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Dundee Precious Metals Inc.

Revised NI 43-101 Technical Report

Ada Tepe Deposit, Krumovgrad Project, Bulgaria

By
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And

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And

Simon Meik (QP)
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And

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BAI, C.Eng

For:
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Approved:

Galen White
"signed and sealed"
____
Galen White
Managing Director - UK

Certificate of Qualified Person – Galen White

As a Qualified Person of this Technical Report on the Ada Tepe Deposit of Dundee Precious Metals Inc., Bulgaria, I, Galen White do hereby certify that:

1) I am a Director and Principal Consultant of CSA Global (UK) Ltd, and carried out this assignment for CSA Global (UK) Ltd, 2 Peel House, Barttelot Road, Horsham, West Sussex, RH12 1DE, UK Telephone +44 1403 255 969, e-mail: galen.white@csaglobal.com

2) The Technical Report to which this certificate applies is titled "Revised NI 43-101 Technical Report – Ada Tepe Deposit, Krumovgrad Project, Bulgaria" and is dated 7th November 2017 with an effective date of 21st March 2014.

3) I hold a BSc (Hons) degree in Geology from the University of Portsmouth, England (1996) and am a registered Fellow in good standing of the Australasian Institute of Mining and Metallurgy (FAusIMM Membership Number 226041) and a Fellow of the Geological Society, London (Member Number 1003505). I am familiar with NI 43-101 and, by reason of education, experience in exploration, evaluation and mining of epithermal and vein hosted mineral deposits in Europe and Australia, and professional registration; I fulfil the requirements of a Qualified Person as defined in NI 43-101. My experience includes 17 years in mineral exploration and resource evaluation.

4) I visited the project that is the subject of this Technical Report, between 5th-9th November 2012 for a combined total of 5 days.

5) I am responsible for the following sections of this Technical Report; Sections 1-12, 14, 20.1-20.7, 20.9, 23, 24.1-24.9, 24.12 and 25-27.

6) I am independent of the issuer as described in Section 1.5 of NI 43-101.

7) I have had prior involvement with the property that is the subject of this Technical Report, being a visit to the project in November 2012 for the purposes of reviewing data and validating the Mineral Resource Estimate, as a precursor to the preparation of this Technical Report.

8) I have read NI 43-101 and the parts of the Technical Report I am responsible for have been prepared in compliance with this instrument.

9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 7th day of November 2017.

Galen White
"signed and sealed"
Galen White BSc (Hons), FAusIMM, FGS
Director and Principal Geologist, CSA Global (UK) Ltd.

Certificate of Qualified Person – Julian Bennett

As a Qualified Person of this Technical Report on the Ada Tepe Deposit of Dundee Precious Metals Inc., Bulgaria, I, Julian Bennett do hereby certify that:

1) I am an Independent Mining Consultant to CSA Global (UK) Ltd, and carried out this assignment for CSA Global (UK) Ltd, 2 Peel House, Barttelot Road, Horsham, West Sussex, RH12 1DE, UK Telephone +44 1403 255 969, e-mail: www.csaglobal.com

2) The Technical Report to which this certificate applies is titled "Revised NI 43-101 Technical Report – Ada Tepe Deposit, Krumovgrad Project, Bulgaria" and is dated 7th November 2017 with an effective date of 21st March 2014.

3) I hold a BSc degree in Mining from the Royal School of Mines, London. I am a Fellow of the UK Institute of Materials, Minerals and Mining and am a Chartered Engineer. I am familiar with NI 43-101 and, by reason of education, experience in the evaluation and mining of epithermal and vein hosted mineral deposits in Europe, Africa, Asia, Australia and the Americas, and professional registration; I fulfil the requirements of a Qualified Person as defined in NI 43-101. My work experience includes more than 50 years in the mining industry.

4) I have not visited the project site that is the subject of this Technical Report, however I visited the offices of the company in Sofia, Bulgaria between 24th and 28th February 2014 for a total of 5 days, during which time I held discussions with the Technical Team involved in the project which is the subject of this Technical Report.

5) I am responsible for the following sections of this Technical Report; Sections 15, 16, 18.2-18.12, 21, 22, 24.10.2 and 24.11.

6) I am independent of the issuer as described in Section 1.5 of NI 43-101.

7) I have had no prior involvement with the property that is the subject of this Technical Report.

8) I have read NI 43-101 and the parts of the Technical Report I am responsible for have been prepared in compliance with this instrument.

9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make this report not misleading.

Dated this 7th day of November 2017.

Julian Bennett
"signed and sealed"

Julian Bennett BSc, ARSM, FIMMM, CEng
Mining Consultant to CSA Global (UK) Ltd

Certificate of Qualified Person – Simon Meik

As a Qualified Person of this Technical Report on the Ada Tepe Deposit of Dundee Precious Metals Inc., Bulgaria, I, Simon Meik do hereby certify that:

1) I am an Independent Mineral Processing Consultant, who, as at the effective date of this report was in full time employment with the parent company of Balkan Mineral and Mining EAD, 26 Bacho Kiro, Sofia 1000, Bulgaria.

2) The Technical Report to which this certificate applies is titled "Revised NI 43-101 Technical Report – Ada Tepe Deposit, Krumovgrad Project, Bulgaria" and is dated 7th November 2017 with an effective date of 21st March 2014.

3) I hold a BSc degree and PhD in Minerals Engineering from the University of Birmingham, UK. I am a Chartered Professional Member of the Australasian Institute of Mining and Metallurgy (FAusIMM (CP) Membership Number 106146). I am familiar with NI 43-101 and by reason of my 35 years in the mining industry, work experience since graduation and professional registration, I fulfil the requirements of a Qualified Person as defined in NI 43-101.

4) I have visited the project site that is the subject of this Technical Report at various times between 2004 and 2013, the last visit taking place between 5th and 6th December 2013 for a duration of two days.

5) I am responsible for the following sections of this Technical Report; Sections 13, 17, 19, 24.10.1, 24.10.3 and 24.10.5.

6) I am not independent of the issuer as described in Section 1.5 of NI 43-101.

7) Prior involvement with the property that is the subject of this Technical Report since 2004 has included supervision of all metallurgical testwork, process development and overview of plant engineering designs together with "in-country" coordination of the 2005 EIA and contributions to the preparation of the 2005 and 2011 Definitive Feasibility Studies, and subsequent Technical Reports to 2014.

8) I have read NI 43-101 and the parts of the Technical Report I am responsible for have been prepared in compliance with this instrument.

9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make this report not misleading.

Dated this 7th day of November, 2017.

Simon Meik
"signed and sealed"

Simon Meik BSc (Hons), PhD, FAusIMM(CP)
Independent Metallurgical Consultant

Certificate of Qualified Person – Peter Corrigan

As a Qualified Person of this Technical Report on the Ada Tepe Deposit of Dundee Precious Metals Inc., Bulgaria, I, Peter Corrigan do hereby certify that:

1) I hold the positions of Associate and Ground Engineer Service Manager for Golder Associates (UK) Ltd/Golder Associates Ireland Ltd. I have been the Project Manager for Golder for the Intergrated Mine Waste Facility Phase Design/Detailed Design project and have led the design of the IWMF.

2) The Technical Report to which this certificate applies is titled "Revised NI 43-101 Technical Report – Ada Tepe Deposit, Krumovgrad Project, Bulgaria" and is dated 7th November 2017 with an effective date of 21st March 2014.

3) I hold a BAI (Baccalaureus in Arte Ingeniaria) degree in Civil, Structural and Environmental Engineering from Trinity College Dublin, Ireland. I am a Chartered Engineer with Engineers Ireland (Membership Number 040106). I am familiar with NI 43-101 and by reason of my 15 years in the mining industry, work experience since graduation and professional registration, I fulfil the requirements of a Qualified Person as defined in NI 43-101.

4) I have visited the project site that is the subject of this Technical Report in October 2012.

5) I am responsible for the following sections of this Technical Report; Section 18.1, 20.8 and 24.10.4.

6) I am independent of the issuer as described in Section 1.5 of NI 43-101.

7) I have had prior involvement with the property that is the subject of this Technical Report prior to the commencement of the Technical Design Phase in Septebmer 2012.

8) I have read NI 43-101 and the parts of the Technical Report I am responsible for have been prepared in compliance with this instrument.

9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make this report not misleading.

Dated this 7th day of November, 2017.

Peter Corrigan
"signed and sealed"

Peter Corrigan – Associate and Ground Engineering Service Manager
Golder Associates (UK) Ltd/Golder Associates Ireland Ltd.

Contents

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Glossary

USD	United States of America dollars
µm	Micrometre, or 0.000001m
%	Percent
2D	Two-dimensional model or data
3D	Three-dimensional model or data
AAS	Atomic Absorption Spectrometry
AIRCORE	Air core/kit bit destruction drilling
Ag	Silver grade measured in parts per million
Au	Gold grade measured in parts per million
BMM	Balkan Mineral & Mining
CC	Correlation Coefficient
NI 43-101	Canadian National Instrument 43-101
CIL	Carbon-in-leach
cm	centimetre
CV	Coefficient of Variation. In statistics, the normalized variation value in a sample population
DDH	Diamond Drilling Hole
DTM	Digital Terrain Model. Three-dimensional wireframe surface model, for example, topography
E (X)	Easting. Coordinate axis (X) for meter based Projection, typically UTM. Refers specifically to meters east of a reference point (0,0)
EV	Expected Value
FA	Fire assay
g	gram
g/m3	grams per cubic metre
g/t	grams per tonne
HARD	Half the Absolute Relative Difference - used for comparing pairs
HQ2	Size of diamond drill rod/bit/core
hr	hours
HRD	Half Relative Difference - used for comparing pairs
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
IRR	Internal rate of return
ISO	International Standards Organisation
JORC	Joint Ore Reserves Committee (The AusIMM)
kg	Kilogram
kg/t	kilogram per tonne
km	kilometre
km2	square kilometre
koz	kilo-ounce
kW	kilowatts

ktonnes	kilotonne
kWhr/t	kilowatt hours per tonne
L/hr/m2	litres per hour per square metre
M	Million
m	metre
M2	square metre
mE	metres East
Ma	Million years
MIK	Multiple Indicator Kriging
Ml	Millilitre
mm	millimetre
mN	metres North
Moz	Million ounces
Mtpa	Million tonnes per annum
mRL	metres Relative Level
N (Y)	Northing. Coordinate axis (Y) for meter based Projection, typically UTM. Refers specifically to meters north of a reference point (0,0)
NPV	Net present value or net present worth (NPW)
NQ2	size of diamond drill rod/bit/core
°C	Celsius degrees
OK	Ordinary Kriging
oz	Troy ounce (31.1034768 grams)
P80 -75 pm	Measure of pulverisation. 80% passing 75 microns
ppb	parts per billion
ppm	parts per million
Psi	Pounds per square inch
PVC	Polyvinyl chloride
QA/QC	Quality Assurance Quality Control
Q-Q	Quantile-quantile
RC	Reverse circulation drilling method
RL (Z)	Reduced level. Elevation of the collar of a drill hole, a trench or a pit bench above the sea level
ROM	Run of Mine
RQD	Rock quality designation - measure of rock competency
RSG	RSG Global
SD	standard deviation
SG	Specific gravity
SGS	Societe Generale de Surveillance International laboratory group
SMU	Selective mining unit
t	tonnes
t/m3	tonnes per cubic metre
tpa	tonnes per annum
TM	Trademark
w:o	waste to ore ratio

1 Executive Summary

1.1 Introduction

At the request of Dundee Precious Metals Krumovgrad ("DPMKr"), a 100% owned subsidiary of Dundee Precious Metals Inc. ("DPM") CSA Global (UK) Ltd ("CSA") was requested to compile a Technical Report on the Krumovgrad Gold Project, the material asset being the Ada Tepe Deposit located in south-eastern Bulgaria. This report is to comply with disclosure and reporting requirements set forth in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101"), Companion Policy 43-101CP, and Form 43-101F1.

This Technical Report discloses material changes to some components of the Definative Feasibility Study ("DFS") and is an update to an earlier Technical Report dated January 11 2012, which summarised the DFS completed in November 2011.

This updated report pertains to the main deposit that comprises the Krumovgrad Project, namely the Ada Tepe deposit.

This report is an updated (revised) Technical Report which, at the request of the Ontario Securities Commission ("OSC") contains additional disclosure in the following areas;

1. In accordance with NI 43-101 S 2.2 (b), a statement that disclosure of Mineral Resources are inclusive of Mineral Reserves (see Section 14).

2. In accordance with Item 22 (b) of Form 43-101F1, inclusion of annual cash flow forecasts as at the effective date of this report (see Section 22).

1.2 Project Description and Location

1.2.1 Summary

The Krumovgrad Project is a planned 850,000 tonnes per year ("tpa") open pit gold mine located in Bulgaria which is consistent with existing permitting and environmental submissions and is financially viable. The mill facilities and mine will be developed, constructed, and operated by DPMKr, a wholly owned subsidiary of DPM. The size of the project footprint has been minimised as much as possible and the footprint and its anticipated buffer will be approximately 85 ha.

1.2.2 Mineral Rights and Tenement Description

The then Balkan Mineral & Mining EAD ("BMM", now "DPMKr"), a 100% subsidiary of DPM, was awarded the Krumovgrad Licence area (130 km2) on the June 12, 2000 in accordance with the Agreement of Prospecting and Exploration reached with the Bulgarian Ministry of Economy.

The Mining Licence ("Krumovgrad Concession") covers an area of 1370 hectares (13.7 sq.km) and includes the area of the Krumovgrad mining operation and the immediate surrounds. DPMKr has 100% ownership of the Project, which operates under a concession agreement that was granted by the Republic of Bulgaria in 2012 for a period of 30 years. Under Bulgarian regulations, the Mining Licence area is applied for on the basis of geographical coordinates.

In June 11, 2005 an Additional Agreement resulting from a term extension enforced a reduction of the Krumovgrad prospect area from 113 km2 to 100 km2 and an extension of the term by another two years, i.e. until June 13, 2007.

In March 2013, the Krumovgrad Municipal Council (KMC) again considered the Company's application and a majority of the Council voted against the approval of the Terms of Reference and the preparation of the DDP. The decision has been formally presented to the Governor of the Region, who will either: (i) confirm the decision of the KMC; (ii) deny the decision of the KMC and return the matter to the council for further consideration; or (iii) appeal the KMC's decision to the Court. A decision is expected by the Chairman before April 22, 2014.

The Company expects that the KMC will proceed in accordance with the direction of the Administrative Court. The Company currently expects that the appeal process being exhausted, the KMC will issue a favourable decision on the Detailed Development Plan ("DDP"), terms of reference, which in turn is expected to facilitate securing the remaining local permits.

For further detail regarding the Tenement Description, Permitting history for the Krumovgrad licence, please refer to Chapters 4.3 and 4.4.

1.2.3 Environmental Liabilities

DPM is not aware, nor has it been made aware, of any other significant environmental liability associated with the Krumovgrad Project

1.2.4 Royalties

The Company will pay a royalty to the Bulgarian government, at a variable royalty rate applied to the gross value of the gold and silver metals combined in the ore mined. The royalty rate depends on the profitability of the operation. At a pre-tax profit to sales ratio of 10% or less, the royalty rate will be 1.44% of the value of the metals. At a pre-tax profit to sales ratio of 50% or more, the royalty rate will be 4% of the value of the metals. At intermediate levels of profitability, the royalty rate will vary on a sliding scale between 1.44% and 4% in a linear fashion. At a gold price of $1,250/oz and a silver price of $23/oz, the royalty rate will be in the order of 2. 5% of the gross value of gold and silver metals contained in the ore produced from the mine.

1.3 Accessibility, Local Resources and Infrastructure

The licence area is located in the East Rhodope, approximately 320 km (by road) southeast of Sofia, in the Kardjali District, 3km south of the regional township of Krumovgrad (25° 39' 15"E and 41° 26' 15"N). Access to the general area is available at all times of the year, by sealed roads to Krumovgrad. Access within the licence area is good, with all-weather surface roads transecting the project area. Secondary roads are unsurfaced but generally accessible year round with four-wheel drive vehicles.

Krumovgrad is located approximately 320 km by paved road southeast of Sofia, which is serviced by a modern International airport. A second International Airport is located in the city of Plovdiv located approximately 106 km northwest of Krumovgrad.

The project area belongs to the Continental-Mediterranean climatic type, the main feature of which is its proximity to the climate of Subtropical Europe, featuring markedly higher winter and substantially lower summer precipitation. Winters are mild, but during intensive cold spells temperatures may fall to -13°C. Summers are hot, reaching 36°C in warmer spells and exceeding 40°C in some locations.

The average annual precipitation is 703.5 mm. The bulk of this falls in autumn and winter, occasionally as snow in the coldest months. The highest rainfall occurs in December (96.9 mm average) and the lowest in August (24.1 mm). Estimated 1:100 year rainfall events are 117.3 mm for 24 hours duration, and 184.1 mm for 72 hours. Probable Maximum Precipitation ("PMP") estimates are up to 383.4 mm for 24 hours and 605.4 mm for 72 hours. Average annual evaporation is 1050.8 mm, similar overall to annual rainfall in magnitude, but opposite in seasonal sense.

Small villages are dispersed widely throughout the licence area involved in subsistence farming, particularly livestock and the growing of tobacco on the poorly developed soils characteristic of the region. The other main land use within the licence area is state controlled forestry. The population of Bulgaria is largely non-practicing Eastern Orthodox Christian (85%) with a Turkish Muslim minority predominantly residing in the southeast of the country, including the licence area.

Infrastructure in the area is good, with paved roads, power and water resources available within close proximity to the Project.

The Krumovgrad District is around 230 m above mean sea level and is characterised by a rugged landscape. The Ada Tepe deposit is located in an area of moderate, hilly topography abutting a major regional river system. The project area is readily accessible at all times of the year.

1.4 History

The Ada Tepe prospect was the subject of only very brief attention in previous State funded exploration in the early to mid-1990s, by GeoEngineering of Assenovgrad, and Geology & Geophysics of Sofia

On June 12, 2000, BMM (a 100% subsidiary of Dundee) was awarded the Krumovgrad Licence area (113 km2) in accordance with the Agreement of Prospecting and Exploration reached with the Ministry of Economy. Work by BMM included diamond and RC drilling, rock chip sampling, bulk density measurements, and detailed geological mapping.

1.5 Geological Setting

The Krumovgrad region is located within the East Rhodope which comprises the eastern portion of a large metamorphic complex termed the Rhodope Massif. The massif underwent Upper Cretaceous extension leading to uplift and formation of the Kessebir metamorphic core complex. This event was accompanied by low-angle detachment faulting, by graben development, and formation of sedimentary basins. The basins to the north of the Kessebir core complex contain Palaeocene terrestrial sediments that are transitional upwards into marine sediments.

The Ada Tepe deposit is approximately 600 m long (N-S), and up to 350 m wide (E-W). The wall zone is up to 30 m thick. The thickness of the Upper Zone vein mineralization is very variable, from less than 1 m thick, to more than 30 m thick. The Wall Zone exhibits very good continuity. The Upper Zone vein system exhibits less continuity than the Wall Zone, necessitating the higher drilling density that has been applied during the delineation of the Ada Tepe deposit.

1.6 Mineralisation and Deposit Types

Gold and silver mineralisation in the Krumovgrad Licence area is predominantly hosted within the Sharovo Formation proximal to the unconformable fault contact or 'detachment' with the underlying basement rocks of the Kessebir core complex. Sediments within the Sharovo Formation typically form laterally discontinuous lenses ranging from coarse breccia to fine sands with variable clay content. Upward variations in the stratigraphy of the Krumovgrad Group reflect progression from a high-energy environment, breccia-conglomerates and coarse sandstones through to the lower energy siltstones and limestones characteristic of increasing basin maturity.

Review by RSG Global in 2004 suggests that the second-order structural control for mineralisation, after that of the detachment, is the proximity of NE-SW transfer faults. These structures dip steeply, allowing more direct access of fluids from deeper levels than the shallow dipping extensional structures. As such, the shallow extensional structures may represent trap sites or structures that accommodate local lateral fluid flow away from the transfer structures. Closely spaced transfer structures may also be important for localizing mineralisation as the presence of shear couples can enhance brecciation of the intervening rock, or may act to produce tensional sites. For example, dextral shear on the NW-SE striking faults bounding Ada Tepe may have been responsible for facilitating epithermal vein emplacement within E-W tensional sites.

1.7 Exploration

Since the mid-1990s, previous operators have undertaken:

- Geological mapping, trenching and drilling over the nearby Surnak prospect
- Minor trenching on the Skalak and Kuklitsa prospects
- Regional soil sampling programme at an average sample grid 250 m x 50 m across the SE Rhodopes
- Magnetic and IP surveys.

Since June 2000, BMM has conducted detailed exploration of the Ada Tepe prospect, including:

- Establishing accurate survey and topography control
- Detailed surface trenching and channel sampling and geological mapping.

1.8 Drilling and Surveying

Sub-surface mineral resource delineation at the Ada Tepe deposit has been undertaken by a combination of RC and diamond drilling, completed in four drilling programmes between late 2000 and late 2004.

From June 2000 until March 2002, all exploration data collection at Ada Tepe was undertaken by BMM, under the upper management of Navan. From April 2002 to the end of 2004, exploration at Ada Tepe was undertaken under the management of RSG Global in close consultation with BMM field staff, Navan upper management until 30 September 2003, and subsequently DPM upper management.

Trenches and drill access road cut exposures were routinely channel sampled since the commencement of detailed exploration at Ada Tepe in mid-2000. The channel sampling was undertaken predominantly on N-S orientated traverses coinciding with the 25 m spaced drill traverses. Prior to March 2002, a variety of sample intervals were used, primarily controlled by changes in geology.

RC samples were routinely collected at 1 m intervals and the cuttings split with a Jones riffle splitter. The bags of cuttings were routinely weighed prior to taking the sub-sample via the Jones riffle splitter.

The diamond drilling and core sampling procedures employed for mineral resource delineation at the Ada Tepe deposit are summarised in Section 0. Similar to the RC drilling, the diamond drilling at Ada Tepe has been performed using strict quality control procedures.

1.9 Sampling and Analysis

Sample preparation and assay of analysis samples from the Ada Tepe Deposit has been carried out at two principal independent internationally accredited laboratory firms:

- OMAC Laboratories Ltd ("OMAC") of Ireland (2000 - 2001).
- SGS Laboratories of Perth (Welshpool), Western Australia, Gura Rosiei (near the Rosia Montana mine site), Romania and Chelopech, (part of Chelopech Mine) Bulgaria (2002 - 2004).

In addition, umpire assay analyses of approximately 5% of the routine exploration samples from the second and third exploration programmes were performed by two internationally accredited laboratories.

The following routine procedures were used to prepare the trench, RC drilling, and core samples for analysis:

- Dry samples at 105°C.
- Jaw crush core and trench samples to minus 6 mm.
- Pulverize all samples in a LM5 crusher to 95% passing 75 µm. Complete sieve analysis on 1 in 20 samples.
- Clean bowl and puck of the LM5 with compressed air after each sample, and with a barren flush after every 20th sample, or as required to remove residue build-up.
- Complete barren flushes after DPMKr specified samples anticipated to contain high grade mineralisation.

Additional quality control employed by DPMKr for external monitoring of the precision and accuracy of the assay analyses completed during exploration programmes 2 and 3 include:

- Submission of internationally accredited gold and silver standards, produced by Rocklabs of New Zealand, routinely inserted into the sample stream at a frequency of 1 in 20 routine exploration samples.
- Insertion of a blank sample (beach sand) before the first sample of each drillhole from March 2004 onwards.

- Routine collection of duplicate RC drill sample splits at a frequency of 1 in 20 routine samples.

- Routine collection of duplicate channel samples at a frequency of 1 in 20, approximately 20 cm above the primary channel sample location.

- Collection of a duplicate sample split after jaw crushing of trench and core samples at a frequency of 1 in 20.

- A total of 66,427 successful gold analyses and 62,067 silver analyses have been completed on routine exploration samples from Ada Tepe (excluding any QA/QC samples).

- Most sample analyses have been completed at the SGS Gura Rosiei lab (gold - 61%, silver - 60%), followed by SGS Welshpool (gold - 32%, silver - 34%), followed by OMAC (gold - 5%, Silver - 2%), followed by SGS Chelopech (gold - 3%, Silver - 3%).

1.10 Data Verification

CSA completed the following data verification;

- During the 2012 site visit, CSA reviewed the following;

    - Logging information contained in the database for the project, and verified this information (exported as graphic drill logs from acQuire software) against several drill holes inspected at the time. In addition;

    - The contents of the 2005 and 2012 Technical Reports, specifically those sections that detail the methodology employed to complete the Mineral Resource Estimate, the input data, parameters and assumptions used and to check that these are robust and honour both the underlying data characteristics and the geological model for the project;

    - Loading and validating the Drillhole and Trench Database, block model, 2D wireframe surfaces and 3D geological and mineralised domain wireframes;

    - Tying these reviews to an on-site inspection of project geology, drill core, data collection protocols and communication with DPM project staff.

- CSA used the software programme QAQCReporter ("QAQCR") to produce QA/QC reports in order to review the accuracy and precision of the assayed QA/QC material and samples.

- CSA completed checks on standards and blanks submitted to SGS Gura Rosiei, SGS Welshpool and SGS Chelopech, via the preparation of validation control plots.

- Internal laboratory blanks and standards from SGS Gura Rosiei and SGS Chelopech were not available for CSA review. Internal standards for SGS Welshpool and those submitted by DPMKr were reviewed.

- CSA reviewed the results of the check analyses (duplicates, repeats and pulp splits) detailed below and duplicate scatter plots were created.

- CSA reviewed the results of the inter-laboratory assaying performed at Genalysis Perth and ALS Vancouver and whilst the above tables could not be replicated exactly, CSA considers that, on the basis of checks completed, the inter-laboratory precision and relative accuracy of the primary and umpire gold and silver datasets to be acceptable for the purpose of mineral resource estimation.

- CSA considers the drillhole collars, trench and channel sample locations at Ada Tepe to be accurately located in three dimensions for the purposes of mineral resource estimation.

- CSA has taken receipt of, and reviewed this topographic surface and the trench/collar points used in its construction and believes it to be valid for use in constraining the Mineral Resource block model.

- Review of the results of verification twin drilling.

1.11 Mineral Processing and Metallurgical Testing

Three distinct phases of testing have been undertaken in the evaluation of the mineralization present in the deposit at Ade Tepe. In summary, these contributions were:

- 2005 DFS – The basis of the program was to develop an industry standard gold extraction process, The AMMTEC Laboratory, Perth, Australia, was selected to conduct the physical characterisation, comminution, leaching and cyanide detoxification testwork programmes. Samples were also sent to; MinnovEX Technologies (now SGS) in Toronto, Canada for Comminution characterisation and variability testing; for mineralogical examination; Outokumpu Technologies for thickening, and Larox Pty Ltd for filtration; and to Coffey for tailings characterisation. Results from this program confirmed that all the samples tested were considered “free-milling” and amenable to gold production by conventional cyanidation processes, combined with appropriate cyanide destruction to levels well below European and World standards at the time.

- The 2012 mining study essentially reinvented the project following the rejection of the original investment proposal by the local community and government authorities. At the expense of a reduction (8 - 10%) in recovery compared with the original and conventional cyanide leach circuit, the project was ‘re-engineered’ using a more conventional flotation process, combined with the introduction of a combined mine waste and flotation tailings facility (Integrated Mine Waste Facility – “IMWF”). The process evolved from the 2005 flotation scoping testing which demonstrated that at the CIL circuit grind (P80 of 75 µm) between 60% and 80% of the gold could be recovered to a flotation concentrate. The 2009/11 testing program was developed to confirm the potential of both a) physical recovery processes (flotation and gravity) as the primary method of precious metal concentration, and b) the ultimate integration of high-density (or "paste") settled tailings from the process into an overall waste deposition strategy which incorporates the mine waste.

In 2012/2013, the bulk of the testing program was confirmatory, mainly the dispatch of appropriate samples to recognised testing institutions for mechanical design tests, including - materials handling flow characteristics, slurry rheology determinations, additional confirmatory settling and paste thickening testing. Several confirmatory flotation programs continued at SGS.

1.12 Mineral Resource Estimates

The Ada Tepe deposit was the subject of only very brief attention by previous State funded exploration in the early to mid-1990s carried out by GeoEngineering of Assenovgrad, and Geology & Geophysics of Sofia. Since June 2000, DPM has conducted detailed exploration of the Ada Tepe prospect including the establishment of more accurate survey control over the licence area, surveying of the surface topography, detailed surface trenching and channel sampling, diamond and reverse circulation ("RC") drilling, rock chip sampling, bulk density measurements, and detailed geological mapping. To date, the drilling forms a notional 25 mN by 25 mE grid over the entire deposit, while closer spaced drilling on a notional 12.5 mN by 12.5 mE grid has been completed over two rectangular sub-regions of the deposit.

The Ada Tepe deposit is a low-sulphidation adularia-sericite gold-silver epithermal deposit formed during the Neogene within the Southern Rhodope tectonic zone and located within Palaeocene sedimentary rocks overlying the northeastern end of the Kessebir core complex. Two major styles of mineralisation are apparent at Ada Tepe:

- Initial stage of mineralisation hosted by a massive, shallow-dipping (15 degrees north) siliceous body forming the hangingwall to the detachment and defining the contact between the core complex and overlying sedimentary rocks. This mineralisation is termed the "Wall Zone" by local geologists and displays multiple stages of veining and brecciation.

- Second phase of mineralisation represented by steep dipping veins that exhibit textures indicative of formation within an epithermal environment. These veins have a predominant east-west strike, cross-cut the shallow-dipping siliceous Wall Zone mineralisation and extend upwards into the sedimentary breccia unit above the Wall Zone. This mineralisation is referred to as the "Upper Zone" by local geologists.

Based on observations of the geology during the site visit and using all of the available geological and grade information, suitable lithology, oxidation and mineralised domain boundaries were interpreted and wireframe modelled to constrain the mineral resource estimation for the Ada Tepe deposit.

Interpretation and digitising of all constraining boundaries was undertaken on north-south orientated cross sections coinciding with the drill traverses. The resultant digitised boundaries were used to construct wireframe surfaces or solids defining the 3-D geometry of each interpreted feature.

Data from surface channel sampling, and surface RC and diamond drilling have been applied to mineral resource estimation. From June 2000 until March 2002, all exploration data collection at Ada Tepe was undertaken by BMM, under the upper management of the previous owners, Navan Mining PLC (“Navan”). From April 2002 to 2004, exploration at Ada Tepe was undertaken under the management of RSG Global (“RSG”) in close consultation with BMM field staff, Navan upper management until 30 September 2003, and subsequently Dundee upper management as summarised in the table below. There was no reported exploration work carried out between 2005 and 2012.

Mineralised domain boundaries for the purpose of constraining mineral resource estimation were interpreted and modelled based on the geological logging, surface mapping and interpreted geological and structural controls. In addition to these geological constraints, a notional 0.2 g/t Au lower cutoff grade was also applied to demarcate anomalous mineralisation where appropriate.

The mineral resource model is based on detailed statistical and geostatistical investigations generated using 3 m composite data subdivided by the geological interpretation. A sub-blocked block model was constructed using 12.5 mE x 12.5 mN x 5 mRL parent cell dimensions and sub-blocking down to minimum 2.5 m cubic dimensions along the modelled wireframe surfaces representing the geological interpretation and surface topography.

The principal method used to estimate mineral resource gold grades for the 'Wall Zone' was Ordinary Kriging (“OK”). Multiple Indicator Kriging (“MIK”) was used to produce a selective mining unit (“SMU”) mineral resource estimate for gold in the 'Upper Zone' domain. Estimation of silver grades in the mineral resource block model has been undertaken by linear regression from the block model gold estimates. Detailed visual and statistical review of the mineral resource was completed as part of routine validation by RSG, and the mineral resource was considered globally robust.

As part of review work undertaken by CSA during a visit to the project in November 2012, augmented by recent data reviews completed in February and March 2014, the estimate of Mineral Resources completed by RSG has been investigated and several validations and checks completed to confirm the validity and reliability of the Mineral Resource Estimate.

CSA completed the following validation checks on the RSG Mineral Resource block model;

- Swath Plots depicting model tonnes, input declustered composite gold grade, output block model gold grade and drill metres per slice, for the Wall Zone.

- Regularisation of blocks within the RSG block model file so that this block model could be used for mine planning work completed by DPM in 2013. A block model report was then generated at a variety of cut-offs and compared to the RSG model as a check. During this regularisation work, the underlying grade interpolations completed by RSG were not changed. The regularisation process simply assigned block proportions (from the RSG model) of tonnage and grade by majority class, material type and oxidation, to the regularised blocks. The resulting model is referred to as the "CSA block model".

- A check estimate of the CSA block model using Uniform Conditioning, completed for Domains 10, 11 and 12 within the Upper Zone. These domains are considered representative of the grade distribution within the Upper Zone and together account for 27% of contained metal within the Upper Zone.

- On-screen visual comparisons of the RSG block model grades (via MIK) and the CSA Uniform Conditioning check estimate block model grades, for domains 10, 11 and 12.

During discussions between CSA and DPM in November 2012, DPM commented that the RSG block model, as presented, was not suitable from mine planning work being undertaken at that time and planned for 2013. Specifically;

- The block model presented by RSG contains tonnage and grade above cut-off proportion fields for each block of the sub-blocked model. DPM mine planning software (GEMS) requires a regularised block model as the input file, with a single code for material type, oxidation and class. As such, difficulties were encountered when trying to report from the block model, tonnages and grade above cut-off and by material type, class or oxidation.

- Accordingly, CSA considered the approach to regularisation of blocks, taking in to account the following;

    - The regularised model would contain additional proportion and accumulation fields to account for relative block and attribute proportions across boundaries (geological, oxidation and mineralised domain).

    - Care would need to be given to the resulting tonnage, grade and metal proportions across the Wall Zone and Upper Zone boundary since these two grade domains have been estimated using different methods (MIK and OK).

    - The inclusion of all grade thresholds, regularised block proportion information of overburden, Wall Zone, and Upper Zone domains and the three oxidation domains would result in an unwieldy block model with an excessive number of fields to account for the sub-block accumulation function, and this might have an effect on the resultant reported tonnage, grade and metal above each of the cut-offs, and for each material type or class.

    - The accumulation of block proportions to the regularised blocks showed that it was impossible to replicate exact figures of tonnage and grade above cut-off when reporting by class, oxidation or material type since there is no way of knowing how much of the grade above cut-off is contained in the block proportion of class, oxidation or lithology as independent variables. This becomes critical at the boundaries of material type and class over which a regularised block might sit.

- CSA believes a better option to address this "edge effect" is to assign a "block majority" field to the model. The block majority simply assigns the dominant class, oxidation, domain and materially type code to each regularised block.

CSA considers the regularised model to be reliable and valid for use in mine planning work and that the model adequately honours the RSG reported global resource at each cut-off. The CSA regularised model (mikclsmod.csv) was presented to DPM and used in mine planning work completed in 2013-14 and for conversion to Mineral Reserves.

Uniform Conditioning ("UC") is a recoverable estimation method. Like MIK, it estimates tonnes and grade above cut-offs for a given SMU size. It's advantageous where drill spacing is widely enough spaced that grades have to be estimated into blocks larger than SMU sizes, but where a recoverable estimation is required or might be desirable. It has advantages over Ordinary Kriging ("OK") in situations like these because reporting above cut-offs using an OK estimate is unreliable if blocks are too small relative to the grid spacing. OK grades tend to be increasingly smoothed when blocks are larger. Metal content is therefore lower for OK at most grade cut-offs.

UC differs from MIK in that it uses the SMU dimensions, the variogram model (in this case the RSG variogram models) and the in-situ grade to estimate the recoverable resource. The change of support correction required for MIK (from global to recoverable resource) is a mathematical correction which can be arbitrary and can often rely on empirical correction factors. MIK can be favoured for some types of deposits, but UC is often considered advantageous over MIK for the following reasons:

- It is quick to implement as there is no requirement for the generation of many variograms above cut-offs, which is a requirement of MIK. As one gets to higher cut-offs, often the data support drops off considerably, making variograms for those cut-offs less reliable.

- It includes a more robust change of support. It relies on the variogram of the underlying data to inform the change of support (from large blocks to small SMUs) rather than the more arbitrary change of support methods used in MIK which often have to be modified using empirical factors.

- A UC model can be localised to SMU sized blocks using a process called Localised Uniform Conditioning ("LUC"). This moves away from the fairly unwieldy format of MIK models where mine planning can become difficult when faced with a block model file that contains fields of proportions and grades above cut-off. A LUC model has a single grade per SMU, while perfectly representing the Grade Tonnage Curve of each larger panel block.

- Often in precious metal deposits there are areas of higher and lower grade, between which the grade profile is not a sharp contact but a gradational one. In these instances, and Ada Tepe is an example, UC is preferred to MIK. MIK is based on the assumption that grade classes are independent of each other (mosaic). UC relies on the assumption that grades are gradational (diffusive) i.e. generally, to move from high grades to low grades, one must move through intermediate grades.

The process by which the UC estimate was completed in summarised below.

- Flagging the raw assays with domain wireframes.
- Compositing flagged raw assays to 1 m. A preliminary review on the impact of composite lengths was completed by CSA. The results informed the decision taken by CSA to composite to 1 m for the check estimate. This differed from the 3 m (including residuals >1 m) used by RSG. The reasons for this were based on 1 m being the dominant sampling interval. In addition, in a review of the residuals formed by 1 m compositing, the statistics showed that no bias was being introduced by using residuals and 1 m composites. This was considered preferable over the removal of <1 m residuals from 3 m composites, as in the RSG estimate. A review of residuals between 1 and 3 m show grades of approximately 25% less (globally) than those seen in 3 m (and 1 m) composites. CSA believes that through the use of 1 m composites, grade bias is reduced.
- Top-cuts were used to control the influence of high grade outliers. Top cuts of 100 g/t Au and 46 g/t Ag were imposed during Ordinary Kriging.
- Variography was completed on uncut 1 m composites that had been de-clustered using cell de-clustering. Rotations reflected those seen in the RSG models. The nugget was high for Au which also reflects the results seen in the RSG model. Approximately 85% of the Au variability was modelled to be within the first 15 m. The nugget for Ag was approximately 20%, with approximately 60% of variability within the first 15 m.
- Ordinary Kriging was used to estimate in-situ grades of parent blocks (12.5 m x 12.5 m x 5 m), while Uniform Conditioning was used to estimate recoverable grades and tonnes of each parent block based on an SMU size of 6.25 m x 6.25 m x 2.5 m as per the RSG model.
- Soft boundaries were used between domains to minimise potential high-grading. This is in contrast to the method used by RSG which instead used an envelope around the wireframes of domains 11 and 12.
- Validation comprised reviewing the block models spatially plan view and cross sections.
- The Kriged results prior to the UC estimation were reviewed against input composites to ensure that the global estimate was representative of the underlying data and swath plots were generated so that results could be compared spatially.
- CSA believes that the results of the check estimate adequately supports the results of the portion of the model estimated using MIK. The higher grades seen at the lower grade cut-offs (including the reporting cut-off) may reflect the inclusion of lower grade residual data (1 to 3 m) in the RSG estimate. Residuals included in the check estimate reflected the

same grades as 1m composites and raw data. However, in the context of the overall resource, and classification, these differences are not considered material.

The results of the UC estimate is summarised in Table 1.

Table 1. Comparison Table of the CSA UC Check Estimate against the CSA Block Model

CUTOFF	Tonnes_MIK	Au g/t_MIK	Ag g/t_MIK	KOz Au_MIK	AgKoz_MIK
0.5	1,547,738	2.78	1.60	138.40	79.79
0.6	**1,354,148**	**3.10**	**1.73**	**134.99**	**75.38**
0.7	1,204,207	3.41	1.85	131.89	71.78
0.8	1,085,762	3.70	1.97	129.06	68.80
0.9	987,029	3.98	2.09	126.39	66.19
1	903,120	4.27	2.20	123.85	63.87
CUTOFF	**Tonnes_UC**	**Au g/t_UC**	**Ag g/t_UC**	**KOz Au_UC**	**AgKoz_UC**
0.5	1,465,568	3.05	1.71	143.85	80.37
0.6	**1,302,099**	**3.37**	**1.82**	**140.98**	**76.22**
0.7	1,172,885	3.67	1.93	138.28	72.72
0.8	1,068,738	3.95	2.03	135.78	69.73
0.9	981,309	4.23	2.13	133.40	67.09
1	907,653	4.49	2.22	131.15	64.75
CUTOFF	**Tonnes_UC**	**Au g/t_UC**	**Ag g/t_UC**	**KOz Au_UC**	**AgKoz_UC**
0.5	-5%	10%	6%	4%	1%
0.6	**-4%**	**9%**	**5%**	**4%**	**1%**
0.7	-3%	8%	4%	5%	1%
0.8	-2%	7%	3%	5%	1%
0.9	-1%	6%	2%	6%	1%
1	1%	5%	1%	6%	1%

The UC check estimate compares favourably to the CSA block model interpolated by RSG using MIK. Additionally, CSA makes the following comment relating to the check estimate;

- The check estimate reports 4% less tonnes at a 9% higher gold grade, resulting in a 4% increase in contained gold within the check domains, at a 0.6 g/t Au cut-off (the cut-off used to report Mineral Resources and Mineral Reserves in the Upper Zone).

- The check estimate reports 4% less tonnes at a 5% higher silver grade, resulting in a 1% increase in contained silver within the check domains, at a 0.6 g/t Au cut-off.

- UC has validated the MIK results suggests the variogram models and the linear regression of silver to be valid.

Table 2. CIM Mineral Resource Statement – Ada Tepe Deposit – Upper Zone and Overburden (CSA, 2013)

Dundee Precious Metals - Krumovgrad Ada Tepe Mineral Resource Estimate as at 31st December, 2013 Upper Zone and Overburden reported at a 0.6 g/t Au cut-off					
Resource Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Metal Content	
				Au (Moz)	Ag (Moz)
Measured	1.1	3.46	1.91	0.125	0.069
Indicated	3.9	2.86	1.70	0.357	0.212
Total M+I	5.0	2.99	1.75	0.482	0.281
Inferred	0.3	1.31	1.06	0.013	0.011

The MRE has estimates gold in to mineral domains via MIK in to blocks with dimensions 12.5 m x 12.5 m x 5 m. Silver estimation is via linear regression from gold values.

MRE is reported using a gold cut-off of 0.6g/t for classified resources in the Upper Zone and Overburden.

Measured and Indicated Mineral Resources are inclusive of Proven and Probable Mineral Reserves.

Tonnages are rounded to the nearest 0.1 million tonnes to reflect this as an estimate.

Metal content is rounded to the nearest 1000ozs to reflect this as an estimate.

Table 3. CIM Mineral Resource Statement – Ada Tepe Deposit – Wall Zone (CSA, 2013)

Dundee Precious Metals - Krumovgrad Ada Tepe Mineral Resource Estimate as at 31st December, 2013 Wall Domain reported at a 0.8 g/t Au cut-off					
Resource Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Metal Content	
				Au (Moz)	Ag (Moz)
Measured	1.7	6.32	3.27	0.353	0.183
Indicated	0.2	4.28	2.38	0.024	0.014
Total M+I	1.9	6.13	3.19	0.377	0.196
Inferred	0.0	0.87	0.88	0.000	0.000

The MRE has estimates gold in to mineral domains via OK in to blocks with dimensions 12.5 m x 12.5 m x 5 m. Silver estimation is via linear regression from gold values.

MRE is reported using a gold cut-off of 0.8g/t for classified resources in the Wall Zone.

Measured and Indicated Mineral Resources are inclusive of Proven and Probable Mineral Reserves.

Tonnages are rounded to the nearest 0.1 million tonnes to reflect this as an estimate.

Metal content is rounded to the nearest 1000ozs to reflect this as an estimate.

Based on the mineral resource estimate, Measured, Indicated and Inferred Mineral Resources have been defined in accordance with the criteria set out in NI 43-101. The criteria used to categorise the mineral resources include the robustness of the input data, the confidence in the geological interpretation, the distance from data and geostatistical service variables such as estimation variance.

CSA believes that the Mineral Resource Estimate classification approach adopted by RSG to be valid. CSA has updated the tabulation of Mineral Resources, following review work completed in 2012 and 2014 but has not materially or significantly changed the estimate from that reported by RSG. To assist with mine planning works CSA took receipt of the Mineral Resource block model and regularised blocks in the model such that reporting from the model could be undertaken by material type (something difficult to achieve using the proportioned block model created by RSG). The underlying grade interpolations completed by RSG have not been modified in any way, the only difference being how the tonnages and grade from the model are reported.

CSA believes the estimate of Mineral Resources would not be materially affected by changes to metallurgical, environmental, permitting, legal, title, socio-economic, marketing or political circumstances. CSA believes the risks regarding permitting and socio-economic factors to be low. CSA has relied on information provided by DPM as regards legal and environmental risks (See Section 3).

1.13 Mining Operations

1.13.1 Mineral Reserves

The Mineral Reserves for the Krumovgrad Project have been estimated assuming a conventional open cut, drill, blast, load and haul operation. Rock will be drilled and blasted, loaded by hydraulic excavators into diesel haul trucks, similar to many small tonnage open pit gold mining operations throughout the world. The mining equipment will be owner operated and will be maintained under contracts with the equipment suppliers.

The planning process consisted of a pit optimisation followed by an open pit design, leading to a life of mine ("LOM") production schedule and capital and operating cost estimations. The main differences in relation to the January 11 2012 report study (the previous report) are the use of updated economic parameters such as metal prices, metallurgical recoveries, royalty and discount rates, plus a revised production schedule based on an improved mine excavation sequence. As in the previous report, a diluted resource model was used to account for mining dilution and expected level of selectivity. The final pit limit was defined based on a maximum NPV criterion using a gold price of USD 1,250/oz and a silver price of USD 23/oz. Metallurgical recoveries of 85% and 70% for gold and silver, respectively, were used and are based on documented metallurgical test work. It should be noted that the pit optimisations were done using a gold price of USD 1,350/oz. However, the subsequent financial analysis has been done using a gold price of USD 1,250/oz.

The open pit was designed taking into consideration the geotechnical recommendations as derived in the previous study with an additional change to flatter slopes in two areas of suspect pit wall stability. Four incremental cutbacks were designed to reach the final pit configuration and a long term production schedule was optimized to provide a maximum total mill feed of 0.84 million tonnes per year (Mtpa). This resulted in an 8 year LOM with maximum annual rock movement of approximately 3.2 Mt per year. In order to improve the ounces profile the plan makes use of a low grade stockpile with maximum capacity for 0.5 Mt.

The mine planning process outline above resulted on the Mineral Reserve estimates outlined in Table 4 below. The numbers are appropriate for the purpose of public reporting in that they provide an acceptable prediction of the material available to mine. In particular, dilution and mining recovery have been properly addressed. Tonnes, grades and metal content are reported using variable economic cut-off grades based on documented costs and prices.

Table 4. Ada Tepe Deposit – CIM Mineral Reserve Summary as at 21st March 2014

Mineral Reserve Summary					
Category	Tonnes	Gold		Silver	
		Grade	Ounces	Grade	Ounces
	(Mt)	(g/t)	(Koz)	(g/t)	(Koz)
Proven	2.59	5.39	449	2.82	235
Probable	3.61	3.08	358	1.79	208
Total	**6.20**	**4.04**	**807**	**2.22**	**443**

Reserves have been estimated using a gold cut-off of 0.6 g/t for the Upper Zone, and 0.8 g/t for the Wall Zone. Probable ore includes low grade ore that is processed at the completion of mining operations. Mineral Reserves are estimates using USD 1,250/oz Au and USD 23/oz Ag.

This reserve estimate has been determined and reported in accordance with NI 43-101, 'Standards of Disclosure for Mineral Projects' (the "Instrument", June 2011) and the classifications adopted by CIM Council in November 2010.

The Mineral Reserves at Krumovgrad have been estimated by including a number of technical, economic and other factors. A change to any of the inputs would therefore have some effect on the overall results. Concerning mining and metallurgical factors, it is CSA's belief that sufficient work has been done by DPM to ensure that these are not likely to have any significant or material effect on Mineral Reserves.

However, CSA relies on information as presented in Section 3 of this Technical Report as regards legal and environmental considerations.

1.13.2 Development and Operations

The timeframe for implementing the Project, once all necessary permitting is in place; from start through to practical completion of the process plant in readiness for ore commissioning is estimated to take 125 weeks.

Over the life of mine, the project will produce 686,000 ozs of gold in concentrate. This production is based on the Mineral Reserve over a mine life of 8 years. The plant is planned to treat a maximum of 0.84 Million tonnes per year (Mtpa) of ore over the 8 year mine life, including the processing of stockpiled low grade ore at the end of the Project.

Development of the Project will require the acquisition of land and buildings from a variety of impacted stakeholders. The size of the Project footprint has been minimised as much as possible and the footprint and its anticipated buffer will be approximately 85 ha. The overall site plan is presented in Figure 1.

The operation is planned to use conventional mining methods to mine ore, low grade and waste. The mining equipment proposed for the mining operation includes a 23.7 m3 back hoe excavator and off-highway haul trucks with a payload capacity of 40 tonnes. Provision has been made for drilling and blasting from the initial benches. The open pit will operate for 2 shifts per day to minimise the noise impacts on the local communities.

The process selected as a result of the testwork program comprises crushing and milling of the ROM ore followed by froth flotation to produce a gold and silver bearing concentrate. The process plant will operate 24 hours per day, 7 days per week, except for ore crushing which will operate only for 12 hours per day. The plant is designed to process approximately 100 tonnes/hour at an operating availability of 91.3%.

Metallurgical recoveries of 85% and 70% for gold and silver, respectively, were used for the feasibility assessments.

The project will employ approximately 230 people on site engaged in the administration, mining, and processing operations.

The process plant will be located on the side of the Ada Tepe hill, adjacent to the Integrated Mine Waste Facility (“IMWF”) and approximately 1 km south of the open pit. The milling and flotation areas will be in a building which also incorporates maintenance facilities for the plant, as well as warehouse, plant offices and change rooms. The mining fleet and other company vehicle maintenance will be done in a separate building about 600 m north of the process plant.

Process plant tailings will be thickened to a paste of maximum solids content ranging between 56%wt and 68%wt and will be disposed of in the IMWF, along with waste rock from the mine.

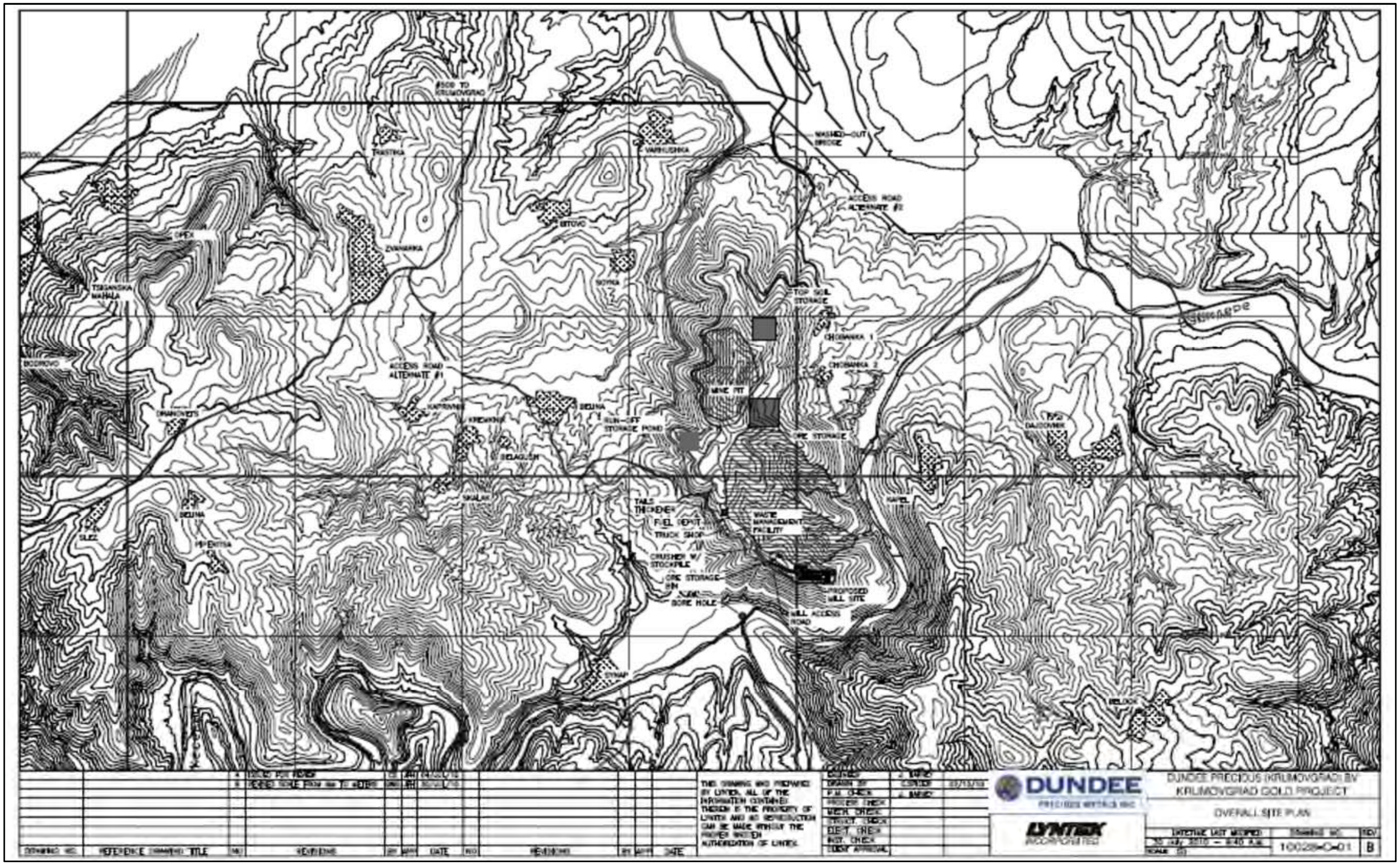

Figure 1. Overall Site Plan

1.13.3 Project Implementation

The timeframe for implementing the project through to practical completion of the process plant in readiness for ore commissioning extends from the present through to Q4 2016/Q1 2017. This schedule allows for land acquisition, the completion of the permitting and approvals, engineering design, procurement of materials and equipment, and construction of all facilities on site, including pre- production activity at the open pit mine.

The development capital cost of the project has been estimated at USD 164.1 Million and is based on a hybrid EPCM (engineering, procurement, construction and management) implementation strategy. A renowned international consultant has been involved since July 2012 in the delivery of the “EP” component of the project, and DPM Krumovgrad’s experience and “know how” in Bulgaria will be used to optimise the detail engineering, permitting and execution of the “CM” component of the project. An Owner’s team commensurate with the stage of development will gradually be built up and award a number of consulting agreements to various consultants to supplement the Owner’s team based on a defined scope of deliverables and implementation schedule. The estimate includes Owner's costs, working capital, and a contingency of approximately USD 30.8 Million.

A work force of approximately 300 people at its peak will be engaged on site during the construction period.

1.14 Environmental Studies

Under Bulgarian environmental regulations, the mining projects are required to comply with an EIA process as a key part of project permitting. The content and depth of the EIA is compliant with the Bulgarian Environmental Protection Act. The Bulgarian environmental legislation is fully harmonized with EU one.

The EIA systematically assesses project impacts in relation to the physical, biological and human environmental components, taking account of activities that will take place during the construction, operation and closure phases. Consideration was also given to alternative options for technology (mining, processing and waste management) and to the location of facilities (process plant, IMWF and RPWR).

The EIA report comprise of two major appendices. The first one is the Assessment on the Compatibility of Conservation Objectives of the Protected Zone Eastern Rhodope and Protected Zone Krumovitza with the Investment Proposal. This assessment has been prepared pursuant to the Bulgarian Law on Biodiversity, and the Regulation on Requirements for Conducting a Compatibility Assessment (“CA”) between Plans, Programs, Projects, as well as Investment Proposals and the Conservation Objectives of Protected Zones (“PZ”). Assessment of compatibility of the investment proposal with the object and purpose of protected areas is done according to the requirements of the European ecological network - NATURA 2000.

The second appendix of the EIA is The Mining Waste Management Plan. This was developed in connection with the URA, SG 23/12.03.1999, last amendment and elaboration in SG 70/8.08.2008 and the Regulation on the Specific Requirements to Mining Waste Management, SG 10/6.02.2009.

The Bulgarian Minister of Environment and Waters has signed a resolution № 18-8, 11/2011 approving the EIA for the Company's Krumovgrad Gold Project in Bulgaria. The resolution is still not in force because it was appealed by environmental NGOs.

The EIA study carried out indicates that the Project may be constructed, operated and closed in an environmentally acceptable manner. The implementation of the project was approved by environmental authority under execution of specific conditions described in the EIA Decision. The requested measures are connected with discharged water quality, level of dust and noise emission, mine waste management and protection of wild life.

1.15 Capital and Operating Costs

In calculating the returns from the Project, the following fundamental assumptions have been made:

- Metal prices of USD 1,250/oz for gold and USD 23/oz for silver will be maintained throughout the life of the Project.
- Metal price and currency hedging is excluded.
- Gold and silver recoveries of 85% and 70%, respectively, based on average testwork performance on all ore types.
- All production will be sold in the period in which it is produced.
- The life of the Project will be 8 years from commencement of processing operations, including the processing of stockpiled low grade material at the end of the project.

Evaluation has encompassed a 125 week development and construction period prior to commencement of ore feed to the mill, through to the end of ore processing.

The key project statistics for the life of the Project are summarised for both scenarios modelled (850 ktpa) in Table 5 to Table 8 following, the content of which is:

- Project Results Summary
- Production and Revenue Summary
- Operating and Capital Cost Summary

Table 5. Project Results Summary

Item	Unit	LOM
Throughput	Mt/a	0.85
Project Life	Years	8
Gold Price	USD/oz	1,250
Silver Price	USD/oz	23
Project Economics		
NPV at a discount rate of 7.5%	USD M	143.9
Internal Rate of Return	%	26.3
Payback Period	years	2.5

Table 6. Production and Revenue Summary

Production and Revenue		
Item	**Unit**	**LOM**
Throughput	Mt/a	0.85
Wall zone		
Total quantity of ore mined/milled	t	1,592,780
Gold grade	g/t	6.74
Silver grade	g/t	3.46
Upper zone		
Total quantity of ore mined/milled	t	4,611,315
Gold grade	g/t	3.11
Silver grade	g/t	1.79
Total ore		
Total quantity of ore mined/milled	t	6,204,095
Gold grade	g/t	4.04
Silver grade	g/t	2.22
Waste mined	t	16,241,716
Metallurgical recoveries		
Gold recovery	%	85
Silver recovery	%	70
Metal content		
Gold in concentrate	ozs	685,549
Silver in concentrate	ozs	309,915
Total Net Revenue	USD M	792.8
Site EBITDA *	USD M	519.02
* Before community support		

Table 7. Annual Cash Flow Forecast

Annual Cash Flow Forecast													
	Unit	Year -3	Year-2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total / Avg
Total ore	t				1,070,601	1,133,305	877,977	744,774	698,340	566,205	654,444	458,450	6,204,095
Au Grade	g/t				2.77	4.47	4.94	2.96	6.63	4.30	3.26	2.87	4.04
Ag Grade	g/t				1.65	2.36	2.57	1.74	3.41	2.37	1.91	1.75	2.22
Waste	t				2,240,899	1,014,695	1,156,023	2,466,226	2,512,660	2,644,794	2,441,248	1,765,172	16,241,716
Mined rock material	t				3,311,500	2,148,000	2,034,000	3,210,999	3,210,999	3,210,999	3,095,691	2,223,621	22,445,811
Strip Ratio	w:o				2.1:1	0.9:1	1.3:1	3.3:1	3.6:1	4.7:1	3.7:1	3.9:1	2.6:1
Milled Ore	t				809,023	803,026	812,586	836,608	773,311	785,464	788,800	595,276	6,204,095
Au in Feed	g/t				3.42	5.01	5.59	3.40	6.06	3.29	2.83	2.37	4.04
Ag in Feed	g/t				1.92	2.60	2.84	1.91	3.16	1.93	1.72	1.53	2.22
Gold-Silver Conc Produced (dmt)	t				3,618	5,264	5,936	3,721	6,123	3,383	2,914	1,844	32,805
Au recovery	%				85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%
Ag recovery	%				70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%
Au Recovered	oz				75,613	110,015	124,060	77,760	127,964	70,694	60,902	38,541	685,549
Ag Recovered	oz				34,997	46,951	51,886	36,043	54,955	34,067	30,581	20,434	309,915
Net Revenue	US$MM				$87.5	$127.2	$143.4	$89.9	$147.9	$81.8	$70.5	$44.6	$793
Mining Costs	US$/t mined				$2.84	$3.74	$4.05	$3.02	$3.48	$3.01	$3.32	$3.85	$3.34
Mining Costs	US$/t ore				$11.62	$10.01	$10.14	$11.61	$14.47	$12.30	$13.03	$14.37	$12.10
Processing Costs	US$/t ore				$21.84	$21.43	$22.32	$20.70	$22.15	$22.37	$21.74	$22.11	$21.81
Tailings Costs	US$/t ore				$2.70	$1.61	$2.40	$2.95	$3.23	$3.39	$2.39	$0.99	$2.50
Royalty	US$/t ore				$2.14	$5.40	$8.26	$4.18	$4.54	$4.83	$1.58	-	$4.00
Cash Cost	US$/t ore				$41.92	$42.08	$46.71	$42.92	$48.14	$46.51	$42.31	$42.18	$44.13
Cash Cost (incl. royalty) - by-product	US$/oz				$438	$297	$296	$451	$281	$506	$536	$639	$389

Annual Cash Flow Forecast													
	Unit	Year -3	Year-2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total / Avg
Cash Cost (incl. royalty) - co-product	US$/oz				$445	$305	$304	$458	$289	$512	$543	$645	$396
Mining Costs	US$MM				$9.4	$8.0	$8.2	$9.7	$11.2	$9.7	$10.3	$8.6	$75.1
Processing Costs	US$MM				$17.7	$17.2	$18.1	$17.3	$17.1	$17.6	$17.1	$13.2	$135.3
Tailings Costs	US$MM				$2.2	$1.3	$2.0	$2.5	$2.5	$2.7	$1.9	$0.6	$15.5
G&A Costs	US$MM				$2.9	$2.9	$2.9	$2.9	$2.9	$2.8	$2.8	$2.8	$23.0
Royalty	US$MM				$1.7	$4.3	$6.7	$3.5	$3.5	$3.8	$1.2	-	$24.8
Cash Operating Costs	US$MM				$33.9	$33.8	$38.0	$35.9	$37.2	$36.5	$33.4	$25.1	$273.8
EBITDA	US$MM				$53.5	$93.4	$105.4	$54.0	$110.7	$45.3	$37.1	$19.5	$519.0
Support to local communities	US$MM	–	–	($3.0)	($1.0)	($1.0)	($3.1)	($2.6)	($5.4)	($2.0)	($1.7)	–	($19.8)
Taxation	US$MM	–	–	$2.2	($1.0)	($5.2)	($6.2)	($0.7)	($8.5)	($2.0)	$0.2	$0.7	($20.6)
Operating Cash Flow	US$MM	–	–	($0.8)	$51.5	$87.2	$96.2	$50.7	$96.7	$41.2	$35.6	$20.2	$478.6
Less: Change in Working capital	US$MM	–	–	($2.7)	$17.3	$3.2	$1.3	($4.3)	$4.8	($5.5)	($0.9)	($9.1)	$4.2
Less: Capex		–	–	–	–	–	–	–	–	–	–	–	–
Development	US$MM	$23.0	$65.6	$75.5	–	–	–	–	–	–	–	–	$164.1
Sustaining / Closure	US$MM	–	–	–	$2.0	$3.0	$3.4	$2.0	$2.0	$2.0	$1.0	$12.0	$27.2
Total Capex	US$MM	$23.0	$65.6	$75.5	$2.0	$3.0	$3.4	$2.0	$2.0	$2.0	$1.0	$12.0	$191.3
Free Cash Flow	US$MM	($23.0)	($65.6)	($73.5)	$32.2	$80.9	$91.5	$53.1	$89.9	$44.8	$35.5	$17.3	$283.1

Development includes all pre-production activities

Table 8. Operating and Capital Cost Summary

Item	Unit	LOM
Throughput	Mt/a	0.85
Total cash cost per tonne of ore processed	USD/t	44.13
mining cash cost per tonne of ore	USD/t	12.1
processing cash cost per tonne of ore processed	USD/t	21.81
tailings treatment & IMWF	USD/t	2.50
royalty per tonne of ore processed	USD/t	4.01
admin cash cost per tonne processed	USD/t	3.71
Total Cost of production:	USD M	273.81
Mining	USD M	75.1
Processing	USD M	135.3
Tailings treatment & IMWF	USD M	15.5
Royalty	USD M	24.8
Administrative costs	USD M	23.0
Capital Costs		
Owner's cost	USD M	15.1
EPCM	USD M	9.2
Process plant and buildings	USD M	73.7
Mobile and Mining Equipment	USD M	12.6
Land Acquisition	USD M	4.6
Integrated Mine Waste facility and Haulroads	USD M	33.4
Contingency (12.5%)	USD M	15.5
Total	USD M	164.1
Sustaining/Deferred capital		
Sustaining capital	USD M	12.5
Closure and rehabilitation costs	USD M	14.7

1.16 Financial Summary

Over the LOM, on-site cash operating costs (not including royalty) are projected to be USD 249 Million, at an average of USD 40.13 per tonne processed. Total inclusive costs are projected to be USD 44.13 per tonne of ore, or USD 389 per ounce of gold equivalent.

Dundee will pay a royalty to the Bulgarian government, which is expected to be 2.57% of the gross value of ore produced from the mine, totalling USD 24.84 Million.

Based on USD 1,250 per ounce price for gold and USD 23 per ounce for silver, the Project produces an internal rate of return ("IRR") of 26.3% on a 100% equity funded basis, and an after tax net present value ("NPV") of USD 143.9 Million at a discount rate of 7.5%. For a gold price of USD 1000 per ounce, the sensitivity analysis shows the equivalent IRR will be 16.2% with an NPV of USD 58.9 Million.

Sensitivity analysis has been conducted to assess the effects of changes in key parameters upon the IRR, NPV and the period required for payback of the initial capital. The analysis encompasses the range of +/-10%, 20%, and 30% of the following key parameters:

- Gold price
- Gold recovery
- Aggregate operating costs
- Capital costs

The effect of these scenarios on the IRR and NPV are presented in Figure 2 to Figure 4.

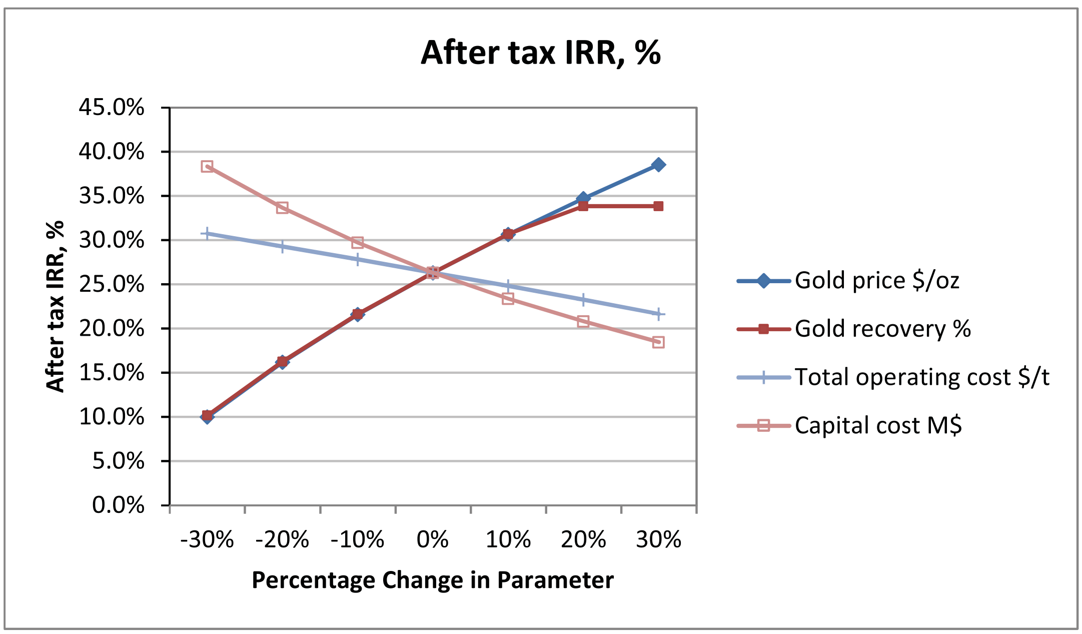

Figure 2. After tax IRR, %

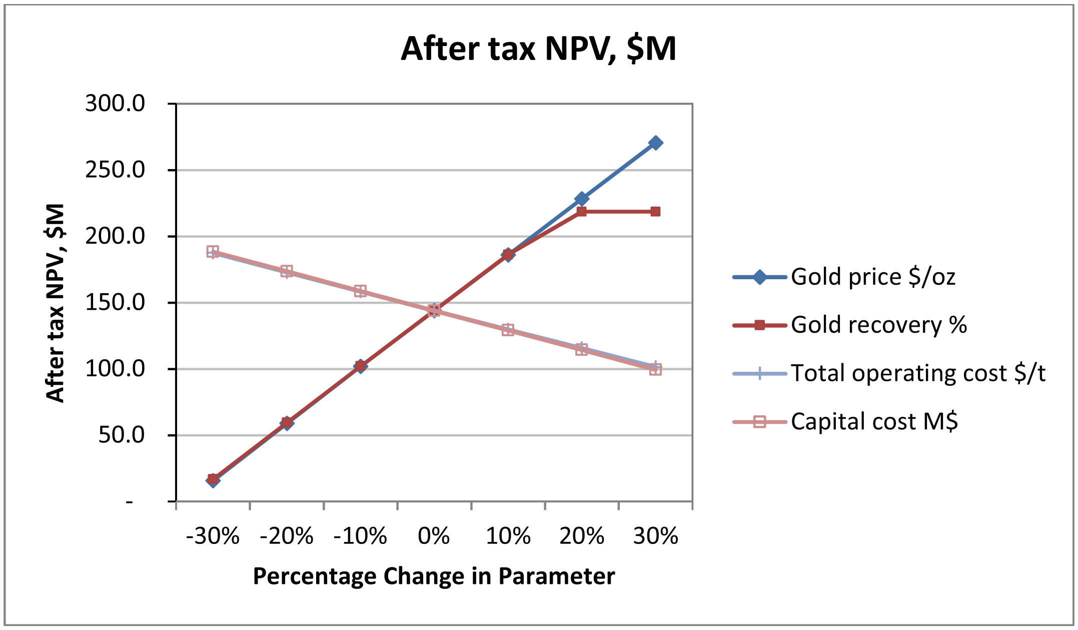

Figure 3. After tax NPV, USD M

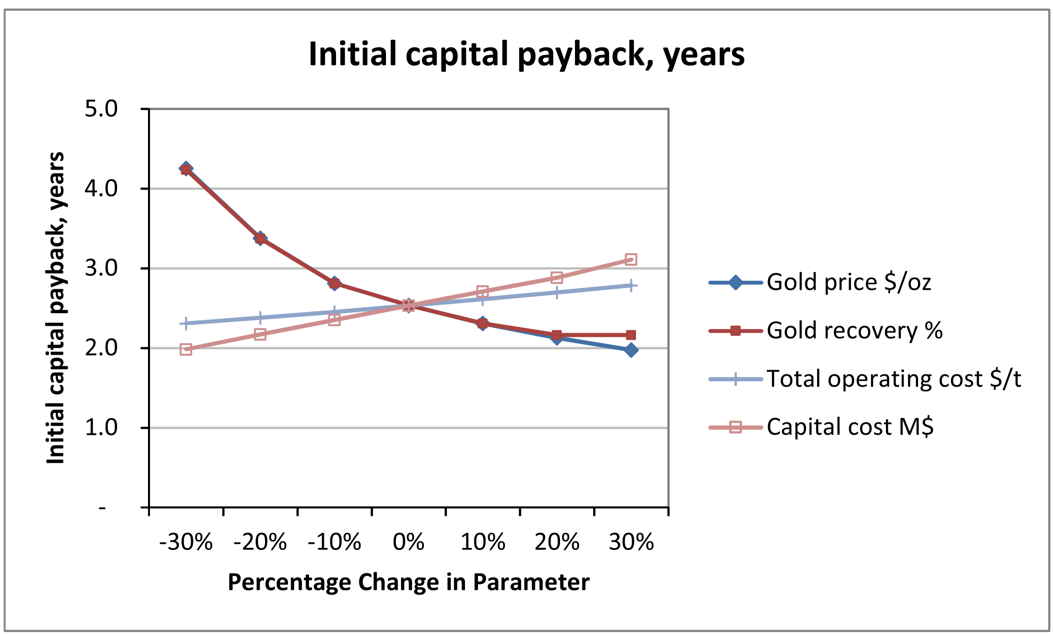

Figure 4. Initial Capital Payback – years

1.17 Conclusions

BMM conducted detailed exploration of the Ada Tepe prospect between 2000 and 2004. 52.9 km of drilling, and 18.3 km of surface trenching was completed, with more than 66,000 individual assay intervals and 5,700 bulk density determinations. This has resulted in a strong level of confidence in the data on which the resource is based. The mine plan proposed shows a high conversion of resources to reserves at the cut-off grades selected.

The extent of the data collected through this exploration program and the quality control standards used provide the basis for a high level of confidence on the potential of this project. Furthermore, the project has been demonstrated to be technically and commercially viable if designed, constructed, commissioned and operated as specified in the mining study. Opportunities still exist in most areas of the project to optimize various components that will be more rigorously investigated during the detailed engineering phase and where appropriate will be incorporated into the project.

CSA makes the following concluding comment as regards drilling, sampling and data verification;

- *CSA verified the location of some trench and drill access road cut exposures during the 2012 site visit and was able to confirm the positions of some sampling points.*

- It is the opinion of CSA that the RC drilling and associated sampling was completed to high industry standards. This opinion is informed by a review of data collection procedures, protocols and metadata contained in the database for the project, as reviewed during the 2012 site visit.

- *CSA observed this pulp library facility during the 2012 site visit and performed random spot checks of sample numbers and compared these with data contained in the project database. No issues were detected.*

- During the 2012 site visit CSA reviewed logging information contained in the database for the project, and verified this information (exported as graphic drill logs from acQuire software) against several drill holes inspected at the time. No issues were detected and CSA verifies that the information stated as being collected has been captured.

- CSA used the software programme QAQCReporter (“QAQCR”) to produce QA/QC reports in order to review the accuracy and precision of the assayed QA/QC material and samples. No significant issues of bias or fatal flaws were noted.

- Results of all of the internal laboratory blanks and standards were not available for review by CSA, however the results of the BMM submitted blanks and standards assayed show no evidence of systematic bias. CSA considers that the gold and silver standards analysed by the SGS Gura Rosiei laboratory to be accurate and appropriate for mineral resource estimation studies.

- CSA considers that the gold and silver standards analysed by the SGS Welshpool laboratory to be accurate and appropriate for mineral resource estimation studies.

- Internal laboratory blanks and standards were not all available for CSA review. Those submitted by BMM were reviewed. CSA notes that accuracy and precision exhibited by SGS Chelopech was, at times, poorer than the other laboratory results, however no significant bias is noted and CSA considers that the gold and silver standards analysed by the SGS - Chelopech laboratory to be accurate and appropriate for mineral resource estimation studies.

- CSA reviewed the results of the check analyses (duplicates, repeats and pulp splits). No significant bias or material issue were detected.

- CSA considers that Industry Standard acceptable levels of precision are reported for all of the sampling stages for the purpose of mineral resource estimation.

- CSA, having reviewed the results of twin core sampling, agrees with the conclusions reached by RSG Global.

CSA makes the following concluding comment as regards to Mineral Resource Estimation;

- CSA considers the approach used by RSG to model mineralisation, lithological and oxidation boundaries to be valid for the current level of study.

- CSA considers the approach of defining a zone of dilution to be a valid one which honours the mining approaches being considered and facilitates the reliable use of MIK as a grade interpolation method.

- CSA agrees with the approach adopted by RSG for the consideration of grade capping.

- CSA agrees with comments made by RSG as regards data clustering and agrees that cell declustering is a valid and meaningful approach to arriving at reliable and unbiased mean grades for each domain.

- CSA has reviewed density data and considered the application of assigned densities to material types to be valid.

- CSA considers the global validations completed by RSG to be valid. As an additional check CSA completed a validation estimate on selected domains within the Upper Zone using Uniform Conditioning as a check against MIK.

- CSA has reviewed the relationship between gold and silver grades and believes the assumptions regarding the appropriateness of estimating silver grade through linear regression from gold grade in the model, to be valid.The results of the Uniform Conditioning check estimate for silver, completed by CSA on several domains in the Upper

Zone compares favourably to the results obtained by RSG and goes some way to validate the approach taken by RSG to estimate silver grade.

- CSA completed the following validation checks on the RSG Mineral Resource block model;
    - Swath Plots depicting model tonnes, input declustered composite gold grade, output block model gold grade and drill metres per slice, for the Wall Zone.
    - Regularisation of blocks within the RSG block model file so that this block model could be used for mine planning work completed by DPM in 2013.
    - A check estimate of the CSA block model using Uniform Conditioning, completed for Domains 10, 11 and 12 within the Upper Zone. These domains are considered representative of the grade distribution within the Upper Zone and together account for 27% of contained metal within the Upper Zone.
    - On-screen visual comparisons of the RSG block model grades (via MIK) and the CSA Uniform Conditioning check estimate block model grades, for domains 10, 11 and 12.
- The UC check estimate compares favourably to the CSA block model interpolated by RSG using MIK. Additionally, CSA makes the following comment relating to the check estimate;
    - The check estimate reports 4% less tonnes at a 9% higher gold grade, resulting in a 4% increase in contained gold within the check domains, at a 0.6 g/t Au cut-off (the cut-off used to report Mineral Resources and Mineral Reserves in the Upper Zone).
    - The check estimate reports 4% less tonnes at a 5% higher silver grade, resulting in a 1% increase in contained silver within the check domains, at a 0.6 g/t Au cut-off.
    - UC has validated the MIK results suggests the variogram models and the linear regression of silver to be valid.
- The Mineral Resource Estimates for the Ada Tepe deposit have been categorised in accordance with the criteria laid out in the NI 43-101. A combination of Measured, Indicated and Inferred Mineral Resources have been defined using definitive criteria determined during the validation of the grade estimates, with detailed consideration of the NI 43-101 categorisation guidelines.
- On an after tax basis, based on USD 1,250 per ounce price for gold and USD 23 per ounce for silver and on the treatment of 850,000 tpa of ore, the project produces an IRR of 26.3% on a 100% equity funded basis and an after tax NPV of USD 143.9 million at a discount rate of 7.5%. The anticipated project payback time is 2 and a half years. For a gold price

of USD 1,625 per ounce, the sensitivity analysis shows the equivalent IRR will rise to 38.6% with an NPV of USD 270.5 million.

- No risks are considered to be fatal flaws in the context of the Krumovgrad Project, although further actions to improve risk profile will be undertaken.

1.18 Recommendations

The following recommendations are made, as regards the Ada Tepe Deposit;

- A review of logged oxidation and therefore the interpretation of oxidation surfaces used to flag the density model may require sensitivity review to aid both improvements in the tonnage model, and the metallurgical process.
- The current mineral resource estimate reports a selective mining model for the upper zone and assumes that a high density of high quality grade control (such as angled RC drilling) will be available. The current close spaced drilling has identified significant short scale variability and RSG considered that appropriate ore block delineation methods such as conditional simulation and probabilistic ore block selection are applied in conjunction with the RC grade control drilling. CSA agrees with this approach.
- The results of the economic analysis indicate that exploitation of the Ada Tepe gold deposit is economically viable and should proceed.
- Key areas for further action to be undertaken in 2014 include:
  - confirming construction cost assumptions
  - further focus on management of permitting
  - continuing assessment of the training needs and the planning for the integration of the maximum numbers of personnel local to Krumovgrad
  - continuation of the CSR plan to further engage the local community and, municipal, regional and national political support

2 Introduction

2.1 Terms of Reference - CSA Global (UK)

At the request of Dundee Precious Metals Krumovgrad ("DPMKr"), a 100% owned subsidiary of Dundee Precious Metals Inc. ("DPM") CSA Global (UK) Ltd ("CSA") was requested to compile a Technical Report on the Krumovgrad Gold Project, the material asset being the Ada Tepe Deposit located in south-eastern Bulgaria. This report is to comply with disclosure and reporting requirements set forth in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101"), Companion Policy 43-101CP, and Form 43-101F1.

This Technical Report discloses material changes to some components of the Mining Study and is an update to an earlier Technical Report dated January 11 2012, which summarised a Mining Study completed in November 2011.

This updated report pertains to the main deposit that comprises the Krumovgrad Project, namely the Ada Tepe deposit.

This report is an updated (revised) Technical Report which, at the request of the Ontario Securities Commission ("OSC") contains additional disclosure in the following areas;

3. In accordance with NI 43-101 S 2.2 (b), a statement that disclosure of Mineral Resources are inclusive of Mineral Reserves (see Section 14).

4. In accordance with Item 22 (b) of Form 43-101F1, inclusion of annual cash flow forecasts as at the effective date of this report (see Section 22).

2.2 Location

The mine site is located at Ada Tepe, approximately 3 km south of the town of Krumovgrad in the Kardjali District of Bulgaria.

2.3 Ownership

Balkan Mineral & Mining AD ("BMM", now "DPMKr"), a 100% subsidiary of DPM, was awarded the Krumovgrad Licence area (130 km2) on June 12, 2000 in accordance with the Agreement of Prospecting and Exploration reached with the Bulgarian Ministry of Economy. BMM is now known as Dundee Precious Metals Krumovgrad (DPMKr), as of 2nd July 2013. The licence area is located in the East Rhodope, approximately 320 km (by road) southeast of Sofia, in the Kardjali District, immediately south of the regional township of Krumovgrad (Latitude 25° 39' 15" and Longitude 41° 26' 15").

The Mining Licence ("Krumovgrad Concession") covers an area of 1370 hectares (13.7 sq.km) and includes the area of the Krumovgrad mining operation and the immediate surrounds. DPMKr has 100% ownership of the Project, which operates under a concession agreement that was granted by the Republic of Bulgaria in 2012 for a period of 30 years. Under Bulgarian regulations, the Mining Licence area is applied for on the basis of geographical coordinates.

2.4 Project Status

Bulgaria's legal framework for conducting business has been rapidly improving since enactment of the Commerce Act in 1991. With the accession to the European Union ("EU") in 2007, further positive changes are taking place in Bulgaria. While the legal and permitting requirements are both complicated and time consuming, the desire for change has created a collaborative environment for the government/regulatory agencies and the investor to work together to satisfy objectives of mutual benefit.

2.5 Principal Sources of Information

The authors have relied extensively on information compiled as a result of the mining study for the Krumovgrad Project which was completed by DPM between 2012 and 2014. This information was presented to, and reviewed by Mr. Julian Bennett – Associate Principal Mining Engineer during a visit to Sofia, Bulgaria in February 2014 and formed the basis of subsequent desk-top study culminating in the preparation of this Technical Report.

In addition, a current visit to the Krumovgrad Project was undertaken in November 2012 by Mr. Galen White – Principal Geologist of CSA Global (UK) Ltd for the purposes of project evaluation, geological review, data/procedural review and discussion with DPM staff. A review of the current estimate of Mineral Resources has also been undertaken by CSA.

A full listing of the principal sources of information is included in Section 27 of this report.

The preparation of the updated NI 43-101 report has been coordinated and completed by DPM, in conjunction with various specialist consultants required to complete all aspects of the updated study. Those retained directly for the Project were;

- DPM - Project Owner and Manager;
- Mr. Galen White (QP) – Geology and Resources
- Mr. Julian Bennett (QP) – Mining and Reserves
- Mr. Simon Meik (QP) – Metallurgy and Processing

In addition, significant study work completed by the following consultants, as summarised in the 2012 Report, has been reviewed as part of the current study.

- Golder Associates – mineable reserve, mine design and engineering; paste plant; mine waste and process tailings storage (integrated mine waste facility); and water management;
- Lyntek Incorporated – engineering and plant design; and preparation of the DFS
- SGS Canada – testwork, flowsheet development and communition and flotation circuit design.

Remaining portions of NI 43-101 report, including the Financial Analysis, were prepared by DPM and reviewed by CSA.

2.6 Site Visit

2.6.1 Site Visit Geology

Between November 2012 and January 2013 Mr. Galen White completed a desk-top study of available data and literature relating to geology and resources of the Krumovgrad Project, including a site visit to the project between the 5th and 9th November 2012 (White, 2012).

The objectives of the site visit and subsequent desk-top review were to;

- Review available information relating to the Mineral Resource Estimate and to essentially peer review this work to confirm the suitability of the Mineral Resource Estimate and associated block model for use as an input in to pit optimisation and mine planning work as part of the update to the mining study. In addition DPM required CSA to take ownership of the Mineral Resource Model and to provide whatever assistance, including sensitivity analyses, which may be required to assist mine planning work.

During the site visit the following was reviewed;

- Reviewing the contents of the Technical Report, specifically those sections that detail the methodology employed to complete the Mineral Resource Estimate, the input data, parameters and assumptions used and to check that these are robust and honour both the underlying data characteristics and the geological model for the project;
- Loading and validating the Drillhole and Trench Database, block model, 2D wireframe surfaces and 3D geological and mineralised domain wireframes;
- Tying these reviews to an on-site inspection of project geology, drill core, data collection protocols and communication with DPM project staff.

The following conclusions were drawn from the site visit;

- The MRE and supporting documentation has been completed to a high standard and all steps of the resource estimation process are well document, well-reasoned and honour the geological observations, grade characteristics and probable mining scenarios being considered.

- CSA was able to interrogate and validate the grade model and replicate reported figures of tonnage and grade quoted in the 2012 Technical Report. In addition CSA were able to verify all of the geological information contained in the reports via on-site inspection and further supports the geological model proposed for the Krumovgrad Project.

- The Mineral Resource Estimate is considered reliable and robust for the purposes of current mine planning.

- Resource tonnage is considered reliable. Density assignation has been informed by significant surface and sub-surface sampling such that average domain densities are adequate.

- Data collection practises and procedures reviewed were of a high standard, and in some aspects exceed Industry standard. Data QA/QC is well documented and suggests that grade data is both accurate and precise, and that geological information captured is pertinent and thorough.

2.6.2 Site Visit Mining

No visit has been made to site by the mining engineer, since no construction work has taken place and no mining facilities exist at present. The visit of the mining engineer consisted of a 5 day period in the Sofia office of DPM, in February 2014 during which time, in-depth discussions took place with the project's main personnel, data and information was studied, reviewed and information was freely exchanged.

2.7 Independence

The external contributors to this study neither have nor have had previously any material interest in DPM or related entities or interests. The relationship with DPM is solely one of professional association between client and independent consultant. This report is prepared in return for fees based upon agreed commercial rates and the payment of these fees is in no way contingent on the results of this report.

2.8 Forward Looking Information

This Technical Report contains “forward-looking information” or "forward-looking statements" that involve a number of risks and uncertainties. Forward-looking information and forward-looking statements include, but are not limited to, statements with respect to the future prices of gold and other metals, the estimation of Mineral Resources and Reserves, the realisation of mineral estimates, the timing and amount of estimated future production, costs of production, capital expenditures, costs (including capital costs, operating costs, cash cost per oz and per lb and other costs) and timing of the development of new mineral deposits, success of exploration activities, permitting time lines, LOM, rates of production, annual revenues, IRR, NPV, currency fluctuations, requirements for additional capital, government regulation of mining operations, environmental risks, unanticipated reclamation expenses, title disputes or claims, limitations on insurance coverage and timing and possible outcome of pending litigation.

Often, but not always, forward-looking statements can be identified by the use of words such as “plans”, “expects”, or “does not expect”, “is expected”, “budget”, “scheduled”, “estimates”, “forecasts”, “intends”, “anticipates”, or “does not anticipate”, or “believes”, or variations of such words and phrases or state that certain actions, events or results “may”, “could”, “would”, “might” or “will” be taken, occur or be achieved.

Forward-looking statements are based on the opinions, estimates and assumptions of contributors to this report. Certain key assumptions are discussed in more detail. Forward Looking statements involve known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements of DPM to be materially different from any other future results, performance or achievements expressed or implied by the forward-looking statements.

Such factors include, among others: the actual results of current exploration activities; actual results of reclamation activities; conclusions of economic evaluations; changes in project parameters as plans continue to be refined; future prices of gold and other metals; possible variations in ore grade or recovery rates; failure of plant, equipment or processes to operate as anticipated; accidents, labour disputes and other risks of the mining industry; delays in obtaining governmental approvals or financing or in the completion of development or construction activities, fluctuations in metal prices, as well as those risk factors discussed or referred to in this report and in DPM’s latest annual information form under the heading "Risk Factors" and other documents filed from time to time with the securities regulatory authorities in all provinces and territories of Canada and available at www.sedar.com.

There may be other factors than those identified that could cause actual actions, events or results to differ materially from those described in forward-looking statements, there may be other factors that cause actions, events or results not to be anticipated, estimated or intended. There can be no assurance that forward-looking statements will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements. Accordingly, readers are cautioned not to place undue reliance on forward-looking statements.

3 Reliance on Other Experts

This report is based on information contained in the Krumovgrad Project mining study dated November 2011, and the various consultants' reports supporting this study. Dundee has compiled information contained in this report from these sources, and also from equipment vendors and service suppliers Dundee and the authors have also relied upon others for information contained within this report. Information from third party sources is quoted either as a report in the text or referenced. In particular, Dundee's technical teams have updated a number of components of the mining study during January – February 2014. The results of these updated factors are included in this report.

Aside from Dundee, the authors of this report are not qualified to provide extensive comment on legal, permitting or financial issues associated with the Krumovgrad Project and included in this report. The assessment of data pertaining to these aspects relies heavily on information provided by Dundee.

The assessment of metallurgical and mineral processing aspects of the Krumovgrad Project, included in Sections 14 and 18, is based substantially on information and reports provided as part of the mining study and compiled by SGS Canada, Inc.

The authors of this Technical Report have reviewed available company documentation relating to the project and other public and private information as listed in the "References" section at the end of this Report. In addition, this information has been augmented by first-hand review and on-site observation and data collection conducted by the authors.

The Qualified Persons take responsibility for the content of this Technical Report and believe it is accurate and complete in all material aspects. However CSA is not responsible for nor has undertaken any due diligence regarding the non-geological/mining technical aspects of this report, which includes;

- Licencing and Tenure Information. This information has been reviewed.
- Environmental Study Information. This information has been reviewed.

4 Property Description and Location

The information set out in this section is taken from the 2012 Report where it remains current, augmented by updated comment from the author of this Technical Report, where relevant.

4.1 Background Information

4.1.1 Bulgaria

Bulgaria is a Slavic Republic in south-eastern Europe, bounded to the north by Romania, to the west by Serbia and Macedonia, to the south by Greece and Turkey and to the east by the Black Sea. The population is largely Bulgarian Orthodox Christian (~85%), with a Turkish and Muslim minority (~13%). The capital city is Sofia and the national population is approximately 8 million.

Bulgaria became a member of the EU on January 1, 2007 and has been a member of NATO since April 2004. The local currency, the BGN, has been pegged to the Euro (1.95583 BGN/EUR€) since 1999.

Bulgaria has achieved macro-economic stability, with a low inflation rate and a medium basic interest rate. Economic growth was estimated as 0.5% in 2013, down from 0.8% estimated in 2012. Inflation was 3.0% in 2012 and 1.5% in 2013. Both figures are estimated.

Educational standards within the country are high. Mineral exploration was important under the communist regime, resulting in a large pool of well qualified geologists and technical staff. The historical lead-zinc mining industry in the Eastern Rhodopes is also a source of both skilled and unskilled personnel.

Bulgaria is well serviced by facilities and infrastructure. Large towns usually have the facilities normally found in western European countries. The country is served by an extensive network of paved roads, except in the most mountainous districts. There is a comprehensive rail network.

4.1.2 Krumovgrad Project

The Krumovgrad Project is a planned 850,000 tonnes per year ("tpa") open pit gold mine located in Bulgaria which is consistent with existing permitting and environmental submissions and is financially viable. The mill facilities and mine will be developed, constructed, and operated by DPMKr, a wholly owned subsidiary of DPM. The size of the project footprint has been minimised as much as possible and the footprint and its anticipated buffer will be approximately 85 ha.

4.2 Project Location

The licence area is located in the East Rhodope, approximately 320 km (by road) southeast of Sofia, in the Kardjali District immediately south of the regional township of Krumovgrad (25° 39' 15"E and 41° 26' 15"N). Figure 5 displays the location of the licence area in the context of Bulgaria and the surrounding region. Krumovgrad is located approximately 320 km by paved road southeast of Sofia and some 15 km north of the border with Greece.

The Ada Tepe deposit is located 3 km south of the Krumovgrad town site and trends in a north south direction. The deposit area comprises of hilly topography abutting a major regional river system. The Project area is readily accessible at all times of the year.

Figure 5. Location Plan of the Krumovgrad Gold Project Area

4.3 Tenement Description

The Company is entitled to the prospecting and exploration of underground mineral resources – metal minerals in the indicated prospect area pursuant to an Agreement with the Minister of Economy, Energy and Tourism ("MEET"), concluded on June 12, 2000 (referred to hereinafter as the "Agreement").

The Agreement was concluded after the successful permitting process was finalized which included a tender for a prospecting and exploration permit announced in a publication of the State Gazette pursuant to Order No PD–16–460 dated 3 September 1999 of the Minister of Industry. In compliance with the outcome of the tender, MEET granted to the Company Permit No. 1 dated May 9, 2000 under which the Company may undertake prospecting and exploration works at its own expense in Krumovgrad prospect within an area of 130 km2.

Initially the Agreement was signed for a period of 3 (three) years starting from its effective date, i.e. until June 12, 2003.

The provision laid down in Art. 31, par.3 of the Underground Mineral Resources Act ("URA") provides an extension option for the prospecting and exploration period by 2 years each, provided the specific provisions under the concluded Agreement are observed and the work program is completed.

The Company has requested such a term extension twice and as a result the following were concluded:

- Additional Agreement No. 1 to the Agreement dated June 11, 2003, under which the Krumovgrad prospect area was reduced from 130 km2 to 113 km2 and the term of the Agreement was extended by two years – until June 13, 2005; and

- Additional Agreement No. 3 to the Agreement dated June 11, 2005 which enforced another reduction of the Krumovgrad prospect area from 113 km2 to 100 km2 and an extension of the term by another two years, i.e. until June 13, 2007.

In the meantime, on April 25, 2001 the Company registered the geological discovery of the Khan Krum deposit at the Minister of Environment and Waters ("MoEW") featuring gold mineral resources located in the Krumovgrad prospect area. Subsequently, the Company made a new geological discovery – Kandilka and declared it before the MoEW (an application dated 08.05.2001 to the MoEW). On the grounds of this provision and in compliance with the Agreement, the Company requested and was granted a term extension of the Agreement by another year and, to that end, Additional Agreement No 4 dated July 4, 2007 was concluded and the Agreement was extended for a period until June 13, 2008.

Based on the prospecting and exploration permit, the Company submitted to the MoEW Application with Incoming No ZNPB-1149/27.04.2007 for the registration and certification of a commercial discovery: Khan Krum deposit of gold ores comprising the prospects of Ada Tepe, Surnak, Skalak, Sinap, Kuklitsa and Kupel.

On August 28, 2009 the MoEW formally issued the Certificate for Commercial Discovery No 0417/28.08.2009 for the Khan Krum deposit, including the reserves, mineral resources and coordinates. This entitles DPM to plan and develop the resources and reserves commercially and is valid for 30 years. It protects DPM's right to the site while permitting and licencing continues.

The then Balkan Mineral & Mining EAD ("BMM" or the "Company"), a 100% subsidiary of Dundee, was awarded the Krumovgrad Licence area (130 km2) on the June 12, 2000 in accordance with the Agreement of Prospecting and Exploration reached with the Bulgarian Ministry of Economy. The licence area is located in the East Rhodope, approximately 320 km (by road) southeast of Sofia, in the Kardjali District, immediately south of the regional township of Krumovgrad (Latitude 25° 39' 15" and Longitude 41° 26' 15").

Bulgaria's legal framework for conducting business has been rapidly improving since enactment of the Commerce Act in 1991. With the accession to the European Union ("EU") in 2007, further positive changes are taking place in Bulgaria. While the legal and permitting requirements are both complicated and time consuming, the desire for change has created a collaborative environment for the government/regulatory agencies and the investor to work together to satisfy objectives of mutual benefit.

4.4 Permits

4.4.1 Timeline for Approval Actions.

The following outlines major milestones reached in the applications/permits required for the operation of the mine.

June 2, 2007	Original Concession Application
August 28, 2009	Commercial Discovery Certificate ("Certificate") Issued for the Krumovgrad Gold Project by Ministry of Environment and Waters ("MoEW").
June 4, 2010	Ministry of Economy, Energy and Tourism ("MEET") process initiated to obtain necessary approvals to open concession procedure. Company requested to update June 2007 Application. Confirmed consent, granted in December 2007, for opening the concession proceedings subject to an archaeological clearance of the site.
August 9, 2010	Upon a recommendation by the Ministry of Culture ("MoC"), BMM entered into a framework agreement with the National Institute of Archaeology and Museum at the Bulgarian Academy of Sciences ("IAM-BAS").
October 7, 2010	Filed updated documentation with the MEET to obtain a mining concession for the Krumovgrad Gold Project.
October 29, 2010	BMM submits an Environmental Impact Assessment ("EIA") report to the MoEW.
January 4, 2011	MoEW approve the quality of BMM's EIA report.
February 11, 2011	Council of Ministers ("COM") approves the granting of the mining concession to BMM.
March 22, 2011	Mining concession appealed by Krumovgrad municipality and three Non-Governmental Organizations ("NGO's").
July 11-22, 2011	Four public hearings were held on the EIA report in Krumovgrad Municipality.

August 23, 2011	BMM is officially notified by the Supreme Administrative Court ("SAC") that the court hearing regarding the mining concession appeal is scheduled for February 1, 2012.
September 1, 2011	BMM submit a request to the SAC to appoint an earlier date for the court hearing.
September 19, 2011	Supreme Expert Environmental Council ("SEEC") votes in favour of the EIA resolution including the pre-emption of execution of the EIA.
September 21, 2011	BMM receive confirmation from SAC that the court hearing for the mining concession will be held on October 19, 2011 instead of February 1, 2012.
October 5, 2011	BMM receive a letter from the Minister of Environment and Waters ("MoEW") announcing that, following an objection filed by the mayor of Ovchari village, the MoEW has required that the EIA be reconsidered at a subsequent session of the SEEC. BMM submitted an appeal to the SAC against silent refusal of the Minister of Environment and Waters to approve the EIA.
October 19, 2011	At the SAC hearing all the parties – Krumovgrad Municipality, NGOs and BMM presented in brief their arguments in support and against the claims. BMM stated a position that the claims of both Krumovgrad Municipality and NGOs are inadmissible due to the fact that none of these entities are officially party to the concession granting procedure, therefore do not have legal interest in the proceedings. Arguing for the inadmissibility of the claims, BMM requested termination of court proceedings. BMM arguments were supported by both CoM and MEET, who separately laid down the same arguments verbally and in writing. The court decided that the issue of admissibility of the Krumovgrad Municipality and NGOs' claims is predetermining for the litigation proceeding and announced that before considering the claims on their merits, will review the admissibility issue in a closed session. Formal notification of the outcome was pending.
November 8, 2011	SEEC vote in favour for a second time of the EIA resolution including the pre-emption of execution of the EIA.
November 24, 2011	BMM received the EIA resolution with pre-emptive execution signed by the Minister of Environment and Waters on November 23, 2011.
November 28, 2011	Pre-emptive execution of the EIA is appealed by Krumovgrad municipality and three NGO's.
December 8, 2011	The 5-judge panel of the SAC started litigation in response to the appeals from two NGO's and Krumovgrad Municipality challenging the ruling for termination of the concession court case of the 3-judge panel.
December 9, 2011	EIA resolution is appealed by three NGOs.
September 13, 2012	The three member panel of SAC issued a ruling dismissing the appeals filed against the EIA resolution.
October 3, 2012	BMM submitted before the Mayor of Krumovgrad Municipality an Application for approval of the draft ToR and for issuance of permission to prepare the DDP. Following a request by the Municipal Council, some corrections of the documentation were made and it was resubmitted to include amendments of the SDA following the update of the Krumovgrad Municipal Council (KMC).

January 25, 2013	The KMC did not issue a decision on the application within the statutory deadline. Therefore BMM filed an appeal against a silent refusal of the part of KMC.
March 4, 2013	The 5-member SAC panel issued a Ruling upholding the Ruling of the 3-member panel, which rejected the appeals by the NGOs against EIA Resolution. This ruling was final and was not subject to appeal and the EIA Resolution entered in force.
April 30, 2013	The Administrative Court in Kardzhali issued a Ruling repealing the Council's silent refusal to decide on the application and referred the application back to the Council with instructions to issue an explicit decision in due term.
May 29, 2013	At its session the KMC reviewed the ToR of DDP and issued a decision, refusing to approve it and to allow preparation of DDP, because of alleged lack of competence to decide on this issue.
October 4, 2013	The Administrative Court in Khardzhali issued a ruling, which overturned the refusal of the KMC to issue permission for the preparation of the detailed development plan ("DDP") and returned the matter to the MC to decide on the Company's request within one month.
December 18, 2013	The ruling of the Administrative Court was appealed by the KMC before the 3-judge panel of the Court, and the appeal was dismissed by the court.
January 14, 2014	The KMC made a final appeal to the 5-judge panel of the Court.
February 10, 2014	The ruling of the 5-judge panel, which cannot be appealed, was issued in favour of the Company.
March 7, 2014	The KMC again considered the Company's application and a majority of the Council voted against the approval of the Terms of Reference and the preparation of the DDP. The decision has been formally presented to the Governor of the Region, who will either: (i) confirm the decision of the KMC; (ii) deny the decision of the KMC and return the matter to the council for further consideration; or (iii) appeal the KMC's decision to the Court. A decision is expected by the Chairman before April 22, 2014.

The Company expects that the KMC will proceed in accordance with the direction of the Administrative Court. The Company currently expects that the appeal process being exhausted, the KMC will issue a favourable decision on the Detailed Development Plan ("DDP"), terms of reference, which in turn is expected to facilitate securing the remaining local permits, which include;

1. Water Consumption and Discharge Permits;
2. Clearing Permit for cutting the trees within the concession area footprint; and
3. Finally, an investment design for the project should be approved and construction permit obtained.

The KMC is required to render a decision within one month of notice of the final ruling by the Court.

If necessary, DPM will defer its scheduled ramp-up and take steps to limit spending to only those activities necessary to secure the remaining local permits. As a result of previous delays, commissioning of the Project and the hand-over to operations are currently expected to occur during the fourth quarter of 2016 or early in the first quarter of 2017.

Project Implementation can be divided into four main phases, from the Pre-construction period through to entering full operations. The mining study has identified the endorsements, approvals, permits and/or licenses required for each of the four phases of the Project, which are highlighted as follows.

4.4.2 Pre-Construction

Included in this phase are the provisional permits require to proceed on the Project, including - Granting of a concession, Change of Use of Land, Detailed Development Plan (“DDP”), Water Management Endorsement, Permits for storage of hazardous substances. A positive statement by the Fire Safety and Rescue Departments, coordination of the investment projects with the MoC and preliminary agreements with the service providers (Electrical Power Producers, etc.) in order to connect the construction to the infrastructure networks. These shall be issued upon completion of the engineering phase of the Project and are the basis for issuance of a Construction Permit.

The consultant has to prepare a Compliance Assessment of the Proposed Project which is to include the essential requirements in constructions: land ownership, use of land, availability of permits issued by the regulatory authorities and the service providers and the qualifications and licenses of the designers.

Finally, based on the consultant’s report, the Chief Municipal Architect has to approve the investment project and issue a Construction Permit.

4.4.3 Construction

The second phase commences after the Construction Permit has been issued and continues until all construction and equipment installation is finished and mechanical completion is achieved. During this period the activities are focused on obtaining of permits for importation of equipment, reagents and consumables and permits for export of the future finished production, as well as conclusion of final agreements with the service providers for connection to the technical infrastructure networks and preparation of agreements for electricity, water, fuel, etc. consumption and supply. The main steps in phase two are as follows:

Independent construction control during the construction stage shall be carried out by a licensed consulting firm that will be responsible for compliance with procedures.

- Lawful start of construction.
- Implementation of the directive instructions to the Project.

- Documenting of construction.
- Use of appropriate construction materials.
- Safe execution of construction.

The Directorate of National Construction Control ("DNSK" and "RDNSK") - this state agency under the Ministry of Regional Development and Public Works is authorized to exercise compliance of construction works with the approved investment proposal and the applicable legislation, regulations, standards and norms in the Republic of Bulgaria. The Directorate has the authority to shut down illegal construction sites and also sites for non-compliance.

4.4.4 Pre-Operations

The third phase commences after construction and engineering works and concludes with 72 hours testing of the equipment and plants to demonstrate production capacity and achievement of designed environment parameters. The Bulgarian legislation does not determine a deadline for finishing of the commissioning, achievement of designed plant capacity and implementation of all recommendations for protection of the environment, from the EIA. The main steps in phase three with the appropriate authorities are as follows:

Fire Protection Statement - the local fire protection authorities; Sanitary Statement - the regional branch of Hygiene-Epidemiological Inspectorate, under the Ministry of Health. The rest of the regulatory authorities (Civil Protection Dept., RIEW, service providers) will have representatives in the State Acceptance Commission. In principle, these authorities do not issue preliminary statements.

Upon completion of construction, obtaining the abovementioned preliminary statements and concluding the final agreements for connecting the project infrastructure to the providers' networks, the Consultant shall prepare a final report.

Four months are considered to be a reasonable period for finalising the 3rd stage.

4.4.5 Operations

The fourth phase includes procedures for organizing a State Acceptance Commission which will include all regulatory authorities related to the respective construction, signing of Acceptance Protocol Form #16 and obtaining an Operating Permit. One month is considered as a reasonable period for implementation of these activities. Only when the Operational Permit has been issued can the plant be considered as legitimate and commissioned. Then the documentation completed during the previous stages can be archived.

4.5 Royalties

The Company will pay a royalty to the Bulgarian government, at a variable royalty rate applied to the gross value of the gold and silver metals combined in the ore mined. The royalty rate depends on the profitability of the operation. At a pre-tax profit to sales ratio of 10% or less, the royalty rate will be 1.44% of the value of the metals. At a pre-tax profit to sales ratio of 50% or more, the royalty rate will be 4% of the value of the metals. At intermediate levels of profitability, the royalty rate will vary on a sliding scale between 1.44% and 4% in a linear fashion. At a gold price of $1,250/oz and a silver price of $23/oz, the royalty rate will be in the order of 2.5% of the gross value of gold and silver metals contained in the ore produced from the mine.

4.6 Environmental Liabilities

An EIA statement was issued by the Bulgarian Minister of Environment and Water. The statement includes a number of conditions which have to be implemented during detailed design, construction, operation, closure and rehabilitation stages of the project. The Closure and Rehabilitation study was also approved by the Minister of Economy and Energy.

DPM is not aware, nor has it been made aware, of any other significant environmental liability associated with the Krumovgrad Project.

4.7 Other Risk

DPM is not aware of any other significant factors or risks that may affect access, title or their right or ability to perform the contemplated work on the property.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The information set out in this section is taken from the 2012 Report, has been reviewed by the author of this Technical Report and remains current.

5.1 Access

The town of Krumovgrad is approximately 320 km southeast by paved road from the capital of Bulgaria, Sofia, which is serviced by a modern International airport. A second International Airport is located in the city of Plovdiv located approximately 106 km northwest of Krumovgrad. The Ada Tepe deposit is located some 3 km south of Krumovgrad. Access to the general area is available at all times of the year, by sealed roads to Krumovgrad. Access within the licence area is good, with all-weather surface roads transecting the project area. Secondary roads are unsurfaced but generally accessible with four-wheel drive vehicle year round.

5.2 Climate

The project area belongs to the Continental-Mediterranean climatic type, the main feature of which is its proximity to the climate of Subtropical Europe, featuring markedly higher winter and substantially lower summer precipitation. Winters are mild, but during intensive cold spells temperatures may fall to -13°C. Summers are hot, reaching 36°C in warmer spells and exceeding 40°C in some locations.

The average annual precipitation is 703.5 mm. The bulk of this falls in autumn and winter, occasionally as snow in the coldest months. The highest rainfall occurs in December (96.9 mm average) and the lowest in August (24.1 mm). Estimated 1:100 year rainfall events are 117.3 mm for 24 hours duration, and 184.1 mm for 72 hours. Probable Maximum Precipitation ("PMP") estimates are up to 383.4 mm for 24 hours and 605.4 mm for 72 hours. Average annual evaporation is 1050.8 mm, similar overall to annual rainfall in magnitude, but opposite in seasonal sense.

5.3 Local Resources

Small villages are dispersed widely throughout the licence area involved in subsistence farming, particularly livestock and the growing of tobacco on the poorly developed soils characteristic of the region. The other main land use within the licence area is state controlled forestry. The population of Bulgaria is largely non-practicing Eastern Orthodox Christian (85%) with a Turkish

Muslim minority predominantly residing in the southeast of the country, including the licence area.

5.4 Infrastructure

Infrastructure in the area is good, with paved roads, power and water resources available within close proximity to the Project.

5.5 Physiography

The Krumovgrad District is around 230 m above mean sea level and is characterised by a rugged landscape. The Ada Tepe deposit is located in an area of moderate, hilly topography abutting a major regional river system. The project area is readily accessible at all times of the year.

6 History

The information set out in this section is taken from the 2012 Report, has been reviewed by the author of this Technical Report and remains current.

6.1 Summary

The Ada Tepe prospect was the subject of only very brief attention in previous State funded exploration in the early to mid-1990s, by GeoEngineering of Assenovgrad, and Geology & Geophysics of Sofia

On June 12, 2000, BMM (a 100% subsidiary of DPM, now referred to as DPMKr) was awarded the Krumovgrad Licence area (113 km2) in accordance with the Agreement of Prospecting and Exploration reached with the Ministry of Economy. Work by BMM included diamond and RC drilling, rock chip sampling, bulk density measurements, and detailed geological mapping.

6.2 Previous Mineral Resource Estimates

Two published Mineral Resource Estimates for the Ada Tepe deposit, both compliant with disclosure and reporting requirements set forth in NI 43-101, Companion Policy 43-101CP, and Form 43-101F1 at the time of reporting, have been completed to date and both were prepared by RSG, in October 2002 and in November 2004. Both mineral resource models were estimated using a combination of OK and MIK estimation methods. Table 9 presents the 2002 Mineral Resource Estimate, reported using a 1.0 g/t Au lower cutoff grade. Table 10 provides the grade tonnage report for the November 2004 estimate.

CSA took receipt of and reviewed the 2004 Mineral Resource Estimate block model and completed validations on this model. There has been no additional resource definition drilling or sampling since the preparation of this Mineral Resource Estimate. CSA modified this model in 2012 via block regularisation to provide a more user-friendly model for mine planning purposes. The underlying grade estimate has not been changed. Details of CSA review work and the presentation of the current Mineral Resource Estimate is contained in Section 14.

Table 9. Ada Tepe Prospect - October 2002 Mineral Resource Estimate

Wall Zone: Ordinary Kriged Gold and Silver Estimates (25 mE x 25 mN x 5 mRL Size Blocks) Upper Zones: Gold Estimate by SMU Emulation of 5 mE x 5 mN x 2.5 mRL Size Blocks, and Silver Estimate by Ordinary Kriging (25 mE x 25 mN x 5 mRL Size Blocks) - Cutoff Grade : 1.0 g/t Au						
Zone	**JORC/NI 43-101 Mineral Resource Category**	**Tonnes (Mtonnes)**	**Gold**		**Silver**	
			Grade (g/t)	**Ounces (x 1000)**	**Grade (g/t)**	**Ounces (x 1000)**
Wall	Indicated	1.48	7.3	350	4.3	207
Total Indicated	Indicated	1.48	7.3	350	4.3	207
Wall	Inferred	0.40	4.2	54	2.9	37
Upper - North	Inferred	0.77	3.0	74	0.9	30
Upper - South	Inferred	3.50	3.8	431	0.9	153
Total Inferred	Inferred	4.67	3.7	559	1.5	220

Table 10. Grade Tonnage Distribution of 2004 Mineral Resource Estimate Sub-divided by Mineralised Zones and Resource Category

(12.5 mE x 12.5 mN x 5 mRL Size Blocks for Wall Zone and Overburden, and SMU emulation of 6.25 mE x 5 mN x 2.5 mRL Size Blocks for Upper Zone)

CIM Mineral Resource Category	Lower Cutoff (Au g/t)	Wall Zone Tonnes (Mt)	Wall Zone Gold (g/t)	Wall Zone Gold (Koz)	Wall Zone Silver (g/t)	Wall Zone Silver (Koz)	Upper Zone Tonnes (Mt)	Upper Zone Gold (g/t)	Upper Zone Gold (Koz)	Upper Zone Silver (g/t)	Upper Zone Silver (Koz)	Overburden Tonnes (Koz)	Overburden Gold (g/t)	Overburden Gold (Koz)	Overburden Silver (g/t)	Overburden Silver (Koz)	Combined Zones Tonnes (Mt)	Combined Zones Gold (g/t)	Combined Zones Gold (Koz)	Combined Zones Silver (g/t)	Combined Zones Silver (Koz)
Measured	0.5	0.95	7.4	227	4	115	-	-	-	-	-	-	-	-	-	-	0.95	7.4	227	4	115
	1.0	0.92	7.6	226	4	114	-	-	-	-	-	-	-	-	-	-	0.92	7.6	226	4	114
	1.2	0.91	7.7	225	4	113	-	-	-	-	-	-	-	-	-	-	0.91	7.7	225	4	113
	1.5	0.87	8.0	224	4	112	-	-	-	-	-	-	-	-	-	-	0.87	8.0	224	4	112
	2.0	0.82	8.4	221	4	110	-	-	-	-	-	-	-	-	-	-	0.82	8.4	221	4	110
Indicated	0.5	1.31	4.6	196	3	107	3.80	3.1	373	2	215	-	-	-	-	-	5.11	3.5	569	2	322
	0.7	1.23	4.9	194	3	105	3.06	3.7	359	2	198	-	-	-	-	-	4.28	4.0	553	2	302
	1.0	1.12	5.3	191	3	102	2.36	4.5	340	2	179	-	-	-	-	-	3.48	4.8	531	3	280
	1.2	1.05	5.6	189	3	100	2.05	5.0	330	3	169	-	-	-	-	-	3.10	5.2	518	3	269
	1.5	0.96	6.0	185	3	97	1.71	5.7	315	3	158	-	-	-	-	-	2.68	5.8	500	3	254
	2.0	0.84	6.6	178	3	92	1.33	6.9	294	3	143	-	-	-	-	-	2.17	6.8	472	3	234
Measured + Indicated	0.5	2.27	5.8	422	3	222	3.80	3.1	373	2	215	-	-	-	-	-	6.06	4.1	795	2	437
	0.7	2.17	6.0	420	3	219	3.06	3.7	359	2	198	-	-	-	-	-	5.22	4.6	779	2	417
	1.0	2.04	6.4	417	3	216	2.36	4.5	340	2	179	-	-	-	-	-	4.40	5.4	757	3	394
	1.2	1.95	6.6	414	3	213	2.05	5.0	330	3	169	-	-	-	-	-	4.01	5.8	743	3	382
	1.5	1.83	6.9	409	4	209	1.71	5.7	315	3	158	-	-	-	-	-	3.55	6.3	724	3	367
	2.0	1.67	7.5	399	4	202	1.33	6.9	294	3	143	-	-	-	-	-	2.99	7.2	693	4	345
Inferred	0.5	0.18	1.7	9	1	7	2.07	1.7	112	1	81	0.36	1.3	15	1	12	2.61	1.6	136	1	99
	0.7	0.12	2.1	8	1	6	1.39	2.2	99	1	64	0.28	1.4	13	1	10	1 .79	2.1	121	1	79
	1.0	0.07	3.0	7	2	4	0.87	3.1	85	2	49	0.21	1.7	11	1	8	1.15	2.8	104	2	62
	1.2	0.06	3.3	7	2	4	0.70	3.5	79	2	44	0.17	1.8	10	1	7	0.93	3.2	96	2	55
	1.5	0.05	3.7	6	2	4	0.55	4.2	73	3	39	0.12	2.0	8	1	5	0.72	3.8	87	2	48

(12.5 mE x 12.5 mN x 5 mRL Size Blocks for Wall Zone and Overburden, and SMU emulation of 6.25 mE x 5 mN x 2.5 mRL Size Blocks for Upper Zone)																					
CIM Mineral Resource Category	Lower Cutoff (Au g/t)	Wall Zone					Upper Zone					Overburden					Combined Zones				
		Tonnes	Gold		Silver		Tonnes	Gold		Silver		Tonnes	Gold		Silver		Tonnes	Gold		Silver	
		(Mt)	(g/t)	(Koz)	(g/t)	(Koz)	(Mt)	(g/t)	(Koz)	(g/t)	(Koz)	(Koz)	(g/t)	(Koz)	(g/t)	(Koz)	(Mt)	(g/t)	(Koz)	(g/t)	(Koz)
	2.0	0.04	4.4	6	2	3	0.39	5.1	643	3	3.3	0.05	2.4	3	1	2	0.47	4.8	73	2	3.8

7 Geological Setting and Mineralisation

The information set out in this section is taken from the 2012 Report, has been reviewed by the author of this Technical Report and remains current.

7.1 Regional Geology

The Krumovgrad region is located within the East Rhodope which comprises the eastern portion of a large metamorphic complex termed the Rhodope Massif. The massif underwent Upper Cretaceous extension leading to uplift and formation of the Kessebir metamorphic core complex. This event was accompanied low-angle detachment faulting, by graben development, and formation of sedimentary basins. The basins to the north of the Kessebir core complex contain Palaeocene terrestrial sediments that are transitional upwards into marine sediments. Figure 6 displays a plan of the regional geology of the Krumovgrad region.

Basement rocks in the Krumovgrad area consist of Precambrian and Paleozoic metasediments, gneisses, and amphibolites of the Kessebir metamorphic core complex. The basement is unconformably overlain by Tertiary (Paleocene-Eocene) conglomerates, sandstones, siltstones and limestones of the Krumovgrad Group. The basal Sharovo Formation, the primary host to gold-silver mineralisation within the Krumovgrad Licence area, consists of a chaotic accumulation of coarse to fine breccia and sands typically sourced from the adjacent basement, and regarded to have been deposited during rapid uplift of the Kessebir core complex.

Felsic to intermediate volcanism began in the Upper Eocene and progressed episodically until the Upper Oligocene. Several Pb-Zn (Au-Ag) epithermal vein deposits are related to volcanoes formed during this period including Zvezdel and Madjarovo, which are situated 15 km west and 25 km northeast of Krumovgrad respectively. More recent Neogene-Quaternary sedimentary cover occurs throughout the region.

The structural architecture of the Krumovgrad area reflects several stages of extensional deformation associated with uplift of the Kessebir core complex. Extensional faults generally strike E-W to NW-SE and dip shallow to steep towards the N-NE. Extension on these structures was accommodated by the formation of N-S to NE-SW striking, steeply dipping transfer faults. The contact between the core complex and the overlying sedimentary rocks of the Krumovgrad Group is commonly a shallow northeast-dipping fault which has been interpreted as a major district wide detachment. All prospects currently identified to date are located on or very close to the basement/sediment contact and are generally associated with topographic highs. Figure 6 displays the regional geology of the Krumovgrad area.

The 'detachment' structure has had a protracted history, initiating in the late Cretaceous and undergoing numerous stages of reactivation that pre-date and post-date identified epithermal mineralisation in the Krumovgrad area. The most conspicuous stage is also the youngest, evident in diamond drill core as a metre-scale, poorly indurated cataclasite that exhibits well developed fabric asymmetries indicative of non-coaxial shearing.

The final stage of extensional deformation associated with evolution of the Kessebir Dome is represented by a N-S to NW-SE trending graben. The orientation of the graben, and of faults that cross-cut it suggest that the extension direction for graben opening has also been oriented approximately N(E)-S(W), similar to that during pre-graben extension.

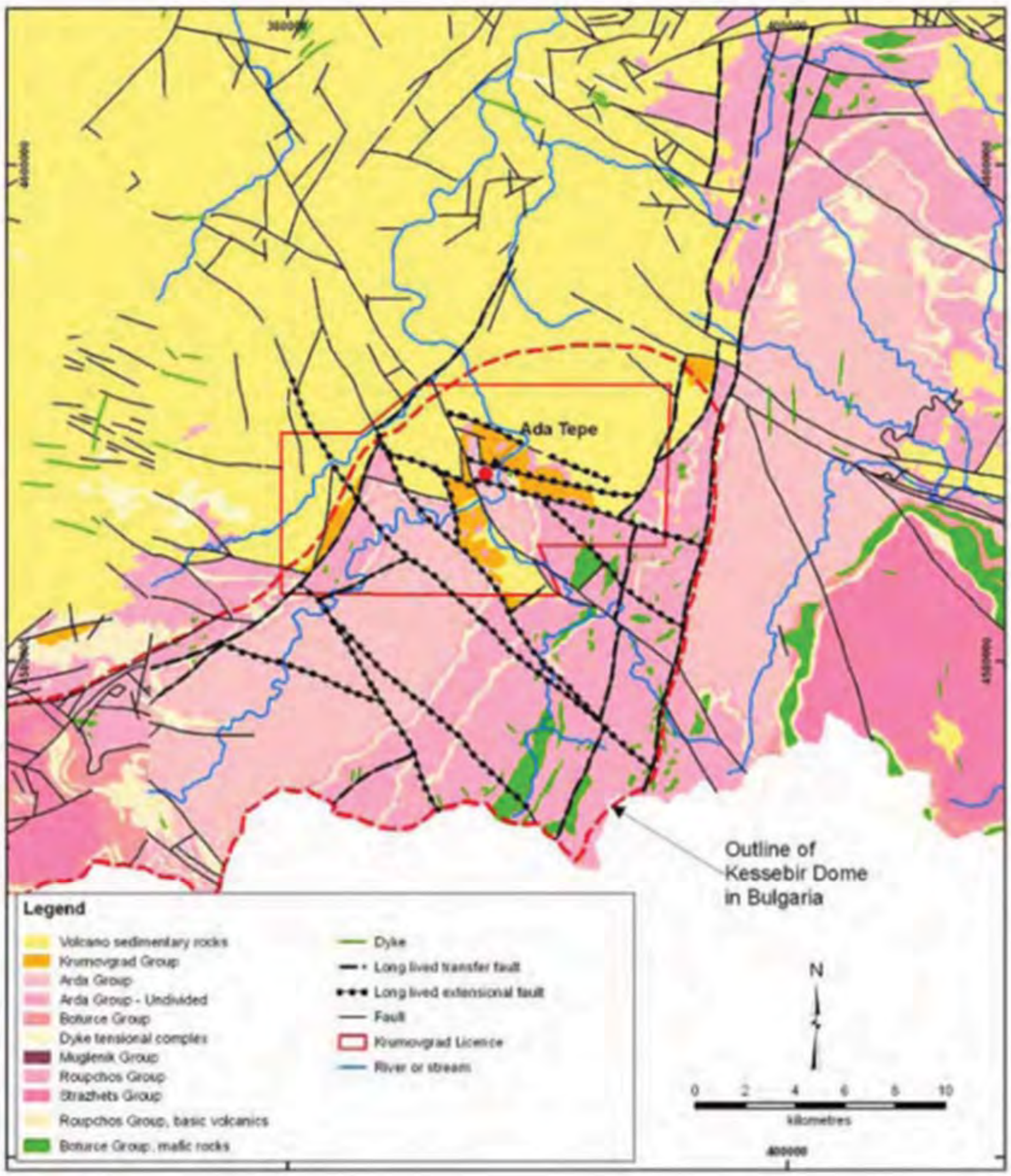

Figure 6. Regional Geology of Krumovgrad Area

7.2 Local Geology and Mineralisation Controls

Gold and silver mineralisation in the Krumovgrad Licence area is predominantly hosted within the Sharovo Formation proximal to the unconformable fault contact or 'detachment' with the underlying basement rocks of the Kessebir core complex. Sediments within the Sharovo Formation typically form laterally discontinuous lenses ranging from coarse breccia to fine sands with variable clay content. Upward variations in the stratigraphy of the Krumovgrad Group reflect progression from a high-energy environment, breccia-conglomerates and coarse sandstones through to the lower energy siltstones and limestones characteristic of increasing basin maturity. The location of the Ada Tepe deposit and other prospect areas in Krumovgrad Licence are displayed in Figure 7.

The dominant structure at the Ada Tepe deposit is the 'detachment' structure that separates the Kessebir core complex rocks (basement) from the overlying sedimentary rocks, which forms a 10° to 15° north dipping lower structural bounding surface to the deposit. The deposit is bound to the north and south by approximately NE-SW striking, steep dipping faults.

Review by RSG Global in 2004 suggests that the second-order structural control for mineralisation, after that of the detachment, is the proximity of NE-SW transfer faults. These structures dip steeply, allowing more direct access of fluids from deeper levels than the shallow dipping extensional structures. As such, the shallow extensional structures may represent trap sites or structures that accommodate local lateral fluid flow away from the transfer structures. Closely spaced transfer structures may also be important for localizing mineralisation as the presence of shear couples can enhance brecciation of the intervening rock, or may act to produce tensional sites. For example, dextral shear on the NW-SE striking faults bounding Ada Tepe may have been responsible for facilitating epithermal vein emplacement within E-W tensional sites.

Another important control on the localisation of mineralisation is exhibited by the sedimentary rocks overlying the detachment. The Shavarovo Formation is a highly porous, poorly sorted breccia that is bound on its contact with the Kessebir core complex by the detachment. The porosity of this unit is interpreted as a second-order control in localising fluid flow.

Mapping and structural data from diamond drill core indicate that the latter stages of movement along the 'detachment' structure at Ada Tepe post-dates emplacement of epithermal mineralisation and were responsible for translation of the hangingwall sediments (and deposit) towards 030°. Consequently, any basement feeder structures are interpreted as lying to the south and may have been eroded or are concealed beneath later graben sediment fill.

The Ada Tepe deposit is approximately 600 m long (N-S), and up to 350 m wide (E-W). The wall zone is up to 30 m thick. The thickness of the Upper Zone vein mineralization is very variable, from less than 1 m thick, to more than 30 m thick. The Wall Zone exhibits very good continuity. The Upper Zone vein system exhibits less continuity than the Wall Zone, necessitating the higher drilling density that has been applied during the delineation of the Ada Tepe deposit.

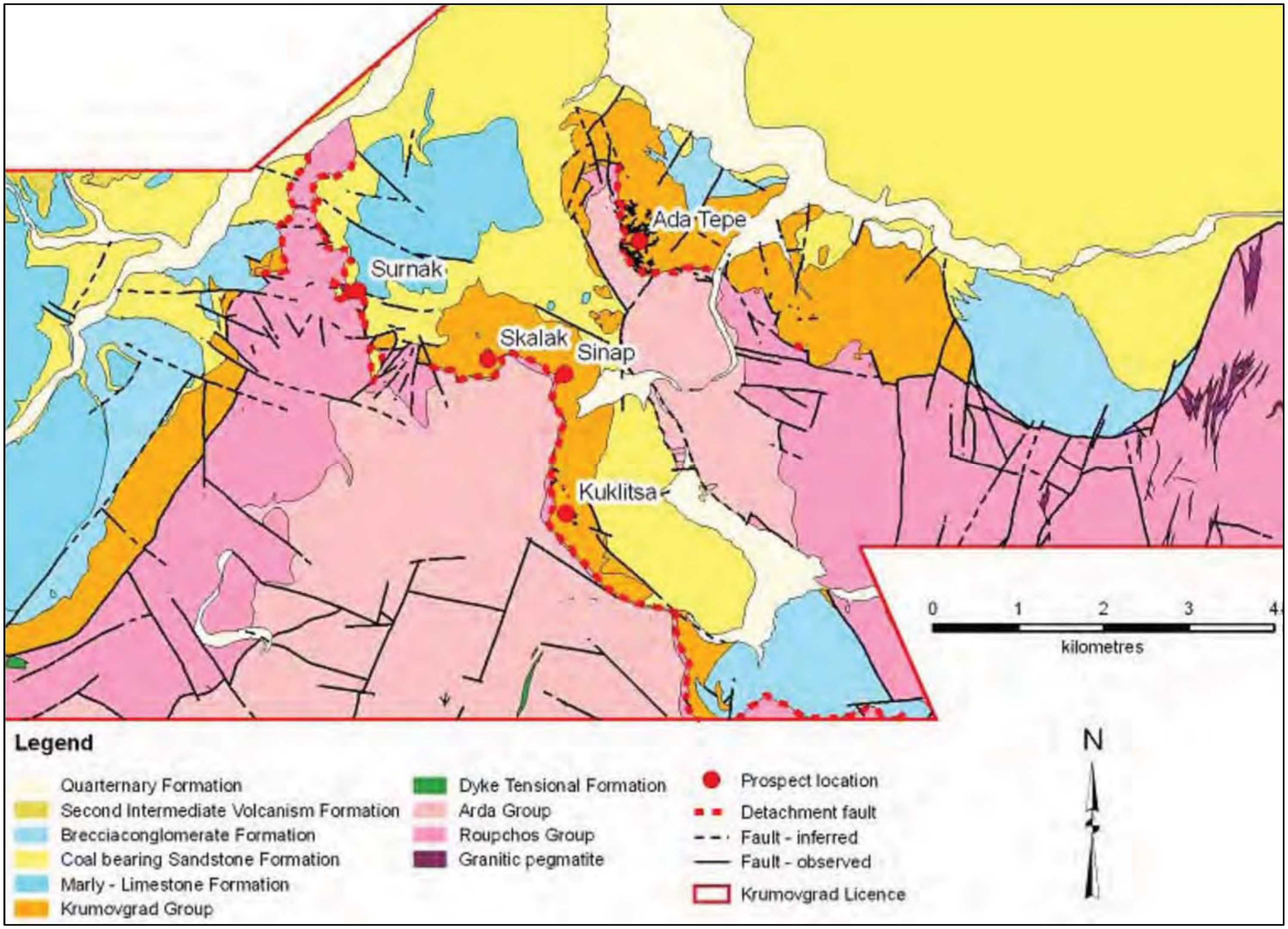

Figure 7. Krumovgrad Project Geology

8 Deposit Types

The information set out in this section is taken from the 2012 Report, has been reviewed by the author of this Technical Report and remains current.

8.1 Deposit Setting

The Ada Tepe deposit is a prime example of a high level epithermal gold-silver deposit, formed during the Neogene within the Southern Rhodope tectonic zone. It is characterised as a low-sulphidation epithermal gold silver deposit. These deposit types are common throughout the world and form in association with volcanic arcs along subduction zones on plate boundaries. Epithermal gold-silver deposits are often associated with deeper porphyry-related copper-gold mineralisation. Major porphyry-epithermal belts elsewhere include the Carpathian Belt in Europe, the Andes Mountains in South America and the Indonesian Archipelago.

The Ada Tepe deposit is a low-sulphidation adularia-sericite gold-silver epithermal deposit located within Palaeocene sedimentary rocks overlying the northeastern end of the Kessebir core complex. Two major styles of mineralisation are apparent at Ada Tepe:

- Initial stage of mineralisation hosted by a massive, shallow-dipping (15o north) siliceous body forming the hangingwall to the detachment and defining the contact between the core complex and overlying sedimentary rocks. This mineralisation is termed the "Wall Zone" by local geologists and displays multiple stages of veining and brecciation.

- Second phase of mineralisation represented by steep dipping veins that exhibit textures indicative of formation within an epithermal environment. These veins have a predominant east-west strike, cross-cut the shallow-dipping siliceous Wall Zone mineralisation and extend upwards into the sedimentary breccia unit above the Wall Zone. This phase of mineralisation has been locally termed the "Upper Zone".

The initial stage Wall Zone mineralisation is interpreted to be associated with early silica flooding and relatively low gold grades. However regions of the Wall Zone through which well-developed Upper Zone vein mineralisation passes are typically thicker, more intensely brecciated and contain epithermal vein and hydraulic breccia infill textures and associated high gold grades not present in regions where Upper Zone vein mineralisation is absent. These thick strongly continuous regions of high grade Wall Zone mineralisation generally thin and diminish in grade away from and between regions of well-developed Upper Zone vein mineralisation.

Typical epithermal textures present at the Ada Tepe deposit include the following:

- Crustiform and colloform banding;

- Chalcedonic banding;
- Bladed silica replacement textures after carbonate;
- Compositionally zoned crystals;
- Hydraulic breccia textures;
- Late-stage carbonate veins

The textural style and grade of mineralisation at Ada Tepe, high grades in association with open-space fill textures, such as bladed silica replacement after carbonate (i.e. evidence of boiling), hydrothermal breccias and also the presence of sinter material, suggests proximity to the paleosurface and a low-sulphidation nature of mineralisation.

9 Exploration

The information set out in this section is taken from the 2012 Report, has been reviewed by the author of this Technical Report and remains current.

9.1 Exploration History

In the early to mid-1990s, GeoEngineering of Assenovgrad carried out an extensive programme of geological mapping, trenching and drilling over the nearby Surnak prospect together with a minor trenching on the Skalak and Kuklitsa prospects. Geology & Geophysics included the entire licence area in a SE Rhodopes regional soil sampling programme (average sample grid 250 m x 50 m) conducted during the early-mid 1990s. Magnetic and IP surveys were also conducted across the prospect. The results of this early work showed the presence of gold soil geochemical anomalies of significant intensity and extent over the Krumovgrad License area, and a variety of geophysical anomalies.

Since June 2000, BMM has conducted detailed exploration of the Ada Tepe prospect. This exploration included the establishment of more accurate survey control over the licence area, surveying of the surface topography, detailed surface trenching and channel sampling. The programme also included diamond and RC drilling, rock chip sampling, bulk density measurements, and detailed geological mapping. To date, the drilling forms a notional 25 mN by 25 mE grid over the entire deposit, and closer spaced drilling on a notional 12.5 mN by 12.5 mE grid has been completed over two rectangular sub-regions of the deposit.

Sub-surface mineral resource delineation at the Ada Tepe deposit has been undertaken by a combination of RC and diamond drilling, completed in four drilling programmes between late 2000 and late 2004. In addition to this extensive trenching has been undertaken.

10 Drilling

The information set out in this section is taken from the 2012 Report, has been reviewed by the author of this Technical Report and remains current. No additional resource development drilling has been completed over the project since that disclosed in the 2012 Report.

10.1 Summary

The initial drilling programme, undertaken in 2000 and 2001, comprised 74 diamond drillholes completed on a notional 50 m by 50 m grid. Most holes were declined 65° towards the southwest (230° azimuth), and remaining holes were drilled vertically or declined towards the northeast. The drilling was completed by BMM, Bulgarian drilling contractors GEOPS, and Romanian-based drilling contractors RB Drilling, predominantly using Boyles-BBS37, CKB-4 and BOBY-150 coring rigs. Mostly HQ (78%) and to a lesser extent NQ (20%), and minor PQ (2%) size core was collected.

The second drilling programme was carried out in May through August 2002 and included 17 diamond drillholes and 54 RC drillholes mostly declined 60° towards the south along grid north-south orientated drill traverses on a notional 50 m by 50 m pattern. The drilling orientation was changed from the grid southwest orientation used in the initial programme to optimise drill intersections in the predominant east-west trending veins in the Upper Zone and shallow north dipping Wall Zone mineralisation. All drilling in the second programme was completed by a Bulgarian drilling contractor, International Drilling Services using DT1000 and CM1200 multipurpose rigs, with HQ (62%) and HQ-3 (38%) size core collected. All RC drilling was completed using a 125 mm face sampling hammer drill bit.

The third and most substantial drilling programme was undertaken between September 2003 and June 2004, by Drilling Services Bulgaria (formerly International Drilling Services) using DT1000 and CM1200 multi-purpose rigs, and by GEOPS using Diamec282 and Boyls-BBS37 coring rigs. The programme comprised 137 diamond holes (including 94 completely cored and 35 diamond tail mineral resource holes, 5 'wild cat' exploration holes and 8 metallurgical holes) and 333 RC holes (including 298 complete mineral resource holes and 35 pre-collar holes). This programme resulted in a notional drilling density of 25 mE by 25 mN over the majority of the deposit, with most of the holes declined 60° towards the south and several scissor holes declined 60° towards the north and north west. In addition, RC infill drilling was completed to a notional 12.5 m by 12.5 m hole spacing in two selected areas in the south-western and central- western regions of the deposit to allow investigation of the close spaced variability of gold and silver assay grades. The diamond drilling collected PQ-3 (23%), HQ-3 (57%), HQ (4%) and NQ-3 (16%) size core, while all RC drilling was completed using a 125 mm face sampling hammer drill bit.

The fourth drilling programme was undertaken between late October 2004 and mid November 2004 by Drilling Services Bulgaria using a DT1000 multi-purpose rig. The programme comprised 36 RC drillholes designed to selectively infill strongly mineralised zones within the southern third and to a lesser extent the northern flank of the deposit. All drilling was completed using a 125 mm face sampling hammer drill bit, with drillholes inclined 60° towards northerly and southerly directions, and a variety of scissor orientations.

Table 11 provides a summary of the exploration applied to mineral resource estimation. Details of the procedures and equipment used to maximise core recovery and the quality of RC samples are subsequently provided in Section 10.2. The location of the drillhole collars, colour coded by drilling programme and method is displayed in Figure 8. All drilling at Ada Tepe has been completed by reputable Bulgarian and Romanian based drilling contractors using appropriate equipment.

Table 11. Summary of Exploration Drilling and Channel Sampling Used for Mineral Resource Estimation

Exploration Programme	Method	Type	Num Holes / Trenches	Average Length	Total Metres	Number of Assays
1 (2000-2001)	DH Trench	Complete N/A	70 200	83.7 34.1	5,856.3 6,824.7	5,423 4,319
2	RC	Complete Precollar	53 1	84.0 94.0	4,452.0 94.0	4,513
(2002)	DH	Complete	17	96.9	1,646.8	1,642
	Trench	N/A	77	51.8	3,988.0	3,507
3 (2003-Sept 2004)	RC	Complete Precollar	299 34	80.8 92.2	24,167.0 3,134.1	26,901
	DH	Complete Tail	102 35	101.3 32.1	10,336.4 1,125.0	10,861
	Trench	N/A	148	50.6	7,487.1	6,944
4 - (Sept to Nov 2004)	RC	Complete	36	58.2	2,095.0	2,065
Combined	RC	Complete Precollar	388 35	79.2 92.2	30,714.0 3,228.1	33,479
	DH	Complete Tail	189 35	94.4 32.1	17,839.5 1,125.0	17,926
Total	Trench		425	43.1	18,299.8	1 4,770
	Drilling		612	86.4	52,906.6	51,405
	Trench		425	43.1	18,299.8	1 4,770
	Combined		1,037	68.7	71,206.4	66,175

Note: additional trench data was collected in November and December 2004, however this comprised extension to existing trenches.

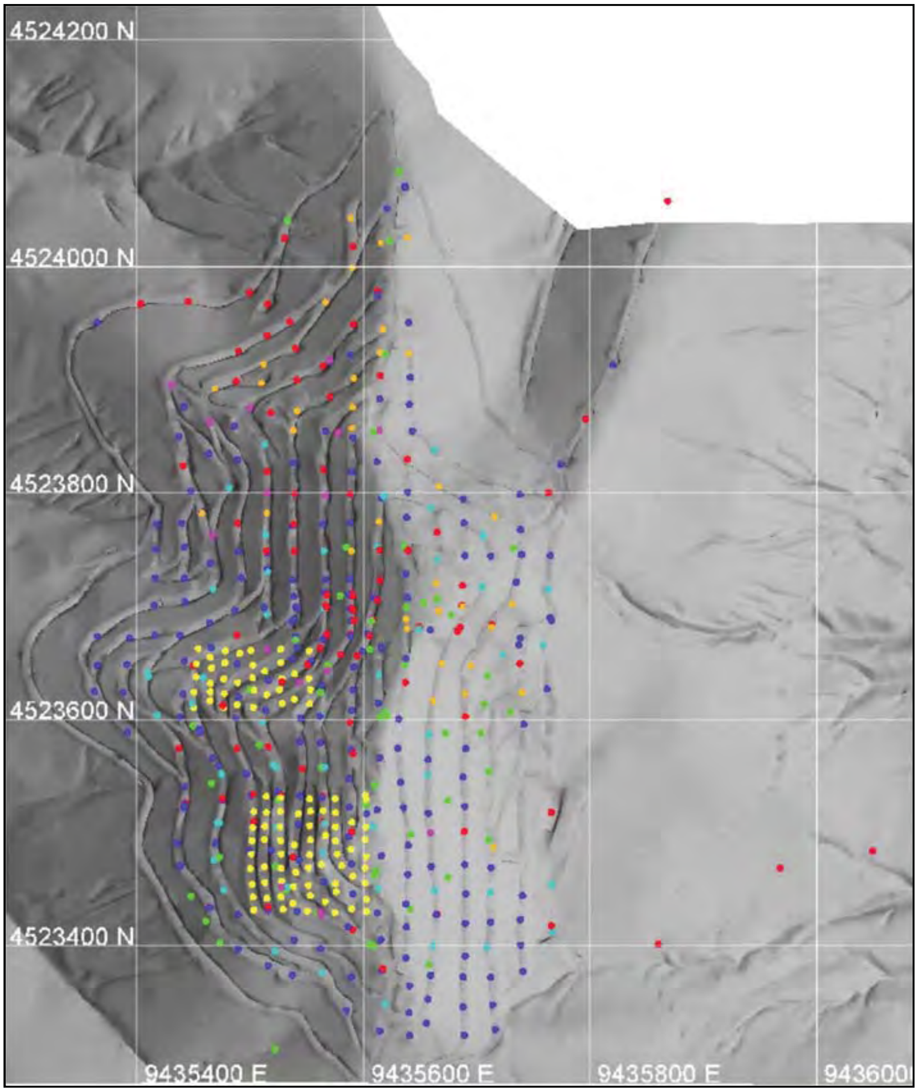

Figure 8. Drillhole Collar Locations

Navan : DDH [2000-01] Green, DDH [2002]-Magenta, RC [2002] – Light Blue
Dundee: RC Dark Blue [2003-04] and Yellow [2004], DDH [2004] Red and Orange

10.2 Sampling

The information set out in this section is largely taken from the 2012 Report. This information has been reviewed and some checks made by the author of this Technical Report and information remain current. CSA comment and observations from the 2012 site visit to the project are included in this section where relevant. There are some minor discrepancies noted between information summarised by RSG and that derived from the database and following CSA review, however these are considered minor and are not material or significant so as to have introduced bias or bring in to question the validity of sample data.

10.2.1 Introduction

From June 2000 until March 2002, all exploration data collection at Ada Tepe was undertaken by BMM, under the upper management of Navan. From April 2002 to the end of 2004, exploration at Ada Tepe was undertaken under the management of RSG Global in close consultation with BMM field staff, Navan upper management until 30 September 2003, and subsequently DPM upper management.

10.2.2 Channel Sampling

Trenches and drill access road cut exposures were routinely channel sampled since the commencement of detailed exploration at Ada Tepe in mid-2000. The channel sampling was undertaken predominantly on N-S orientated traverses coinciding with the 25m spaced drill traverses (Figure 9). Prior to March 2002, a variety of sample intervals were used, primarily controlled by changes in geology. In April 2002, RSG Global initiated the use of a standard RSG Global channel sampling method which is summarised below:

- All surfaces to be channel sampled are cleaned of loose debris prior to beginning sampling;
- The channel sampling line and channel interval (1 metre) for each sample are marked up, using spray paint, by geologists prior to the initiation of sampling.
- Each channel sample is chiselled out over standard width and depth to avoid sampling bias due to variations in rock hardness.
- Channel samples are routinely weighed to ensure that a constant sample weight of approximately 3 kg is collected (approximating half HQ core).
- A duplicate channel sample, located approximately 20cm to 25cm above the standard channel run is routinely collected over 5% of the sample intervals to enable statistical assessment of sampling errors.

Some 425 surface channels have been excavated at Ada Tepe from which a total of 14,770 channel samples have been collected representing a total of 18,299.8 m of sampling.

Weights were recorded for 8,988 channel samples. While there is considerable variation in the sample weights for the range of sample interval lengths (Figure 10), the sample weights generally increase with increasing interval lengths consistent with 3 kg samples being collected over 1 m intervals. In addition, there is no evidence of any bias in the sample gold grades relating to the variations in the sample weights (Figure 11).

CSA verified the location of some trench and drill access road cut exposures during the 2012 site visit and was able to confirm the positions of some sampling points.

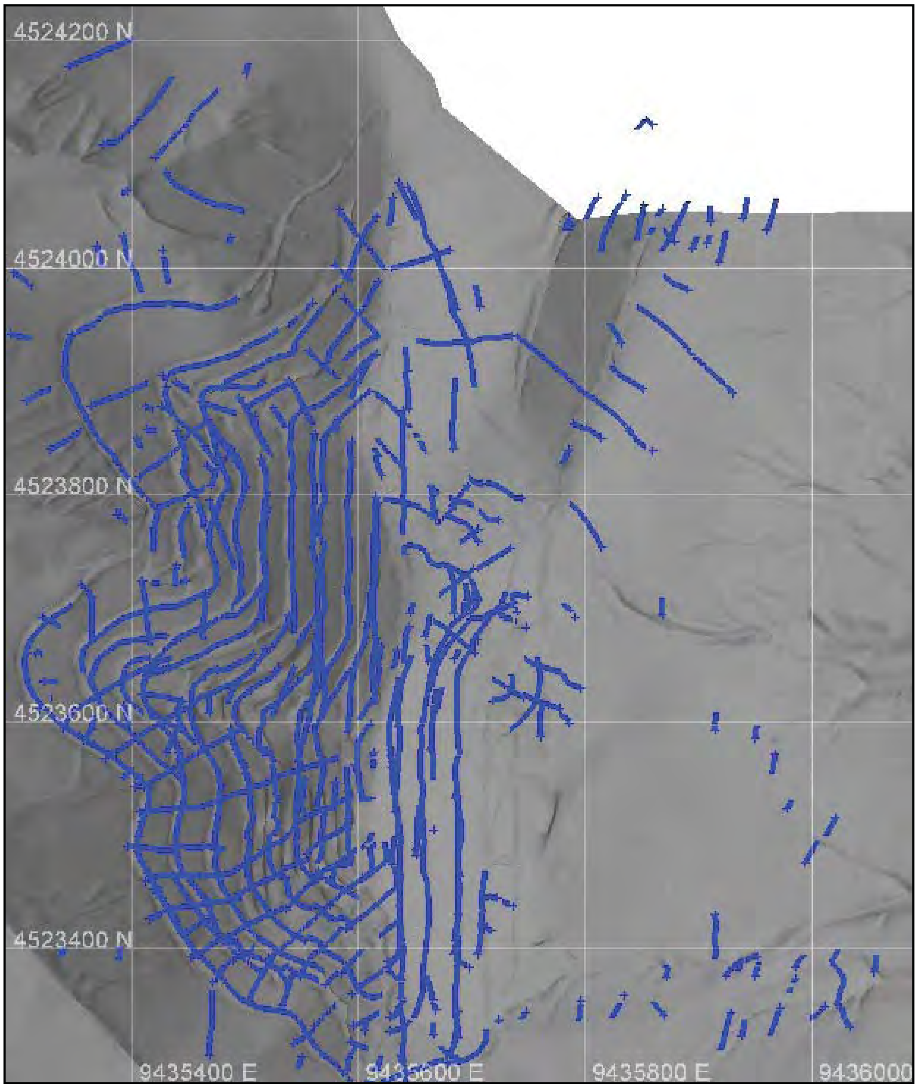

Figure 9. Ada Tepe Trench and Channel Sampling Locations

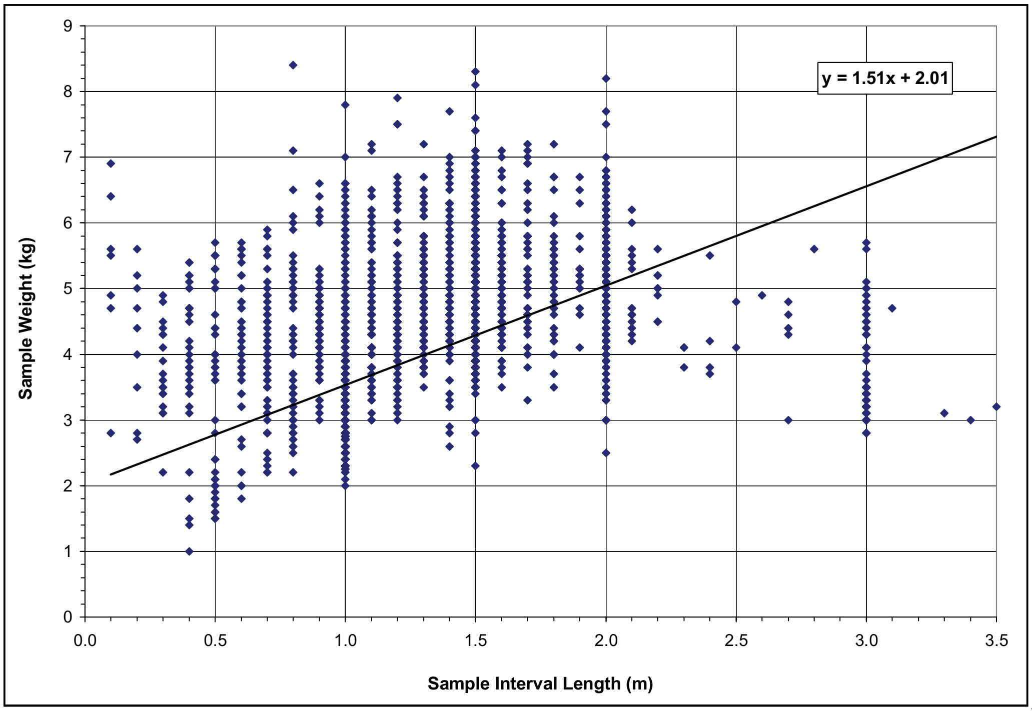

Figure 10. Channel Sample Interval Lengths versus Sample Weights

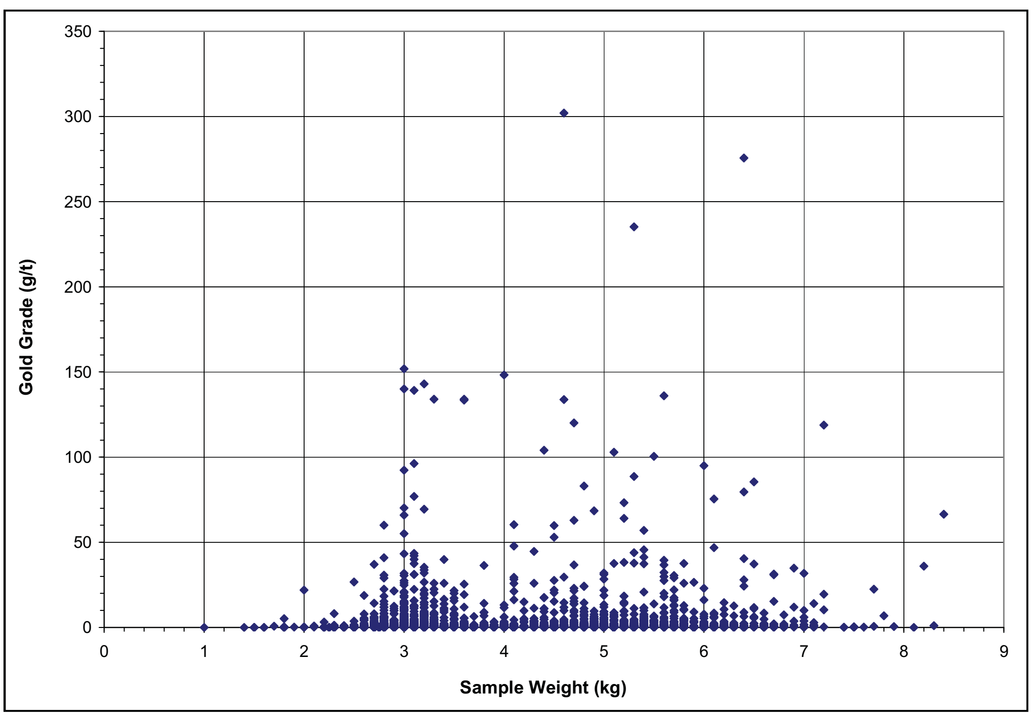

Figure 11. Channel Sample Gold Assay Grades versus Sample Weights

10.2.3 RC Drilling

RC samples were routinely collected at 1m intervals and the cuttings split with a Jones riffle splitter. The bags of cuttings were routinely weighed prior to taking the sub-sample via the Jones riffle splitter. The RC sampling process as applied during drilling at Ada Tepe is summarised in the flow sheet presented as Figure 12.

Stringent quality control procedures have been used for all RC drilling completed at Ada Tepe including:

- Use of face sampling hammers.
- The collar position of each hole was conditioned with a 3 m to 6 m casing of PVC.
- Each metre sample was collected in a fresh plastic bag labelled with hole ID and depth (from-to). On completion of the 1 m drill advance, each RC drill sample was weighed and the weight recorded.
- Routine blowbacks were undertaken (i.e. each metre is cleared into the sample bag prior to moving on to the next metre of advance).
- The cyclone was cleaned at each rod change and at the first occurrence of basement lithologies.
- On a rod change any material in the hole was cleared before the first new metre sample is collected.
- The riffle splitter is cleaned with compressed air between each sample (using compressed air guns attached to the drilling rig).
- Duplicate field samples were collected every 20 m by riffle splitting the bag of cuttings once more.
- On the extremely rare occasions that moisture was encountered in the hole (note the emphasis on moisture rather than wet drilling conditions), the hole was 'conditioned' or dried prior to advancing further.
- Water misting systems were used to suppress dust from the cyclone chimney and the outside return hose.
- Additional compressed air boosters were routinely used to enhance RC sample recoveries and to ensure that samples below the water table remained dry.

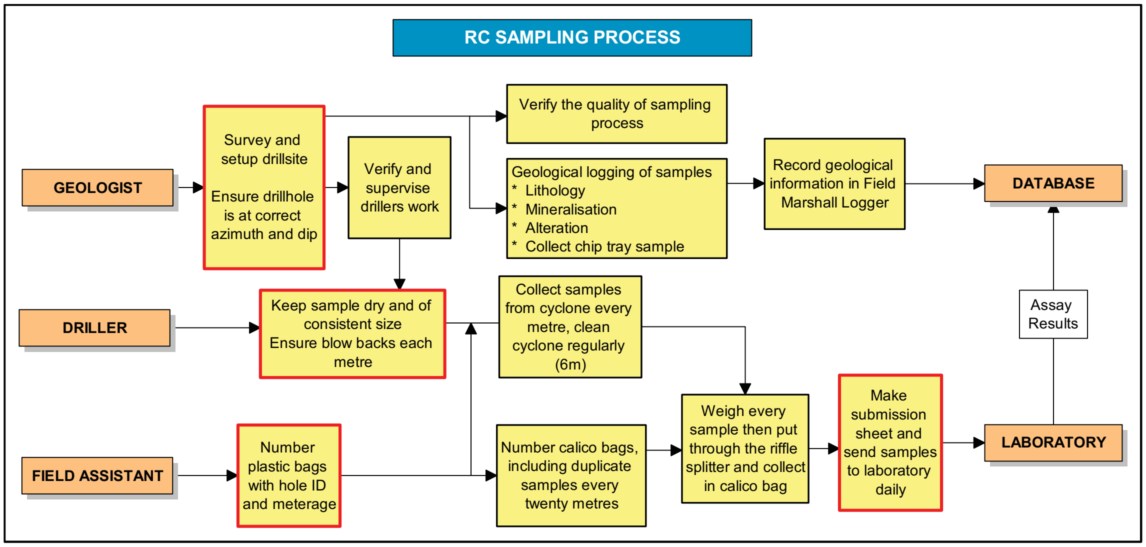

Figure 12. RC Sampling Process

The RC drilling was supervised at all times by a well trained geologist, assisted by 4 field assistants, who operated the sample splitter and weighed each metre sample bag. A representative portion of RC cuttings from each successive metre was also sieved clean and stored in neatly labelled chip trays.

It is the opinion of CSA that the RC drilling and associated sampling was completed to high industry standards. This opinion is informed by a review of data collection procedures, protocols and metadata contained in the database for the project, as reviewed during the 2012 site visit.

RC drill sample weights were routinely measured as part of the standard RSG Global RC drilling procedures. Statistical analysis was undertaken based on 3 m composites of the RC sample weight data for the deposit host breccia conglomerate unit converted to percent recovery data using theoretical sample weights of approximately 26 kg (per metre) for strongly to moderately oxidised conglomerate, 29 kg for weakly oxidised conglomerate, and 30 kg for fresh conglomerate.

Average RC sample recoveries ranged from 70% in strongly to moderately oxidised conglomerate to 75% in fresh conglomerate, and average 72% for all RC intersections of conglomerate. The trend of increasing recovery with decreasing oxidation is also reflected in the core recovery data (discussed in Section 10.2.4). It was the opinion of RSG Global that in its experience of using both RC and diamond drilling for mineral resource delineation, average theoretical RC sample recoveries are typically 20% less than the average core recoveries. This was supported for the Ada Tepe deposit by comparing theoretical RC sample percent recovery data with corresponding core recovery data for 24 successful RC-diamond twin hole pairs representing approximately 1,865 m of comparative drilling. The average 73% theoretical sample recovery for the RC drilling is 21% less than the achieved average core recovery of 92%.

The average RC sample recoveries achieved in the different oxidation zones at the Ada Tepe deposit are consistent with those achieved by diamond drilling, and are considered to be of industry accepted standards for the ground conditions and styles of mineralisation in the deposit. Figure 13 displays a scatter plot of the RC sample weights versus gold grade. There is no evidence that anomalously low or high sample weights are associated with high (or low) gold grades.

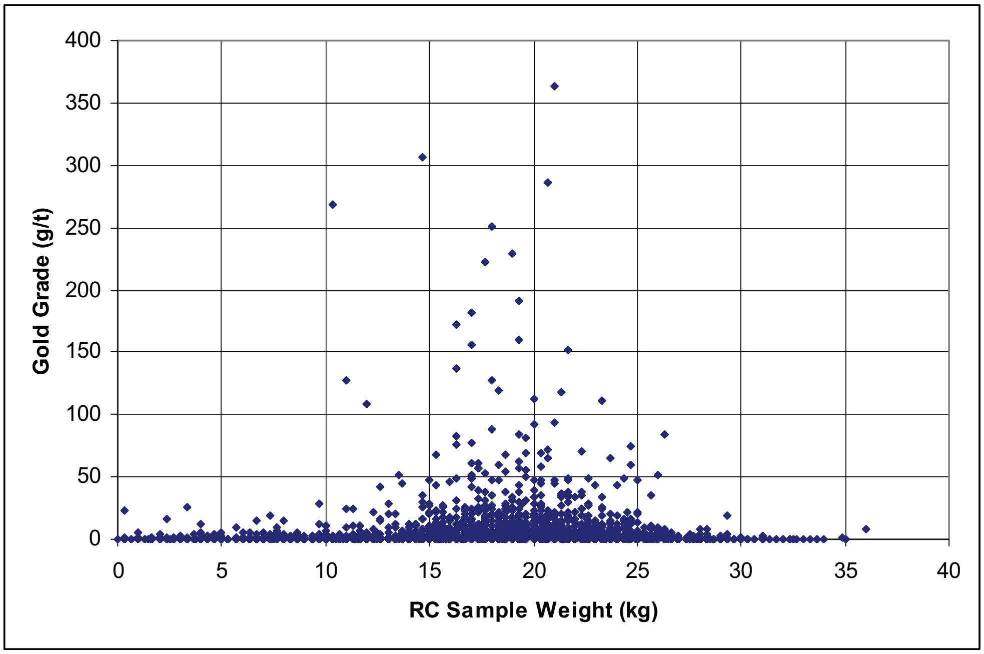

Figure 13. RC Sample Weight versus Gold Grade

10.2.4 Diamond Drilling

The diamond drilling and core sampling procedures employed for mineral resource delineation at the Ada Tepe deposit are summarised in Figure 14. Similar to the RC drilling, the diamond drilling at Ada Tepe has been performed using strict quality control procedures. The key technical considerations used for the majority of the diamond drilling at Ada Tepe are as follows:

- The majority of drilling was completed using 1.5 m PQ and HQ triple tube, however, core diameter reduction to NQ triple tube was used if ground conditions were deemed unstable. Where possible, PQ core was used to drill approximately 5 m past the base of oxidation, there after the hole was continued with HQ.

In cases of poor core recovery, the drill runs were reduced from 1.5 m to less than 0.5 m.

- Specialised drilling muds and polymers were routinely used to maximise core recovery.

- Triple tube splits and core lifters were washed prior to reuse in successive drill runs.

- Drill core was orientated every 3 m using the spear method.
- Wooden core blocks were placed between runs, recording the length of the run and/or core loss.
- Forced breaks made by the drillers were marked on the core with a red cross.

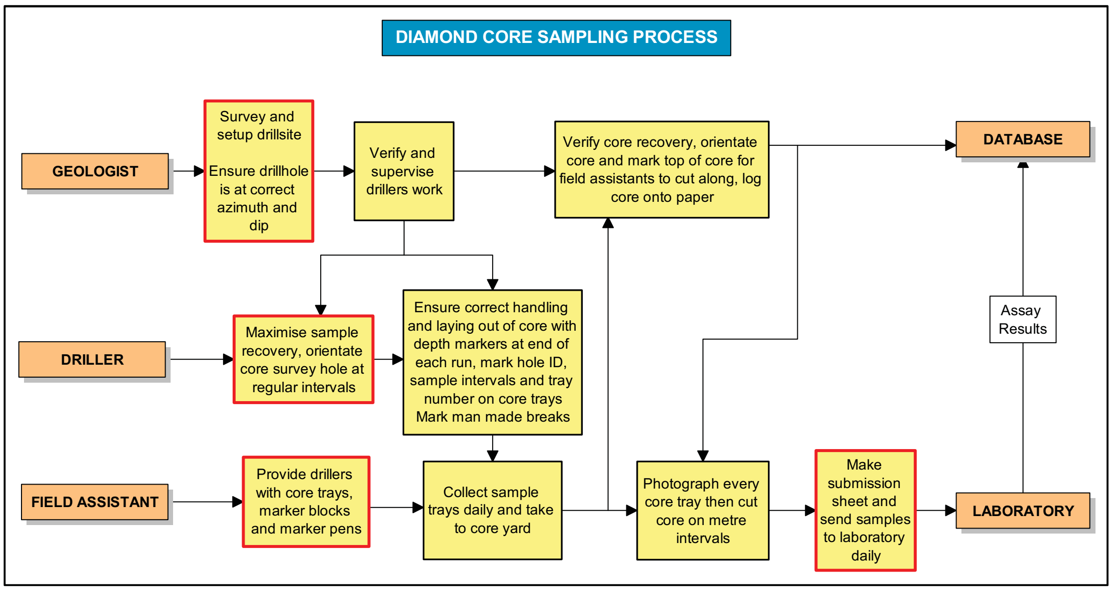

Figure 14. Diamond Core Sampling Process

The diamond core was marked up with a longitudinal line coinciding with the orientation mark when available, or otherwise perpendicular to the dominant structural fabric in the core. The core was then marked off predominantly at 1 m intervals for sampling and then cut in half lengthways using diamond core saws. Mostly half core samples have been submitted for sample preparation and laboratory analysis. Field duplicates were produced from the same half-core sample following jaw crushing at a frequency of 1 in 20 samples.

Statistical analysis was undertaken based on 3 m composites of the core recovery data for the host breccia conglomerate unit. Average core recoveries range from 89% in strongly to moderately oxidised conglomerate to 99% in fresh conglomerate, and average 94% for all diamond core intersections of conglomerate, as presented in Table 12. It is considered that these recoveries are good for the ground conditions at Ada Tepe.

Table 12. Summary Statistics of 3m Composite

Percentage Core Recovery Data for Breccia Conglomerate				
	Completely and Moderately Oxidised	**Weakly Oxidised**	**Fresh**	**Combined**
Number	2,533	836	1,744	5,113

Percentage Core Recovery Data for Breccia Conglomerate				
	Completely and Moderately Oxidised	Weakly Oxidised	Fresh	Combined
Minimum	0	0	20	0
Maximum	100	100	100	100
Mean	89	97	99	94
Median	97	100	100	100
Std Dev	18	11	4	15
Variance	337	122	18	216
Coeff Var	0.21	0.11	0.04	0.16

Figure 15 displays a scatter plot of percentage core recovery versus gold grade for the diamond drilling at Ada Tepe. There is no evidence that low core recoveries are associated with high gold grades and in fact, the opposite appears to be the norm.

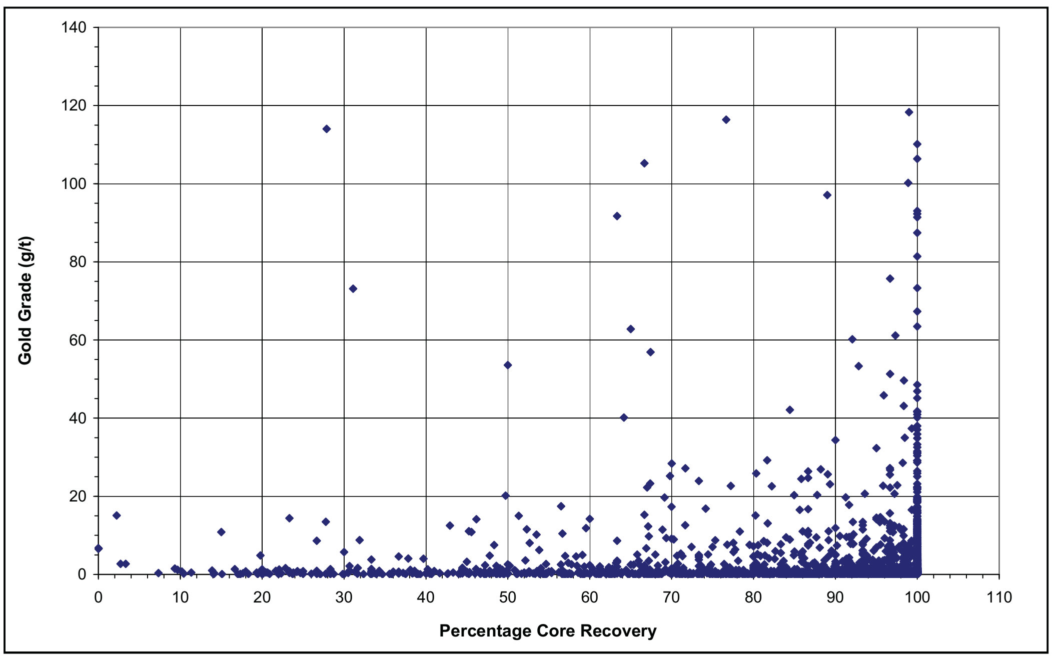

Figure 15. Percentage Core Recovery versus Gold Grade

11 Sample Preparation, Analysis and Security

The information set out in this section is taken from the 2012 Report, has been reviewed by the author of this Technical Report and remains current. CSA comment and observations from data reviews completed in November 2012 and March 2014, are included in this section where relevant.

Sample preparation and assay of analysis samples from the Ada Tepe Deposit has been carried out at two principal independent internationally accredited laboratory firms:

- OMAC Laboratories Ltd ("OMAC") of Ireland (2000 - 2001).
- SGS Laboratories of Perth (Welshpool), Western Australia, Gura Rosiei (near the Rosia Montana mine site), Romania and Chelopech, (part of Chelopech Mine) Bulgaria (2002 - 2004).

In addition, umpire assay analyses of approximately 5% of the routine exploration samples from the second and third exploration programmes were performed by two internationally accredited laboratories:

- Genalysis Laboratory Services, Maddington, Western Australia, Australia (2002 and 2004); ISO9002:1984 and ISO17025.
- ALS Chemex, Vancouver, British Columbia, Canada (2004); ISO9001:2000 and ISO17025.

Samples from the 2000 and 2002 trench sampling and drilling were transported either to the OMAC or SGS Gura Rosiei facilities for both sample preparation and analysis.

Initiation of the 2003 drilling and associated trench sampling included the establishment of an SGS sample preparation facility in Krumovgrad within a fully secured and enclosed core farm and RC sample storage facility, with 24 hour security. With the exception of the first 600 RC samples from the 2003/04 drilling programme (transported to the SGS Gura Rosiei facility for both sample preparation and analysis), all subsequent samples from the third and fourth drilling programmes underwent sample preparation at the SGS facility in Krumovgrad, and subsequent transport to the SGS Gura Rosiei (Romania), SGS Welshpool (Western Australia) or SGS Chelopech (Bulgaria) laboratories for assay analysis.

The following routine procedures were used to prepare the trench, RC drilling, and core samples for analysis:

- Dry samples at 105°C.
- Jaw crush core and trench samples to minus 6 mm.
- Pulverize all samples in a LM5 crusher to 95% passing 75 ^m. Complete sieve analysis on 1 in 20 samples.
- Clean bowl and puck of the LM5 with compressed air after each sample, and with a barren flush after every 20th sample, or as required to remove residue build-up.
- Complete barren flushes after BMM specified samples anticipated to contain high grade mineralisation.

A pulp library is maintained of all samples prepared by SGS, Krumovgrad, which are stored in a locked room in the BMM core shed.

CSA observed this pulp library facility during the 2012 site visit and performed random spot checks of sample numbers and compared these with data contained in the project database. No issues were detected.

Description of the assay techniques and the quality control used by the primary and umpire laboratories during each the following periods of exploration activity are provided in Table 13 and Table 14.

- Programme 1: Navan Mining PLC, 2000 - 2001 exploration drilling and trenching.
- Programme 2: Navan Mining PLC, 2002 drilling and trenching.
- Programme 3: Dundee Precious Metals EAD, 2003-2004 feasibility study drilling and trenching.
- Programme 4: Dundee Precious Metals EAD, November 2004 feasibility study additional infill RC drilling and trenching (note, additional trenching completed during field season of Programme 3).

Additional quality control employed by BMM for external monitoring of the precision and accuracy of the assay analyses completed during exploration programmes 2 and 3 include:

- Submission of internationally accredited gold and silver standards, produced by Rocklabs of New Zealand, routinely inserted into the sample stream at a frequency of 1 in 20 routine exploration samples.
- Insertion of a blank sample (beach sand) before the first sample of each drillhole from March 2004 onwards.

- Routine collection of duplicate RC drill sample splits at a frequency of 1 in 20 routine samples.

- Routine collection of duplicate channel samples at a frequency of 1 in 20, approximately 20 cm above the primary channel sample location.

- Collection of a duplicate sample split after jaw crushing of trench and core samples at a frequency of 1 in 20.

Summary details of the gold and silver analyses completed on routine exploration samples from Ada Tepe are provided in Table 15 and Table 16. RSG Global made the following comments on the basis of these tabulations compiled from the supplied exploration database:

- A total of 66,427 successful gold analyses and 62,067 silver analyses have been completed on routine exploration samples from Ada Tepe (excluding any QA/QC samples).

- Most sample analyses have been completed at the SGS Gura Rosiei lab (gold - 61%, silver - 60%), followed by SGS Welshpool (gold - 32%, silver - 34%), followed by OMAC (gold - 5%, Silver - 2%), followed by SGS Chelopech (gold - 3%, Silver - 3%).

- Not all samples were analysed for silver in the initial exploration programme:

    - None of the trench samples (1,954) submitted to OMAC were analysed.

    - 72% of the trench samples (1,707) submitted to SGS Gura Rosiei were not analysed.

    - 15% of the core samples (627) submitted to SGS Gura Rosiei were not analysed.

    - 5% of the core samples (66) submitted to OMAC were not analysed.

CSA notes that there are only four core samples (< 0.3% total) in the database that were submitted to OMAC and not analysed for silver.

CSA completed checks of information summarised above and that tabulated below. Minor discrepancies were noted but CSA considers these neither material nor significant.

Table 13. Analytical Procedures - Primary Laboratories

Analytical Procedures - Primary Laboratories				
Element	**Lab Code**	**Sample Types**	**Detection Limit**	**Procedure**
SGS Chelopech, Bulgaria - Programme 4 Only				
Gold (Au)	F650	Primary, Repeat, & Split	>0.01ppm	50gm fire assay, lead collection, with AAS finish
Silver (Ag)	A117	Primary, Repeat, & Split	1ppm to 100ppm	0.3gm charge dissolved in 15ml (x50 dilution factor) two acid aqua regia digest (hydrochloric and nitric acid) with AAS finish
	A104	Primary, Repeat, & Split	>100ppm	025.gm charge dissolved in 50ml (x200 dilution factor) four acid digest (hydrofluoric, perchloric, hydrochloric, and nitric acid) with AAS finish

Within each batch of 50 samples internal lab QA/QC checks consist of 3 repeats, 3 second splits, 2 standards, and 1 blank

SGS Analabs - Welshpool, Australia - Programme 3 Only				
Gold (Au)	F650	Primary, Repeat, & Split	>0.01ppm	50gm fire assay, lead collection, with AAS finish (gravimetric finish on samples nominally greater than 80g/t)
Silver (Ag)	A117	Primary, Repeat, & Split	1ppm to 100ppm	0.4gm charge dissolved in 20ml (x50 dilution factor) two acid aqua regia digest (hydrochloric and nitric acid) with AAS finish
	A119	Primary, Repeat, & Split	>100ppm	0.25gm charge dissolved in 500ml (x2000 dilution factor) four acid digest (hydrofluoric, perchloric, hydrochloric, and nitric acid) with AAS finish

Within each batch of 50 samples internal lab QA/QC checks consist of 2 repeats, 3 second splits, 2 standards, and 1 blank

SGS Analabs - Gura Rosiei, Romania - Programmes 2 and 3				
Gold (Au)	F650	Primary, Repeat, & Split	>0.01ppm	50gm fire assay, lead collection, with AAS finish
Silver (Ag)	A108	Primary, Repeat, & Split	1ppm to 300ppm	0.5gm charge dissolved in 25ml (x50 dilution factor) two acid aqua regia digest (hydrochloric and nitric acid) with AAS finish
	A108	Primary, Repeat, & Split	>300ppm	0.25gm charge dissolved in 25ml (x100 dilution factor) two acid digest (hydrochloric and nitric acid) with AAS finish

Analytical Procedures - Primary Laboratories				
Element	**Lab Code**	**Sample Types**	**Detection Limit**	**Procedure**

Within each batch of 50 samples internal lab QA/QC checks consist of 4 repeats, 3 second splits, 2 standards, and 1 blank

OMAC Laboratories, Galway, Ireland - Programme 1 Only				
Gold (Au)	Au4	Primary	>0.01ppm	30gm fire assay, lead collection, with AAS finish
	Au5	Repeat & Re-repeat	>0.01ppm	30gm two acid aqua regia digest (hydrochloric and nitric acid) mixed in a MIBK solution (hydrocarbon compound) with AAS finish
	Au6	Primary	>0.01 ppm	30gm two acid aqua regia digest (hydrochloric and nitric acid) with AAS finish. 10% of samples checked with fire assay
Silver (Ag)	GAR	Primary & Repeat	0.5ppm to 200ppm	0.2gm charge dissolved in 10ml (x50 dilution factor) two acid aqua regia digest (hydrochloric and nitric acid) with AAS finish.
	BM2	Primary & Repeat	>1500ppm	1gm charge dissolved in 200ml (x200 dilution factor) two acid digest (hydrochloric and nitric acid) with AAS finish.

Within a normal batch of 50 samples internal lab QA/QC checks consists of 4 repeats, 1 standard, and 1 blank

Table 14. Analytical Procedures – Umpire Laboratories

Analytical Procedures – Umpire Laboratories				
Genalysis – Maddinton, Australia – Programmes 2, 3 and 4				
Gold (Au)	FA50	Primary	>0.01ppm	50gm fire assay, lead collection, with AAS finish
	FA25	Repeat	>0.01ppm	25gm fire assay, lead collection, with AAS finish
Silver (Ag)	B/AAS	Primary & Repeat	0.1ppm to 100ppm	10gm charge, dissolved in 50ml (x5 dilution factor) two acid aqua regia digest (hydrochloric and nitric acid) with AAS finish
	B/AAS	Primary & Repeat	>100ppm	1gm charge dissolved in 50ml (x50 dilution factor) two acid aqua regia digest. Alternatively, a four acid digest is used (hydrofluoric, perchloric, hydrochloric, and nitric acid) with AAS finish
Internal lab QA/QC consists of one blank inserted at the start of each job, one standard/blank randomly inserted at a ratio of one standard/blank every 26 samples, plus one standard/blank inserted at the end of the job. 10% of gold samples are repeated, consisting of a routine duplicate every 25 samples and 6% repeats on selected assay results. In addition, laser sizing has been completed on every 10th sample.				
ALS Chemex – Vancouver, Canada 0 – Programmes 3 and 4				

Gold (Au)	Au-AA26	Primary & Repeat	>0.01ppm to 100ppm	50gm fire assay, lead collection, with AAS finish
	Au-GRA22	Primary	0.05ppm to 1000ppm	50gm fire assay, lead collection with gravimetric finish
Silver (Ag)	Ag-AA45	Primary & Repeat	0.2ppm to 100ppm	0.5gm charge dissolved in 12.5ml (x25 dilution factor) two acid aqua regia digest with AAS finish
	Ag-AA46	Primary	1ppm to 1500ppm	0.4g charge dissolved in 10ml (x25 dilution factor) aqua regia digest with AAS finish
Within a normal batch of 84 assays internal lab QA/QC checks consist of 2 standards, 3 duplicates, and 1 blank. In addition, every 10th sample has undergone				
SGS Analabs – Welshpool, Australia – Programme 4 Only – Intra Lab Check with SGS Chelopech				
Gold (Au)	FAA005	Primary, Repeat, & Split	>0.01ppm	50gm fire assay, lead collection, with AAS finish (gravimetric finish on samples nominally greater than 80g/t)
Silver (Ag)	AAS12s	Primary, Repeat, & Split	1ppm to 100ppm	0.4gm charge dissolved in 20ml (x50 dilution factor) two acid aqua regia digest (hydrochloric and nitric acid) with AAS finish
	AAS42s	Primary, Repeat, & Split	>100ppm	0.25gm charge dissolved in 500ml (x2000 dilution factor) four acid digest (hydrofluoric, perchloric, hydrochloric, and nitric acid) with AAS finish
Within each batch of 50 samples internal lab QA/QC checks consist of 2 repeats, 3 second splits, 2 standards, and 1 blank (Note: FAA005 same method as F650; AAS12s same method as A117; AAS42s same method as A119 - (scheme codes changed during October 2004))				

Table 15. Summary of Primary Sample Gold Analyses Sub-divided by Exploration Programme, Laboratory, Analytical Method and Exploration Sample Type.

Summary of Primary Sample Gold Analyses Sub-divided by Exploration Programme, Laboratory, Analytical Method and Exploration Sample Type					
Exploration Programme	Lab	Analytical Method	Sample Type	Num. of Samples	Percentage of Samples
1	OMAC	Au4 Au6	Core Trench	1,393 1,954	13.9% 19.6%
	SGS_GR	F650 F650	Core Trench	4,282 2,365	42.8% 23.7%
Programme 1 Sub-Total				9,994	100%
2	SGS_GR	F650 F650 F650	Core RC Trench	1,642 4,513 3,507	17.0% 46.7% 36.3%
Programme 2 Sub-Total				9,662	100%

3	SGS_GR	F650 F650 F650	Core RC Trench	8,094 9,833 5,586	18.1% 22.0% 12.5%
	SGS_WP	F650 F650 F650	Core RC Trench	2,767 17,068 1,358	6.2% 38.2% 3.0%
Programme 3 Sub-Total				44,706	100%
4	SGS_CH	F650	RC	2,065	100%
Programme 4 Sub-Total				2,065	0.0%
Combined Programmes	OMAC	Au4 Au6	Core Trench	1,393 1,954	2.1% 2.9%
	SGS_GR	F650 F650 F650	Core RC Trench	14,018 14,346 11,458	21.1% 21.6% 17.3%
	SGS_WP	F650 F650 F650	Core RC Trench	2,767 17,068 1,358	4.2% 25.7% 2.0%
	SGS_CH	F650	RC	2,065	3.1%
Total	All	All	All	66,427	100.0%

Table 16. Summary of Primary Sample Silver Analyses Sub-divided by Exploration Programme, Laboratory, Analytical Method and Exploration Sample Type

Summary of Primary Sample Silver Analyses Sub-divided by Exploration Programme, Laboratory, Analytical Method and Exploration Sample Type					
Exploration Programme	Lab	Analytical Method	Sample Type	Num. of Samples	Percentage of Samples
1	OMAC	GAR	Core	1,327	23.5%
	SGS_GR	A108 A108	Core Trench	3,655 658	64.8% 11.7%
Programme 1 Sub-Total				5,640	100%
2	SGS_GR	A108 A108 A108	Core RC Trench	1,642 4,513 3,498	17.0% 46.8% 36.2%
Programme 2 Sub-Total				9,653	100%
3	SGS_GR	A108 A108 A108	Core RC Trench	8,094 9,833 5,586	18.1% 22.0% 12.5%
	SGS_WP	A117 A117 A117 A119 A119	Core RC Trench Core RC	2,765 17,053 1,358 1 15	6.2% 38.2% 3.0% 0.0% 0.0%
Programme 3 Sub-Total				44,705	100%
4	SGS_CH	A117	RC	2,069	100%
Programme 4 Sub-Total				2,069	100%
Combined Programmes	OMAC	GAR	Core	1,327	2.1%
	SGS_GR	A108 A108 A108	Core RC Trench	13,391 14,346 9,742	21.6% 23.1% 15.7%
	SGS_WP	A117 A117 A117 A119 A119	Core RC Trench Core RC	2,765 17,053 1,358 1 15	4.5% 27.5% 2.2% 0.0% 0.0%
	SGS_CH	A117	Core	2,069	3.3%
Total	All	All	All	62,067	100.0%

12 Data Verification

The information set out in this section is taken from the 2012 Report, has been reviewed by the author of this Technical Report and remains current. CSA comment and observations from verification completed on site in 2012 or subsequent data review, are included in this section where relevant.

12.1 Logging

All surface trenches and channel sampled road cut exposures, RC drilling and diamond core were geologically logged using the standard BMM logging scheme. Geological logging was carried out mostly on a one metre basis, with particular attention to oxidation type, rock type, tectonic/structural fabrics, veining/intensity, alteration/intensity, sulphides/intensity, and moisture content. In addition, the occurrences of voids and/or insufficient samples were recorded. Detailed geological drawings of all channel sampled trenches and road cut exposures have been generated. Geological logging of core was mostly conducted over intervals equal to the sampling interval (generally one metre), with the exception of the first drilling programme when logging intervals were chosen on a geological basis.

All core was geotechnically logged, including RQD, recovery per drill run, and number of fractures per metre. Core recoveries were calculated by comparing the measured length of recovered core with the distance recorded on the core blocks between each drill run. Detailed structure orientation logging was completed for all orientated intervals of core including recording of structure types and associated alpha/beta measurements. Dip and dip direction measurements were also collected for structures exposed by trenching and along channel sampling pathways in road cut exposures.

Due to the difficulty in obtaining robust orientation marks for much of the core, an alternative method was also used to record vein orientation data for the majority of the core. This involved recording the number of veins with alpha angle measurements lying within a series of alpha angle intervals (Alpha 0o-20o, 20o-50o, 50o-70o and 70o-90o) over each sampling interval, thus at least allowing the orientation of vein structures to the core axis to be considered during geological modelling.

The great majority of logging information was collected digitally on palm top Hewlett Packard IPAQ computers using Field Marshall software. All core was photographed, both wet and dry, using a digital camera. The geological, geotechnical and structure orientation logging of the drilling and trenching completed at Ada Tepe has been conducted to high industry standards.

During the 2012 site visit CSA reviewed logging information contained in the database for the project, and verified this information (exported as graphic drill logs from acQuire software) against several drill holes inspected at the time. No issues were detected and CSA verifies that the information stated as being collected has been captured.

12.2 Statistical Analysis of Assay Quality Control Data

12.2.1 Introduction

CSA used the software programme QAQCReporter ("QAQCR") to produce QA/QC reports in order to review the accuracy and precision of the assayed QA/QC material and samples. No significant issues of bias or fatal flaws were noted and plots and comments are included in the relevant sections below

The precision and accuracy of the gold and silver assay data for the Ada Tepe exploration samples have been assessed based on assays of routine quality control samples inserted into the sample stream, both as part of each laboratory's internal quality control procedures, and external monitoring of laboratory performance by BMM. In addition, the reliability of the assay data from the primary laboratories (SGS Gura Rosiei and Welshpool labs) has been further assessed by comparison of the original assay results with umpire assays completed by ALS Chemex and Genalysis.

The quality control data has been assessed statistically using a number of comparative analyses for each dataset. The objectives of these analyses were to determine relative precision and accuracy levels between various sets of assay pairs and the quantum of relative error. The results of the statistical analyses are presented as summary plots, which include the following:

- Thompson and Howarth Plot showing the mean relative percentage error of grouped assay pairs across the entire grade range, used to visualise precision levels by comparing against given control lines.

- Rank % HARD Plot, which ranks all assay pairs in terms of precision levels measured as half of the absolute relative difference from the mean of the assay pairs (% HARD), used to visualise relative precision levels and to determine the percentage of the assay pairs population occurring at a certain precision level.

- Mean vs % HARD Plot, used as another way of illustrating relative precision levels by showing the range of % HARD over the grade range.

- Mean vs %HRD Plot is similar to the above, but the sign is retained, thus allowing negative or positive differences to be computed. This plot gives an overall impression of precision and also shows whether or not there is significant bias between the assay pairs by illustrating the mean percent half relative difference between the assay pairs (mean % HRD).

- Correlation Plot is a simple plot of the value of assay one against assay two. This plot allows an overall visualisation of precision and bias over selected grade ranges. Correlation coefficients are also used.

- Quantile-Quantile ("Q-Q") Plot is a means where the marginal distributions of two datasets can be compared. Similar distributions should be noted if the data is unbiased.
- Standard Control Plot shows the assay results of a particular reference standard over time. The results can be compared to the expected value, and the ±10% precision lines are also plotted, providing a good indication of both precision and accuracy over time.

12.2.2 Assay Accuracy

The accuracy of the gold and silver assay data and the potential for cross contamination of samples during sample preparation has been assessed based on the assay results for the laboratory internal standards and blanks and the BMM submitted blanks and Rocklabs standards. Both oxide and sulphide standards have been used covering the range of the majority of the gold and silver grades evident at the Ada Tepe deposit.

12.2.2.1 OMAC

No assay data are available for the internal laboratory standards and blanks analysed by OMAC during the initial exploration programme.

12.2.2.2 SGS - Gura Rosiei

The following standards and blanks datasets have been used to assess the accuracy of the SGS Gura Rosiei gold and silver assay data:

- Gold assay data for blanks and 3 internal laboratory standards routinely analysed during exploration programme 3.
- Gold assay data for BMM submitted blanks from July 2004 to August 2004 (exploration programme 3).
- Gold assay data for 15 Rocklabs standards (with 10 or greater assay results) submitted by BMM throughout exploration programmes 2 and 3.
- Silver assay data for blanks and 2 internal laboratory standards routinely analysed during exploration programme 3.
- Silver assay data for 1 Rocklabs standard submitted by BMM throughout exploration programmes 2 and 3, and 2 additional standards submitted by BMM during August 2004.

CSA completed checks on standards and blanks submitted to SGS Gura Rosiei, via the preparation of validation control plots. A selection of these for which significant samples were submitted, are illustrated in Figure 17 to Figure 19 below.

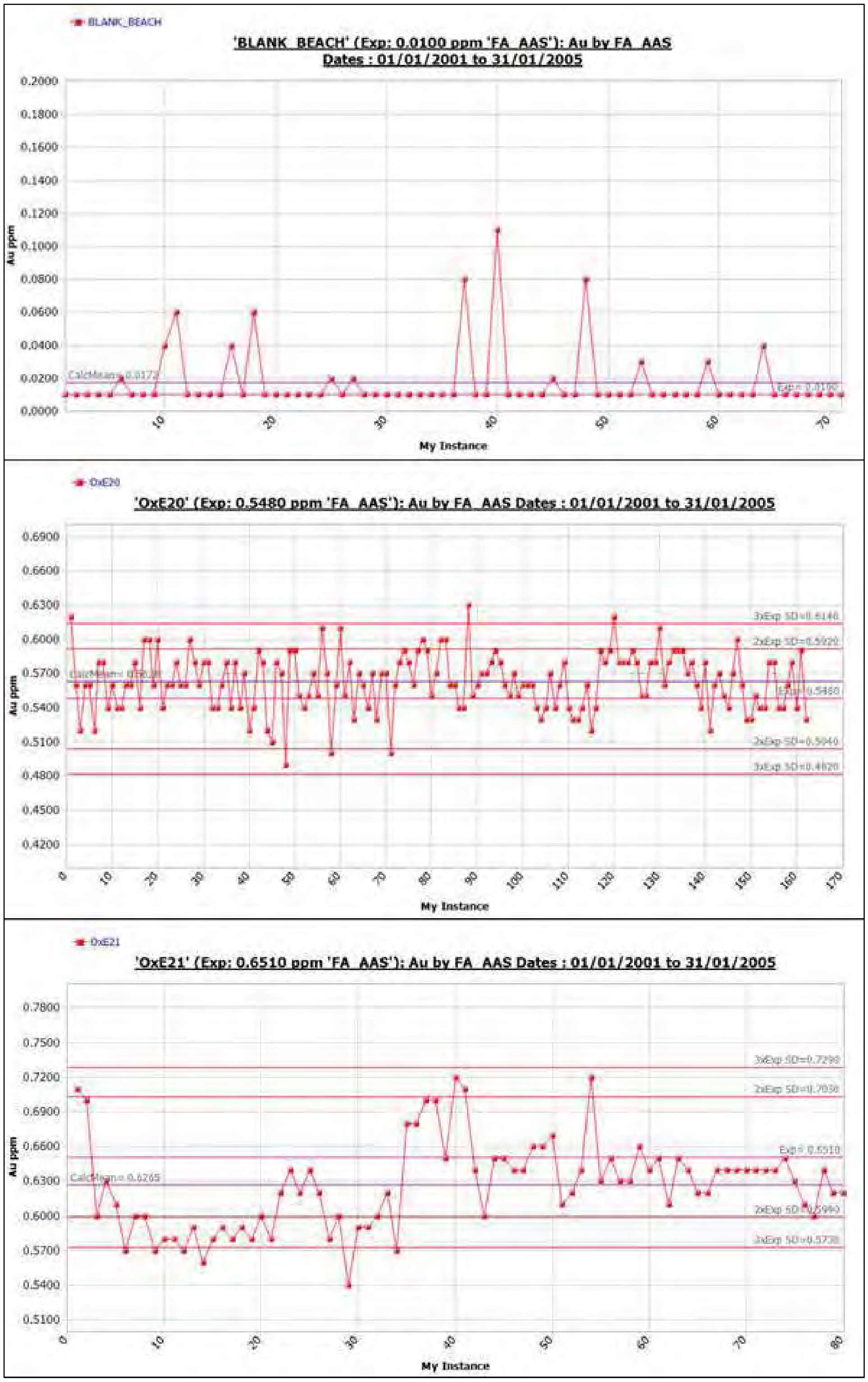

Figure 16. Gold Blank and Standard Results (Blank Beach, OxE20, OxE21) from SGS - Gura Rosiei (CSA)

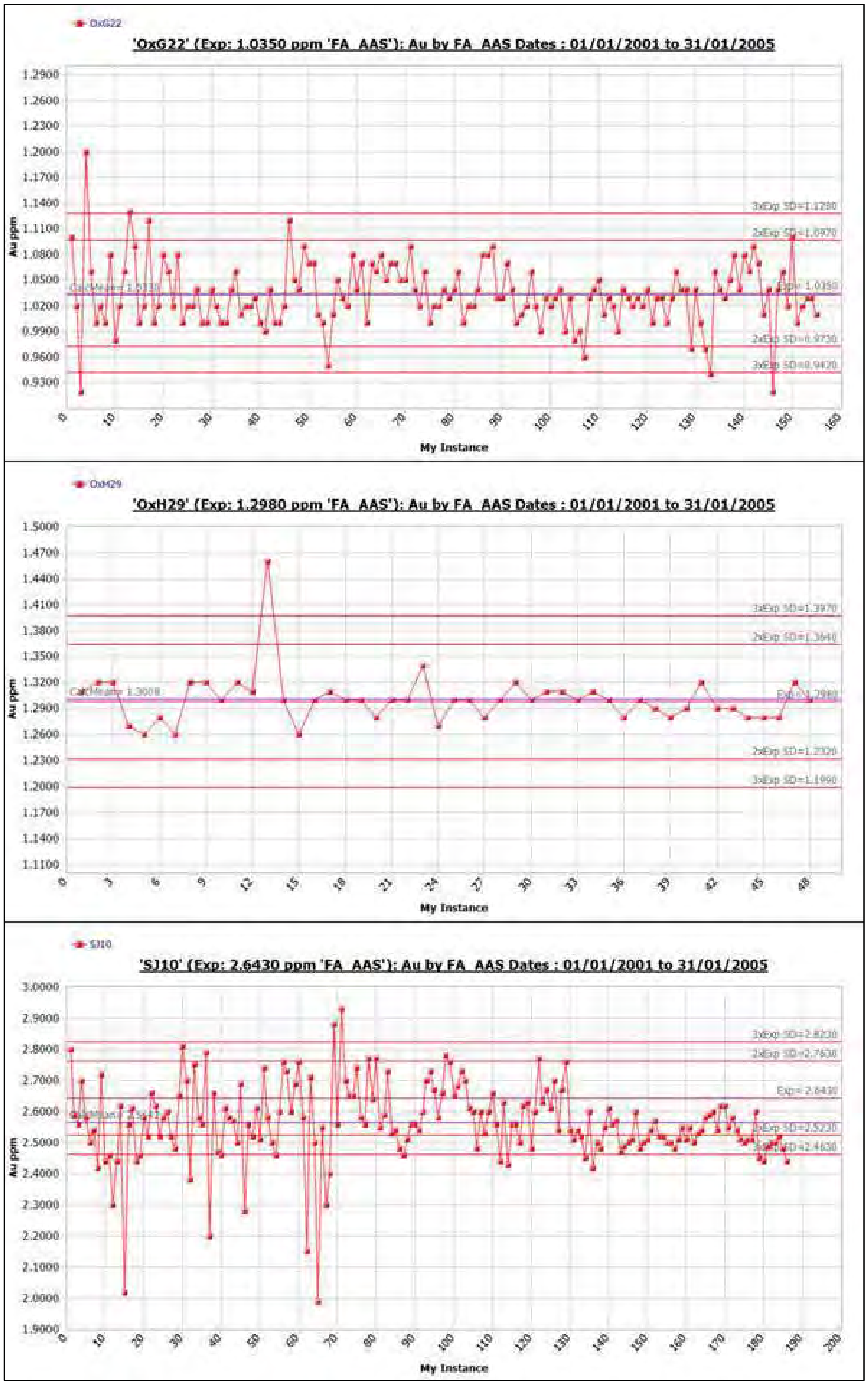

Figure 17. Gold Standard Results (OxG22, OxH29, SJ10) from SGS - Gura Rosiei (CSA)

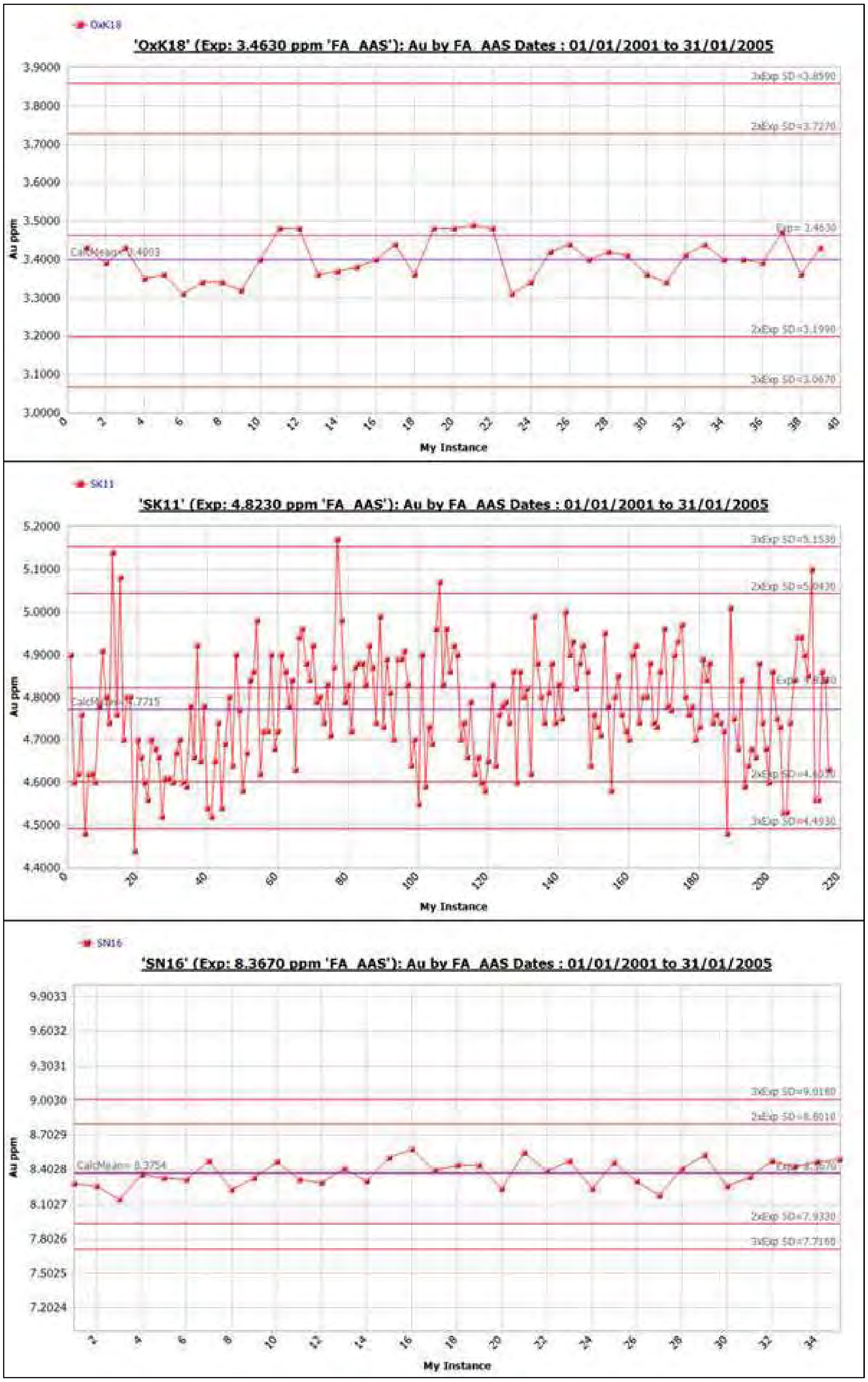

Figure 18. Gold Standard Results (OxK18, SK11, SN16) from SGS - Gura Rosiei (CSA)

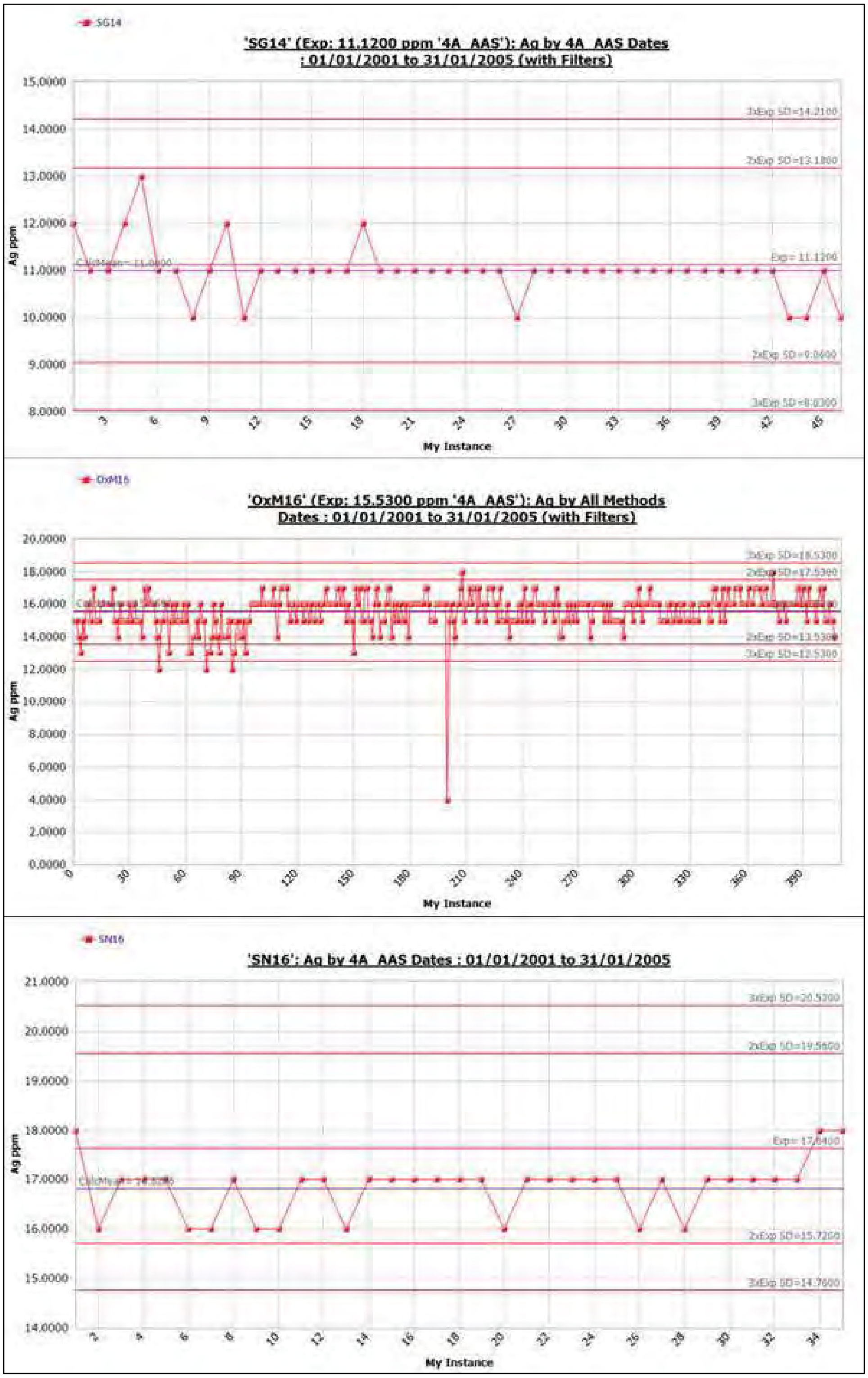

Figure 19. Silver Standard Results (SG14, OxM16, SN16) from SGS - Gura Rosiei (CSA)

RSG commented that the results of the statistical analysis of the blanks and standards analysed by the SGS Gura Rosiei facility can be summarised as follows:

- The vast majority of the blanks report gold and silver assays at or near the detection limits and hence, there is little evidence of cross contamination of samples.
- The vast majority of the gold and silver assays of the laboratory and BMM submitted standards are within ±10% of the expected standard values.
- There is no evidence of significant bias in gold and silver assays for any of the analysed standards.

Results of the internal laboratory blanks and standards were not available for review by CSA, however the results of the BMM submitted blanks and standards assayed show no evidence of systematic bias. CSA considers that the gold and silver standards analysed by the SGS Gura Rosiei laboratory to be accurate and appropriate for mineral resource estimation studies.

12.2.2.3 SGS Welshpool

The accuracy of the SGS Welshpool gold and silver assay data has been assessed based on the following standards and blanks datasets:

- Gold assay data for blanks and 5 internal laboratory standards routinely analysed during exploration programme 3.
- Gold assay data for BMM submitted blanks from May 2004 to August 2004 (exploration programme 3).
- Gold assay data for 10 Rocklabs standards with greater than 10 assay results (12 used in total) submitted by BMM throughout exploration programme 3.
- Silver assay data for blanks and 2 internal laboratory standards routinely analysed during exploration programme 3.
- Silver assay data for 2 Rocklabs standards submitted by BMM throughout exploration programme 3.

CSA completed checks on standards and blanks submitted to SGS Welshpool, via the preparation of validation control plots. A selection of these for which significant samples were submitted, are illustrated in Figure 20 to Figure 24 below.

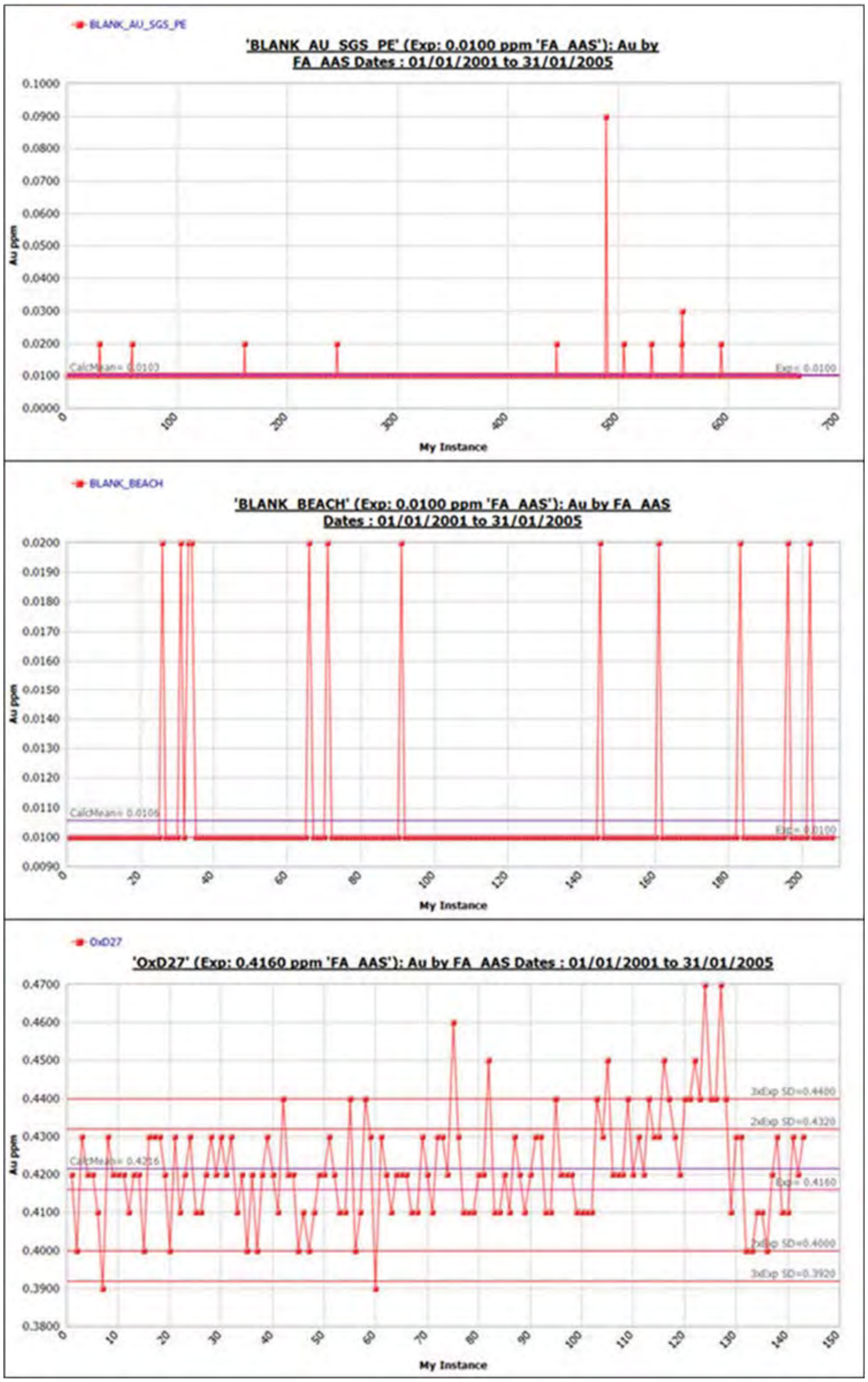

Figure 20. Gold Blank and Standard Results (Blank Au SGS, Blank Beach,OxD27) from SGS – Welshpool (CSA)

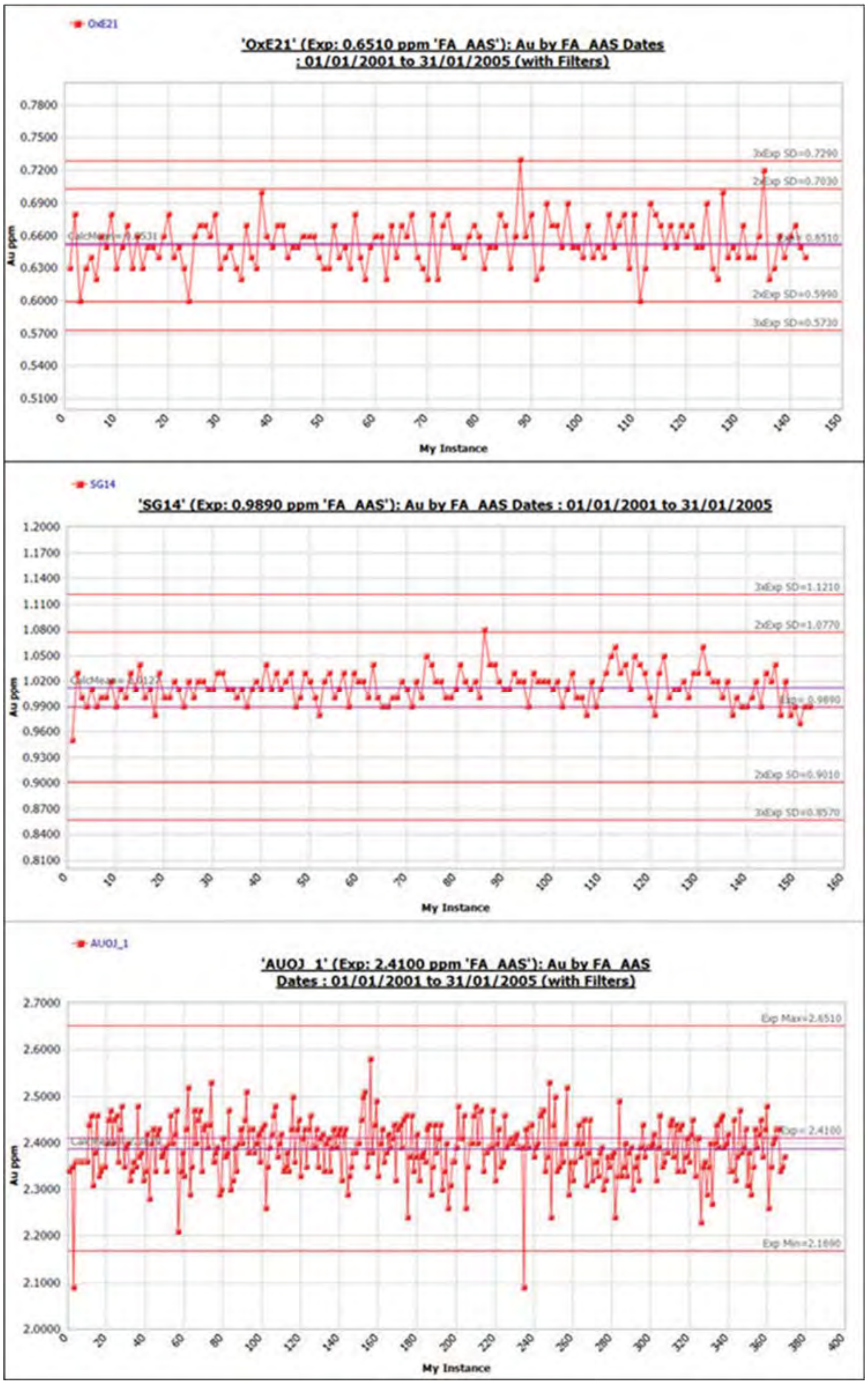

Figure 21. Gold Standard Results (OxE21, SG14, AuOJ 1) from SGS – Welshpool (CSA)

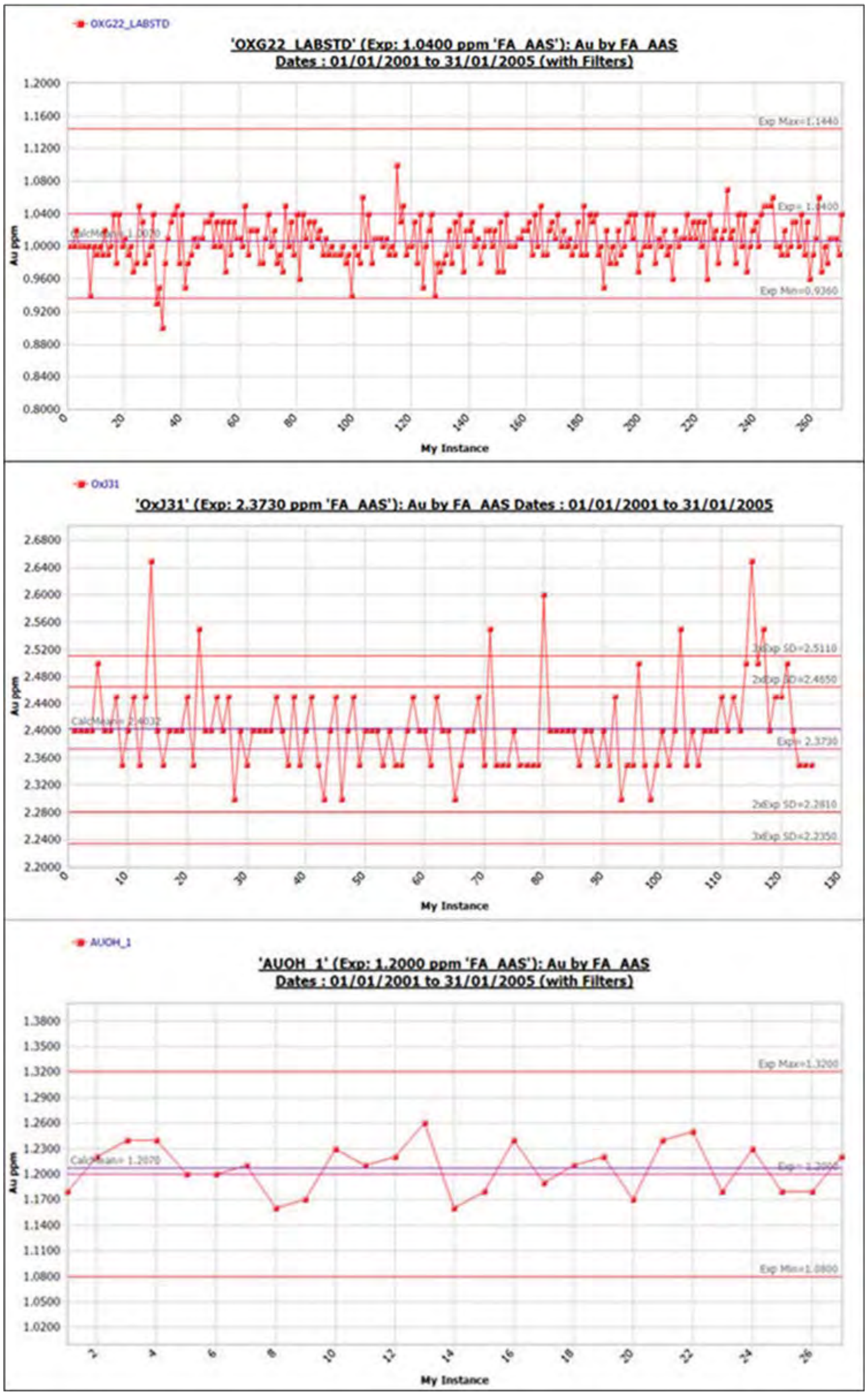

Figure 22. Gold Standard Results (OxG22, OxJ31, AuOH 1) from SGS – Welshpool (CSA)

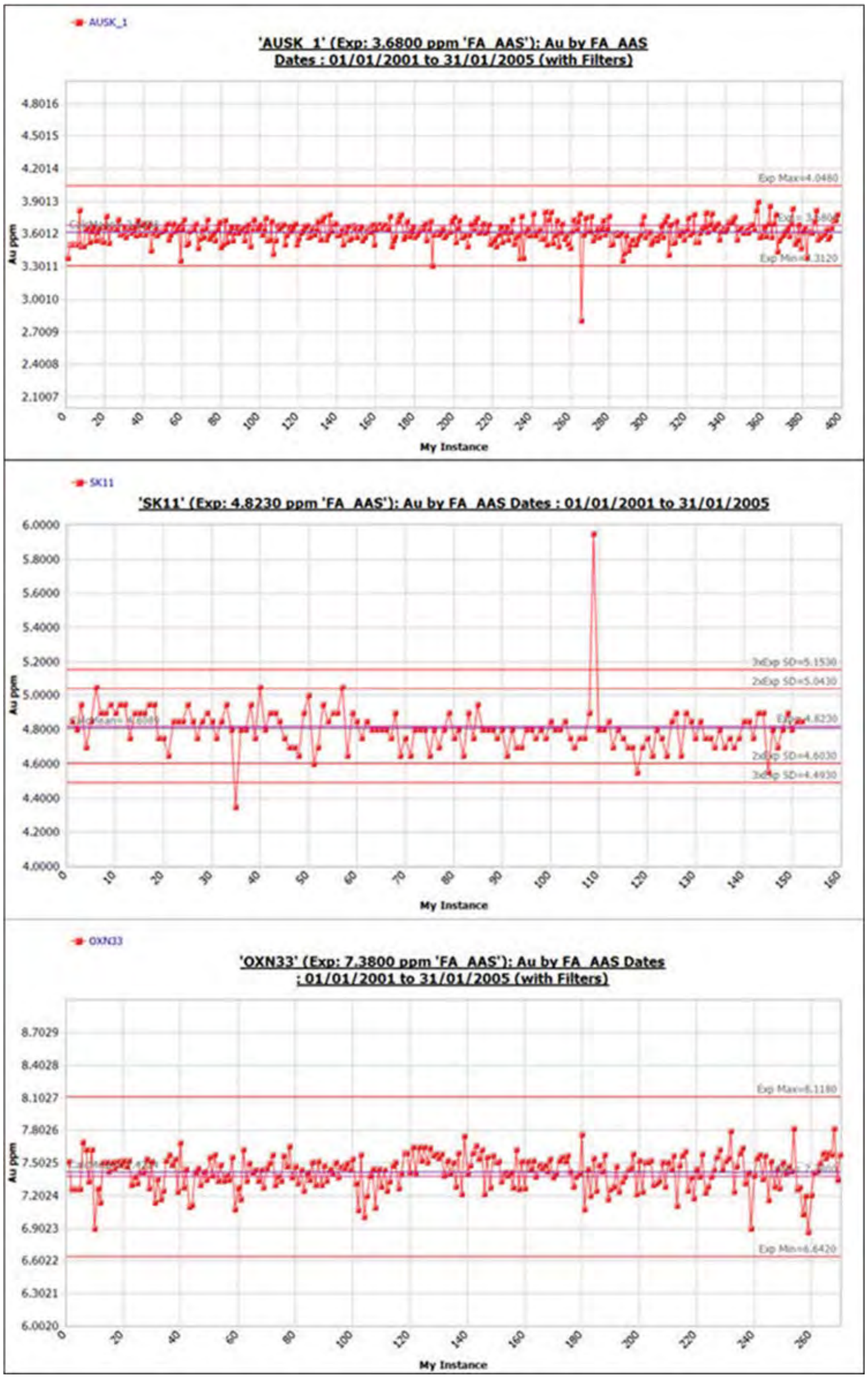

Figure 23. Gold Standard Results (AuSK1, SK11, OxN33) from SGS – Welshpool (CSA)

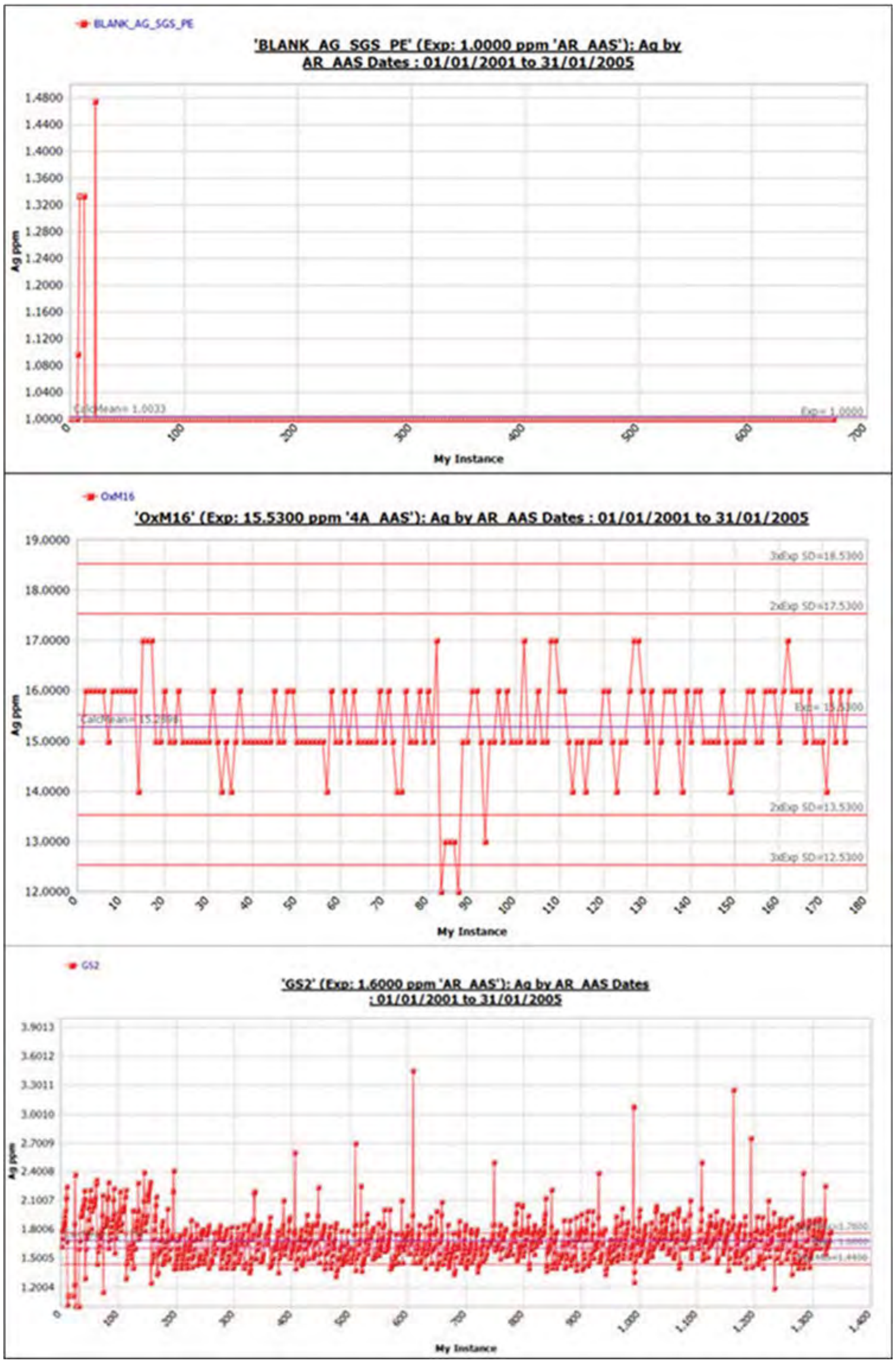

Figure 24. Silver Blank and Standard Results (Blank Ag SGS, Om16, GS2) from SGS – Welshpool (CSA)

The results of the statistical analysis of the blanks and standards analysed by the SGS Welshpool laboratory can be summarised as follows:

- The vast majority of the blanks report gold and silver assays at or near the detection limits and hence, there is little evidence of cross contamination of samples.
- The vast majority of the gold and silver assays of the laboratory and BMM submitted standards are within ±10% of the expected standard values.
- There is no evidence of significant bias in gold and silver assays for any of the analysed standards.

CSA considers that the gold and silver standards analysed by the SGS Welshpool laboratory to be accurate and appropriate for mineral resource estimation studies.

12.2.2.4 SGS Chelopech

The accuracy of the SGS Chelopech gold and silver assay data has been assessed based on the following standards and blanks datasets:

- Gold assay data for blanks and 2 internal laboratory standards routinely analysed during exploration programme 4.
- Gold assay data for BMM submitted blanks from November 2004 to December 2004 (exploration programme 4).
- Gold assay data for 6 Rocklabs standards with greater than 10 assay results (14 used in total) submitted by BMM throughout exploration programme 4.
- Silver assay data for blanks and 2 internal laboratory standards routinely analysed during exploration programme 4.
- Silver assay data for 1 Rocklabs standards with greater than 8 assays results (4 standards used in total) submitted by BMM throughout exploration programme 4.

CSA completed checks on standards and blanks submitted to SGS Chelopech, via the preparation of validation control plots. A selection of these for which significant samples were submitted, are illustrated in Figure 25 and Figure 26 below.

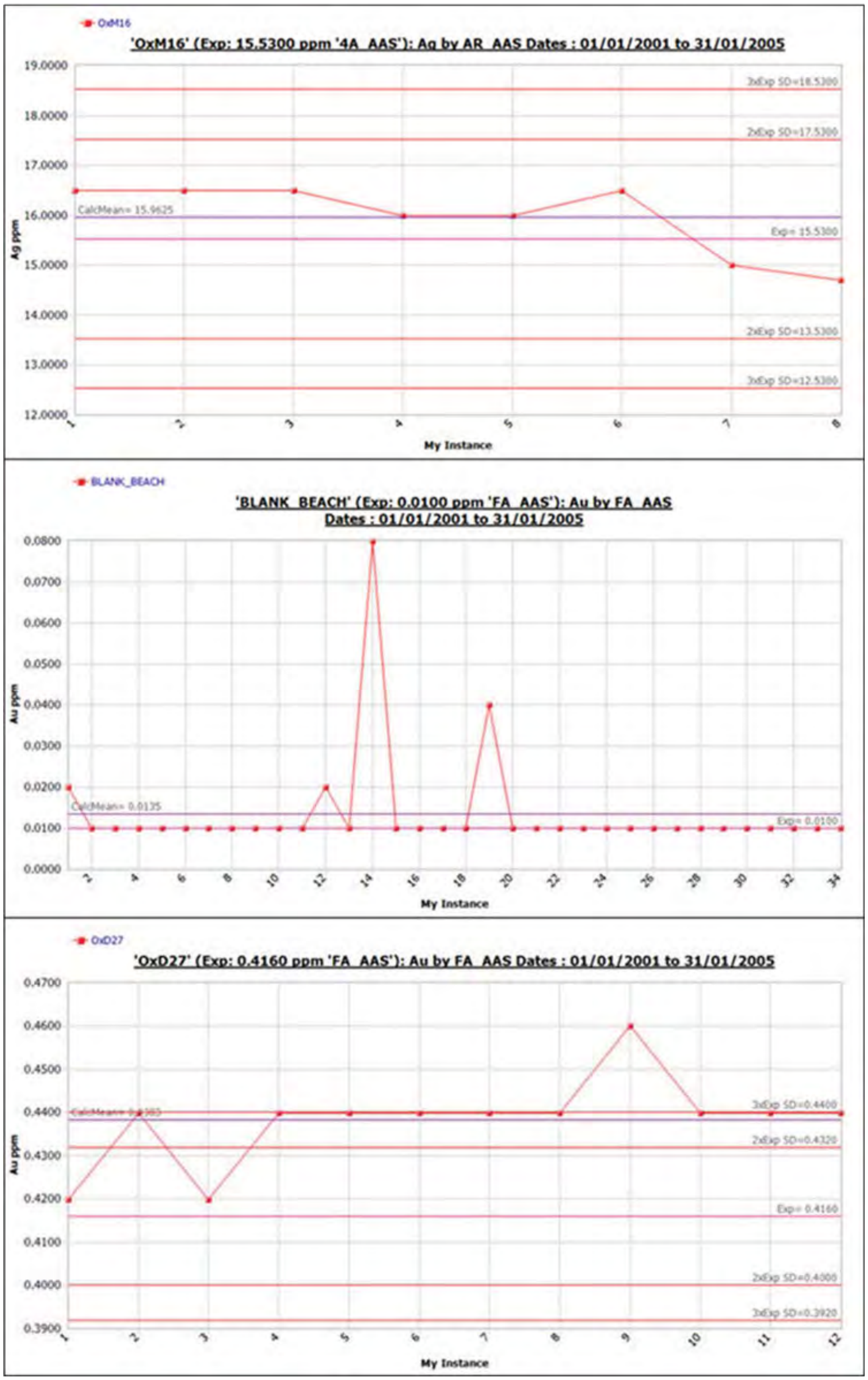

Figure 25. Silver and Gold Standard Results (OxM16, Blank Beach, OxD27) from SGS – Chelopech (CSA)

Figure 26. Gold Standard Results (OxE21, OxL25, Om16) from SGS – Chelopech (CSA)

The results of the statistical analysis of the blanks and standards analysed by the SGS Chelopech laboratory can be summarised as follows:

- The vast majority of the blanks report gold and silver assays at or near the detection limits and hence, there is little evidence of cross contamination of samples.
- The vast majority of the gold and silver assays of the laboratory and BMM submitted standards are within +-10% of the expected standard values.
- There is no evidence of significant bias in gold and silver assays for any of the analysed standards.

Internal laboratory blanks and standards were not available for CSA review. Those submitted by BMM were reviewed. CSA notes that accuracy and precision exhibited by SGS Chelopech was, at times, poorer than the other laboratory results, however no significant bias is noted and CSA considers that the gold and silver standards analysed by the SGS - Chelopech laboratory to be accurate and appropriate for mineral resource estimation studies.

12.2.3 Assay Precision

The precision of the gold and silver assay data for the Ada Tepe deposit has been statistically assessed based on the following comparative sample/data types:

- Duplicate RC and trench samples collected in the field - allows assessment of overall precision, reflecting total combined sampling and analytical errors (field and laboratory).
- Duplicate splits of core and trench samples after jaw crushing - allows assessment of overall laboratory precision, reflecting sample preparation and analytical errors at the laboratory.
- Duplicate pulp splits of samples after pulverisation - allows assessment of laboratory precision inclusive of sampling and analytical errors after sample pulverisation.
- Repeat analyses of laboratory pulp samples - allows assessment of laboratory analytical precision (exclusive of dominant sampling errors).
- Primary versus umpire laboratory analyses of duplicate pulp splits - allows assessment of inter-laboratory precision (and relative accuracy) inclusive of sampling and analytical errors after sample pulverisation.

A statistical summary of gold and silver assays and blcnks for each laboratory are detailed in Table 17 to Table 22.

Table 17. Statistical Summary - SGS Gura Rosiei Laboratory - Gold Assays of Standards and Blanks

Statistical Summary - SGS Gura Rosiei Laboratory - Gold Assays of Standards and Blanks										
Standard Name	Expected Value (EV)	+/-10% (EV) (g/t)	Period of Analyses	No of Analyses	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	% Within +/- 10 of EV	% RSD (from EV)	% Bias (from EV)
Internal Laboratory Standards and Blanks										
BLANK	0.0	N/A	Aug. 2003 - Aug. 2004	612	0.005	0.04	0.006	N/A	N/A	N/A
ST17	0.76	0.684 / 0.836	May 2004 - Aug. 2004	345	0.69	0.84	0.76	99.7	2.83	-0.44
QFA8	1.18	1.062 / 1.298	July 2003 - March 2004	286	1.09	1.28	1.17	100.0	3.01	-1.03
QFA15	1.42	1.278 / 1.562	Aug. 2003 - Sept. 2004	617	1.22	1.53	1.42	99.8	3.16	0.19
BMM Submitted Standards and Blanks										
AuBlank	0.0	N/A	May 2004 - Aug. 2004	20	0.005	0.01	0.006	N/A	N/A	N/A
Beach Sand	0.0	N/A	July 2004 - Aug. 2004	66	0.005	0.11	0.015	N/A	N/A	N/A
OxD27	0.416	0.374 / 0.458	May 2004 - Aug. 2004	40	0.40	0.48	0.42	92.5	4.31	1.74
OxE20	0.548	0.493 / 0.603	May 2003 - June 2004	162	0.49	0.63	0.56	95.7	4.43	2.71
OxE21	0.651	0.586 / 0.716	Jan. 2004 - Aug.2004	76	0.54	0.72	0.63	80.3	6.29	-3.71
OxG22	1.035	0.932 / 1.139	May 2003 - June 2004	156	0.92	1.20	1.03	98.1	3.64	-0.26
OxH29	1.298	1.168 / 1.428	Jan. 2003 - June 2004	50	1.22	1.46	1.30	98.0	2.38	0.06
OxJ24	2.427	2.184 / 2.670	June 2002 - Aug. 2002	124	2.24	2.75	2.46	97.6	3.94	1.55
OxJ31	2.373	2.136 / 2.610	April 2004 - Aug. 2004	20	2.25	2.54	2.43	100.0	2.37	2.28
OxK18	3.463	3.117 / 3.809	August 2004	32	3.31	3.49	3.40	100.0	1.59	-1.89
OxL25	5.852	5.267 / 6.437	April 2004 - Aug.2004	51	5.65	6.01	5.83	100.0	1.39	-0.45
OxM16	15.15	13.635 / 16.665	April 2002 - Aug.2004	405	13.24	16.90	14.97	98.3	4.38	-1.19
SG14	0.989	0.890 / 1.088	April 2004 - Aug. 2004	42	0.98	1.06	1.01	100.0	1.71	2.00
SJ10	2.643	2.379 / 2.907	April 2002 - Aug. 2004	181	1.99	2.93	2.57	95.6	4.94	-2.88
SK11	4.823	4.341 / 5.305	May 2003 - Aug. 2004	222	4.30	5.17	4.77	99.6	2.82	-1.13
SN16	8.367	7.530 / 9.204	August 2004	30	8.14	8.58	8.36	100.0	1.34	-0.03

Note: RSD = Relative Standard Deviation

Table 18. Statistical Summary - SGS Gura Rosiei Laboratory – Silver Assays of Standards and Blanks

Statistical Summary - SGS Gura Rosiei Laboratory – Silver Assays of Standards and Blanks										
Standard Name	Expected Value (EV)	+/-10% (EV) (g/t)	Period of Analyses	No of Analyses	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	% Within +/- 10 of EV	% RSD (from EV)	% Bias (from EV)
Internal Laboratory Standards and Blanks										
BLANK	0.0	N/A	April 2003 – Aug. 2004	634	0.5	2.0	0.5	N/A	N/A	N/A
BM997_1	0.0	5.0 - 7.0	April 2003 – Aug. 2004	643	5	8	6.3	99.8	7.6	4.09
GBM997_3	6.09	41.0 - 55.0	April 2003 – Aug. 2004	636	44	60	49	99.8	3.4	1.3
BMM Submitted Standards and Blanks										
OxM16	15.53	13.0 / 18.0	April 2002 – Aug. 2004	405	4	18	15.6	99.0	7.2	0.4
SG14	11.12	9.0 / 13.0	April 2002 – Aug. 2004	42	10	13	11.1	100.0	4.6	-0.4
SN16	17.64	15.0 / 20.0	August 2004	28	16	18	16.7	100.0	3.1	-5.1

Table 19. Statistical Summary - SGS Welshpool Laboratory – Gold Assays of Standards and Blanks

Statistical Summary - SGS Welshpool Laboratory – Gold Assays of Standards and Blanks										
Standard Name	Expected Value (EV)	+/-10% (EV) (g/t)	Period of Analyses	No of Analyses	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	% Within +/- 10 of EV	% RSD (from EV)	% Bias (from EV)
Internal Laboratory Standards and Blanks										
Gold BLANK	0.0	N/A	May 2004 - Sept. 2004	738	0.00	0.20	0.003	N/A	N/A	N/A
AUOH_1	1.197	1.077 / 1.317	Aug. 2004 - Sept. 2004	30	1.04	1.26	1.20	96.7	3.39	0.45
AUOJ_1	2.413	2.172 / 2.654	May 2004 - Aug. 2004	405	2.09	2.58	2.39	99.5	2.48	-1.01
AUSK_1	3.680	3.312 / 4.048	May 2004 - Sept. 2004	437	2.81	3.90	3.62	99.5	2.63	-1.56
OxG22	1.035	0.932 / 1.139	May 2004 - Sept. 2004	301	0.90	1.10	1.01	98.7	2.70	-2.69
OxN33	7.378	6.640/ 8.116	May 2004 - Sept. 2004	301	6.84	7.82	7.42	100.0	2.14	0.57
BMM Submitted Standards and Blanks										
AuBlank1	0.0	N/A	May 2004 - Sept. 2004	57	0.005	0.02	0.008	N/A	N/A	N/A
Beach Sand	0.0	N/A	May 2004 - Aug. 2004	200	0.005	0.03	0.007	N/A	N/A	N/A
OxD27	0.416	0.374 / 0.458	May 2004 - Sept. 2004	130	0.39	0.47	0.42	97.7	3.37	1.46
OxE21	0.651	0.586 / 0.716	May 2004 - Sept. 2004	127	0.60	0.73	0.65	98.4	3.47	0.43
OxG22	1.035	0.931 / 1.138	June 2004 - July 2004	28	0.99	1.06	1.02	100.0	1.69	-1.17
OxJ31	2.373	2.136 / 2.610	May 2004 - Sept. 2004	126	2.30	2.65	2.45	98.4	2.72	1.34
OxL25	5.852	5.267 / 6.437	May 2004 - Sept. 200	112	4.80	6.25	5.82	99.1	2.80	-0.54
OxM16	15.15	13.635 / 16.665	4 June 2004 - Sept. 2004	144	14.7	15.9	15.42	100.0	1.57	1.77
SG14	0.989	0.890 / 1.088	June 2004 - Sept. 200	134	0.95	1.08	1.05	100.0	1.86	2.58
SJ10	2.643	2.379 / 2.907	4 May 2004 - Sept. 2004	127	2.40	2.75	2.60	100.0	2.28	-1.73
SK11	4.823	4.341 / 5.305	June 2004 - Sept. 2004	150	4.35	5.95	4.81	99.3	2.90	-0.29

Table 20. Statistical Summary - SGS Welshpool Laboratory – Silver Assays of Standards and Blanks

Statistical Summary - SGS Welshpool Laboratory – Silver Assays of Standards and Blanks										
Standard Name	Expected Value (EV)	+/-10% (EV) (g/t)	Period of Analyses	No of Analyses	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	% Within +/- 10 of EV	% RSD (from EV)	% Bias (from EV)
Internal Laboratory Standards and Blanks										
AGBLANK	0.0	N/A	May 2004 – Sept. 2004	756	0.0	1.5	N/A	N/A	N/A	N/A
GS2	1.6	1.0 – 2.0	May 2004 – Sept. 2004	1,458	0.0	3.5	93.9	13.5	7.6	4.8
STDTEL	12.5	11.0 - 14.0	September 2004	38	12.2	12.8	100.0	2.7	3.4	2.2
BMM Submitted Standards and Blanks										
OxM16	15.53	13.0 / 18.0	June 2004 – Sept. 2004	149	12	17	98.7	5.8	-1.7	0.4
SG14	11.12	9.0 / 13.0	June 2004 – Sept. 2004	134	8	12	99.3	5.9	-4.6	-0.4

Table 21. Statistical Summary - SGS Chelopech Laboratory - Gold Assays of Standards and Blanks

Statistical Summary - SGS Chelopech Laboratory - Gold Assays of Standards and Blanks										
Standard Name	Expected Value (EV)	+/-10% (EV) (g/t)	Period of Analyses	No of Analyses	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	% Within +/- 10 of EV	% RSD (from EV)	% Bias (from EV)
Internal Laboratory Standards and Blanks										
BLANK	0.0	N/A	Nov. 2004 - Dec. 2004	64	0.01	0.04	0.01	N/A	N/A	N/A
ST05	2.36	2.12/ 2.60	Nov. 2004 - Dec. 2004	58	2.20	2.45	2.34	100	2.34	-0.73
G902-6	4.23	3.81 / 4.65	Nov. 2004 - Dec. 2004	58	4.00	4.50	4.23	100	2.44	0.02
BMM Submitted Standards and Blanks										
AuBlank	0.0	N/A	Nov. 2004 - Dec. 2004	11	0.005	0.01	0.008	N/A	N/A	N/A
Beach Sand	0.0	N/A	Nov. 2004 - Dec. 2004	36	0.005	0.08	0.013	N/A	N/A	N/A
OxD27	0.416	0.374 / 0.458	Nov. 2004 - Dec. 2004	12	0.42	0.46	0.44	91.7	2.25	5.40
OxE21	0.651	0.586 / 0.716	Nov. 2004 - Dec. 2004	9	0.68	0.70	0.69	100	1.33	5.6
OxL25	5.852	5.267 / 6.437	Nov. 2004 - Dec. 2004	14	5.39	6.28	5.82	100	3.76	-0.56
OxM16	15.15	13.635 / 16.665	November 2004	9	16.08	16.08	15.33	100	2.68	1.19
SJ10	2.643	2.379 / 2.907	Nov. 2004 - Dec. 2004	9	2.72	2.72	2.53	100	4.41	-4.19

Table 22. Statistical Summary - SGS Chelopech Laboratory – Silver Assays of Standards and Blanks

Statistical Summary - SGS Chelopech Laboratory – Silver Assays of Standards and Blanks										
Standard Name	Expected Value (EV)	+/-10% (EV) (g/t)	Period of Analyses	No of Analyses	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	% Within +/- 10 of EV	% RSD (from EV)	% Bias (from EV)
Internal Laboratory Standards and Blanks										
BLANK	0.0	N/A	Nov. 2004 - Dec. 2004	64	0.5	0.5	0.5	N/A	N/A	N/A
GBM900-1c	6.08	5.0 / 7.0	Nov. 2004 - Dec. 2004	65	5.5	7.0	6.08	100	4.29	0.00
GBM399-6	15.5	13.0 / 18.0	Nov. 2004 - Dec. 2004	63	14.7	17.0	15.66	100	3.00	1.02
BMM Submitted Standards and Blanks										
OxM16	15.53	13.0 / 18.0	Nov. 2004 - Dec. 2004	8	14.7	16.5	16.0	100	4.25	2.78

The order of the comparative data types listed above reflects the successive removal of compounding error thus allowing the precision associated with each stage in the sampling process (field and laboratory), the sample preparation process and finally the sample analyses, to be assessed. Details of the available datasets and results of the statistical analyses are summarised below for each laboratory. Statistical analysis of the gold datasets has considered only the assay data greater than or equal to 10 times the SGS analytical detection limit (i.e. data at or above 0.1 g/t Au). Similarly, a lower selection threshold of 5 times the SGS analytical detection for silver (i.e. data at or above 5 g/t Ag) has been used for statistical analysis of the silver assay datasets.

CSA reviewed the results of the check analyses (duplicates, repeats and pulp splits) detailed below and duplicate scatter plots were created. No significant bias or material issue were detected.

12.2.3.1 OMAC

The precision of the gold analyses completed by OMAC was assessed based on the following datasets:

- Routine laboratory repeat assay data for core and trench samples.
- Limited number of duplicate trench samples.

The levels of precision reported for both the laboratory repeat gold assays and assays of the duplicate trench samples (Table 23) are consistent with those reported based on the analogous datasets for the SGS Gura Rosiei and Welshpool facilities.

Table 23. Summary Data Precision OMAC Gold Assay Data

Summary Data Precision OMAC Gold Assay Data									
Data Comparison	Number of Data Pairs			Mean % Hard			Median % Hard		
	Trenches/ Channels	RC Chips	DDH Core	Trenches/ Channels	RC Chips	DDH Core	Trenches/ Channels	RC Chips	DDH Core
Duplicate Field Samples	43	-	-	23.9	-	-	15.6	-	-
Duplicate Crusher Splits	-	-	-	-	-	-	-	-	-
Duplicate Pulp Splits	-	-	-	-	-	-	-	-	-
Laboratory Repeats	242	-	142	3.1	-	3.0	2.5	-	2.1

No routine comparative QA/QC assay data are available in relation to the silver analyses completed by OMAC.

12.2.3.2 SGS Gura Rosiei

Assessment of the precision of the gold assays for samples analysed at the SGS Gura Rosiei facility was based on the following available datasets:

- Routine duplicate RC sample splits and trench sample twins.
- Routine duplicate splits of jaw crushed core and trench samples
- Routine duplicate splits of pulverised core, RC, and trench samples.
- Routine laboratory repeat assay data for core, RC, and trench samples.

The results of the statistical analysis of the routine comparative QA/QC gold assay data displayed in Table 24 can be summarised as follows:

- Increasing levels of precision are reported in relation to each successive sampling stage approaching final laboratory analysis (as expected);
- Similar %HARD values (precision levels) are reported for the assay datasets for the different input sample types after entry into the sample preparation stream (duplicate crusher splits onwards).
- The lowest precision (±28%) is reported for the duplicate (twin) trench samples, reflecting the collection of samples from distinctively separate locations (20cm apart), and the closed spaced variability of the gold grades at Ada Tepe.

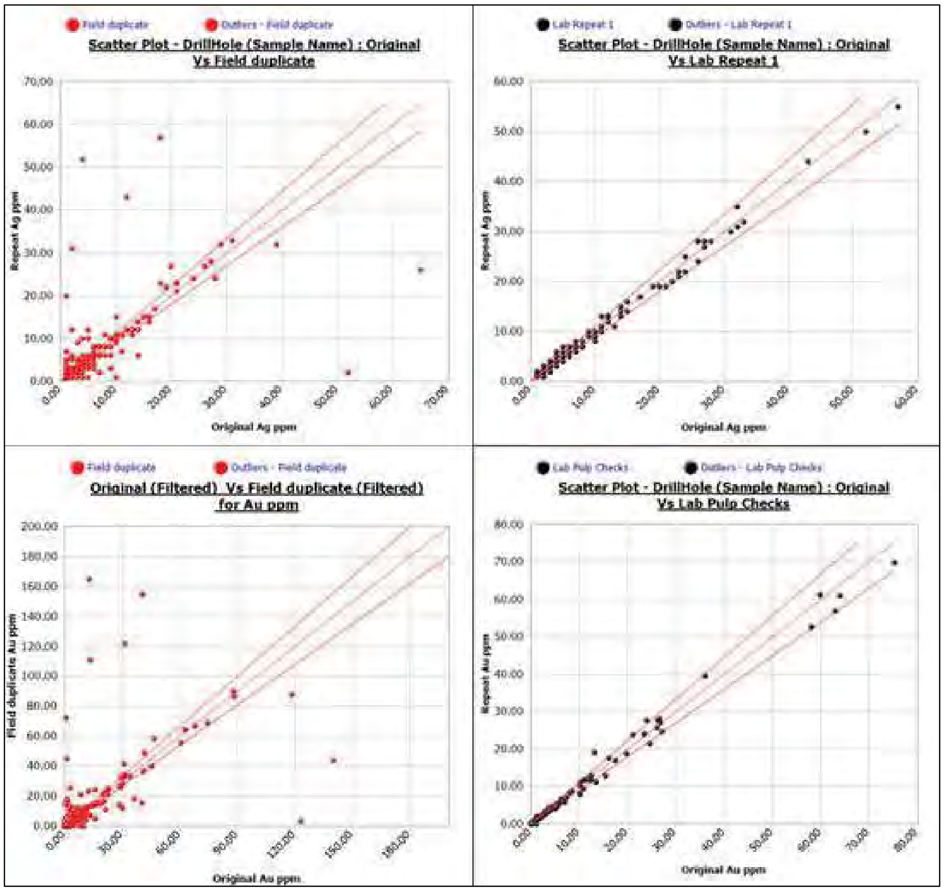

Figure 27. Scatter plots for duplicates analysed at SGS Gura Rosiei (CSA)

CSA considers that Industry Standard acceptable levels of precision are reported for all of the sampling stages for the purpose of mineral resource estimation. Figure 27 shows scatter plots for field duplicates, laboratory repeats and laboratory pulp splits analysed at SGS Gura Rosiei. As expected, the field duplicate pairs (red dots) exhibit a poorer correlation than the laboratory splits and repeats but no significant bias is apparent.

Table 24. Summary of Data Precision SGS Romania Gold and Silver Assay Data

Summary of Data Precision SGS Romania Gold and Silver Assay Data									
Data Comparison	Number of Data Pairs			Mean % Hard			Median % Hard		
	Trenches / Channels	RC Chips	DDH Core	Trenches / Channels	RC Chips	DDH Core	Trenches / Channels	RC Chips	DDH Core
Gold Analyses									
Duplicate Field Samples	374	406	-	27.8	9.2	-	21.4	5.5	-
Duplicate Crusher Splits	156	-	225	5.3	-	6.9	4.0	-	4.2
Duplicate Pulp Splits	372	544	357	4.3	4.2	4.2	3.0	3.0	2.8
Laboratory Repeats	1,244	1,527	1,432	4.1	3.7	3.7	2.9	2.6	24
Silver Analyses									
Duplicate Field Samples	26	44	-	33.6	5.3	-	26.8	4.2	-
Duplicate Crusher Splits	9	-	27	4.2	-	5.9	0.0	-	4.8
Laboratory Repeats	213	511	336	3.11	3.2	3.0	1.61	2.0	1.5

The precision of the silver analyses has been assessed predominantly on the basis of laboratory repeat assay data. A relatively small number of assay pairs were also available for duplicate RC and trench field samples and duplicate jaw crusher splits of core and trench samples. Industry acceptable levels of precision are reported for all of the data comparisons, as indicated in Table 24.

12.2.3.3 SGS Welshpool

The precision of the gold assays for samples analysed at the SGS Welshpool lab was assessed based on the following datasets:

- Routine duplicate RC sample splits and trench sample twins.
- Routine duplicate splits of jaw crushed core and trench samples.
- Routine duplicate splits of pulverised core, RC, and trench samples.
- Routine laboratory repeat assay data for core, RC, and trench samples.

Allowing for variations in the number of data pairs, similar levels of precision are reported for the SGS Welshpool gold assays for the different sample input sample types and downstream sample groups as those reported by SGS Gura Rosiei lab and SGS Chelopech, with industry acceptable levels of precision evident for all of the sampling stages.

Industry acceptable levels of precision for the Welshpool silver assays are also evident based on the relatively small duplicate field sample and laboratory repeat data datasets, as presented in Table 25.

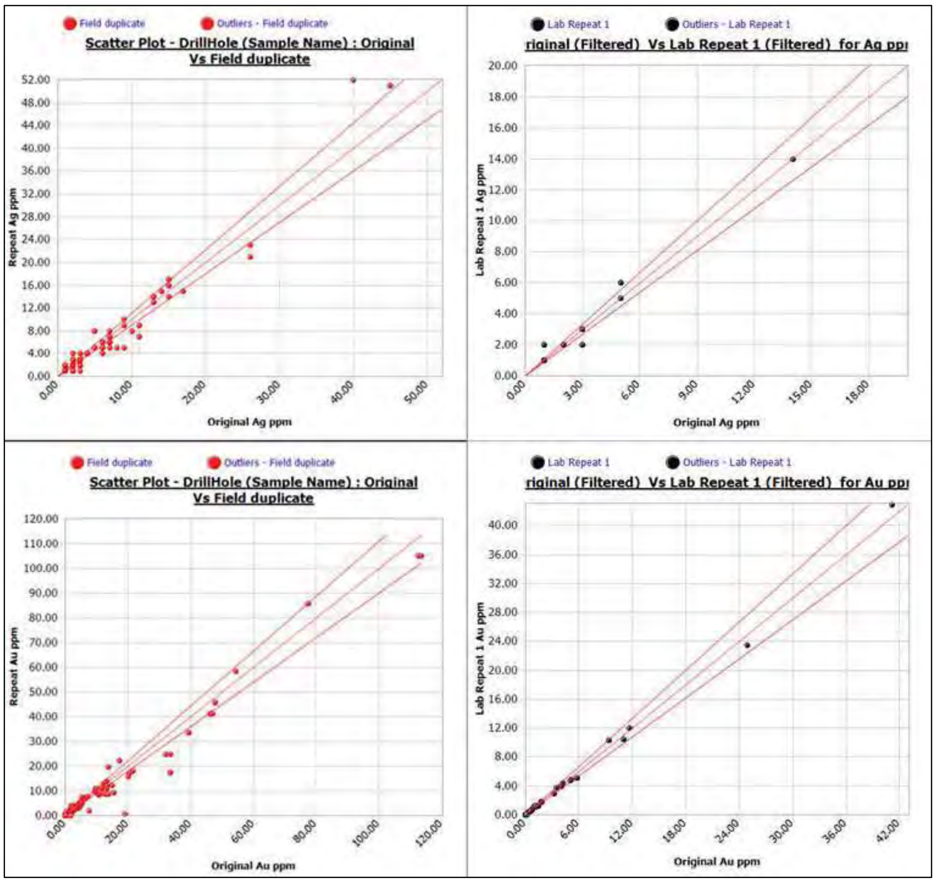

Figure 28. Scatter plots for duplicates analysed at SGS Welshpool (CSA)

CSA considers that Industry Standard acceptable levels of precision are reported for all of the sampling stages for the purpose of mineral resource estimation. Figure 28 shows scatter plots for gold and silver field duplicate and lab repeat pairs analysed at SGS Welshpool. A slight bias to the original in the gold field duplicates can be observed between 15 and 50 ppm gold, but overall no significant biases were noted.

Table 25. Summary of Data Precision SGS Welshpool Gold and Silver Assay Data

Summary of Data Precision SGS Welshpool Gold and Silver Assay Data									
Data Comparison	Number of Data Pairs			Mean % Hard			Median % Hard		
	Trenches / Channels	RC Chips	DDH Core	Trenches / Channels	RC Chips	DDH Core	Trenches / Channels	RC Chips	DDH Core
Gold Analyses									
Duplicate Field Samples	23	390	-	24.6	8.9	-	18.8	5.0	-
Duplicate Crusher Splits	-	-	41	-	-	4.2	-	-	3.9
Duplicate Pulp Splits	34	551	41	4.5	4.7	5.1	4.6	3.5	4.4
Laboratory Repeats	25	564	68	2.8	2.8	2.9	3.0	3.0	2.6
Silver Analyses									
Duplicate Field Samples	-	32	6	-	6.8	5.1	-	6.3	5.0
Duplicate Crusher Splits	-	-	-	-	-	-	-	-	-
Laboratory Repeats	-	62	8	-	3.0	3.6	-	0.0	4.1

12.2.3.4 SGS Chelopech

The precision of the gold assays for samples analysed at the SGS Chelopech lab was assessed based on the following datasets:

- Routine duplicate RC sample splits
- Routine duplicate splits of pulverised core, RC
- Routine laboratory repeat assay data for RC samples

Allowing for variations in the number of data pairs, similar levels of precision are reported for the SGS Chelopech gold assays for the different sample input sample types and downstream sample groups as those reported by SGS Gura Rosiei and Welshpool, with industry acceptable levels of precision evident for all of the sampling stages, as presented in Table 24 and Table 25. Industry acceptable levels of precision for the Chelopech silver assays are also evident based on the relatively small duplicate field sample and laboratory repeat data datasets, as presented in Table 26.

CSA considers that Industry Standard acceptable levels of precision are reported for all of the sampling stages for the purpose of mineral resource estimation.

Table 26. Summary of Data Precision SGS Chelopech Gold and Silver Assay Data

Summary of Data Precision SGS Chelopech Gold and Silver Assay Data									
Data Comparison	Number of Data Pairs			Mean % Hard			Median % Hard		
	Trenches / Channels	RC Chips	DDH Core	Trenches / Channels	RC Chips	DDH Core	Trenches / Channels	RC Chips	DDH Core
Gold Analyses									
Duplicate Field Samples	-	61	-	-	8.6	-	-	4.8	-
Duplicate Crusher Splits	-	-	-	-	-	-	-		-
Duplicate Pulp Splits	-	78	-	-	2.7	-	-	2.4	-
Laboratory Repeats	-	131	-	-	3.7	-	-	1.7	-
Silver Analyses									
Duplicate Field Samples	-	5	-	-	4.0	-	-	4.7	-
Duplicate Crusher Splits	-	-	-	-	-	-	-		-
Duplicate Pulp Splits		11	-	-	2.5	-	-	1.5	-
Laboratory Repeats	-	39	-	-	2.8	-	-	2.7	-

12.2.3.5 Inter-Laboratory Precision and Relative Accuracy

The inter-laboratory precision and relative accuracy between the primary laboratories (SGS, Chelopech, Gura Rosiei and Welshpool) and umpire laboratories (Genalysis, Perth and ALS Chemex, Vancouver) and inter-laboratory checks between different SGS laboratories (SGS Chelopech and SGS Welshpool) were assessed based on the primary and umpire laboratory assays of the duplicate pulp splits collected during the second and fourth exploration programmes at Ada Tepe. Minor instances of mislabelled samples were noted and omitted from the dataset. Oxide and sulphide standards were inserted between every 20th umpire sample and certified blanks between every 50th umpire sample.

All primary and umpire assay datasets for both gold and silver have been compared, as listed below:

- SGS Gura Rosiei primary assays versus Genalysis umpire assays for the second exploration programme.
- SGS Gura Rosiei primary assays versus Genalysis umpire assays for the third exploration programme.

- SGS Gura Rosiei primary assays versus ALS Chemex umpire assays for the third exploration programme.
- SGS Welshpool primary assays versus Genalysis umpire assays for the third exploration programme.
- SGS Welshpool primary assays versus ALS Chemex umpire assays for the third exploration programme.
- SGS Chelopech primary assays versus Genalysis umpire assays for the fourth exploration programme.
- SGS Chelopech primary assays versus ALS Chemex umpire assays for the fourth exploration programme.
- SGS Chelopech primary assays versus SGS Welshpool inter-laboratory (umpire) assays for the fourth exploration programme.
- Genalysis umpire assays versus ALS Chemex umpire assays for the third and fourth exploration programme, split into three groups based on the samples originally analysed at the SGS Gura Rosiei, or, SGS Welshpool labs, or, SGS Chelopech laboratories.

Relatively high levels of inter-laboratory precision are evident for the gold analyses between primary and umpire laboratories, as indicated by the mean %HARD values, which range from 4.23 to 7.03 (Table 27). In addition, there is no evidence of bias between the compared datasets, as indicated by the mean %HRD values, which range from -5.46 to 5.62. Excluding the Genalysis versus the SGS Chelopech umpire assays results in improved mean %HRD values that range between -1.93 and 1.00.

Table 27. Summary Statistics – Comparison of Primary and Umpire Laboratory Gold Assays

Summary Statistics – Comparison of Primary and Umpire Laboratory Gold Assays											
Statistic	**2002**	**2003-2004**									
	SG_-GR vs Gen	**SGS_GR vs ALS**	**SGS_GR vs Gen**	**SGS_GR vs ALS (Prim. SGS_GR)**	**SGS_WP vs ALS**	**SGS_WP vs Gen**	**Gen vs ALS (prim SGS_WP)**	**SGS_CH vs Gen**	**SGS_CH vs ALS**	**Gen vs ALS (Prim. SGS_CH)**	**SGS_CH vs SGS_WP**
Number	190	908	920	911	893	896	900	100	100	99	515
Linear Correlation	0.96	0.99	0.96	0.97	0.99	1.00	0.99	1.01	1.01	1.01	1.00
Rank Correlation	1.00	0.99	0.99	1.00	0.99	1.00	1.00	1.00	1.00	1.00	0.99
Mean %HARD	6.19	5.71	5.55	4.46	5.23	4.85	4.23	7.03	5.50	6.66	6.13
Median %HARD	4.34	3.74	3.45	2.86	3.63	3.44	2.91	4.42	3.64	4.90	4.65
Mean %HRD	-0.04	0.52	-0.02	0.60	-0.90	-1.93	1.00	-5.46	0.29	5.62	0.60
Median %HRD	0.08	-0.06	-0.26	0.31	-0.96	-1.71	0.56	-3.67	0.00	4.06	0.00

The primary and umpire silver analyses also show good levels of inter-laboratory precision, with mean %HARD values for the compared datasets ranging from 3.48 to 6.57 (Table 28). However, the %HRD values reported for different data comparisons indicate the ALS Chemex and SGS Welshpool silver assays tend to be marginally lower grade than the Genalysis and SGS Gura Rosiei silver assays, coinciding with %HRD values ranging from 2.40 to 3.50. It is not considered that this is a material issue which would affect the mineral resource estimation.

CSA reviewed the results of the interlaboratory assaying performed at Genalysis Perth and ALS Vancouver and whilst the above tables could not be replicated exactly, CSA considers that, on the basis of checks completed, the inter-laboratory precision and relative accuracy of the primary and umpire gold and silver datasets to be acceptable for the purpose of mineral resource estimation.

Table 28. Summary Statistics – Comparison of Primary and Umpire Laboratory Silver Assays >=5.0g/t Ag

Summary Statistics – Comparison of Primary and Umpire Laboratory Silver Assays >=5.0g/t Ag											
Statistic	2002	2003-2004									
	SG_-GR vs Gen	SGS_GR vs ALS	SGS_GR vs Gen	SGS_GR vs ALS (Prim. SGS_GR)	SGS_WP vs ALS	SGS_WP vs Gen	Gen vs ALS (prim SGS_WP)	SGS_CH vs Gen	SGS_CH vs ALS	Gen vs ALS (Prim. SGS_CH)	SGS_CH vs SGS_WP
Number	55	215	231	215	170	178	171	17	17	17	53
Linear Correlation	0.99	1.00	0.98	0.98	1.00	1.00	1.00	1.06	1.06	1.06	1.01
Rank Correlation	1.00	0.99	0.99	0.98	0.99	0.99	0.98	1.02	1.02	1.04	1.00
Mean %HARD	6.57	4.49	4.83	5.94	4.41	6.17	5.50	4.06	3.96	3.48	4.25
Median %HARD	5.66	3.29	4.06	5.47	3.83	5.14	4.48	3.64	4.00	2.36	4.35
Mean %HRD	-3.59	2.40	-1.09	3.50	-0.43	-3.84	3.57	-0.36	-0.82	-0.89	0.73
Median %HRD	-2.91	1.89	-1.14	3.68	-0.10	-4.07	3.00	0.00	0.47	0.61	0.00

12.3 Grid Control and Survey

12.3.1 Introduction

All surveying at Ada Tepe is conducted using the Bulgarian National Coordinate System (BNCS), a close variant of the Stereo 70 system. The BNCS divides the country into roughly quarters, with the BG5 zone covering the SE quadrant of Bulgaria, including the Krumovgrad area.

12.3.2 Survey Control Audit

In May 2002, Australian surveying group Spectrum Surveys and Mapping Pty Ltd ('SSM') were contracted to conduct an audit of the survey control at Ada Tepe prior to further exploration being undertaken. The activities undertaken and results, as provided in the SSM report of the audit, are summarised below.

Audit surveying activities were conducted using two geodetic accuracy GPS units, with one unit used as a base station positioned at a known (Bulgarian) trigonometric station, and the other used to investigate and establish control points at various locations in the License area. Most investigations were undertaken in rapid static mode, however, spot checks on existing control points and drillhole collars were also conducted using the Real Time Kinematic (RTK) function on the GPS.

The following surveying activities were undertaken during the audit:

- Investigation of the Bulgarian Government 1970 Grid system to evaluate its suitability for the future of the mining and exploration program.
- Verification of existing government survey control points.
- Audit of the integrity of existing local exploration survey control points.
- Establishment of 11 new high order survey control points throughout the Kumovgrad Licence area (five at Ada Tepe).

The following key observations and conclusions are provided in the audit report produced by SSM:

- There are no published geodetic parameters for the 1970 grid or the level datum.
- The 1970 system is a non-earthed (planar) grid system based on four separate zones on which there is co-ordinate overlap. Geodetic corrections are applied to surveys to obtain the reduced co-ordinates.
- A closed static GPS survey of seven government survey control points distributed over the License area indicated that all points surveyed agree to within centimetres of the Bulgarian government supplied horizontal UTM co-ordinates, and the levels also agree within centimetres at all control points surrounding the Ada Tepe and Surnak prospects.
- The accuracy of the local survey control points at Ada Tepe checked by RTK GPS indicate that the government control point at the top of Ada Tepe hill gives reduced coordinates within 0.12 m horizontal and 0.1 m elevation of the specified location. However, the accuracy of various additional survey points checked steadily diminishes with increased distance away from this point.
- For optimal survey control the reference system for Ada Tepe and the Krumovgrad Licence should change to a planar UTM system using a central point as the origin which has the same co-ordinates in both planar UTM and real world UTM and a common reference bearing.

12.3.3 Routine Surveying

All surveying of the surface topography and exploration sites at Ada Tepe were carried out by a government licensed contractor, Dimiter Motrev of Geocom Ltd using the survey control established by SSM during the 2002 survey audit. All surveying was conducted using two electronic total station instruments. Geocom has established a dynamic net of some 183 survey stations covering the Ada Tepe hill using the five survey control points established by SSM during the 2002 survey audit. All of the Geocom survey stations have been established on closed survey loops.

12.3.4 Topography

Topographic control over the Ada Tepe prospect area has been established on the basis of the following surveying activities:

- Numerous ground traverses.
- Surveying of all drill access roads, including crest and toe locations in regions of cut and fill.
- Surveying of all trenches.
- Surveying of all drill collars.

The resultant array of survey points forms at least a notional 25 mE by 25 mN grid over the entire prospect area. However, additional survey points along the drill access roads, drainage gullies and flanks of the Ada Tepe hill result in a closer spacing of topographic control points over much of the prospect area. Geocom has generated 2 m topographic contours covering the Ada Tepe prospect area based on the array of survey control points current through the end of August 2004. The 3D coordinates of all survey points, the 2 m contour data, and the breakline string data defining the crest and toe positions of the drill access roads were supplied to BMM and RSG Global for review and development of a topographic model for use in mineral resource modelling.

RSG Global has conducted a detailed review of the supplied topographic data for Ada Tepe and identified no material inconsistencies between supplied topographic datasets. A 3- dimensional wireframe model of the topography with no tree canopy effects was subsequently constructed based on the 2 m contour and breakline datasets. RSG Global considers this model to be of sufficient accuracy for mineral resource modelling purposes. Figure 29 displays a perspective view of the topographic model of the Ada Tepe prospect.

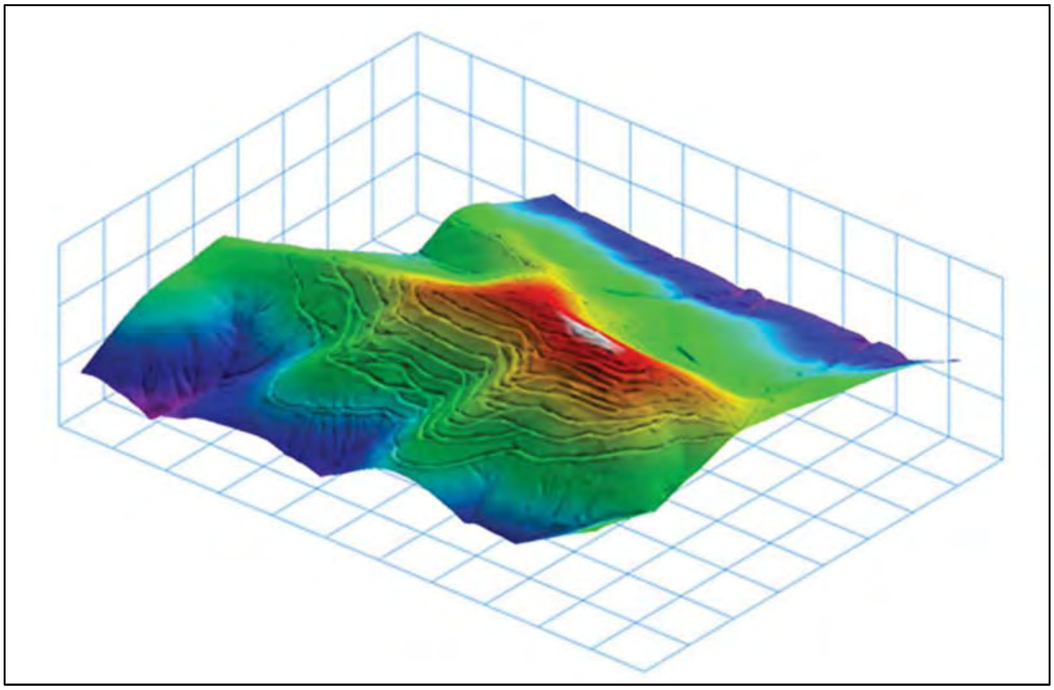

Figure 29. Perspective View of the Topographic Model at the Ada Tepe Prospect

CSA has taken receipt of, and reviewed this topographic surface and the trench/collar points used in its construction and believes it to be valid for use in constraining the Mineral Resource block model.

12.3.5 Drillhole Collar and Trench Locations

The preserved drillhole collars from the 2000-2001 diamond drilling and all drillhole collars from the 2002 through 2004 drilling at Ada Tepe were surveyed on the basis of the SSM established survey control. All channel sampled surface trenches and road cut exposures were also surveyed on the basis of SSM survey control. Any non-preserved drillhole collars from the 2000-2001 programme and trench sampling completed over the same period were surveyed based on the pre-audit survey control. The results of the audit and the RSG Global cross checking of the location and elevation of proximal early and more recent exploration sites suggest that the location accuracy of the earlier exploration sites is to industry accepted standards.

A consistent approach has been used to survey continuous channel sampling along trenches and road cut exposures. This involves surveying the start and end points of a trench or channel, and all intermediate points where the azimuth or dip of the trench or channel changes. The resultant survey coordinates were used to calculate azimuth and dip values for the surveyed positions along each channel, allowing the trenches and channels to be treated as pseudo- drillholes for mineral resource modelling.

CSA considers the drillhole collars, trench and channel sample locations at Ada Tepe to be accurately located in three dimensions for the purposes of mineral resource estimation.

Figure 8 displays the drillhole collar locations colour coded by drilling programme and method, while the locations of the surface trench and channel sampling are displayed in Figure 9.

12.3.6 Downhole Surveying

Downhole surveying was routinely conducted using Eastman single shot camera, Tropari, or Sperry Sun multi-shot instruments. Survey measurements were recorded at downhole intervals typically ranging from 25 m to 50 m. Some 580 of the 614 drillholes (94%) and 90% of the total drilling metreage have been surveyed. Downhole deviations from the drillhole collar azimuth and dip measurements are typically small, primarily due to the shallow depth of the great majority of the drillholes.

CSA considers the downhole orientations of the drillholes and 3-D locations of the drill samples are considered to be accurately located for the purposes of mineral resource estimation.

12.4 Bulk Density

A review of bulk density data undertaken prior to the 2002 exploration programme managed by RSG Global indicated that inappropriate techniques had been used (water immersion without sealing and air pycnometer) such that a density reading closer to the specific gravity rather than the in situ bulk density had been measured. As a result, all pre-2002 density data was rejected and collection of new bulk density data based on diamond drill core and trench grab samples collected from March 2002 onwards was initiated. Bulk density measurements were routinely collected from core billets at approximately 3 m down intervals and trench grab samples collected at 5 m intervals.

All bulk density measurements were completed by an ISO 9002 rated laboratory, Evrotest Kontrol, in Sofia using an ISO 9002 approved method of wax sealed water immersion bulk density measurement. The laboratory and the technique used were inspected by RSG Global prior to the beginning of routine sample submissions for density analysis from Ada Tepe. A total of 5,764 bulk density measurements are available for the Ada Tepe deposit covering all of the major rock types and variations in oxidation and weathering at locations distributed throughout the deposit. In addition to the bulk density determinations, a suit of tests were completed to ascertain the residue moisture in the core billets (53 oxide samples and 10 fresh samples). Negligible moisture was recorded, with the average moisture for both oxide and fresh samples being <1%.

12.5 Verification Sampling

Substantial secondary verification sampling of the primary trench/channel sampling and drilling at the Ada Tepe deposit was completed, including the following:

- Twinning of four 2001 diamond drillholes in 2002 with 366 m of triple tube coring.

- Twinning of 194 m of the initial 2000 and 2001 channel sampling in 2002 using the RSG Global channel sampling procedures.

- Successful twinning of 24 RC drillholes from the second and third drilling programmes with triple tube coring (1,865 m).

12.5.1 Analysis of Twin Trench Sampling Data

The 194m of twin trench sampling was completed along various segments of 13 trenches, with the twin samples collected over the same intervals as the original samples. The average gold grade of the original and twinned segments of continuous of trench sampling has been calculated, both uncut and with a 10 g/t Au upper cut applied, as presented in Table 29. The compared intersection grades are generally similar with no systematic bias evident in favour of the original or twin intersection grades. On this basis, it is considered the trench sampling data collected during the initial exploration programme to be suitable for use in mineral resource estimation.

CSA, having reviewed the results of twin trench sampling, agrees with the conclusions reached by RSG.

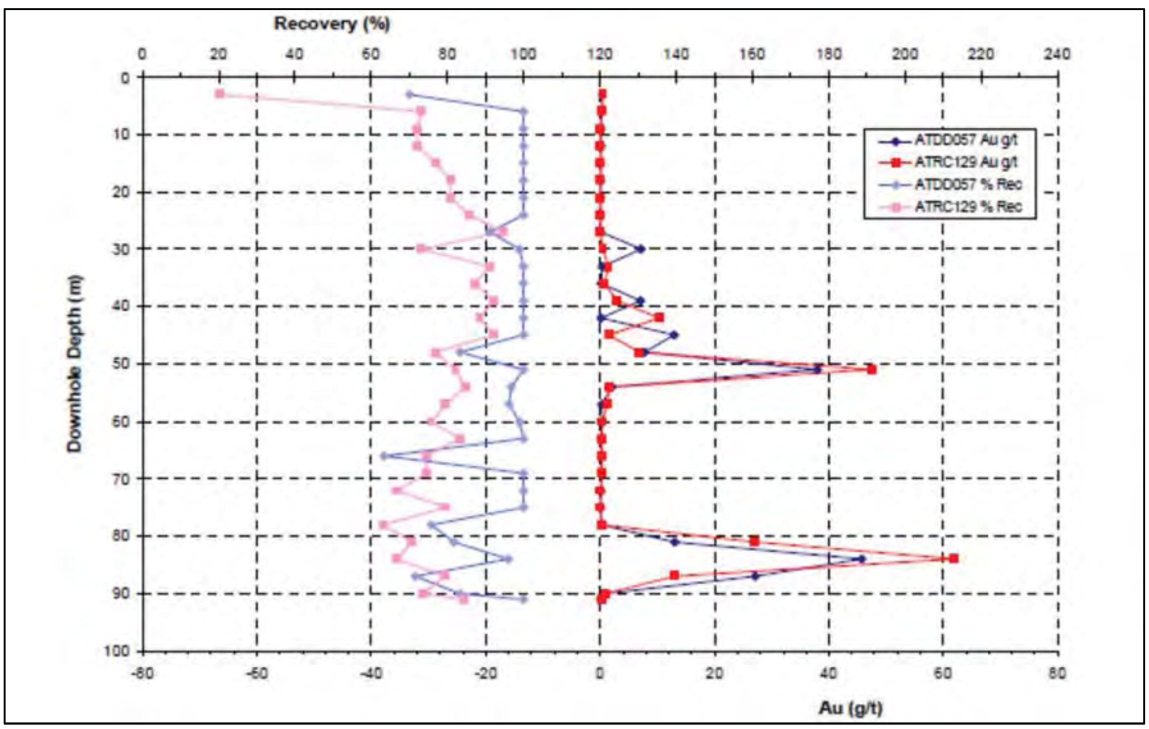

Figure 30. Grade and Recovery versus Depth Plot for Diamond and RC Twin Drillholes ATDD057 and ATRC129

Table 29. Uncut and Cut Mean Gold Grade Data for Twin Trench Sampling Intersections

Uncut and Cut Mean Gold Grade Data for Twin Trench Sampling Intersections							
Trench ID	From	To	Metres	Uncut Mean Gold Grade (g/t)		Cut Mean Gold Grade (10g/t Au)	
				Original	Twin	Original	Twin
AT102-6	57.8	65.4	7.6	1.0	1.9	1.0	1.8
AT108	23.2	50.8	27.6	10.2	9.7	5.3	4.6
AT110-1	17	27	10	0.3	0.5	0.3	0.5
AT110-2	6	36.1	30.1	11.3	12.9	5.1	4.8
AT121-1	72.7	79.4	6.7	2.5	3.4	2.5	2.9
AT133	34.9	42	7.1	13.9	17.1	5.1	5.5
AT149	0	23.7	23.7	0.2	0.2	0.2	0.2
AT149-1	0	5	5	0.1	0.2	0.1	0.2
AT156	79.7	84	4.3	55.3	132.1	6.7	10.0
AT182	6	15.2	9.2	3.5	2.1	3.5	2.1
AT187	0	11.4	11.4	5.9	2.4	1.9	1.7
AT187	28.6	45.8	17.2	13.4	9.6	2.4	2.9
AT194-2	82.8	104.7	21.9	1.7	1.1	0.9	1.0
AT196	0	3.6	3.6	0.9	0.7	0.9	0.7
AT196	11.1	19.9	8.8	5.5	3.4	2.6	2.5
Total	N/A	N/A	194.2	7.3	8.6	2.8	2.8

12.5.2 Analysis of Twin Core Hole Data

Comparison of the original and twin diamond drillhole intersections (Table 30), both uncut and using a 10 g/t Au upper cut, indicates the original core hole intersections are higher grade than the twin hole intersections, but with much reduced differences evident between the intersections based on the cut grade data. Plots of the sample gold assay grades versus downhole depth for each of the four twin pairs indicate that the high grade mineralised intersections are of similar length in the original and twin holes. Furthermore, not all of the mineralised intersections are dominated by higher sample grades in the initial drillholes. RSG Global also noted that the separation between the surveyed downhole traces of the twin pairs AT1045 and ATDD014, and AT1079 and ATDD015 range from approximately 2 m at the top of the holes to greater than 9 m at depth, and as such, differences in the sample and mineralised intersection grades at depth are not unexpected.

In summary, when the effects of a relatively small number of very high grade samples are removed by cutting, and the significant separation between some of the original and twin drillhole traces are considered, the length and tenor of the mineralised intersections in the original core holes is generally confirmed by triple tube twin holes.

CSA, having reviewed the results of twin core sampling, agrees with the conclusions reached by RSG Global.

Table 30. Uncut and Cut Mean Gold Grade Data for Twin Core Holes

Uncut and Cut Mean Gold Grade Data for Twin Core Holes								
Original Hole	Twin Hole	From	To	Metres	Uncut Au (g/t)		Cut (10g/t Au)	
					Original	Twin	Original	Twin
ATI 012	ATDD011	3	59	54	18.5	10.7	4.8	4.8
ATI 059	ATDD012	0	66	65	1.8	0.7	1.5	0.7
AT1045	ATDD014	1	107	1 06	4.6	2.3	1.5	1.2
AT1079	ATDD015	0	130	130	1.7	1.6	1.2	1.1
Total	Total	N/A	N/A	355	5.1	3.0	1.9	1.6

12.5.3 Analysis of RC and Diamond Twin Hole Data

The 24 successful RC and diamond twin drillhole pairs have been compared on the basis of 3 m composites of the gold assay data over the twinned downhole segments of each twin hole pair. Similar to the diamond twin holes and the duplicate channel samples, a high level of short scale variability is noted, as is typical for epithermal gold systems. The important findings of the RC versus DDH twin drilling review are summarised as follows:

- The mineralised intersection lengths, their relative downhole positions and the associated sample recoveries show reasonable correlation for the majority of twin hole pairs, as illustrated in the example plot displayed in Figure 30.

- A high degree of grade variation is noted between the mineralised intercepts (see Table 31). Although a larger percentage of RC holes are relatively higher tenor (consistent with that described above for the DDH versus DDH twin dataset), a significant number of diamond intercepts are also noted to be higher grade than the corresponding RC twin. This would support the finding of high short-scale variability and not bias.

- Small intercepts (<= 3 m) identified in the RC or the DDH twin are often not repeated in the adjacent drillhole.

It is considered that the twin drilling adequately replicates the original drillhole intercepts in terms of magnitude and location, when consideration is given to the mineralisation style.

CSA agrees with the conclusion reached by RSG.

Table 31. Comparison of Uncut and Cut Gold 3m Composites Reverse Circulation versus Diamond Drillhole Twins

Comparison of Uncut and Cut Gold 3m Composites Reverse Circulation versus Diamond Drillhole Twins								
DDH Hole	RC Hole	From	To	Length (m)	Uncut Au (g/t)		Cut (10g/t Au)	
					DDH	RC	DDH	RC
ATDD026	ATRC013	3	54	51	2.83	7.38	2.83	7.03
		75	80.3	5.3	17.11	3.68	16.64	3.68
ATDD030	ATRC003	0	36	36	3.68	2.03	3.68	2.03
		123	137	14	1.69	1.17	1.69	1.17
ATDD032	ATRC002	0	9	9	2.02	2.41	2.02	2.41
		1 05	111	6	1.14	4.68	1.14	4.68
		120	131.6	11.6	3.87	2.74	3.87	2.74
ATDD034	ATRC007	60	93	33	7.89	6.63	7.76	6.63
ATDD057	ATRC129	27	60	33	6.99	6.85	6.26	5.26
		78	90	12	21.72	25.81	17.76	17.80
ATDD058	ATRC062	15	20	5	0.40	31.85	0.40	29.67
ATDD059	ATRC059	0	21	21	7.48	6.74	7.48	6.74
ATTDD060	ATRC132	15	24	9	0.47	0.61	0.47	0.61
ATTDD061	ATRC074	0	12	12	9.06	13.80	8.43	10.11
		45	66	21	0.48	0.42	0.48	0.42
ATDD073	ATRC117	0	33	33	15.16	18.32	8.01	6.53
		45	51	6	0.13	1.46	0.13	1.46
		63	108	45	5.07	5.63	4.75	5.33
ATDD074	ATRC118	12	66	54	30.12	60.07	11.21	15.55
		81	109.7	28.7	4.04	11.40	4.04	5.71
ATDD077	ATRC273	66	70	4	0.21	15.95	0.21	15.95
ATDD078	ATRC339	0	18	18	0.97	20.51	0.97	5.64
ATDD081	ATRC143	9	15	6	0.73	0.31	0.73	0.31
		33	81	48	8.37	11.57	5.67	8.90
ATDD083	ATRC081	0	18	18	0.84	0.87	0.84	0.87
		24	39	15	5.37	6.65	5.37	6.65
ATDD085	ATRC111	0	15	15	6.25	7.94	6.25	7.94
		27	68.2	41.2	6.17	17.24	6.17	10.76
ATDD087	ATRC097	15	27	12	27.80	5.10	7.77	5.10
		54	87	33	1.96	2.81	1.96	2.81
ATDD088	ATRC135	6	18	12	1.93	5.26	1.93	5.26
		27	45	18	6.00	3.75	6.00	3.75

Comparison of Uncut and Cut Gold 3m Composites Reverse Circulation versus Diamond Drillhole Twins								
DDH Hole	RC Hole	From	To	Length (m)	Uncut Au (g/t)		Cut (10g/t Au)	
					DDH	RC	DDH	RC
ATDD089	ATRC357	0	54	54	0.55	12.57	0.55	4.37
		69	85.5	16.5	25.51	11.37	10.79	9.42
ATDD091	ATRC270	42	66	24	1.66	1.65	1.66	1.65
ATDD092	ATRC016	0	18	18	1.61	1.63	1.61	1.63
		27	78	51	5.98	11.20	4.40	3.79
ATDD101	ATRC335	0	12	12	0.78	2.34	0.78	2.34
		18	24	6	0.98	0.03	0.98	0.03
ATDD105	ATRC140	15	30	15	1.47	1.14	1.47	1.14
		57	63	6	5.02	0.24	5.02	0.24
		90	112	22	21.86	11.28	14.38	11.28
Note: Selection completed on the basis of >2 composites and a minimum grade of 0.5g/t Au for either/or original and twin hole.								

12.6 Site Visit – Geology and Resources

12.6.1 Summary

At the request of Edgar Urbaez, Corporate Director Technical Services - DPM, CSA Principal Geologist and Managing Director, Mr Galen White completed a desktop review of available data and literature pertaining to the Ada Tepe gold Project, Krumovgrad, Bulgaria which was augmented by a site visit to the project between 5th and 9th November 2012 (CSA, 2012). Within this context CSA was asked to review available information relating to the Mineral Resource Estimate contained in the NI 43-101 Technical Report for the project (RSG, 2005), and to essentially peer review this work to confirm the suitability of the Mineral Resource Estimate block model for use as an input in to planned and refined pit optimisation and mine planning work. In addition DPM required CSA to take ownership of the resource model and to provide whatever assistance, including sensitivity analysis, which may be required to assist mine planning work.

To meet these objectives, CSA focussed on;

- Reviewing the contents of the 2005 and 2012 Technical Reports, specifically those sections that detail the methodology employed to complete the Mineral Resource Estimate, the input data, parameters and assumptions used and to check that these are robust and honour both the underlying data characteristics and the geological model for the project;

- Loading and validating the Drillhole and Trench Database, block model, 2D wireframe surfaces and 3D geological and mineralised domain wireframes;

- Tying these reviews to an on-site inspection of project geology, drill core, data collection protocols and communication with DPM project staff.

All data and literature provided by DPM was reviewed prior to undertaking the site visit and referenced additionally on site. The more pertinent data reviewed included;

- An NI 43-101 Technical Report prepared by RSG in 2005 which details the MRE.
- An NI 43-101 Technical Report document prepared by DPM in 2012.
- The RSG block model file, exported from Vulcan software to a .csv file.
- A Drill Hole and Trench Database for the Ada Tepe project, which includes collar, assay, survey, geology, structure and density data.
- Oxidation surface wireframes as .dxf files.
- Geology and mineralisation 3D domains as .dxf files.
- Constraining wireframes for grade constraining, interpolation and classification.

During the site visit, the following was achieved;

- The project and local geology characteristics were discussed with DPM staff who were involved in drilling and sampling programs between 2000 and 2004.
- An inspection of the geological sequence over the project, through field excursions to road side outcrop and trenches.
- A review of continuous (top of hole to bottom of hole) drill core from ATDD006, 023, 073, 085 and 097, being representative of mineralised zones encountered over the project and of the drilled deposit (spatial representivity). This review was undertaken with reference to hard copy graphical logs of geology, sampling and assay data.
- A review of the database held for the project (hosted in acQuire software), spot checks against hard copy assay certificates and review of data collection procedures and protocols adopted during data collection activities.

12.6.2 Conclusions and Recommendations

Conclusions from the work undertaken by CSA at that time included:

- The view that the Mineral Resource Estimate and supporting documentation has been completed to a high standard and all steps of the resource estimation process were well documented, well-reasoned and honour the geological observations, grade characteristics and probable mining scenarios that were being considered.

- CSA was able to interrogate and validate the grade model and replicate reported figures of tonnage and grade quoted in the 2012 Technical Report. In addition CSA were able to verify all of the geological information contained in the reports via on-site inspection, which further supports the geological model proposed for Ada Tepe.

- During the review, core from several drill holes were viewed in conjunction with 3D drilling data on screen in Micromine software, as well as drill hole log printouts from acQuire. Good examples of mineralisation styles were noted and logged lithological and mineralised zones were verified and the logging found to be of a detailed standard.

- The Mineral Resource Estimate is considered reliable and robust for the purposes of mine planning.

- Resource tonnage is considered reliable. Density assignation has been informed by significant surface and sub-surface sampling such that average domain densities are adequate. However, a review of logged oxidation and therefore the interpretation of oxidation surfaces, used to flag the density model may require sensitivity review to aid improvements in the tonnage model.

- CSA has sufficient confidence in the Mineral Resource Estimate to be able to assist DPM in any required sensitivity analysis on the model, and ultimately take ownership of the model during current and future mine planning activities.

- Data collection practises and procedures currently being employed, and that were employed prior to the Mineral Resource Estimate are of a high standard, and in some aspects exceed Industry standard. Data QA/QC is well documented and suggests that grade data is both accurate and precise, and that geological information captured is pertinent and thorough.

- Resource augmentation potential at Ada Tepe is low, however there are several other prospects in the local area that may be amenable as satellite projects. A brief discussion on site covering these prospects suggests that only Surnak contains significant potential to warrant further exploration to define significant additional gold ounces.

12.7 Site Visit – Mining

No visit has been made to site by the mining engineer, since no construction work has taken place and no mining facilities exist at present. The visit of the mining engineer consisted of a 5 day period in the Sofia office of DPM, during which time, in-depth discussions took place with the project's main personnel, data and information was studied, reviewed and information was freely exchanged.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

Three distinct phases of testing have been undertaken in the evaluation of the mineralization present in the deposit at Ade Tepe. In summary, these contributions were:

- 2005 DFS – The basis of the program was to develop an industry standard gold extraction process, The AMMTEC Laboratory, Perth, Australia, was selected to conduct the physical characterisation, comminution, leaching and cyanide detoxification testwork programmes. Samples were also sent to; MinnovEX Technologies (now SGS) in Toronto, Canada for Comminution characterisation and variability testing; for mineralogical examination; Outokumpu Technologies for thickening, and Larox Pty Ltd for filtration; and to Coffey for tailings characterisation. Results from this program confirmed that all the samples tested were considered “free-milling” and amenable to gold production by conventional cyanidation

- The 2012 DFS update essentially reinvented the project following the rejection of the original investment proposal by the local community and government authorities. At the expense of a reduction (8 - 10%) in recovery compared with the original and conventional cyanide leach circuit, the project was ‘re-engineered’ using a more conventional flotation process, combined with the introduction of a combined mine waste and flotation tailings facility (Integrated Mine Waste Facility – IMWF). The process evolved from the 2005 flotation scoping testing which demonstrated that at the CIL circuit grind (P80 of 75 µm) between 60% and 80% of the gold could be recovered to a flotation concentrate. The 2009/11 testing program was developed to confirm the potential of both a) physical recovery processes (flotation and gravity) as the primary method of precious metal concentration, and b) the ultimate integration of high-density (or “paste”) settled tailings from the process into an overall waste deposition strategy which incorporates the mine waste.

In 2012/2013, the bulk of the testing program was confirmatory, mainly the dispatch of appropriate samples to recognised testing institutions for mechanical design tests, including - materials handling flow characteristics, slurry rheology determinations, additional confirmatory settling and paste thickening testing. Several confirmatory flotation programs continued at SGS.

13.2 Ore Characterisation – 2009 to 2013

The SGS laboratory in Lakefield, Ontario was selected as the testing facility for the supplementary comminution, and the flotation development programs. A comprehensive test program was undertaken for this program, samples for which were taken from the extensive store of HQ, NQ and PQ drill core. The locations were selected to cover the range of mineralization present across the zones, to include composites and sufficient samples to determine the extent of any variability in performance.

The test program focused on the two main ore zones; the "softest" – the Oxidized Upper Zone and the "hardest", the Fresh Wall Zone. Composites of each were examined, and in addition, individual samples for variability testing, were also tested. The samples were selected in consultation with the exploration team from the remainder of the HQ, NQ and PQ drill core available. Most of the samples were split from the existing half core H- and NQ material, into quarter core.

13.3 Gold Occurrence

The 2005 study concluded that the Gold was predominantly present in the ore as fine, and sometimes, liberated particles of Electrum. The concentration of sulphides and base metals were very low; and two dominant ore lithologies were present - the 'brecciated' Upper zone and the 'siliceous' Wall zone. Subdivisions of these (oxidised or fresh) indicating the degree of weathering that had occurred were examined and the subsequent metallurgical and comminution testwork programmes confirmed only minor variations in the physical characteristics within each subdivision. However, substantial differences in overall "hardness" between the two main ore zone classifications were noted.

The overall geology, ore zones, and lithologies remained unchanged for the 2009/10 study, and these were considered in the selection of the remaining samples. As part of the characterisation process, a gold deportment study (SGS Canada, Project 12251-002) was carried out on selected concentrates from the two main composite samples ground to a P80 of approximately 35 microns. The bulk of the gold was found to be present as electrum, and with average equivalent diameters of 7 and 5 microns for the Upper oxide and Wall particles respectively. The Wall samples are slightly finer overall (maximum size 14 microns), compared to a maximum of 25 microns for the upper Oxide. Approximately two thirds of the gold particles for the two composites were recorded as liberated (>95% free mineral), and attached (80 – 95%), with the remaining being locked (<80% free). This confirmed the relatively fine grind requirement to achieve liberation and subsequent recovery by physical methods.

13.4 Comminution

To confirm the extent of any variations in the hardness characteristics, additional tests were carried out to supplement the original (2005) comminution testwork programme. These were conducted on the Upper Oxide and Fresh Wall Zone composites and variability samples which showed the widest range of characteristics. The full ranges of tests were completed, which confirmed the original findings that the Upper Oxide samples were moderately competent and abrasive, while those of the Wall were relatively much harder and more abrasive.

Together with the standard tests for grindability determinations undertaken in the 2005 program, 32 individual samples in total have been tested for SAG Power Index (SPI) and Bond Ball Mill Work Index (BWi) measurements. SPI is a measure of the hardness of the ore from the perspective of semi-autogenous milling. The Bond Work Index characterizes the ore hardness with respect to ball milling, with the Modified Bond (ModBond) method being used on all the 32 samples. Four full Bond Index measurements were made in 2005 to calibrate the Modified Bond results on the first 14 samples at a P100 of 106 microns. A further 6 full Bond Index measurements were made in late 2009 to calibrate the 18 additional samples at a P100 of 75 microns.

As the test program evolved and it became clearer that a significantly finer grind than the original 2005 DFS circuit design required, additional testing was completed to determine the relative parameters to properly size the equipment capable of producing the grind sizes required. A single composite was subjected to a full Bond test at both P100 of 75 and 38 microns to account for the significantly finer mill product size required for the new circuit. Further samples were sent to Metso (York, Pennsylvania) for supplementary fine grinding tests, which confirmed the appropriate engineering assumptions to be used for the fine grinding (tertiary) milling circuit. The additional grinding test results obtained confirmed the range of hardness previously measured, reflected in the primary and secondary mill specific power requirements, which remained unchanged from the 2005 design.

13.5 Flotation

The flotation development program was also conducted on the samples from the same zones, representing the majority of the ore expected from the deposit, and carried out in several phases.

The 2009/10program concentrated on developing the grind/recovery relationships for flotation, together with the development of the optimum circuit. Following the initial 2005 scoping work and several ‘proof of concept’ phases, a significant sampling program on drill core covering the extent of mineralization was designed to cover the range of ore zones covered in the earlier characterisation program. An additional 20 individual intervals of the upper Oxide material, and 12 individual samples of the Wall were selected, and prepared at SGS. A number of these were combined into composites for the development program, and also individually tested to determine the effects of variability.

The test programs took place in several phases over the period between September 2009 and May 2010. The bulk of the development work took place on main composites (UpOx and Wall) of individual samples from each of the two ore types (20 for the Upper Oxide and 11 for the Wall). A wide range of flotation parameters, including grind size, reagent optimization, Rougher, Rougher/Cleaner (both in open circuit and locked cycle), and kinetic tests were undertaken through the program, with the main conclusion being that overall performance was very similar for each ore type, and both were completely dependent on grind size. The overall results showing the clear trends of improving performance with increasing fineness of grind is summarized in Figure 31Figure 31 and Figure 32. This also includes the results of the two main locked cycle tests (LCT) carried out on the main composites.

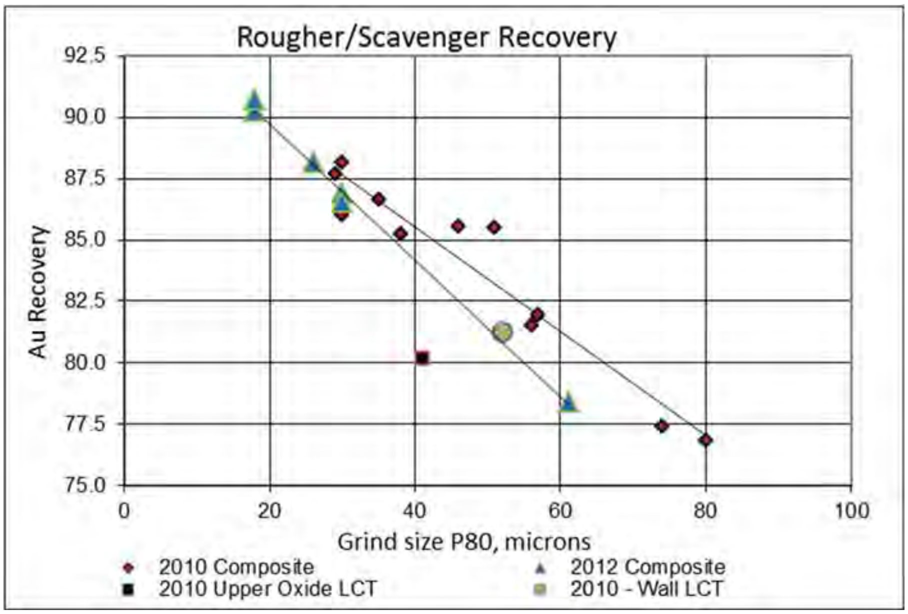

Figure 31. Summary Grind Size – Rougher/Scavenger Flotation Recovery of the Main Composites

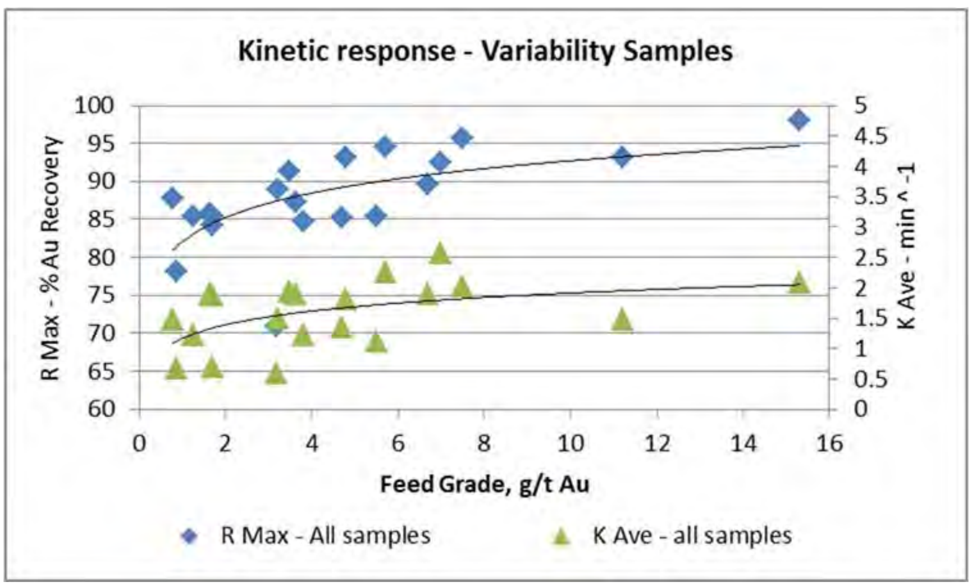

Figure 32. Kinetic Response of the variability samples tested in 2010

13.6 Tailings

The tailings thickening work was performed on a number of samples from each at a variety of grind sizes. The bulk work was completed on the worst case sample (most oxidized) to ensure the dewatering design parameters would be adequate under all operating conditions. Drill core samples representing the various lithological ore zones were prepared in batches to the various grind sizes required for testing for thickening and “high density” tailings (“paste”) production. This was performed by Golder Pastec in Sudbury, Ontario, and Burnaby, British Columbia, and the work undertaken included Material Characterisation (sizing and mineralogy), Dewatering Characteristics (achievable solids concentrations, yield stress and viscosity, and Rheological properties. Specific products from these programs were also sent to several thickener and filter suppliers for “pilot” plant characterization as the basis of design and subsequent quotations for the various unit process selections.

13.7 2012 Comminution and Flotation Circuit Design

To complement the physical testing programs, additional engineering studies were completed by SGS Canada, which confirmed the original grinding circuit characteristics, and developed the flotation circuit requirements. These studies utilized the basic testwork characteristics, which for Comminution were: individual sample rock hardness from primary, secondary and tertiary grinding mill perspectives – completed with the proprietary Comminution Economic Evaluation Program (CEET2®) (SGS Canada, Nov 2010) ; and for Flotation were individual kinetic parameters, to design the appropriate flowsheets – Flotation Economic Evaluation Program – FLEET2® (SGS Canada, Feb 2011).

Both of the studies involved the geostatistical estimation of the relative parameters distributed to the total blocks of the mine block model. On the basis of the annual mine plan, the circuit production estimates were then produced on a year by basis to confirm the capabilities of the selected equipment to process the variations in parameters on an annual basis . The geostatistical approach allows for the estimation of the precision (statistical error) in the block values, thereby improving the reliability of the production forecasts.

Using the design requirements of required throughput and final grind size, the required specific energy for each of the comminution circuit stages were predicted, which are then used as the minimum requirements for the engineering design. SGS add appropriate safety factors for drive losses and a margin for the orebody uncertainty, and complete a power requirement for the particular circuit studied. Similarly, the flotation circuit performance predictions indicated that, on average, a concentrate grade of 250 g/t Au could be produced at 86% recovery over the LOM from ore grade of 3.36 g/t Au. Monte Carlo simulations were used to determine the risk of error in the forecast results arising from lack of precision in the estimation of head grade and related kinetic parameters for different periods of operation.

Taking into account these uncertainties, the value for gold recovery used for the purposes of the 2012 study was 85%, to a concentrate grade of 200 g/t.

14 Mineral Resource Estimates

The following section describes the methodology, parameters and key assumptions regarding the preparation of the Mineral Resource Estimate contained in the 2012 Technical Report, which CSA considers to be current. CSA has reviewed the methodology, parameters and assumptions used in its creation and consider the estimate of resources reliable and suitable for inclusion in the recently updated mining study work.

The information in the following sections is taken largely from the 2012 Report and summarises the work of RSG, which CSA considers valid. Additional comment by CSA following validation review work is also presented in the following sections along with a summary of Mineral Resource validation (Section 14.7) and steps taken by CSA to modify the RSG block model (regularisation of blocks) to be more user-friendly in subsequent mine planning work (Section 14.7.2). It should be noted that during this regularisation work the underlying grade interpolation results have not been changed, and the underlying grade interpolations of RSG are honoured in the CSA model. A table of CIM compliant Mineral Resources for the project, prepared by CSA is contained in Section 14.8.2.

14.1 RSG Database Development

14.1.1 Introduction

The database for the Ada Tepe deposit was compiled by BMM technical staff on site and independently validated by RSG Global geological staff in Krumovgrad and Perth. The data from the initial two exploration programmes was compiled on site using Microsoft Excel, Gemcom, Microsoft Access, and Micromine software. In 2003, the then current database was transferred into a full relational database using the acQuire software product. Most of the data from the 2004 exploration programme was compiled and validated on site using acQuire.

The database trench and drillhole information for the Mineral Resource Estimate was exported by the BMM site database manager to a series of comma delimited files and forwarded to the RSG Global Perth office, along with a series of excel files containing the quality control data collected for the Ada Tepe deposit, and detailed contour, breakline and survey points defining the surface topography over the deposit. Detailed review and validation of the supplied trench and drillhole database files was undertaken using MS Excel, Micromine and GeoAccess software products, prior to loading separate trench and drillhole databases into the Vulcan software package, wherein further visual validation of the data was carried prior to the commencement of mineral resource modelling.

14.1.2 Validation of Supplied Database

The following database validation activities were carried out in 2004 by RSG:

- Ensure compatibility of total hole depth data in the collar, survey, assay, geology, structure orientation, vein orientation class, recovery/RQD, and density drillhole database files and similar trench database files, excluding recovery/RQD and vein orientation class files, not relevant to the trenches.
- Checking of drillhole survey data for unusual or suspect downhole deviations.
- Cross-checking of laboratory assay certificates against the digital assay data for the top 100 ranked gold assays for each of the primary laboratories, OMAC, SGS Gura Rosiei, and SGS Welshpool, and 15% of all the gold assays completed by the primary laboratories.
- Replacement of less than detection limit values (such as <0.01) with a numeric value of half the detection limit.
- Checking of geology logging codes.
- Calculation of strike and dip values from alpha and beta structure orientation logging for oriented core (measured directly for trenches), and 3-D coordinates of the structures logged in both the trenches and drillholes.
- Calculation of percentage core recovery values from run length and recovered core metreage data.
- Checking for suspect mineralised drillhole intersections.
- Checking of drillhole collar and trench sampling trace locations against the surface topography.

A limited number of mineralised intercepts were identified from within the basement rocks. This mineralisation is generally characterised by a limited spatial extent and is difficult to correlate between sections, and as such, all mineralised basement intercepts were removed from the mineral resource dataset. No anomalies were identified in the manual cross-checks of the digital assay data and no material errors were identified during database validation activities.

CSA completed similar reviews during the site visit in 2012 (Section 12.6) and some aspects more recently in March 2014 and found no anomalies or material errors.

14.1.3 Vulcan Database Development

The validated drillhole and trench database files were imported into the Vulcan software package as three databases:

- Drillhole database, adapkru16.dhd.isis, containing collar, survey, assay, geology, vein class, density, and recovery/RQD tables.
- Trench database, adapkru16.trd.isis, containing collar, survey, assay, geology and density tables.
- Structural database, adadbftek.isis, containing structure type, 3-D location and structural orientation data for both drillholes and trenches.

Statistics of drilling and sampling used for the Mineral Resource Estimation are displayed in Table 11.

14.2 RSG Geology and Mineralised Domain Modelling

14.2.1 Introduction

Based on observations of the geology during the site visit and using all of the available geological and grade information, suitable lithology, oxidation and mineralised domain boundaries were interpreted and wireframe modelled to constrain the Mineral Resource Estimation for the Ada Tepe deposit. Interpretation and digitising of all constraining boundaries was undertaken on north-south orientated cross sections coinciding with the drill traverses. The resultant digitised boundaries were then used to construct wireframe surfaces or solids defining the 3-D geometry of each interpreted feature. The interpretation and wireframe models were developed by RSG Global using the Vulcan software package.

14.2.2 Lithology Boundaries

Two principal lithologies occur at Ada Tepe, predominantly unmineralised basement metamorphic rocks mostly comprised of gneiss and amphibolite, and the variably mineralised overlying sedimentary rocks of the Shavarovo Formation, comprised of irregular lenses of sedimentary breccia and sandstone. The shallow northerly dipping boundary between the basement and overlying sedimentary rocks is often represented by a thin clay zone with mylonitic textures, which can contain mineralised vein fragments.

When the clay mylonitic zone is absent, the basement - sediment contact has been interpreted and wireframe modelled at the logged position of the contact above unmineralised clay zone intersections but below some of the mineralised clay zone intersections where considered appropriate on the basis of the intersections in the adjacent drillholes.

While generally little to no surface talus or overburden occurs over most of the Ada Tepe deposit, some overburden, mostly associated with ancient surface mine workings, termed 'The Quarry', occurs along the eastern side of the Ada Tepe hill. A basal surface to this overburden was interpreted and modelled based on the drillhole and trench logging for the area.

14.2.3 Oxidation/Weathering Boundaries

Weathering and oxidation at Ada Tepe have been logged with strong, moderate, or weak oxidation/weathering or fresh rock recorded for each logging interval. Based on a detailed review of the spatial distribution of the logging and associated variations in bulk density, RSG Global interpreted and modelled wireframe boundaries subdividing the weathering profile into a combined strongly and moderately oxidised zone, weakly oxidised zone and fresh (unoxidised) rock zone.

Strong to moderate oxidation in the sedimentary breccia is typically associated with the partial weathering of variably altered clasts of basement and near complete oxidation of sulphides. This is generally associated only with minor reductions in bulk density compared to the fresh sedimentary breccia due to the substantial degree of silicification and veining throughout the deposit. The depth of strong to moderate oxidation at Ada Tepe typically ranges from 25 m to 75 m depth, with the deepest weathering present beneath the top of the Ada Tepe hill, and decreasing towards the north and down the east and west flanks of the hill. Weak oxidation and weathering typically extends an additional 10 m to 25 m below the base of strong to moderate weathering.

CSA considers the approach used by RSG to model lithological and oxidation boundaries to be valid for the current level of study.

14.2.4 Mineralised Domain Boundaries

Mineralised domain boundaries for constraining the Mineral Resource Estimate was interpreted and modelled based on the geological logging, surface mapping and interpreted geological and structural controls. The interpretation reflects the following two principal styles of mineralisation recognised at Ada Tepe:

- Wall Zone mineralisation occurring within a shallow north dipping zone of brecciated vein material and variably silicified sedimentary breccia immediately above the basement - sediment contact. Generally thicker more intensely brecciated regions of the Wall Zone, through which well-developed Upper Zone vein mineralisation passes, contain epithermal vein and hydraulic breccia infill textures and associated high gold grades not present in regions where Upper Zone vein mineralisation is absent. These thick strongly continuous regions of high grade Wall Zone mineralisation generally thin and diminish in grade away from and between regions of well-developed Upper Zone vein mineralisation.

- Upper Zone mineralisation predominantly occurring within east-west striking, steep dipping veins, transitional downwards into the Wall Zone.

14.2.4.1 Wall Zone Domain

The wireframe surface of the basement - sediment contact was used as the lower bounding surface to the Wall Zone domain. An upper bounding wireframe surface was constructed based on digitised cross sectional lines defined using the following generalised parameters:

- Individual drillhole intersections of Wall Zone mineralisation interpreted by BMM geologists on site.
- Intersections of massive silicified breccia-conglomerate and re-brecciated early phase vein mineralisation.
- Shallow dipping structural orientation data for vein mineralisation near the basement-sediment contact.
- A notional 0.2 g/t Au lower cutoff grade.
- Use of 2 m minimum downhole intersections.
- Avoiding large changes in the interpreted thickness of the Wall Zone mineralisation amongst adjacent drillholes.

The resultant wireframe surface of the interpreted upper boundary of the Wall Zone mineralisation reflects the selection of a best fit boundary, often through mineralisation that is strongly transitional between the Wall and Upper Zones. However, this surface was used as a hard boundary during mineral resource grade estimation, as the transitional nature of the mineralisation between the Wall Zone and the Upper Zone is often absent, particularly where Upper Zone mineralisation is poorly developed. In summary, the modelled surfaces representing both the lower and upper confining surfaces of the Wall Zone mineralisation were considered robust and suitable for constraining mineral resource estimation for the Wall Zone.

CSA agrees with this methodology and considers it a valid approach.

A wireframe solid model of the Wall Zone mineralisation was generated by intersecting the wireframe surfaces of the basement-sediment contact and upper confining boundary. The lateral limits of the resultant wireframe solid reflect an interpreted tapering of the Wall Zone mineralisation converging towards the basement - sediment contact. Representative plan view, cross-section and 3-D perspective views of the wireframe solid model of the Wall Zone domain are displayed in Figure 33 to Figure 35.

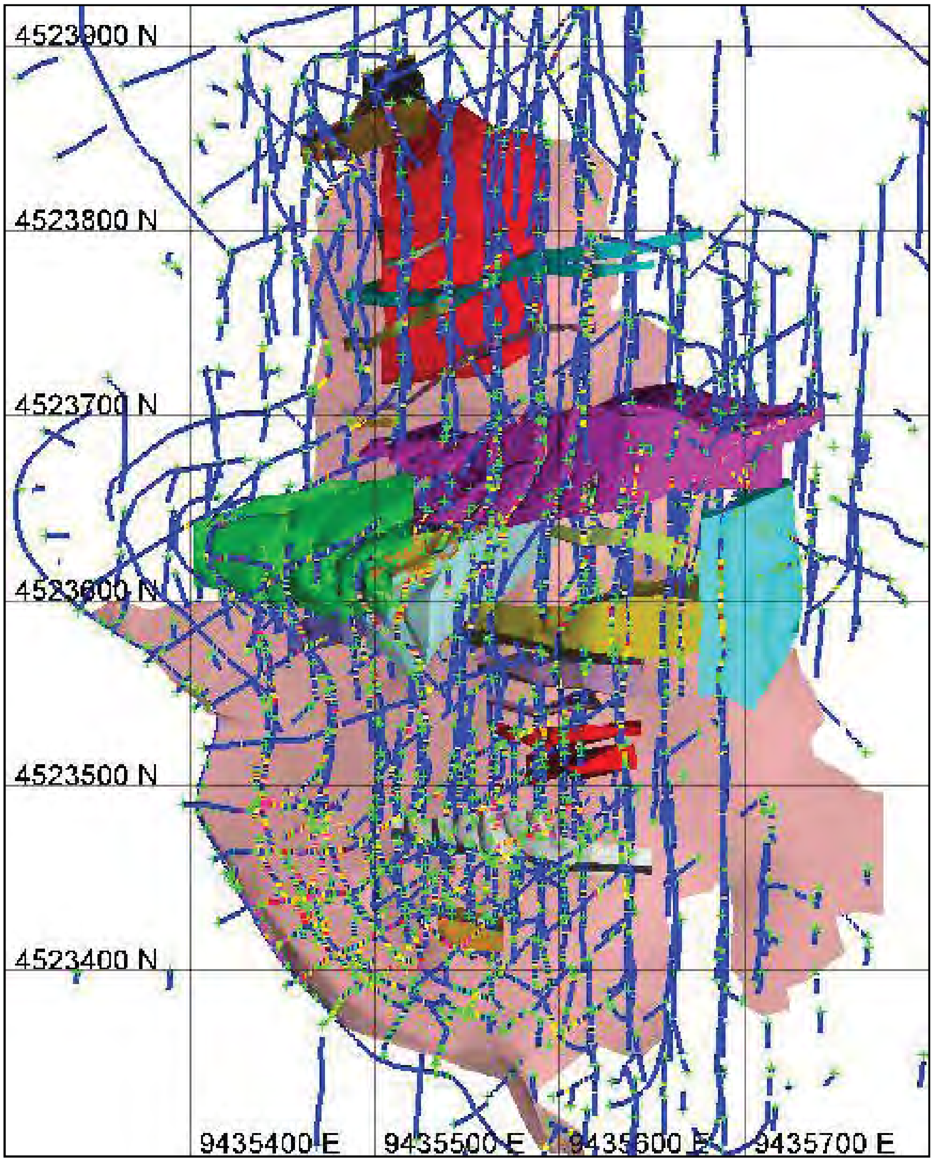

Figure 33. Plan View of Drilling, Trench Sampling and Modelled High Grade Domains

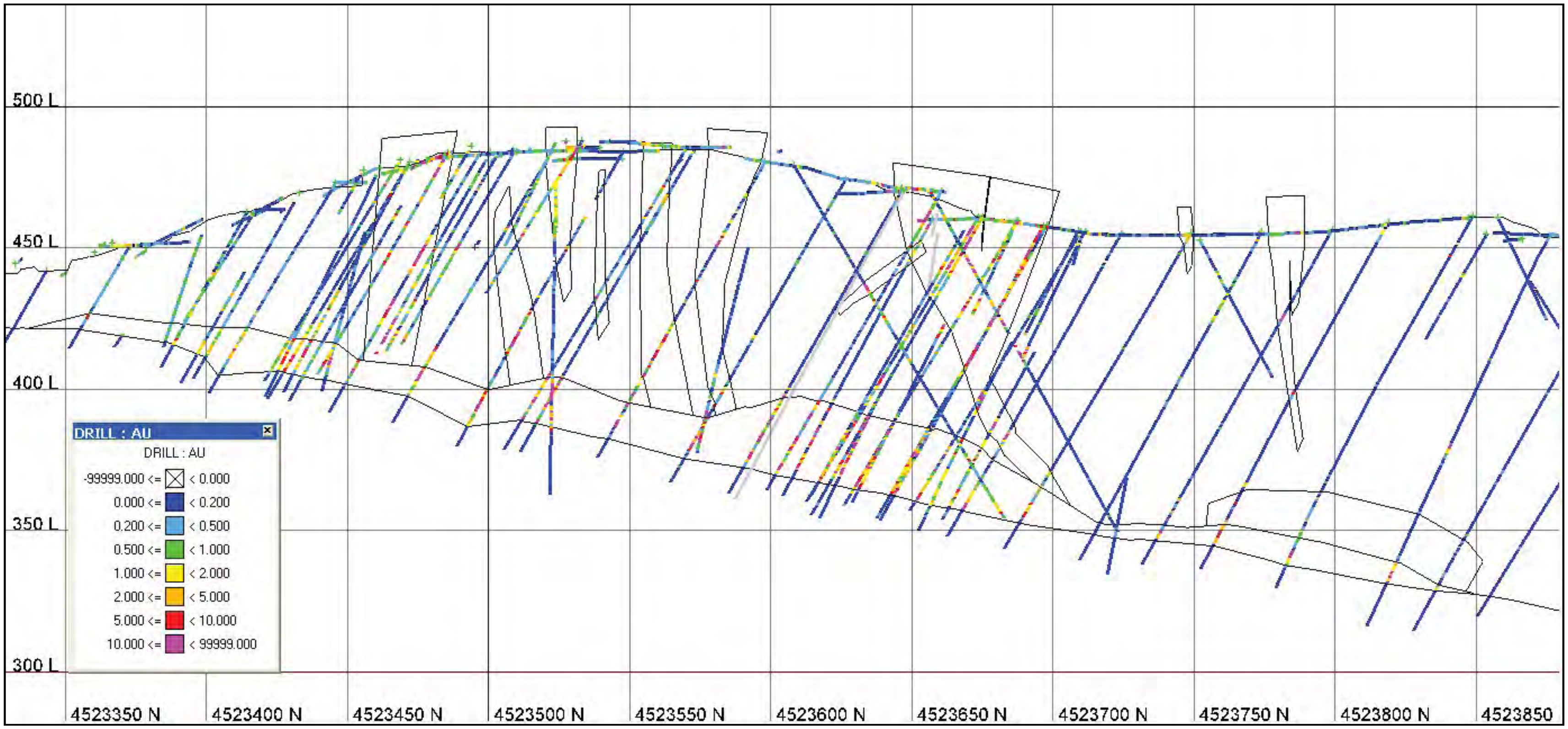

Figure 34. Cross-sectional View of Drilling, Trench Sampling and Modelled High Grade Domains at 94355590E

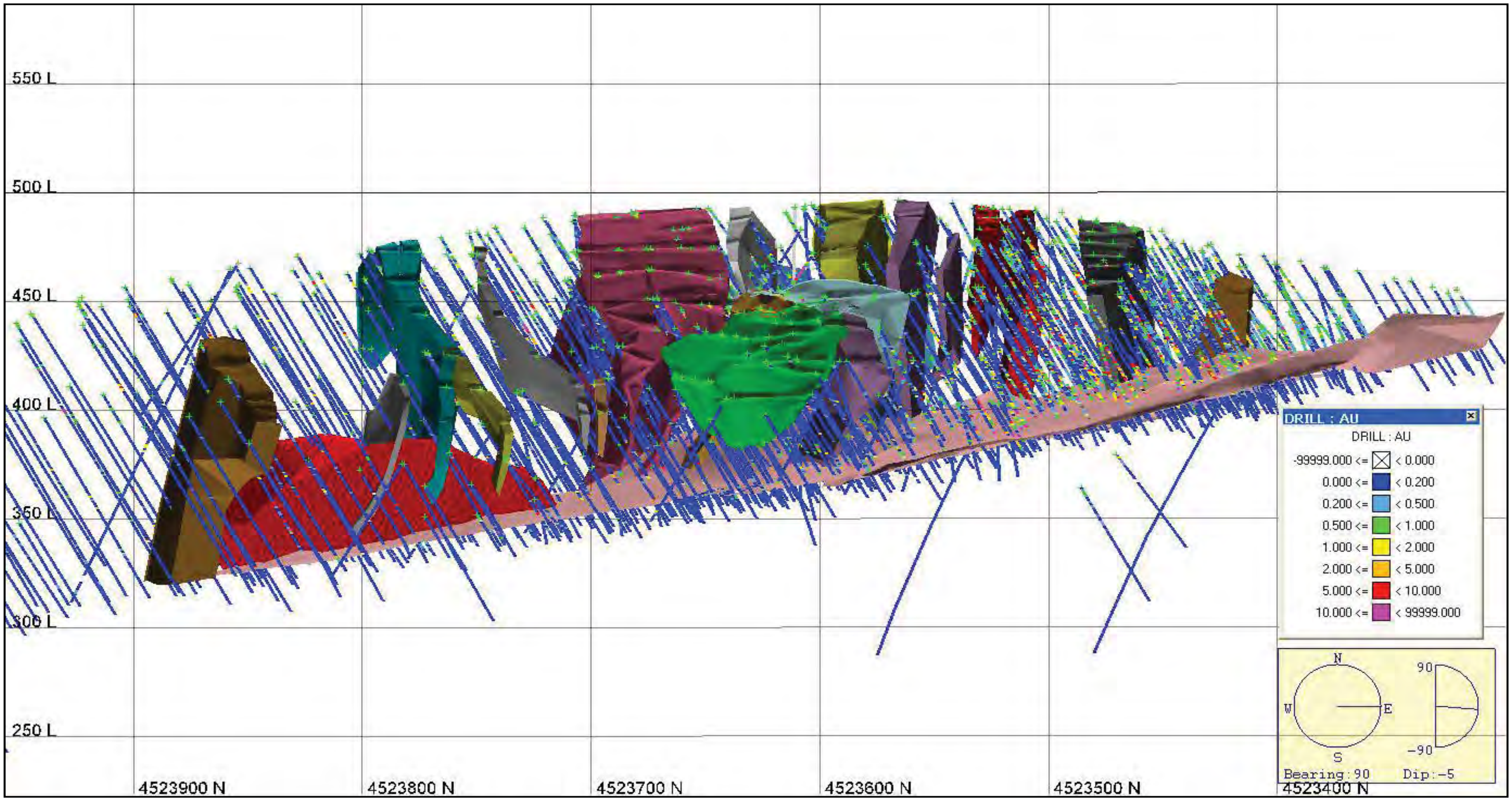

Figure 35. 3-D Perspective View (Towards the East) of Drilling and Modelled High Grade Mineralised Domains

14.2.4.2 Upper Zone Domains

Review of all the available geological and structural orientation data for the Upper Zone mineralisation resulted in the development of detailed mineralised domain constraints in regions where discrete packages of vein mineralisation could be interpreted between adjacent drillholes over multiple drill traverses, while broader boundaries were modelled capturing all of the remaining mineralisation located outside the interpreted limits of the detailed domains. The primary goal of the interpretation was to separate zones of distinctively higher grade mineralisation from the surrounding lower tenor mineralisation and barren rock.

CSA considers this to be a valid approach based on the observed mineralisation characteristics.

The following generalised parameters were used to interpret and wireframe model the spatial limits of the detailed high grade domains in the Upper Zone:

- Structural orientation data for veining and faulting based on trench mapping and oriented core.
- Surface mapping of vein and fault traces.
- Orientation class data for veining based on alpha angle measurements for un-oriented core.
- Presence of silicification and veining.
- A notional 0.2 g/t Au lower cutoff grade.
- Use of 4 m minimum downhole intersections.
- Incorporation of internal and external dilution as required to produce robust 3-D continuous boundaries.
- Termination of domains halfway to an adjacent sub-grade or barren drillhole intersection, at the top of the modelled Wall Zone (domain 1), at the basement-sediment contact where no Wall Zone mineralisation was interpreted, or at other positions based on the interpreted structure and geology.

In aggregate, 26 mineralised domains were interpreted, representing 24 Upper Zone domains and a single Wall Zone domain, plus a separate zone constraining the mineralised overburden. Separate wireframes (19 domains) were modelled for the higher grade mineralisation while a further 5 domains captured the remaining mineralisation outside the higher grade envelopes. Representative plan, cross section and 3-D perspective views of the modelled high grade domains are presented in Figure 33 to Figure 35.

While other zone interpretations are possible, it is believed that the current domains honour the orientation of the vein mineralisation in the Upper Zone, and achieves the goal of separating most of the zones of higher grade mineralisation from the surrounding lower tenor mineralisation and/or barren rock. The geological and mineralisation domains have been validated both visually and by checks of the wireframe integrity prior to database and block model coding. The wireframes are considered robust.

CSA agrees with this statement.

In addition to the domain coding, a diluted domain coding was completed for the Upper Zone domains 11, 12, 21-24, 31-35, 41 and 42. The dilution was based on the domain wireframes expanded approximately 2.5m in 3-D, and was completed to ensure sufficient dilution was included for the more tightly constrained high-grade domains only. Note this coding has been applied to the drillhole and trench data sets only and is therefore not available in the block model. The mineralisation domain interpretation was coded to the drillhole and channel database in preparation for compositing (Table 32). Data or blocks lying outside the domains were assigned a domain code of zero.

CSA considers the approach of defining a zone of dilution to be a valid one which honours the mining approaches being considered and facilitates the reliable use of MIK as a grade interpolation method.

14.3 Statistical Analysis

14.3.1 Introduction

Statistical analysis was undertaken based on 3 m composites of the gold and silver assay data, and the bulk density data for the drilling and trench sampling completed at Ada Tepe. The activities completed in this phase of the study were as follows:

- Coding of the drillhole and trench sampling databases based on the geological interpretation.
- Compositing of the drillhole and trench sampling data to 3m unit lengths.
- Compilation of descriptive statistics and histogram plots of the composite gold, silver and bulk density datasets, subdivided by the geological interpretation.
- Outlier grade analysis and determination of upper cuts for gold and silver.
- Assessment of data clustering and calculation of declustered grade statistics.
- Compilation of conditional statistics and determination of indicator thresholds for MIK of gold in the Upper Zone domain groups.

- Correlation analysis between composite gold and silver datasets subdivided by estimation domain/group.

14.3.2 Data Coding

The wireframe models of the lithology, oxidation and mineralised domains were used to assign a series of codes into the drillhole and trench sampling databases (Table 32).

Table 32. Domain Coding of Vulcan Drillhole and Trench Databases

Domain Coding of Vulcan Drillhole and Trench Databases							
Database Coding							
Domain		Wireframe		Variable		Fill	Priority
Type	Description	Name (*.00t)	Description	Name	Code	Direction	(lowest =1)
Lithology	Basement (default) Sedimentary Breccia Overburden Air	N/A base_breccia base_overburden_dec04 topo_rlimexp	N/A Base of Wall Zone Base of overburden Surface topography	Lith lith lith lith	100 200 300 1000	Z (default) Z (above) Z (above) Z (above)	1 2 3 4
Oxidation	Fresh (default) Weak oxidation Weak oxidation Strong/Mod oxidation Air	N/A bo_wox_solid_dec04 bo_wox_surf_dec04 bo_mox_surf_dec04 topo_rlimexp	N/A Weak oxidation solid Base of weak oxidation Base of moderate oxid. Surface topography	oxid oxid oxid oxid oxid	50 60 60 70 0	Z (default) Z (inside) Z (above) Z (above) Z (above)	1 2 2 3 4
	Wall Zone (domain 1) UZ group 1 domain 11 UZ group 1 domain 12	wall_0904 dom11_dec04 dom12 dec04	Wall zone solid UZ group 1 domain 1 solid UZ group 1 domain 2 solid	domain domain domain	1 11 12	Z (inside) Z (inside) Z (inside)	70 11 12
	UZ group 2 domain 21 UZ group 2 domain 22 UZ group 2 domain 23 UZ group 2 domain 24 UZ group 2 domain 25	dom21_dec04 dom22_dec04 dom23_dec04 dom24_dec04 dom25_dec04	UZ group 2 domain 1 solid UZ group 2 domain 2 solid UZ group 2 domain 3 solid UZ group 2 domain 4 solid UZ group 2 domain 5 solid	domain domain domain domain domain	21 22 23 24 25	Z (inside) Z (inside) Z (inside) Z (inside) Z (inside)	22 20 21 19 18

Domain Coding of Vulcan Drillhole and Trench Databases							
Database Coding							
Domain		Wireframe		Variable		Fill	Priority
Type	Description	Name (*.00t)	Description	Name	Code	Direction	(lowest =1)
High Grade Domains	UZ group 3 domain 31	dom31_dec04	UZ group 3 domain 1 solid	domain	31	Z (inside)	13
	UZ group 3 domain 32	dom32_dec04	UZ group 3 domain 2 solid	domain	32	Z (inside)	14
	UZ group 3 domain 33	dom33_dec04	UZ group 3 domain 3 solid	domain	33	Z (inside)	16
	UZ group 3 domain 34	dom34_dec04	UZ group 3 domain 4 solid	domain	34	Z (inside)	15
	UZ group 3 domain 35	dom35_dec04	UZ group 3 domain 5 solid	domain	35	Z (inside)	17
	UZ group 4 domain 41	dom41_dec04	UZ group 4 domain 1 solid	domain	41	Z (inside)	29
	UZ group 4 domain 42	dom42_dec04	UZ group 4 domain 2 solid	domain	42	Z (inside)	28
	UZ group 4 domain 43	dom43_dec04	UZ group 4 domain 3 solid	domain	43	Z (inside)	24
	UZ group 4 domain 44	dom44_dec04	UZ group 4 domain 4 solid	domain	44	Z (inside)	23
	UZ group 4 domain 45	dom45_dec04	UZ group 4 domain 5 solid	domain	45	Z (inside)	27
	UZ group 4 domain 46	dom46_dec04	UZ group 4 domain 6 solid	domain	46	Z (inside)	25
	UZ group 4 domain 47	dom47 dec04	UZ group 4 domain 7 solid	doman	47	Z (inside)	26
	Air	topo_rlimexp	Surface topography	domain	0	Z (above)	80
		to_wall2_dec04.00t	Top of Wall Zone	domain	0	Z (above)	60
Background (Lower Grade) Domain Groups	UZ domain domain 10	grp_dom1	UZ domain group 1 solid	domain	10	Z (inside)	3
	UZ domain domain 20	grp_dom2m	UZ domain group 2 solid	domain	20	Z (inside)	4
	UZ domain domain 30	grp_dom3m	UZ domain group 3 solid	domain	30	Z (inside)	5
	UZ domain domain 40	grp_dom4m	UZ domain group 4 solid	domain	40	Z (inside)	2
	UZ domain domain 50	grp_dom5m	UZ domain group 5 solid	domain	50	Z (inside)	1

The lithology, oxidation, mineralised domain and mineralised region coding assigned to the drillhole and trench samples was visually compared with the corresponding wireframe boundaries in cross section and plan views to ensure all coding was robust. No errors were detected.

CSA replicated this step as a validation check and found no errors.

14.3.3 Compositing

The lengths of the samples in the validated Vulcan drillhole and trench databases were statistically assessed in advance of compositing. Approximately 90% of the drillhole samples and 65% of the trench samples have been collected over 1 m intervals. Most of the remaining samples have been collected over various intervals ranging 1 m to 3 m. A 3 m unit length was used for data compositing to approximate the envisaged mining bench height of 2.5 m. The geological interpretation coded to the database was assigned to the composites. Any composites less than 1 m long were excluded for the remainder of the study.

CSA recreated the composite dataset used in the Mineral Resource Estimate. In addition a review was undertaken to investigate the likely impact on the grade estimate when <1 m composites are excluded from the dataset. Excluded composites represent 4% of the total dataset and 2% and 1% of the Upper Domain and Wall Domain composite dataset respectively. Given the small proportion this represents and given the observed grade variability, being particularly low in the Wall Domain the exclusion of <1m composites is not considered to have a material effect on the subsequent grade estimation, and any effect is likely to occur at the margins of domains (edge effect).

14.3.4 Statistical Analysis

14.3.4.1 Descriptive Statistics

Descriptive statistics of the composite gold and silver data subdivided by mineralisation domain is presented in Table 33. The highest mean grade is reported for the Upper Zone domain 42, albeit that few data exist for this domain. The coefficient of variation values (CV; calculated by dividing the standard deviation by the arithmetic mean) is elevated for the majority of domains indicating possible significant distortion of the mean grade by outlier composites. Box plots (Figure 36) of these datasets indicate they all form positively skewed distributions typical of gold deposits.

Descriptive statistics of the composite gold data for the Wall Zone and the combined Upper Zone, subdivided by the modelled oxidation profile are displayed in Table 34. Comparative log probability plots grouped by oxidation are provided for the Wall Zone and the Upper Zone as Figure 37 and Figure 38 respectively. The Upper Zone datasets show a consistent decrease in grade downwards through the oxidation profile, although little evidence exists for lateral dispersion due to oxidation. It is interpreted that the increased grade with increased oxidation correlation to be a function of the veining and fracturing (higher grade mineralisation) providing pathways for water migration and therefore an increase oxidation.

The implications of sub-dividing the modelled Upper Zone domain groups with hard weathering zone boundaries during mineral resource estimation was considered, however, as described above, the oxidation appears not to control the distribution or location of mineralisation but is a function of the increased level of fracturing associated with the mineralised zones. As such, the

Mineral Resource Estimation was undertaken subdivided by the "inside" versus "outside" regions of high grade domains within each of the Upper Zone domain groups, without weathering zone sub-division.

Table 33. Summary Statistics of 3 m Gold and Silver (g/t) Composites Sub-divided by Modelled Mineralised Domain

Region	Domain	Variable	Number	Minimum	Maximum	Mean	Median	Std Dev	Coeff Var
Wall Zone	1	Au Ag	1,686 1,618	0.005 0.300	129.62 66.00	6.62 4.23	2.16 2.17	12.44 5.78	1.88 1.37
	10	Au	3,474	0.007	305.90	1.69	0.29	9.22	5.46
	(BG)	Ag	2,926	0.200	121.67	1.26	0.67	4.08	3.23
	11	Au	392	0.020	286.36	8.06	0.90	27.38	3.40
	(HG)	Ag	347	0.267	352.67	5.03	1 .00	22.60	4.49
	12	Au	115	0.237	133.54	7.54	1 .86	16.32	2.16
	(HG)	Ag	109	0.353	60.37	4.24	1 .83	7.78	1.84
	20	Au	3,381	0.005	33.45	0.26	0.08	1.18	4.60
	(HG)	Ag	3,193	0.100	34.30	0.71	0.50	0.81	1.14
	21	Au	1,043	0.020	363.72	5.44	0.79	18.97	3.49
	(HG)	Ag	915	0.200	129.33	2.90	1 .00	7.61	2.62
	22	Au	161	0.040	35.93	3.55	1 .33	5.90	1.66
	(HG)	Ag	145	0.500	18.33	1.84	1 .00	2.56	1.40
	23	Au	113	0.040	151.28	6.18	0.97	20.27	3.28
	(HG)	Ag	104	0.500	54.83	3.01	0.83	7.20	2.39
	24	Au	69	0.050	65.75	3.80	0.83	9.06	2.38
	(HG)	Ag	49	0.500	8.17	1.62	1.25	1.64	1.01
	25	Au	528	0.008	112.07	1.66	0.16	7.79	4.68
	(HG)	Ag	474	0.500	41.00	1.28	0.67	2.82	2.20
	30	Au	2,067	0.005	250.70	0.42	0.14	5.76	13.82
	(HG)	Ag	1,876	0.100	81.67	0.71	0.50	2.03	2.86
	31	Au	136	0.103	46.44	3.27	1.23	6.96	2.13
	(HG)	Ag	123	0.500	29.33	3.02	1 .85	3.79	1.26
	32	Au	143	0.067	76.41	2.17	0.58	6.95	3.20
Upper Zone	(HG)	Ag	133	0.500	52.67	2.74	1.17	6.29	2.30
	33	Au	249	0.037	60.80	2.15	0.62	5.37	2.49
	(HG)	Ag	214	0.280	112.00	3.10	1 .00	10.72	3.45
	34	Au	58	0.132	27.99	2.90	1.22	4.29	1.48
	(HG)	Ag	58	0.500	14.47	1.93	1 .00	2.21	1.15
	35	Au	48	0.127	136.97	11.25	1 .33	24.20	2.15
	(HG)	Ag	33	0.450	58.67	6.64	0.85	12.84	1.93

Region	Domain	Variable	Number	Minimum	Maximum	Mean	Median	Std Dev	Coeff Var
	40	Au	4,400	0.005	53.56	0.20	0.06	1.29	6.34
	(HG)	Ag	4,124	0.500	56.33	0.75	0.50	1.22	1.62
	41	Au	127	0.007	109.93	6.20	0.81	14.91	2.40
	(HG)	Ag	116	0.500	37.33	3.24	1 .33	5.13	1.58
	42	Au	18	0.105	187.27	18.32	2.22	43.96	2.40
	(HG)	Ag	13	0.500	43.17	7.61	2.08	11.90	1.56
	43	Au	33	0.030	10.60	2.39	1 .46	2.48	1.04
	(HG)	Ag	27	0.500	3.33	1.58	1 .08	1.00	0.64
	44	Au	14	0.023	10.08	2.63	0.74	3.48	1.32
	(HG)	Ag	12	0.500	4.67	1.82	1 .00	1.47	0.81
	45	Au	18	0.030	56.59	9.95	2.92	15.29	1.54
	(HG)	Ag	17	0.500	23.50	5.14	3.08	5.93	1.15
	46	Au	91	0.013	30.38	2.61	0.28	5.16	1.98
	(HG)	Ag	79	0.500	10.83	1.96	1 .33	1.88	0.96
	47	Au	293	0.013	20.61	0.74	0.21	1.95	2.65
	(HG)	Ag	293	0.500	12.67	1.36	0.83	1.42	1.05
	50	Au	765	0.005	17.35	0.26	0.15	0.97	3.69
	(HG)	Ag	636	0.500	6.00	0.66	0.50	0.47	0.71
Over-burden	100	Au Ag	149 144	0.030 0.500	25.50 9.00	1.34 1.03	0.65 0.50	2.86 1.05	2.13 1.03

Table 34. Summary Statistics of Composite Gold Data for Wall Zone and Upper Zone Domain High Grade Domains Sub-divided by Oxidation

Summary Statistics of Composite Gold Data for Wall Zone and Upper Zone Domain High Grade Domains Sub-divided by Oxidation						
	Wall Zone			Upper Zone High Grade Domains		
	Oxide	Trans	Fresh	Oxide	Trans	Fresh
Number	630	654	402	12,529	1 ,939	3,268
Minimum	0.017	0.005	0.06	0.005	0.005	0.005
Maximum	114.0	91.4	129.6	363.72	76.41	83.41
Mean	7.75	4.10	8.95	1.81	0.64	0.25
Std Dev	12.35	9.72	15.48	10.24	3.38	1.89
Coeff Var	0.02	0.02	0.04	5.67	5.28	7.58

Note: Excludes overburden composites

Figure 36. Box Plots of Composite Gold Data for Primary Mineralised Regions:-Wall Zone (red), Upper Zone (blue) and Overburden (green)

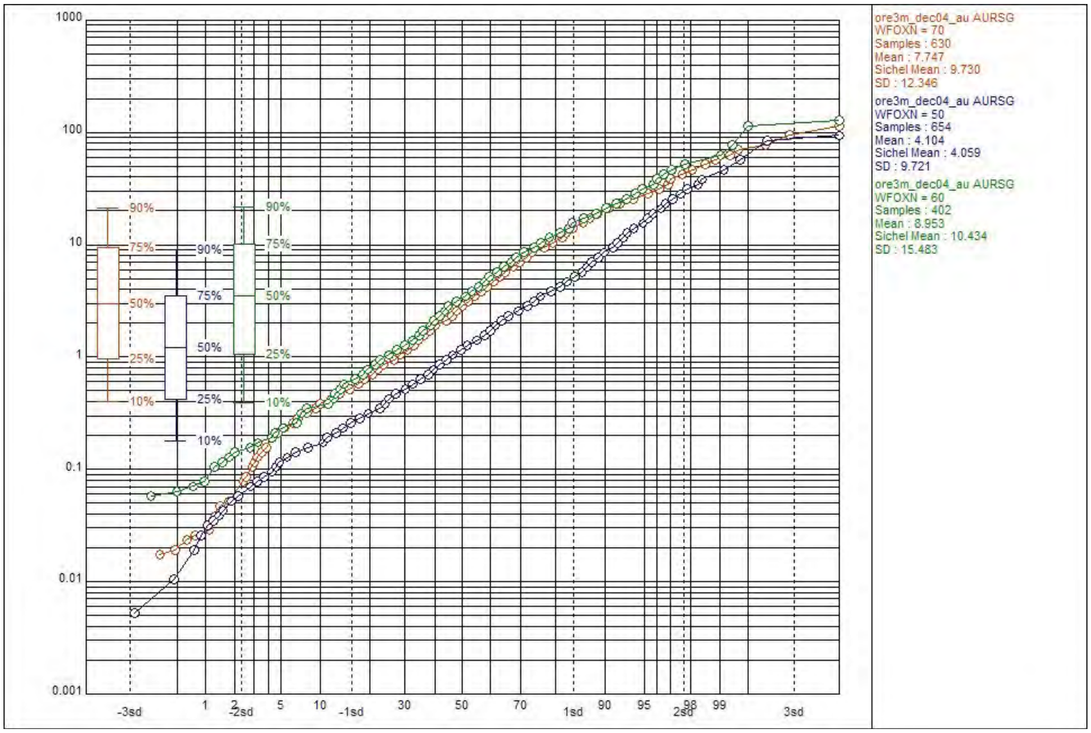

Figure 37. Log Probability Plots of Composite Gold Data for Wall Zone Sub-divided by Oxidation Profile

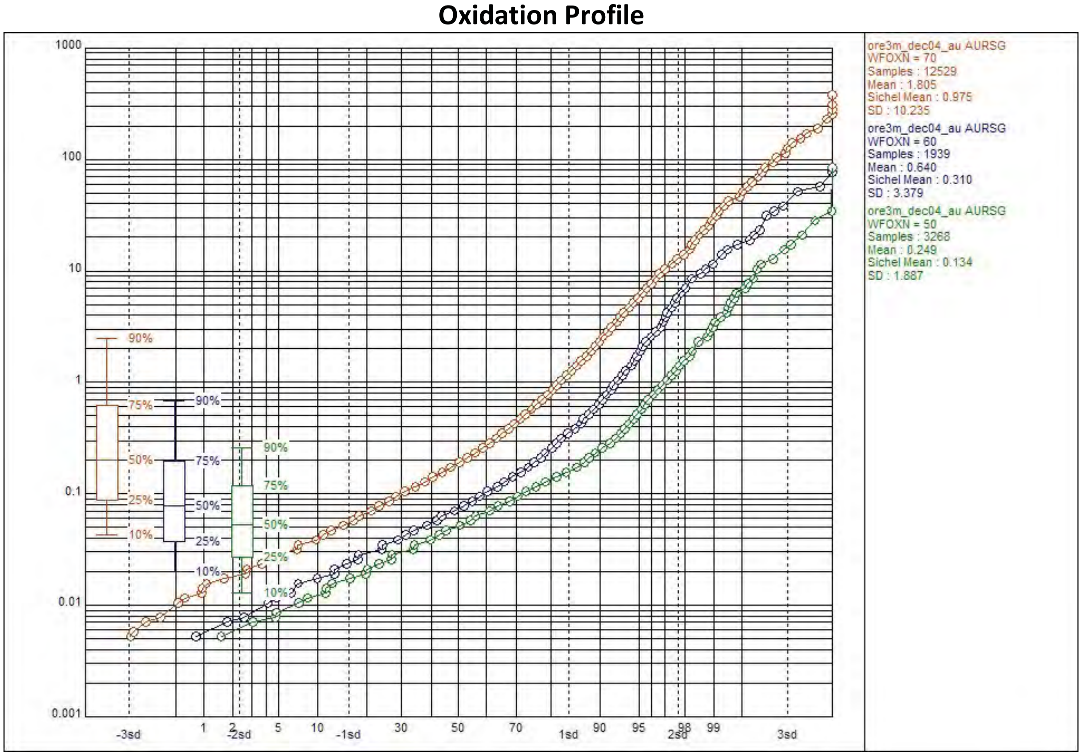

Figure 38. Log Probability Plots of Composite Gold Data for Upper Zone Domains Sub-divided by Oxidation Profile

14.3.4.2 Outlier Analysis and Determination of Upper Cuts

Assessment of the high grade gold composites was completed to determine the requirement for high grade cutting for each of the input datasets to be used for mineral resource estimation. The approach taken to the assessment of the high grade composites and outliers is summarised as follows:

- Detailed review of histograms and probability plots, with significant breaks in populations used to interpret possible outliers.
- Investigation of clustering of the higher-grade data. High grade data that are clustered were considered to be real while high grade composites not clustered with other high grade data were considered to be possible outliers, requiring further consideration via cutting.
- The ranking of the composite data and the investigation of the influence of individual composites on the mean and standard deviation (mean versus std-dev plots).

Based on this assessment, a series of high-grade cuts or caps were determined and applied to mineral resource estimation as presented in Table 35 for gold. The silver data was not cut as grade was regressed from gold (Section 14.6.4). It should be noted that no upper cut was considered necessary for the Wall Zone, due to the absence of outliers. The upper cuts applied to the Upper Zone domain group datasets result in cut mean grades ranging from 4% to 33% less than the uncut mean grades.

CSA agrees with the approach adopted by RSG for the consideration of grade capping.

Table 35. Gold (g/t) Outlier Analysis – Sub-divided by Estimation Domain Groups

Gold (g/t) Outlier Analysis – Sub-divided by Estimation Domain Groups									
Domain(s)	**Cut/Uncut**	**Count**	**Min**	**Max**	**Mean**	**Median**	**Std. Dev**	**CV**	**Mean Diff (%)**
Wall	Uncut	1,686	0.01	129.62	6.62	2.16	12.44	1 .88	
	Cut	1,686	0.01	129.62	6.62	2.16	12.44	1 .88	0%
10	Uncut	3,474	0.01	305.9	1.69	0.29	9.22	5.45	
	Cut	3,474	0.01	93	1.53	0.29	5.94	3.88	9%
11 to 12	Uncut	507	0.02	286.36	7.94	1 .08	25.26	3.18	
	Cut	507	0.02	109	6.86	1 .08	16.55	2.41	14%
20	Uncut	3,381	0.01	33.45	0.26	0.08	1.18	4.6	
	Cut	3,381	0.01	15	0.25	0.08	0.92	3.76	4%
21 to 25	Uncut	1 ,91 4	0.01	363.72	4.22	0.64	15.67	3.71	
	Cut	1 , 91 4	0.01	119	4	0.64	12.45	3.11	5%
30	Uncut	2,067	0.01	250.7	0.42	0.14	5.76	13.81	
	Cut	2,067	0.01	19	0.28	0.14	0.95	3.39	33%
31 to 35	Uncut	634	0.04	136.97	3.15	0.78	9.11	2.89	
	Cut	634	0.04	51	2.95	0.78	7.12	2.41	6%
40	Uncut	4,400	0.01	53.56	0.2	0.06	1.29	6.34	
	Cut	4,400	0.01	16	0.19	0.06	0.8	4.27	5%
41 to 47	Uncut	594	0.01	187.27	3.14	0.37	11.36	3.62	
	Cut	594	0.01	55	2.79	0.37	7.57	2.71	11%
50	Uncut	765	0.01	17.35	0.26	0.15	0.97	3.68	
	Cut	765	0.01	3	0.21	0.15	0.3	1.4	19%

14.3.4.3 Declustered Grade Statistics

Significant clustering of the exploration sample data is evident at Ada Tepe resulting from the two regions of close spaced drilling completed to investigate the short range variability of the gold mineralisation, the drilling of 'geological' holes investigating the geometry of zones of high grade gold mineralisation and regions of intersecting trench sampling. Cell declustering has been undertaken to assess the effects of the data clustering on the global mean grade of each of the defined estimation domains / groups. Table 36 presents a comparison of the naïve and declustered mean grades and CVs for each of the estimation domains/groups based on the cut composite gold datasets. It is evident that declustering results in significantly reduced mean grades, in particular for the Wall Zone and most of the Upper Zone high grade domain groups. The declustering also results in marginally increased CV values compared to the naive datasets.

CSA agrees with comments made by RSG and agrees that cell declustering is a valid and meaningful approach to arriving at reliable and unbiased mean grades for each domain.

14.3.4.4 Conditional Statistics

Conditional statistics were generated for Upper Zone domain groups, as MIK was considered an appropriate estimation method for the Upper Zone gold mineralisation. The conditional statistics were used to determine the appropriate indicator thresholds and intra-class mean grades to allow post processing. Initial conditional statistics were compiled based on the 10th, 20th, 30th, 40th, 50th, 60th, 70th, 75th, 80th, 85th, 90th, 92.5th, 95th, 97th and 99th percentile of the composite data >=0.1 g/t Au for each of the Upper Zone domain group regions. The 0.1 g/t Au lower threshold was used to focus the statistics on the data representing mineralised material.

Table 36. Comparison of Naïve and Declustered Cut 3 m Composite Gold (g/t) Datasets

Domain(s)		Count	Min	Max	Mean	Median	Std. Dev	CV
Wall	Naive	1,686	0.01	129.62	6.62	2.16	12.44	1 .88
	Declustered	1,686	0.01	129.62	4.97	2.16	10.99	2.21
10 to 12	Naive	3,981	0.01	109	2.21	0.34	8.3	3.75
	Declustered	3,981	0.01	1 09	1.78	0.34	6.94	3.91
20 to 25	Naive	5,295	0.01	119	1.6	0.13	7.74	4.83
	Declustered	5,295	0.01	119	1.45	0.13	6.95	4.78
30 to 35	Naive	2,701	0.01	51	0.91	0.18	3.72	4.11
	Declustered	2,701	0.01	51	0.94	0.18	3.83	4.08
40 to 47	Naive	716	0.01	55	2.35	0.29	6.96	2.96
	Declustered	716	0.01	55	2.57	0.29	7.49	2.92
50	Naive	765	0.01	3	0.21	0.15	0.3	1.4
	Declustered	765	0.01	3	0.23	0.15	0.34	1.49

14.3.4.5 Correlation Analysis of Composite Gold and Silver Data

For this study, the correlation was not updated and gold-silver regression formulae developed for the previous study (Gossage, B. L. and Ridley, J., November 2004) was adopted. Gold and silver showed good correlation when elevated gold or silver grades existed, albeit correlation is poor at lower grades, with silver significant in terms of value only at elevated grades. It was therefore considered acceptable to determine silver grades based on the gold silver correlation.

The scatter plots for were reviewed and all significant outlier pairs were excluded prior to the determination of the linear regression equations to be used to estimate mineral resource silver grades. A table of the linear regression equations used in the estimate are presented in Table 51.

14.3.4.6 Bulk Density Data

Detailed statistical analysis of the bulk density data for the Ada Tepe deposit has been undertaken to determine appropriate density assignments for mineral resource grade tonnage reporting. Initial investigations undertaken subdivided by the geological logging concluded that the modelled geological constraints allow appropriate grouping of the bulk density data.

Table 37 displays summary statistics of the bulk density grouped by the modelled primary lithological boundaries and sub-divisions of the weathering profile. The statistics for these data groupings can be summarised as follows:

- Very few data are available to determined robust mean density values for oxidised basement rocks or overburden. This is not considered a material issue for the basement rocks, as no significant gold mineralisation (identified to date) occurs in the basement rocks at Ada Tepe. It is also anticipated that little to no basement material will be mined due to its position below the gold-silver mineralisation at Ada Tepe.

- The histogram plots indicate the bulk density data for each of the data groupings (with sufficient data) are normally distributed with most of the bulk density values being within 5% to 10% of the mean density reported for each data group.

- There is little to no difference between the mean and median bulk density values reported for each data group (with sufficient data).

- It is considered that the mean bulk density value reported for the overburden material (based on 7 data) to be too high for unconsolidated breccia conglomerate material. A 2.0 t/m3 bulk density value was selected as being more appropriate, and was applied for reporting overburden mineral resources.

Table 37. Summary Statistics of Bulk Density Data Grouped By Major Rock Types

Summary Statistics of Bulk Density Data Grouped By Major Rock Types									
	Breccia Conglomerate			Basement			Overburden		
	Slight to Moderate Oxidation	Weak Oxidation	Fresh	Slight to Moderate Oxidation	Weak Oxidation	Fresh	Slight to Moderate Oxidation	Weak Oxidation	Fresh
Number	2,164	747	1,627	7	10	675	7	0	0
Minimum	1.52	1.91	2.07	1.99	2.17	1.99	2.14	-	-
Maximum	2.93	2.95	3.11	2.60	2.72	2.97	2.52	-	-
Mean	2.26	2.48	2.53	2.29	2.51	2.61	2.32	-	-
Median	2.26	2.50	2.54	2.19	2.52	2.62	2.31	-	-
Std Dev	0.17	0.13	0.10	0.21	0.18	0.13	0.14	-	-
Coeff Var	0.08	0.05	0.04	0.09	0.07	0.05	0.06	-	-

Notes: 1) Mean density values highlighted in yellow used for mineral resource reporting.
2) A density of 2.00t/m3 was used to report the mineral resource in overburden.

The mean density assignments have been applied to the mineral resource block model for mineral resource grade tonnage reporting.

CSA has reviewed density data and considered the application of assigned densities to material types to be valid.

14.4 Variography

14.4.1 Introduction

No update to the variography has been completed for this study and the variography from the November 2004 mineral resource estimate (Gossage, B. L. and Ridley, J., November 2004) was adopted. Preliminary investigation indicated little change in the experimental semi- variography, and summaries of the November 2004 variography are provided below. The term "variogram" is used here as a generic word to designate the function characterising the variability of variables versus the distance between two samples.

14.4.2 Upper Zones - Domain Group 10, 20, 30, 40 and 50

Grade and indicator variography was generated for domains 10 to 12, 20 to 25, 30 to 35, 40 to 47, and 50. The domain groupings captured geological and statistically similar mineralisation. A number of indicator variograms (6 or 7 thresholds) were generated and modelled, representing the lower, middle, and upper parts of the grade distribution. Separate grade variograms have been generated and modelled for the grouped diluted high grade and lower grade data sets. A combination of 2 and 3 spherical structures have been fitted to the grade and indicator variography.

The indicator and grade variograms are presented in the following tables:

- Domains 10 to 12 Table 38
- Domains 20 to 25 Table 39
- Domains 30 to 35 Table 40
- Domains 40 to 47 Table 41
- Domain 50 Table 42

The domains 10 through 12 variography have been modelled with the major axis orientated towards 280°, a shallow westerly plunge and a vertical dip. This orientation is consistent with the interpreted geology and mineralisation controls. The grade variography is characterised by a high nugget effect (the nugget variance divided by the total sill variance) and a significant amount of short-scale variability. The overall ranges fitted to the variography are relatively short (maximum range of 60 m). The relative nugget of the indicator variography ranges from a minimum of 30% to a maximum of 60%. Similar to the grade variography, the overall ranges fitted to the indicator variograms are relatively short (50 m) with short scale major axis ranges of 12 m corresponding to the spacing of the grade control drilling.

Variogram models, similar to those modelled for domains 10 to 12 have been fitted to domains 20 to 25, 30 to 35, and 40 to 47, with high nugget effects and significant short scale variability for both the grade and indicator variography. The domains 20 to 25 variography has been modelled with the major axis orientated towards 070° and a moderate (75°) northerly dip. The grade variography has been modelled with a 50% and 54% relative nugget for the grouped high grade domains and the lower grade domains respectively. While extended overall ranges of approximately 100 m have been fitted, the short scale variability dominates the variography.

Table 38. Indicator Variogram Model Parameter Upper Zone Domains 10 to 12

Indicator Variogram Model Parameters - Upper Zone Domains 10 to 12																	
Variogram Type or Threshold	**Domain**	**Co**	**Rotation**			**C1**	**Range 1 (m)**			**C2**	**Range 2 (m)**			**C3**	**Range 3 (m)**		
			X	**Y**	**Z**		**X**	**Y**	**Z**		**X**	**Y**	**Z**		**X**	**Y**	**Z**
Au	Diluted Domains 11 & 12	0.45	100	20	90	0.2	14	12	6	0.13	14	26	12	0.22	60	26	12
	Domain 10	0.45	100	20	90	0.37	14	8	11	0.18	37	24	31	-	-	-	-
Indicator Number	Indicator Threshold	Indicator Variograms															
2	0.19	0.3	1 00	20	-90	0.3	12	9	8	0.4	50	32	40				
3	0.33	0.3	1 00	20	-90	0.3	12	9	8	0.4	50	32	38				
5	0.64	0.35	1 00	20	-90	0.3	12	9	6	0.35	50	32	32				
7	1.4	0.4	1 00	20	-90	0.27	12	10	6	0.33	50	32	31				
9	2.96	0.44	1 00	20	-90	0.29	12	14	6	0.26	50	32	28				
11	6.72	0.5	1 00	20	-90	0.29	12	16	6	0.21	50	32	28				
13	18.56	0.6	100	20	-90	0.29	12	12	3.5	0.11	50	25	14				

Table 39. Indicator Variogram Model Parameters Upper Zone Domains 20 to 25

Indicator Variogram Model Parameters - Upper Zone Domains 20 to 25																	
Variogram Type or Threshold	Domain	Co	Rotation			C1	Range 1 (m)			C2	Range 2 (m)			C3	Range 3 (m)		
			X	Y	Z		X	Y	Z		X	Y	Z		X	Y	Z
Au	Diluted Domains 21 to 25	0.5	70	0	75	0.28	11	8	6	0.12	25	25	18	0.1	100	50	40
	Domain 20	0.54	70	0	75	0.25	18	8	11	0.15	100	37	40	0.06	200	100	100
Indicator Number	Indicator Threshold	Indicator Variograms															
2	0.16	0.25	70	0	75	0.35	12	12	7	0.4	1 60	85	60				
4	0.40	0.27	70	0	75	0.38	12	12	7.5	0.35	1 25	85	38				
6	0.98	0.35	70	0	75	0.35	12	12	8	0.3	70	65	23				
8	1.86	0.43	70	0	75	0.32	12	12	8	0.25	70	60	20				
10	5.75	0.48	70	0	75	0.33	12	12	5.5	0.19	70	60	13				
12	11.65	0.57	70	0	75	0.29	12	12	6.5	0.14	50	57	11				
13	21.32	0.57	70	0	75	0.29	12	12	7	0.14	30	57	7				

The variography generated and modelled for domains 30 to 35, and 40 to 47 was orientated approximately east-west with a steep dip. Relative nugget effects fitted to the domain 30 grade variography is 41% for the high-grade domains (31 to 35) and 49% for the lower grade domain. No strong anisotropy is noted with similar overall ranges noted for all three axis although the major and semi-major axis are significantly large for the short range structure, which has been fitted at approximately the drillhole section spacing of 25 m. The overall range fitted for the high-grade domains is 57 m, 50 m and 38 m respectively for the major, semi-major and minor axis respectively. Similar variography has been generated and fitted for domain 40.

Domain 50 captures more erratic mineralisation which is difficult to interpret and is therefore considered to be of lower confidence. A single indicator threshold has been investigated with variography. Both the indicator and grade variography was generated and modelled with an easterly dip and steep dip, consistent with previous discussed upper mineralisation domains. A relative nugget of 43% was modelled for the grade variogram, with overall ranges of 41 m, 40 m and 10 m fitted for the major, semi-major and minor axis, respectively.

The Upper Zone variography is considered consistent with the interpreted controls on mineralisation with a high degree of short scale variability between intercepts from adjacent drillholes and section lines. The implication of the moderate to high nugget mineralisation is an expected moderate to high level of smoothing of kriged grade. In addition, direct or linear estimation of small blocks suitable for mine planning is not appropriate. It is considered that a non-linear approach is appropriate for grade estimation and, as such, selected MIK.

In addition to the estimation implications discusses above, the variography indicates that a relative close spacing of grade control drilling will be required. It is considered that angled RC drilling is the most suitable drilling method for grade control.

14.4.3 Wall Zone and Overburden

The Wall Zone is considered to be well defined and is likely to be mined in total, thus a selective mining estimate has not been produced and only grade variography was generated and modelled. The wall zone variography has been modelled towards 310° with a shallow westerly dip and a shallow north-westerly plunge. A moderate 40° nugget effect was modelled with overall ranges of 45 m, 58 m and 9 m fitted for the major, semi-major and minor axis respectively (see Table 43). A short range structure with ranges of 20 m, 16 m and 3 m fitted for the major, semi-major and minor axis respectively has also been modelled. An assumed model has been fitted for the overburden to enable estimation.

Table 40. Indicator Variogram Model Parameters - Upper Zone Domains 30, 31, 32, 33, 34 & 35

Indicator Variogram Model Parameters - Upper Zone Domains 30, 31, 32, 33, 34 & 35													
Variogram Type or Threshold	**Domain**	**Co**	**Rotation**			**C1**	**Range 1 (m)**			**C2**	**Range 2 (m)**		
			X	**Y**	**Z**		**X**	**Y**	**Z**		**X**	**Y**	**Z**
Au	Diluted Domains 31, 32, 33, 34 & 35	0.41	85	0	-90	0.27	20	25	8	0.32	57	50	38
	Domain 30	0.49	85	0	-90	0.35	26	27	7	0.16	95	75	50
Indicator Number	Indicator Threshold	Indicator Variograms											
2	0.14	0.3	85	0	-90	0.33	12	22	12	0.37	90	70	65
4	0.33	0.25	85	0	-90	0.35	15	32	6.5	0.4	65	62	37
6	0.63	0.3	85	0	-90	0.35	16	22	6.5	0.35	56	58	35
8	1.14	0.35	85	0	-90	0.38	18	12	7	0.27	50	53	27
10	2.85	0.45	85	0	-90	0.3	17	12	7	0.25	42	45	22
12	5.75	0.55	85	0	-90	0.3	12	12	6	0.15	40	40	14
13	9.30	0.6	85	0	-90	0.25	12	12	6	0.15	35	35	11

Table 41. Gold Indicator Variogram Model Parameters - Upper Zone Domains 40, 41, 42, 43, 44, 45, 46 & 47

Gold Indicator Variogram Model Parameters - Upper Zone Domains 40, 41, 42, 43, 44, 45, 46 & 47													
Variogram Type or Threshold	Domain	Co	Rotation			C1	Range 1 (m)			C2	Range 2 (m)		
			X	Y	Z		X	Y	Z		X	Y	Z
Au	Diluted Domains 41, 42, 43, 44, 45, 46 & 47	0.46	74	0	80	0.36	20	10	3	0.18	42	25	8
	Domain 40	0.63	74	0	80	0.29	15	10	3	0.08	30	25	8
Indicator Number	Indicator Threshold	Indicator Variograms											
2	0.15	0.33	74	0	80	0.29	12	12	8	0.38	75	53	53
4	0.30	0.4	74	0	80	0.29	12	12	8	0.31	55	50	40
6	0.57	0.48	74	0	80	0.25	12	12	8	0.27	50	50	26
8	1.25	0.55	74	0	80	0.2	12	12	4	0.25	45	50	19
10	3.00	0.6	74	0	80	0.15	12	12	4	0.25	40	40	14
12	10.43	0.65	74	0	80	0.1	12	12	4.5	0.25	40	40	4.5

Table 42. Gold Indicator Variogram Model Parameters - Upper Zone Domain Group 50

Gold Indicator Variogram Model Parameters - Upper Zone Domain Group 50													
	Threshold	Co	Rotation			C1	Range 1 (m)			C2	Range 2 (m)		
			X	Y	Z		X	Y	Z		X	Y	Z
Grade Indicator	N/A 0.52	0.43 0.45	80 80	0 0	-90 -90	0.26 0.17	41 12	12 12	10 3	0.31 0.38	41 55	40 45	10 8

Table 43. Variogram Model Performance – Gold Wall Zone

Variogram Model Performance – Gold Wall Zone																
Domain	Co	Rotation			C1	Range 1 (m)			C2	Range 2 (m)			C3	Range 3 (m)		
		X	Y	Z		X	Y	Z		X	Y	Z		X	Y	Z
Wall	0.4	310	-5	-10	0.25	20	16	3	0.35	45	58	9	-	-	-	-
Overburden	0.4	85	0	-90	0.35	15	15	15	0.25	50	50	50	-	-	-	-

Note: Variography for overburden material is assumed.

CSA has reviewed the variogram parameters as documented and believe they honour the underlying directions of grade continuity, and the geological model. CSA considers the assumptions made with respect to the application of variogram parameters to domains with insufficient data, to be valid. CSA has used some variogram models as inputs in to the check estimate of a selection of Upper Zone domains. The results of this work (14.7.3.2) provide some additional support to the validity of the variogram models.

14.5 Block Model Development

14.5.1 Introduction

A block model has been produced, using the Vulcan mining software package. The block model contains sufficient variables to record the results of OK and MIK grade estimates and other required parameters.

14.5.2 Block Construction Parameters

Table 44 summarises the extents of the Ada Tepe block model. The mineral resource block model was developed using block dimensions of 12.5 m Easting by 12.5 m Northing by 5 mRL with sub-blocking to 2.5 m cubic dimensions for the purpose of providing appropriate definition of the topographic surface, geological and mineralisation zone boundaries.

Table 44. Block Model Dimensions

Block Model Dimensions					
	Origin	Extent	Number	Block Size	
				Parent	Sub-block
East	9,435,287.5	562.5	45	12.5	2.5
North	4,523,275.0	712.5	57	12.5	2.5
Elevation	280.0	230.0	46	5	2.5

14.5.3 Block Model Validation

The block model has been extensively validated against the geological model wireframes and the surface topography. The model has been validated by viewing in multiple orientations using the 3-D viewing tools in Vulcan.

14.5.4 Bulk Density Assignment

The mean bulk density data determined as a result of the statistical assessment has been incorporated into the block model as displayed in Table 45.

Table 45. Block Model Bulk Density Assignments

Block Model Bulk Density Assignments		
Rock Type	**Oxidation**	**Density**
Overburden	N/A	2.00
Breccia Conglomerate	Strong/Moderate Weak Fresh	2.26 2.48 2.53
Basement	Strong/Moderate Weak Fresh	2.30 2.50 2.61

14.6 Grade Estimation

14.6.1 Introduction

Mineral resource estimation for the Ada Tepe Deposit has been undertaken using MIK and OK as the principal estimation methods for gold. Post processing of the MIK results was undertaken to produce a whole block or E-type mean estimate and selective mining unit (SMU) emulation for gold. Silver grades have been estimated by linear regression from the gold grade estimates for all estimation scenarios.

The Wall Zone is considered well constrained and the majority of the zone is likely to be above the economic cutoff of an open cut mining operation. As such, OK is considered the appropriate estimation method. The Upper Zone mineralisation is less amenable to linear estimation techniques such as OK as the interpretation of discrete mineralisation zones is problematic. As such, the Upper Zone domains have been estimated using MIK such that a recoverable mineral resource model (SMU estimates) can be adequately replicated. In addition, MIK is considered appropriate due to the high level of short scale variability present, the highly skewed gold grade distributions and high grades.

CSA considers the basis upon which RSG estimated Mineral Resources using OK and MIK to be valid.

All primary estimates were completed using the Vulcan implementation of the GSLib indicator kriging, ordinary kriging and inverse distance algorithms, while post-processing of the MIK results was completed using RSG Global software.

14.6.2 Ordinary Kriging - Gold

The preferred gold estimates for the Wall Zone and overburden have been completed by OK using a staged sample search (Table 46). The sample search orientation parameters (for the Wall Zone) reflect the mineralised domain interpretation and variography, while the staged estimation approach is designed to force relatively more input data to be used to estimate regions well informed by drilling, and less input data to be used in regions of lower density drilling. In addition, a maximum of 8 composites from any one drillhole were used to restrict the effects of data clustering. The applied sample search parameters are the result of detailed search neighbourhood testing.

Table 46. Sample Search Parameters - Ordinary Kriging of Gold Grades for Wall and Overburden Zones

Sample Search Parameters - Ordinary Kriging of Gold Grades for Wall and Overburden Zones										
		Search Orientation			Search Radii			Number of Samples		
Domain	Pass	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis (m)	Semi-Major Axis (m)	Minor Axis (m)	Min	Max	Max / Hole
Wall Zone	1	310	-5	-10	35	45	10	24	32	8
	2	310	-5	-10	80	80	30	16	32	8
	3	310	-5	-10	100	100	40	8	32	8
Overburden	1	85	0	-90	50	50	50	16	24	8
	2	85	0	-90	100	100	100	4	16	8

Estimation for the Wall Zone has used the variogram model parameters determined from grade variography, as discussed in Section 14.4, while estimation for the overburden has used assumed variography with isotropic spatial characteristics, also noted in Section 14.4. Estimates for both the Wall Zone and overburden have been completed using whole block discretisation of 4 points in the east dimension, 6 points in the north dimension, and 2 points in the RL dimension for a total of 58 discretisation points per whole block estimate. Any sub-blocks within the 3-D limit of each whole block were assigned the whole block grade estimate. Domain control was used for both the input composite data and block selections, wherein only data for a particular zone was used to generate grade estimates for the same zone.

A detailed visual and statistical review of the OK gold estimates for the Wall Zone and overburden was conducted including:

- A comparison of the mean kriged grade estimate versus the mean of the composite dataset, including weighting where appropriate to account for data clustering (see Table 47).

- Visual comparison of the input composites data with the block grade estimates in various cross section views and in plan.

Acceptable levels of mean grade reproduction are noted between the block model and input composite data for both the Wall Zone and overburden when edge effects in the declustering for the Wall Zone and large variations in the data spacing for the overburden are considered.

CSA agrees with this conclusion. CSA has prepared validation Swath Plots for the Wall zone as a check, and these are contained in Figure 39 to Figure 41 (Section 14.7.1).

Table 47. Wall and Overburden Zones Comparison of Ordinary Kriged and Composite Mean Grades (Using Upper Cuts)

Wall and Overburden Zones Comparison of Ordinary Kriged and Composite Mean Grades (Using Upper Cuts)			
Domain	OK Mean Grade (Au g/t)	Composites Mean Grade (Au g/t)	
		Naive	Declustered
Wall Zone Overburden	5.42 0.93	6.62 1.12	4.97 1.04

Note: Declustering completed by 1/n weighting applying declustering cell dimensions of 30 mE x 30 mN x 10 mRL.

14.6.3 Multiple Indicator Kriging - Gold

MIK estimation was undertaken based on the 3 m composite gold data within each of the five defined domain groups of the Upper Zone at Ada Tepe. Separate estimates were generated for the regions lying inside versus outside the modelled high grade domains in order to prevent propagation of high grades into the lower grade regions, which often contain significant volumes of unmineralised material. MIK estimation in the high grade domain groups was completed using diluted input datasets which include the data located inside and within 2.5 m outside of the modelled high grade domain boundaries. This approach was used to ensure robust estimates were generated, inclusive of the lower grades along the margins of the high grade domains, and to therefore allow robust SMU emulation to be undertaken.

MIK was completed using a minimum of 10 and maximum of 14 indicator thresholds selected to adequately partition both the sample population and metal distribution within each of the Upper Zone domain groups, as discussed in Section 14.3.4. Sample neighbourhood testing was completed using the derived variography to determine an appropriate search strategy for MIK estimation. This resulted in the use of a staged sample search strategy, as summarised in Table 48. The effects of data clustering were minimised by using a maximum of 8 composites from any single drillhole to complete a block estimate. Estimation within each of the Upper Zone domain groups used the variogram model parameters determined from indicator variography, as discussed in Section 14.4.

MIK estimates for all of the domain groups were completed using whole block discretisation of 4 points in the east dimension, 6 points in the north dimension, and 2 points in the RL dimension for a total of 58 discretisation points per whole block estimate. Any sub-blocks within the 3-D limit of each whole block were assigned the whole block MIK estimates. Domain control was used for both the input composite data and block selections, wherein only the diluted data for the high grade domains within each domain group were used to estimate blocks within the high grade domains of the same group. This same approach was employed for the selection of input data and blocks for estimation within the lower grade Upper Zone domains.

Table 48. Sample Search Parameters – Multiple Indicator Kriging of Upper Zone Domain Groups

Sample Search Parameters – Multiple Indicator Kriging of Upper Zone Domain Groups										
		Search Orientation			Search Radii			Number of Samples		
Group	Pass	Bearing (Z)	Plunge (Y)	Dip (X)	Major Axis (m)	Semi-Major Axis (m)	Minor Axis (m)	Min	Max	Max /Hole
Domains	1	100	20	-90	35.0	30.0	10.0	24	32	8
11 & 12	2	100	20	-90	70.0	70.0	30.0	16	32	8
	1	100	20	-90	35.0	30.0	10.0	24	32	8
Domain 10										
	2	100	20	-90	70.0	70.0	30.0	16	32	8
Domains	1	70	0	75	35.0	35.0	15.0	24	32	8
21-25	2	70	0	75	70.0	70.0	30.0	16	32	8
	1	70	0	75	35.0	35.0	15.0	24	32	8
Domain 20										
	2	70	0	75	70.0	70.0	30.0	16	32	8
	1	85	0	-90	35.0	35.0	12.0	24	32	8
Domains										
31-35	2	85	0	-90	75.0	75.0	25.0	16	32	8
	3	85	0	-90	75.0	75.0	25.0	8	32	8
	1	85	0	-90	35.0	35.0	15.0	24	32	8
Domain 30	2	85	0	-90	75.0	75.0	30.0	16	32	8
	3	85	0	-90	75.0	75.0	30.0	8	32	8
Domains 41, 42, 46,	1	74	0	80	35.0	35.0	15.0	24	32	8
47	2	74	0	80	75.0	75.0	30.0	16	32	8
Domains 42, 43, 44,	1	74	0	80	35.0	35.0	15.0	24	32	8
(soft boundaries)	2	74	0	80	50.0	50.0	22.0	16	32	8
	1	74	0	80	40.0	40.0	15.0	24	32	8
Domain D40	2	74	0	80	80.0	80.0	30.0	16	32	8
	3	74	0	80	100.0	100.0	40.0	8	32	8
	1	80	0	-90	45.0	45.0	15.0	24	32	8

Domain 50	2	80	0	-90	80.0	70.0	25.0	16	32	8
	3	80	0	-90	90.0	90.0	30.0	8	32	8

The MIK (gold) estimates for each of the Upper Zone domain groups have been processed to produce both whole block E-type estimates and SMU estimates, with the later emulating 6.25 mE by 5 mN by 2.5 mRL size blocks. The MIK estimates for the diluted high grade domains and lower grade regions within each domain group were processed separately based on the corresponding declustered intra-class mean grades (between indicator thresholds), and the grade variogram models summarised in Section 14.4.

Variance adjustment factors were calculated and used to determine appropriate change of support ratios (Table 49) to emulate a 6.25 mE by 5 mN by 2.5 mRL SMU via an indirect log normal change of support. The resultant grade tonnage distribution for the high grade and external regions of each domain group have been compared with a corresponding global change of support distribution generated using the discrete gaussian change of support method. A reasonable level of reproducibility is evident in the comparative grade tonnage curves.

Table 49. SMU – Variance Adjustment Ratios

SMU – Variance Adjustment Ratios	
Domain Group	**Variance Adjustment Ratio**
11 & 12	0.144
10	0.112
21, 22, 23, 24 & 25	0.126
20	0.105
31, 32, 33, 34 & 35	0.184
30	0.109
41, 42, 43, 44 & 45	0.096
40	0.043
50	0.136

A detailed visual and statistical review of the whole block and SMU estimate was conducted including:

- A comparison of the block model whole block estimate versus the mean of the composite dataset, including weighting where appropriate to account for data clustering (see Table 50).
- A comparison of the grade estimate against a theoretical change of support analysis generated via the discrete gaussian change of support.
- Visual comparison of the input composites data with the block grade estimates in various cross section views and in plan.

The block model whole block estimates for each of the domain groups report mean grades showing acceptable reproduction of the corresponding declustered composite datasets. It should be noted that block model mean grades for the high grade domains reflect the inclusion of lower grade composites data located outside the high grade domains limits (within 2.5 m) as dilution during estimation, but do not reflect the additional volume (at lower grade) associated with these composites. As such, the block model mean grade for each of the high grade domain groups correctly lies between the corresponding declustered diluted and undiluted mean grades.

Table 50. Upper Zone Domain Group Comparison of Block Model (MIK Etype Mean) and Composite Mean Gold Grades

Upper Zone Domain Group Comparison of Block Model (MIK Etype Mean) and Composite Mean Gold Grades					
Domain Group	Block Model Mean Grade (Au g/t)	Composites Mean Grade (Au g/t)			
		Naive Diluted	Declustered Diluted	Naive Undiluted	Declustered Undiluted
11 & 12	4.91	4.9	3.77	6.86	5.44
10	1.23	N/A	N/A	1.53	1.24
21, 22, 23, 24 & 25	3.09	3.26	2.39	4	3.63
20	0.17	N/A	N/A	0.25	0.22
31, 32, 33, 34 & 35	2.06	2.03	1.94	2.95	3.1
30	0.2	N/A	N/A	0.28	0.28
41, 42, 43, 44 & 45	1.57	2.35	2.16	2.79	2.99
40	0.16	N/A	N/A	0.19	0.17
50	0.23	N/A	N/A	0.21	0.23

Acceptable levels of reproducibility are noted between the input composites data and the block estimates on the basis of visual review. On this basis and other validation checks, it is believed that the MIK whole block and selective mining unit gold estimates are appropriate and robust.

CSA considers the global validation to be reliable. As an additional check CSA completed a validation estimate on selected domains within the Upper Zone using Uniform Conditioning as a check against MIK. The results of this work are summarised in Section 14.7.3.

14.6.4 Silver Grade Estimates by Linear Regression

Estimation of silver grades in the mineral resource block model has been undertaken by linear regression from the block model gold estimates. This is considered a valid method on the basis of the acceptable correlation coefficients reported between the 3 m composite gold and silver grade datasets for the diluted high grade and residual regions within each of the Upper Zone domain groups, and similar datasets for the Wall Zone and overburden. Silver estimates have been assigned into the mineral resource model to accompany all of the different gold estimates (OK and MIK) using the regression equations displayed in Table 51.

Table 51. Derivation of Silver Grade Estimates by Linear Regression from Gold

Derivation of Silver Grade Estimates by Linear Regression from Gold		
Zone	**Domain Group**	**Regression**
Upper Zone	DG1, HG Domains	ag = (au * 0.449) + 0.5
	DG1, Outside HG Domains	ag = (au * 0.367) + 0.5
	DG2, HG Domains	ag = (au * 0.388) + 0.5
	DG2, Outside HG Domains	ag = (au * 0.436) + 0.5
	DG3, HG Domains	ag = (au * 0.500) + 0.5
	DG3, Outside HG Domains	ag = (au * 0.447) + 0.5
	DG4, HG Domains	ag = (au * 0.426) + 0.5
	DG4, Outside HG Domains	ag = (au * 0.456) + 0.5
	DG5, Total	ag = (au * 0.581) + 0.5
Wall Zone	N/A	ag = (au * 0.439) + 0.5
Overburden	N/A	ag = (au * 0.412) + 0.5

The resultant block model silver grade estimates based on the 2004 preferred whole block gold estimates (MIK and OK) have been visually checked in various cross section and plan views, and by comparing the block model mean silver grades with the declustered cut composite silver data for each of the Upper Zone domain groups, the Wall Zone and overburden. The comparison between the block model and declustered composites datasets (Table 52) shows that there is reasonable agreement between the higher grade diluted domains. In regions of poorer correlation, the model reproduction of the input statistics is less robust.

Table 52. Comparison of Block Model (Regressed) and Composite Mean Silver Grades

Comparison of Block Model (Regressed) and Composite Mean Silver Grades						
Zone	Domain Group	Block Model Grade (Au g/t)	Composites Mean Grade (Au g/t)			
			Naive Diluted	Declustered Diluted	Naive Undiluted	Declustered Undiluted
Upper Zone	11, 12	2.71	3.09	2.48	4.2	3.57
	10	0.95	N/A	N/A	1.17	1.08
	21, 22, 23, 24 & 25	1.7	1.92	1.55	2.26	2.17
	20	0.57	N/A	N/A	0.7	0.69
	31, 32, 33, 34 & 35	1.53	2.1	1.91	2.83	2.81
	30	0.59	N/A	N/A	0.66	0.66
	41, 42, 43, 44, 45, 46 & 47	1.17	1.82	1.68	2.05	2.04
	40	0.57	N/A	N/A	0.74	0.74
	50	0.63	N/A	N/A	0.66	0.67
Wall Zone	N/A	2.88	N/A	N/A	4.23	3.42
Overburden	N/A	0.89	N/A	N/A	1.03	1.03

Note: Declustering completed by 1/n weighting applying declustering cell dimensions of 30 mE x 30 mN x 10 mRL.

CSA has reviewed the relationship between gold and silver grades and believes the assumptions regarding the appropriateness of estimating silver grade through linear regression from gold grade in the model, to be valid. The results of the Uniform Conditioning check estimate for silver, completed by CSA on several domains in the Upper Zone (14.7.3) compares favourably to the results obtained by RSG and goes some way to validate the approach taken by RSG to estimate silver grade.

14.7 CSA Validations of the RSG Block Model

CSA completed the following validation checks on the RSG Mineral Resource block model;

- Swath Plots depicting model tonnes, input declustered composite gold grade, output block model gold grade and drill metres per slice, for the Wall Zone.

- Regularisation of blocks within the RSG block model file so that this block model could be used for mine planning work completed by DPM in 2013. A block model report was then generated at a variety of cut-offs and compared to the RSG model as a check. During this regularisation work, the underlying grade interpolations completed by RSG were not changed. The regularisation process simply assigned block proportions (from the RSG model) of tonnage and grade by majority class, material type and oxidation, to the regularised blocks. The resulting model is referred to as the “CSA block model”.

- A check estimate of the CSA block model using Uniform Conditioning, completed for Domains 10, 11 and 12 within the Upper Zone. These domains are considered representative of the grade distribution within the Upper Zone and together account for 27% of contained metal within the Upper Zone.

- On-screen visual comparisons of the RSG block model grades (via MIK) and the CSA Uniform Conditioning check estimate block model grades, for domains 10, 11 and 12.

14.7.1 Wall Zone Swath Plots

Figure 39 to Figure 41 show the comparison of input and output gold grade through the RSG model, by Northing, Easting and Bench;

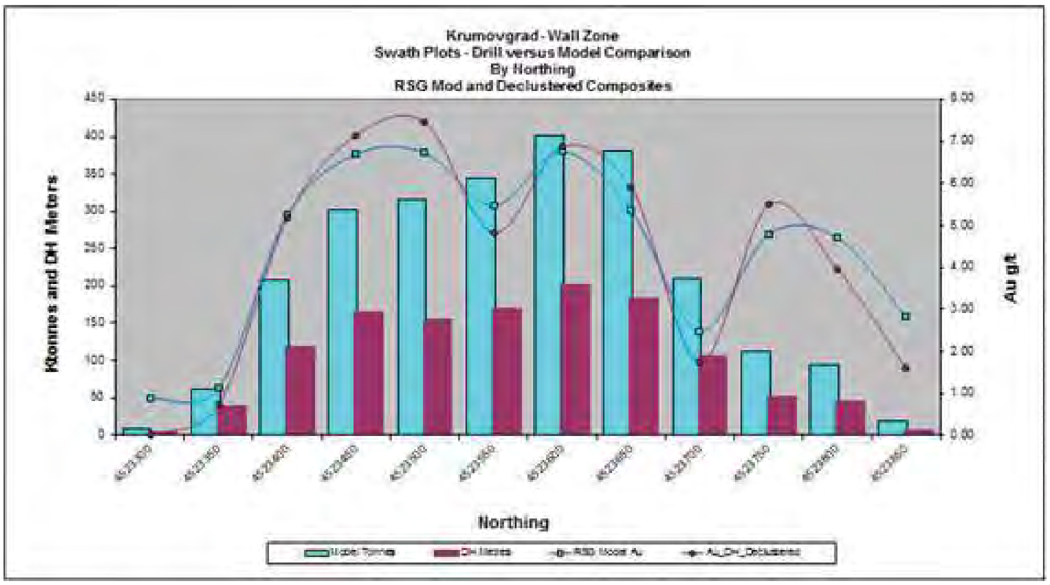

Figure 39. Swath Plot – Wall Zone by Northing (CSA, 2014)

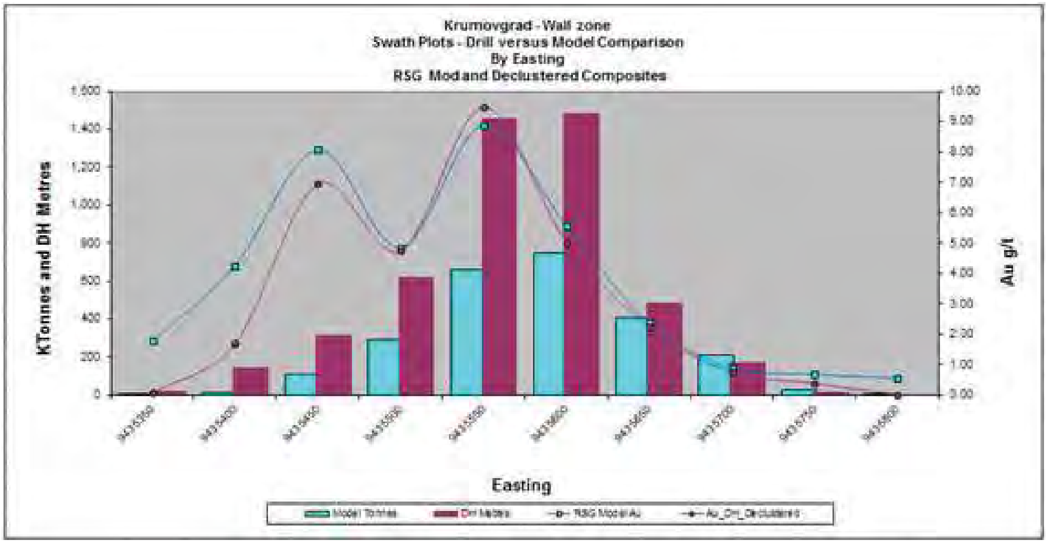

Figure 40. Swath Plot – Wall Zone by Easting (CSA, 2014)

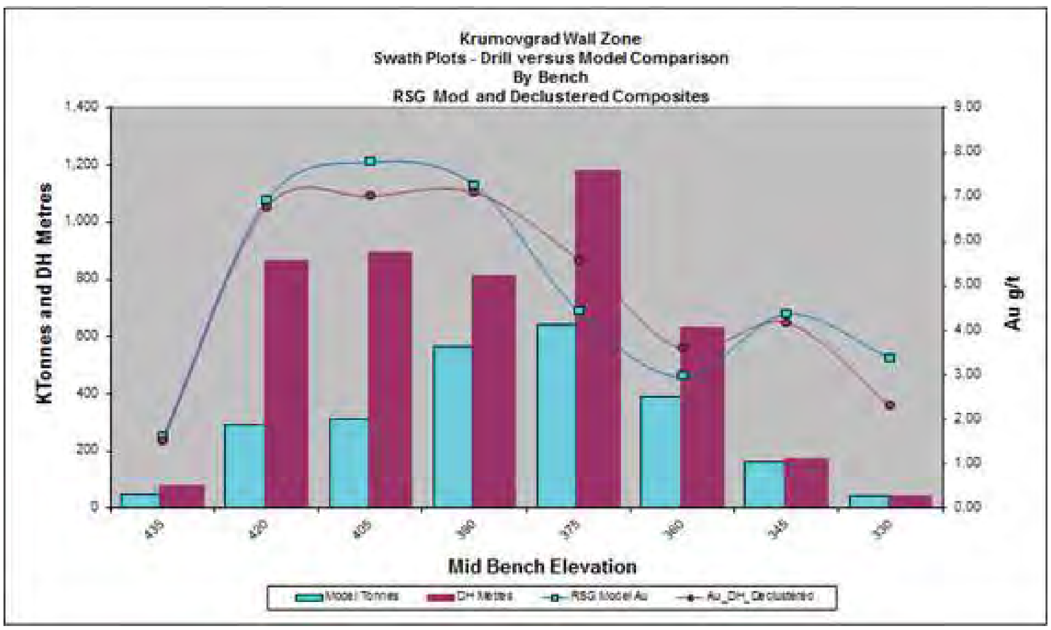

Figure 41. Swath Plot – Wall Zone by Bench (CSA, 2014)

The results of the swath plots show a good comparison between input declustered composite grade and output block model grade. The underlying grade trends through the model honour the trends of the input data. A degree of grade smoothing in the model is evident and expected and closer comparison is evident in areas of the model with higher sample support.

14.7.2 Regularisation of the block model

During discussions between CSA and DPM in November 2012, DPM commented that the RSG block model, as presented, was not suitable from mine planning work being undertaken at that time and planned for 2013. Specifically;

- The block model presented by RSG contains tonnage and grade above cut-off proportion fields for each block of the sub-blocked model. DPM mine planning software (GEMS) requires a regularised block model as the input file, with a single code for material type, oxidation and class. As such, difficulties were encountered when trying to report from the block model, tonnages and grade above cut-off and by material type, class or oxidation.

- Accordingly, CSA considered the approach to regularisation of blocks, taking in to account the following;

    - The regularised model would contain additional proportion and accumulation fields to account for relative block and attribute proportions across boundaries (geological, oxidation and mineralised domain).

    - Care would need to be given to the resulting tonnage, grade and metal proportions across the Wall Zone and Upper Zone boundary since these two grade domains have been estimated using different methods (MIK and OK).

    - The inclusion of all grade thresholds, regularised block proportion information of overburden, Wall Zone, and Upper Zone domains and the three oxidation domains would result in an unwieldy block model with an excessive number of fields to account for the sub-block accumulation function, and this might have an effect on the resultant reported tonnage, grade and metal above each of the cut-offs, and for each material type or class.

    - The accumulation of block proportions to the regularised blocks showed that it was impossible to replicate exact figures of tonnage and grade above cut-off when reporting by class, oxidation or material type since there is no way of knowing how much of the grade above cut-off is contained in the block proportion of class, oxidation or lithology as independent variables. This becomes critical at the boundaries of material type and class over which a regularised block might sit.
- CSA believes a better option to address this “edge effect” is to assign a “block majority” field to the model. The block majority simply assigns the dominant class, oxidation, domain and materially type code to each regularised block.

The resulting regularised block model report was then compared to the RSG tabulated results. The comparison results are presented in Table 53 below;

Table 53. Comparison of the CSA Regularised Block Model with the RSG Block Model

Au	RSG Model (2004)							CSA Regularised Model (2012)			Difference		
											CSA/RSG	CSA/RSG	CSA/RSG
Cut-off	Class Code	Class	Tonnes	Au g/t	Ag g/t	Mt	Kozs (Au)	Mt	Au g/t	Kozs (Au)	Mt	Au g/t	Kozs (Au)
0.5	1	Measured	3300201	4.85	2.59	3.30	515	3.12	4.82	483	-6%	-1%	-6%
0.5	2	Indicated	4687281	2.54	1.57	4.69	383	4.71	2.59	393	1%	2%	3%
0.5	3	Inferred	403393	1.17	0.99	0.40	15	0.35	1.25	14	-14%	7%	-8%
0.5	**Total**	**All**	**8390874**	**3.38**	**1.94**	**8.39**	**913**	**8.18**	**3.38**	**889**	**-3%**	**0%**	**-3%**
0.6	1	Measured	3138643	5.07	2.69	3.14	512	2.95	5.06	480	-6%	0%	-6%
0.6	2	Indicated	4043400	2.86	1.70	4.04	371	4.08	2.91	381	1%	2%	3%
0.6	3	Inferred	378106	1.21	1.00	0.38	15	0.32	1.30	13	-15%	7%	-9%
0.6	**Total**	**All**	**7560148**	**3.69**	**2.08**	**7.56**	**898**	**7.36**	**3.7**	**875**	**-3%**	**0%**	**-3%**
0.7	1	Measured	2988763	5.29	2.79	2.99	509	2.80	5.30	477	-6%	0%	-6%
0.7	2	Indicated	3552432	3.16	1.83	3.55	361	3.60	3.21	371	1%	1%	3%
0.7	3	Inferred	302988	1.35	1.06	0.30	13	0.26	1.45	12	-14%	8%	-8%
0.7	**Total**	**All**	**6844183**	**4.01**	**2.21**	**6.84**	**883**	**6.66**	**4.02**	**861**	**-3%**	**0%**	**-2%**
0.8	1	Measured	2858176	5.50	2.88	2.86	506	2.67	5.53	474	-7%	0%	-6%
0.8	2	Indicated	3165662	3.46	1.95	3.17	352	3.22	3.50	362	2%	1%	3%
0.8	3	Inferred	260091	1.45	1.10	0.26	12	0.22	1.58	11	-16%	9%	-8%
0.8	**Total**	**All**	**6283929**	**4.31**	**2.34**	**6.28**	**870**	**6.11**	**4.31**	**847**	**-3%**	**0%**	**-3%**
0.9	1	Measured	2753216	5.68	2.96	2.75	503	2.56	5.72	471	-7%	1%	-6%
0.9	2	Indicated	2843390	3.75	2.07	2.84	343	2.90	3.79	354	2%	1%	3%
0.9	3	Inferred	233477	1.52	1.13	0.23	11	0.20	1.67	11	-15%	10%	-7%
0.9	**Total**	**All**	**5830083**	**4.57**	**2.45**	**5.83**	**857**	**5.66**	**4.59**	**835**	**-3%**	**0%**	**-3%**
1	1	Measured	2650257	5.86	3.04	2.65	500	2.46	5.91	468	-7%	1%	-6%
1	2	Indicated	2571978	4.05	2.19	2.57	335	2.63	4.08	346	2%	1%	3%
1	3	Inferred	212620	1.57	1.15	0.21	11	0.18	1.73	10	-15%	10%	-6%
1	**Total**	**All**	**5434855**	**4.84**	**2.56**	**5.43**	**845**	**5.27**	**4.86**	**823**	**-3%**	**0%**	**-3%**

The CSA regularised block model reports;

- 3% less tonnes at the same grade and results in 3% less contained metal as compared to the RSG model, for all grade cut-offs.
- The CSA model estimates 6-7% less tonnes at near identical grade and results in 6-7% less contained metals than the RSG model, in the Measured Category.
- The CSA model estimates 1-2% more tonnes at a 1-2% higher grade, resulting in 3% higher contained metal than the RSG model, in the Indicated category
- The CSA model estimates 14-15% less tonnes at a 7-8% higher grade, resulting in 6-9% less contained metal than the RSG model, in the Inferred category. It should be noted that the Inferred portion of the Mineral Resource accounts for 4% of total resource tonnage, and negligible contained metal.

CSA considers the regularised model to be reliable and valid for use in mine planning work and that the model adequately honours the RSG reported global resource at each cut-off. The CSA regularised model (mikclsmod.csv) was presented to DPM and used in mine planning work completed in 2013-14 and for conversion to Mineral Reserves.

14.7.3 Uniform Conditioning Check Estimate – Upper Zone Domains 10, 11 & 12

14.7.3.1 Introduction

Uniform Conditioning ("UC") is a recoverable estimation method. Like MIK, it estimates tonnes and grade above cut-offs for a given SMU size. It's advantageous where drill spacing is widely enough spaced that grades have to be estimated into blocks larger than SMU sizes, but where a recoverable estimation is required or might be desirable. It has advantages over Ordinary Kriging ("OK") in situations like these because reporting above cut-offs using an OK estimate is unreliable if blocks are too small relative to the grid spacing. OK grades tend to be increasingly smoothed when blocks are larger. Metal content is therefore lower for OK at most grade cut-offs.

UC differs from MIK in that it uses the SMU dimensions, the variogram model (in this case the RSG variogram models) and the in-situ grade to estimate the recoverable resource. The change of support correction required for MIK (from global to recoverable resource) is a mathematical correction which can be arbitrary and can often rely on empirical correction factors. MIK can be favoured for some types of deposits, but UC is often considered advantageous over MIK for the following reasons:

- It is quick to implement as there is no requirement for the generation of many variograms above cut-offs, which is a requirement of MIK. As one gets to higher cut-offs, often the data support drops off considerably, making variograms for those cut-offs less reliable.

- It includes a more robust change of support. It relies on the variogram of the underlying data to inform the change of support (from large blocks to small SMUs) rather than the more arbitrary change of support methods used in MIK which often have to be modified using empirical factors.

- A UC model can be localised to SMU sized blocks using a process called Localised Uniform Conditioning ("LUC"). This moves away from the fairly unwieldy format of MIK models where mine planning can become difficult when faced with a block model file that contains fields of proportions and grades above cut-off. A LUC model has a single grade per SMU, while perfectly representing the Grade Tonnage Curve of each larger panel block.

- Often in precious metal deposits there are areas of higher and lower grade, between which the grade profile is not a sharp contact but a gradational one. In these instances, and Ada Tepe is an example, UC is preferred to MIK. MIK is based on the assumption that grade classes are independent of each other (mosaic). UC relies on the assumption that grades are gradational (diffusive) i.e. generally, to move from high grades to low grades, one must move through intermediate grades.

14.7.3.2 CSA UC Check Estimate

The process by which the UC estimate was completed in summarised below.

- Flagging the raw assays with domain wireframes.
- Compositing flagged raw assays to 1 m. A preliminary review on the impact of composite lengths was completed by CSA. The results informed the decision taken by CSA to composite to 1 m for the check estimate. This differed from the 3 m (including residuals >1 m) used by RSG. The reasons for this were based on 1 m being the dominant sampling interval. In addition, in a review of the residuals formed by 1 m compositing, the statistics showed that no bias was being introduced by using residuals and 1m composites. This was considered preferable over the removal of <1 m residuals from 3 m composites, as in the RSG estimate. A review of residuals between 1 and 3 m show grades of approximately 25% less (globally) than those seen in 3 m (and 1 m) composites. CSA believes that through the use of 1 m composites, grade bias is reduced.
- Top-cuts were used to control the influence of high grade outliers. Top cuts of 100 g/t Au and 46 g/t Ag were imposed during Ordinary Kriging.
- Variography was completed on uncut 1 m composites that had been de-clustered using cell de-clustering. Rotations reflected those seen in the RSG models. The nugget was high for Au which also reflects the results seen in the RSG model. Approximately 85% of the Au variability was modelled to be within the first 15 m. The nugget for Ag was approximately 20%, with approximately 60% of variability within the first 15 m.
- Ordinary Kriging was used to estimate in-situ grades of parent blocks (12.5 m x 12.5 m x 5 m), while Uniform Conditioning was used to estimate recoverable grades and tonnes of each parent block based on an SMU size of 6.25 m x 6.25 m x 2.5 m as per the RSG model.
- Soft boundaries were used between domains to minimise potential high-grading. This is in contrast to the method used by RSG which instead used an envelope around the wireframes of domains 11 and 12.
- Validation comprised reviewing the block models spatially plan view and cross sections.
- The Kriged results prior to the UC estimation were reviewed against input composites to ensure that the global estimate was representative of the underlying data and swath plots were generated so that results could be compared spatially.
- CSA believes that the results of the check estimate adequately supports the results of the portion of the model estimated using MIK. The higher grades seen at the lower grade cut-offs (including the reporting cut-off) may reflect the inclusion of lower grade residual data (1 to 3 m) in the RSG estimate. Residuals included in the check estimate reflected the same grades as 1m composites and raw data. However, in the context of the overall resource, and classification, these differences are not considered material.

The results of the UC estimate is summarised in Table 54 below, and illustrated in Figure 42 and Figure 43.

Table 54. Comparison Table of the CSA UC Check Estimate against the CSA Block Model

CUTOFF	Tonnes_MIK	Au g/t_MIK	Ag g/t_MIK	KOz Au_MIK	AgKoz_MIK
0.5	1,547,738	2.78	1.60	138.40	79.79
0.6	**1,354,148**	**3.10**	**1.73**	**134.99**	**75.38**
0.7	1,204,207	3.41	1.85	131.89	71.78
0.8	1,085,762	3.70	1.97	129.06	68.80
0.9	987,029	3.98	2.09	126.39	66.19
1	903,120	4.27	2.20	123.85	63.87
CUTOFF	**Tonnes_UC**	**Au g/t_UC**	**Ag g/t_UC**	**KOz Au_UC**	**AgKoz_UC**
0.5	1,465,568	3.05	1.71	143.85	80.37
0.6	**1,302,099**	**3.37**	**1.82**	**140.98**	**76.22**
0.7	1,172,885	3.67	1.93	138.28	72.72
0.8	1,068,738	3.95	2.03	135.78	69.73
0.9	981,309	4.23	2.13	133.40	67.09
1	907,653	4.49	2.22	131.15	64.75
CUTOFF	**Tonnes_UC**	**Au g/t_UC**	**Ag g/t_UC**	**KOz Au_UC**	**AgKoz_UC**
0.5	-5%	10%	6%	4%	1%
0.6	**-4%**	**9%**	**5%**	**4%**	**1%**
0.7	-3%	8%	4%	5%	1%
0.8	-2%	7%	3%	5%	1%
0.9	-1%	6%	2%	6%	1%
1	1%	5%	1%	6%	1%

The UC check estimate compares favourably to the CSA block model interpolated by RSG using MIK. Additionally, CSA makes the following comment relating to the check estimate;

- The check estimate reports 4% less tonnes at a 9% higher gold grade, resulting in a 4% increase in contained gold within the check domains, at a 0.6 g/t Au cut-off (the cut-off used to report Mineral Resources and Mineral Reserves in the Upper Zone).
- The check estimate reports 4% less tonnes at a 5% higher silver grade, resulting in a 1% increase in contained silver within the check domains, at a 0.6 g/t Au cut-off.
- UC has validated the MIK results suggests the variogram models and the linear regression of silver to be valid.

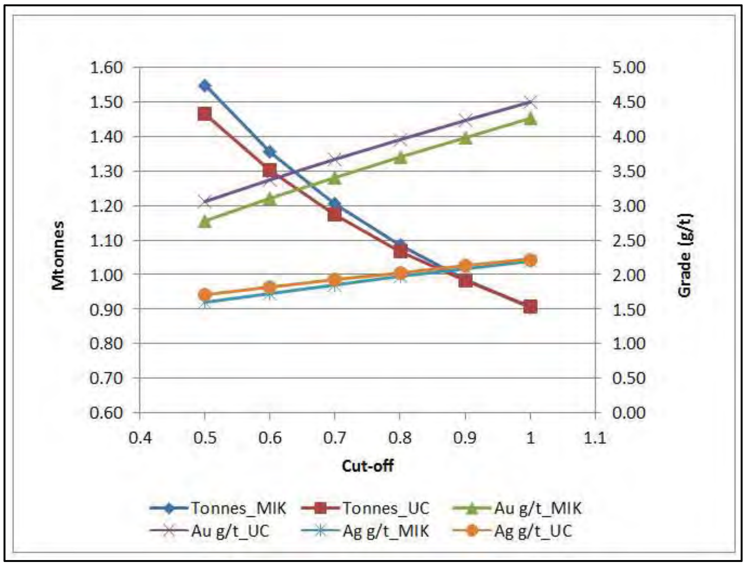

Figure 42. Grade-Tonnage Curve for Au and Ag – MIK and UC estimates (CSA, 2014)

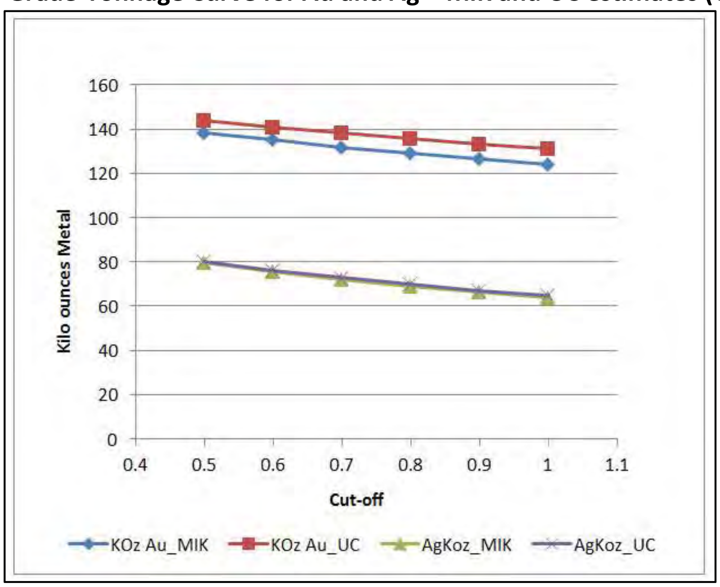

Figure 43. Metal Inventory – MIK and UC estimates (CSA, 2014)

14.8 Mineral Resource Reporting

14.8.1 Introduction

The Mineral Resource Estimates for the Ada Tepe deposit have been categorised in accordance with the criteria laid out in the NI 43-101. A combination of Measured, Indicated and Inferred Mineral Resources have been defined using definitive criteria determined during the validation of the grade estimates, with detailed consideration of the NI 43-101 categorisation guidelines.

14.8.2 Criteria for Mineral Resource Categorisation

The Mineral Resource Estimates have been classified as a combination of Measured, Indicated and Inferred Mineral Resources based on the confidence level of the key criteria that were considered during mineral resource classification, which include robustness of the geological model, reliability, accuracy and precision of input data, grade interpolation parameters and data spacing.

14.8.2.1 Measured Mineral Resources

- Wall Zone blocks with an OK gold estimate located at a distance of <=12.5 m to the nearest composite used for estimation using a minimum of 16 composites (max of 32) collected from at least 2 drillholes.

- The Upper Zone blocks are located within two regions defined by grade control drilling completed at 12.5 m x 12. 5m drill spacing.

14.8.2.2 Indicated Mineral Resources

- Wall Zone blocks with an OK gold estimate located at a distance of <=30m to the nearest composite used for estimation, and not classified as a Measured Mineral Resource. This estimate required a minimum of 16 composites (max of 32) collected from at least 2 drillholes.

- Upper Zone blocks with MIK estimates located within the region drilled at a notional 25 mE x 25 mN drill density, and not classified as a Measured Mineral Resource.

14.8.2.3 Inferred Mineral Resources

- Wall Zone blocks with an OK gold estimate not classified as a Measured or Indicated Mineral Resource.

- Upper Zone blocks for all domain groups with MIK estimates not classified as a Measured or Indicated Mineral Resource.

- Overburden blocks with OK gold estimates.

Table 55. CIM Mineral Resource Statement – Ada Tepe Deposit – Upper Zone and Overburden (CSA, 2013)

Dundee Precious Metals - Krumovgrad Ada Tepe Mineral Resource Estimate as at 31st December, 2013 Upper Zone and Overburden reported at a 0.6 g/t Au cut-off					
Resource Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Metal Content	
				Au (Moz)	Ag (Moz)
Measured	1.1	3.46	1.91	0.125	0.069
Indicated	3.9	2.86	1.70	0.357	0.212
Total M+I	5.0	2.99	1.75	0.482	0.281
Inferred	0.3	1.31	1.06	0.013	0.011

The MRE has estimates gold in to mineral domains via MIK in to blocks with dimensions 12.5 m x 12.5 m x 5 m. Silver estimation is via linear regression from gold values.

MRE is reported using a gold cut-off of 0.6g/t for classified resources in the Upper Zone and Overburden.

Measured and Indicated Mineral Resources are inclusive of Proven and Probable Mineral Reserves.

Tonnages are rounded to the nearest 0.1 million tonnes to reflect this as an estimate.

Metal content is rounded to the nearest 1000ozs to reflect this as an estimate.

Table 56. CIM Mineral Resource Statement – Ada Tepe Deposit – Wall Zone (CSA, 2013)

Dundee Precious Metals - Krumovgrad Ada Tepe Mineral Resource Estimate as at 31st December, 2013 Wall Domain reported at a 0.8 g/t Au cut-off					
Resource Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Metal Content	
				Au (Moz)	Ag (Moz)
Measured	1.7	6.32	3.27	0.353	0.183
Indicated	0.2	4.28	2.38	0.024	0.014
Total M+I	1.9	6.13	3.19	0.377	0.196
Inferred	0.0	0.87	0.88	0.000	0.000

The MRE has estimates gold in to mineral domains via OK in to blocks with dimensions 12.5 m x 12.5 m x 5 m. Silver estimation is via linear regression from gold values.

MRE is reported using a gold cut-off of 0.8g/t for classified resources in the Wall Zone.

Measured and Indicated Mineral Resources are inclusive of Proven and Probable Mineral Reserves.

Tonnages are rounded to the nearest 0.1 million tonnes to reflect this as an estimate.

Metal content is rounded to the nearest 1000ozs to reflect this as an estimate.

CSA believes the estimate of Mineral Resources would not be materially affected by changes to metallurgical, environmental, permitting, legal, title, socio-economic, marketing or political circumstances. CSA believes the risks regarding permitting and socio-economic factors to be low. CSA has relied on information provided by DPM as regards legal and environmental risks (See Section 3).

15 Mineral Reserve Estimates

The mining method is a conventional open cut, drill, blast, load and haul operation, using hydraulic excavators and haul trucks to mine the material. The mining equipment will be owner operated and will be maintained under a contract with the equipment supplier.

The mine planning update consisted of a pit optimisation followed by open pit design, long term production scheduling and cost estimation. The main differences in relation to the previous study were: (a) the use of updated economic parameters such as metal prices, metallurgical recoveries, royalty and discount rate; and (b) adoption of a diluted model were used to account for operational mine dilution and expected level of selectivity. The final pit limit was defined based on a maximum NPV criterion using a gold price of USD 1,350/oz and a silver price of USD 23/oz.

The optimisation process is discussed in Section 16.

The open pit was designed taking into consideration the geotechnical recommendations by George Orr and Associates (2005). The updated slope design has also taken into consideration the weathered rock material in the NE corner of the pit, near the surface, and the presence of historical waste dumps in the SE corner of the pit, also near the surface. Four incremental cutbacks were designed (a) to give early and consistent access to the Wall zone material and (b) to ensure the provision of sufficient waste rock in the early stages of the operation to enable construction of the cells on the IMWF. Consideration has also been taken into account of the restricted stockpile area for both ROM and low grade material. A new life of mine (LOM) mining schedule has been prepared on the basis of mill throughput of 105 t/hr for upper zone and 90 t/hr for wall zone ore. This gives a mine life of 8 years, with maximum annual rock movement of approximately 3.2 Mt. Low grade material, between 0.6 and 0.8 g/t is stockpiled on a separate area with maximum capacity 0.5 Mt. Much of this is planned to be delivered to the process plant in the latter years of the mine's life.

The estimated Mineral Reserve figures are presented in Table 57 and Table 58. The numbers are appropriate for the purpose of public reporting in that they provide an acceptable prediction of the material available to mine.

This reserve estimate has been determined and reported in accordance with NI 43-101, 'Standards of Disclosure for Mineral Projects' (the Instrument, June 2011) and the classifications adopted by CIM Council in 2010.

Table 57. Ada Tepe Deposit – CIM compliant Ore Mineral Reserves Summary (DPM, 2014) as at 21st March 2014

Mineral Reserve Summary					
Category	Tonnes (Mt)	Gold		Silver	
		Grade (g/t)	Ounces (Koz)	Grade (g/t)	Ounces (Koz)
Proven	2.59	5.39	449	2.82	235
Probable	3.61	3.08	358	1.79	208
Total	6.20	4.04	807	2.22	443

**Reserves estimated using a cut-off of 0.6 g/t Au for the Upper Zone and 0.8 g/t Au for the Wall Zone.*

Table 58. Ada Tepe Deposit – CIM compliant Ore Mineral Reserves Summary (by material type) as at 21st March 2014

CIM Compliant Mineral Reserves (31st March 2014)					
Category	**Wall Zone (0.8 g/t cut-off)**				
	Tonnes (Mt)	**Gold**		**Silver**	
		Grade (g/t)	**Ounces (Koz)**	**Grade (g/t)**	**Ounces (Koz)**
Proven	1.48	6.83	325	3.50	166
Probable	0.11	5.54	20	2.93	11
Total	1.59	6.74	345	3.46	177
Category	Upper Zone (0.6 g/t cut-off)				
	Tonnes (Mt)	Gold		Silver	
		Grade (g/t)	Ounces (Koz)	Grade (g/t)	Ounces (Koz)
Proven	1.11	3.46	124	1.91	68
Probable	3.50	3.00	337	1.75	197
Total	4.61	3.11	461	1.79	266

Reserves have been estimated using a gold cut-off of 0.6 g/t for the Upper Zone, and 0.8 g/t for the Wall Zone. Probable ore includes low grade ore that is processed at the completion of mining operations. Mineral Reserves are estimates using USD 1,250/oz Au and USD 23/oz Ag.

The Mineral Reserves at Krumovgrad have been estimated by including a number of technical, economic and other factors. A change to any of the inputs would therefore have some effect on the overall results. Concerning mining and metallurgical factors, it is CSA's belief that sufficient work has been done by DPM to ensure that these are not likely to have any significant or material effect on Mineral Reserves.

However, CSA relies on information as presented in Section 3 of this Technical Report as regards legal and environmental considerations.

16 Mining Methods

16.1 Mine Design

The geology and physical attributes of the deposit suggest that open pit mining is appropriate for Krumovgrad. Hydraulic excavators and off-highway haul trucks are proposed. All pit mobile equipment will be diesel powered.

To ensure maximum mining recovery while minimizing dilution, the bench height has been set at 5 m with ore extraction split into two flitches of 2.5 m. Mining operations will be carried out on two 8.5 hour shifts with minimal work, mainly on equipment maintenance and servicing, during the hours of darkness, given the close proximity of the Krumovgrad town site.

The procedure used to develop a mine plan and estimate mineral reserves for the Krumovgrad Gold Project included the following stages:

- Validation and regularisation of blocks within the RSG block model by CSA to 12.5 m by 12.5 m by 5 m blocks size using the MIK estimates for the Upper Zone which accounts for dilution and ore loss in the grade estimation.
- Add a mining cost adjustment factor to each block to quantify the incremental mining cost with depth;
- Pit optimisation to define the final pit limits;
- Design final pit and intermediate cutbacks taking into account operational and geotechnical considerations;
- Production of a mine schedule using GEOVIA MineSched software;
- Detailed design of the pit configuration at the end of each period;
- Design the pit configuration at the end of each month for the first period;
- Develop haulage profiles to calculate truck cycle times and productivities;
- Determine mine equipment requirements for each period;
- Calculate mining capital expenditure and operating costs.

16.2 Geotechnical Parameters

In 2005 George Orr Associates were commissioned to conduct a geotechnical study of the Krumovgrad project and investigate parameters to be used for a pit design for Ada Tepe. This report (Orr, 2005) forms the basis of the geotechnical considerations for the pit designs conducted by DPMKr in 2012. The report focuses on the rock quality for the different rock types. 'Key to this was the groundwater table, which is located at depths below ground surface between 2 m and 85 m, at elevations RL 403 and RL 370. The report indicated that the presence of groundwater resulted in weathering of the relatively shallow rock.

Due to the interpreted highly variable nature of rock weathering and alteration (which has resulted in a variable degree of local strengthening and weakening of the wall rocks), the consultants considered that it was not possible to realistically subdivide the proposed pit into distinct geotechnical domains. The following tables (Table 59 and Table 60) show the geotechnical parameters used for the 2012 mining study open pit design.

Table 59. Interim pit slope parameters

Interim pit slope parameters			
RL (m)	Face angle (deg)	Bench height (m)	Berm width (m)
+ 430	45	10	5
400 to 430	60	10	5
below 400	60	20	5

Table 60. Final pit slope parameters

Final pit slope parameters			
RL (m)	Face angle (deg)	Bench height (m)	Berm width (m)
+ 430	45	10	5
400 to 430	60	10	6
below 400	65	10	5

In 2013 Golder Associates conducted an open pit mining review (Golder, 20134) of the pit designs completed by DPMKr for the January 2012 mining study. It concluded that in general the geotechnical parameters used in the open pit designs were conservative in nature, but two areas of the pit (dark blue areas in Figure 44 below) failed to meet a minimum target Factor of Safety (FoS) of 1.2 for Inter Ramp Angles (IRA), also, it was identified that the pit designs intercepted historical mine waste on the eastern wall of the pit.

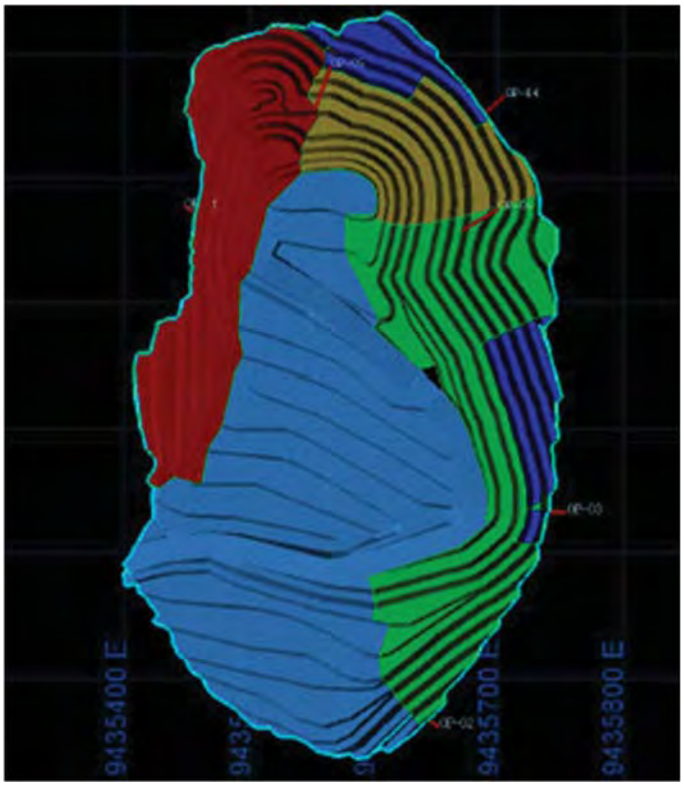

Figure 44. Ada Tepe open pit design. Areas depicted in dark blue indicate areas that Golder Associates identify as having low factor of safety. (Golder, 20134)

Golder Associates recommended that the historical mine waste be excavated so as not to form part of the pit wall; also, the mining study open pit designs be re-designed in these areas using updated Bench Face Angles (BFA) of 35 degrees instead of the original 45 degrees for weathered material, with an increase in bench height and berm widths from 10 m and 5 m to 15 m and 7.5 m respectively for the whole design (Golder, 20134).

In November 2013 DPM redesigned the open pit with the 35 degree BFA's for the areas identified as not reaching the minimum target FoS. Golder Associates conducted an open pit design validation (Golder, 20136) using the updated open pit mine design completed by DPM, this report concluded that the target FOS had been achieved for the redesigned areas.

16.3 Optimisation Parameters

16.3.1 Mining and Processing Costs

The cost estimates are in-house calculations based on costs for similar size operations belonging to Dundee, the reference mining cost is USD 2.96/t for all pit material, ore or waste. A mining cost adjustment factor (MCAF) of USD 0.01/t/m was applied. The reference level is RL 390m, below which the pit material has an incremental cost rate equivalent to the MCAF and material above reference level, 70% of the MCAF was applied. The mining costs used in the pit optimisation were based on an owner mining scenario with Dundee purchasing and operating all mining equipment. Therefore, no allowances were made for major equipment capital costs, escalation or taxation in the calculation of the cash flow values in the pit optimisation costs. A summary of pit optimisation parameters can be found in Table 61.

The processing costs details are as follows:

- Processing operating cost for Wall Zone material: USD 22.60/t treated.
- Processing operating cost for Upper Zone material: USD 17.50/t treated.
- Tailings treatment & IMWF cost: USD 2.50/t treated.
- G&A: USD 3.71/t treated.
- Rehabilitation: USD 0.75/t treated.

16.3.2 Selling cost

The selling cost is related to trading expenses, shipping, administration, taxation, etc. The cost of USD 6.22 per ounce has been provided by DPM.

16.3.3 Metal Prices and Discount Rate

The metal prices used for the purpose of pit optimisation were:

- Gold price: USD 1,350/oz; and
- Silver price: USD 23/oz.

For the calculations of NPV, that are required to select the ultimate pit limit, a discount rate of 7.5% was used.

16.3.4 Royalty

A royalty payment of 2.57% of the value of the contained metal within the ore mined, will be payable.

16.3.5 Processing Parameters

The key process parameters required for pit optimisation are the metallurgical recovery and the plant throughput. The parameters adopted in this study were:

- Gold recovery of 85% and silver recovery of 70% in both zones;
- For NPV calculations the plant throughput has been fixed at 0.85Mt per year;
- No other plant parameters or any penalty elements were specified.

Table 61. Summary of Pit Optimisation Parameters

Optimisation Parameters	Units	
Gold Price	USD/oz.	1350
Silver Price	USD/oz.	23
Selling Cost	USD/oz.	6.22
Royalty	%	2.51
Gold Recovery	%	85
Silver Recovery	%	70
Processing (Wall Zone)	USD/t	22.70
Processing (Upper Zone)	USD/t	18.00
Tailings Treatment & IMWF Cost	USD/t	2.50
G&A	USD/t	3.71
Rehabilitation	USD/t	0.75
Reference Mining Cost	USD/t(mat)	3.34
Cut-off Grade (Wall Zone)	g/t	0.80
Cut-off Grade (Upper Zone)	g/t	0.60
Mining Cost Adjustment Factor Vertical (MCAFV) The MCAFV is USD 0.01/t/m with depth from reference elevation. The base level to apply the MCAFV was defined as RL 390m, below which the material has an incremental cost rate equivalent to the MCAFV. For material above the base level, 70% of the MCAFV was applied.		

16.3.6 Pit Shell Selection

The optimum pit shell selection criterion was based on the maximum discounted cash flow (NPV). Pit shell number 30 has been selected as the ultimate pit limit which lies between the maximum discounted cash flow for the best case scenario and worst case scenario. This shell contains approximately 22.8 Mt of rock (ore and waste) and 6.4 Mt of ore with an average grade of 4.1 g/t for gold and 2.2 g/t for silver.

16.4 Pit Design

The selected economic pit shell was used to drive the pit design work. The pit design was checked against the Whittle shells to ensure they match reasonably well. In general, the pit differs horizontally from the shell by 10 to 15 metres, which is considered to be acceptable. The final design contained about 3.2% less ore and 0.8% less waste than the Whittle shell. This is considered reasonable considering the requirement for ramps and other practical mining considerations. A three-D view of the final pit design is presented in Figure 45.

The mining operation is planned to be carried out via three intermediate, and a final fourth cutback (Phases 1 to 4). Mining of the cutbacks is developed towards the north, reaching the final design limit in the south, east and west for each of Phases 1 to 3 (Figure 46). The final pit is up to 115 meters deep and has a width at surface of more than 350 meters. The elevation of the deepest bench in the fourth cutback is 340 mRL and the maximum elevation of the pit benches is 455 mRL.

The cutbacks were designed with 20 m wide ramps to allow more space at the bottom of the pit. The pit ramp exits the pit at south west.

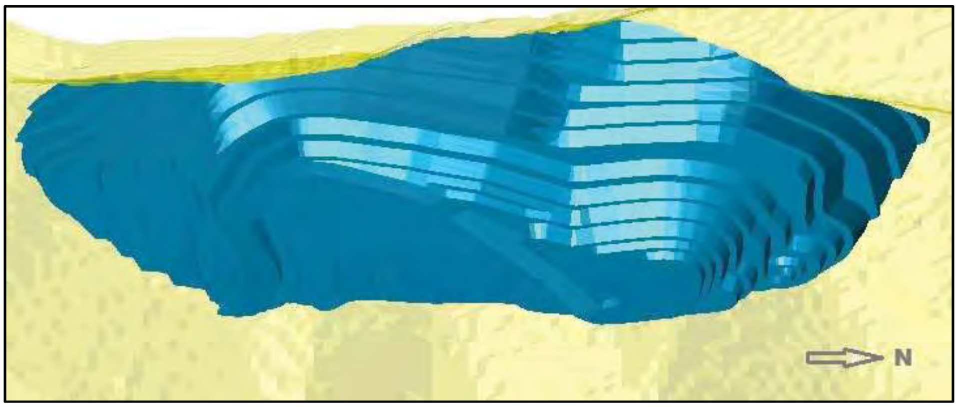

Figure 45. Final Pit Design 3-D View

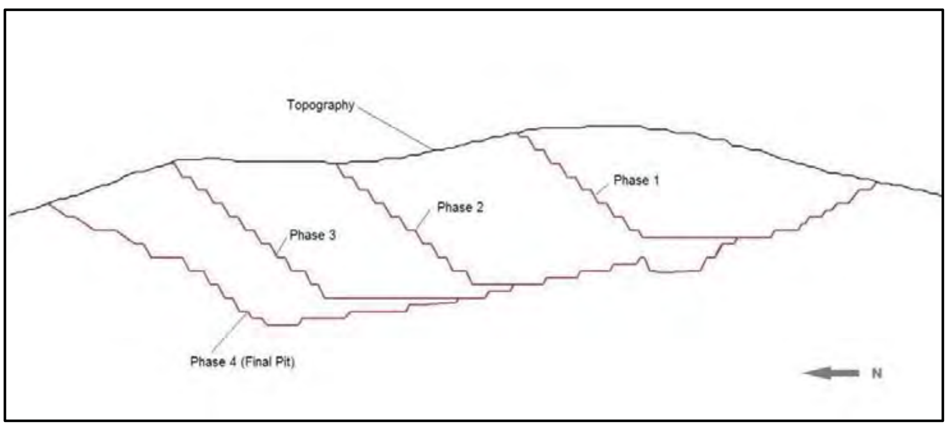

Figure 46. Mining Approach

16.5 Production Scheduling

The mine schedule was derived using GEOVIA MineSched software. The key constraints applied to the schedule were the designed mining cutbacks and a limited ROM pad surface area of about 20,000 m2. The criteria used in the construction of the mine production schedule were:

- Mine the cutbacks in order of best value earliest.

- Respect the geotechnical considerations and other operational constraints.
- Ensure continuity of ore exposure to provide the necessary plant feed requirements.
- Waste requirement to build Integrated Mine Waste Facility (IMWF) cells ahead of tailings deposition.

The ore feed continuity is ensured by an appropriate balance between the rate of ore extraction and waste removal as well as the use of stockpile.

The base information for the preparation of the mine plan was the following:

- Mineral Resource Base: Measured and Indicated Mineral Resources inside the final pit
- Mine operating days: 365 days/year
- Plant operating days: 365 days/year
- Mill throughput: 105 t/hr for upper and 90 t/hr for wall zone ore
- Maximum rock movement: 3.2 Mt per year
- Low Grade Stockpile capacity: 500 kt
- Cut-off grade strategy: Upper zone 0.6 g/t Au; Wall zone 0.8 g/t Au

Table 62 presents a summary of quantities for the production schedule developed for a mill throughput of 105 t/hr for upper zone and 90 t/hr for wall zone ore. The maximum annual plant feed is 0.84 Mt. The information in this table shows that:

- Based on the mineral reserves contained in the selected final pit limit a mine plan has been developed that is able to feed the plant for a period of 8 years.
- The mine plan amounts to a total mineral reserve of 6.20 Mt with an average grade of 4.04 g/t Au and average waste:ore ratio of 2.62.

The suggested mine schedule achieves a balanced rock movement to allow a uniform fleet size throughout the mine life.

Table 62. Production Schedule for 0.85 Mtpa

Production Schedule																	
Year	Rock	Total Ore	Total Waste	HGO Stockpile	LGO Stockpile	Strip	Ore to Process										
							wall			upper			Total Ore				
	Mt	Mt	Mt	Mt	Mt	t:t	Mt	Au (g/t)	Ag (g/t)	Mt	Au (g/t)	Ag (g/t)	Mt	Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
-1	0.65	0.23	0.42	0.17	0.06	1.83	0.00	0.00	0.0	0.00	0.00	0.0	0.00	0.00	0.0	0	0
1	2.66	0.84	1.82	0.02	0.25	2.16	0.01	1.85	1.3	0.80	3.45	1.9	0.81	3.42	1.9	89	50
2	2.15	1.13	1.01	0.21	0.38	0.90	0.22	6.49	3.3	0.58	4.46	2.3	0.80	5.01	2.6	129	67
3	2.03	0.88	1.16	0.17	0.49	1.32	0.18	9.77	4.8	0.64	4.42	2.3	0.81	5.59	2.8	146	74
4	3.21	0.74	2.47	0.00	0.57	3.31	0.02	9.59	4.7	0.82	3.26	1.9	0.84	3.40	1.9	91	51
5	3.21	0.70	2.51	0.00	0.49	3.60	0.40	9.53	4.7	0.37	2.35	1.5	0.77	6.06	3.2	151	79
6	3.21	0.57	2.64	0.00	0.27	4.67	0.33	5.27	2.8	0.46	1.89	1.3	0.79	3.29	1.9	83	49
7	3.10	0.65	2.44	0.00	0.14	3.73	0.32	3.95	2.2	0.47	2.06	1.4	0.79	2.83	1.7	72	44
8	2.22	0.46	1.77	0.00	0.00	3.85	0.12	4.91	2.7	0.48	1.73	1.2	0.60	2.37	1.5	45	29
Total	**22.45**	**6.20**	**16.24**			**2.62**	**1.59**	**6.74**	**3.46**	**4.61**	**3.11**	**1.8**	**6.20**	**4.04**	**2.2**	**807**	**443**

16.6 Mining Equipment

A relatively small scale mining fleet was selected to suit the proposed production rate and the selective mining requirements. Rigid and articulated 40 t trucks were selected together with 3.7 m³ hydraulic backhoes. It is planned to use the rigid trucks for ore transport, and the articulated trucks for the waste haul, where more manoeuvrability is required. Both ore and waste will be drilled using a Tamrock DP1100. Additionally, four types of support equipment were defined, which will contribute to the performance of the production equipment.

The mining fleet was selected for a capacity of 8,000 tpd ore and waste in year one and a ramp up to 9,500 tpd from year four. The equipment list is provided in Table 63. It was determined that the equipment will be owner operated, with the maintenance done through a MARC contract. The equipment models were selected based on their operational factors and capacities. Another important factor considered was the level of service available in Bulgaria.

Table 63. Mine Equipment List

Mine Equipment List					Unit Cost	Total
Item	**Description**	**Make**	**Model**	**Units**	USD	USD
Drill Rig	Top hammer: 89 – 129 mm	Tamrock	DP1100	1	509,621	509,621
Drill Rig (backup)	Top hammer: 89 – 129 mm	Tamrock	DP1100	1	509,621	509,621
Excavator	3.7 m³ bucket capacity	Komatsu	PC700-8	2	519,250	1,038,500
Excavator (IMWF)	2 m³ capacity class	Komatsu	PC350LC-8	1	306,534	306,534
Rigid Truck	40 tonnes capacity	Komatsu	HD405-7	4	517,963	2,071,850
Articulated Truck	40 tonnes capacity	Komatsu	HM400-3	4	452,500	1,810,000
FEL	6.4 m³ capacity class	Komatsu	WA600-6	1	679,241	679,241
Bulldozer	230 kW class	Komatsu	D155AX-6	2	519,219	1,038,438
Grader	100 kW – 120 kW class	Komatsu	GD555-5	1	277,111	277,111
Water Truck	10 klitre	IVECO	AD 260T36	1	133,165	133,165
IT (tool carrier)	130 kW – 150 kW class	Komatsu	WA250PZ-6	1	162,728	162,728
					Total	**8,536,809**

16.7 Capital Expenditure

Based on the mining equipment list and recommendation of the previous study, vendors were contacted and requested to supply information on their mining equipment and their capability to provide support on site. The RFQ included two parts, the first one requesting pricing and the second one requesting maintenance of equipment under a fully maintained and repaired by the OEM under a maintenance and repair contract (MARC). RFQs were sent to the Komatsu dealer (all equipment less drill rigs) and the Tamrock dealer. Definitive equipment selection for Eastern Europe should also include consideration of existing equipment and supplier support in the region and the skill levels to operate and maintain the equipment. The proposed OEMs have a good presence in Bulgaria.

A 3.7 m3 backhoe configured hydraulic excavator was selected as the primary loading unit as it provides lower costs, better mining selectivity and higher breakout force when compared to the FEL. A fleet of 2 excavators will load into dump trucks which are in turn selected based on this configuration. A haul truck capacity of 40 tonnes was selected on the basis of matching/fit with the excavator, ramp width and gradient design suitability for the pit size. The incorporation of a FEL to the fleet aims at providing a backup for the ROM loader and to be used where a FEL is better suited for operations, such as loading and rehandling topsoil.

Blast-hole drill selection is primarily dictated by the powder factor required for the hardness of the rock, leading to the determination of the drill pattern parameters and the drill and blast bench height. Blasthole drilling will be performed using a top hammer drill capable of drilling 89 mm to 102 mm blast holes. In addition, the drill rig will also provide for pre-split drilling as recommended by the geotechnical review.

The ancillary or load and haul support equipment of dozers, graders and water trucks, have been selected to be a suitable match to the load and haul fleet and to match other duties such as road maintenance and the expected quantity of waste dump. Ancillary surface equipment requirements include two bulldozers, a grader, water truck and small loader (tool carriers).

All dedicated mining equipment will be purchased new and specified to comply with EU requirements for noise and emission levels. The estimated mining capital cost related to the mining equipment corresponds to approximately USD 9.0 million.

16.8 Operating Cost

The mining cost modelling was carried out for the duration of the project as specified in the LOM schedule. The scope of the mining costs includes delivery of ore to the run-of-mine (ROM) pad and waste dumping at the IMWF. The mining cost includes the re-handling cost related to reclaiming low grade material from stockpiles and feeding into to the primary crusher.

The base case assumption is that mining will be carried out as an owner mining operation, given the comparatively low labour rates, mining expertise within the country, requirement for selective mining and commitments to employ local labour where possible. Realistic and achievable performance in terms of productivity, mechanical availability and utilisation has been determined from experience at other similar operations, including data supplied by suppliers.

Dundee operates the Chelopech underground copper mine in central Bulgaria. The following information was supplied from this operation and used to estimate the mine operating costs.

- Fuel, oil and lubricant prices
- Explosives and accessory prices
- Earthmoving tyres prices as applicable
- Labour costs of operating manning

Detailed cost modelling was carried out using a spreadsheet based model developed by Golder. After determining the equipment fleet required for the LOM and estimating the equipment's operating hours, the manpower costs and equipment supplies were estimated. The resulting unit operating cost per item is presented in Table 64.

Table 64. Mining Operating Cost by Item

Mining Cost	USD/t(ore)	USD/t(mat)	% of Total
Labour and Overheads	2.72	0.75	22%
Fuel and Lube	4.59	1.27	38%
Drilling Consumables	0.16	0.04	1%
Blasting	1.06	0.29	9%
Maintenance	2.04	0.56	17%
Grade Control Cost	1.36	0.38	11%
Others	0.18	0.05	1%
Total	12.10	3.34	100%

17 Recovery Methods

17.1 Recovery Methods and Process Design - Introduction

The 2012 mining study project described a mine and process facility with design treatment rates of nominal 0.85 and 1.1 Mtpa. The process flowsheet incorporated a relatively fine (P80 30 microns) primary grind, recovery of the precious metals by flotation, and the tailings being thickened and co- deposited with the mining waste in an integrated mine waste facility. No changes were made to the original concepts, however the base assumptions for each of different components were interrogated to ensure the optimum production through the life of the operation.

The mine plan was optimised to advance the processing of the wall ore (the highest grade material, but also the hardest) as much as practicable (Section 16). An iterative series of plant trade-off studies based on this occurred throughout the design optimisation phase in 2012 between DPM's project consultants, AMEC Engineers, and the owner's team. The outcome formed the basis for the final design (Macromet, Process Design Review, April 2013), and this sought to minimise the equipment requirements while maximising the production profile throughout the life of the operation. All aspects of the project were considered in this process which has enabled the plant equipment and overall infrastructure requirements to be optimised. From the engineering perspective, the requirement to achieve the relatively fine primary grind from the relatively hard ore types present presented several challenges and were the subject of considerable study during the design phase.

17.2 General Design Basis

The Krumovgrad process plant and associated service facilities will take run-of-mine ore from the mine and will produce a gold bearing concentrate for shipment to a smelter. The proposed process encompasses crushing and grinding of the run-of-mine ore, followed by froth flotation to produce a gold bearing concentrate. Tailings will be thickened to a sufficient density to enable deposition in the IMWF.

The proposed process plant design is based on a metallurgical flowsheet with unit operations that are well proven in mineral processing operations worldwide. The key criteria for equipment selection have been the suitability for duty, reliability and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements while maintaining a compact footprint.

The key project and ore specific criteria for the plant design are:

- Treatment of a maximum of 0.85 tpa of ore for each year of operation.
- Operation of the crushing plant on a 12 hours/day basis; mill operations on a 24 hours/day basis. Surge capacity is provided in a 3,000 t capacity Silo located between the circuits.
- Design availability of 91.3% with standby equipment in critical areas.

- Sufficiently automated plant control to minimise the need for operator interface on a continuous basis but allow manual override and control if required

17.2.1 Design Criteria Summary

A summary of the important parameters used as the basis of the important plant unit processes is provided in Table 65. (Ref. Project Design Criteria Document – KGP100-2000-1100-DCS-0001).

Table 65. Krumovgrad Gold Project - Summary Design Parameters

Criteria		Units	General	
Ore Throughput	Maximum Annual	t/y	840,000	
Design Recovery (Range)	Master Composite Basis	Au %	85.0 (83.5 - 88.7)	
		Ag %	70.0 (54.6 - 77.8)	
Primary Grinding	Grind Size, P80	µm	30	
Flotation Circuit - Stages	Rougher/Scavenger	Stages	8	
	First Cl., Cl. Scavenger	Stages	9	
	Second Cleaner	Stages	4	
Concentrate Regrind	Grind Size, P80	µm	15	
Final Concentrate	Design	g/t	200	
	Expected	g/t	650	
			Upper	Wall
Life of Mine (LOM)		t	4,611,315	1,593,000
Plant Availability	Design	%	91.3	
	Nominal Throughput	t/h	105	90
Design Feed Grades	Gold	g/t	3.7	6.7
	Silver	g/t	2.1	3.5
Physical Characteristics	Impact Work Index	kWh/t	12.5	16.1
	Rod Mill Work Index	kWh/t	16.1	22.8
	Ball Mill Work Index	kWh/t	17.0	20.0
	SAG Power Index (Range)	Mins	45 - 96	76 - 149
JKMRC Functions	A and b		49.5 and 1.24	83.5 and 0.35
	Average UCS	Mpa	29	117
	Abrasion Index	g	0.28	0.60
Tailings Thickening	Thickener Flux,	t/m2.h	0.641	
	U/F, Design Solids Density	%w/w	56	
	Maximum Measured	%w/w		68

17.3 Grinding Circuit Development

The original circuit design proceeded along the conventional SAG/Ball Mill grinding circuit flowsheet, with a third stage of grinding incorporating a vertical style ball mill – selected to take advantage of the more efficient energy utilization these types of mill offer compared to corresponding ball mill inefficiencies at the finer end of the size range.

Several trade-off studies evolved reviewing various options for optimising the circuit. Final selection was a single stage SAG mill in closed circuit with cyclones producing a product P80 of 125 microns, followed by the tertiary stage of a vertical style mill. Combining the conventional SAG and ball mill proved to be the most cost effective (both Capital and Operating) circuit. Single stage primary mills are not unusual in this application, however the successful performance of the single stage SAG mill at DPM's Chelopech operation generated a high level of confidence in the approach (Jobson et al, 2012).

Actual plant throughput will be limited by the proportion of the wall component of ore treated in any one year.

17.4 Flotation Circuit Development and Design

The 2012 mining study design incorporated the flowsheet developed from the extensive test program completed at SGS and the subsequent FLEET circuit modelling program. This specified the equipment required to achieve the predicted metal recovery for the two throughput options being considered for the project at that time. The first iteration of the plant design undertaken in 2012 incorporated the mining study conventional tank cell design for the flotation circuit, which was the base case for the first round of capital cost analysis. One of the subsequent trade-off studies included a layout comparison of the tank cells and a new style of flotation cell (the Staged Flotation Reactor - SRF).

In parallel to the Ade Tepe test program, DPM had incorporated a production size unit of into the current Chelopech cleaner circuit in mid-2012. This offers some significant advantages over conventional 'tank' cell designs, including reduced floor area requirement, reduced circuit operating costs (power and air demand), together with some process advantages in most applications (Woodgrove Technologies, Feb 2013).

Because of overall site layout restrictions space in the plant area was at a premium, and the study outcome confirmed some of the advantages claimed, but of particular importance to the project was the potential reduction in floor area (approx. 30% of the original flotation circuit footprint), DPM elected to continue with the SFR approach and these units have been incorporated in the final design.

Chelopech completed the installation of an upgraded Cleaner Circuit (second and third cleaners) in their primary production circuit in mid-2013, and have installed a new recovery circuit for downstream recovery of Pyrite from the cleaner tailings stream. This circuit is very similar in equipment sizing to the Rougher circuit proposed for Krumovgrad.

17.5 Process Description

17.5.1 General Design Basis

The complete plant block flowsheet is illustrated in Figure 47. This reflects the overall facilities, the most important ones of which are described below

17.5.1.1 Crushing

All ore will be reclaimed from the run-of mine pad by a front-end loader (FEL) and fed to the feed bin. This will be fitted with a static grizzly to prevent oversize rocks from entering the bin.

Ore will be drawn from the ROM bin via a variable speed feeder and discharged into a jaw crusher to produce a crushed product P100 of 200 mm and P80 of 125 mm. The primary crusher product will discharge onto a conveyor belt which transfers the product to the Coarse Ore Silo for storage prior to the grinding circuit.

17.5.1.2 Grinding Circuit

Ore will be withdrawn from the silo by one of two apron feeders below, and fed onto the grinding circuit feed conveyor. The grinding circuit comprises of a single stage open circuit SAG mill with pebble crushing in closed circuit with cyclones to produce a product of P80 of 100 - 75 µm. The SAG mill will discharge over a trommel screen, with the oversize (pebbles and steel pieces) discharging onto the first of two conveyors feeding the pebble crusher. The tramp steel will be removed using an overhead belt magnet and the pebbles will feed into a cone crusher to produce a crushed product P100 of 16 mm. The pebble crusher product will discharge back onto the mill feed conveyor belt.

The trommel undersize slurry will gravitate to the mill discharge hopper where it is diluted with water and pumped to a cluster of hydrocyclones for classification. The overflow from the hydrocyclones (ground product) will gravitate through a trash removal screen to the tertiary grinding circuit, while the hydrocyclone underflow will be returned by gravity to the SAG mill for further grinding. The trash screen undersize slurry will gravitate to the combined cyclone feed hopper prior to final classification in hydrocyclones. The underflow will form the feed to the tertiary mills.

The tertiary grinding circuit comprises of two vertical mills in parallel, which have been selected for its more efficient use of energy for fine grinding (compared to a conventional ball mill). The discharge product will be pumped to a cluster of hydrocyclones for classification. The overflow from the hydrocyclones (the final grinding circuit product) will gravitate as feed to the flotation circuit, at the required grind size P80 of 30 microns.

17.5.1.3 Flotation Circuit

Concentrate from the first stage of the Rougher/Scavenging flotation (4 units for each stage) will be sent for regrind in the tertiary grinding circuit (to a P80 of 15 µm), and then cleaned in the first cleaner (2 units) to produce final concentrate. Tails from the Rougher circuit will be ‘scavenged’ and the concentrate recycled back to the Rougher feed. Tails from the first cleaner are retreated in the cleaner/scavenger bank (5 units), the tails from which return to the feed to the rougher. The Scavenger cleaner concentrate will be recleaned in the second cleaner (4 units) producing final concentrate, with the tails being recycled back to the feed of the cleaner/scavenger bank.

17.5.1.4 Concentrate Handling

Final concentrate will be thickened, and then dewatered in a pressure filter with the product being stored in bags prior to shipment from site in sealed containers.

17.5.1.5 Tailings

The discharge from the Scavenger Flotation bank will be pumped to the head of the IMWF, where it will be dewatered in a deep cone thickener to produce an underflow with a minimum of 56% w/w solids. The thickened underflow slurry is conveyed by gravity pipeline and combined with mine waste and placed in the disposal area of the IMWF. The thickener overflow water is returned to the reclaim water tank back at in the mill circuit area.

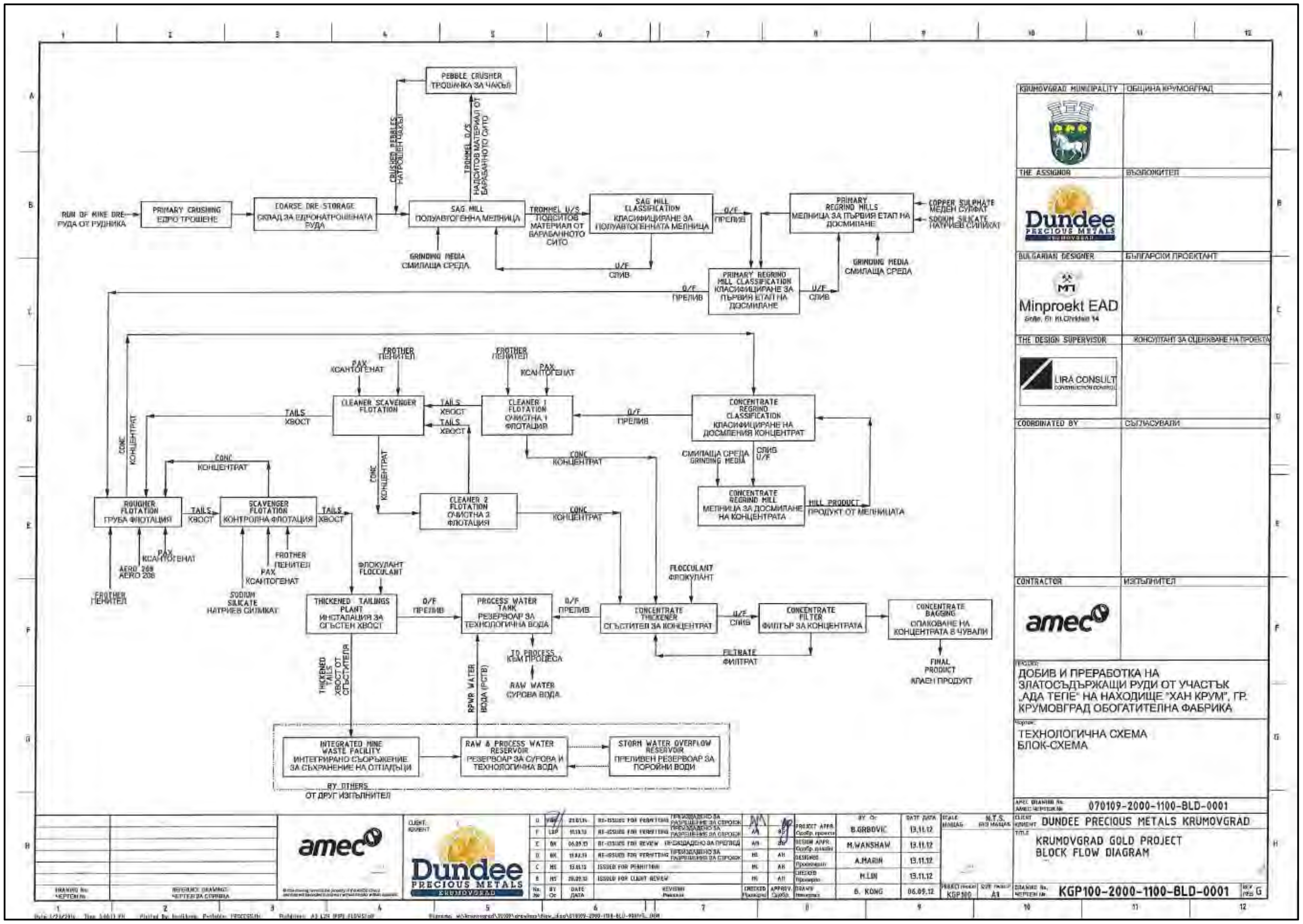

Figure 47: Overall Plant Flowsheet

18 Project Infrastructure

18.1 Integrated Mine Waste Facility

18.1.1 Background and Site Selection

The concept of a conventional slurry disposal facility as proposed in the 2005 mining study has been replaced with an IMWF which will receive both the thickened tailings and the mine waste rock from the Ada Tepe pit. The tailings storage location was revised to minimise land use and the environmental footprint. Two sites were initially identified for a potential IMWF, located north and south of the open pit respectively. Preliminary capacity assessments as well as optimisation of the mine and road layout resulted in selection of the south site.

18.1.2 General Description

The concept of the IMWF is to place thickened tailings into cells constructed from mine rock. The mine rock provides strength required for overall stability and also internal drainage. Water reporting to the underdrain will be pumped to the Raw and Process Water Reservoir ("RPWR") located southwest of the open pit. The IMWF will be constructed within two small valleys, being operated as two separate facilities early in the life of the project and later merging into a single facility as operations progress. Rehabilitation of the lower slopes of the IMWF will begin during the early stages of mine operation.

The IMWF structures required for commencement of mining operations will be constructed from the soil and rock excavated to create the platform for the process plant and the roads on the mine site. Once the mining operation begins, the mine rock will be trucked from the open-pit to the IMWF, dumped and spread to construct containment cells for the tailings. Tailings will be thickened in the tailings thickening plant to the maximum practical amount, and then conveyed by pump and pipeline to the containment cells. The IMWF will be a fully drained facility and will not contain a water pond at any time during its operation. A system of under-drains will be constructed along the axis of each small surface water channel in the footprint of the IMWF and these drains will discharge to one of two sumps located at the toe of the facility.

The IMWF will be constructed from the bottom up, with mine wastes placed on starting platforms at the bottom of the valley at approximately 300 m elevation and then progressively built up in benches during the mine life to elevation 450 m. This will allow the lower, completed sections of the facility to be reclaimed and closed during the life of the mining operation.

Given the economic parameters used for this study, (i.e. 0.6 g/t COG), 15.1 million tonnes of mine rock and 6.2 millions tonnes of tailings will be stored within the IMWF over 8 years during the life of the mine.

A dual reservoir system has been developed which has resulted in the mine being able to adopt a zero discharge water management strategy. The two reservoirs are the RPWR and the Storm Water Overflow Reservoir (SWOR). These two reservoirs are adjacent to each other and have differing functions with regard to water management, these being management of process water and storage of storm water and pit inflows.

18.1.3 Closure and Rehabilitation

Closure of the Krumovgrad IMWF will involve conventional practice for mine rock facilities.

Drainage into the IMWF is collected in an under-drain system that prevents the build-up of a water table within the rock and tailings. Water draining from or through the IMWF will exit at the toe of the ravines. During operations, water reporting to the sumps at the toe of the ravines will be pumped to the RPWR prior to clarification for use for mill make-up. Following operations, the sediment load reporting to the sumps will be monitored. When the sediment load is of an acceptable level, drainage will be allowed to enter the river directly.

The IMWF will be constructed from the bottom up with horizontal benches at 10 m vertical intervals with the intervening slope constructed at 2.5H: 1V. During operations, the external faces of the completed portions of the IMWF can be covered with topsoil and vegetated. This means that the majority of the IMWF can be rehabilitated prior to the end of the mining operations.

18.2 Water Management

The project water management plan is central to maintaining an appropriate environmental and operational performance for the project. The principle adopted for site water management is to intercept and divert away water flowing towards operational areas and intercept water in contact with operational areas. This contact water may then be used in the Project or discharged in line with discharge consents. In operation, the process plant will source its water mainly from recycle of decant water. Make-up water will be taken from a proposed borehole well to be located approximately 0.3 km southwest of the process plant near the Krumovitsa River.

The project water management plan has been developed to ensure minimum impact on the surrounding community users. To manage the water balance across the site a dual water storage system will be used. This system is composed of a Raw and Process Water Reservoir (RPWR) and a Storm Water Overflow Reservoir (SWOR) which will be used to collect and temporarily store water. The dual reservoirs have a capacity of 130,450 m3.

All surface water within the processing facilities area will be collected in a channel that diverts the water into the IMWF north sump, and this water will then be pumped back to the RPWR. The groundwater and surface water reporting to the open pit will be collected in a sump and pumped directly to the SWOR. Rainfall that infiltrates into the IMWF and the water expelled from the tailings during consolidation will be directed to an underdrain system. These drains will discharge to one of the two sumps located at the toe of the facility. The collected water will be pumped to the RPWR. The IMWF will be a fully drained facility and will not contain a water pond.

18.2.1 Water Supply

The water balance is positive on an annual basis however on a 100 dry year there could be a shortage. For that reason a backup source of freshwater has been allowed for in the form of a borehole well located southwest of the process plant near the Krumovitsa River.

18.3 Communications

The mine site will be linked to the public network in town of Krumovgrad using fibre optic cable which will support both data and voice communications. A repeater system will provide the infrastructure to enable hand-held and mobile radio sets to communicate around the site.

18.4 Access Road and In-plant Construction

The proposed access road to the plant site is an existing secondary paved road approximately 2 km in length which runs from Zvanarka through Pobedam, two small villages located near by the site. This secondary road connects with the main road leading to the town of Krumovgrad. In anticipation of increased traffic on the section of the secondary road between Zvanarka and Pobeda, 7 pull-out areas will be incorporated into the existing road to facilitate vehicle passing. The road will be upgraded to accommodate heavy vehicles. The second portion of the access road from the paved road to the plant will follow an existing dirt road for approximately 950 m. This road will be widened and paved to minimize dust emission.

On site a 950 m long road will connect to an exit of the mine open-pit and provide access to the crusher area. The main section of this road will be comprised of a 20 m wide running surface with 2 m high by 3 m wide berms located on the down slope side of the road. It will be surfaced with gravel maintained by frequent grading and water sprinkling.

The IMWF embankment access roads will provide access from the open pit to two embankment dams. One road will be 1.9 km length and will connect the open pit with the north embankment construction site. A second road 760 m long will connect to the south embankment construction. It will be surfaced with gravel maintained by frequent grading and sprinkled as necessary.

18.5 Effluent

Sewage from the various plant site buildings will be dealt with by means of a packaged Tertiary Wastewater Treatment System. Waste such as hydrocarbons from equipment maintenance and chemical waste from the laboratory will be collected and stored for collection by contractors who will remove from site and dispose of in accordance with the applicable regulations. Office waste and waste from the meals areas will be collected by a cleaning contractor who will dispose of the waste materials in a solid Municipal landfill site.

18.6 Fuel Storage and Distribution

Diesel fuel storage will be provided to supply fuel to process equipment, light vehicles, the mining fleet and mobile plant and equipment. All fuel required at the plant site will be delivered in tanker trucks by commercial suppliers. The fuel storage area will be bunded to prevent spillage of fuel contaminating the site area or watercourses. Minor quantities of petrol that may be required can be obtained from local fuel distributors.

18.7 Vehicle Washdown Facilities

A vehicle washdown facility will be provided adjacent to the diesel fuel refuelling area. It will comprise a bunded concrete slab sloping to a settling sump. Captured rainfall and diesel spillage from the adjacent diesel refuelling facility will also be directed to this sump. A sump pump will transfer dirty water to an oil/water separator.

18.8 Power Supply and Reticulation

The plant electrical power will be supplied by the local power authority via a proposed underground high voltage cable supplied from the Krumovgrad 110 kV / 20 kV Substation. A 20 kV main substation will be established at the plant site to facilitate power distribution to various areas within the plant. Within the main substation, a tariff metering system will be established to allow for reading of whole of plant power consumption.

18.9 Buildings

Infrastructure buildings are classified as either architectural, control rooms or industrial. Architectural buildings include administration offices and ablution facilities. Control rooms include the crusher control room and the main process plant control room. Industrial buildings include workshops, warehouses and buildings that house process equipment.

The assessment of building requirements has been based on the number of personnel required in each area and the functions required in each particular area. These buildings will be constructed of reinforced block-work or brick. Roofing will be corrugated steel and floors of elevated timber or on ground concrete. Local construction materials will be used to the maximum extent possible.

18.10 Fire Protection

Fire protection will consist of the provision of fire hydrants, fire hose reel cabinets and fire extinguishers placed strategically around the facilities in accordance with the requirements of the relevant regulations. Firefighting water will be supplied from a dedicated volume in the fresh water reservoir. Water is gravity fed to firewater pumps at the process plant. Jockey, duty and diesel-powered standby pumps will be provided.

Various types of fire extinguishers will be provided in areas where water as a means of fire control is undesirable. These include MCCs and control rooms.

18.11 Security

All persons entering the Process Plant and mine facilities areas will be required to pass through the continuously manned boom gate adjacent to the administration building on the access road. Security guards located within the administration building will control all entry and exit of vehicles and personnel. Search and inspection of personnel, bags and items leaving the plant will be carried out at this facility.

A stock fence will be constructed around the all project facilities including the process plant, Integrated Mine Waste Facility, mine, and raw and process water reservoir. Security fencing with lockable access gates will be installed locally around the remote pumping facilities and the explosives magazine.

Additional security fencing will be provided around the warehouse yard. All security fencing around the key areas will be 2.4 m high wire chain mesh cyclone type fencing with 4 strand barbed wire.

18.12 Operations Administration

18.12.1 Recruitment of Personnel

Personnel will be recruited locally to fill the majority of the available positions. Given the requisite skills, or the ability to quickly acquire such skills, Bulgarian nationals living in the Krumovgrad district would be ideally placed to fill the available positions. Key personnel will be employed sufficiently early in the implementation schedule to achieve an effective involvement in the development of operating and training programs/procedures.

Expatriate labour will be sourced to fill those positions requiring experience in gold plant and related operations. Expatriate positions will account for a very small, but essential, proportion of the total personnel employed.

To ensure efficient continuous production from operations after commissioning, personnel numbers in the first years of operation will be higher than subsequent years. Over this period personnel numbers will gradually be reduced to a level that will sustain future operations at the designed plant capacity.

18.12.2 Manning

Manning numbers are presented in Table 66 below. These numbers include the contract laboratory workers in the Process Plant operations. In addition to these numbers, additional people will be employed at site by other contractors, including the mining equipment contract maintenance workers.

Table 66. Krumovgrad Gold Project Annual Manning Schedule

Krumovgrad Gold Project Annual Manning Schedule	
Department	**Employees**
Administration	40
Process Plant + Maintenance	116
IMWF	10

Krumovgrad Gold Project Annual Manning Schedule	
Department	**Employees**
Mine + Maintenance	87
Total	253

19 Market Studies and Contracts

The process flowsheet for the current proposal has been thoroughly tested and a robust circuit has been developed for the project. At the time of writing, sufficient quantities of concentrate were produced to confirm the predicted quality of the concentrate, however insufficient volumes were produced for any significant marketing studies to be completed.

Dundee has considerable experience in producing, shipping and negotiating with metallurgical facilities and their owners worldwide. Preliminary analysis and discussions with four purchasers of such concentrates have indicated that the proposed quality will be readily saleable in at least several locations in Europe, as well as Canada and China. Dundee’s wholly owned facility in Namibia (DPMTsumeb) will also be considered for placement of the product, should it be required in the future. Indicative terms are typical of current market conditions, with payables of 95% for the gold grades that will be produced and typical treatment charges of USD150/wmt.

All of the potential purchasers have stated they will wait for production quantities on which to discuss final terms, and all expressed their interest in being involved in such discussions at the appropriate time.

20 Environmental Studies, Permitting and Social/Community Impact

Under Bulgarian environmental regulations, the mining projects are required to comply with an EIA process as a key part of project permitting. The content and depth of the EIA is compliant with the Bulgarian Environmental Protection Act. The Bulgarian environmental legislation is fully harmonized with EU one.

The EIA systematically assesses project impacts in relation to the physical, biological and human environmental components, taking account of activities that will take place during the construction, operation and closure phases. Consideration was also given to alternative options for technology (mining, processing and waste management) and to the location of facilities (process plant, IMWF and RPWR).

The EIA report comprise of two major appendices. The first one is the Assessment on the Compatibility of Conservation Objectives of the Protected Zone Eastern Rhodope and Protected Zone Krumovitza with the Investment Proposal. This assessment has been prepared pursuant to the Bulgarian Law on Biodiversity, and the Regulation on Requirements for Conducting a Compatibility Assessment (“CA”) between Plans, Programs, Projects, as well as Investment Proposals and the Conservation Objectives of Protected Zones (“PZ”). Assessment of compatibility of the investment proposal with the object and purpose of protected areas is done according to the requirements of the European ecological network - NATURA 2000.

The second appendix of the EIA is The Mining Waste Management Plan. This was developed in connection with the URA, SG 23/12.03.1999, last amendment and elaboration in SG 70/8.08.2008 and the Regulation on the Specific Requirements to Mining Waste Management, SG 10/6.02.2009.

The Bulgarian Minister of Environment and Waters has signed a resolution № 18-8, 11/2011 approving the EIA for the Company's Krumovgrad Gold Project in Bulgaria. The resolution was appealed by environmental NGOs. The final decision on EIA resolution by the 5- member panel of the Supreme Administrative Court (SAC) was issued on March, 4th 2013 in favour of the Company. This decision is final and EIA Resolution is not subject to any further appeals.

The European Commission has published in 2014 a report and a case study on the permitting procedure and compatibility of activities with the Natura 2000 requirements, aiming at improving these processes in Europe. The Compatibility Assessment of the Krumovgrad Project was published as one of 12 cases for best practices on the continent.

20.1 Community Impacts, Including Air Quality

The setting of the Project is typically rural, the site being located 3 km from a small town with no significant existing sources of industrial emissions that could affect people's health and enjoyment of the environment. The Project will be the potential source of emissions that are important in regard to community health and amenity, including gases, fumes, dust, noise and blasting vibration. The nature of these emissions is well understood and design of the Project includes specific mitigation against non-fugitive and fugitive emissions, such as dust and noise. Provided that this mitigation strategy is appropriately adopted, no significant impact is forecast for the local community regards their health or enjoyment. It is however likely that from time to time, a local nuisance impact will be experienced by adjacent communities, particularly in regard to dust during dry, windy weather. Such impacts may be managed by good operational practices coupled with an effective community liaison scheme.

Site traffic during construction and operation will lead to a significant increase in trucks on local public roads. Attention will be placed on vehicle routing and site access to avoid undue increase in traffic accidents.

The Project will have no significant impact on community health. Health of the workforce will be protected by industry-standard measures for avoidance of exposure to harmful emissions and (where that is not possible) wearing of personal protective equipment.

20.2 Surface Water

The region is located within a climatic belt that experiences both Mediterranean and Southern Bulgarian (Central European) climatic influences with most rain (and occasional snow) in the winter period. Rainfall tends to be experienced as marked events and surface water flows therefore show great variation. The Project site is drained by the Krumovitsa River which flows at the foot of Ada Tepe, past the town of Krumovgrad. This river is part of a system that crosses into Greece about 50 km downstream from the site. Water quality is relatively good with lack of industrial pollution and the town of Krumovgrad sources its supply from alluvial gravels below the river bed. This abstraction points are protected by a sanitary protection zones.

The Project presents a potential source of contamination of surface water and disturbance of surface water flows. The design of the Project mitigates against surface water impact using the following strategies:

- Thickening of the tailings before discharge to the IMWF and recycling of the supernatant water, which will reduce evaporation losses (compared to deposition of tailings in a TMF);
- Recycling of the mine and IMWF drainage waters back into the process;
- Adoption of a project water supply scheme that maximizes use of recycle water and draws fresh water from site area inflow;
- Maximising recycling and minimizing environmental discharge towards “zero discharge”;
- Proposed adoption of an environmental management plan that includes procedures for spill avoidance, containment and treatment as well as various housekeeping measures that are regarded as international best practice;

- Progressive closure and rehabilitation of IMWF which will protect erosion of the slopes and will reduce suspended solids in rain water.

It is forecast that the proposed mitigation will enable the Project to be constructed, operated and closed without significant impact on surface water flows and quality, i.e., other users of the surface water (including communities, businesses and wildlife) will not be significantly affected. This forecast is based on assessment under normal and extreme (flood) events. The proposed closure strategy will include re-instatement of a permanent surface drainage system, including closure of the Integrated Mine Waste facility and a pit lake in the final mine void.

20.3 Groundwater

The Project site and adjacent land is underlain by Palaeogene rocks that contain no significant aquifers. Local communities draw groundwater from shallow wells and as noted above, the town of Krumovgrad sources its water from the Krumovitsa Valley alluvial aquifer about 3km from Ada Tepe. Mining at Ada Tepe and abstraction of water for the Project will have no significant impact on groundwater resources. The proposed mitigation measures, especially in regard to water management generally and management of wastes, in particular, are forecast to prevent groundwater contamination in the short and long term (post-closure).

The Company obtained Permit № 31530328/04.03.2013 for abstraction of groundwater resources through construction of new water abstraction facility – one tube-and-shaft well with drainage branch, located on land plot No. 000281 on the land of Skalak village, Krumovgrad Municipality, Kardzhali District. The permitted annual quantity is 70 000m3. The permit is in force and valid for 10 years.

20.4 Soils and Land Use

Soils are in general low in fertility and very shallow over the hilly areas. The chemistry of the soils appears to reflect underlying geology and mineralisation and there are indications of certain areas having naturally elevated levels of heavy metals, including arsenic. These levels are not significant having regard to potential for pollution of surrounding land (e.g., by dust blow during soil stripping operations), but existing soils geochemistry is an important consideration for re-use of soils recovered from the stripping of operational areas.

Land take for construction of the Project will have an insignificant impact on land use, particularly over the road for access to the mine site. The operational areas are mainly on state forest land. Rehabilitation will restore a forest land, mainly of value as nature conservation and recreational land.

20.5 Wildlife

The project site is located within a region well documented as containing a very diverse and interesting terrestrial ecology. Various sites and areas have designated protected status, but none of these lie within proximity of the Project site and all fall outside a nominal area of influence.

The entire project area lies within the footprint of Natura 2000 protected site known as BG 0001032 Rhodopes East under Council Directive 92/43 on the Conservation of Natural Habitats of Wild Fauna and Flora. BG 0002012 Krumovitsa, which is a protected site under Council Directive 79/409/EEC on the Conservation of Wild Birds, is in close proximity to the project area.

The CA performed for all habitats and species were prepared. The results from this CA allow for a conclusion to be made - that the project is compatible with the goals of protection of protected sites, as the degree of impact onto environmental components is insignificant.

20.6 Cultural Heritage

Archaeology investigation, walkover survey, and literature research has indicated the presence of numerous sites of archaeological interest in the vicinity, some lying within the project boundary but no archaeological objects of expositional or architectural value have been found so far. A few of these sites will be directly affected by land take for the project. Consultation indicates that survey, recording and removal for preservation of any artefacts found will be appropriate and acceptable. It is possible that other sites will be discovered during construction and operation, and these will be appropriately dealt with in full consultation with the authorities and their officers.

20.7 Social Impacts

Krumovgrad Municipality has unfavourable demographic and social resource characteristics as compared to national statistics. These may be summarized as follows:

- Poor access to health care;
- Limited educational resources;
- Limited employment and skills training opportunity;
- High levels of unemployment.

The inward investment and significantly increased employment opportunity afforded by the Project comprises, at minimum, a local and regionally significant positive impact. Mitigation against negative "boom and bust" impacts (such as pressure on housing and education resources in the boom and unemployment in the bust) is provided for in a preliminary social management plan that incorporates such important features as provision for community resource development; appropriate recruitment and training programmes; and inward investment providing funds for improved health and education.

20.8 Closure Plan and Rehabilitation

Closure of the Krumovgrad IMWF will involve conventional practice for mine rock facilities.

Drainage into the IMWF is collected in an underdrain system that prevents the buildup of a water table within the rock and tailings. Water draining from or through the IMWF will exit at the toe of the ravines. During operations, water reporting to the sumps at the toe of the ravines will be pumped to the RPWR prior to clarification for use for mill make-up or release to the environment. Following operations, the sediment load reporting to the sumps will be monitored. When the sediment load is of an acceptable level, drainage will be allowed to enter the river directly.

The IMWF will be constructed from the bottom up with horizontal benches at 10-m vertical intervals with the intervening slope constructed at 2.5H: 1V. During operations, the external faces of the completed portions of the IMWF can be covered with topsoil and vegetated. This means that the majority of the IMWF can be covered and reclaimed prior to the end of the mining operations.

20.9 Recommendations

The EIA study carried out indicates that the Project may be constructed, operated and closed in an environmentally acceptable manner. The implementation of the project was approved by environmental authority under execution of specific conditions described in the EIA Decision. The requested measures are connected with discharged water quality, level of dust and noise emission, mine waste management and protection of wild life.

21 Capital and Operating Costs

21.1 Capital Costs

The capital cost estimate for the project is estimated at USD 164.1 million. A summary of the capital cost estimate is included in Table 67.

The total capital cost estimate for the project includes the design, construction and/or procurement of the following items:

- Open Pit Mine and Haul Roads
- Plant Access Roads
- Overhead Power Supply from Substation located near the City of Krumovgrad
- Mining Equipment
- Gold/Silver Ore Processing Facility to treat 0.85Mt/a of ore throughput
- Deep Cone (Tailings) Thickener Facility
- Integrated Mine Waste Facility (IMWF)
- Run-off Water Storage Facility
- Ancillary Buildings
- Engineering, Procurement, Construction Management, Permitting and Commissioning and Start-up
- Working Capital
- Owner's Costs

Table 67: Capital Cost Summary

Cost Element	850 kt/a Estimated Cost (USDx000)
Crushing	7.7
Grinding	18.0
Flotation	12.5
Concentrate Handling	3.4
IMWF	24.8
IMWF Haul Roads	8.6
Return , Process and Storm Water Reservoirs	3.3
Water Supply	4.0
Plant Air System and Reagent Mixing	1.6
Buildings + Fuel Depot	10.0
Electrical and Instrumentation	3.4
Mobile and Mining Equipment	12.6
Land Acquisition	4.6
Commissioning and Temporary facilities	9.8
Subtotal Direct Costs	124.3
EPCM	9.2
Owner's Costs	15.1
Contingency (12.5%)	15.5
Total Capital Cost	164.1

Capital cost estimates were provided as follows:

- Amec provided all capital estimates for the ore processing facilities, power and communications for the overall site, ancillary buildings, other infrastructure items, details and quantities for the main access road, supplementary site roads and diversion trenches, sustaining capital, and working capital.
- Golder Associates provided capital costs for mining equipment, mine pre-production stripping, preparation earthworks for IMWF, haul roads and the site runoff water storage pond.
- Golder PasteTec provided capital estimates for the IWMF tailings distribution piping, valve and pumping systems, and the deep cone thickener facility.
- Owner's costs have been provided by DPMKr, supported by experience from the recent expansion of its Chelopech Operation in Bulgaria.
- Amec consolidated all capital cost information and determined the appropriate accuracy percentages for the total Capital Cost Estimate.

21.2 Operating Cost Estimate

The operating cost estimate is presented in United States dollars (USD) and uses prices obtained in the fourth quarter of 2013. This estimate includes all site related operating costs associated with mining and processing ore to produce a gold/silver concentrate that will be shipped to an outside smelter/refinery for final treatment.

The operating cost estimate excludes concentrate shipping, insurance and smelting/refining costs, escalation, corporate overhead charges, financing costs, royalties, income taxes or similar taxes and fees as well as expenditures classified as capital, sustaining capital, rehabilitation and closure costs or any Owner's Costs. Some of these elements are included in the financial analysis.

Operating cost estimates were prepared by a number of consultants for the various components of the Project. Major contributions to the estimate were made by:

- DPMKr provided information relating to the present operation at Chelopech, and gave advice in relation to local suppliers, labour rates, labour loadings, manning schedules and security needs
- Amec provided the operating cost components associated with the Process Plant.
- Golder Associates provided the operating cost components associated with the Open Pit Mine.
- Golder PasteTec provided the operating cost components associated with the Tailings Treatment Facility and the Integrated Mine Waste Facility (IMWF).
- Amec combined these inputs into an overall operating cost estimate for the project.
- Table 68 summarizes the estimated annual Krumovgrad Project operating costs for the open pit mining and the ore processing at a treatment rate of 850ktpa over the life of the mine. This table also summarizes the Tailings Treatment and General and Administrative (G&A) costs for the life of the mine.

Table 68. Operating Cost Summary - 850 ktpa scenario

Item	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
Total Tonnes Processed	809 023	803 026	812 586	836 608	773 311	785 464	788 800	595 276	6 204 095
Au Contained	95 202	162 889	139 372	70 854	148 883	78 326	68 685	42 317	806 528
Au Recovered	75 613	110 015	124 060	77 760	127 964	70 694	60 902	38 541	685 549
Ag Contained	56 797	85 893	72 595	41 662	76 666	43 110	40 284	25 728	442 735
Ag Recovered	34 997	46 951	51 886	36 043	54 955	34 067	30 581	20 434	309 915
Au Equivalent Contained	96 338	164 607	140 824	71 687	150 417	79 188	69 491	42 832	815 382
Oz Au Equivalent Recovered	76 313	110 954	125 097	78 481	129 064	71 375	61 514	38 950	691 747
Mining Costs	9 400	8 036	8 241	9 711	11 190	9 659	10 282	8 551	75 068
Processing Costs	17 671	17 213	18 133	17 314	17 127	17 574	17 147	13 160	135 339
Tailings Treatment & IMWF Costs	2 184	1 293	1 951	2 467	2 496	2 664	1 883	587	15 525
General & Administration	2 927	2 917	2 917	2 917	2 902	2 839	2 809	2 809	23 036
Total Bare Operating Cost	32 181	29 458	31 242	32 409	33 715	32 736	32 121	25 107	248 969
Royalty	1 732	4 336	6 714	3 497	3 514	3 795	1 250	0	24 838
Total Estimated Operating Cost	33 914	33 794	37 956	35 906	37 229	36 531	33 370	25 107	273 807
Operating Cost Per tonne Processed	$41.92	$42.08	$46.71	$42.92	$48.14	$46.51	$42.31	$42.18	$44.13

22 Economic Analysis

22.1 Assumptions

In calculating the returns from the Project, the following fundamental assumptions have been made:

- Metal prices of USD 1,250/oz for gold and USD 23/oz for silver will be maintained throughout the life of the Project.
- Metal price and currency hedging is excluded.
- Gold and silver recoveries of 85% and 70%, respectively, based on average testwork performance on all ore types.
- All production will be sold in the period in which it is produced.
- The life of the Project will be 8 years from commencement of operation, including the processing of stockpiled low grade ore at the end of the project.

Evaluation has encompassed a 125 week development and construction period prior to commencement of ore feed to the mill, through to the end of ore processing.

22.2 Currency, Escalation and Exchange Rates

Analysis has been conducted in United States dollars (USD) rather than Bulgarian Lev (BGN), since it is considered to be the standard currency for project evaluation by the owner of the project. Base Exchange rates used for the evaluation of the Project are USD 1.25/EUR and BGN 1.95583/EUR. Effects of significant shifts in these exchange rates are considered as part of sensitivity analysis. The analysis has been conducted excluding escalation of both metal prices and capital and operating costs.

22.3 Taxation

The financial analysis has been conducted after tax.

22.4 Funding

Financial analysis is based on the Project being wholly equity funded.

22.5 Summary of Results

The key project statistics for the life of the Project are summarised for both scenarios modelled (850 ktpa) in the tables following, the content of which is:

- Table 69 Project Results Summary
- Table 70 Production and Revenue Summary
- Table 71 Operating and Capital Cost Summary

Table 69. Project Results Summary

Project Results		
Item	**Unit**	**LOM**
Throughput	Mt/a	0.85
Project Life	Years	8
Gold Price	USD/oz	1,250
Silver Price	USD/oz	23
Project Economics		
NPV at a discount rate of 7.5%	USD M	143.9
Internal Rate of Return	%	26.3
Payback Period	years	2.5

Table 70. Production and Revenue Summary

Production and Revenue		
Item	**Unit**	**LOM**
Throughput	Mt/a	0.85
Wall zone		
Total quantity of ore mined/milled	t	1,592,780
Gold grade	g/t	6.74
Silver grade	g/t	3.46
Upper zone		
Total quantity of ore mined/milled	t	4,611,315
Gold grade	g/t	3.11
Silver grade	g/t	1.79
Total ore		
Total quantity of ore mined/milled	t	6,204,095
Gold grade	g/t	4.04
Silver grade	g/t	2.22
Waste mined	t	16,241,716
Metallurgical recoveries		
Gold recovery	%	85
Silver recovery	%	70
Metal content		
Gold in concentrate	ozs	685,549
Silver in concentrate	ozs	309,915
Total Net Revenue	USD M	792.8
Site EBITDA *	USD M	519.02
* Before community support		

Table 71. Operating and Capital Cost Summary

Item	Unit	LOM
Throughput	Mt/a	0.85
Total cash cost per tonne of ore processed	USD/t	44.13
mining cash cost per tonne of ore	USD/t	12.10
processing cash cost per tonne of ore processed	USD/t	21.81
tailings treatment & IMWF	USD/t	2.50
royalty per tonne of ore processed	USD/t	4.01
admin cash cost per tonne processed	USD/t	3.71
Total Cost of production:	USD M	273.81
Mining	USD M	75.1
Processing	USD M	135.3
Tailings treatment	USD M	15.5
Royalty	USD M	24.8
Administrative costs	USD M	23.0
Capital Costs		
Owner's cost	USD M	15.1
EPCM	USD M	9.2
Process plant and buildings	USD M	73.7
Mobile and Mining Equipment	USD M	12.6
Land Acquisition	USD M	4.6
Integrated waste management facility and Haulroads	USD M	33.4
Contingency (12.5%)	USD M	15.5
Total	USD M	164.1
Sustaining capital	USD M	12.5
Closure and rehabilitation costs	USD M	14.7

22.6 Project Economics

As detailed in Table 71 based on treatment of 850 ktpa from the Ada Tepe open pit, to produce gold/silver concentrate, and on the basis that the project is wholly equity funded, the Project is anticipated to yield, the following:

- After Tax IRR: 26.3%;
- NPV: USD 143.9 million at a discount rate of 7.5%;
- Initial capital payback: 2.5 years after commencement of production.

At average grades of 4.04 grams gold per tonne of ore and 2.22 grams of silver per tonne of ore, and with average metallurgical recoveries of 85% for gold and 70% for silver, the 6.2 Mt of ore processed for the project is calculated to produce concentrate containing 685,549 oz of gold and 309,915 oz of silver.

At constant metal prices over the life of the project of USD 1,250/oz gold and USD 23/oz silver, and allowing for transport, treatment and refining of concentrate, it is anticipated that USD 792.83 million will be realised in net revenue over the life of the Project, for an initial capital outlay of USD 164.1 million. Cash Operating Costs are anticipated to average USD 44.13/t of ore treated over the life of the project. USD 24.8 million will be paid to the Government of Bulgaria in the form of Concession fees/royalties.

Over the LOM on-site cash operating costs are projected to be USD 249 Million, at an average of USD 40.13 per tonne processed.

DPMK will pay a royalty to the Bulgarian government, which is expected to be 2.57% of the gross value of ore produced from the mine, totalling USD 24.8 Million.

Table 72 summarises the project cash flow forecast for the life of mine.

Table 72. Annual Cash Flow Forecast

Annual Cash Flow Forecast													
	Unit	Year -3	Year-2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total / Avg
Total ore	t				1,070,601	1,133,305	877,977	744,774	698,340	566,205	654,444	458,450	6,204,095
Au Grade	g/t				2.77	4.47	4.94	2.96	6.63	4.30	3.26	2.87	4.04
Ag Grade	g/t				1.65	2.36	2.57	1.74	3.41	2.37	1.91	1.75	2.22
Waste	t				2,240,899	1,014,695	1,156,023	2,466,226	2,512,660	2,644,794	2,441,248	1,765,172	16,241,716
Mined rock material	t				3,311,500	2,148,000	2,034,000	3,210,999	3,210,999	3,210,999	3,095,691	2,223,621	22,445,811
Strip Ratio	w:o				2.1:1	0.9:1	1.3:1	3.3:1	3.6:1	4.7:1	3.7:1	3.9:1	2.6:1
Milled Ore	t				809,023	803,026	812,586	836,608	773,311	785,464	788,800	595,276	6,204,095
Au in Feed	g/t				3.42	5.01	5.59	3.40	6.06	3.29	2.83	2.37	4.04
Ag in Feed	g/t				1.92	2.60	2.84	1.91	3.16	1.93	1.72	1.53	2.22
Gold-Silver Conc Produced (dmt)	t				3,618	5,264	5,936	3,721	6,123	3,383	2,914	1,844	32,805
Au recovery	%				85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%
Ag recovery	%				70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%
Au Recovered	oz				75,613	110,015	124,060	77,760	127,964	70,694	60,902	38,541	685,549
Ag Recovered	oz				34,997	46,951	51,886	36,043	54,955	34,067	30,581	20,434	309,915
Net Revenue	US$M				$87.5	$127.2	$143.4	$89.9	$147.9	$81.8	$70.5	$44.6	$793
Mining Costs	USD$/t mined				$2.84	$3.74	$4.05	$3.02	$3.48	$3.01	$3.32	$3.85	$3.34
Mining Costs	USD$/t ore				$11.62	$10.01	$10.14	$11.61	$14.47	$12.30	$13.03	$14.37	$12.10
Processing Costs	USD$/t ore				$21.84	$21.43	$22.32	$20.70	$22.15	$22.37	$21.74	$22.11	$21.81
Tailings Costs	USD$/t ore				$2.70	$1.61	$2.40	$2.95	$3.23	$3.39	$2.39	$0.99	$2.50
Royalty	USD$/t ore				$2.14	$5.40	$8.26	$4.18	$4.54	$4.83	$1.58	-	$4.00
Cash Cost	USD$/t ore				$41.92	$42.08	$46.71	$42.92	$48.14	$46.51	$42.31	$42.18	$44.13
Cash Cost (incl. royalty) - by-product	USD$/oz				$438	$297	$296	$451	$281	$506	$536	$639	$389

Annual Cash Flow Forecast													
	Unit	Year -3	Year-2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total / Avg
Cash Cost (incl. royalty) - co-product	USD$/oz				$445	$305	$304	$458	$289	$512	$543	$645	$396
Mining Costs	USD$M				$9.4	$8.0	$8.2	$9.7	$11.2	$9.7	$10.3	$8.6	$75.1
Processing Costs	USD$M				$17.7	$17.2	$18.1	$17.3	$17.1	$17.6	$17.1	$13.2	$135.3
Tailings Costs	USD$M				$2.2	$1.3	$2.0	$2.5	$2.5	$2.7	$1.9	$0.6	$15.5
G&A Costs	USD$M				$2.9	$2.9	$2.9	$2.9	$2.9	$2.8	$2.8	$2.8	$23.0
Royalty	USD$M				$1.7	$4.3	$6.7	$3.5	$3.5	$3.8	$1.2	-	$24.8
Cash Operating Costs	USD$M				$33.9	$33.8	$38.0	$35.9	$37.2	$36.5	$33.4	$25.1	$273.8
EBITDA	USD$M				$53.5	$93.4	$105.4	$54.0	$110.7	$45.3	$37.1	$19.5	$519.0
Support to local communities	USD$M	–	–	($3.0)	($1.0)	($1.0)	($3.1)	($2.6)	($5.4)	($2.0)	($1.7)	–	($19.8)
Taxation	USD$M	–	–	$2.2	($1.0)	($5.2)	($6.2)	($0.7)	($8.5)	($2.0)	$0.2	$0.7	($20.6)
Operating Cash Flow	USD$M	–	–	($0.8)	$51.5	$87.2	$96.2	$50.7	$96.7	$41.2	$35.6	$20.2	$478.6
Less: Change in Working capital	USD$M	–	–	($2.7)	$17.3	$3.2	$1.3	($4.3)	$4.8	($5.5)	($0.9)	($9.1)	$4.2
Less: Capex		–	–	–	–	–	–	–	–	–	–	–	–
Development	USD$M	$23.0	$65.6	$75.5	–	–	–	–	–	–	–	–	$164.1
Sustaining / Closure	USD$M	–	–	–	$2.0	$3.0	$3.4	$2.0	$2.0	$2.0	$1.0	$12.0	$27.2
Total Capex	USD$M	$23.0	$65.6	$75.5	$2.0	$3.0	$3.4	$2.0	$2.0	$2.0	$1.0	$12.0	$191.3
Free Cash Flow	USD$M	($23.0)	($65.6)	($73.5)	$32.2	$80.9	$91.5	$53.1	$89.9	$44.8	$35.5	$17.3	$283.1

22.7 Sensitivity Analysis

Sensitivity analysis has been conducted to assess the effects of changes in key parameters upon the IRR, NPV and the period required for payback of the initial capital. The analysis encompasses the range of +/-10%, 20% and 30% of the following key parameters:

- Gold price
- Gold recovery
- Aggregate operating costs
- Capital costs

In assessing the sensitivity of the Project returns, each of these parameters is varied independently of the others. Combined beneficial or adverse variations in any of these parameters will therefore have a more marked effect on the economics of the project than the individual variations considered.

23 Adjacent Properties

The nearest adjacent property held by another company is Rosino property currently held by Hereward Ventures PLC. Gorubso-Kardjali AD holds the Sedefche exploration licence to the immediate west of the Krumovgrad licence where they are exploring for gold.

24 Other Relevant Data and Information

24.1 Company information

BMM is a joint stock company, solely owned by Dundee Precious (Krumovgrad) BV, a subsidiary of Dundee. BMM was incorporated in September 1997. Dundee acquired all shares in the Company from its previous owner on September 30, 2003.

24.2 Business Legislation

The legal framework for conducting business in Bulgaria has evolved rapidly since 1991 when the new Constitution of Republic of Bulgaria proclaimed the main principles of the market economy. They include the inviolability of the private property, free business initiative and equal conditions for performing economic activities for all individuals and legal persons. In the same year the Commerce Act of Bulgaria was enacted. It provides for the rules applicable to different types of commercial companies with regard to incorporation procedures and documents, capital and shares, shareholders, management bodies, resolutions, mergers, liquidation and insolvency and defined the requirements applicable to different types of business transactions.

24.3 Mining Legislation

The Concession Act of 1995 and the URA of 1999 establish the conditions and the procedures for prospecting, exploration and mining of underground mineral resources located on the territory of the Republic of Bulgaria. The URA defines the main rights and obligations of the permit/concession holders related to the development of the underground mineral resources.

24.4 Taxation

The taxation of corporate income and profits is governed by the Corporate Income Tax Act ("CITA"). With the accession of Bulgaria to the EU on January 1, 2007, a new CITA was adopted to meet the necessity of harmonization of Bulgarian taxation legislation with the requirements of the European directives concerning direct taxation. Under the CITA, all resident companies and partnerships (including non-incorporated business), as well as permanent establishments of non-residents, are liable to corporate income tax of 10%. For tax purposes, a company is considered resident in Bulgaria if it is incorporated (registered) pursuant to Bulgarian legislation. Resident companies are also any companies incorporated under Council Regulation (EC) No 2157/2001, and any cooperative society, incorporated under Council Regulation (EC) No 1435/2003, whose registered office is situated in the country and is entered into a Bulgarian register. The Bulgarian branches of non-residents are deemed resident companies for tax purposes. There are no group taxation rules. Tax anti-avoidance rules cover transfer pricing and related persons.

Most goods and services are subject to a 20% VAT rate. Any person (legal or physical, resident or non-resident) who has a taxable turnover of at least BGN 50,000 during the preceding twelve months is obliged to register for VAT purposes. Only VAT registered persons may charge VAT on taxable supplies and recover input VAT charged to them.

24.5 Customs Duties

Customs duties are payable on the importation of goods and products to Bulgaria.

The Single Market of the EU was built in the course of three decades in compliance with the founding documents. A full EU member, on January 1, 2007, Bulgaria became also an equal participant in the Single Market of the EU. Likewise, domestic legislation in the respective areas was brought into conformity with the legislation of the Community – the acquis communautaire. Bulgaria is also a member of the World Trade Organization. Following Bulgaria's accession to the EU, a number of changes occurred in the foreign trade and customs regime in regard to exports and imports of goods. The new developments concerned the direct application of Community acquis, which regulates the common procedures, tariff and non-tariff measures (prohibitions and restrictions) on exports and imports of goods "to" and "from" non-member states and uniform customs control instruments.

Bulgarian customs legislation is harmonized with the European customs regime. The imports of products are subject to customs duties at rates determined in the Customs Tariff approved by the Government. At its accession to the EU, Bulgaria eliminated customs duties in its trade with the other EU Member States and started applying the Common Customs Tariff of the EU in its trade with non-member states. The Common Customs Tariff requires levying of the same duties on products, imported from third countries. In addition, it is used by the EU as an instrument for regulation of international trade. The EU continues to adapt the Common Customs Tariff to the results of negotiations for tariff reduction within the framework of the General Agreement on Tariffs and Trade (GATT). Bulgaria has preferential tariff agreements (free trade agreements) with EU, EFTA, CEFTA, Turkey, Israel, Macedonia, Albania, Serbia and Montenegro, which may result in certain tariff rates being reduced or eliminated. The preferential tariff rates apply to products originating from the respective party to the agreement and are subject to submission of an evidence of origin.

24.6 Social Security/Health Insurance Contributions

The main legal instruments in the field of social security and health insurance regimes are the Social Security Code, the Health Insurance Act, the Law on the Budget of the Social Security Fund and the Law on the Budget of the Health Insurance.

Legislation requires that all employees are covered by the social security system. The system includes coverage for a group of social risks, which are general illness, work accidents, occupational diseases, maternity, disability, unemployment, retirement and death. Every employee who was employed for more than 5 working days or 40 working hours during a calendar month have to be secured against all social risks.

The social security / health insurance contributions are based on the employee gross monthly remuneration. However, the legislation provides for a minimum and a maximum limit of the amount, used as a base for calculating the social security/ health insurance contributions. The minimum amount varies for different professions (for different mining activities it varies between BGN 270 to BGN 600). The maximum amount for 2011 is BGN 2000. These amounts are usually changed every year.

24.7 Foreign Investment

24.7.1 National Treatment

The Bulgarian Constitution and the PIA provide for national treatment to foreign investors which means that foreign investors are entitled to perform commercial activities in the country under the same provisions applicable to Bulgarian investors except where otherwise is provided by law. In particular this principle covers the whole range of economic and legal forms of activities for accomplishing entrepreneurial businesses. The national treatment of foreign investors allows for the possibility foreign investors to participation in the process of privatisation and acquisition of shares, debentures, treasury bonds and other kinds of securities.

24.7.2 Most Favoured Nation Status

Bulgaria is signatory to a number of bilateral treaties on promotion and mutual protection of foreign investment which provide, further to the national treatment regime, for the most favoured nation status of the investment made by entities and individuals from one of the contracting countries on the territory of the other contracting country.

24.7.3 Priority of International Treaties

When international treaties to which Bulgaria is a party provide for more favourable terms and conditions for foreign investment, these terms have precedence over the national rules. This guiding principle finds expression in the treaties for protection of foreign investments and especially in the agreements for the elimination of double taxation regulations. The international treaties on mutual protection of foreign investment always include an extended concept of a foreign direct investment, and the application of this concept has priority over the Bulgarian legislation. National treatment applies to foreign investors, which means that foreign persons are entitled to invest in Bulgaria under the terms and conditions provided to Bulgarian investors except as otherwise is provided by law.

24.7.4 Guarantees against Adverse Changes of the Legislation

The PIA stipulates that foreign investment made prior to adoption of amendments in the law, which impose statutory restrictions only with regard to foreign investments, shall not be affected by these restrictions.

The URA (Art.63) provides for protection of investments in prospecting, exploration and concession activities against changes in the legislation which result in restriction of rights to, or material damages for, the holder of prospecting and exploration permit or mining concession. In cases where such changes have been adopted, the permit or concession holder is entitled to request amendment of the exploration or concession contract to restore its rights and interests.

24.8 Land Ownership

24.8.1 Legislative Framework

The major legislative acts governing the real estate and real estate transactions in Bulgaria are the Bulgarian Constitution, Property Act, State Property Act, Municipal Property Act, Civil Procedures Code, Investments Promotion Act, Spatial Development Act, Obligations and Contracts Act, Forestry Act and Ownership and Use of Agricultural Land Act.

24.8.2 General Rules

24.8.2.1 Types of Rights over Real Estate

Rights less than full ownership include a real right to use a third party's real estate, a real right to construct over third party real-estate, long-term lease, etc. For the purposes of the concession, full ownership is most suitable to assure undisturbed operation for the life of the mine. Limited rights may be appropriate in the case of temporary roads or other secondary purposes.

24.8.2.2 Direct Acquisition of Real Estate in Bulgaria by Foreigners and Foreign Companies

In Bulgaria foreign citizens and foreign companies can directly acquire buildings, premises within a building and limited property rights (e.g., a construction right, right of use), but not land.

Special rules are provided for the citizens and entities of the member states of the EU and the European Economic Area ("EEA"). According to the Accession Act of Bulgaria to the EU, Bulgaria, upon its discretion, can keep the restrictions for acquisition of land by citizens and entities from the member states:

- for five years starting from January 1, 2007 – for the land provided for second residence; and
- for seven years starting from January 1, 2007 – for agricultural land, forests and forest land.

On March 20, 2007 changes to the Bulgarian Ownership Act, Forestry Act, Protected Areas Act and Agricultural Land Ownership Act were promulgated in the State Gazette. The changes reflect the provisions of the Accession Act of Bulgaria to the EU into the national legislation.

24.8.2.3 Indirect Acquisition of Real Estate in Bulgaria by Foreign Companies or Foreigners

Indirectly, foreign companies and foreign citizens can acquire any type of real estate, including land, by registering a Bulgarian company to act as acquirer. It is possible for such a company to be 100% owned by the foreign investor.

24.8.2.4 The Transaction

The general rule under Bulgarian law is that transactions involving real estate (e.g., a purchase, exchange, etc.) should be executed with a notary deed before a registered notary in the region where the real estate is located. The form of notary deed is mandatory not only for transactions for transfer of ownership title over real estate properties, but also regarding establishment of limited property rights over real estate properties (e.g., construction right, right of use, etc.).

After execution of the deed, the notary is obliged to register the transaction into the Property Register at the Registry Agency in order to make the title of the acquirer defendable against third parties.

A notary deed is not required for disposal of state or municipal property or in privatization transactions where the simple written form is sufficient for a valid title transfer. There are also special rules and procedures governing the acquisition of real estate arising from enforcement, insolvency and similar procedures, and for in-kind contributions of real estate.

24.8.2.5 The Price

The purchase price is freely negotiable and may be stipulated and paid in BGN or in any other currency.

24.8.2.6 Title Review (Real Estate Legal Due Diligence)

Before purchasing real estate it is recommended that the buyer ensure verification of the ownership status of the targeted real estate, including that there is/are:

- a clean, valid and ownership title held by the seller. The seller has to be, and his predecessors should have been the valid owner of the targeted real estate in order to avoid any risk of rescinding or annulment of the transaction. Usually, this title review covers the last 10 years since the maximum acquisitive prescription term in Bulgaria is 10 years;

- no liens or encumbrances over the property. The buyer should be fully aware as to whether there are any registered liens and/or encumbrances over the targeted real estate, e.g., mortgages, interlocutory injunctions, going-concern pledges, limited property rights established in favour of third parties. A general principle in Bulgarian law is that liens and encumbrances "follow the property", i.e., the registered liens and encumbrances can been forced against the new owner;

- no other registered rights in favour of third parties – if there are registered rental or lease agreements over the targeted real estate then the buyer shall be bound by them until the expiration of their term; and

- no court or restitution claims.

24.9 Land Acquisition

24.9.1 Introduction

Development of the project will require the acquisition of land and buildings from a variety of impacted stakeholders. The size of the project footprint or the Project Impact Area (PIA) has been minimized as much as possible and the project footprint and its anticipated buffer will be approximately 85 ha.

Based on its intended use the land within the claimed concession area comprises agricultural land, forest land and urbanized areas. Several uninhabited buildings (currently owned by individuals), the 'Svezhest' tourist lodge (in poor condition and unusable) and also a student lodge with adjacent bungalows are located within the concession area. Both lodges are owned by the Krumovgrad municipality. Both the tourist lodge and student lodge will not be usable as intended once the concession activities commence, therefore DPMKr will have to hold negotiations with the municipality to acquire them.

The majority of the project area is covered by forests which are state property included in the national forest fund. There is also agricultural land owned by the state, as well as areas owned by the municipality The access road which will need to be upgraded and widened is the only area of the project that is partly located on privately owned agricultural land near the entrance to the site.

24.9.2 Legal and Institutional Framework

The state forest lands within the concession area are private state property. These lands are part of the State Forest Fund and the ownership can be transferred to third parties after the land use is changed and the lands are excluded from the State Forest Fund. To change the land use DPMKr will file an application for prior approval by a Committee with the Executive Forestry Agency. The sale is finalized by the Minister of Agriculture and Foods or an authorized officer under the State Property Act.

The acquisition, management and disposal of municipal property that is not forest land are performed by the Mayor under the general direction and control of the Municipal Council. The procedure for acquisition of ownership or limited real estate rights is set out in the Municipal Property Act and relevant Municipal Council's Regulation in accordance with the provisions of the relevant laws.

Citizens and legal persons can dispose of their land in accordance with the requirements of the applicable law.

In compliance with the provisions laid down in the URA, DPMKr will hold independent negotiations in order to acquire rights over the required areas.

Unsuccessful negotiations may serve as grounds for initiating a compulsory expropriation procedure for the respective land. This would be done at the request of DPMKr in compliance with the provisions laid down in the URA and the Spatial Development Act (”SDA”). At this stage DPMKr does not believe this procedure will be required.

24.9.3 Acquisition Strategy

Because the completion of title documentation will be complex and lengthy, including identifying and locating owners and heirs, the process will be started as soon as possible. The hamlets in the PIA need to be surveyed in order to create a complete and accurate cadastre. The cadastral surveys will be carried out by a licensed surveyor contracted with the Company, under a formal procedure according to legislation. The procedure involves publication in State Gazette, adjudication of boundaries on the ground, and the opportunity for appeal by affected parties.

24.9.4 Redesignation of Land for Project Implementation

The granting of a concession does not affect the permitted use of land on the surface nor change its designation or planning status. Procedures must be taken under the Spatial Planning Law, Preservation of Agricultural Land Law, and Forest Law. Forest land must be "excluded" from that status. Agricultural land must be redesignated for the mine use. The hamlets within the mine footprint must lose their settlement status. The planning status of the territory must be changed to be consistent with construction of the mine and its facilities, thus providing a basis for construction approval. All of these matters involve the approval of a DDP.

24.10 Project Implementation

24.10.1 General

Dundee's intention for the Project implementation strategy is to go forward with a hybrid Engineering, Procurement, Construction and Management ("EPCM") structure where a renowned international consultant will be involved in the delivery of the "EP" component of the Project, and Dundee's experience and "know how" in Bulgaria will be used to optimize the detail engineering, permitting and execution of the "CM" component of the Project. An Owner's team commensurate with the stage of development will gradually be built up and award a number of consulting agreements to various consultants to supplement the Owner's team based on a defined scope of engineering deliverables and implementation schedule.

These contracts are to include:

- Contract an engineering consultant to undertake the preliminary engineering (EP) of the process plant and associated infrastructure. The detailed engineering, including preparation of associated permit packages, would be performed by the Owner's team with input as required from the consultant. The Owner's team will include licensed Bulgarian engineers, who would be responsible for the signing off of all detailed engineering drawings, calculations notes and documentation as required under Bulgarian legislation.

- Contract a mining and tailings facility engineering consultant, who would be responsible for the preliminary engineering of the Paste plant and IMWF with associated haul roads and main plant access road. This consultant would also be responsible for coordinating the geotechnical aspects of the plant site earthworks requirements and assist in the procurement of all the mining equipment required for the Project. The detailed engineering, including preparation of associated permit packages, would be performed by the Owner's team with input as required from the consultant. The consultant would be present for the QA/QC function during civil construction of the facility and may also assist in the initial grade control and mine scheduling functions prior to the engagement of the Owner's mining engineering team.

- Permitting would be an integrated function of the Owner's team, which would assist DPMKr through the multitude of permits required for Project development.

- An environmental consultant would be engaged (or a dedicated Environmental Manager appointed) to assist DPMKr in ensuring compliance with all aspects of the EIA.

- An engineering peer review process would be performed under the direction of the Owner's team.

The various consultants must be experienced in the design and construction of projects of this type and would preferentially have had experience in Bulgaria. For this reason Dundee may elect to continue with the various consultants involved for the various aspects of the Project that have completed the mining study.

24.10.2 Mining

The mining implementation plan will detail how Dundee will commence owner mining. The plan includes a schedule and general requirements to complete the following:

- Update of mineral resources and then re-optimize the mine design and scheduling process, based on the latest cost variables.
- Finalising the selection of the optimum mining equipment in terms of equipment performance, capital and operating costs and supplier maintenance costs and support.
- Additional technical work on geotechnical, dewatering and stream diversion options.
- Sourcing and selecting suitable personnel to carry out the implementation plan and to manage the mining operations.
- Ensuring employment conditions for significant additional personnel.
- Implementing appropriate infrastructure to support the new mining operations.
- Reviewing and supplementing existing personnel resources from site service departments such that the owner mining option is adequately supported.

24.10.3 Process Plant and Paste Plant

24.10.3.1 General

The most suitable approach to develop the Project will be to engage two main engineering consultants. One consultant will develop the process and basic design and another to procure equipment and to provide support in the detailed engineering, construction and commissioning phases of the process plant, paste plant and associated infrastructure and services. The activities will be led by the Owner's team Project Manager and Engineering Manager.

Detailed engineering, preparation of associated permit packages and construction tenders will be performed by members working under the direction of the Owner's team Engineering Manager. This approach ensures that the work is performed without duplication, and there is a direct transfer of information throughout all stages of the Project. Equipment specifications procurement packages are critical to the engineering schedule to ensure vendor information is incorporated into the design prior to completion of the permitting packages.

When the focus of work shifts to site, the Owner's Site Manager will assume responsibility for the construction phase with support provided by:

- On site by various discipline supervisors;
- Owner's team Engineering Manager and Project Controls/Administrative Manager supported by engineers and administrative staff; and,
- Two main engineering consultants will provide representatives on site as required supported by additional staff at their respective home offices.

In accordance with the Bulgarian legislative requirements, there will be statutory supervision by licensed engineers and the independent construction control consultant.

When the Project reaches the point of practical completion, the plant will be handed over to the Commissioning Manager. The Commissioning Manager will be supported by the Site Manager until the plant is successfully commissioned and handed over to Operations.

It is planned that DPMKr will set up an "Owner's" Project Management Team to manage the Project throughout all stages of engineering, construction and commissioning.

24.10.3.2 Contracting Strategy

The contracting strategy employed for the Project will largely depend on the types and proficiencies of the local contractors, the requirement to utilize local contracts, and their ability to meet the Project requirements.

For the Krumovgrad Gold Project, the permitting requirements dictate that most of the engineering will need to be completed for the process plant prior to a permit for construction being obtained. However, the recent experience at the Chelopech Mine Expansion indicate that preliminary works for Earthworks and Foundations can be authorized in stages following the approval of the construction permit applications. The preliminary construction approval (Stage 1) would be followed by the technical permit application (Stage 2) when the engineering/permitting process is completed for the whole Project. This still needs to be validated, and DPMKr is reviewing measures to minimize the effect of this on the overall schedule, but at this time, it has been assumed that we will need to comply with the strict interpretation of the law. The assumption is that the engineering design must be substantially complete prior to the construction permit application which will enable good lump sum style contracts for construction to be obtained. It is currently envisaged that the construction permits can be applied for in stages in accordance with the construction priorities as listed below.

- Main access road and bulk earthworks.
- Communications tower and associated facilities.
- Permanent Administration Office facility.
- Main power supply.
- Reagent store.
- Core shed and laboratory building.
- Process plant office, workshop, store and ablutions.
- Crushing plant.
- Main process plant including process plant building.
- Paste Plant.
- Integrated Mine Waste Facility and return water systems

During the mining study, a number of Bulgarian construction contractors were requested to supply rates for the construction estimate. It was evident from this review and from the recent experience at the Chelopech Mine Expansion Project that Bulgarian contractors have the capability to perform all the general fabrication and construction activities required for the Project. It was also evident that these contractors would need some guidance through the tendering process and extensive supervision during fabrication and construction to ensure quality and time requirements are met as these contractors are not familiar with construction of this type of mineral processing facility. It was also evident that although contractors may not have previously been familiar with schedule of Unit Rates style of contracting, the Chelopech Expansion Project has been executed on that basis with positive results. These contracts are much more straightforward and easier to manage, from both sides.

It was concluded that horizontal packaging of the majority of contracts on a schedule of Unit Rates basis for the construction of the major project works, would provide the following advantages.

- A horizontal contract will permit more direct contractor expertise to be used with better control of contractor services. Subcontractors normally maintain strength in one major discipline and as such may not be capable of performing all aspects of a vertically integrated contract. These works could be subcontracted out to other groups over which the Prime Contractor may not have effective control.

- The horizontal package contract should provide a more cost competitive bid as generally the contractor will be comfortable in one area for tendering.
- It will allow smaller local contractors to competitively bid for the work.

24.10.4 Integrated Mine Waste Facility

DPMKR will use an integrated approach to mine waste management to minimize the footprint required for mine waste disposal. The concept of an IMWF is to place thickened tailings within cells constructed from mine rock.

The IMWF will be constructed in stages during the LOM. The IMWF will be built from the bottom up, with upper parts of the facility founded on previously placed mine rock and tailings. The thickened tailings will be transported via pipeline for discharge into the mine rock cells. Access for the pipeline and other aspects of the IMWF will be constructed using the mining fleet. The Initial or Stage 1 facility will involve the construction of the two starter platforms, access haul roads and associated works, and is planned to take place during the dry season. The facility will be raised and extended in a multitude of stages in order to provide the design storage capacity.

Materials for construction of the IMWF embankments will be sourced from bulk earthwork on the plant site and internal roads, within the IMWF basin and the Ada Tepe Pit (mine waste). In addition, there is a requirement to source aggregate materials for incorporation in the embankment drainage layers. These materials will be sourced outside the Project area from commercial sources. The Stage 1 IMWF construction is expected to take approximately eight to nine months.

The Stage 1 IMWF embankment earthworks would utilize the mining fleet for the placement of materials. In subsequent upstream raises of the IMWF facility, the mining fleet will be utilized to construct the IMWF embankment raises.

Construction of a water storage reservoir embankment will also be required for the operation. This will proceed in parallel with the plant site access road construction. The embankment construction for the water storage is expected to take approximately five to six months. It is envisaged that the water storage embankments and associated works will be constructed by a civil contractor prior to delivery of the mining fleet.

24.10.5 Overall Project Schedule

A Project Schedule has been developed to design, construct and commission the processing plant and associated facilities, and incorporating the land acquisition and permitting activities.

The Project Schedule indicates an overall project duration of 125 weeks from award of the contract to the Engineer to Practical Completion (end of pre-commissioning).

The schedule activity durations for procurement and construction are based on quoted manufacturing and installation periods and previous experience working on similar projects

24.11 Risk Assessment

The Risk Assessment for this project was performed in 2011.

CSA do not consider that there is any change to the risk profile that needs recording in 2014 other than the following;

- Concerning the item Risk ID 41, delays to the project due to the archaeology investigation at Ada Tepe, this work is now complete and it is probable that there will be no delay to the project as a result of this work.

The risk review was workshop based. Participants were selected for their seniority, knowledge, expertise and involvement in the operation. They included DPMKr project team members and key project consultants from Golder and Lyntek. The risk review was facilitated by DPMKr's Corporate Director, Health Safety and Environment.

The objectives of this risk review were to:

- Provide a structured forum for communication on risk issues;
- Identify, assess and rank significant project risks;
- Review the general control environment and identify gaps in existing controls;
- Establish the basic framework for a control improvement program.

24.11.1 Risk Review

The following process was adopted for the risk review:

- Lyntek and Golder gave detailed presentations on the design work done so far and the main limitations and constraints.
- Risk review workshop was held to identify risks and propose appropriate mitigation measures.
- Risk & mitigation review meetings were held to review all risk items mitigation measures. These were recorded in the risk matrix.
- Further refinement of mitigation actions established the basis for re-ranking the project risks and for managing the implementation of each control. The controls are focused on

reducing the effect of all risk issues with a particular focus on reducing those that are ranked high and extreme.

24.11.2 Project Objectives Considered

Risk is the chance of something happening that could impact upon the achievement of project objectives.

The purpose of the Project is to deliver mining operations and associated facilities which maximize NPV within the project constraints (including 850,000 tpa throughput requirements). In order to achieve this, specific Project objectives were identified. The project outcome objectives are to:

- Minimize cost
- Maximize revenue
- Minimize harm to the environment
- Maximise positive health and safety outcomes
- Optimum relations with community and workforce
- Compliance with government requirements
- The project delivery objectives are:
- Cost Effectiveness
- Project Start up
- Project permits
- Environment outcomes
- Health and safety outcomes
- Community outcomes

24.11.3 Risk Assessment Outcomes

The risk profile explains important risks that could impact the achievement of each objective by describing the risk event and analysing the realistic worst case consequences and the likelihood of those consequences being realized.

The workshop team identified 52 risk issues which were categorized into the following initial inherent risk rankings. However, risk ID 41 is no longer considered to be a factor, and has been removed from the list; resulting in 51 risk issues.

- 11 low
- 6 moderate
- 17 high
- 17 extreme

After consideration of the proposed mitigation measures following residual risk rankings resulted:

- 11 low risks
- 19 moderate
- 17 high
- 4 extreme (Table 73)

The project risks are of two types:

- Project delivery risks which relate to permitting and regulation conditions, schedule delays, capital cost overruns and mine development delay.
- Project outcomes risks which relate to safety culture and approach, plant performance and environmental issues.

Table 73. Four Extreme Residual Risks and Mitigation Strategies

Risk ID	Risk Event	Mitigation Strategies
33	Political Opposition to the project	Set Up a strategy for communication with politicians

34	NGO's opposition to the project	Emphasize environmental concerns have been minimized Point out the economic benefits for the people Review of documents. Detailed discussions, analysis Communication/coordination with government authorities Common strategy
47	Land Acquisition delays	Prioritize property acquisition Land acquisition strategy in place Continuous communication with government and community
53	Vehicle Accidents during construction	Management focus on safety Create a traffic management Plan HSE induction Daily monitoring Risk assessment and JSA

One important observation is that there are significant risks related to country political and community attitudes to the project. By contrast, there are no significant technical issues of significance, indicating that this is a "soft issues" project requiring focus on proactively managing stakeholders and perception about the project.

In addition to the risks identified by DPM, and reviewed by CSA, CSA would draw attention to the risks posed by the construction schedule of the IMWF. The structure for the IMWF will be built up with waste rock forming the cells prior to deposition of the tails material. At all times, enough waste must be delivered to the facility for cell construction. There is, however, little space for stockpiling waste if a surplus arises. Therefore, the timing of waste delivery will always be a risk factor which may affect the deposition of process tailings.

No risks are considered to be fatal flaws in the context of the Krumovgrad Project, although further actions to improve risks should be undertaken to progressively improve the risk profile.

Key areas for further action concern firming construction costs, better understanding of and planning for operational workforce capability; further focus on management of permitting; approaches to engage the community and ensure political support, manage the archaeological works, ensure better understanding of the IMWF from the local authorities and focus on safety during construction.

24.12 Abbreviations

A full listing of abbreviations used in this report is provided in the Glossary at the beginning of this document.

25 Interpretation and Conclusions

DPMKr conducted detailed exploration of the Ada Tepe prospect between 2000 and 2004. 52.9 km of drilling, and 18.3 km of surface trenching was completed, with more than 66,000 individual assay intervals and 5,700 bulk density determinations. This has resulted in a strong level of confidence in the data on which the resource is based. The mine plan proposed shows a high conversion of resources to reserves at the cut-off grades selected.

The extent of the data collected through this exploration program and the quality control standards used provide the basis for a high level of confidence on the potential of this project. Furthermore, the project has been demonstrated to be technically and commercially viable if designed, constructed, commissioned and operated as specified in the mining study. Opportunities still exist in most areas of the project to optimize various components that will be more rigorously investigated during the detailed engineering phase and where appropriate will be incorporated into the project.

CSA makes the following concluding comment as regards drilling, sampling and data verification;

- *CSA verified the location of some trench and drill access road cut exposures during the 2012 site visit and was able to confirm the positions of some sampling points.*

- It is the opinion of CSA that the RC drilling and associated sampling was completed to high industry standards. This opinion is informed by a review of data collection procedures, protocols and metadata contained in the database for the project, as reviewed during the 2012 site visit.

- *CSA observed this pulp library facility during the 2012 site visit and performed random spot checks of sample numbers and compared these with data contained in the project database. No issues were detected.*

- During the 2012 site visit CSA reviewed logging information contained in the database for the project, and verified this information (exported as graphic drill logs from acQuire software) against several drill holes inspected at the time. No issues were detected and CSA verifies that the information stated as being collected has been captured.

- CSA used the software programme QAQCReporter ("QAQCR") to produce QA/QC reports in order to review the accuracy and precision of the assayed QA/QC material and samples. No significant issues of bias or fatal flaws were noted.

- *Results of the internal laboratory blanks and standards were not available for review by CSA, however the results of the DPMKr submitted blanks and standards assayed show no evidence of systematic bias. CSA considers that the gold and silver standards analysed by* *the SGS Gura Rosiei laboratory to be accurate and appropriate for mineral resource estimation studies.*

- *CSA considers that the gold and silver standards analysed by the SGS Welshpool laboratory to be accurate and appropriate for mineral resource estimation studies.*

- *Internal laboratory blanks and standards were not available for CSA review. Those submitted by DPMKr were reviewed. CSA notes that accuracy and precision exhibited by SGS Chelopech was, at times, poorer than the other laboratory results, however no significant bias is noted and CSA considers that the gold and silver standards analysed by the SGS - Chelopech laboratory to be accurate and appropriate for mineral resource estimation studies.*

- CSA reviewed the results of the check analyses (duplicates, repeats and pulp splits). No significant bias or material issue were detected.

- *CSA considers that Industry Standard acceptable levels of precision are reported for all of the sampling stages for the purpose of mineral resource estimation.*

- *CSA, having reviewed the results of twin core sampling, agrees with the conclusions reached by RSG Global.*

CSA makes the following concluding comment as regards to Mineral Resource Estimation;

- CSA considers the approach used by RSG to model mineralisation, lithological and oxidation boundaries to be valid for the current level of study.
- *CSA considers the approach of defining a zone of dilution to be a valid one which honours the mining approaches being considered and facilitates the reliable use of MIK as a grade interpolation method.*

- CSA agrees with the approach adopted by RSG for the consideration of grade capping.

- CSA agrees with comments made by RSG as regards data clustering and agrees that cell declustering is a valid and meaningful approach to arriving at reliable and unbiased mean grades for each domain.

- CSA has reviewed density data and considered the application of assigned densities to material types to be valid.

- CSA considers the global validations completed by RSG to be valid. As an additional check CSA completed a validation estimate on selected domains within the Upper Zone using Uniform Conditioning as a check against MIK.

- CSA has reviewed the relationship between gold and silver grades and believes the assumptions regarding the appropriateness of estimating silver grade through linear regression from gold grade in the model, to be valid.The results of the Uniform Conditioning check estimate for silver, completed by CSA on several domains in the Upper Zone compares favourably to the results obtained by RSG and goes some way to validate the approach taken by RSG to estimate silver grade.

- CSA completed the following validation checks on the RSG Mineral Resource block model;
  - Swath Plots depicting model tonnes, input declustered composite gold grade, output block model gold grade and drill metres per slice, for the Wall Zone.
  - Regularisation of blocks within the RSG block model file so that this block model could be used for mine planning work completed by DPM in 2013.
  - A check estimate of the CSA block model using Uniform Conditioning, completed for Domains 10, 11 and 12 within the Upper Zone. These domains are considered representative of the grade distribution within the Upper Zone and together account for 27% of contained metal within the Upper Zone.
  - On-screen visual comparisons of the RSG block model grades (via MIK) and the CSA Uniform Conditioning check estimate block model grades, for domains 10, 11 and 12.
- The UC check estimate compares favourably to the CSA block model interpolated by RSG using MIK. Additionally, CSA makes the following comment relating to the check estimate;
  - The check estimate reports 4% less tonnes at a 9% higher gold grade, resulting in a 4% increase in contained gold within the check domains, at a 0.6 g/t Au cut-off (the cut-off used to report Mineral Resources and Mineral Reserves in the Upper Zone).
  - The check estimate reports 4% less tonnes at a 5% higher silver grade, resulting in a 1% increase in contained silver within the check domains, at a 0.6 g/t Au cut-off.
  - UC has validated the MIK results suggests the variogram models and the linear regression of silver to be valid.
- The Mineral Resource Estimates for the Ada Tepe deposit have been categorised in accordance with the criteria laid out in the NI 43-101. A combination of Measured, Indicated and Inferred Mineral Resources have been defined using definitive criteria determined during the validation of the grade estimates, with detailed consideration of the NI 43-101 categorisation guidelines.

- On an after tax basis, based on USD 1,250 per ounce price for gold and USD 23 per ounce for silver and on the treatment of 850,000 tpa of ore, the project produces an IRR of 26.3% on a 100% equity funded basis and an after tax NPV of USD 143.9 million at a discount rate of 7.5%. The anticipated project payback time is 2 and a half years. For a gold price of USD 1,625 per ounce, the sensitivity analysis shows the equivalent IRR will rise to 38.6% with an NPV of USD 270.5 million.

- No risks are considered to be fatal flaws in the context of the Krumovgrad Project, although further actions to improve risk profile will be undertaken.

26 Recommendations

The following recommendations are made, as regards the Ada Tepe Deposit;

- A review of logged oxidation and therefore the interpretation of oxidation surfaces used to flag the density model may require sensitivity review to aid both improvements in the tonnage model, and the metallurgical process.

- The current mineral resource estimate reports a selective mining model for the upper zone and assumes that a high density of high quality grade control (such as angled RC drilling) will be available. The current close spaced drilling has identified significant short scale variability and RSG considered that appropriate ore block delineation methods such as conditional simulation and probabilistic ore block selection are applied in conjunction with the RC grade control drilling. CSA agrees with this approach.

- The results of the economic analysis indicate that exploitation of the Ada Tepe gold deposit is economically viable and should proceed.

- Key areas for further action to be undertaken in 2014 include:

    - confirming construction cost assumptions through additional trade-off studies

    - further focus on management of permitting

    - production of sufficient concentrate to complete marketing studies

    - better understanding of, and planning for, operational workforce capability

    - Continuation of the CSR plan to further engage the local community and, municipal, regional and national political support

27 References

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