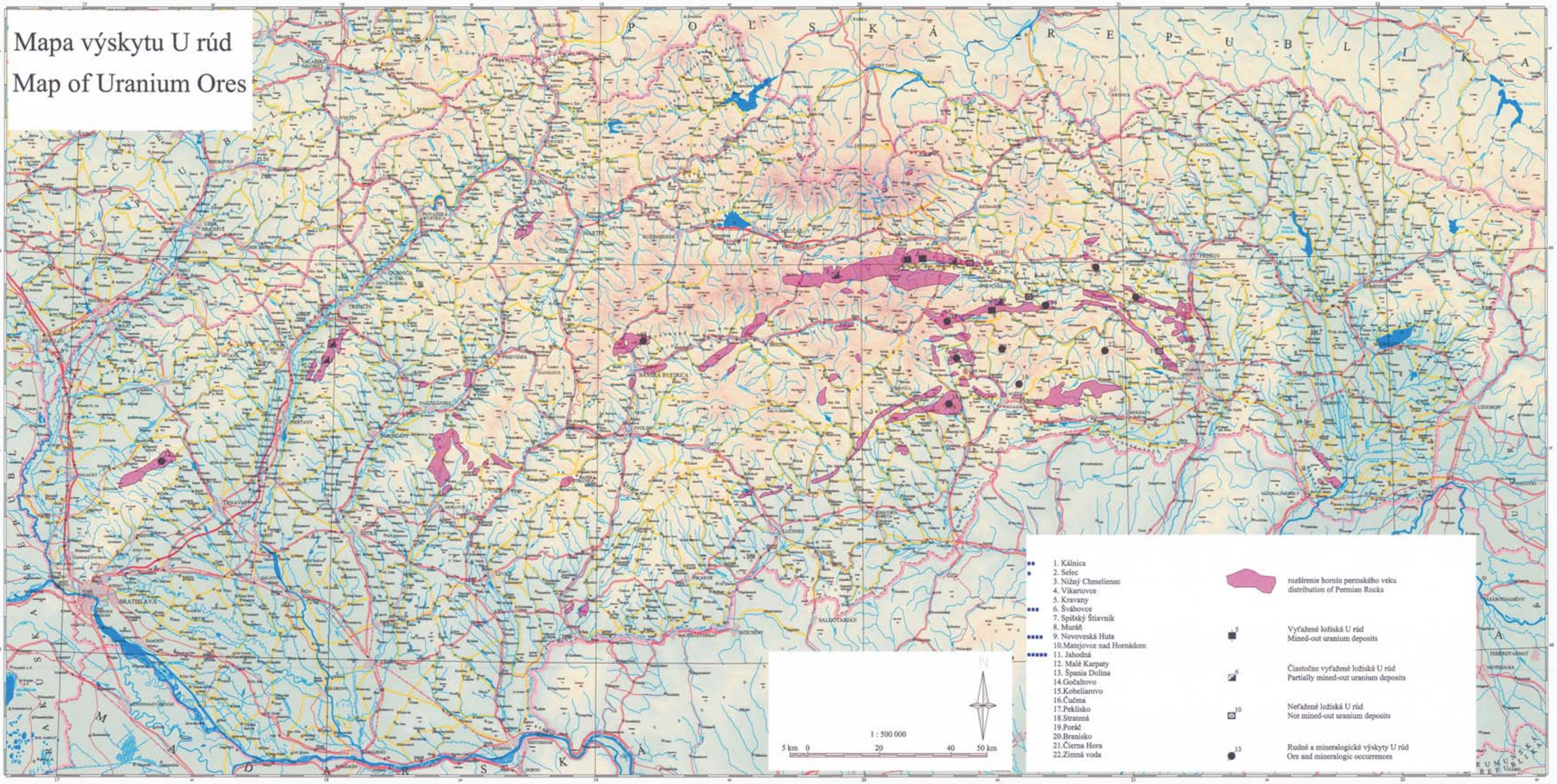
MAP 1- SLOVAKIA ROAD MAP- scale 1:500,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia.

A C A Howe International Limited



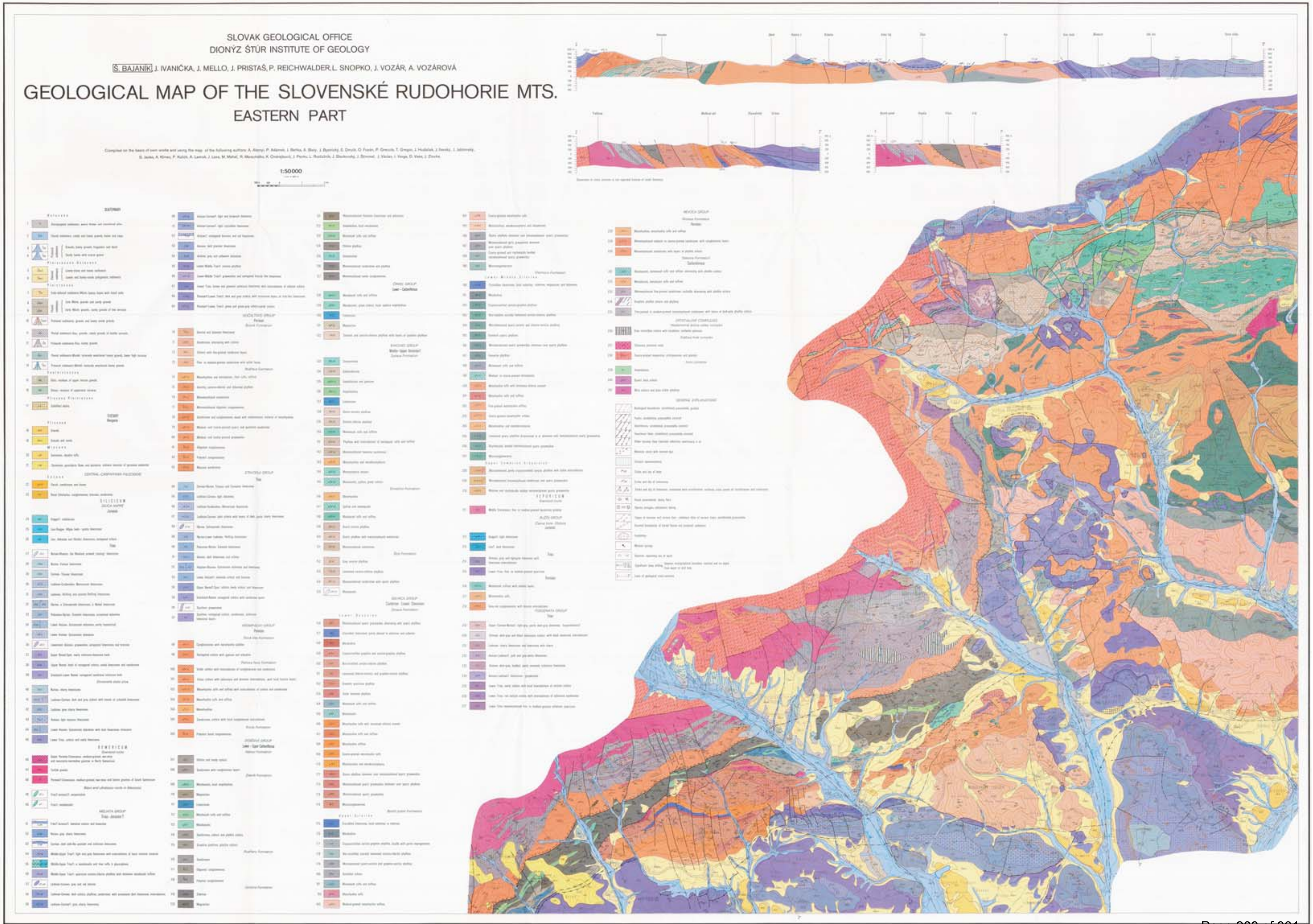
MAP 2- SLOVAKIA GEOLOGY MAP- scale 1: 500,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia.

A C A Howe International Limited



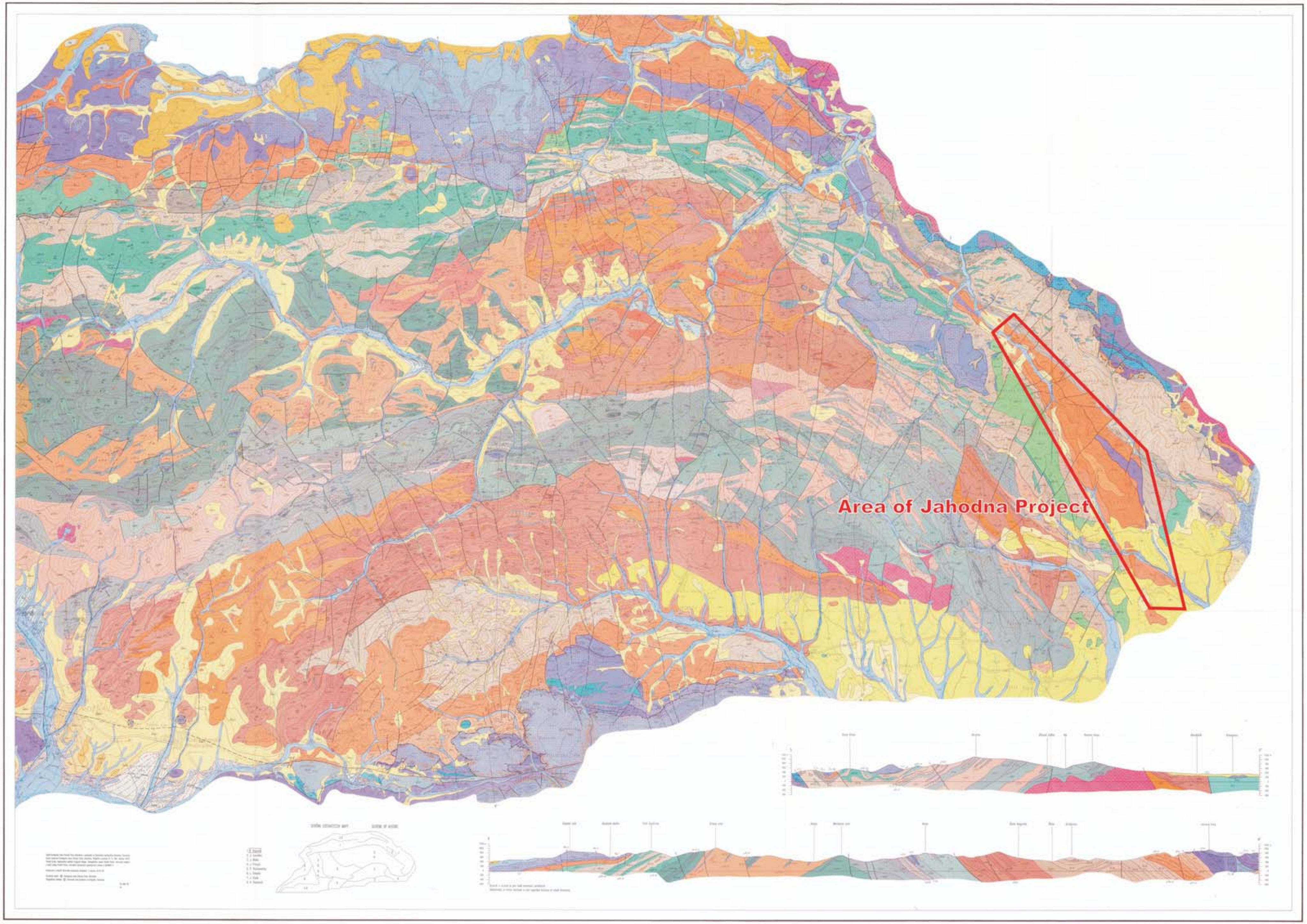
MAP 3- SLOVAKIAN URANIUM DEPOSITS- scale 1:500,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited



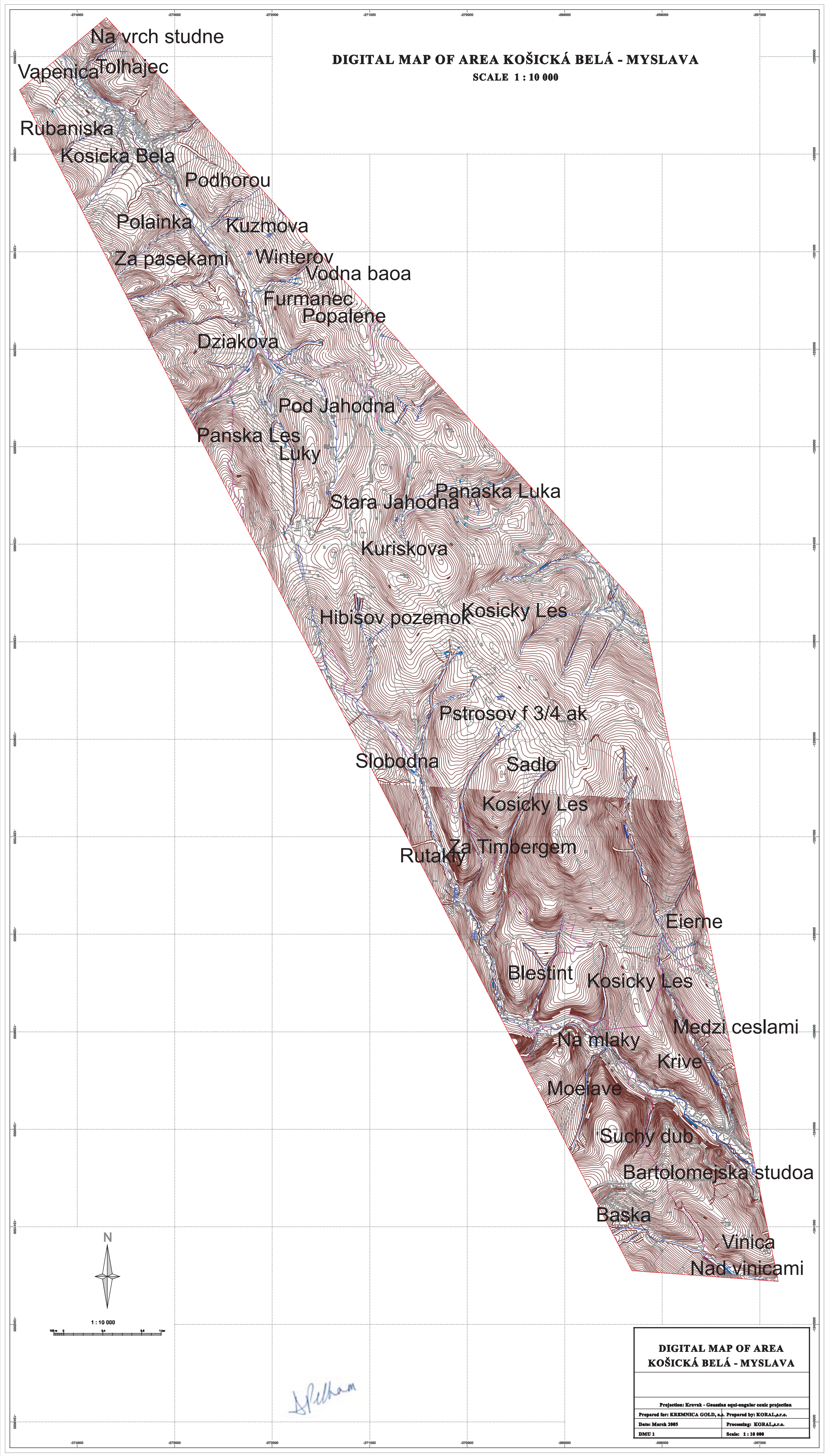
MAP 4- SLOVENSKE RUDOHORIE, WEST SHEET- scale 1:50,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited



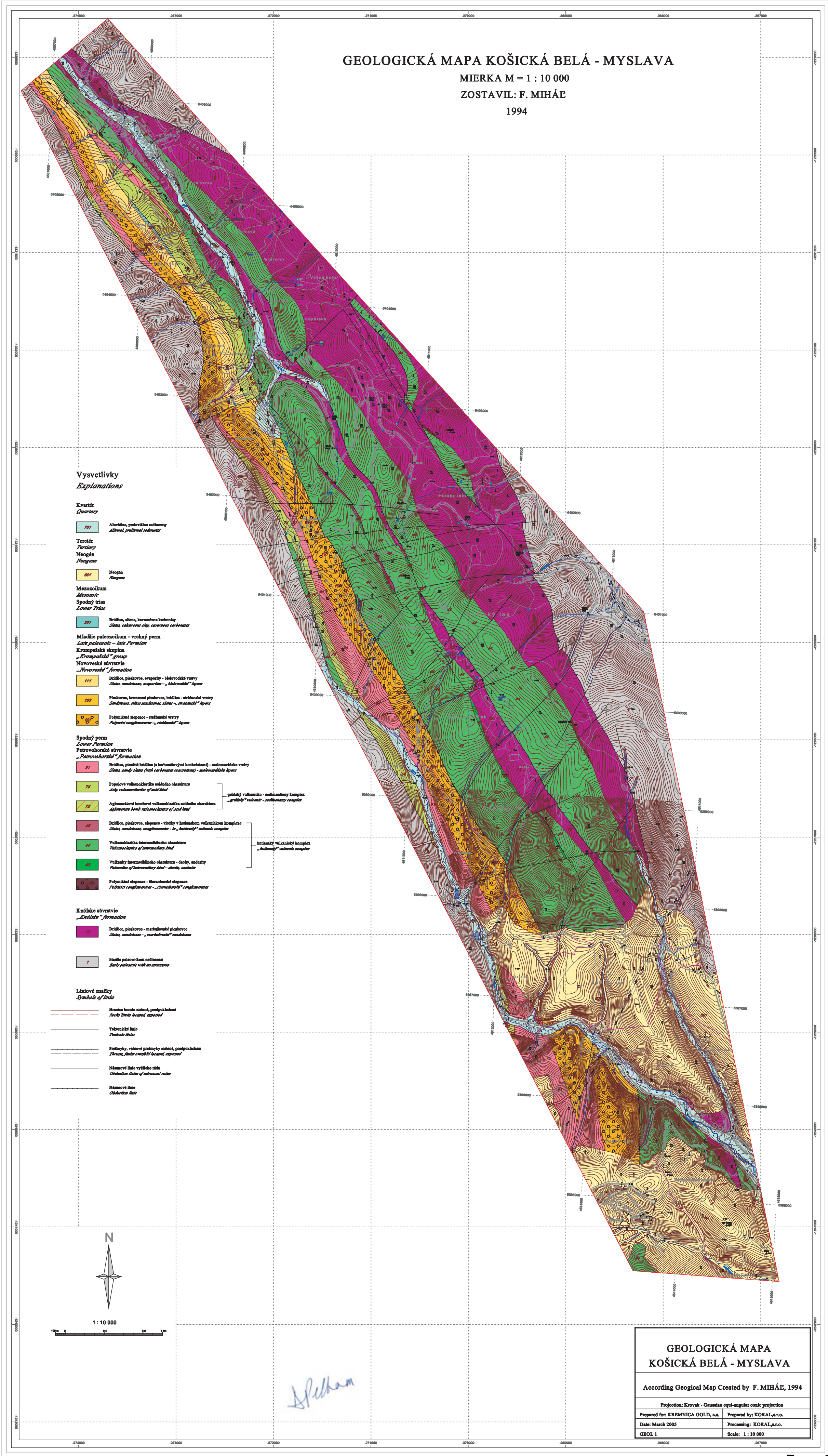
MAP 5- SLOVENSKE RUDOHORIE, EAST SHEET- scale 1:50,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited

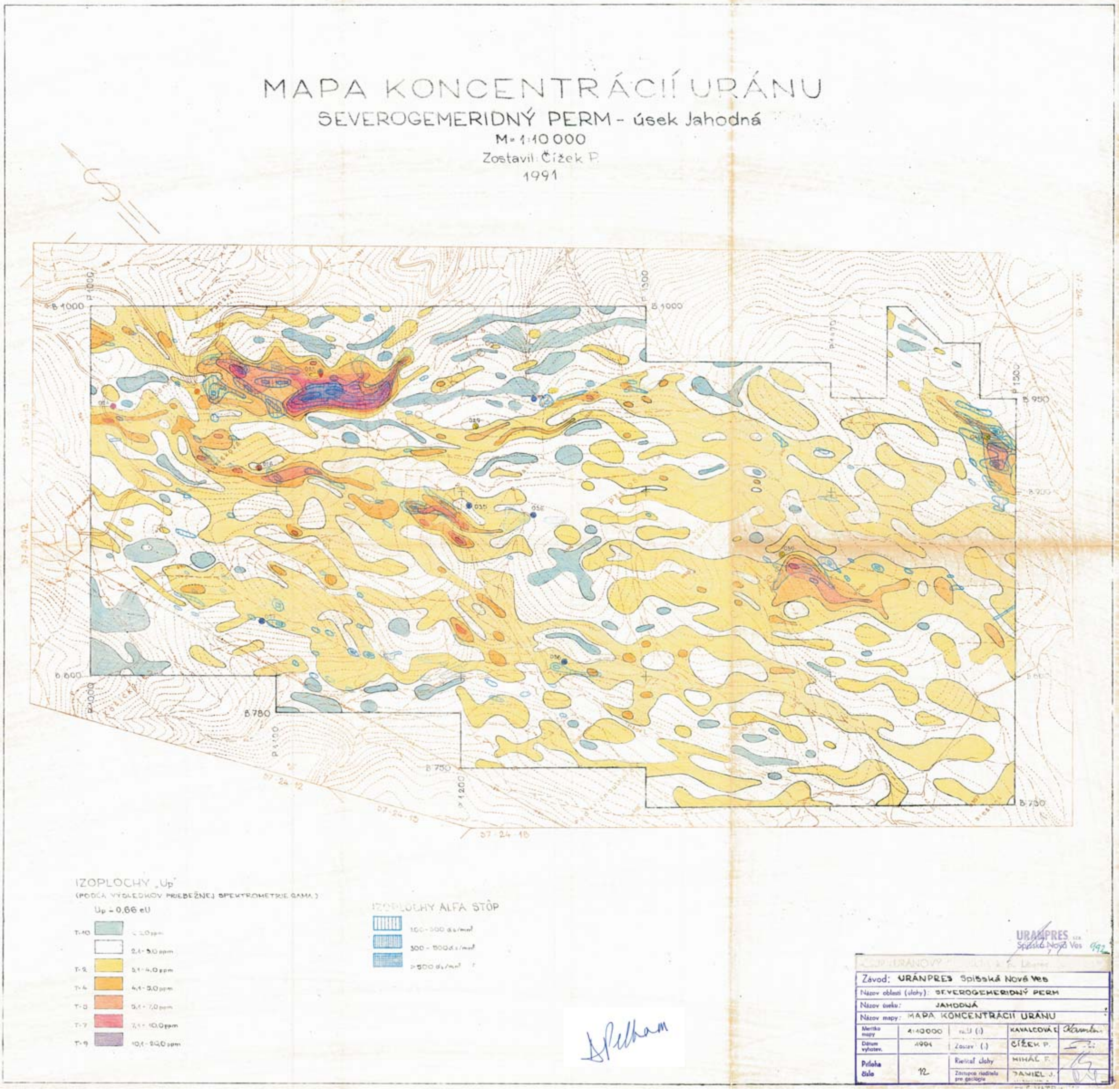


MAP 6- JAHODNA TOPOGRAPHIC MAP- scale 1:10,000
For Tournigan Gold Corporation, Jahodna
Uranium Project, Slovakia

A C A Howe International Limited

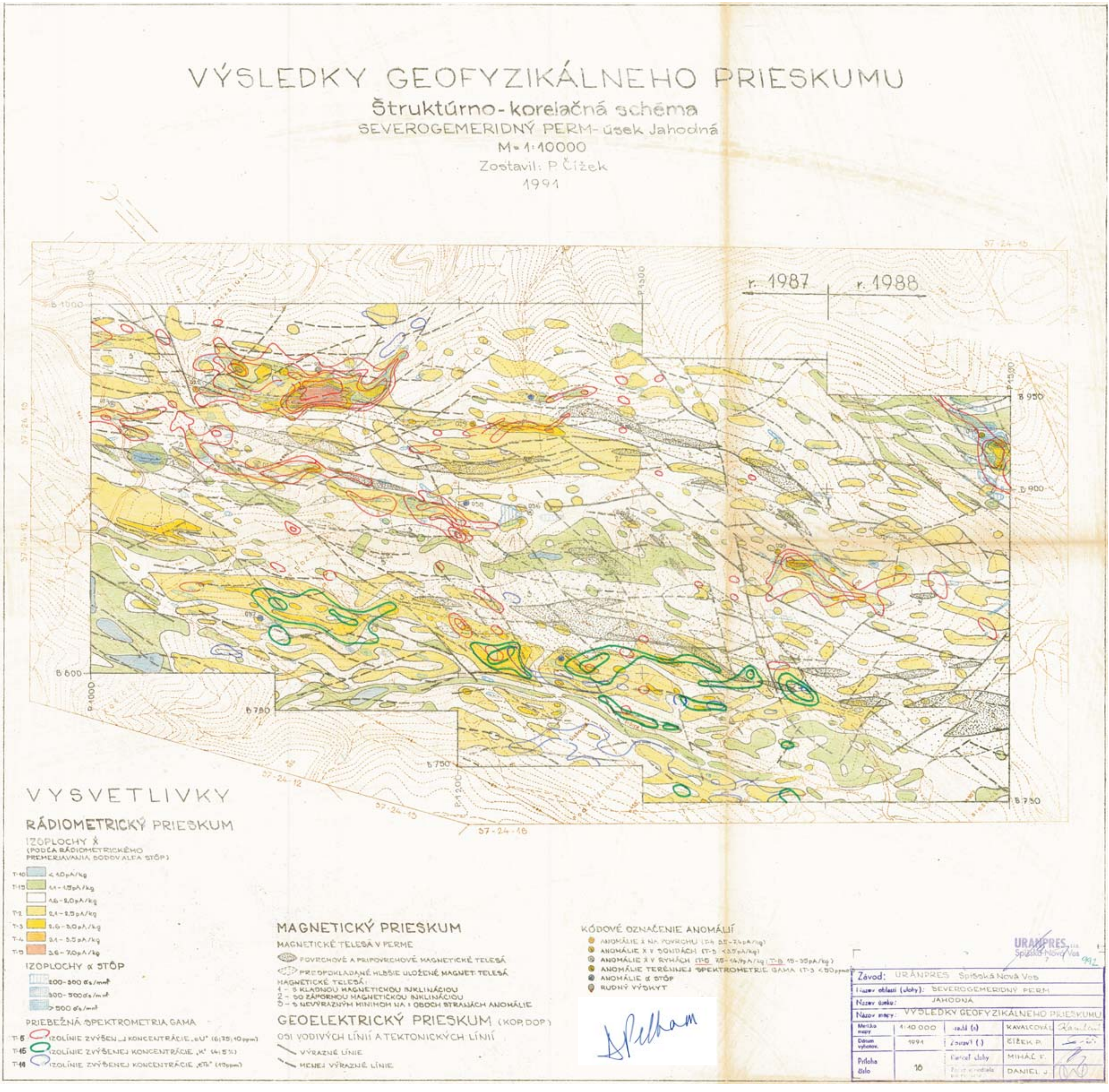


MAP 7- JAHODNA GEOLOGICAL MAP - scale 1:10,000
For Tournigan Gold Corporation, Jahodna
Uranium Project, Slovakia



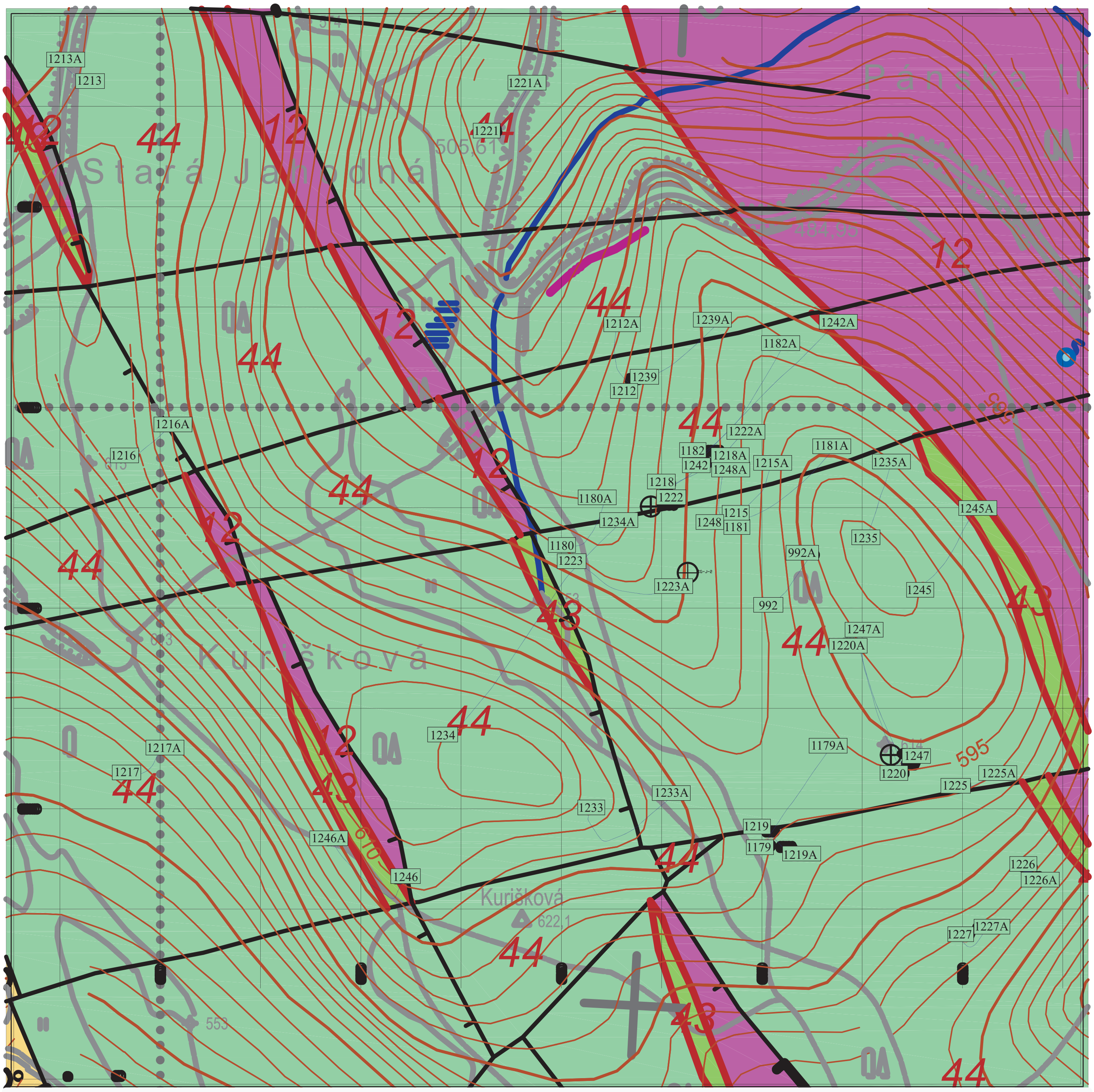
MAP 8- JAHODNA- GAMMA & ALPHA READINGS
- scale 1:10,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited

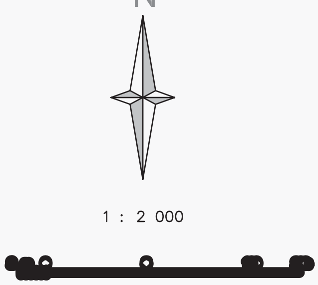


MAP 9- JAHODNA- GEOPHYSICAL COMPILATION PLAN- scale 1:10,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited



12	Bridlice, pieskovce - markušovské pieskovce *Slates, sandstones - „markušovské" sandstones*
44	Vulkanoklastika intermediálneho charakteru *Vulcanoclastics of intermediary kind*
43	Vulkanity intermediálneho charakteru - dacity, andezity *Vulcanites of intermediary kind – dacite, andesite*



GEOLOGICKÁ MAPA KOŠICKÁ BELÁ - MYSLAVA

According Geogical Map Created by F. MIHÁĽ, 1994

Projection: Krovak - Gaussian equi-angular conic projection

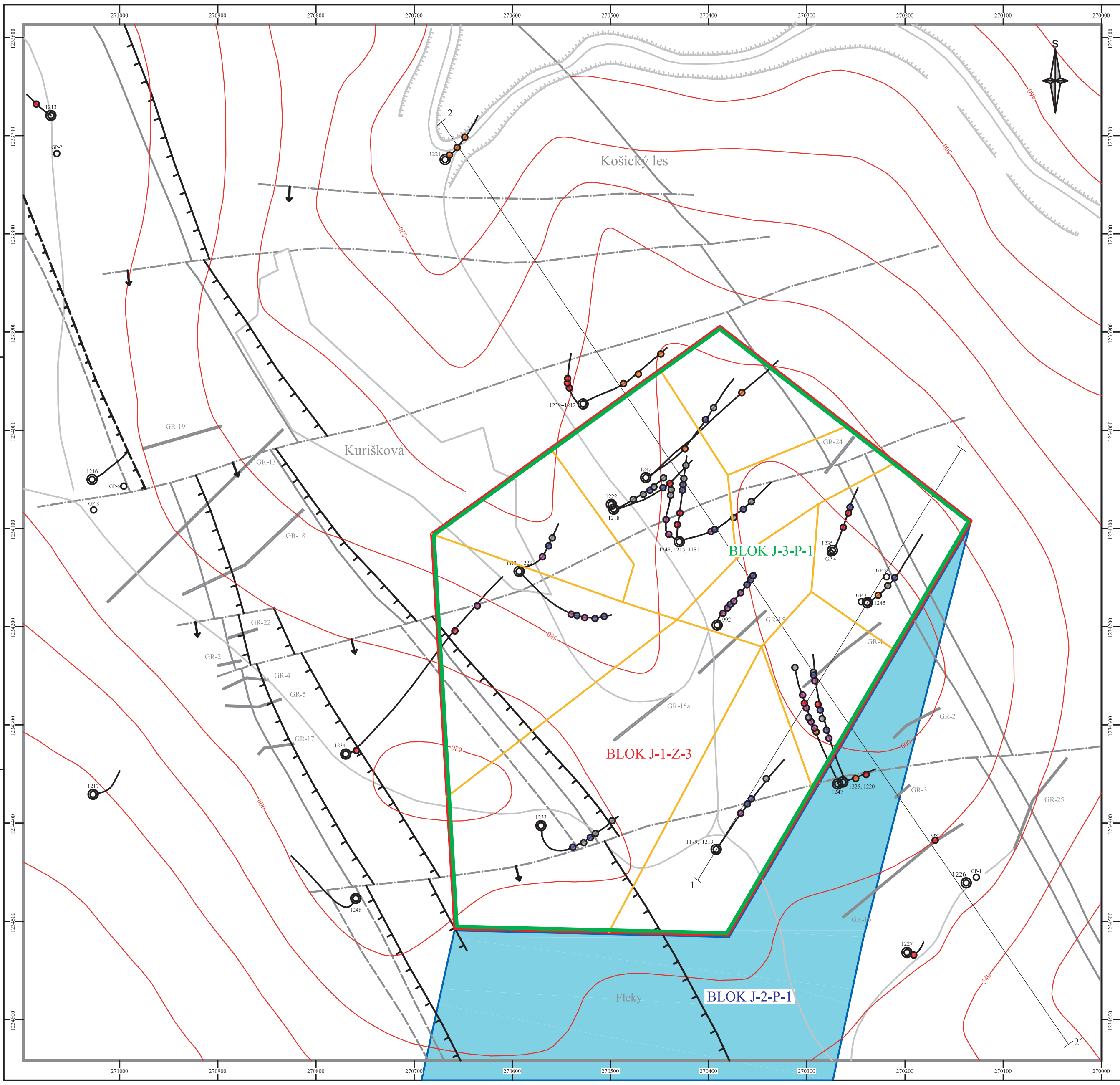
Prepared for: KREMNICA GOLD, a.s.	Prepared by: KORAL,s.r.o.
Date: March 2005	Processing: KORAL,s.r.o.
GEOL 2	Scale: 1 : 2 000

MAP 10- JAHODNA - GEOLOGICAL MAP- scale 1:2,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

MAPA BLOKU ZÁSOB, VARIANT I. A BLOKOV PROGNÓZNYCH ZDROJOV, LOŽISKO KOŠICE I.

M = 1 : 2 000

Zostavil: F. Miháľ, 1991 a J. Daniel, 2005



Vysvetlivky:

- pôsobnosť vrtov
- hranice bloku J - 1 - Z - 3
- hranice bloku J - 2 - P - 1
- hranice bloku J - 3 - P - 1

Poznámka: iné vysvetlivky sú na prílohe č. 1

Blok J-1-Z-3 Variant I.	S = 339 500 m^2 m = 2.38 c = 0.304% V = 808 010 m^3 Q = 2 197 787t K = 6 681t U

Príloha č. 2

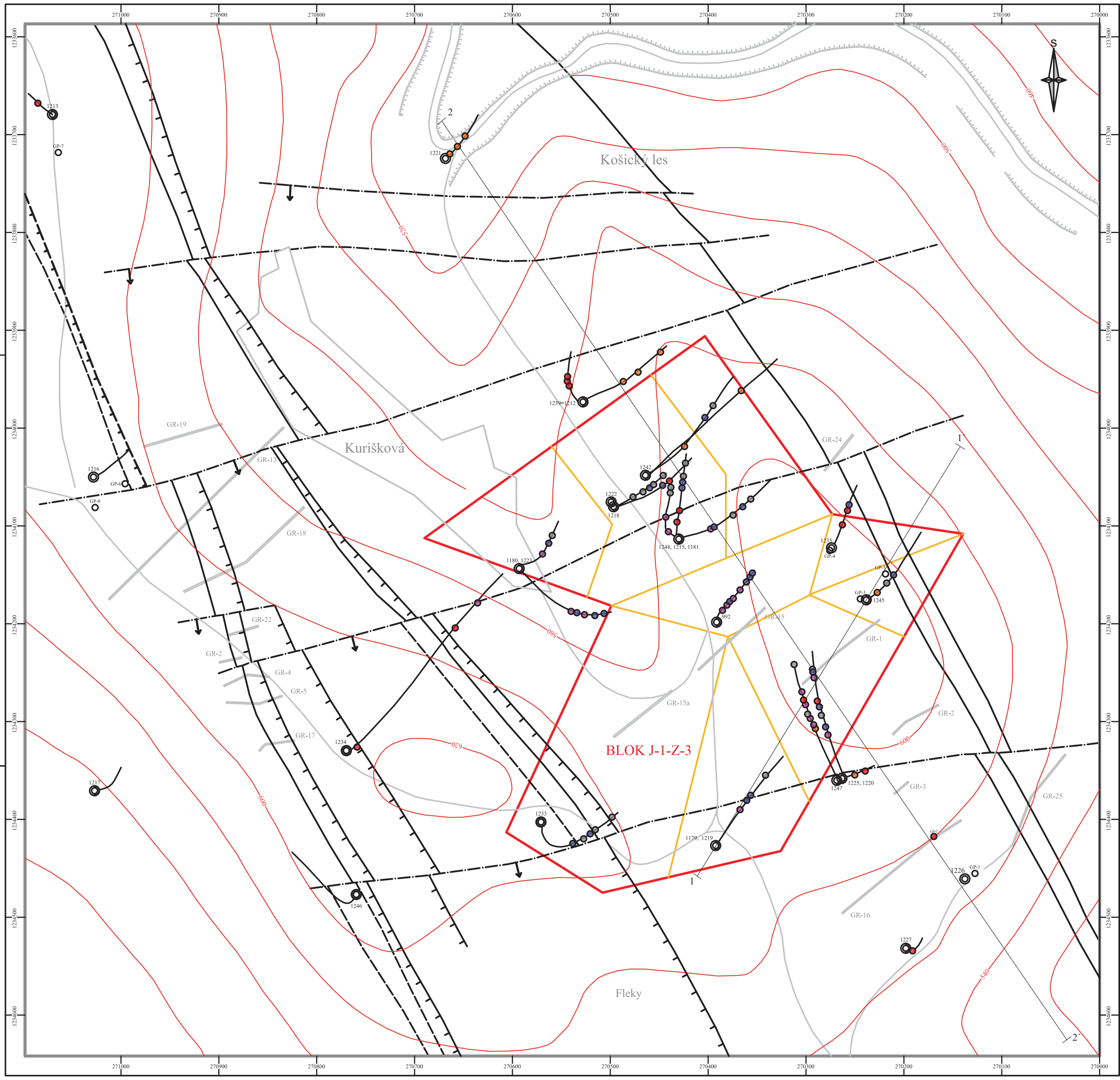
MAP 11- JAHODNA RESERVES- RESERVE BLOCKS
- VARIANT I - scale 1:2,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited

MAPA BLOKU ZÁSOB, VARIANT II., LOŽISKO KOŠICE I.

M = 1 : 2 000

Zostavil: F. Miháľ, 1991 a J. Daniel, 2005



Vysvetlivky:

- hranice bloku J-1-Z-3
- pôsobnosť vrtov

Poznámka: iné vysvetlivky sú na prílohe č. 1

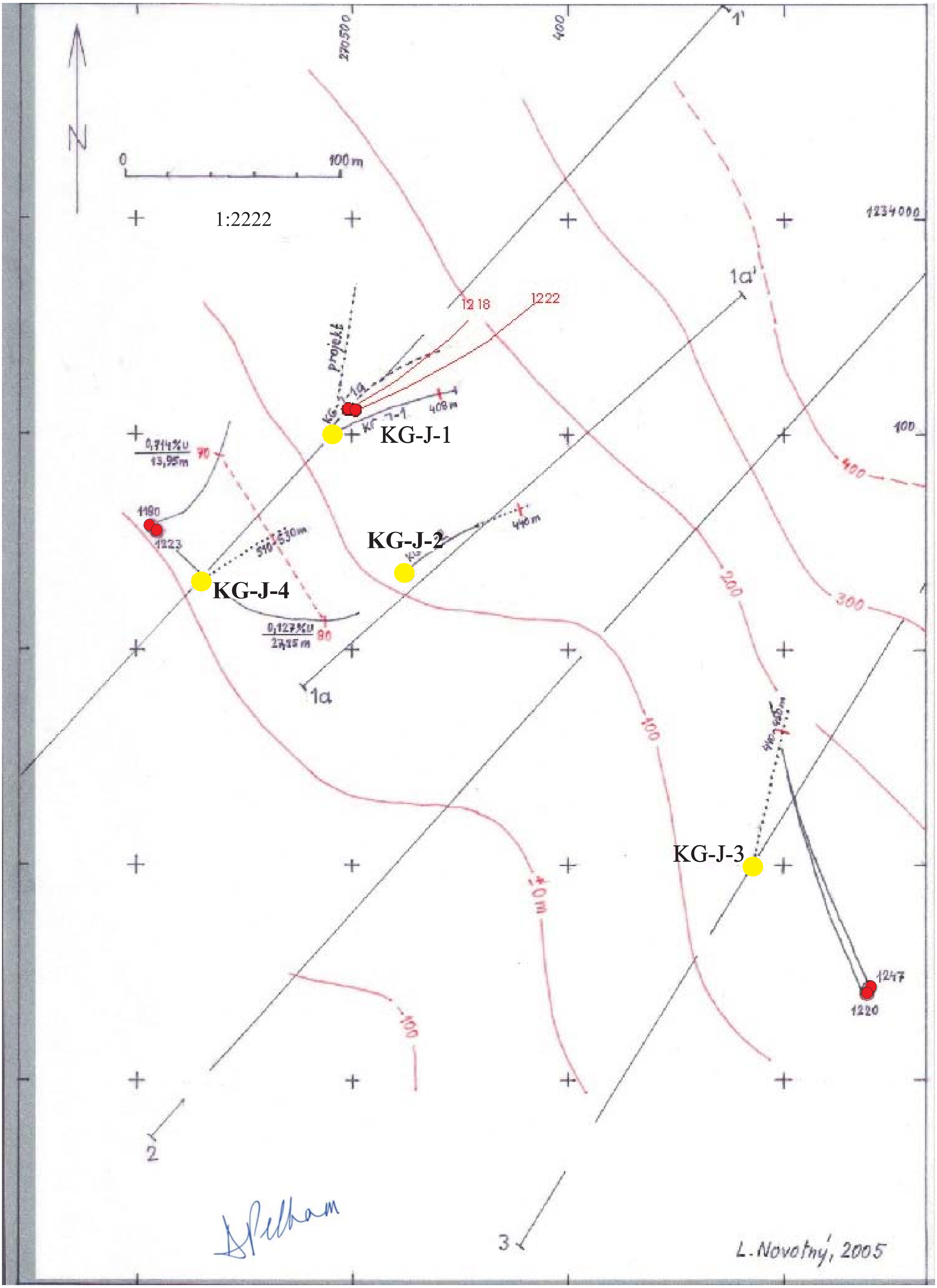
Blok J-1-Z-3	
Variant II.	S = 234 350 m^2 m = 2.19 c = 0.472% V = 513 226 m^3 Q = 1 395 975t K = 6 589t U

Príloha č. 3

MAP 12- JAHODNA RESERVES- RESERVE BLOCKS - VARIANT II - scale 1:2,000

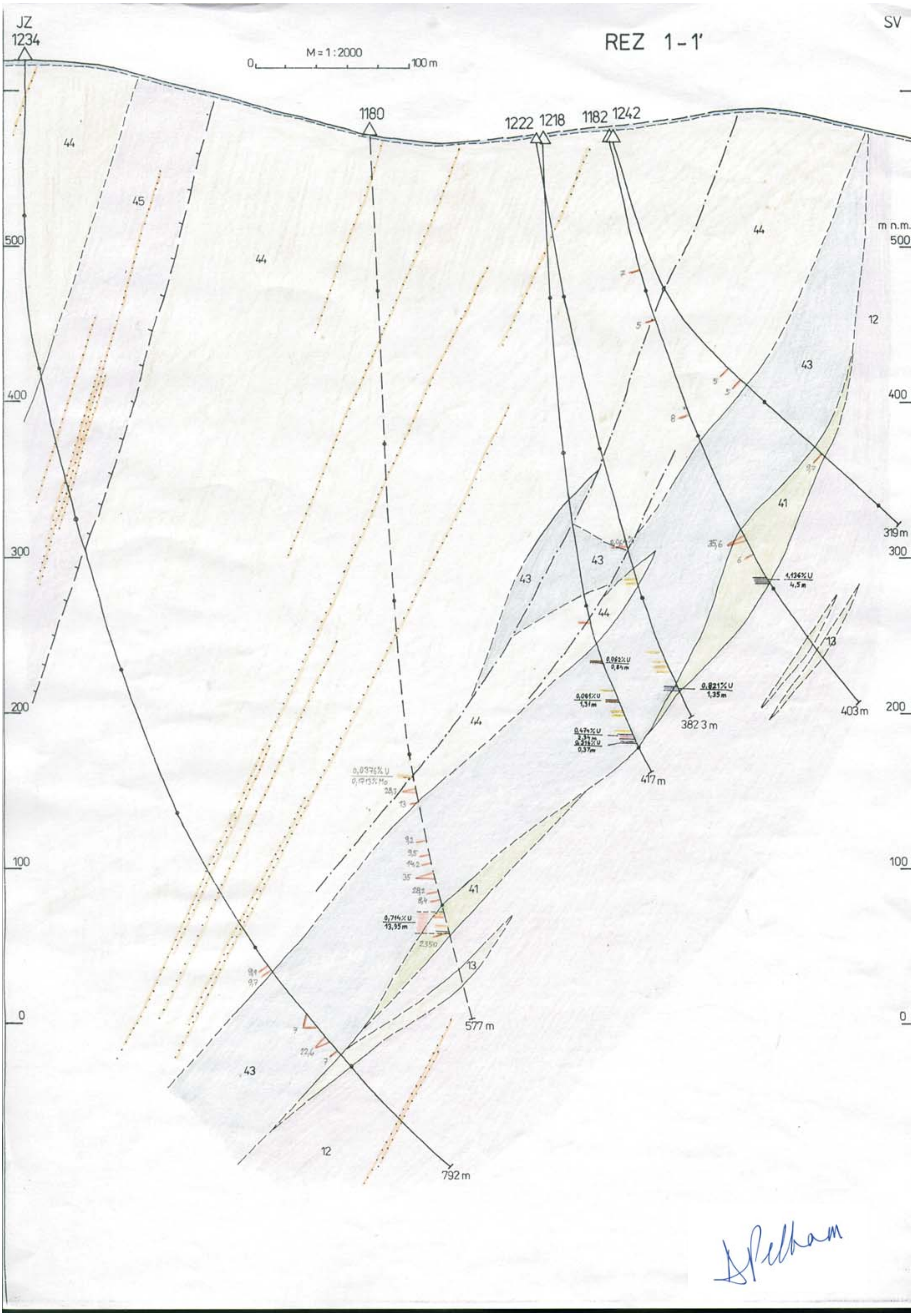
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited



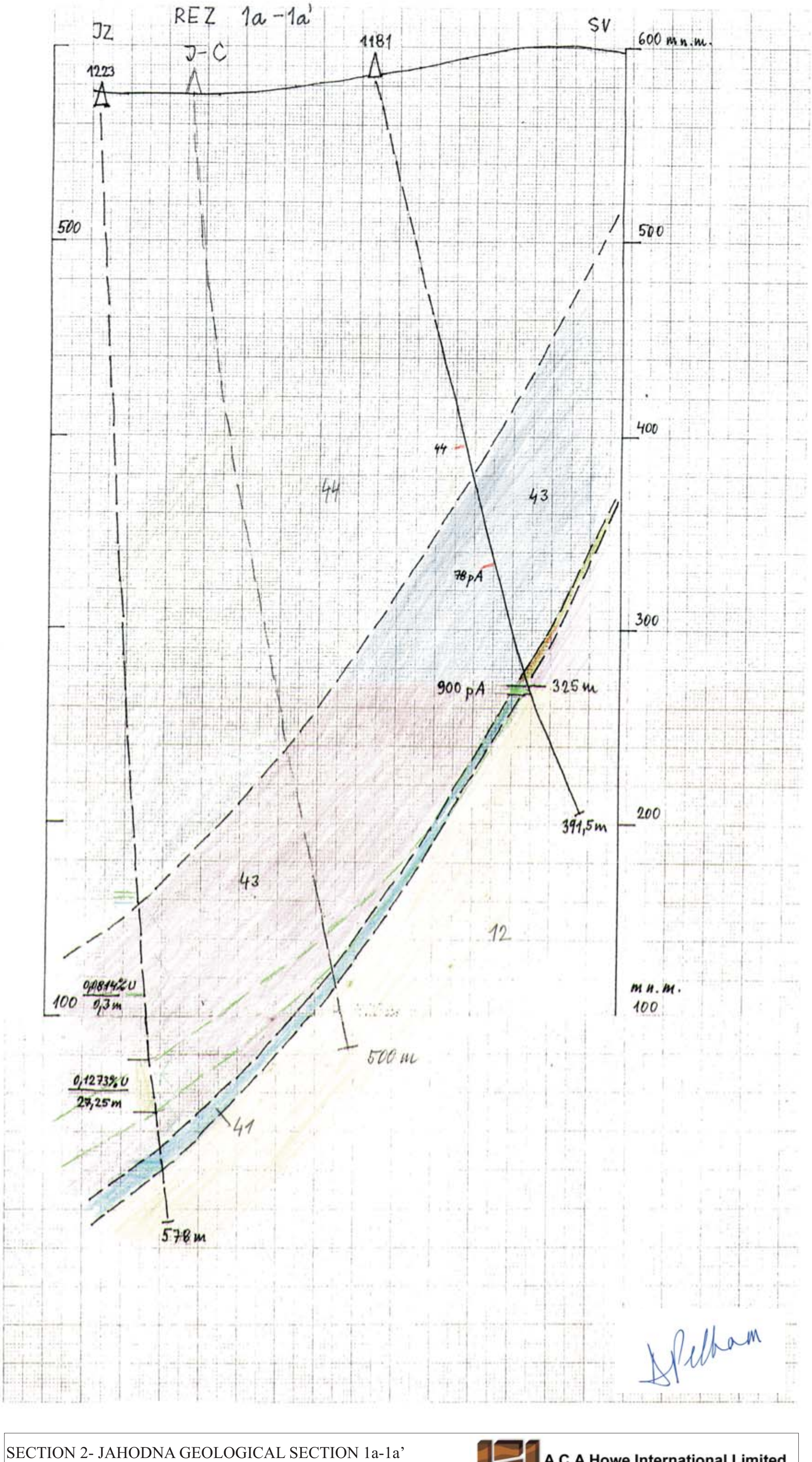
MAP 13- JAHODNA- TOURNIGAN 2005/06 PHASE 1 DRILLING
For Tournigan Gold Corp, Jahodna Uranium Project, Slovakia

A C A Howe International Limited



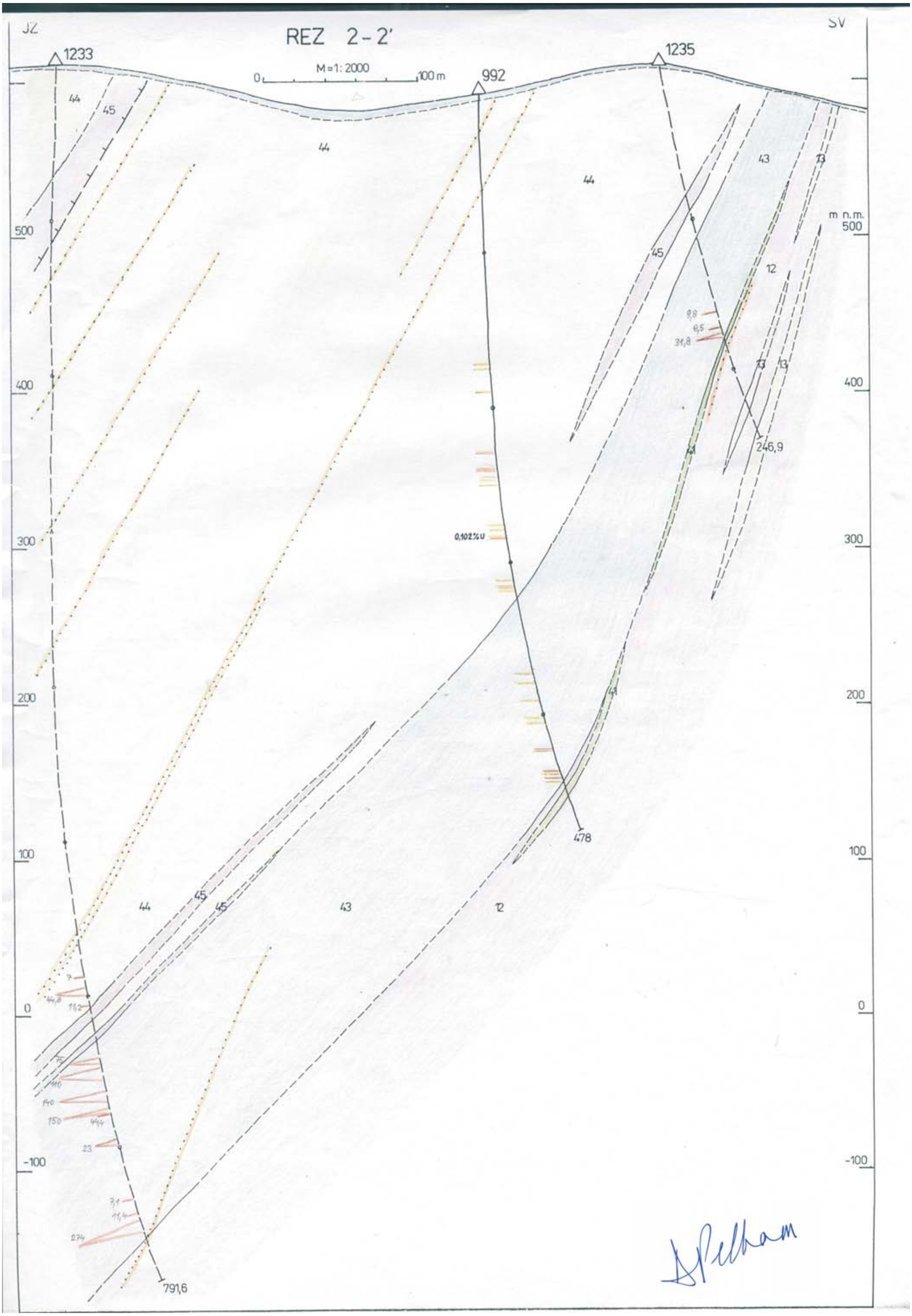
SECTION 1- JAHODNA- GEOLOGICAL SECTION 1-1'
- scale 1:2,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited

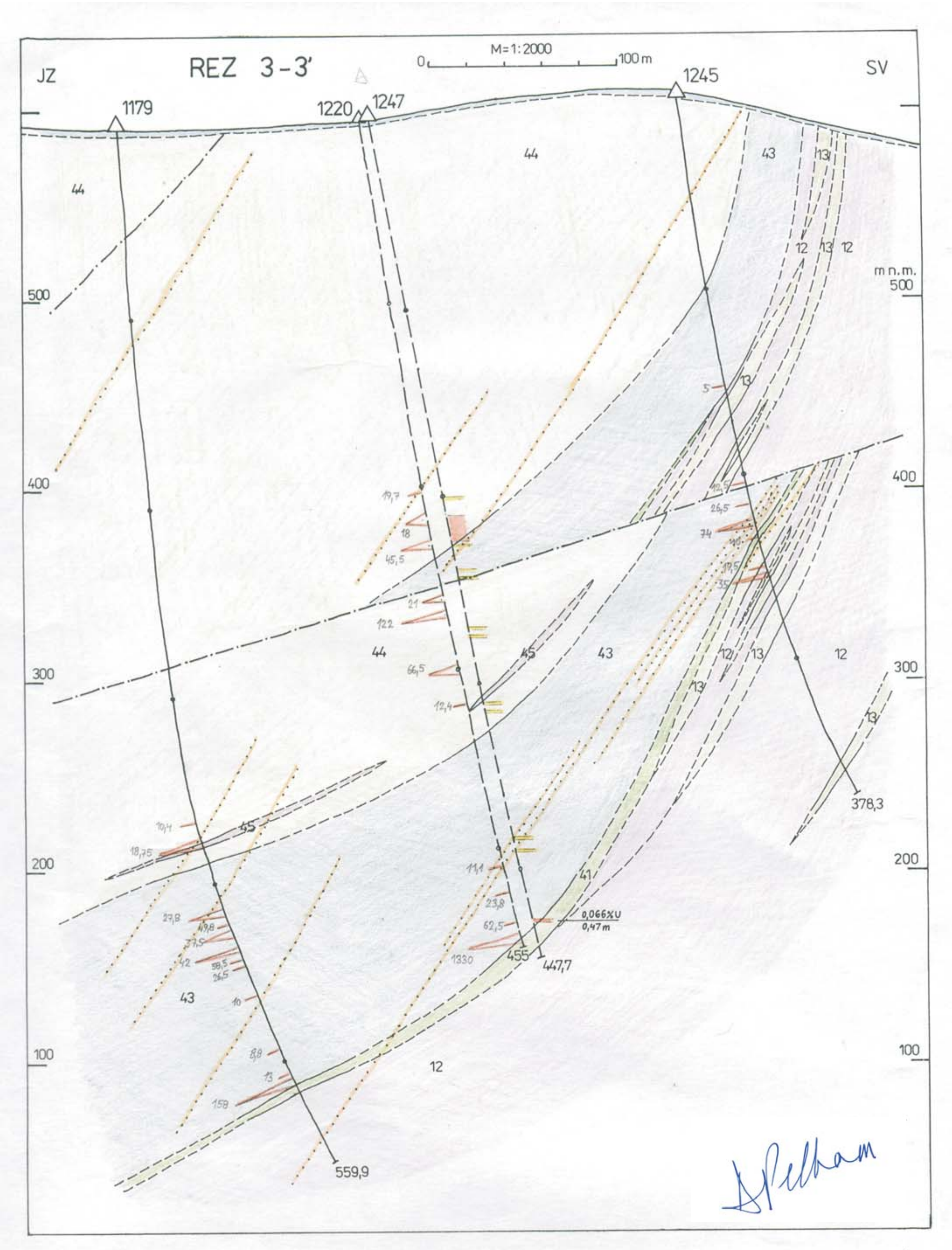


SECTION 2- JAHODNA GEOLOGICAL SECTION 1a-1a'
- scale 1:2,000
For Tourigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited

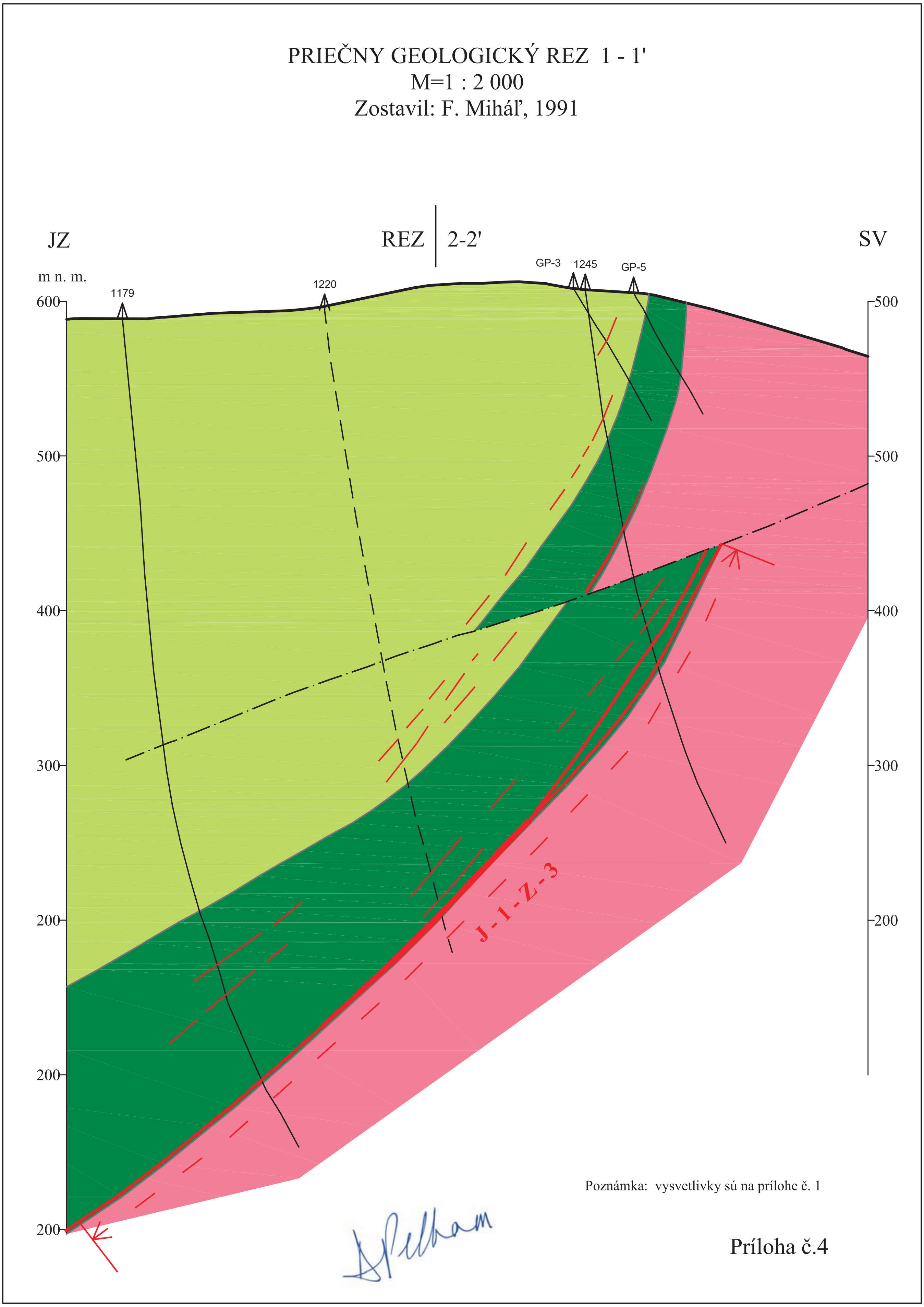


SECTION 3- JAHODNA- GEOLOGICAL SECTION 2-2'
- scale 1:2,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia



SECTION 4- JAHODNA- GEOLOGICAL SECTION 3-3'
- scale 1:2,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

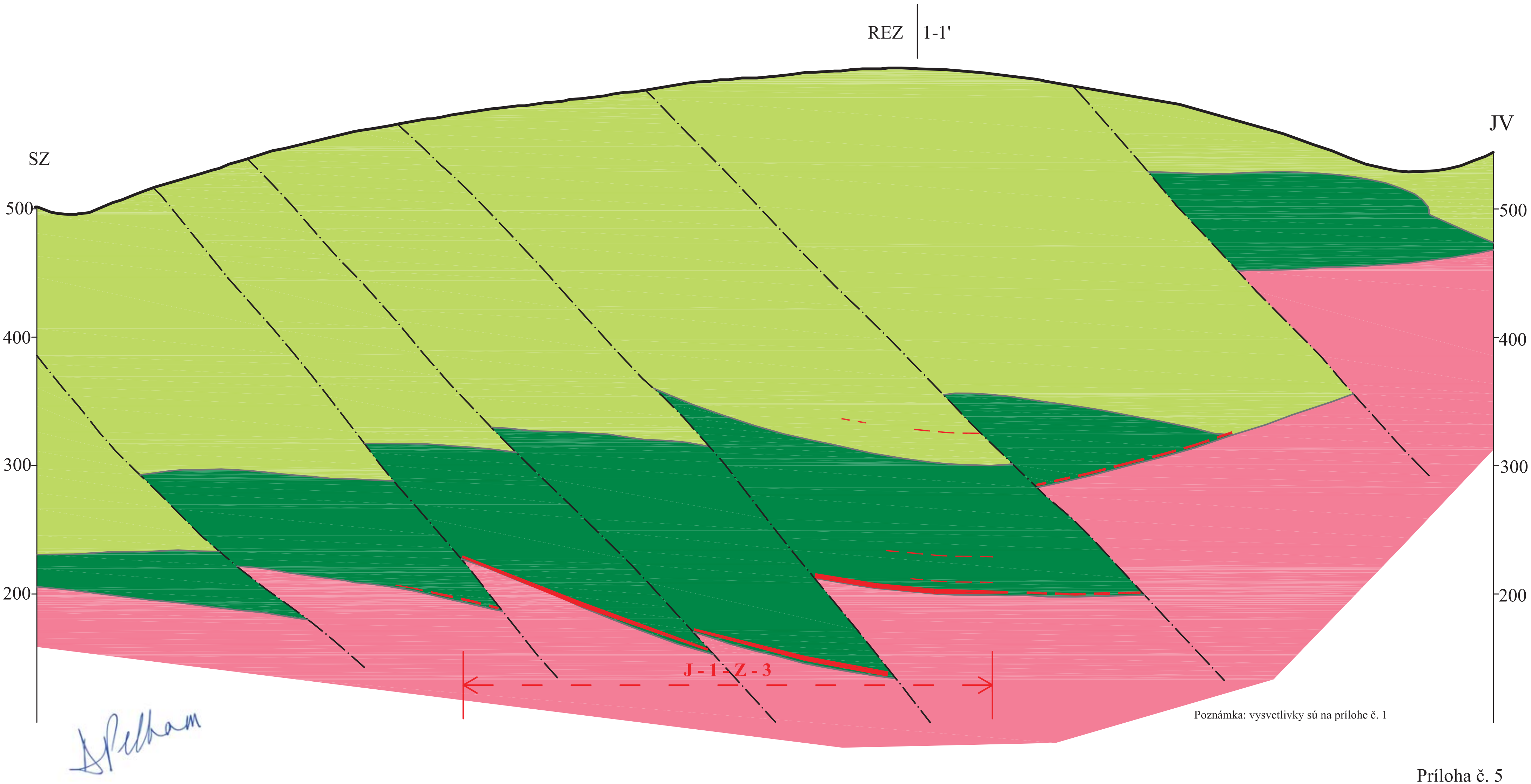
A C A Howe International Limited



SECTION 5- JAHODNA DEPOSIT- CROSS SECTION 1-1'
- scale 1:2,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia.

A C A Howe International Limited

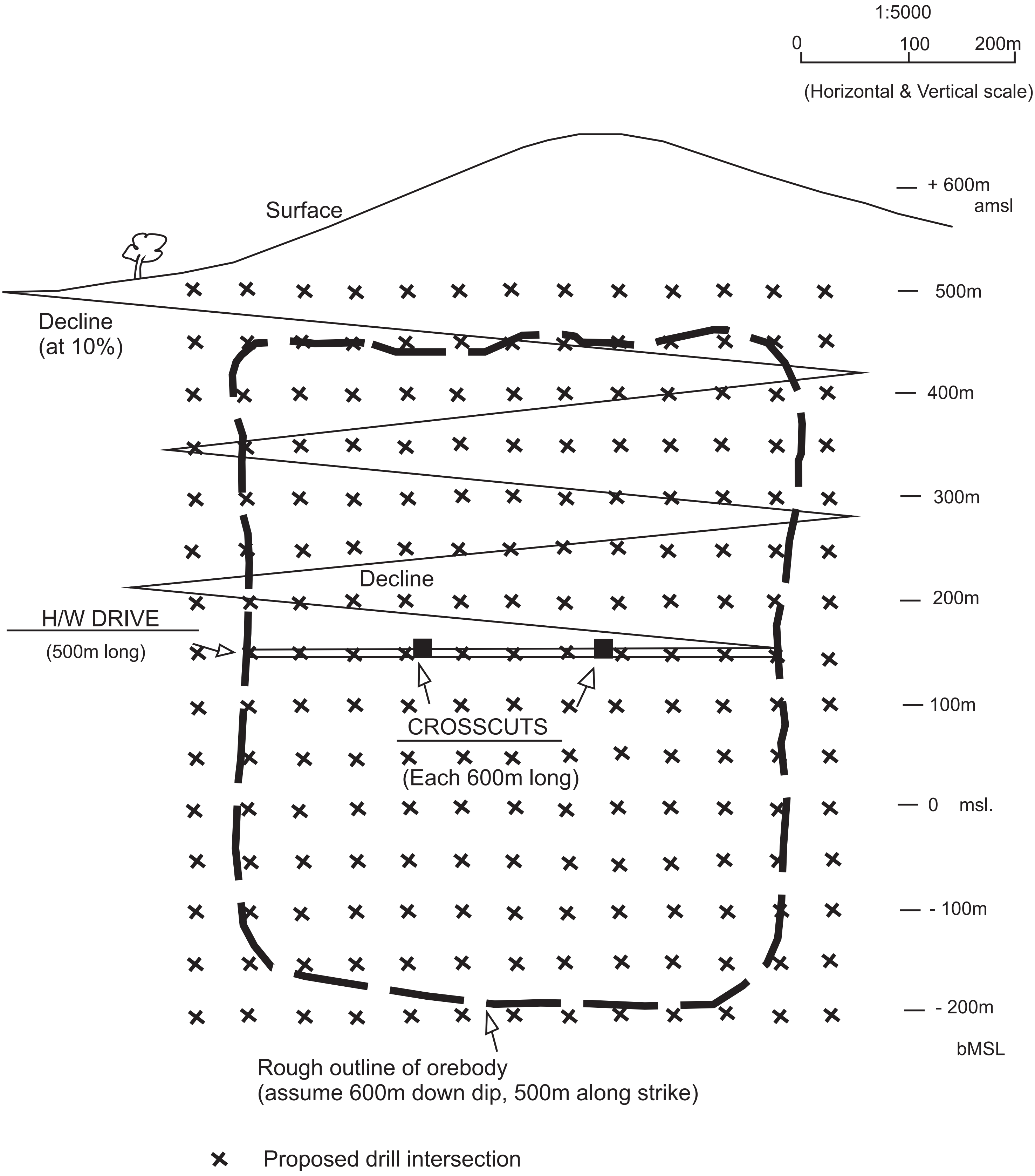
POZDĹŽNÝ GEOLOGICKÝ REZ 2 - 2'
M=1 : 2 000
Zostavil: F. Miháľ, 1991



Príloha č. 5

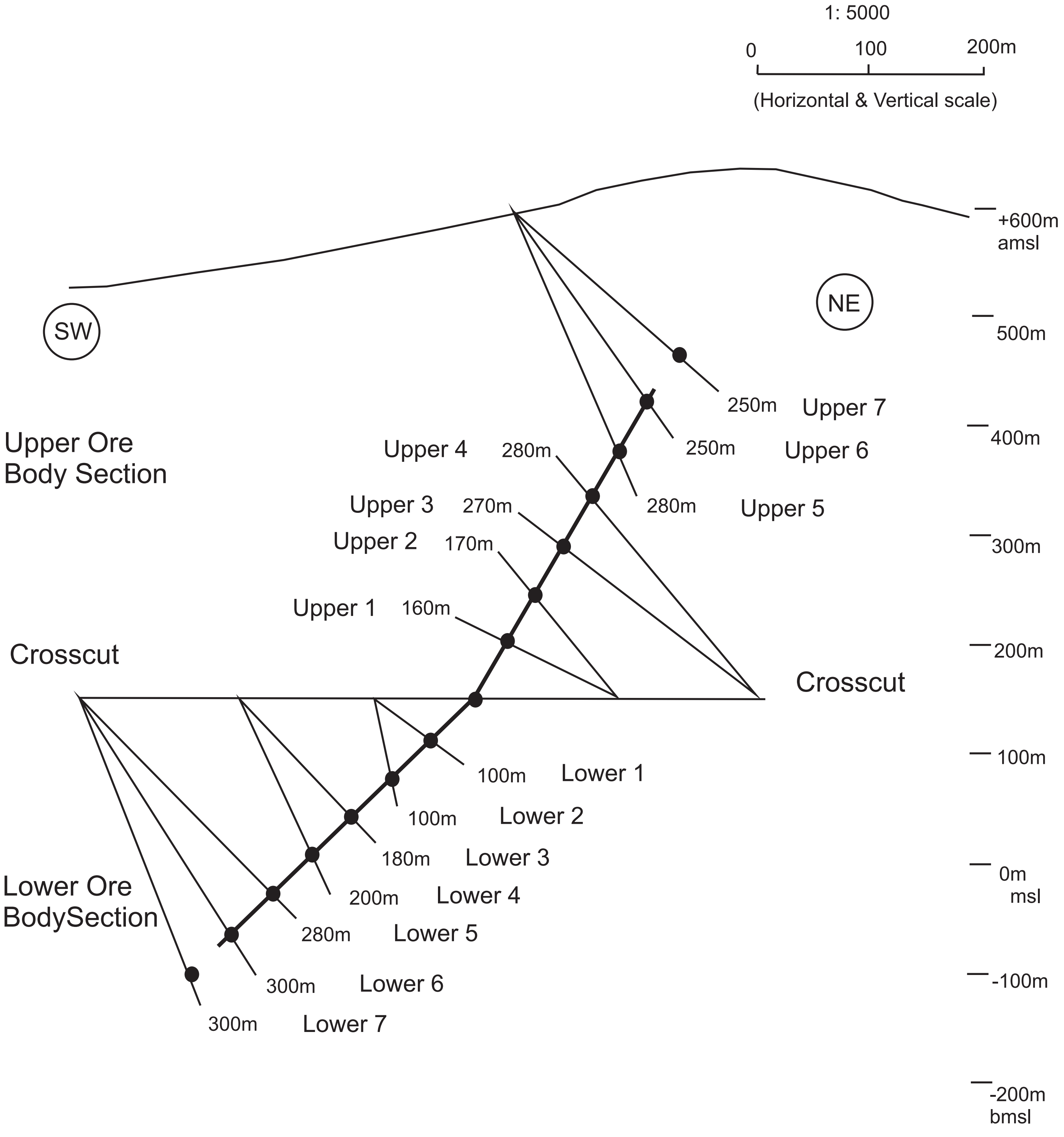
SECTION 6- JAHODNA DEPOSIT- CROSS SECTION 2-2'- scale 1:2,000
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia.

A C A Howe International Limited



SECTION 7 - SIMPLIFIED LONGITUDINAL SECTION
To show proposed decline position. 1:5000
For Tournigan Gold Corporation Jahodna Uranium Project, Slovakia

A C A Howe International Limited



SECTION 8 - SIMPLIFIED CROSS SECTION (SW - NE)
To show proposed underground drilling layout. 1:5000
For Tournigan Gold Corporation Jahodna Uranium Project, Slovakia

A C A Howe International Limited

APPENDIX 2.

Jahodna Report - Plates

Plate 1 - Jahodna Area - deciduous woodland

Plate 2 - Jahodna - Drill Hole KGJ1 - core boxes and drill

Plate 3 - Jahodna - Drill Hole KGJ1 - drill and Uaz logging van

Plate 4 - Jahodna - Drill Hole KGJ1 - Longyear drill

Plate 5 - Jahodna - Drill Hole KGJ1 - Praga 6WD trucks

Plate 6 - Jahodna - Drill Hole KGJ1 - Uaz 4WD logging van

ACA HOWE INTERNATIONAL LTD

Plate 7 - Jahodna - Drill Hole KGJ1- core boxes and trucks

Plate 8 - Jahodna - Drill Hole KGJ1 - core boxes (some covered)

Plate 9 - Jahodna - Drill Hole KGJ1 - ZRUP portable gamma logger

Plate 10 - Jahodna - Drill Hole KGJ1- Tectonised reformed tuff (75m)

Plate 11 - Jahodna - Drill Hole KGJ1- Competent andesitic tuff, foliation at 30 deg to Core Axis (198m)

Plate 12 - Jahodna - Drill Hole KGJ1- Pink Quartz Dolomite Vein (285m)

Plate 13 - Jahodna - Drill Hole KGJ1- hematitic zone with gamma kick (312m)

Plate 14 - Jahodna - Drill Hole KGJ2 - Prospector Drill

Plate 15 - Jahodna - KORAL Logging van

Plate 16 - Jahodna - KORAL Logging van and sonde

Plate 17 - Jahodna - F. Pomorsky with portable Gamma Logger

Plate 18 - Jahodna - F. Pomorsky checking radioactivity in Knolske Formation sediments

APPENDIX 3.

MICROMINE STUDY OF THE JAHODNA URANIUM RESOURCE, SLOVAK REPUBLIC

for
TOURNIGAN GOLD CORPORATION

by
ACA HOWE INTERNATIONAL LTD

March 28, 2006

Berkhamsted
Herts, UK

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EXECUTIVE SUMMARY

A polygonal wireframe estimate was calculated for the Jahodna Resource (December 2005) using data from thirteen (13) historical drillholes over the project area. The results are contained in Table 1 and are given for a cut-off of 0.030%U.

Cut-off grade	Tonnage	Grade (%U)	SG	U (lbs)
0.030%U	1,100,363	0.55	2.72	13,344,645

TABLE 1. GLOBAL POLYGONAL WIREFRAME ESTIMATE (DEC05)

Following re-evaluation of the Jahodna deposit and the drilling of three (3) new drill holes at Jahodna (KG-J-1, KG-J-1a and KG-J-2) an updated polygonal wireframe estimate was calculated in February 2006 that included the additional drilling data. The updated resource estimate is contained in Table 2 below:

Cut-off grade	Tonnage	Grade (%U)	SG	U (lbs)
0.030%U	1,1256,088	0.56	2.72	15,510,172

TABLE 2. GLOBAL POLYGONAL WIREFRAME ESTIMATE (FEB06)

The February resource estimate predicts a 14% increase in tonnes, 2% increase in grade and an overall 16% increased in contained U pounds (lbs) compared with the December 2005 estimate. Given the narrow and often high-grade nature of the deposit, a cut-off of 0.030%U is sensible and should be used in future estimations and refined as new drilling is undertaken, if necessary.

1. INTRODUCTION

Following acquisition of historical drill hole data over the Jahodna uranium project, drill hole data was imported in to Micromine software for the purposes of producing a Polygonal Wireframe Resource Estimate (PWRE) for comparison with historical resource estimates reported for the project. This resource calculation was completed in December 2005. In addition, following the drilling of three new holes at the Jahodna Project, the PWRE was calculated again (February 2006) to include the new drill hole data.

2. DATA USED IN MODELLING

Limited historical raw data was presented for the study and consisted of collar, survey and assay data for thirteen (13) drillholes. This historical data was assumed to be correct and no checks were made to confirm its validity. Assay data consisted of two tables, the first contained downhole mineralised intervals at a 0.015% U lower cut-off and the second, mineralised intervals at a 0.030% U lower cut-off , the latter being used for this resource estimate. Following the drilling of new holes at Jahodna (KG-J-1, KG-J-1a and KG-J-2), geochemical assay data for the new holes was added to the assay tables in Micromine (shown as shaded in the tables 3-4 below).

Table 4 also contains true thickness values for each mineralised interval A review of drill hole traces through the mineralised envelope indicates that drilling is largely sub-perpendicular to the mineralised zone and true thickness values are, on average 15% less than drill hole interval thicknesses.

Full down hole survey data, geophysical data and geochemical assay data was received for the new drilling and reviewed prior to incorporating the data into the new model. Downhole survey data was directly imported in to Micromine from excel spreadsheet data provided by Tournigan. All drill hole positions are shown on the accompanied Jahodna Drillhole Plan.

TABLE 3. COLLAR DATA

Hole ID	Depth (m)	Dip	Azimuth	Northing	Easting	RL
VRT_992	470.00	-89.3	25	-1234199.06	-270390.03	590.50
VRT_1179	558.60	-85.5	25	-1234432.75	-270395.30	589.97
VRT_1180	573.00	-90.0	29	-1234142.74	-270593.21	571.38
VRT_1181	390.20	-79.0	75	-1234113.37	-270426.76	576.91
VRT_1182	403.00	-76.6	53	-1234049.28	-270463.45	568.07
VRT_1215	448.00	-86.5	45	-1234114.44	-270430.49	576.51
VRT_1218	405.00	-90.0	68	-1234081.17	-270494.99	566.28
VRT_1220	452.00	-75.0	336	-1234360.28	-270263.07	594.20
VRT_1222	381.00	-78.9	50	-1234084.32	-270496.03	566.95
VRT_1223	580.00	-90.0	165	-1234144.29	-270590.99	571.57
VRT_1233	780.00	-90.0	0	-1234404.37	-270573.31	610.87
VRT_1247	439.00	-74.1	338	-1234356.92	-270260.29	594.79
VRT_1248	412.50	-85.9	296	-1234114.82	-270429.25	576.77
KG-J-1	439.00	-85.0	40	-1234093.73	-270513.97	565.57
KG-J-1a	442.00	-88.5	5	-1234092.02	-270512.46	565.67
KG-J-2	480.00	-88.1	40	-1234165.13	-270473.30	575.41

Hole ID	From (m)	To (m)	Interval (m)	True Thickness (m)	U%
VRT_992	440.03	440.70	0.67	0.62	0.055
VRT_1179	512.01	513.00	0.99	0.97	0.116
VRT_1180	510.51	518.50	7.99	7.55	0.494
VRT_1181	318.39	322.00	3.61	3.49	0.164
VRT_1182	301.23	305.30	4.07	4.01	1.092
VRT_1215	402.58	404.20	1.62	1.48	1.111
VRT_1218	395.21	398.10	2.89	2.80	0.413
VRT_1220	404.62	408.10	3.48	3.23	0.475
VRT_1222	361.85	363.20	1.35	1.31	0.821
VRT_1233	708.35	709.20	0.85	0.82	0.175
VRT_1247	429.22	429.60	0.38	0.36	0.075
VRT_1248	400.43	401.50	1.07	1.00	0.676
KG-J-1	406.90	408.10	1.20	0.99	0.387
KG-J-1a	424.00	424.90	0.90	0.77	8.829
KG-J-2	450.90	452.00	1.10	0.93	1.140

TABLE 4 ASSAY DATA

3. DATA VALIDATION

Data files were created in Micromine for collar, assay and survey data and then a validation function performed to check for data errors. No errors were detected.

4. WIREFRAMING AND INTERPRETATION

Drillholes and assay data were plotted on screen and checked against historical plans and hand drawn cross sections to confirm hole paths and the position of mineralised intervals downhole.

A series of cross sections were then produced (1-1', 1a-1a', 2-2' and 3-3') for variant 1 and 2 replicating historical hand drawn sections (roughly NE-SW). In addition an extra cross section was produced termed 1b-1b' and isn't included in historical sections. Mineralised intervals on each section were then joined up, honouring the geometry of the mineralised zone as interpreted in historical cross sections. Cross sections with the interpreted mineralisation outline at a 0.030% U lower cut-off are contained in Sections 1-5 below along with a drill hole plan (Plan 1) showing the recent drilling (February 2006)

The mineralised zones were found to be striking NW-SE and shallowly dipping to the SW with the dip becoming steeper to the NE. This geometry mirrors that of historical interpretations.

String sections were then produced for each cross section and mineralised intervals extended to the NE and SW typically by half the distance to the next drillhole. Mineralised zones were also extended to the NW and SE by 60m (the average section spacing over the project) and closed off. These strings were then joined up to produce 3-D mineralised wireframe envelopes for the February 2006 model, which were then validated and a volumes reported for all.

Figure 1 shows the mineralised wireframe and drill hole positions. Figure 2 shows the mineralised wireframe looking NW and clearly shows the steep to shallowly dip to the SW, of the mineralised envelope.

5. POLYGONAL WIREFRAME ESTIMATION

A polygonal wireframe estimate was then performed on the 3-D wireframe and a volume, tonnage and grade reported.

The Polygonal Estimation method was chosen as data was limited, and as a simple method of determining a resource figure that could be compared to historical calculations and check their validity.

Polygonal estimation is a linear estimation technique, based on the linear weighted average of all drillhole assay intervals, applied to a given volume (in this case the wireframe).

Parameters used in calculating the Polygonal Wireframe Estimate were as follows;

Wireframe (Dec05 model):		variant2_alljan06
Wireframe (Feb06 model):		variant2_allfeb06mod
Imput files:	Assay:	jahodna_lowcutcomp, jahodna_highcutcomp
	Survey:	jahodna_survey
	Collar:	jahodna_collar
S.G		2.7

6. COMMENT

The February 2006 model predicts a 14% increase in tonnes, 2% increase in grade and an overall 16% increased in contained U pounds (lbs) compared to the December 2005 model.

Validation of the February model and associated cross sections and 3-d interpretation was undertaken upon completion of the model to check the validity of the model and also to explain the variability that exists between both models. Cross sections for Variant 1 are attached here (section 1-1', 1a-1a', 1b-1b', 2-2' and 3-3')

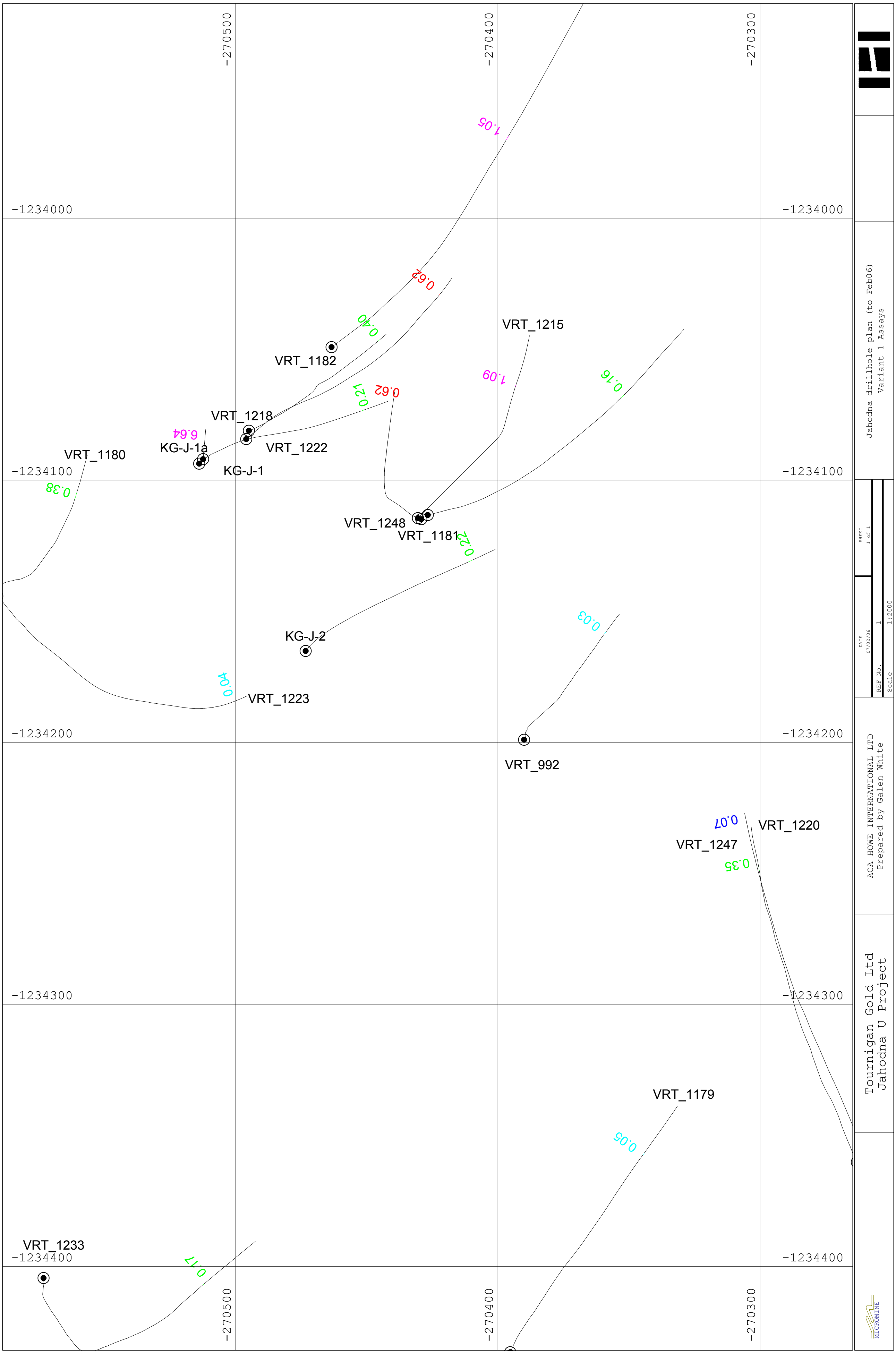
The February 2006 model using a 0.030% lower cut-off shows an increase in tonnage, grade and contained U Lbs compared to the December 2005 model. The increase in tonnes is due to mineralised zone thickness increases on sections 1a-1a' and 1b-1b' by the addition of holes KG-J-1 and KG-J-2. The grade and therefore contained U has increased largely because the grades of hole KG-J-1a and KG-J-2 are the two highest of all the holes and have therefore significantly upgraded the resource.

Given the narrow and often high-grade nature of the deposit, the cut-off of 0.030% U should be used in future estimations and refined as new drilling is undertaken, if necessary.

The Polygonal Estimation method of interpolation introduces conditional bias, i.e. the high grades are overstated, and the low grades are understated, and assumes extraordinary grade continuity between sample points. This means that the resource estimate may overstate the grade (i.e. the higher grade values will have more influence on the overall grade than lower values). When considering the dimensions of the deposit (500m × 500m × 2.5m) which is only informed by 16 grade points, the majority clustered in the NW of the deposit, then high grades will have a large influence on the overall grade.

It was decided to run a polygonal model because of the lack of data with which to calculate a resource (only 16 grade intervals) but with added drill data, a more refined interpolation method (IDW or Kriging) may be undertaken and will provide a more refined grade-tonnage scenario. The polygonal estimates have succeeded in confirming the likely tonnage of the deposit (which compares favourably to historic calculations) but the grade of the deposit may be overstated.

Though simplistic, this method is adequate for the purposes of comparison with historical calculations and for use at the current stage of advancement of the Jahodna deposit.



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KG-J-2
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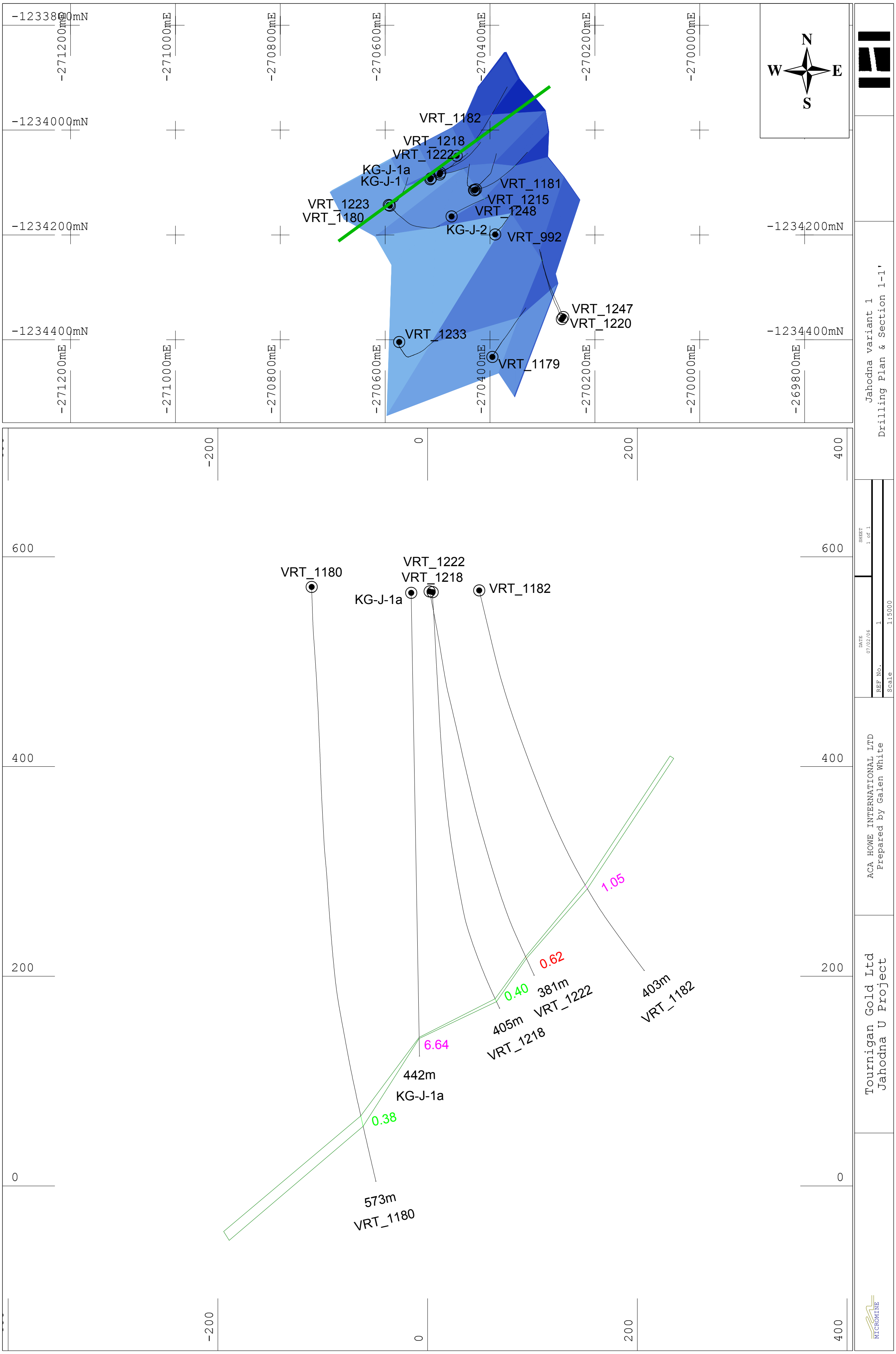
0.17
Jahodna drillhole plan (to Feb06)
Variant 1 Assays
SHEET
1 of 1
DATE
REF No.
1
Scale
1:2000
ACA HOWE INTERNATIONAL LTD
Prepared by Galen White
Tournigan Gold Ltd
Jahodna U Project
MICROMINE


Tournigan Gold Ltd
Jahodna U Project
ACA HOWE INTERNATIONAL LTD
Prepared by Galen White
Jahodna variant 1
Drilling Plan & Section 1-1'
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E
S
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VRT_1218
VRT_1222
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KG-J-2
VRT_992
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VRT_1220
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VRT_1179
-200
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200
400
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1.05
0.62
0.40
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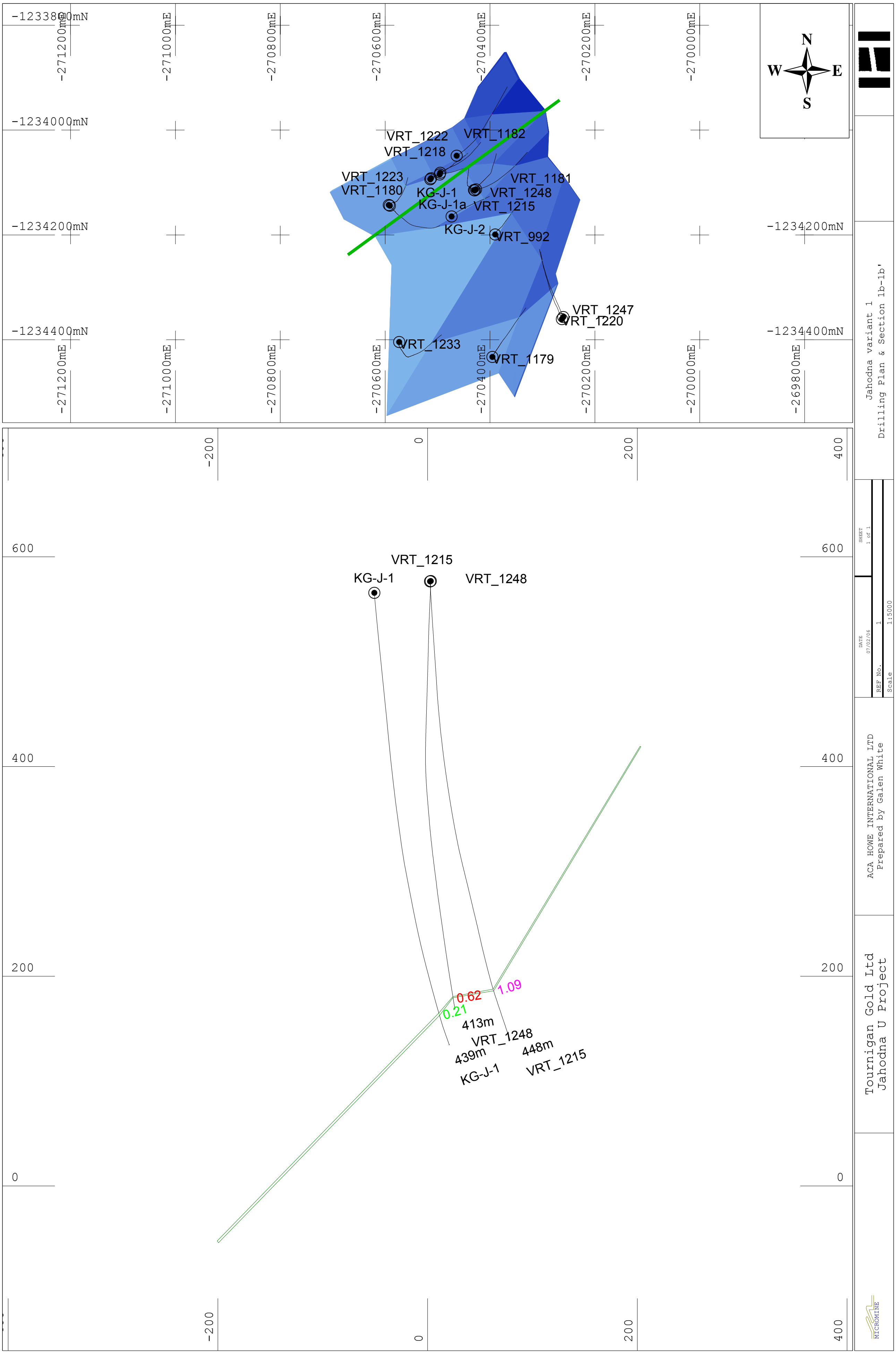

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VRT_1182
VRT_1218
VRT_1223
VRT_1180
KG-J-1
KG-J-1a
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VRT_1248
VRT_1215
KG-J-2
VRT_992
VRT_1247
VRT_1220
VRT_1233
VRT_1179
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0
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400
600
VRT_1215
KG-J-1
VRT_1248
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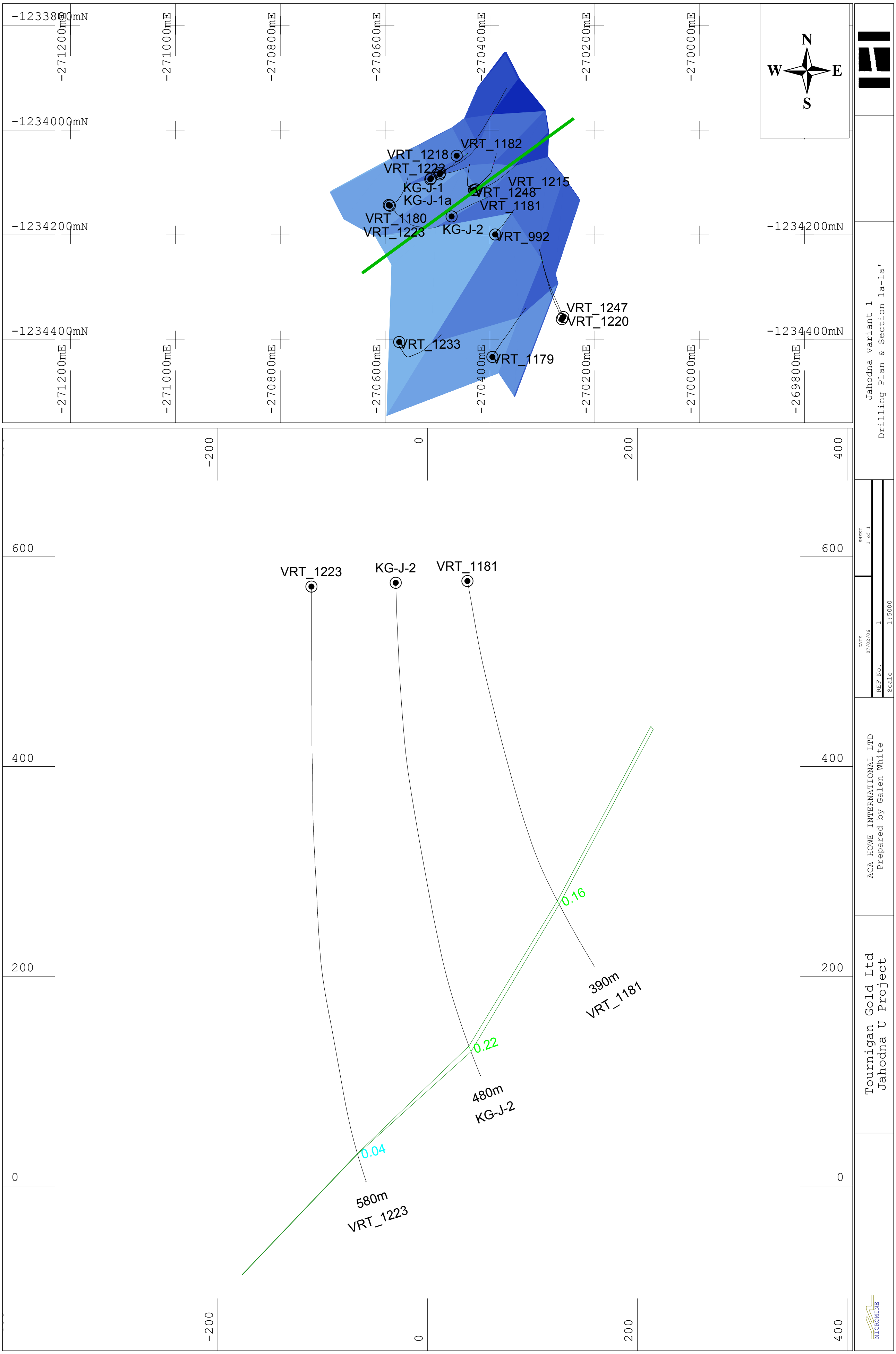
0.21
413m
VRT_1248
439m
448m
KG-J-1
VRT_1215
Jahodna variant 1
Drilling Plan & Section 1b-1b'
SHEET 1 of 1
DATE 07/02/06
REF No. 1
Scale 1:5000
ACA HOWE INTERNATIONAL LTD
Prepared by Galen White
Tournigan Gold Ltd
Jahodna U Project
MICROMINE


Tournigan Gold Ltd
Jahodna U Project
ACA HOWE INTERNATIONAL LTD
Prepared by Galen White
Jahodna variant 1
Drilling Plan & Section 1a-1a'
SHEET 1 of 1
DATE 07/02/06
REF No. 1
Scale 1:5000
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VRT_1218
VRT_1222
KG-J-1
KG-J-1a
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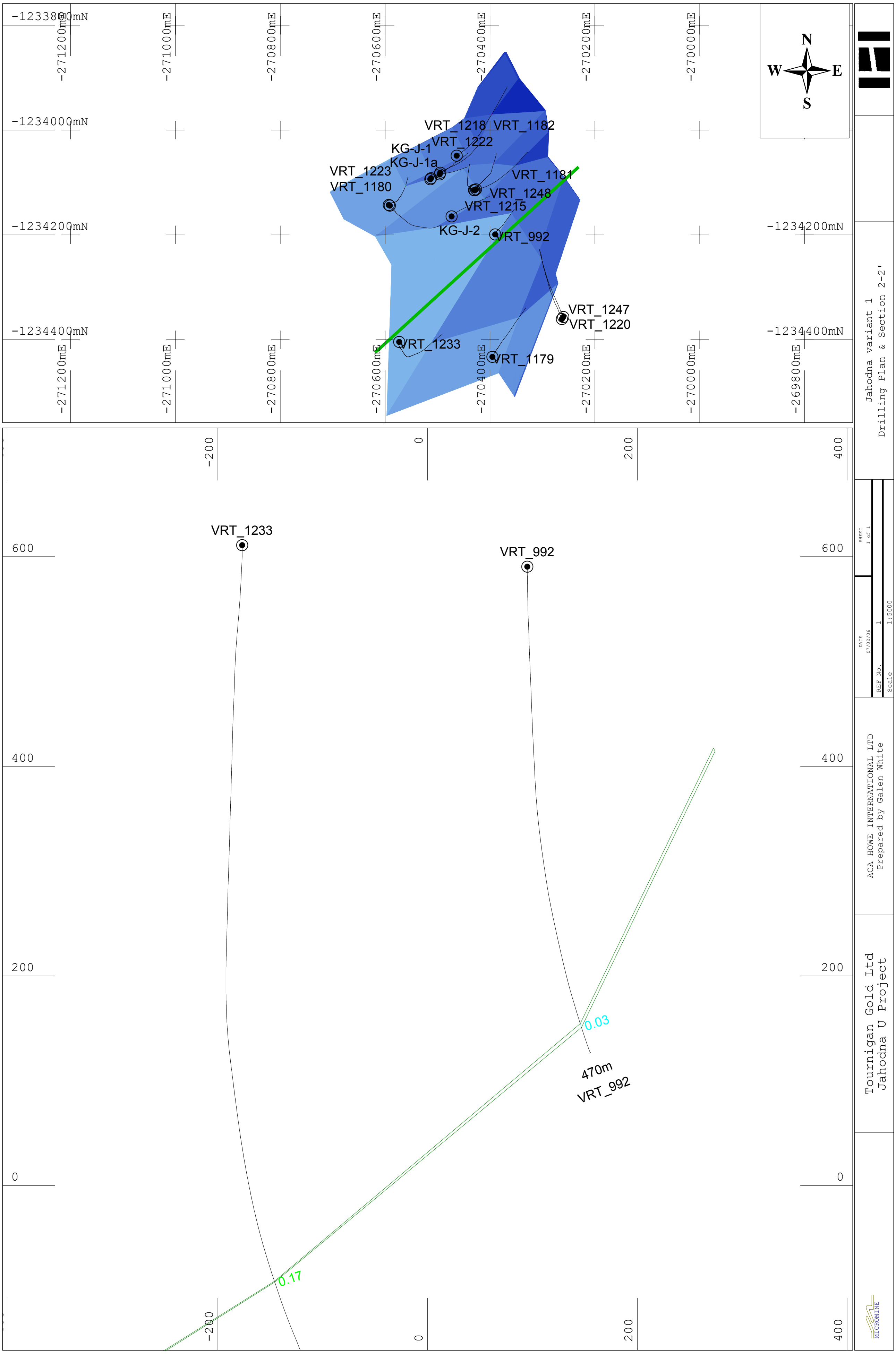

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400
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VRT_992
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VRT_992
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Jahodna variant 1
Drilling Plan & Section 2-2'
SHEET 1 of 1
DATE 07/02/06
REF No. 1
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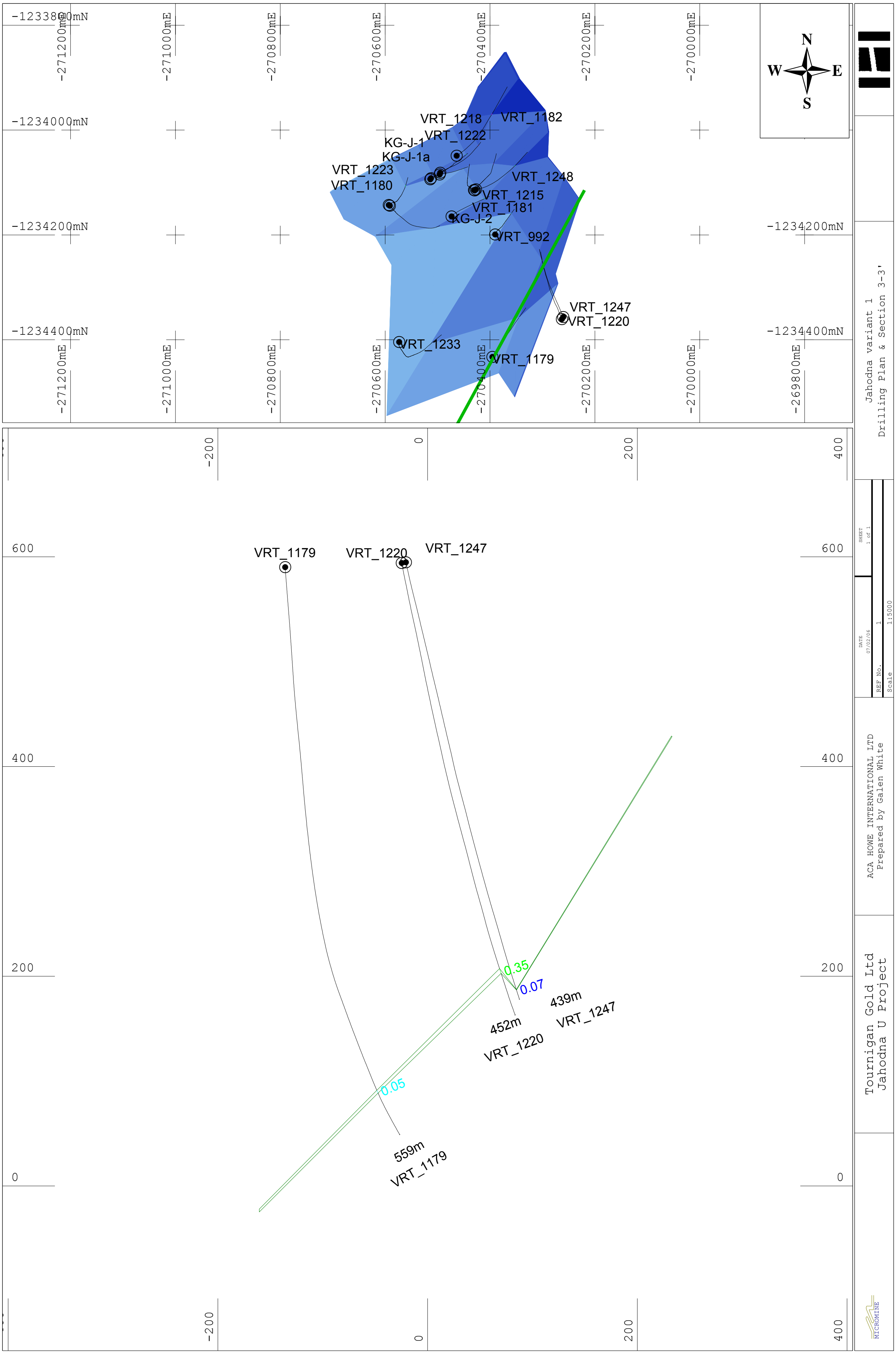
ACA HOWE INTERNATIONAL LTD
Prepared by Galen White
Tournigan Gold Ltd
Jahodna U Project
MICROMINE



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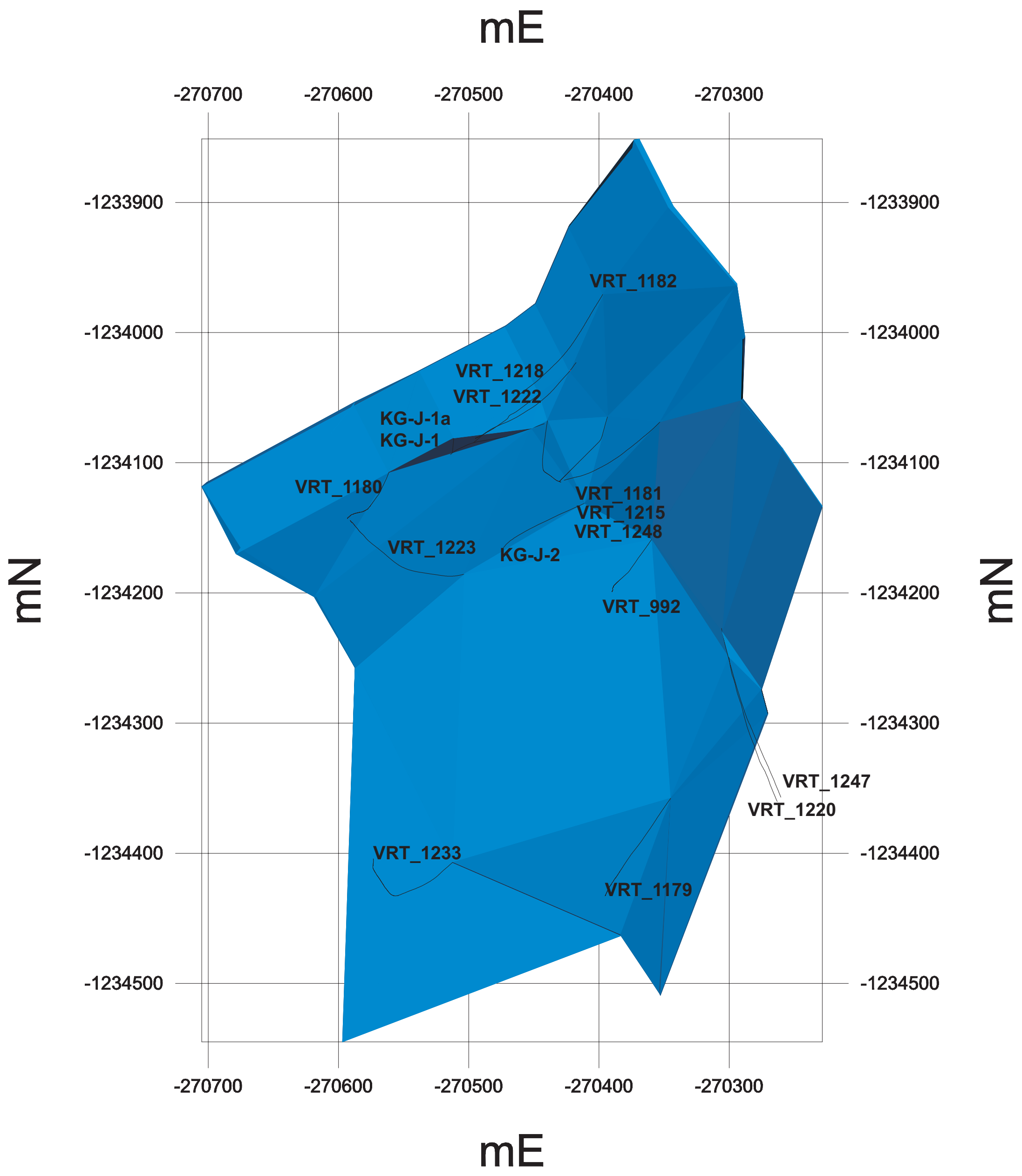
Jahodna variant 1
Drilling Plan & Section 3-3'
Scale 1:5000
ACA HOWE INTERNATIONAL LTD
Prepared by Galen White
Tournigan Gold Ltd
Jahodna U Project
MICROMINE



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Jahodna variant 1
Wireframe and Drill Hole Plan

SHEET 1 of 1
DATE 07/02/06
REF No. 1
Scale 1:1000

ACA HOWE INTERNATIONAL LTD
Prepared by Galen White

Tournigan Gold Ltd
Jahodna U Project

FIGURE 1 - JAHODNA - VARIANT 1 WIREFRAME - PLAN VIEW
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited

MICROMINE

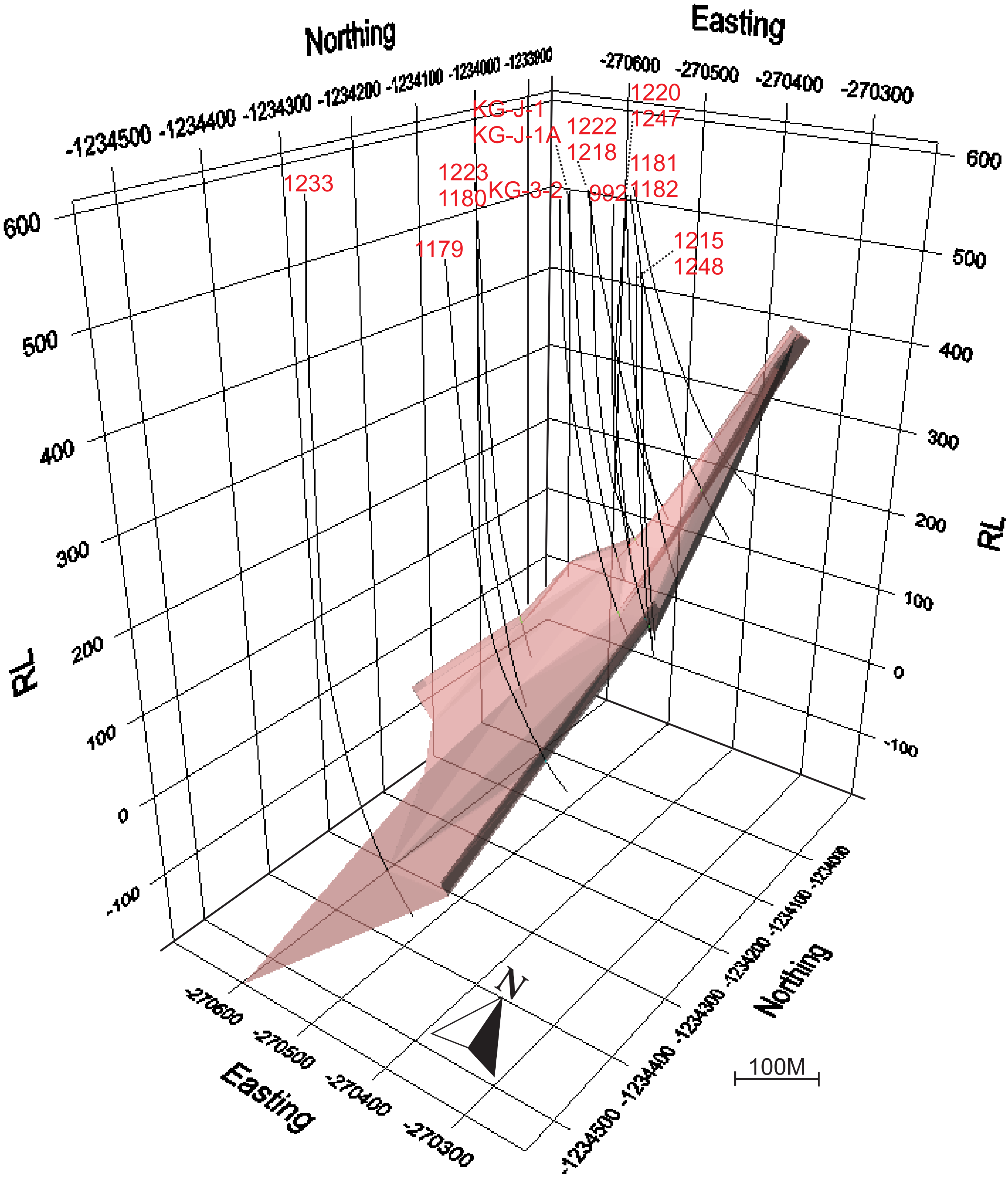

FIGURE 2 - JAHODNA VARIANT 1 - WIREFRAME CROSS SECTION LOOKING NORTHWEST
For Tournigan Gold Corporation, Jahodna Uranium Project, Slovakia

A C A Howe International Limited

Signatures

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

Tournigan Gold Corporation
(Registrant)

December 12, 2006	By: /s/ James Walchuck
Date	James Walchuck, President and CEO