WRITTEN STATEMENT

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Re: TERMS OF REFERENCE OF THE ENVIRONMENTAL IMPACT STATEMENT FOR THE PROJECT PROPOSAL AND MINE WASTE MANAGEMENT PLAN.

PART: PROCESS OPTIONS FOR PROCESSING OF ADA TEPE ORES, KHAN KRUM DEPOSIT.

INPUT DATA

The project proposal considers two alternative options for processing of the ores mined from the Ada Tepe portion of the Khan Krum deposit:

Option 1: Processing of the ore to gold-silver concentrate as the end product based on a combined flowsheet of flotation and gravity separation;

Option 2: Processing of the ore to end metals (dore gold alloy) using cyanide leaching of gold and silver.

It should be pointed out that this is the second submission of a project proposal, an EIS TOR document and a Mine Waste Management Plan. The first one was made in 2005 and the EIS was completed in April of the same year by a team of experts led by Dr. Y. Pelovsky and Dr. N. Pipkov.

The 2010 proposal includes a Metallurgical Testwork section, which discusses the combined flotation and gravity separation testwork program completed on Ada Tepe samples. As a result, a new EIS is under preparation, which should be submitted to the Ministry of Environment and Waters. This EIS would primarily discuss Option 1.. Since BMM's project proposal considers both options, this statement will discuss and compare both process alternatives in detail. The reason is to select the option that ensures better technical, economic and financial results in full compliance with the environmental standards and regulations.

RESERVES AND RESOURCES

The submission including the project proposal, the EIS TOR and the Mine Waste Management Plan are based on the Ada Tepe Resource/Reserve Statement dated 01.09.2004. The Statement reports:

- Possible reserves (code 122) @ 0.9 g/t cut-off; 1,493,000 t @ 7.3 g/t Au and 4.3 g/t Ag; 17.294 t Au metal and 7.503 t Ag metal;
- Measured resources (code 331) @ 0.9 g/t cut-off; 7,292,000 t @ 2.4 g/t Au and 1.0 g/t Ag; 17.294 t Au metal and 7.503 t Ag metal;

The overview of the deposit included in the Mine Waste Management Plan prepared by BMM in July 2010 indicates the following:

The Ada Tepe deposit can be classified as a high-grade, shallow, low-sulphidation epithermal style gold-silver deposit. Hydrothermal alteration with strictly controlled gold-bearing mineralization is found at Ada Tepe. The hydrothermal alteration is the strongest in the sediments at the base of the Paleogene complex. Alteration consists of intensive silicification, silica, quartz-adularia and quartz-carbonate veinlets, argillisation, oxidation. Hydrothermal alteration of the basement metamorphic rocks is weaker - they are argillised and intensely cataclised in immediate proximity to the contact with the sediments.

Mineral Composition of Ada Tepe Ore

The mineralogical characterisation is based on detailed understanding of the mineralogy of the primary ore and gangue minerals. Under a microscope, the following is observed: "The gold is contained in electrum. The native gold occurs as fine impregnations that are corrosively intergrown with quartz and as relics within limonite. Its size ranges from 1-2 µm to 6-8 µm and is observed as micro-agglomerations and gold emulsion within quartz or developed as framing aggregates around clayey sections within the quartz. Native gold is found to contain about 82.5% Au and 17.5% Ag. 0.1 to 0.2% Fe and 0.2 to 0.44% Cu are also identified.

Pyrite occurs in several generations but its head grade in the ore is below 1.0%. Under a microscope, the pyrite is observed to have idiomorphous to hypidomorphous texture, which is occasionally corroded by later quartz and magnetite. There are no indications that pyrite includes submicron gold and therefore it is not regarded as a gold-bearing mineral.

The assays of bulk samplest indicate the following grades of the main components: silica dioxide - 80.2 %, dialuminum trioxide - 5.90 %, diiron trioxide - 3.28%, calcium oxide - 2,80%, sulphur trioxide - 0.22%.

The above testwork data about the composition of the Ada Tpe ore is sufficient to indicate the viable options for processing of that type of ore. These indications should also be confirmed by conducting the necessary metallurgical testwork for assessment of the amenability of this ore to processing.

PROCESS EQUIPMENT

Each of the the two process flowsheets requires the following equipment:

1. COMBINED GRAVITY SEPARATION AND FLOTATION FLOWSHEET:

A jaw crusher with an estimated capacity of 200-250 tph and a discharge end diameter of approx. 150mm. It will be installed at a suitable location in the open pit.

A fully enclosed inclined belt conveyor leading to the feed hoppers in the grinding section.

A primary semi-autogenous grinding mill operating in an open circuit.

Regrinding in a secondary ball mill and a tertiary vertical stirred mill.

Three hydrocyclone classification stages to obtain a typical final grinding product P_{80} = 40 microns.

A cone crusher to handle the pebbles recycled from the semi-autogenous grinding mill.

A screening section for removal of any trash, mostly wooden and plastic waste, from the ore feed.

Moseley gravity separators (separation/shaking tables). Centrifugal machines may also be utilised in the gravity separation circuit. A separate gravity separation sheet is not provided but it is expected that the gravity concentrate would provisionally contain 3,000 to 5,000 g/t Au and 1,000-2,000 g/t Ag.

A flotation circuit consisting of one rougher stage, three cleaner stages and two scavenger stages. The type of flotation banks that is suitable for concentration of low-sulphide ores is not specified. Reagents used - copper sulphate, water glass (sodium silicate), dithiophosphate (Aerofloat 208), PAX (potassium amyl xanthate), frother and flocculant.

Concentrate dewatering - the proposal does not specify the type of equipment intended to be used.

An Integrated Mine Waste Facility, which will be the ultimate mine waste storage facility for co-disposal of mine rock and flotation tailings.

2. HYDROMETALLURGICAL FLOWSHEET FOR RECOVERY OF PRECIOUS METALS USING CYANIDATION.

The flowsheet includes the following main circuits:

A two-stage crushing circuit with a primary jaw crusher and a secondary cone crusher.

A two stage grinding circuit with a primary semi autogenous grinding mill and a secondary (regrinding) ball mill.

A leaching and adsorption circuit, which is a combination of the CIP (carbon-in-pulp) and CIL (carbon-in-leach) processes, performed in one leaching and six adsorption 720 m^3 open top tanks equipped with open launders, airlift-agitators and intertank screens to retain the carbon.

The loaded carbon from the leaching circuit is washed with diluted hydrochloric acid and water, and then fed into the elution column, where the carbon is eluted using a concentrated sodium hydroxide and sodium cyanide solution under high temperature.

After the leaching process, the ore pulp is thickened and subjected to cyanide destruction using the so-called INCO process (INCO SO_2/Air Cyanide Removal Process), which utilises sodium pyrosulphite. The detoxified final tailings are advanced to the TMF while the thickener overflow is recycled back into the process.

The pregnant eluate directly reports to the electrowinning circuit, where the metals deposited onto the cathodes are removed and melted to produce dore gold bullion. The alloy will contain between 70 and 80% gold and 20 to 30% silver.

ANALYSIS OF TESTWORK COMPLETED TO-DATE

The testwork necessary to provide all Ada Tepe process plant design criteria for the detailed design was undertaken for BMM and its parent Dundee Precious Metals by:

1. For Option 1, SGS Canada Inc. in its research and testwork centre in Lakefield, Ontario, Canada in 2009 and 2010.

2. For Option 2, by Ausenco Limited, Australia, in 2004 and 2005.

To select the best option for ore processing requires detailed comparison of all the data from both sources in three main directions: technical and process solutions, economics and finance, and environmental compliance.

First and foremost, it should be noted that there is a difference in the Ada Tepe resource estimates, which give the benchmark for assessment of the proposed process alternatives.

According to the DFS issued in 2005, the resource estimates give measured + indicated resource of 4.4 Mt @ 5.4 g/t Au and 3.0 g/t Ag. Gold metal is 757,000 oz. or 23.545 t and silver is 394,000 oz. or 12.225 t. This is from the 2004 resource estimates.

The 2010 project proposal reports measured resources (code 331) of 7.292 Mt @ 2.4 g/t Au and 1.0 g/t Ag. Gold metal is 17.294 t and silver is 7.503 t. The probable reserves (code122) are 1.493 Mt @ 7.3 g/t Au and 4.3 g/t Ag. Gold metal is 10.893 t and silver is 6.44 t. This is from the Resource/Reserve Statement dated 01.09.2004.

The 2005 metallurgical testwork was carried out on three main composites representing the Oxidised Upper Zone, Oxidised Wall Zone and Fresh Wall Zone. The head assays of the three metallurgical composites are given below:

Metallurgical Composite Head Assay (Ausenco, 2005)

Element	Unit	Fresh Wall Zone	Oxidised Wall Zone	Oxidised Upper Zone
Au	g/t	7.97	13.8	4.07
Ag	g/t	3.10	4.7	5.1
S total	%	0.38	0.05	0.05
S sulphide	%	0.27	0.05	0.05

The testwork for the development of a combined flotation-gravity separation flowsheet was carried out in Lakefield, Canada, in 2009-2010 on two main composites representing the Oxidised Upper Zone and Fresh Wall Zone. Detailed mineralogical analysis was performed on the samples and material from drill cores. The head assays of the two metallurgical composites are given below:

Metallurgical Composite Head Assay (Lakefield, Canada)

Element	Unit	Oxidised Upper Zone	Fresh Wall Zone
Au	g/t	5.84	6.17
Ag	g/t	3.30	3.50
Fe	%	2.41	1.66
S	%	0.06	0.48
Cu	g/t	23.00	23.00

The main composites are called "representative" but the significant variance in Au and Ag grades and between the two resource estimates does not confirm their "representativeness". Each composite weighed between 50 and 80 kg.

According to the regulations that are accepted in our country, one cannot estimate cut-offs, do resource/reserve reporting, design and commission mining and processing facilities without having results from detailed laboratory testwork that are backed up by pilot plant testwork. This requirement is a fair one because such data provides the process plant design criteria for the detailed design, the end operating and economic results and the compliance with the environment protection requirements.

Both studies referred to above provide ore preparation criteria and the results that enable selection of the preferred process flowsheet out of gravity separation, flotation and hydrometallurgy. The hydrometallurgical option includes gold extraction by a conventional cyanidation process, which is used to produce about 90% of the gold worldwide. These studies also consider the quality and tonnage of the end products, as well as the waste storage and water recycling options. Based on this data, each research centre arrives at a different preferred process flowsheet for the processing of the Ada Tepe ore determined by its environmental, technical and economic advantages:

The Australian Ausenco Limited selects gold extraction by a conventional cyanidation process as the preferred option for project development.

SGS Canada Inc, on the other hand, selects a combined flotation-gravity separation flowsheet as the preferred option for project development.

The key criteria that influence the assessment and selection of one of the two proposed options are:

1. The Ada Tepe deposit can be classified as a high-grade, shallow, low-sulphidation epithermal style gold-silver deposit with gold finely intergrown with silica. This means that obtaining gravity and flotation concentrates would be very difficult due to the relatively low concentrate yield from the ore, which would impede the recovery of the precious metals into the concentrates.

2. The combined flotation-gravity separation flowsheet includes a three-stage grinding circuit to obtain a typical P_{80} final grind size of 40 µm. The cyanidation flowsheet includes a two-stage

grinding circuit to obtain a nominal P_{80} grind size of 75 µm. The influence of this variance will be further explained in detail.

3. Section S.5.4 of the feasibility study document (2005) describes the response of the ore to gravity concentration based on the testwork on Ada Tepe samples carried out by Ausenco:

   "Preliminary gravity concentration testwork conducted on the three main composites ground to a 80% passing (P_{80}) size of 350 µm and gave recoveries to gravity concentrate ranging from 3.2% to 5.2%. The oxidised Upper Zone gravity tail sample was reground to a P_{80} of 75 µm and reprocessed. This resulted in additional gravity recovery to concentrate giving an overall recovery to final gravity concentrate of 21.5%. Intensive cyanidation of the gravity concentrate gave gold recoveries between 91% and 99% from the gravity concentrate. Although a gravity recovery of 21.5% at a grind of P_{80} 75 µm is reasonable, to achieve this in an operating plant would be difficult. Given the excellent response of the three main composites to direct cyanidation, the very fine nature of the free gold and the relatively poor response to gravity concentration, there has been no gravity circuit included in the flow sheet."

   Table 11 in the SGS Canada's report "A METALLURGICAL TESTWORK PROGRAM ON ORE SAMPLES FROM THE KRUMOVGRAD DEPOSIT IN BULGARIA, April 2010" summarises the gravity performance for the two master composites. For the UpOx composite, 11.7% of the Au and 6.9% of the Ag in the feed were recovered to a gravity concentrate @ 2421 g/t Au and 841 g/t Ag representing 0.03% of the feed mass. For the Wall composite, 10.6% of the Au and 7.7% of the Ag in the feed were recovered to a gravity concentrate @ 2210 g/t Au and 864 g/t Ag representing 0.03% of the feed mass.

4. Both companies carried out flotation testwork.

   Results from Ausenco's flotation testwork indicate that for samples ground to P_{80} of 75 µm, between 60% and 80% of the gold in the feed is recovered to a flotation concentrate representing 1% to 6% of the feed mass. No concentrate grade information is provided in the report but the balance suggests that for feed headgrades < 3.0 g/t, the resulting concentrate Au grade would be 50 to 100 g/t and for feed headgrades > 4.5 g/t, the resulting concentrate Au grade would be 100 to 150 g/t with no demonstrable improvement in overall recovery.

   The testwork carried out by SGS Canada Inc indicated that at a P_{80} size of 40µm, the recovery for the UpOx composite was 75% on average, while the Au recovery for the Wall composite ranged from 63.9 to 80.5%. The open-cycle testwork using a combined gravity+flotation flowsheet achieved an overall Au recovery of 85% for the UpOx composite and 63.9 to 80.5% for the Wall composite.

   At the conclusion of the batch testwork, a locked-cycle flotation test was carried out. The circuit flowsheet included rougher flotation, 1st cleaner stage and 1st cleaner-scavenger stage (fig. 14). The concentrate obtained from the Wall composite represented 3.6% of the feed mass @ 156 g/t Au, 72.8 g/t Ag and 11.5% S. The recoveries were 81.3% for Au, 71.3% for Ag and 83.2% for S.

The concentrate obtained from the UpOx composite represented 1.9% of the feed mass @ 243 g/t Au, 98.8 g/t Ag and 4.7% S. The recoveries were 80.2% for Au, 58.9% for Ag and 78.2% for S. The headgrades of the feeds selected for testwork were: Wall Comp.: 6.89 g/t Au, 3.67 g/t Ag, 0.5% S; UpOx Com.: 5.51 g/t Au, 3.05 g/t Ag, 0.11% S (Table 19). The grind size was P_{80} of 50 µm for the Wall composite and P_{80} of 40 µm for the UpOx composite. There are no results for the proposed final flowsheet with three cleaner stages, which is attached hereto as Fig. 1.

5. Ausenco carried out detailed cyanidation testwork on the three master composites from Ada Tepe. The testwork programme identified the factors influencing the gold and silver leach recovery: feed quality (headgrades), grind and wt % of solids, residence time, pulp temperature, sodium cyanide concentration, CIL pH.

   The results of the testwork programme indicate that:

   – The leach residue grades show a direct correlation with head grade (i.e. constant recovery).
   – Oxygen addition to the leach significantly increases the initial leach kinetics but does not increase overall leach extraction at 48 hours. The increased leach kinetics with oxygen addition effectively reduces the optimal CIL residence time from 36 hours to 24 hours. The use of oxygen rather than air in the CIL with a 24 hour residence time is used in the selected flowsheet. The Ada Tepe ores are considered low oxygen consumer ores. The highest calculated oxygen consumption over the 24 hour leach period is 19 g O_2/t of solids.
   – Leaching at low temperatures of 5°C and 10°C has minimal impact on either the leach kinetics or the final gold recovery.
   – The optimum CIL pulp density is 40% w/w solids.
   – The free cyanide level in the leach solution is set at 150 mg/L, which should be maintained at all times during the process.
   – The cyanide consumption rate is 0.35 kg/t and the lime consumption rate necessary to maintain the target CIL pH of 10.5 is 1.04 kg/t.
   – Gold leach extraction ranges from 67.0% to 98.0% and averages 94.5% and silver leach extraction ranges from 54.6% to 96.0% and averages 91.5% for the variability samples.
   – The final product is doré bullion containing 70% Au and 30% Ag.
   – Cyanide destruction using the INCO process. The process utilises SO_2 and air as oxidising agents. CN (WAD) are destroyed by oxidising cyanide to cyanate and precipitating the cyano/metal complexes as metal-hydroxide compounds with addition of small amounts of copper sulphate. The required SO_2 is derived by adding sodium metabisulphate to the CIL tailings. The results from semi-continuous bench scale pilot testwork indicated that the SO_2/air process can reduce the CN (WAD) level in tailings pulp to less than 1 mg/l against the EU upper limit of 10 mg/l.
   – The final tailings from the process plant may be deposited in a conventional TMF or an Integrated Mine Waste Facility with 100% reclamation of process/decant water without discharges into the environment.

The cyanidation tests carried out by SGS Canada Inc identified the following optimum feed and process parameters:

- Grind size: P_{80} of 74 µm;
- Residence time: 48 hours;
- CIL pulp density: 32% solids;
- NaCN concentration: 2.0 g/l;
- Pulp temp.: 20 °C
- pH range: 10.5-11;
- NaCN consumption rate: 0.27 to 0.30 kg/t;
- CaO consumption rate: 0.5 to 0.8 kg/t;

Under those conditions, the resulting Au and Ag recoveries were 97.5% and 84.7% respectively for UpOx composite and 93.1% and 84.3% respectively for Wall composite.

The overall cumulative Au and Ag recoveries, across gravity, flotation and cyanidation were 94.2% and 88.2% respectively for UpOx composite and 93.9% and 88.4% respectively for Wall composite.

CONCLUSIONS

1. The submission made this year, which includes a project proposal, EIS TOR and Mine Waste Management Plan, has the objective of having a new Environmental Impact Statement prepared and submitted to the Supreme Expert Panel at the Ministry of Environment and Waters for review and approval. The main issue during the review of the 2005 EIS and the reason for having the final EIA resolution delayed by several years was the proposed gold recovery by a conventional cyanidation process. Now, a combined flotation and gravity separation flowsheet is proposed, which is considered to be safe for the environment and the health of the employees and the local communities. The analysis of the two process flowsheets indicates that their full scale implementation would not cause significant problems. There are several differences that may influence the environmental impact assessment and should be considered carefully before the final assessment is issued.

The first of these factors is the above-mentioned use of the highly toxic alkaline cyanides under Option 2, which would be avoided by selection Option 1. The importance of this difference must be analysed and explained in detail before the final decision is made. According to the 2005 Definitive Feasibility Study by the Australian Ausenco Limited (page 25 of the Project Summary), the testwork results have proven that the INCO process can reduce the CN (WAD) level below the EU target of 1.0 mg/l. According to Dr. Terry Mudder at the Workshop "International Cyanide Management Code - Reality and Perspectives" held on 24.04.2009 in Sofia, it was well established that the INCO process, when applied to solutions containing 260 mg/l free cyanide and 250 mg/l WAD cyanide, could effectively reduce those levels to 0.5 mg/l and 0.05 mg/l respectively. When applied to tailings pulp containing 115 mg/l free cyanide and 110 mg/l WAD cyanide, those levels in the final tailings could be reduced to 0.2 mg/l and 0.5 mg/l respectively. The flowsheet developed by Ausenco ensures that the free cyanide level is maintained on 150 mg/L, which is similar to the second example and means that the process, when applied to Ada Tepe CIL tailings, can reduce the CN level in the tailings pulp to below 0.5 mg/l. An introduction to the International Cyanide Management Code was made at the above workshop. The Code is based on solid methodological ground, which ensures safe operation of the mines that use alkaline cyanides. The Code's principles and standards of practice cover all aspects of production, packaging, handling, transportation, storage, preparation and addition of solutions to the process. It also provides guidance on all modern methods of detoxification of tailings and leach solutions, as well as on specialty instruments for process control and warning of potential emergency situations. The Code also includes requirements for the training of personnel working with cyanide and establishes protocols for continuous control and audits, which ensure and verify the compliance with the Code's principles and standards. The adoption and implementation of the Code by the Bulgarian State together with all mining operations that use cyanides would promote safe implementation of the best available techniques and improve the financial stability of the companies in the business, which will help stabilise the broader economy. By implementing these proposals BMM EAD and the Krumovgrad Gold Project will be pioneers in the implementation of efficient process solutions, which will improve the standard of living of the communities in the area of Krumovgrad.

The second factor that makes a difference between the proposed process options is the elevation of the levels of harmful substances in the wastewaters, which will be collected in the TMF or the IMWF. Harmful heavy metals are found to occur in the mined rock material. These are iron, copper, zinc, nickel, cobalt, lead, arsenic, chromium, manganese, cadmium and bismuth. The migration of these elements in the water is caused by oxidation processes, whose mechanism is dictated by the ambient conditions. The pH of the process tailings is a function of the processes and the reagents used in them. Direct cyanidation is performed in a highly alkaline medium, where the pH of the slurry is 10.5, which is maintained by addition of lime solution. Obviously, secondary oxidation of the minerals that contain heavy metals is not possible in such a medium. The favourable effect of this option is that the process plant water requirements can easily be met from recycling. Lime is commonly used as a medium regulator but it can also be used as a coagulant and combined with small amounts of flocculant it creates the best conditions for decantation and removal of suspended solid slime residues. In an extraction system with flotation, the copper sulphate is used to sulphidise oxide minerals but it can also act as a medium regulator because its solutions are strongly acidic. The data in the report by SGS Canada Inc indicate that the flotation feed is slightly alkaline to almost neutral (pH = 7.5-7.8). Under these conditions, the re-use of process water will reduce the pH level of the medium, which will cause elevation of the concentrations of the above-mentioned harmful heavy metals, some of which will exceed the maximum allowable levels. The only way out would be additional construction of a return water treatment plant, whose cost is high. Under these conditions, any emergency release of wastewaters would potentially cause pollution.

Under this system, all drainage and mine water flows may be directed to the waste management facility without allowing them to migrate into other local water sources.

2. When comparing the end technical and economic results from the full scale implementation of each process option, the following points should be considered:

2.1 All gravity concentration testwork results indicate that inclusion of a gravity separation stage in the flowsheet is not recommendable. On one hand, the gravity recoverable gold, even though it is 21% of the total recoverable gold, responds poorly to gravity concentration, which is obviously due to the native gold particle size range. This same gold is 100% recoverable by direct cyanidation. This means that a combined flowsheet including a gravity circuit will unnecessary inflate the project capital and operating costs.

2.2 The main problem in accepting the suitability of a combined gravity+flotation flowsheet is the considerably lower recoveries of gold and silver that can be achieved. The testwork carried out in Australia and Canada give an unambiguous answer to this. Ausenco's testwork establishes that between 60% and 80% of the gold is recoverable to a flotation concentrate. SGS Canada Inc get nearly identical results. Several final tests carried out without the recommended three-stage cleaning did achieve 84 to 85% recovery of gold to a flotation concentrate. The recovery of silver varied from 65 to 80%. It should be pointed out that the proposed flowsheet including a rougher, two scavenger and three cleaner stages was not subjected to locked-cycle tests at a primary grind size of 40 microns. The CIL extraction system will be capable of achieving end recoveries of 94.5% for gold and 91% for silver.

The fact that neither locked-cycle tests were carried out at the conclusion of the batch testwork nor pilot plant testing of the floatability of the Ada Tepe ore was undertaken gives reason to believe that the precious metal recoveries achieved using the combined gravity+flotation flowsheet (proposed as Option 1) are overestimates. We have, in our country, data from laboratory and pilot plant testwork on similar ores – low-sulphidation gold-quartz ores, where the gold is finely intergrown with silica. The low-grade (below economic cutoff) ore in veins 2 and 5 of the Madzharovo deposit are of the same style. Flotation testwork was carried out on them on a laboratory scale at the NIPRORUDA Institute and on a full scale in the Madzharovo Process Plant. Detailed direct cyanidation testwork was carried out at the Copper Institute in Bor in Yugoslavia in 1966. It was established that the variance in the recoveries of gold and silver achieved through direct cyanidation and through flotation exceeded 25%. Similar testwork was carried out on ores from the Petelov deposit, Panagyurishte, and Sedefche deposit, Eastern Rhodopes. The results indicated 20% variance in the recoveries of gold and silver achieved through both processes. Obviously, a similar conclusion would be drawn from the testwork that is currently carried out on gold ores from the area of Breznik in the western part of Bulgaria. The testwork at this stage, however, is still inconclusive. The above-mentioned flotation testwork on ores from Bulgarian deposits has been undertaken under the leadership and with the direct involvement of the author of this statement. This gives us reason to believe that the full scale implementation of the two process alternatives would give at least 20% variance in the recoveries of gold and silver from Ada Tepe ore in favour of the direct cyanidation process. Given the proven resource at 2.4 g/t Au and 1.0 g/t silver and an average throughput of 1 Mtpa, this would mean a loss of 480 kg, or 15,434 oz. of gold at a current gold price of US$1,301/oz. and 260 kg, or 8,360 oz. of silver at a current silver price of US$21.5/oz. It is essential to remember that these metal amounts will be lost with the tailings in the waste management facility. These losses are the lowest possible because the metal headgrades may increase and the anticipated maximum throughput is 1.1 Mtpa. It should also be remembered that the deposit also includes the satellite deposits of Kupel, Kuklitsa, Sinap, Surnak and Skalak, whose exploration and development future is unclear.

The Investor may live with reduced project economic efficiency but our state is the owner of the underground resource and its interests require that the resource is developed using the most efficient methods and processes and achieving the best economic results in full compliance with the environmental provisions. Those should be the requirements of the Krumovgrad community and local authorities.

2.3 When comparing the two ore processing routes, it should be taken into account that a full-scale gravity-flotation plant would cost more to construct and commission than a cyanide leaching plant. In addition to the redundant gravity circuit, the flotation plant would include a three-stage grinding circuit due to the requirement, though not fully justified, to reduce the ore to P_{80} size of 40 microns. The addition of two ball milling stages and three classification stages will affect the floatability of the non-recovered free gold due to the increased residence time in the mills. It is proven that under these conditions the surfaces of the free gold particles are affected by depositions. This process is further facilitated by the presence of elevated levels of silver, copper and iron in free gold. These metals normally deposit onto the surface of free gold, are readily oxidisable and make the gold difficult to flotate.

The inclusion of a redundant gravity stage and a three-stage grinding circuit will increase the operating costs associated with power consumption and labour.

2.4 The quality of the end product will also affect the financial results. The end product obtained through direct cyanidation is dore gold, which may, at low cost, be further treated to obtain pure gold and silver, which can be sold at their trading prices. The end product of a combined gravity+flotation extraction system is gold-silver concentrate. Its marketing incurs additional costs for transportation to the custom smelter, treatment charges as well as for treatment losses though minimal they are.

It is evident from the arguments above that a direct cyanidation/carbon adsorption/electrowinning flowsheet is preferrable from both economic and environmental point of view.

CONCLUSION

1. It is recommended that BMM EAD should accept the direct cyanidation/carbon adsorption/electrowinning flowsheet for detailed plant design and full scale operation.
2. The Ministry of Environment and Waters should conclude a concession agreement with BMM EAD for Khan Krum deposit after the proposed flowsheet is subjected to pilot plant scale testwork.