Report No. 20200515_MED

PRELIMINARY ECONOMIC ASSESSMENT AND NI 43-101 TECHNICAL REPORT FOR THE MEDGOLD TLAMINO PROJECT LICENCES, SERBIA

PREPARED FOR

MEDGOLD RESOURCES CORP.

BY

Addison Mining Services

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EFFECTIVE DATE

JANUARY 07, 2021

SIGNATURE DATE

MARCH 11, 2021

Document Title	Preliminary Economic Assessment and NI 43-101 Technical Report for the Medgold Tlamino Project Licences, Serbia.
Document Control	Final v1.2
Prepared for	Medgold Resources Corp. Suite 650-200 Burrard St. Vancouver BC, Canada V6C 3L6
Project Number	20200515_MED
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Effective Date	January 07, 2021
Signature Date	March 11, 2021
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Qualified Person **Mining**	**Dr Matthew Randall** – BSc (Hons), PhD, MIMM, CEng, Principal Mining Engineer, Axe Valley Mining Consultants Ltd. Item 16, and relevant subsections of items 1, 18, 21, 25, 26
Qualified Person **Environmental, Permitting and Social**	**Dr Sue Struthers** – BSc, MSc, PhD, FIMMM, CEnv, Principal Environmental Consultant, Skapa Mining Services Ltd. Item 20, and relevant subsections of items 1, 25, 26

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iv. Certificates of Qualified Persons

I, RICHARD JOHN SIDDLE, MGeol (Hons), MSc, MCSM, MAIG, FGS do hereby certify that:

1. I am currently employed as Senior Resource Consultant by; Addison Mining Services Ltd, 13-17 High Beech Road, Loughton, Essex, IG10 4BN
2. I am the Qualified Person for this Technical Report; “Preliminary Economic Assessment and NI 43-101 Technical Report for the Medgold Tlamino Project Licences, Serbia” with the effective date January 07, 2021 and take responsibility for Items 2 to 12, 14, 23 and relevant subsections of items 1, 25, and 26.
3. I graduated with a Master of Geology (Hons) from the University of Leicester, UK, in 2007. In addition, I obtained a Master of Science (merit) in Mining Geology in 2010 from the Camborne School of Mines, University of Exeter, Tremough, Cornwall, UK.
4. I am a member of the Australian Institute of Geoscientists (membership number 6802) and a fellow of the Geological Society of London.
5. I have worked as a geologist for over 10 years since graduation from university. Relevant experience includes 3 years of exploration, drilling supervision and resource development in respect to uranium, gold, silver and base metal deposits in Queensland, New South Wales and Western Australia and 2.5 years as a consulting resource geologist at Micromine Consulting Services. I have since spent 6 years performing resource estimation and geological modelling for Addison Mining Services.
6. I completed a site visit to the Tlamino Project between November 11 and November 14, 2019 in order to assess data collection methodologies, exploration practises, geology and styles of mineralization.
7. I have read the CIM definitions, and definition of “qualified person” as set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements of being a “qualified person” for the purposes of NI 43-101
8. I am independent of the issuer when applying all of the tests in section 1.5 of National Instrument 43-101.
9. I have no prior involvement with the Tlamino Project apart from the provision of independent professional consulting services and performing the studies as contemplated by this report.
10. I have read and am familiar with the CIM definitions, National Instrument 43-101 and Form 43-101F1. The Technical Report has been prepared in compliance with those instruments and form.
11. As of the effective date of the Technical Report, to best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this day March 11, 2021.

Richard Siddle

“Richard Siddle”

I, ANDREW BAMBER, BSc, MASc, PhD, P.Eng do hereby certify that:

1. I am currently employed as Director and Principal Consultant by; Bara Consulting Ltd,17 Central Buildings, Market Place Thirsk, North Yorkshire, YO7 1HD
2. I am the Qualified Person for this Technical Report; “Preliminary Economic Assessment and NI 43-101 Technical Report for the Medgold Tlamino Project Licences, Serbia” with the effective date January 07, 2021 and take responsibility for sections 22, 19 and relevant subsections of items 1, 18, 21, 25 and 26.
3. I graduated with a BSc (Hons) in Engineering from the University of Cape Town in 1993, a Master of Applied Science in Mineral Processing from the University of British Columbia in 2005 and a PhD in Mining Engineering from the University of British Columbia in 2008.
4. I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum Engineers (CIM) in Canada, a registered Professional Engineer with Engineers and Geoscientists BC (EGBC) in Canada, and a member of the IMMM in the UK.
5. I have worked as an engineer in mining since 1989. Relevant experience includes over 25 years on underground and open pit mining operations and projects, over 5 years in design and construction of mining projects, and over 15 years as a consultant in NI 43-101 and JORC-compliant studies in gold, silver, copper, nickel, lead, zinc and tungsten.
6. I have not completed a site visit to the Tlamino Project.
7. I have read the CIM definitions, and definition of “qualified person” as set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements of being a “qualified person” for the purposes of NI 43-101
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10. I have read and am familiar with the CIM definitions, National Instrument 43-101 and Form 43-101F1. The Technical Report has been prepared in compliance with those instruments and form.
11. As of the effective date of the Technical Report, to best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this day March 11, 2021.

“Andrew Bamber”

I, IAN JACKSON, BEng, CEng, ACSM, FIMMM, MSAIMM do hereby certify that:

1. I am currently employed as Process Engineer by Jackson's Unique Mineral Processing Services Ltd, 62 – 64 New Road, Basingstoke, Hampshire, RG21 7PW.
2. I am a Qualified Person for this Technical Report; "Preliminary Economic Assessment and NI 43-101 Technical Report for the Medgold Tlamino Project Licences, Serbia" with the effective date January 07, 2021 and take responsibility for Sections 13, 17 and relevant subsections of items 1, 18, 21, 25, 26.
3. I graduated with a Bachelor of Engineering (Mineral Process Engineering) from the Camborne School of Mines, UK, in 1987.
4. I am a Chartered Engineer, a Fellow of the Institute of Materials, Minerals and Mining and a Member of the Southern African Institute of Mining and Metallurgy.
5. I have worked as a mineral processing engineer for over 30 years since graduation. Relevant experience includes 9 years of operations on gold, PGM and antimony mines in South Africa and 23 years consulting and performing process engineering on projects and feasibility studies of projects to recover gold and silver; PGM; iron; copper, nickel, lead and zinc sulphides; chromite; phosphate; heavy minerals.
6. I have not completed a site visit to the Tlamino Project.
7. I have read the CIM definitions, and definition of "qualified person" as set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements of being a "qualified person" for the purposes of NI 43-101.
8. I am independent of the issuer when applying all of the tests in section 1.5 of National Instrument 43-101.
9. I have no prior involvement with the Tlamino Project.
10. I have read and am familiar with the CIM definitions, National Instrument 43-101 and Form 43-101F1. The Technical Report has been prepared in compliance with those instruments and form.
11. As of the effective date of the Technical Report, to best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
12. Dated this day March 11, 2021.

"Ian Jackson"

I, MATTHEW RANDALL, PhD, BSc (Hons), MIMM, CEng do hereby certify that:

1. I am currently employed as Director and Principal Mining Engineer by Axe Valley Mining Consultants Ltd. 138 High Street, Swanage, Dorset, England, BH19 2PA
2. I am a Qualified Person for this Technical Report; “Preliminary Economic Assessment and NI 43-101 Technical Report for the Medgold Tlamino Project Licences, Serbia” with the effective date January 07, 2021 and take responsibility for Sections 16 and the relevant subsections of items 1, 18, 21, 25, 26.
3. I graduated with a Bachelor of Engineering honours degree from the Camborne School of Mines, UK, in 1978 and subsequently completed a PhD in Rock Mechanics in 1989.
4. I am a Chartered Engineer and a Member of the Institute of Materials, Minerals and Mining.
5. I have worked as a mining engineer for over 35 years since graduation. Relevant experience includes 15 years of operations on gold, copper and Borax mines in South Africa, Spain, Papua New Guinea, and the USA. I also have more than 20 years’ experience in consulting and performing mine engineering on projects and feasibility studies of projects to recover gold and silver, PGM, iron, copper, nickel, lead and zinc sulphides, chromite, talc and various heavy minerals.
6. I have not completed a site visit to the Tlamino Project.
7. I have read the CIM definitions, and definition of “qualified person” as set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements of being a “qualified person” for the purposes of NI 43-101.
8. I am independent of the issuer when applying all the tests in section 1.5 of National Instrument 43-101.
9. I have no prior involvement with the Tlamino Project.
10. I have read and am familiar with the CIM definitions, National Instrument 43-101 and Form 43-101F1. The Technical Report has been prepared in compliance with those instruments and form.
11. As of the effective date of the Technical Report, to best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this day March 11, 2021

“Matthew Randall”

I, Sue Struthers – PhD, MSc, BSc, FIMMM, CEnv do hereby certify that:

1. I am currently employed as Principal Consultant by Skapa Mining Services Ltd. Hillbanks, Burray, Orkney, KW17 2SX
2. I am a Qualified Person for this Technical Report; "Preliminary Economic Assessment and NI 43-101 Technical Report for the Medgold Tlamino Project Licences, Serbia" with the effective date January 07, 2021 and take responsibility for item 20, and relevant subsections of items 1, 25, 26
3. I graduated with an MSc in Mining Geology from the Camborne School of Mines in 1984 and graduated with a PhD from in Environmental Engineering from RMIT, Melbourne in 1998.
4. I am a Chartered Environmentalist and a Fellow of the Institute of Materials, Minerals and Mining and a Member Institute of Environmental Science
5. My experience includes 36 years in the mining industry, 22 of which have been in geological- and mining environmental and social consulting.
6. I have not completed a site visit to the Tlamino Project.
7. I have read the CIM definitions, and definition of "qualified person" as set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements of being a "qualified person" for the purposes of NI 43-101.
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9. I have no prior involvement with the Tlamino Project.
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11. As of the effective date of the Technical Report, to best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this day March 11, 2021.

"Sue Struthers"

1 Executive Summary

1.1 Introduction

This Technical Report was prepared by Addison Mining Services ("AMS") for Medgold Resources Corp. (the "Issuer', "Medgold") of Suite 650-200 Burrard St. Vancouver BC, Canada V6C 3L6. Medgold is the indirect 49% owner of the Tlamino Project to which this report relates. Fortuna Silver Mines Inc. is the direct owner of 51% of the Tlamino Project having met the terms of an earlier option agreement, now cancelled. In January 2020 Medgold subsidiaries entered into a new Option Agreement with Fortuna to acquire Fortuna's 51% interest in the Tlamino Project.

This Technical Report has been compiled by the following independent Qualified Persons (QP).

- Mr Richard Siddle – MSc, MGeol (Hons), FGS, MAIG, Director and Senior Consultant Geologist, Addison Mining Services Ltd. – QP Geology and Resources
- Dr Andrew Bamber – BSc, MASc, PhD, P.Eng, MCIM, Director and Principal Consultant, Bara Consulting Ltd. – QP Financial Analysis
- Mr Ian Jackson – BEng, FIMMM, CEng, Principal Process Engineer, Jackson's Unique Mineral Processing Services Ltd. – QP Mineral Processing
- Dr Matthew Randall – BSc (Hons), PhD, MIMMM, CEng, Principal Mining Engineer, Axe Valley Mining Consultants Ltd. – QP Mining
- Dr Sue Struthers – BSc, MSc, PhD, FIMMM, CEnv, Principal Environmental Consultant, Skapa Mining Services Ltd. – QP Environmental and Social

The study is based on findings of a site visit by AMS, desktop study, data review, data validation, deposit modelling, block model grade interpolation, Mineral Resource estimation, metallurgical testing, pit optimisation, conceptual mine planning and scheduling, cost estimation and preliminary economic analysis.

A site visit was conducted to the Tlamino Project between November 11 and November 14, 2019, by Mr. Siddle as part of a previous study, the purpose of which was to inspect the Property, core processing procedures and to confirm the presence of mineralization. Due to restrictions relating to the COVID-19 pandemic in force at the time of the study, no other QP was able to visit the site.

The Mineral Resources estimated, and the Preliminary Economic Assessment undertaken as part of this study have been reported in accordance with the *National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101)* and *The CIM Definition Standards on Mineral Resources and Reserves (CIM Definition Standards)*. The report has been prepared to be filed on SEDAR in support of a News Release by the Issuer dated 26/01/2021 in which the findings of the Preliminary Economic Assessment were reported.

1.2 Property Description and Location

Medgold holds three Exploration Licenses ("ELs") in south eastern Serbia, forming a contiguous block totalling approximately 199 square kilometres. Two of the ELs – namely Donje Tlamino and Surlica-Dukat – form the Tlamino Property (the "Project") and cover a total area of 192.63 square kilometres. Medgold staked Donje Tlamino and Surlica-Dukat in October 2016 for an initial three years, the ELs were re-issued on September 30, 2020 for a further 3 years. The Project is subject to an option agreement with Fortuna Silver Mines Inc. ("Fortuna"), described below.

The Property falls principally in the district of Pčinja, and the municipalities of Bosilegrad and Trigovište. The central coordinates of the Project are approximately 609500mE/4693000mN (WGS84 UTM34N), or 22.3° east and 42.4° north.

The third EL, Žuti Kamen, has not been subject to exploration work, it did not form part of the Original Option with Fortuna, and the locations of known mineralised zones on the Donje Tlamino and Surlica-Dukat ELs suggest it is unlikely that Žuti Kamen would share any project infrastructure with these two ELs. The Žuti Kamen EL is therefore considered part of the Tlamino Project.

The Property surrounds the Podvirovi Mining Licence and an EL both held by Bosil-Metal d.o.o., a Serbian subsidiary of UK-registered Mineco Limited, a non-related entity to Medgold.

1.2.1 Property Ownership

The Donje Tlamino and the Surlica-Dukat ELs are held by Medgold Istraživanja d.o.o., a Serbian registered company wholly owned by Tlamino Mining Limited of Malta, a wholly owned subsidiary of MGold International Limited, also of Malta. MGold International Limited is wholly owned by Medgold Resources Corp of British Columbia, Canada. Medgold Istraživanja d.o.o. was formerly wholly owned by Medgold Resource Limited of England and Wales, itself wholly owned by Medgold Resources Corp of British Columbia, Canada. The registration of the above change of ownership of Medgold Istraživanja d.o.o. by Tlamino Mining Limited with the Serbian Business Registry is pending.

1.2.2 Property Agreements

In June 2016, Medgold Istraživanja d.o.o. granted Fortuna the right to enter into an option agreement (the "Original Option") on a selected project in Serbia. In March 2017 Fortuna elected to option the Tlamino Project consisting of the Donje Tlamino and the Surlica-Dukat ELs. Under the terms of the Original Option Fortuna was required to spend US$3.0 million to acquire 51 percent of the Project; this milestone was reached during H2 2019. In January 2020 Medgold, Medgold

Istraživanja d.o.o. and Tlamino Mining Limited entered into a new Option Agreement (the "New Agreement") with Fortuna to acquire Fortuna's 51% interest in the Tlamino Project. The Original Option was terminated under the terms of the New Agreement.

The terms of the New Agreement provide Medgold with an exclusive option (the "Option") to purchase Fortuna's 51% interest in the Project for a cash consideration of US$3.468 million. The Option is valid for three years and is exercisable (i) at any time at the election of Medgold prior to the expiry of the term of the Option; or (ii) at the date of completion of a sale by Medgold of a 100% interest in the Project to a third party; or (iii) at the date of completion of a merger between Medgold and a third party, whichever arises soonest.

In the event that Medgold completes a sale of the Project or corporate merger during the term of the Option and receives consideration in excess of US$8.84 million (the "Sale Consideration"), Medgold will pay to Fortuna an asset sale bonus equal to 10.2% of any amount in excess of the Sale Consideration, less all of Medgold's costs related to the sale or corporate merger. No other consideration is due by Medgold to Fortuna under the terms of the Agreement. The monthly Option Fees referred to in the non-binding letter of intent announced on June 18, 2020 have been struck.

Should Medgold not exercise the Option or complete a sale of the Project or corporate merger within the term of the Option Medgold will transfer its undivided 49% interest in the Project to Fortuna for no consideration, such that Fortuna will then hold an undivided 100% interest in the Project.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Property may be accessed via the town of Bosilegrad, which is located approximately 385 kilometres south of Belgrade. The Tlamino property is 20km south-west of Bosilegrad accessed by the no. 444 secondary road, with an additional 2km of graded gravel tracks leading from route 444 to the Property boundary. Smaller unsurfaced tracks currently provide access to the main prospects on the Property.

Both route 444 and the gravel track to Podvirovi are kept open all year round and only typically close for short periods immediately following heavy snowfall. Other tracks within the Property may be inaccessible due to snow accumulation generally limiting the exploration season to April through November.

Medgold has an operational base in the town of Bosilegrad, which serves a population of around 8,000 in the surrounding municipality and hosts an array of services including a healthcare centre, a police station, primary and secondary schools, postal and banking services, a fuel station, a hotel, plus shops and restaurants. The main industries in the area are forestry and related processing of timber products. There are no significant population centres within the Property boundary.

Power distribution in the area occurs via 110kV to Bosilegrad and 35kv voltage transmission lines running along the Golema River to Podvirovi.

1.4 History

Historical exploration work at the Property may be summarized as follows.

- 1930: limited surface and underground exploration was carried out by a private company at Božilovo Ležište. This work included two trenches, one short shaft and three exploratory adits with total underground development of approximately 250 metres.
- 1950 to 1970: Yugoslav State exploration included geochemical sampling, mapping, geophysical surveys, trenching and drilling. Exploration at the time appears to have focused on base metal exploration.
- 2005 – 2012: parts of the Property are known to have been held under an EL by Dundee Plemeniti Metali d.o.o., which was renamed as Avala Resources d.o.o. prior to being acquired by Avala Resources Ltd., (collectively, "Avala"). While the full details of the EL are not known, work is known to have included multiple stages of stream sediment, soil, and rock sampling from which geochemical anomalies were identified at the Karamanica Prospect and Barje Deposit. Avala work culminated in the drilling of four exploration drill holes, totalling 831.2 metres at Barje during 2010 and 2011. These drill holes did not intersect any significant mineralization.

A number of historical mineral inventory estimates have been completed in respect to the Property, none of which were reported in accordance with NI 43-101 or a similar CRIRSCO (the Committee for Mineral Reserves International Reporting Standards)-aligned reporting codes. Such works include classifications which have no equivalent in NI 43-101.

1.5 Geology, Mineralized Zones on the Property and Deposit Types.

The Property is situated in the western part of the Tethyan orogenic belt, a complicated zone of tectonic units and past magmatic activity related to collisional plate tectonic processes active from the mid-Paleozoic through to the modern day.

The geology of the Property can be roughly split into high metamorphic-grade rocks in the north and east, and low metamorphic-grade rocks in the south and west. Numerous older intrusions, represented by orthogneiss, cut both metamorphic sequences but are limited in the area of the

Property to the higher metamorphic grade rocks. Relatively minor areas of sediments and volcanoclastic sequences are preserved in the Property; these are deposited unconformably on the lower-grade metamorphic sequence. The metamorphic and sedimentary sequences are cut by porphyritic dykes and sills and, in the western part of the Property, by volcanic plugs. Quaternary deposits occur in the Property and include unconsolidated talus on the higher ground and alluvial sediments along valley floors.

The Property hosts identified mineralized zones at the Barje Deposit, the Liska Prospect and the Karamanica Prospect. The Barje Deposit is the most advanced mineralized zone within the Property and is the primary subject of this report.

Historical prospecting at Barje located two main areas of outcropping gold and base metal mineralization. Medgold's drilling has confirmed the continuation of mineralization between, and to the west of the discovery outcrops in an area of 700 metres east-west by 350 metres north-south. The mineralization is controlled by a hydrothermal breccia of up to approximately 20 metres in thickness, following a structure inclined by approximately 18° towards the south. The structure cuts a fault-bounded sequence of schist and conglomerate above a dacite sill intruded along a detachment surface at the top of the Crnook basement. While mineralization is strongest in the hydrothermal breccia, a halo of lower-grade mineralization is also found in the overlying rocks. The hydrothermal breccia contains transported clasts of the local wall-rocks cemented by a matrix of quartz ± calcite/siderite and sulphide minerals, including pyrite, arsenopyrite, sphalerite, galena and more rarely chalcopyrite and tennantite. Grains of electrum up to approximately 50 microns in diameter and containing approximately 60% gold and 40% silver, have been observed microscopically within the higher-grade mineralization. A Mineral Resource Estimate for Barje is presented in section 14 of this report.

1.6 Exploration and Drilling

Medgold has completed a desktop study of the geochemical exploration work undertaken by Avala as well as soil geochemistry, lithogeochemistry and geophysical surveys of its own. The analysis of this work was then used to define drill targets at the Barje, Liska, and Karamanica prospects. Medgold then completed 33 diamond drill holes at the Barje Deposit over 4991.5 meters, which identified gold and silver mineralization with lesser amounts of lead, zinc and copper. An initial Mineral Resource Estimate for Barje is presented as part of this Technical Report.

Medgold drilling at the Liska Prospect included 10 drill holes over 2139.4 meters. While this drilling identified the presence of mineralization, the metal grades returned were not considered to be economically significant, or where potentially economic, were interpreted to be isolated with a lack of demonstrated continuity. Drilling of 10 holes at the Karamanica prospect over 1996.5 meters returned only weak mineralization associated variously with fault zones, dark carbonaceous schists, and the margins of porphyritic intrusions. Several geochemical and geophysical targets remain un-drilled on the Karamanica Prospect and further exploration is warranted.

1.7 Data Quality and Verification

The Qualified Person for Geology and Resources has reviewed the drilling data collected by Medgold and considers them reliable for use in Mineral Resource estimation. The data collection practices are considered in line with current industry best practices, and regional exploration data are considered suitable for the generation of follow up drill targets.

1.8 Metallurgical Testing

Two programmes of metallurgical sampling and testing of material from the Barje Deposit have been completed. The initial programme was undertaken in 2019 with the objective of determining precious and base metal recoveries and arsenic deportment to concentrate; a second programme completed during 2020 comprising mineralogical examination and bench-scale gravity concentration, flotation and leaching tests on a range of samples from Barje.

In the first programme, composites were generated from selected coarse drill core reject samples from HBX and Triple X (“XXX”) material. Both composites were taken from unoxidized/unweathered breccia material containing pyrite, arsenopyrite, sphalerite, galena and more rarely chalcopyrite and tennantite. Grains of electrum up to approximately 50 µm in size, containing approximately 60% gold and 40% silver, were observed microscopically within higher-grade zones of mineralization.

Initial metallurgical testing was used to determine baseline metallurgical performance, including recoveries and reagent consumptions for a range of likely extraction routes. Baseline tests include preliminary cyanidation, bulk sulphide flotation, sequential flotation, and diagnostic leaching. No comminution test work was undertaken.

Baseline cyanidation work targeting gold and silver values only was undertaken using standard bottle roll tests on the HBX and XXX composites. Diagnostic leaching of the HBX and XXX composites indicated little of the gold to be cyanide-soluble, with a high proportion contained in either arsenopyrite or pyrite. Additionally, the bottle roll tests suggested that a high proportion of the silver is potentially occluded in galena, inhibiting cyanide recovery of Ag. Reagent consumption was moderate to high indicating poor process economics for this route. Cyanide leaching was therefore not recommended as a treatment route for the Barje breccias.

Sequential gravity and flotation testing was also performed on the composites, and targeted the production of separate Pb and Zn concentrates, as well as a treatable Au-Ag concentrate by gravity. For the HBX composite, gravity testing resulted in recoveries of 16.5% Au and 4.6% Ag at grades of 48.99 g/t Au and 19.3 g/t Ag into a 0.6% concentrate by mass. For the XXX sample, recoveries of 10.0% Au and 4.4% Ag at grades of 66.59 g/t Au and 30.20 g/t Ag into a 1.5% concentrate by mass were achieved. While neither result is particularly high, the presence of gravity recoverable gold and silver in the form of native gold or electrum was indicated and gravity concentration was recommended for inclusion in future testing for flowsheet development.

Results of the sequential float tests suggest Pb and Zn recoveries to separate concentrates are poor and the concentrates produced are unlikely to be marketable.

Baseline bulk sulphide flotation tests were performed to assess production of a bulk polymetallic concentrate which could then be sold for toll treatment. Gold recoveries to concentrate for baseline rougher test were 88.2% and 90.5% for the HBX and XXX composites respectively, with rougher concentrate grades of 16.8 g/t and 36.5 g/t respectively. Silver recovery for the HBX composite was 88.2% with a concentrate grade of 124 g/t. The XXX composite test resulted in 96.4% silver recovery with a concentrate Ag grade of 379 g/t. Copper, lead and zinc recoveries for the HBX composite were 93.2%, 96.5% and 74.2% respectively, with 95.3%, 91.6% and 91.4% respectively for the XXX composite. Concentrate grades were however deemed too low to be of economic interest. Arsenic grades of 5.86% for HBX and 12.10% for XXX were reported for the bulk concentrate.

A second programme of mineralogical examination and metallurgical testing was performed during 2020. Composites formed from coarse drill core rejects of the “HG Breccia” (HG_BX), “LG Schist” (LG_Sch) and “Partially Oxidized“ (OX) material types were tested. The HG_BX material can be considered a composite of the HBX and XXX rock types tested in the first programme of metallurgical testing, together with other hydrothermal breccias containing sulphide mineralization of similar chemical and mineralogical characteristics. The test work had the objective of producing concentrates suitable for toll treatment by pressure oxidation, Albion process or roasting. Optimal concentrate target grades of 45–50 g/t gold and less than 15% arsenic were set during flowsheet development.

Samples of each composite were subjected to mineralogical examination by QEMSCAN Particle Mineral Analysis and TESCAN Trace Mineral Analysis. The mineralogical composition as determined by QEMSCAN is summarised in Table 1.1.

Table 1.1: Mineral Composition of Samples.

Mineral	Unit	HG_BX	LG_Sch	OX
Copper/silver sulphide	%	0.05	0.05	0.03
Galena	%	0.31	0.1	<0.01
Sphalerite	%	1.1	0.23	0.01
Pyrite	%	4.3	2.3	0.2
Arsenopyrite	%	2.5	1.7	0.2
Quartz	%	50.0	49.9	47.1
Feldspars	%	18.5	14.7	17.3
Micas	%	17.1	21.2	24.2
Calcium carbonate	%	1.8	4.2	0.2
Other non-sulphide gangue	%	4.4	5.4	10.7

X-ray spectra indicated that while many of the gold occurrences contained silver, that this varied widely. The percentage of liberated gold ranged between 35% for the LG_Sch composite to 53% for the HG_BX composite. The unliberated gold in these samples was generally associated with sulphide minerals such as sphalerite, pyrite, galena, and arsenopyrite. Gold occurrences were relatively fine, with projected diameters of between 2 µm and 10 µm. Between 92% and 100% of the gold occurrences observed were either liberated, or associated with particles with high sulphide mineral surface exposures, thus indicating high potential for recovery via bulk flotation. The number of gold occurrences detected for each composite was low. Less than 10% of the gold contained in the sub-samples was thus detected, the remaining gold interpreted as being either refractory or sub-microscopic. Spectrally undetected gold present within sulphide minerals would be considered recoverable to a bulk flotation concentrate.

For the OX composite, gold occurred primarily as liberated gold grains. About 21% of the gold was measured in binary form with non-sulphide gangue as inclusions, and as such would not be expected to be recoverable in bulk sulphide flotation.

Gravity concentration was tested on each rock type, the results of which are shown in Table 1.2

Table 1.2: Results of Gravity Concentration Tests.

Sample	Mass recovery %	Au grade g/t	Au recovery %	Ag grade g/t	Ag recovery %
HG_BX	1.2	65.1	19.5	432	7.4
LG_Sch	0.8	33.1	21.7	136	9.9
OX	0.5	20.2	6.3	992	11.2

Whilst the recovery of gold via gravity concentration from HG_BX and LG_Sch composites was moderate, such recovery from the OX composite was poor, and the grades of all three concentrates were low, indicating that gravity concentration has limited application to these material types. These findings are in line with the TESCAN results in that only a small portion of the gold is present as discrete particles. Flotation after gravity concentration was tested and there was no discernible difference between using gravity concentration followed by rougher and cleaner flotation and using flotation alone. Gravity recovery of gold was therefore not considered to be a process option for Barje.

Flotation in rougher and cleaner stages to a bulk Au-Ag concentrate was also tested. This programme focused on increasing the grade of gold in a bulk concentrate from that achieved during the first programme, while also optimizing recovery.

Rougher and cleaner tests were performed on the HG_BX and LG_Sch material. Bulk rougher concentrates were subjected to two stages of cleaning, as shown in Figure 1.1. The rougher concentrates were reground prior to cleaner flotation, the results of which are summarised in Table 1.3.

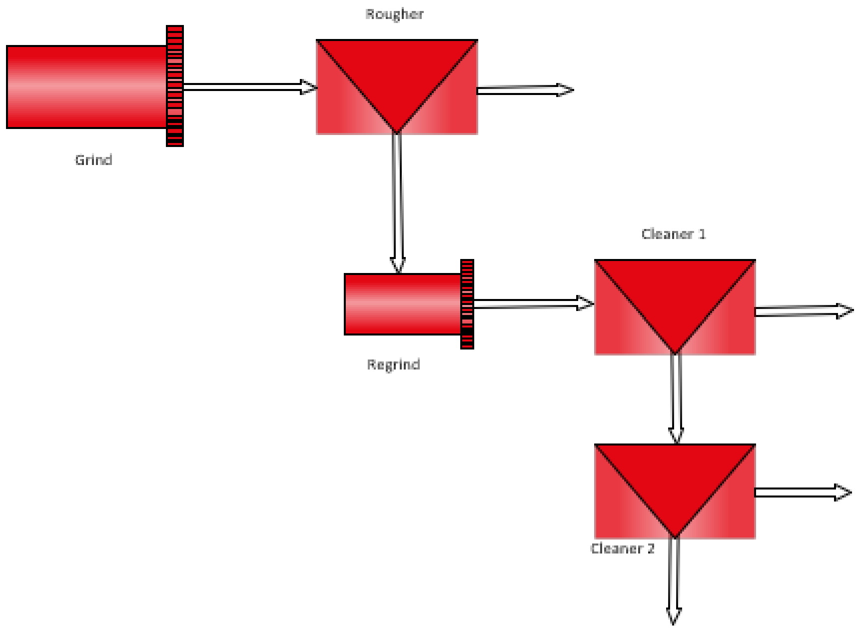


Figure 1.1: Flowsheet of Cleaner Flotation Tests.

Table 1.3: Results of Cleaner Tests.
** Recovery calculated from rougher feed.*

	HG_BX	LG_Sch
Recovery in rougher		
Au%	91.6	88.0
Ag%	93.4	90.5
Cleaner 1 concentrate		
Au g/t	45.9	22.0
Ag g/t	770	199
Cu%	2.81	0.22
Pb%	2.81	1.66
Zn%	8.17	2.75
As%	11.3	16.2
S%	35.3	30.4
Recovery to Cleaner 1 concentrate*		
Au%	85.8	76.5
Ag%	88.8	82.7
Cleaner 2 concentrate		
Au g/t	48.9	24.4
Ag g/t	824	228
Cu%	0.29	0.24
Pb%	3.02	1.94
Zn%	8.80	3.12
As%	11.8	18.0
S%	37.7	34.6
Recovery to Cleaner 2 concentrate*		
Au%	83.4	71.2
Ag%	86.8	79.2

Cleaner flotation testing demonstrated recoveries of 83.4% Au and 86.8% Ag to a sulphide concentrate grading 48.9 g/t Au and 824 g/t Ag and 11.8% As from the HG_BX composite. Recoveries of 71.2% Au and 79.2% Ag to a sulphide concentrate grading 24.4 g/t Au and 228 g/t Ag were demonstrated for the LG_Sch composite. Additionally, cyanide leaching via standard bottle roll tests was evaluated for the OX composite. Tests were performed on coarse drill core rejects (as delivered to the laboratory) and material after grinding to a P_{80} of 78 µm. Results are shown in Table 1.4.

Table 1.4: Results of Cyanide Leaching of OX Material.

Grind size	Gold			Silver		
	Recalc. feed g/t	Leach tail g/t	Extraction %	Recalc. feed g/t	Leach tail g/t	Extraction %
-2 mm	1.62	0.38	76.6	39	15.7	59.5
80% -78 µm	1.59	0.31	80.5	39	7.2	81.6

Bulk sulphide flotation was successful in producing a marketable concentrate from HG_BX and LG_Sch material types but not from the partially oxidized material. Cyanide leaching of the partially oxidized material was moderately successful and, while not considered for this PEA, may warrant further investigation in future studies.

1.9 Mineral Resources

Mineral Resources have been estimated for the Barje Deposit of the Tlamino Project only, and no mineral resources for Liska or Karamanica prospects have been declared. The estimated Mineral Resource for Barje, reported in accordance with NI 43-101 and the CIM Definition Standards, above various break-even cut-off grades for their respective material types is approximately 7.1 Mt at 2.5 g/t Au and 38 g/t Ag in the Inferred category containing 570,000 oz of Au and 8.8 Moz of Ag. This equates to approximately 2.9 g/t AuEq or 670,000 oz AuEq. It is the opinion of the Qualified Person for Geology and Resources that all elements included in the Au Equivalent calculation (gold and silver) have a reasonable prospect of being recovered and sold.

The updated Mineral Resource estimate has an effective date of January 07, 2021 and supersedes the previous initial Mineral Resource estimate, there has been no material change to the Mineral Resource estimate in terms of tonnage, grade and contained metal. See Table 1.5 for further information relating to the Mineral Resource Estimate.

No estimates of Mineral Reserves have been completed. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

The Mineral Resources extend from surface to a depth of approximately 110 m, are laterally extensive over an area of approximately 600 m from east to west and approximately 350 m north to south. Mineralization continues to the west for approximately 100 m where it thins and cover thickness increases, this mineralization is not considered to have a reasonable prospect of economic extraction due to the increased striping ratios and is not included in the Mineral Resource estimate.

The thickness of resource mineralization ranges from approximately 10 to 40 m with some isolated thinner areas. It is closed by bounding faults to the north and south and by drilling to the east and west. Some possibility of identifying additional mineralization via infill drilling in areas where the model is currently interpreted to pinch and in which data are sparse, and in the north-west corner of the area of mineralization both remain. A plan view of the mineralized wireframe models is shown in Figure 1.2 and an example cross section of the resource block model is shown in Figure 1.3.

Additional drilling is required to increase the confidence in the Mineral Resources; increased levels of information brought about by further drilling may serve to either increase or decrease the Mineral Resources.

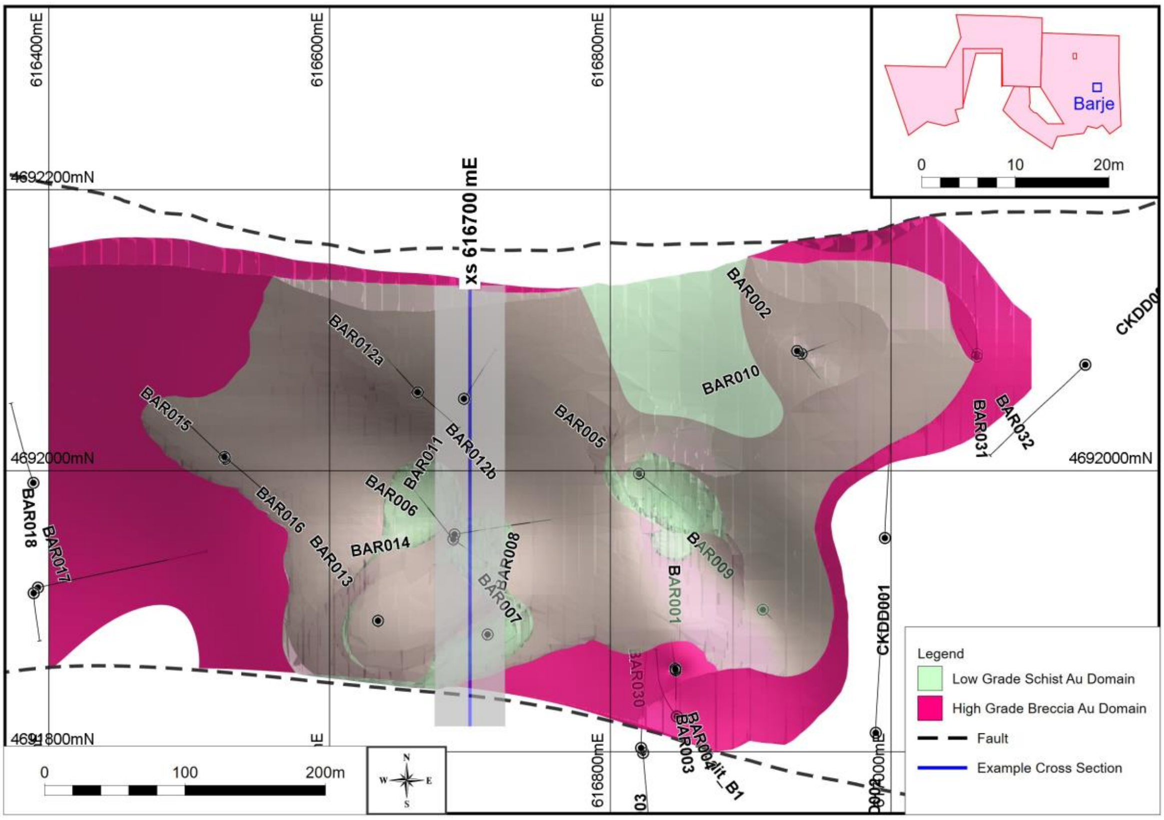


Figure 1.2: Plan view of mineralized wireframes used in resource estimation.
Cross section line shown for 616700 mE.

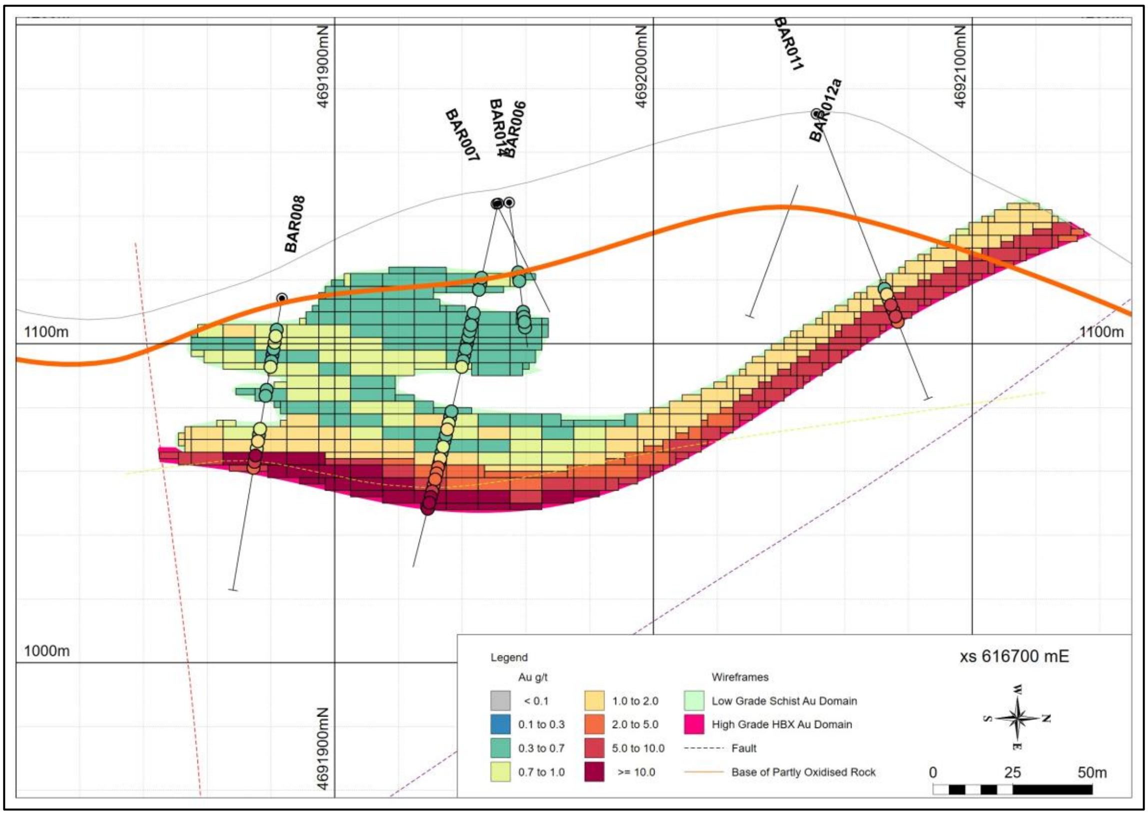


Figure 1.3: Example cross section 616700 mE. Composite data and block model both colour coded to Au grade.

Table 1.5: Mineral Resource Estimate for the Barje Prospect

		AuEq		Au		Ag	
Tonnes	Density	g/t	Contained oz	g/t	Contained oz	g/t	Contained oz
Total Inferred Resources							
7,100,000	2.7	2.9	670,000	2.5	570,000	38	8,800,000
Including							
High Grade Breccia							
3,200,000	2.8	4.7	470,000	3.9	400,000	65	6,700,000
Low Grade Schist							
2,400,000	2.7	1.2	96,000	1.1	88,000	8.4	650,000
Partially Oxidized Material							
1,500,000	2.5	2.1	100,000	1.7	87,000	29	1,400,000

Notes to the Mineral Resource Estimate:

1. The independent Qualified Person for the Mineral Resource Estimate, as defined by NI 43-101, is Mr. Richard Siddle, MSc, MAIG, of Addison Mining Services Ltd since November 2014. The effective date of the Mineral Resource Estimate is January 07, 2021.
2. These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The quantity and grade of reported Inferred Resources in this Mineral Resource Estimate are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as Indicated or Measured, however it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

3. Mineral Resources reported in the above table are presented as undiluted and in-situ for an open-pit scenario and are considered to have reasonable prospects for economic extraction. The Mineral Resources constrained by open pit optimisation.
4. Break even cut-off grades were estimated for each material type of 0.6 g/t, 0.8g/t and 0.5 g/t AuEq for the High Grade Breccia, Low Grade Schist and Partially Oxidized materials respectively, these cut-off grades were used in Resource Reporting. The cut-off grades were calculated on the basis of the following assumptions: a gold price of US$1500/oz, a silver price of US$16.5/oz, mining costs of US$2.3/t, processing costs including tailings disposal of US$10/t for sulphide rock and US$12/t for oxide, G&A costs of US$4/ROMt and transport costs of US$2/ROMt.
5. Per metallurgical test work completed to date, recovery to concentrate after flotation of 85.8% for gold and 84.3% for silver were used for the High Grade Breccia material with 75% payability. For the Low Grade Schist recoveries used were 76.5% for gold and 82.7% for silver with 60% payability. For the Partially Oxidized material 80% recovery via leaching for gold and silver was assumed with 98% payability. 5% gross royalty was applied to both metals.
6. Geological and block models for the Mineral Resource Estimate used data from 33 surface drillholes performed by Medgold in 2018 and 2019; data from four drillholes completed by Avala Resources Ltd., a prior operator, were used to constrain the model though they did not intercept significant mineralization. The drill database was validated prior to resource estimation and QA/QC checks were made using industry-standard control charts for blanks, core duplicates and commercial certified reference material inserted into assay batches by Medgold and by comparison of umpire assays performed at a second laboratory. No QA/QC was possible on the data relating to the drilling by Avala.
7. The geological model as applied to the Mineral Resource Estimate comprises two mineralized domains, a shallowly inclined high-grade hydrothermal breccia unit and a lower-grade schist unit immediately overlying the hydrothermal breccia. Individual wireframes were created for each domain. Weathering domains of fresh and partially oxidized material were defined within the two mineralized domains.
8. The block model was prepared using Micromine version 2020, Services Pack 1, A 10 m x 10 m x 4 m block model was created with sub-blocks of minimum 2 m x 2 m x 2 m on domain boundaries. Grade estimation from drillhole data was carried out for Au, Ag, As, Cu, Pb, Zn, Fe, S using Ordinary Kriging and was validated by comparison of input and output statistics, kriging neighbourhood analysis and by inspection of the assay data and block model in cross section. A gold equivalent (AuEq) grade was calculated for each block using the formula AuEq = ((Ag g/t) x 0.011)) + (Au g/t) for the High Grade Breccia and Partially Oxidized materials and AuEq = ((Ag g/t) x 0.012)) + (Au g/t) for the Low Grade Schist.
9. Bulk density values were calculated for each block of the model based on a broad linear relationship observed between 152 measured bulk density values within the mineralized domains and the assayed values of As, Cu, Fe, S, Pb and Zn. Blocks within the partially oxidized material were assigned a single bulk density value of 2.54 g/ cm^3.
10. Estimates in the above table have been rounded to two significant figures.
11. CIM Definition Standards for Mineral Resources have been followed.
12. The independent Qualified Person for Resources is not aware of any additional known environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues that could materially affect the Mineral Resource Estimate.

1.10 Mining Methods

The Barje Deposit is relatively thick, flat-lying and situated under shallow to medium-depth overburden. Initially both low-cost open pit, as well as underground methods with higher selectivity were considered, however open pit methods were preferred on account of the overall low stripping ratio and generally low RQD of the rock mass Mining via open pit methods using hydraulic excavators and wheel loaders charging articulated dump trucks for haulage of both waste and potentially economic material is therefore projected. Mining activities at Barje will include free-digging of the weathered zones, the blasting of fresh rock, and loading, hauling and dumping of the respective materials, plus mining support activities. The removal and stockpiling of topsoil will be performed prior to mining.

1.10.1 Mine Design

A preliminary geotechnical characterization was undertaken, including logging of resource drill holes geotechnically for RQD, plus visual observation of cores for validation. RQD is generally low to moderate, with values ranging from 20% to 70%, although higher RQD was determined for the calcareous schists. Preliminary overall slope angles of between 37° (West) and 41° (East) were selected. No testing or modelling of pit hydrogeology has been completed to date, however reasonable provisions for pit water management including perimeter dykes and diversion ditches, in-pit water collection ditches, and in-pit pumps and collection systems to transfer water from the open pits to discharge points for settling, and potentially treatment prior to discharge, have been made.

1.10.2 Pit Optimisation

Pit optimisation was undertaken using Datamine Studio NPVS (“NPVS”) and SimSched DBS (“DBS”) software packages. This two-stage process was adopted in order to establish the maximum economic pit limits and the optimal mining sequence within the pit limit. It was recognised that the standard Lerchs-Grossman optimisation methodology of determining the pit limit and mining sequence based on pit shells cannot adequately account for blending constraints or stockpiling with respect to the LG_Sch material. However, by applying a blending constraint in the optimisation stage of the DBS run it was possible to search for alternative mining sequences that maximise value whilst stockpiling the LG_Sch material. The output from DBS is a series of period surfaces (annual in this case) that represent the optimal mining sequence. These surfaces have been used to manually develop the pit stages, which were then imported back into NPVS to produce a hybrid solution that includes the optimised DBS mining sequence as user-defined pushbacks and the optimised NPVS pit limit. This pushback sequence was then re-scheduled in NPVS and the mineral inventory reported.

The Mineral Resource block model was prepared using Micromine software for pit optimisation with the addition of country rock waste blocks to extend beyond the limits of the pit optimisation. The dominant rock type was written to the block model and the mean bulk density values, as estimated from exploration drilling, were applied to each rock type; the bulk density values for the mineral domains were preserved. The block model was then regularized to 5 mE, 5 mN and 2.5 mZ, prior to being exported from Micromine (*.dat) to Datamine (*.dm) format. The block size was selected to represent the minimum Selective Mining Unit (SMU) with the chosen mining equipment. The small block size selection in the Z direction was particularly important for Barje as the mining method needs to allow for mining to the contacts between the LG_Sch and the HG_BX, as well as between the HG_BX and waste on the footwall contact.

1.10.2.1 Optimisation Parameters

The pit optimisation In NPVS was unconstrained by mining boundaries or other physical boundaries. To generate the pit shells the mining rate was set to 600 Ktpa, and a discount rate of 8% was applied. The pit optimisation in DBS also assumed no physical boundaries, a mining rate of 600 Ktpa and a discount rate of 8%. In addition, a constraint was set on blending to limit the ratio of lower-grade LG_Sch to HG_BX mined in the first four years to less than 25%.

Mining costs were estimated at 2.3 US$/t mined based on benchmarking against other contract mining operations in the Balkan region, as well as costing information obtained by Medgold. It was assumed that a significant proportion of the deposit can be mined with either no, or minimal blasting due to the low rock strength. If required, the fresh material will be ripped or lightly blasted with a low powder factor in order to increase productivity. The processing cost was assumed to be between 10.0 US$/t and 14.5 US$/t processed: initial optimisation studies in DBS used the higher value of 14.5 US$/t which was subsequently reduced to 10.0 US$/t later during the study.

Modifying factors for mining recovery and waste dilution were assumed to be accounted for through regularization of the resource block model to a standard block size representative of the selective mining unit. No further factors have been applied.

The selected block size was 5 m x 5 m x 2.5 m and was shown to yield similar tonnage and grade factors to a 5 m x 5 m x 5 m model. The tonnage and grade factors when compared to the un-regularised model are in the order of +5 to +6% in tonnage and -6 to -9% for grade. These factors are considered reasonable for a flat lying deposit with a height of the mineralised section of around 20 to 30m.

Metallurgical factors are specified by dominant material type (HG_BX or LG_Sch) and are calculated as a weighted average where there is a mix of materials in a block (Table 1.6).

Table 1.6: Metallurgical Factors.

ROCK	Units	Au Recovery	Ag Recovery
HG_BX	%	85.8	84.3
LG_Sch	%	76.5	82.7

It was assumed that the oxidised material is stockpiled for the future. No economic value is assigned to this material in the pit optimisation.

Payability factors were assigned for optimisation on the assumption that there will be two separate concentrate streams, one from processing HG_BX (and mixed material) and one from processing LG_Sch (and mixed material). Payability factors were 75% for HG_BX and 60% for LG_Sch, taking into account all downstream costs, including refining. Note that while a 60% payability for LG_Sch was used for the base case pit optimisation, a payability of 40% was used during final economic analysis.

An allowance of 2.0 US$/t Run of Mine (ROM) was allowed for concentrate transport costs to the port.

Other financial parameters considered by modelling included the long-term metal price forecast for Au and Ag, General and Administration (G&A) costs, and Royalties (Table 1.7).

Table 1.7: Metal prices, G&A and Royalty used in pit optimisation.

Parameters	Units	Au Concentrate		Ag Concentrate
Metal Price	US$/oz	1,500		16.5
G&A	US$/t ROM	4.0		4.0
Royalty	%	5.0		5.0

1.10.2.2 Optimisation Results

The DBS schedule optimisation was constrained by the ratio of LG_Sch to HG_BX to limit the LG_Sch processed in the first 4 years to less than 25%. This was achieved by stockpiling up to 1 Mt of low-grade material over the active mining life of 6 years, with two years of subsequent stockpile reclaim. The resulting mining volumes, with smoothed plant feed is shown in Figure 1.4.

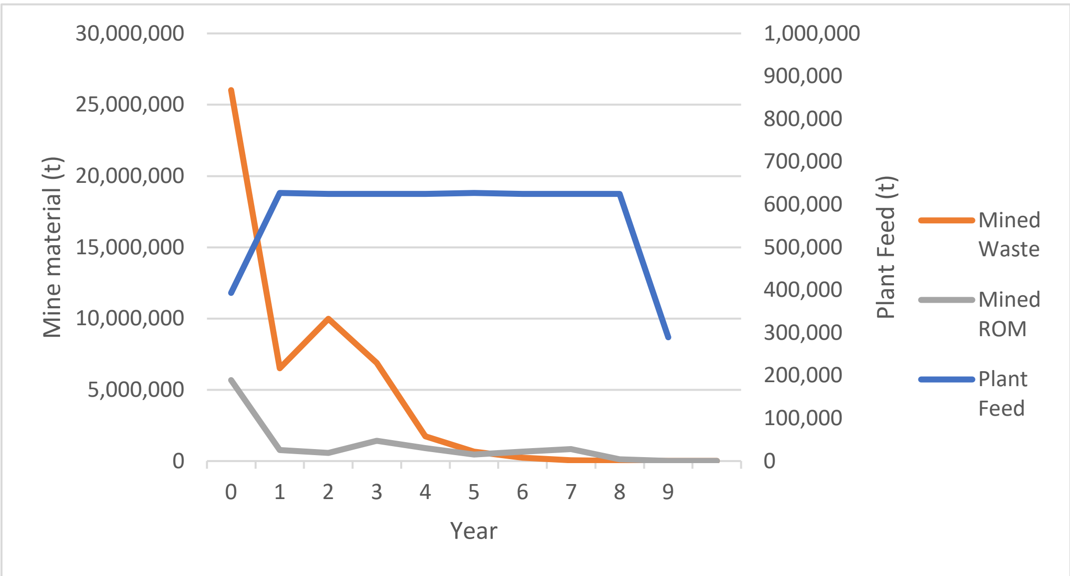


Figure 1.4: ROM production tonnes by period

Mined grades for HG_BX, LG_Sch, and blended plant feed, for both Au and Ag is shown in Figure 1.5.

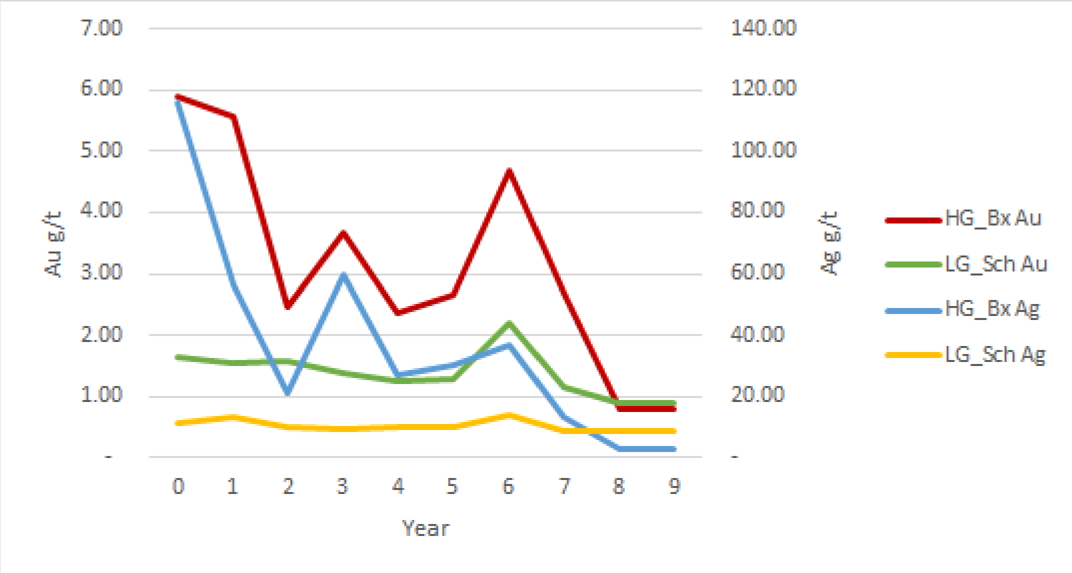


Figure 1.5: Processed grade by period.

ROM is transported from the north of the pit to the low and high grade stockpile areas, then reclaimed for plant feed. Waste rock would be transported to the north of the pit to for storage in the valley to the north. The waste haul for most of the benches will be relatively short and can take advantage of the fact that haul routes may the contours of the hill. This greatly simplifies the ramp systems as there is limited need to establish a permanent ramp system for either plant feed or waste, other than when the pit is well-developed, and mining is below the pit rim elevation at the southern pit exit. At a Price Factor of 1.0 the total in pit mineral inventory is 6.1 Mt @ 2.49 g/t Au and 35.6 g/t Ag. Of this 3.4 Mt @ 3.53 g/t Au and 65.1 g/t Ag is High Grade Breccia and 2.5 Mt @ 1.17 g/t and 9.6 g/t Ag is Low Grade Schist. The overall waste to ROM strip ratio is 4.6:1.

Although the overall pit limit produced by NPVS was considered valid, the mining sequence generated by it for the LG_Sch material was not necessarily optimal were blending constraints on processing the LG_Sch taken into account. In this case it was shown that the DBS mining sequence was better able to limit the amount of LG_Sch mined in the early periods when compared to the solution using NPVS alone.

Although the target of less than 25% LG_Sch plant feed cannot be met without stockpiling at least some of the LG_Sch, it is evident in the NPVS schedule that a large amount of LG_Sch will need to be stockpiled, where this is significantly less with the DBS-generated mining sequence. It is for this reason that the DBS mining sequence for the first 4.6 Mt was chosen to represent the initial mining sequence, whilst the NPVS ultimate pit limit (Price Factor = 1.0) was selected to maximise resource recovery and extend mine life as much as possible (whilst still meeting economic criteria on profitability).

1.10.3 Pit Design

Using the annual surfaces generated by DBS a series of pit expansions (Pit Stages or Pushbacks) were created to follow the general DBS mining sequence and account for the minimum mining width (35m) and other practicalities of mine planning.

The pit limit was selected from the NPVS analysis (Price Factor = 1.0) and was divided up into four Stages with at least six months of plant feed in each stage. This ensured that the vertical advance rate in each stage was kept to below 90m per year, which is regarded as optimal with the selected mining equipment. The general layout of the stages is shown in Figure 1.6 to Figure 1.9.

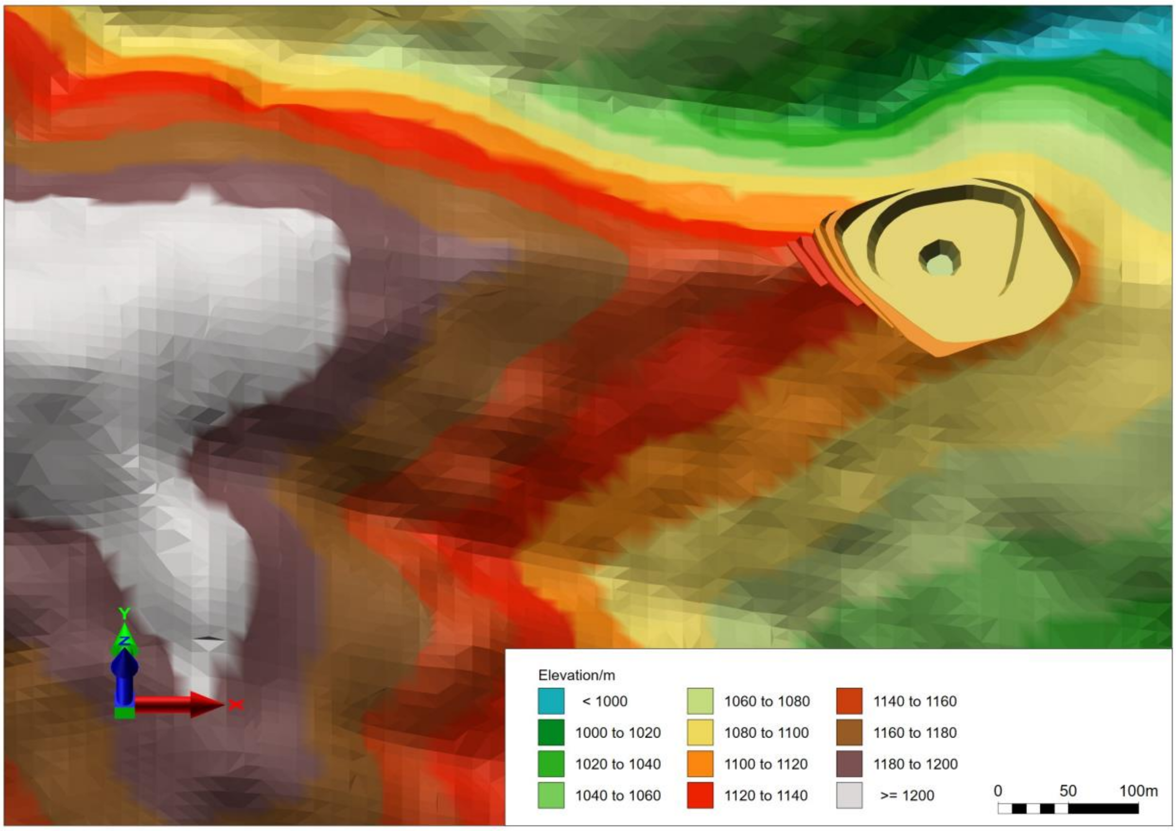


Figure 1.6: Development of Pit Stage Designs Pushback 1

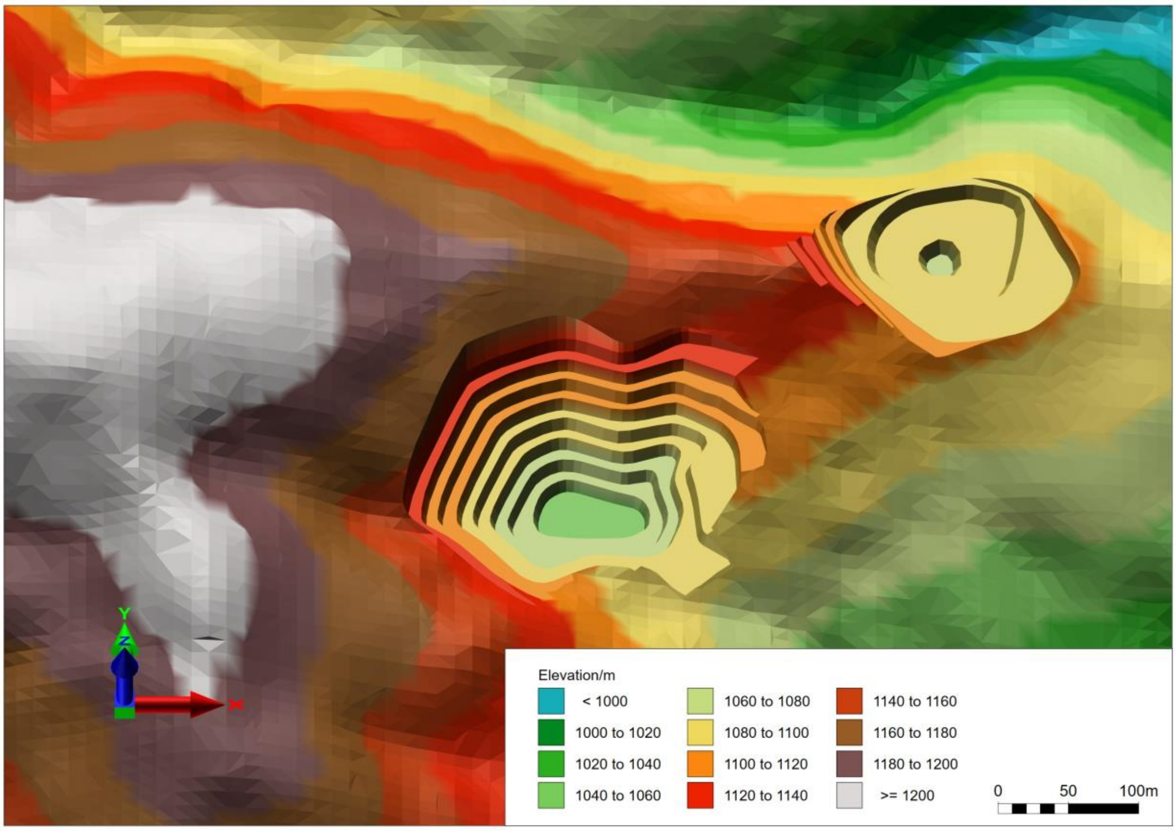


Figure 1.7: Development of Pit Stage Designs Pushback 2

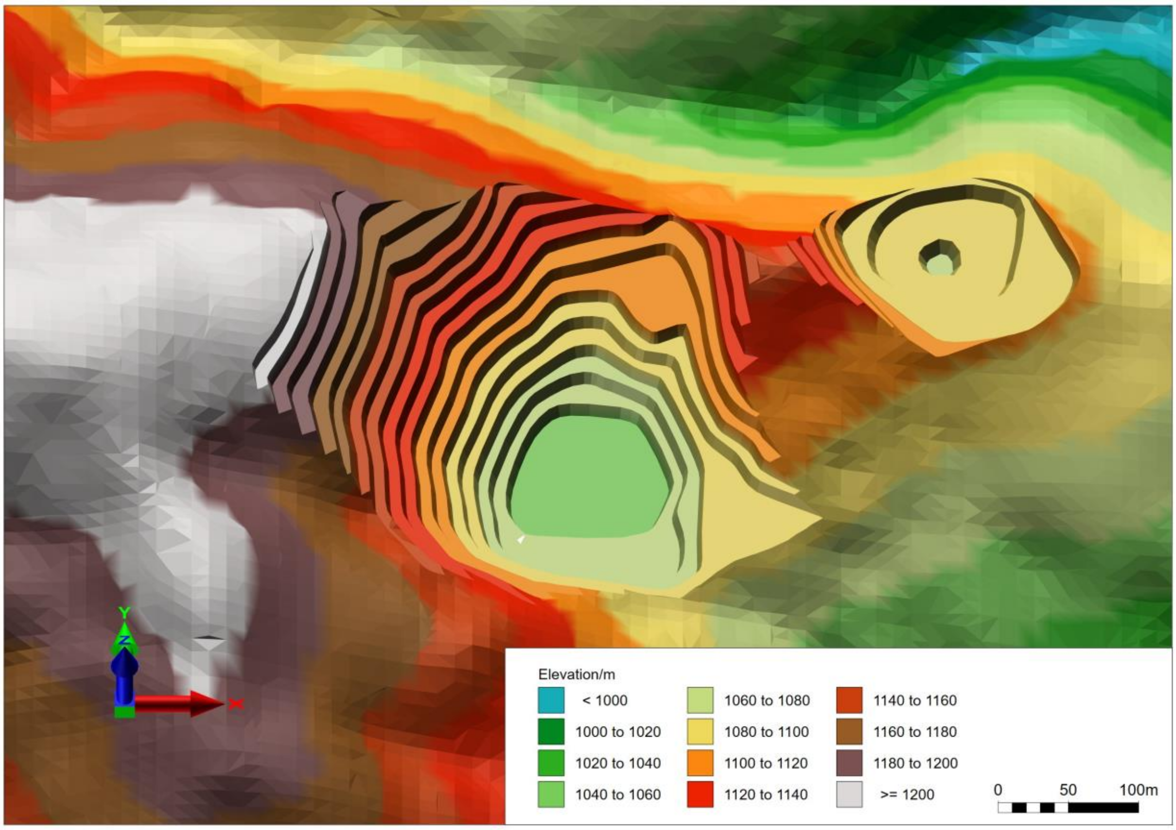


Figure 1.8: Development of Pit Stage Designs Pushback 3

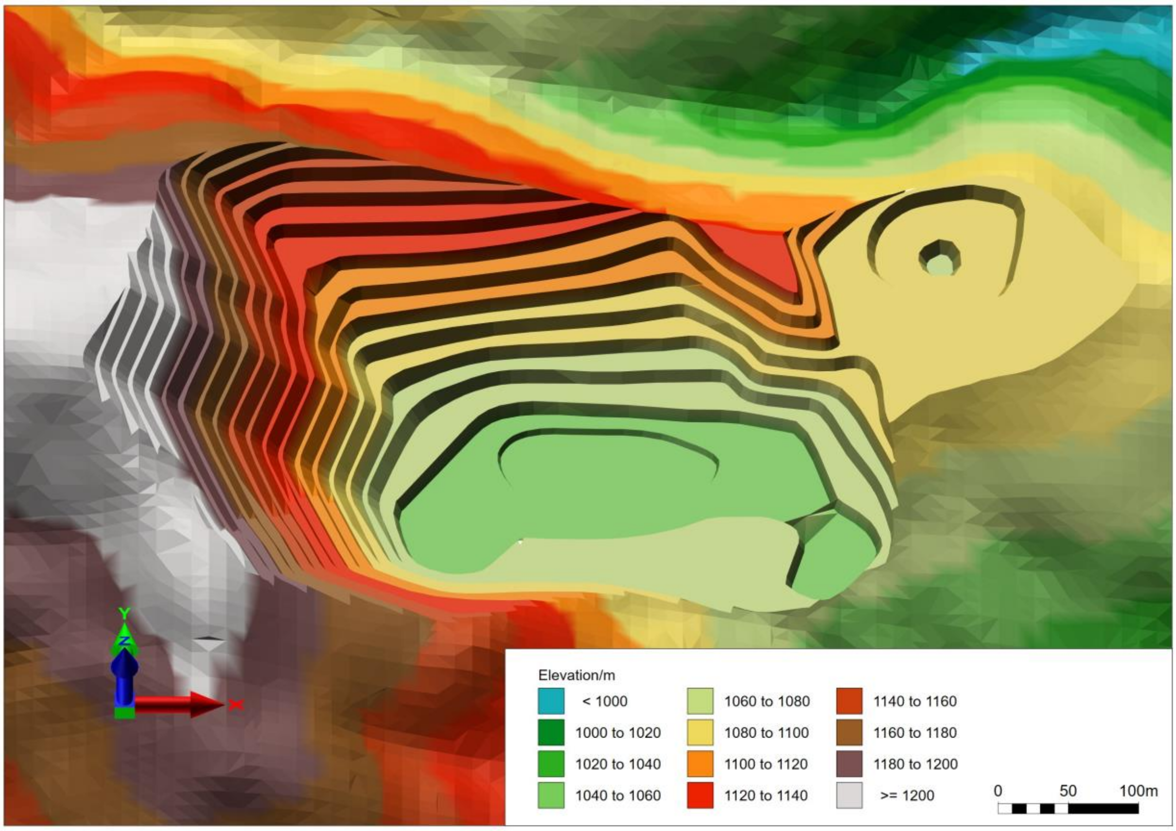


Figure 1.9: Development of Pit Stage Designs Pushback 4

1.10.4 Mine Schedule

Due to the relatively short mine life the pit stages were scheduled in periods of three months to ensure that the mine capacity was smoothed out as much as possible and that the stockpile levels were controlled in order to provide some smoothing of the grade profile.

It should be noted that while the total rock movement appears to peak in Period 1 this is due to the schedule commencing mid-way through Period 0 and there being a relatively short period of the peak mining rate (thirty months excluding pre-strip) with a rapid decline in production rate from Period 4 onwards. The total mine life is almost eight years with two years of stockpile reclaim at the end.

The resulting annualised schedule is summarised in Table 1.8.

Table 1.8: Annualised Mine schedule.
Note: Scheduling Periods are 12 months with a Pre-strip period of 3 months included in Period 0.

Mining Summary	Units	Total	Year									
			0	1	2	3	4	5	6	7	8	9
Total Rock	**t**	**31,700,000**	**7,270,000**	**10,600,000**	**8,300,000**	**2,620,000**	**1,090,000**	**878,000**	**876,000**	**121,000**	**0**	**0**
Total Waste	**t**	**26,000,000**	**6,500,000**	**9,980,000**	**6,880,000**	**1,730,000**	**650,000**	**227,000**	**51,600**	**932**	**0**	**0**
Total ROM	**t**	**5,690,000**	**766,000**	**569,000**	**1,420,000**	**892,000**	**443,000**	**652,000**	**824,000**	**120,000**	**0**	**0**
Plant Feed (All)	**t**	**5,690,000**	**393,000**	**627,000**	**625,000**	**625,000**	**625,000**	**627,000**	**625,000**	**625,000**	**625,000**	**289,000**
	oz Au	480,000	55,000	80,000	45,000	55,000	36,000	44,000	94,000	43,000	17,000	8,000
	oz Ag	7,100,000	1,500,000	1,200,000	470,000	1,300,000	630,000	670,000	740,000	340,000	160,000	75,000
	g/t Au	**2.6**	**4.4**	**4.0**	**2.3**	**2.7**	**1.8**	**2.2**	**4.7**	**2.1**	**0.86**	**0.86**
	g/t Ag	**39**	**120**	**62**	**23**	**63**	**32**	**33**	**37**	**17**	**8.1**	**8.1**
Plant Feed (HG_BX)	**t**	**3,570,000**	**251,000**	**379,000**	**479,000**	**369,000**	**317,000**	**418,000**	**619,000**	**401,000**	**231,000**	**107,000**
	oz Au	390,000	48,000	68,000	38,000	44,000	24,000	36,000	93,000	35,000	5,900	2,700
	oz Ag	6,400,000	940,000	680,000	320,000	710,000	270,000	400,000	730,000	170,000	21,000	9,500
	g/t Au	**2.6**	**4.4**	**4.0**	**2.3**	**2.7**	**1.8**	**2.2**	**4.7**	**2.1**	**0.86**	**0.86**
	g/t Ag	**39**	**120**	**62**	**23**	**63**	**32**	**33**	**37**	**17**	**8.1**	**8.1**
Plant Feed (LG_Sch)	**t**	**2,110,000**	**142,000**	**248,000**	**146,000**	**256,000**	**308,000**	**208,000**	**5,870**	**224,000**	**394,000**	**182,000**
	oz Au	85,000	7,400	12,000	7,400	11,000	12,000	8,600	410	8,300	11,000	5,300
	oz Ag	670,000	50,000	110,000	46,000	79,000	96,000	68,000	2,700	65,000	110,000	49,000
	g/t Au	**2.6**	**4.4**	**4.0**	**2.3**	**2.7**	**1.8**	**2.2**	**4.7**	**2.1**	**0.86**	**0.86**
	g/t Ag	**39**	**120**	**62**	**23**	**63**	**32**	**33**	**37**	**17**	**8.1**	**8.1**
Stockpile IN	**t**	**2,430,000**	**568,000**	**259,000**	**804,000**	**337,000**	**70,700**	**144,000**	**250,000**	**0**	**0**	**0**
Stockpile OUT	**t**	**2,430,000**	**196,000**	**317,000**	**8,780**	**70,000**	**253,000**	**119,000**	**50,800**	**505,000**	**625,000**	**289,000**
Cum Stocks	**t**		**373,000**	**315,000**	**1,110,000**	**1,380,000**	**1,190,000**	**1,220,000**	**1,420,000**	**914,094**	**289,094**	**0**

1.10.5 Waste Storage and Stockpiles

A total waste rock storage capacity of 26 Mt is required. This can be contained within the valley to the north of the pit and will entail a relatively short haul from the upper benches of the stages by developing haul routes to the north around the contour of the hill.

The initial toe of the Waste Rock Storage Facility (WRSF) will need to be established with a compacted foundation keyed into the bedrock. The WRSF can then be developed by backfilling the valley from east to west with lifts of 10m and face angle of 25°. Catch berms will be left at 10m intervals in order that a final profile angle of less than 18° can be obtained by dozing down the faces during rehabilitation.

It should be possible to progressively rehabilitate the WRSF over time with surface topsoil that has been stockpiled separately for this purpose. It may also be possible to consider in-pit disposal of waste once Stage 1 has been mined out, and further in-pit disposal could take place once Stage 3 is mined out.

A water diversion system will be required to divert surface run-off away from the WRSF as the catchment area at the head of the valley is substantial.

The low-grade stockpile will be built up during the life of the mine as while a combination of high and low grade material is processed in the early stage of the life of the mine, and it is estimated that the maximum capacity will be around 1.5 Mt.

The low-grade stockpile should be placed as close as possible to the plant to reduce costs. There is an area to the south east of the pit, next to the southern pit exit, that would provide sufficient capacity for this material. The stockpile has been designed with a face angle of 35° and berm width of 5m on each 10m lift. The stockpile designed capacity, as shown in Figure 1.10, is in excess of 2 Mt.

The general layout of the proposed WRSF and low-grade stockpile, relative to the proposed open pit, is shown in Figure 1.10.

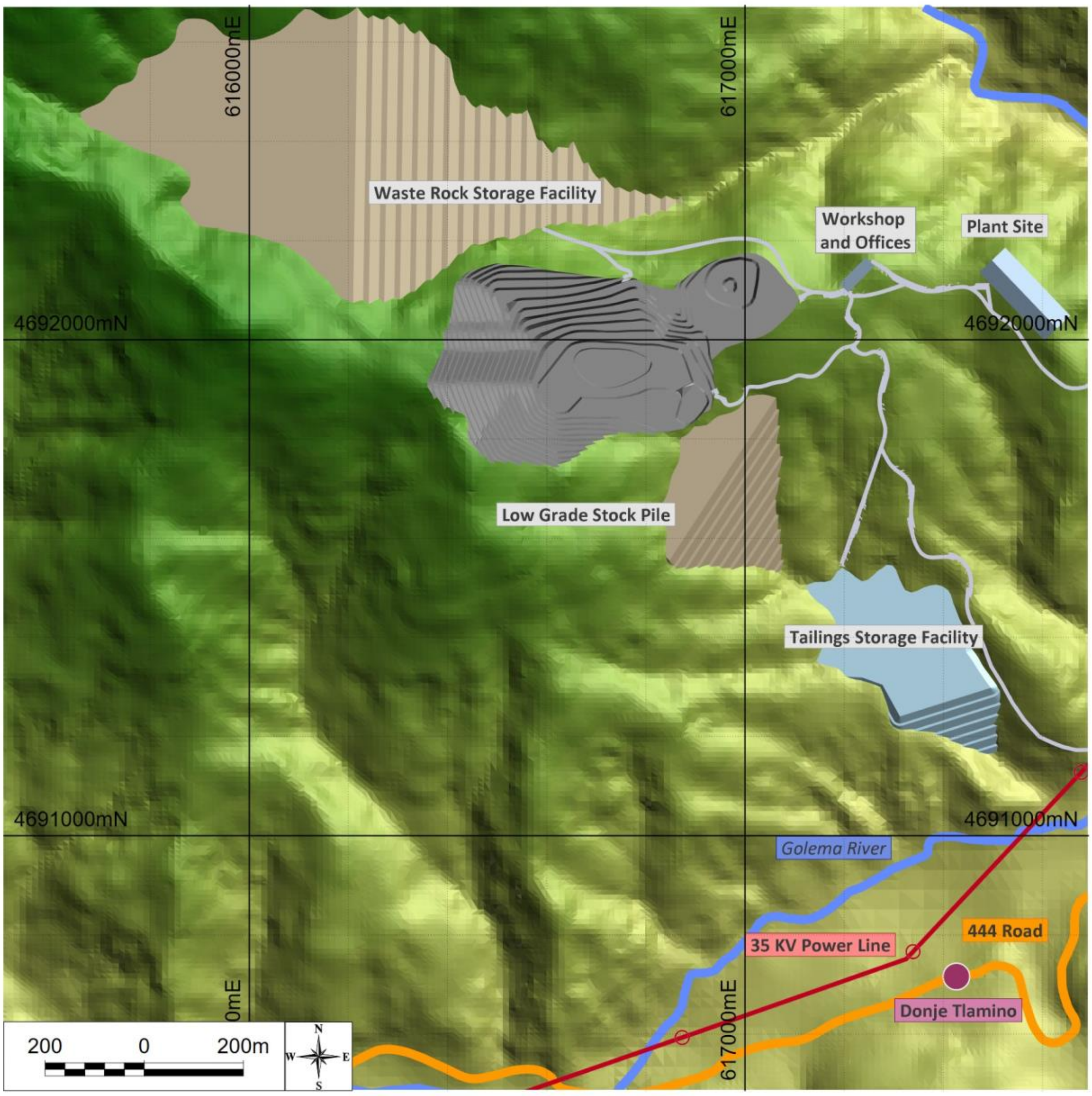


Figure 1.10: Plan View of Pit Limit and location of the WRSF and LG Stockpile

1.10.6 Mining Fleet Requirements

It is expected that the mine will be operated by a mining contractor and the final selection of equipment will be left up to the contractor. However, there is a need to ensure that certain areas of the mine can be mined selectively with 5m, or even 2.5m flitches; a relatively small hydraulic excavator is required for this task. Conversely, the short mine life (< eight years) and stripping ratio of around 4.6:1 requires a peak mining rate of approximately 29,000 tpd to be maintained for at least two to three years. It is recommended that a mix of two to three smaller excavators (3 - 5 m^3 bucket) and one Front End Loader are used in order to give some flexibility and to allow for reclaiming from a ROM stockpile at times.

Based on an average cycle time of 20 minutes for waste and 15 minutes for potentially economic material it is expected that the maximum haulage fleet requirement will consist of 18 x 40 t articulated dump trucks. The haulage times at Barje are relatively short due to the flat lying nature of the deposit and limited pit depth meaning that the majority of the haul routes are on the flat or downhill.

The primary mining operations will be supported by a fleet of support equipment consisting of dozers, graders, water trucks, as well as maintenance and service vehicles. A list of major and support equipment is provided in Table 1.9.

Table 1.9: Fleet Requirements

Equipment	Size/Model	Scheduling Period (Years)										
		0	1	2	3	4	5	6	7	8	9	10
Excavator	5 m^3	3	3	3	1	1	1	1	1			
Wheel Loader	5 m^3	1	1	1	1	1	1	1	1	1	1	
Haul Truck	40 t ATD	14	18	15	6	4	4	4	3	2	2	
Backhoe		1	1	1	1	1	1	1	1			
Track Dozer	D9	1	1	1	1	1	1	1	1			
Grader	12’ Blade	1	1	1	1	1	1	1	1	1	1	
Rubber Tyre Dozer		1	1	1	1	1	1	1	1	1	1	
Water truck	30,000 l	1	1	1	1	1	1	1	1	1	1	
Fuel & Lube		1	1	1	1	1	1	1	1	1	1	
Service truck		1	1	1	1	1	1	1	1	1	1	
Crane	Grove 40 t	1	1	1	1	1	1	1	1	1	1	
Forklift		1	1	1	1	1	1	1	1	1	1	
Welding truck		1	1	1	1	1	1	1	1	1	1	
Personnel van		1	1	1	1	1	1	1	1	1	1	
Pickup truck		4	6	6	6	6	4	4	3	2	2	
Lighting Plants		4	6	6	6	6	6	4	4	4	2	

1.10.7 Mine Labour

Mine labour has been estimated on the basis of two shifts per day covered by a 4-crew roster. The majority of this workforce will be provided by the mining contractor. In addition, there will be a small management team and technical services will be required to manage the mining contractor (Table 1.10).

Table 1.10: Manpower Requirements

Role	Contractor/ Owner	Scheduling Period (Years)										
		0	1	2	3	4	5	6	7	8	9	10
Mine Manager	Both	2	2	2	2	2	2	2	2	2	2	
Mine Supervisor	Contractor	8	8	8	8	8	8	8	8	8	8	
Operators	Contractor	148	180	168	124	116	108	100	92	68	60	
Admin staff	Both	8	8	8	8	8	8	8	4	4	2	
Chief Surveyor	Contractor	1	1	1	1	1	1	1	1			
Surveyors	Contractor	4	4	4	4	4	4	4	2			
Samplers	Contractor	4	4	4	4	4	4	4	2			
Maint Mgr.	Contractor	2	2	2	2	2	2	2	2	2	2	
Mechanics	Contractor	32	32	32	32	24	16	8	8	8	8	
Tyre Bay	Contractor	8	8	8	8	8	8	4	4	4	4	
Chf Geologist	Owner	1	1	1	1	1	1	1	1			
Geologists	Owner	8	8	8	8	8	8	8	4	2	2	
Tech Serv Mgr	Owner	1	1	1	1	1	1	1	1			
Engineers	Both	8	8	8	8	8	8	8	4			
Total		235	267	255	211	195	179	159	135	98	98	

1.11 Recovery Methods

Recovery of gold is via grinding and flotation to a saleable bulk Au-Ag concentrate. The concentrate obtained from the LG_Sch material is of lower grade than that from the HG_BX material and payability of metal content is likely to be lower. The two material types will be processed in the same concentrator but at different times, i.e., on a campaign basis, in order to maximize revenue from the higher-grade material. Laboratory test work has shown that the same grind size and flotation parameters are applicable to both material types and can result in commercially viable products.

Run-of-mine (ROM) material is hauled by trucks and tipped on a storage and blending stockpile. This will facilitate campaigning of the lower-grade material.

Stockpiled ROM is reclaimed by front-end loader and tipped into a bin. A vibrating grizzly feeder (VGF) extracts ROM from the bin and scalps coarse material which is fed to a jaw crusher. Undersize from the VGF joins the jaw crusher discharge and is conveyed to a double deck vibrating screen. Oversize from the top deck is conveyed to a secondary cone crusher and oversize from the lower deck is conveyed to a tertiary cone crusher. The cone crushers are adjacent to the jaw crusher and located above the conveyor which collects the VGF undersize, and the combined crusher discharges are returned to the vibrating screen. A nominal undersize d80 of -10 mm screen has been assumed.

Undersize from the vibrating screen is the final product from the crushing circuit and is conveyed to a storage bin.

Crusher product is extracted from the bin by feeders which discharge onto a conveyor which delivers the material to a ball mill. While no comminution testing has been performed, ball milling with feed prepared by three-stage crushing and screening is assumed as this represents a robust option for this material type. The mill operates in closed circuit with hydrocyclones, the cyclone underflow returning to the mill and the overflow advancing to flotation. A mill circuit product that has 80% passing 80 µm is required, as used in the flotation test work.

Overflow from the mill cyclones enters an agitated tank in which the slurry is conditioned with flotation reagents. Based on the laboratory yest work, potassium amyl xanthate (PAX) is used as a sulphide collector and methyl isobutyl carbinol (MIBC) as a frother.

A rougher flotation stage followed by two stages of cleaner flotation are sufficient to produce acceptable concentrate. Concentrate from the rougher cells is reground before it passes to cleaner cells for upgrading. The rougher concentrate is pumped to a hydrocyclone, the underflow of which is fed to a regrind ball mill while the overflow advances to cleaner flotation. Concentrate slurry from the cleaner cells passes to a recleaner cell. Recleaner concentrate is final concentrate while recleaner tailing enters the first cleaner cell with the rougher concentrate. Cleaner tailing is returned to the rougher. In the laboratory tests the cleaner tailing did not return to the rougher as only single batch tests were performed and, while it is assumed for this PEA that returning the cleaner tail is not detrimental to rougher flotation and will therefore have a positive effect on recovery, this should be confirmed through locked cycle tests. PAX and MIBC are added to the cleaner stages as required.

Recleaner concentrate is the final product. Water is first removed in a conventional thickener; the thickened slurry being stored in an agitated tank before dewatering further by means of a pressure filter. Concentrate filter cake is stored and blended in a shed before transport off site by road. The water recovered by the thickener and filter is returned to the mill circuit.

The rougher flotation tailing is densified in a high-rate thickener, the underflow being stored in an agitated tank before final dewatering by means of a pressure filter. Tailings are deposited in a dry-stack type Tailings Storage Facility (TSF). Thickening and pressure filtration have been used to improve the geotechnical properties and reduce environmental impacts, including maximum recycling of water. Filtered tailings are trucked to the TSF.

1.12 Project Infrastructure

Barje is located 20km south of the town of Bosilegrad, via the 444 sealed road. From route 444, existing gravel tracks lead to the site. The 444 route is serviced by existing 35 kV powerlines running parallel to the road, from Bosilegrad to the town of Podvirovi to the West. The site is flanked by existing rivers, a non-perennial drainage to the North, and the perennial, transboundary Golema river to the south.

Site infrastructure suitable for a 600,000 tpa ROM open pit mine is planned and costed in the present study. New haulage-standard roads will be established from the existing gravel road junction with route 444, up to the site from the south, passing the pit and stockpile sites, and through to the ROM pad at the plant site. Additional roads to the waste rock storage facility and the tailings storage facility location will also be established. Main power will be teed from the existing 35kV powerline along the 444 road to Podvirovi with a 35kV/1000V transformation station and sub-station established near the plant site. A backup power line at 35kV is planned per Serbian mining regulations. Electricity will be reticulated to other main consumers at 1000V.

The main mobile equipment workshop, mine office and changehouse will be located near the pit ramp exit. Structures will comprise steel frameworks with brick walling, sheeted, insulated roofs and standard finish for interiors. Communications and control will be by pervasive WiFi. Pit dewatering will be by semi-permanent submersible pump stations delivering mine water to a common settling pond on surface prior to discharge.

Plant site infrastructure will include a main office, workshop and store with the plant motor control centre and control room located above. A small laboratory for assay/metal accounting and QA/QC will be provided, along with a weighbridge for concentrate accounting. Structures again will be steel framed with brick walling, sheeted insulated roofs and standard fittings.

Fresh water will be sourced from groundwater and pumped to a freshwater tank for use. Potable water will be treated by reverse osmosis prior to distribution. Fresh water will also be pumped to the process water tank for process water make-up. Brown water from changehouses and washrooms in the offices will be routed to a packaged bio-disc sewage system for treatment prior to discharge.

Non-contact surface water will be routed around terraces and structures by suitable berms and culverts for discharge to pre-existing drainage channels. Contact surface water together with excess mine water pumped from the open pit will be collected, settled, and treated if required prior to discharge to pre-existing site drainage.

Tailings storage will be via dry-stack method, with the filter plant located at the plant site and dry tailings cake loaded by FEL and transported by ADT to the TSF for shaping and compaction. Water recirculation will be direct from the tailing thickener and filter to the process water tank for re-use.

1.13 Environmental and Social

The Serbian legal and permitting context, current understanding of site settings and proposed further studies are described, together with initial identification of potential issues and impacts.

The legal framework for mining in Serbia was updated with a new Mining Law in 2015 which has increased the efficiency of the permitting process but with remaining complexity. The framework is aligned with EU regulations and includes a formal Environmental and Social Impact Assessment (ESIA) procedure with minimum 12 months baseline for data collection. A Certificate of Reserves is a prerequisite to obtain the main permits required for mining in Serbia, namely, an Exploitation Field Permit, Mine Works and Facilities Construction Permits, and Approval for Use Permit. The Certificate of Reserves and Exploitation Field Permit both require ESIA Scoping level responses and conditions for Cultural Heritage and Water-approval studies. A permit register with application submission dates, authorising bodies and expiry dates is recommended to avoid permitting delays.

Some initial baseline studies have started at the Barje site, including a surface and groundwater sampling exercise completed by Medgold in 2017, together with desktop investigations and compilation of existing and publicly available data. Additional physical data on surface water quality, air quality and noise are available from baseline studies in the adjacent property at the Mineco Podvirovi deposit <5 km away.

There is an effective weather monitoring network in Serbia from a data collection perspective, however a site weather station will need to be established in order to monitor local variations. Ambient air quality and noise levels are generally good in this rural location, though the area is seismically active with a "medium" earthquake hazard classification. The geology of the area and deposit is generally well understood and mineralogical data for the waste rock, different potential ore types and likely tailings material are being gathered. The target rocks are sulphide/pyrite rich, carbonate poor and with some elevated arsenic content, with potential for acid rock drainage and metal leaching that will each require detailed geochemical investigations and test work.

Monitoring data have been complied for stream sediments, surface- and groundwater quality and shows surface water pH as being above neutral, though with some groundwater samples slightly acidic. Some background exceedances of national limit values were found for arsenic and nickel. Desktop information has been gathered on soils and land-use, surface water hydrology, biodiversity, vegetation, fauna and protected areas, archaeology and cultural heritage, and social aspects with demographics and economic activity. The socio-economic data show general depopulation and economic decline in the area.

Although at an early project stage, Medgold has been engaging with the municipality and local communities since the start of exploration activities, and has a Community Policy, Stakeholder Engagement guidelines, Project Disclosure systems, a Community Assistance Programme and a Grievance Mechanism. A conceptual Closure Plan with cost estimate will need to be developed for submission with the application for the Mine Building Permit together with a bank guarantee.

1.14 Economic Analysis

Preliminary economic analysis has been undertaken for the Barje Deposit of the Tlamino Project. Such analysis considers revenue based on the preliminary mining schedule presented in section 16, metal recoveries as presented in section 13, market factors as presented in section 19 and capital and operating costs as presented in section 21. Estimates and analysis are to scoping level (+-30%).

1.14.1 Capital Costs

Capital costs for mine development, mine infrastructure, process plant, and surface infrastructure including mine offices, control, plant building, common workshop and stores, changehouse, water, powerline and substation, and earthworks including tailings, roads and platforms were estimated. Estimates have been made based on current designs and quotes from recent similar projects by Bara Consulting. Plant capital includes for the design and construction of a 600,000 tpa flotation plant including crushing, grinding, froth flotation, concentrate and tailings handling facilities including filtration of tailings for dry stacking. Infrastructure includes for mine support infrastructure, plant infrastructure, dry stack tailings storage facility, power (including backup 35kV line), water and internal roads. A summary is presented in Table 1.11. Estimates for closure were also assessed during the ESIA review process.

Table 1.11: Capital Cost Estimates.

DESCRIPTION	QTY	UNIT	TOTAL COST (US$M)
Mine Development	3.25	Mt	7.48
Process Plant	600,000	tpa	34.6
Surface Infrastructure			14.0
Indirect Costs	15	%	8.41
Contingency	15	%	9.67
Total			74.2

1.14.2 Operating Costs

A high-level breakdown of operating costs is provided. Estimates have been made based on current designs and quotes from recent similar projects by Bara Consulting. Mine operating costs include ore mining and waste mining at US$2.30/t, and a stockpile reclaim cost for low-grade material of US$1/t equating to US$0.50/ROM tonne. Process costs include crushing, grinding, flotation, concentrate handling and tailings handling (including filtration) for 600,000 tpa flotation feed. G&A includes on-mine administration and general costs. Concentrate transport is costed for delivery of concentrate CIF to customers in China. Details are presented in Table 1.12 below.

Table 1.12: Operating Cost Estimates.

DESCRIPTION	UNIT	COST / UNIT (US$)
MINING		
Mining cost - ROM	t	2.80
Mining cost - Waste	t	2.30
PROCESSING		
Processing	t	11.50
Conc Transport (per ROM t)	t	3.24
General and Admin	t	5.80

1.14.3 Economic Analysis

The economic analysis presented in this report is preliminary in nature and is based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is therefore no certainty that the preliminary economic assessment presented here will be realized.

A long-term Au price of US$1500/oz and a silver price of US$16.50/oz per London Bullion Market Association (LMBA) data were used (LBMA, 2020). Metal payability is 75% for HG_BX and 40% for LG_Sch material types, both net of treatment charges. A summary is presented in Table 1.13.

Table 1.13: Barje Project Key Financial Metrics

Revenue	**458**	**US$M**
Operating Cost	181	US$M
Project Capital Cost	74	US$M
Free Cashflow	153	US$M
LOM C1 Cash Cost	464	US$/oz
LOM AISC	522	US$/oz
Pre-Tax Project NPV8	101	US$M
Post-Tax Project NPV8	86	US$M
Pre-Tax Project IRR	49	%
Post-Tax Project IRR	46	%
Operating Margin	61	%
Peak Funding Requirement	37	US$M
Payback Period	2.0	years

The NPV of the project, at a discount rate of 8%, is US$101M with an IRR of 49%. The operating margin, describing an extremely robust project, is 61%. Upfront capital is US$63M, plus US$10M contingency, with peak funding of US$37M and a payback of two years. Life of Mine C1 cash costs of US$464/oz, and LOM AISC of US$522/oz, would place the project - if operating - in the current lowest quartile cost of global gold production.

1.15 Recommendations

Recommendations include commencing infill drilling work to support a potential conversion of Inferred to Indicated Resources, and field programmes in support of a potential Preliminary Feasibility Study (PFS) on the Barje Deposit. This work would include additional geotechnical and hydrogeological investigation, additional metallurgical testing and commencement of environmental base line studies including air, water, soil, fauna, and flora.

1.15.1 Geology and Resources

Infill drilling on approximately 60 metre centres is recommended to potentially convert Inferred Resources to Indicated Resources at the Barje Deposit. On this basis, approximately 3200 metres of drilling is recommended; results of the drilling should be periodically re-evaluated during the programme to test confidence and appropriateness of drill spacing. Dedicated drillholes for geotechnical and hydrogeological purposes would ideally be completed while the drill rig is on site to avoid remobilisation costs.

Potential exists for a laterally faulted offset of the Barje mineralization to be present on the EL, and additional geological mapping and interpretation may assist in exploring for a faulted continuation of the mineralization. Drilling to date at the Karamanica Prospect has been limited to targets identified mainly by interpretation of geochemical and geophysical datasets. A reassessment of the Karamanica Prospect, considering the additional information gained by drilling in 2019, may lead to additional targets on the Prospect which, if present, would require additional drill testing.

Additional mineralogy and paragenetic studies may also be useful in understanding the genesis of the deposit which may bolster metallurgical studies and improve the understanding of the larger mineralized systems in the area.

1.15.2 Mineral Processing and Metallurgical Testing

Further metallurgical testing is required to support a Pre-Feasibility Study (PFS). Standardised comminution tests will be required to provide data to determine the most effective methods for primary comminution and flotation concentrate regrinding and to enable preliminary sizing of equipment. Additional flotation test work should be completed on composites of the HG_BX and LG_Sch material types in order to evaluate grade and metal concentration vs recovery relationships. Test work on one or more composite samples of both material types is also recommended to evaluate the effects of blending and material mixing during mining on metallurgical response.

Further testing of the OX material by cyanide and cyanide-free leaching (e.g., thiosulphate) warrants further investigation, as does scout testing of biohydrometallurgical and oxidizing leach processes for all material types.

1.15.3 Mining

No immediate mining-specific studies are required for the subsequent recommended work programmes, however information and data that are collected in other suggested work programmes should be incorporated into any future mining studies. This includes further work relating to analysis of the slope stability using geotechnical data from the orientated drill core in combination with hydrogeological data to account for the water table and other hydrogeological parameters.

Metallurgical modelling of the various material types should be improved provided appropriate further metallurgical test work has been completed. Consideration should be given to grade vs. recovery across both main material types and mixed material types. Blending the HG_BX and LG_Sch material types should also be evaluated with respect to final concentrate grade, following which a re-evaluation of mining cut-off grades and the mine schedule should be undertaken; detailed scheduling per month is required to confirm the pre-strip requirements. Formal offtake studies should also be pursued which may also impact the mine schedule and cut-off grades.

1.15.4 Environmental and Social

PFS reporting requires evaluation of project impacts based on results of the baseline studies for the initial permitting process. The baseline studies may not have a full 12 months of monitoring data at the time of PFS, however initial field surveys should be completed. It is therefore recommended to commence ESIA study programmes and baseline data gathering including air, water, soil, fauna and flora studies, urbanization mapping and community consultation, plus development of an impact management programme.

1.15.5 Indicative Budget for Further Work.

An indicative Phase 1 budget for an exploration programme with the intention of converting the majority of Inferred Resources to Indicated Resources is presented in Table 1.14; allowances are included for regional exploration drilling as well as dedicated geotechnical and hydrogeological drillholes and consulting fees to complete a Mineral Resource update. Based on favourable results from Phase 1 it is advised to proceed with Phase 2 recommendations which include additional metallurgical testwork and the commencement of environmental baseline data collection, an indicative budget for which is outlined in Table 1.15. It may be practical to undertake some components of Phase 1 and 2 in parallel, for example initiating low-cost environmental baseline data collection during the next field season.

Table 1.14: Phase 1 indicative costs for additional drilling and MRE update.
Exchange rates; EUR1 = CAD1.5 or USD1.2

Item	Units	Unit Cost	Sub-totals		
			EUR	CAD	USD
Infill diamond drilling	3,200 m	75	240,000	360,000	288,000
Target testing diamond drilling	2,000 m	75	150,000	225,000	180,000
Geotech and Hydro Drilling	600 m	100	60,000	90,000	72,000
Assays of drill core	4,500	55	247,500	371,000	297,000
Staffing – core yard technicians	3 months	6,000	18,000	27,000	22,000
Staffing – geology and professional	6 months	25,000	150,000	225,000	180,000
Overheads, vehicles, core yard, rental	6 months	10,000	60,000	90,000	72,000
Land access and groundworks	1	30,000	30,000	45,000	36,000
Consulting and MRE update	1	50,000	50,000	75,000	60,000
		Sub-total	1,006,000	1,509,000	1,207,000
		Contingency (10%)	101,000	152,000	121,000
		Total	1,107,000	1,661,000	1,328,000

Table 1.15: Phase 2 indicative costs for additional metallurgical testwork and commencement of environmental baseline data collection.
Exchange rates as per Table 1.14.

Item	Units	Unit Cost	Sub-totals		
			EUR	CAD	USD
Additional Scout Metallurgical Tests	1	50,000	50,000	75,000	60,000
Commence EBLS Data collection	12 months	5,000	60,000	90,000	72,000
Consulting and Advisory	1	50,000	50,000	75,000	60,000
		Sub-total	160,000	240,000	192,000
		Contingency (10%)	16,000	24,000	19,000
		Total	176,000	264,000	211,000

2 Introduction

This Technical Report was prepared by Addison Mining Services ("AMS") for Medgold Resources Corp. (the "Issuer', "Medgold") of Suite 650-200 Burrard St. Vancouver BC, Canada V6C 3L6. Medgold is the indirect 49% owner of the Tlamino Project ("Tlamino", the "Project") to which this report relates. Fortuna Silver Mines Inc ("Fortuna") is the direct owner of 51% of Project having met the terms of an earlier option agreement, now cancelled. In January 2020 Medgold subsidiaries entered into a new Option Agreement with Fortuna to acquire Fortuna's 51% interest in Tlamino, details of which are presented in Section 4.4 of this Technical Report.

This Technical Report has been compiled by the following independent Qualified Persons (QP).

- Mr Richard Siddle – MSc, MGeol (Hons), FGS, MAIG, Director and Senior Consultant Geologist, Addison Mining Services Ltd. – QP Geology and Resources
- Dr Andrew Bamber – BSc, MASc, PhD, P.Eng, MCIM, Director and Principal Consultant, Bara Consulting Ltd. – QP Financial Analysis
- Mr Ian Jackson – BEng, FIMMM, CEng, Principal Process Engineer, Jackson's Unique Mineral Processing Services Ltd. – QP Mineral Processing
- Dr Matthew Randall – BSc (Hons), PhD, MIMMM, CEng, Principal Mining Engineer, Axe Valley Mining Consultants Ltd. – QP Mining
- Dr Sue Struthers – BSc, MSc, PhD, FIMMM, CEnv, Principal Environmental Consultant, Skapa Mining Services Ltd. – QP Environmental and Social

The study is based on findings of a site visit by AMS, desktop study, data review, data validation, deposit modelling, block model grade interpolation, Mineral Resource estimation, metallurgical testing, pit optimisation, conceptual mine planning and scheduling, cost estimation and preliminary economic analysis.

A site visit was conducted to the Tlamino Project between November 11 and November 14, 2019, by Mr. Siddle as part of a previous study, the purpose of which was to inspect the Property, core processing procedures and to confirm the presence of mineralization. Due to restrictions relating to the COVID-19 pandemic in force during the course of this study, no other QP was able to visit the site. Medgold has not undertaken any additional exploration activities since the site visit completed by Mr Siddle in November 2019.

The Mineral Resources estimated, and the Preliminary Economic Assessment undertaken as part of this study have been reported in accordance with the *National Instrument 43-101 – Standards of*

Disclosure for Mineral Projects (NI 43-101) and *The CIM Definition Standards on Mineral Resources and Reserves (CIM Definition Standards)*. The report has been prepared to be filed on SEDAR in support of a News Release by the Issuer dated 26/01/2021 in which the findings of the Preliminary Economic Assessment were reported.

2.1 Terms of Reference

AMS was commissioned by Medgold to undertake a Preliminary Economic Assessment with respect to the Barje Deposit, and to prepare a NI 43-101 Technical Report in regard to the Tlamino Project.

2.2 Independence

AMS is an independent geological and mining consultancy based in the United Kingdom. The Qualified Persons for this study, neither have nor hold:

- any rights to subscribe for shares in Medgold Resources Corp. either now or in the future,
- any vested or unvested interests in any concessions held by Medgold,
- any rights to subscribe to any interests in any of the concessions held by Medgold, either now or in the future,
- any vested or unvested interests in either any concessions held by Medgold or any adjacent concessions,
- any right to subscribe to any interests or concessions adjacent to those held by Medgold either now or in the future.

AMS' only financial interest is the right to charge professional fees at normal commercial rates, plus normal overhead costs, for work carried out in connection with the investigations reported herein. Payment of professional fees is neither dependent on project success nor on project financing.

2.3 Units

All units of measurement used in this report are SI metric unless otherwise stated. Tonnages are reported as metric tonnes (t), precious metal values for gold (Au) and silver (Ag) in grams per tonne (g/t) or parts per million (ppm). Other references to geochemical analysis are presented in parts per million (ppm), parts per billion (ppb) or percentage (%) as a function of weight. All ounces are reported as Troy ounces. Millions, where relevant, are designated by the use of 'M'.

Location data were captured and located using the Universal Transverse Mercator (UTM) format. The coordinate system used by the client was UTM Zone 34N WGS84 (EPSG 32634).

2.4 Sources of Information and Data

In the preparation of this Technical Report the QPs have relied upon data provided by Medgold which the QPs have taken steps to verify. Verification work relating to geology and resources is described in Section 12 of this report.

Information relating to the background, history, geology and exploration practices of the Tlamino Project described in Sections 4 to 11 and adjacent properties described in Section 23 has been sourced from a report (Sant, 2019) prepared by Mr Thomas Sant of Exploration Direction Ltd. Mr. Sant has provided consulting services to Medgold since 2017.

Information relating to Mineral Processing and Metallurgical Testing has been sourced from testwork completed by ALS Kamloops (ALS, 2020), and was supervised remotely by the QP for mineral processing.

Information relating to Environmental Studies, Permitting and Social or Community Impact has been sourced from the public domain or supplied by Medgold and reviewed by the QP for environmental and social.

Information relating to operating, capital and transport costs and concentrate payability and specification has been obtained from previous, similar projects or projects in the geographic vicinity, or has been provided by Medgold and reviewed by the QPs for mineral processing, mining, and economic analysis.

Additional sources of information are cited within the document as appropriate; a full list of references is given in Section 27.

2.5 Limitations

In the preparation of this Technical Report the QPs have utilized information provided by Medgold Resources Corp. The QPs have made every reasonable attempt to verify the accuracy and reliability of the data and information received in said information, and to identify areas of possible error or uncertainty. To the best of the QPs knowledge said information contains no omission likely to affect the success of the Project. The QPs take responsibility for the technical content of this report.

This technical report contains the results of Preliminary Economic Assessment based on Inferred Mineral Resources. The Preliminary Economic Assessment is preliminary in nature, Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the Preliminary Economic Assessment will be realized.

The businesses of mineral exploration, development, mining, and production are by their nature associated with significant risks. The success of such projects is dependent on many factors including, but not limited to the size and grade of Mineral Resources, and mining, metallurgical, geotechnical, operational, legal, environmental, marketing, metal pricing, and transportation systematics. This report is based upon metal price history and forecasts at the time of writing. The nature of the mineral exploration, development, mining, and production businesses is such that many factors may be subject to change over relatively short periods of time and as such actual results may be differ significantly.

The interpretations and conclusions presented in this Technical Report are based on the most current and up to date data available to the Qualified Person at the time of writing, however the results are estimates and are subject to change. The QPs make no claim of absolute certainty and as such any financial, economic or investment decisions based on the interpretations and conclusions found in this report will carry an element of risk.

2.6 Forward-looking Statements

Certain statements contained in this Technical Report constitute forward-looking statements within the meaning of Canadian securities legislation. All statements included herein, other than statements of historical fact, are forward-looking statements and include, without limitation, statements about the exploration plans for the Tlamino Project. Often, but not always, these forward looking statements can be identified by the use of words such as “estimate”, “estimates”, “estimated”, “potential”, “open”, “future”, “assumed”, “projected”, “used”, “detailed”, “has been”, “gain”, “upgraded”, “offset”, “limited”, “contained”, “reflecting”, “containing”, “remaining”, “to be”, “periodically”, or statements that events, “could” or “should” occur or be achieved and similar expressions, including negative variations.

Forward-looking statements involve known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements of Medgold to be materially different from any results, performance or achievements expressed or implied by forward-looking statements. Such uncertainties and factors include, among others, the exploration plans for the Tlamino Project; changes in general economic conditions, commodity prices and financial markets; Medgold or any joint venture partner not having the financial ability to meet its exploration and development goals; risks associated with the results of exploration and development activities, estimation of Mineral Resources and the geology, grade and continuity of mineral deposits; metallurgical recovery; geotechnical and hydrological conditions; unanticipated costs and expenses; and such other risks detailed from time to time in the Company’s quarterly and annual filings with securities regulators and available under Medgold's profile on SEDAR at www.sedar.com. Although Medgold and the QPs have attempted to identify important factors 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 to differ from those anticipated, estimated or intended.

Forward-looking statements contained herein are based on the assumptions, beliefs, expectations and opinions of Medgold and the contributing authors to this report, including but not limited to; that the proposed exploration of the Tlamino Project will proceed as recommended in this Technical Report; that there will be no material adverse change affecting Medgold or its properties; and such other assumptions as set out herein. Forward-looking statements are made as of the date hereof and the QPs disclaim any obligation to update any forward-looking statements, whether as a result of new information, future events or results or otherwise, except as required by law. 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, investors should not place undue reliance on forward-looking statements.

3 Reliance on Other Experts

The Qualified Person has not independently verified title to Medgold's assets, nor has it verified the status of Medgold agreements with local landowners and relevant parties but has relied on information supplied by Medgold in this regard. The authors are relying on public documents and information provided by Medgold for the descriptions of title and status of the Property agreements as described in section 4 of this report and as discussed in person during the site visit completed by Mr. Siddle (QP). The QPs have no reason to doubt that the title situation is other than that which was reported to it by Medgold and as described in this report.

4 Property Description and Location

4.1 Property Location

The Property is located in southeast Serbia (Figure 4.1) and falls principally in the district of Pčinja, and the municipalities of Bosilegrad and Trigovište. The central coordinates of the Project are approximately 609500mE/4693000mN (WGS84 UTM34N), or 22.3° east and 42.4° north.

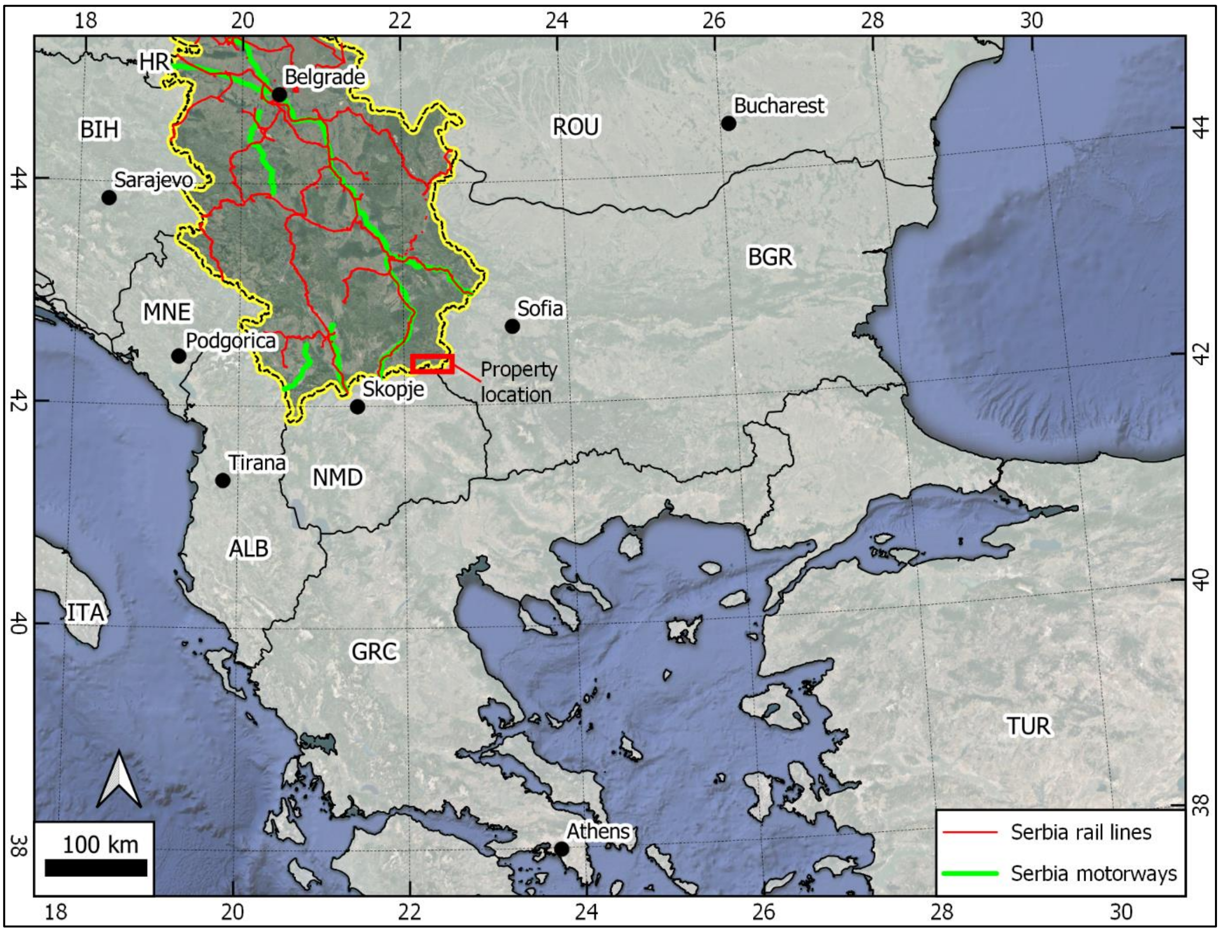


Figure 4.1. Property location.
Property location is outlined in red with Serbia outlined in yellow. ROU – Romania, BRG – Bulgaria, TUR- Turkey, GRC – Greece, NMD – North Macedonia, ALB – Albania, MNE – Montenegro, BIH – Bosnia and Herzegovina, HR – Croatia.

4.2 Property Description

Medgold holds three Exploration Licenses ("ELs") in south eastern Serbia, forming a contiguous block totalling approximately 199 km^2 (Table 4.1 and Figure 4.2). Two of the ELs – namely Donje Tlamino and Surlica-Dukat – form the Tlamino Property (the "Project") and cover a total area of 192.63 km^2. Medgold acquired Donje Tlamino and Surlica-Dukat in October 2016. The Donje Tlamino EL and the Surlica-Dukat EL, were re-issued to Medgold Istraživanja d.o.o on September 30, 2020, by the Serbian Ministry of Mining and Energy (the "MME") following completion of Medgold's initial three-year work-programme ending 31 October 2019 for each area and approval for the subsequent work three-year programmes was granted.

The Project is subject to an Option Agreement with Fortuna, described in section 4.4. As shown in Figure 4.2, the Property surrounds the Podvirovi Mining Licence and an EL both held by Bosil-Metal d.o.o., a Serbian subsidiary of UK-registered Mineco Limited, a non-related entity to Medgold. See section 23 for further information on adjacent properties.

The Žuti Kamen EL was awarded to Medgold by the MME in March 2019, however Medgold was only notified of this decision in October 2020. As such, no exploration work has been completed on the Žuti Kamen EL and it is not subject to technical review in this report. Žuti Kamen did not form part of the Original Option (see Section 4.4) and locations of known mineralised zones on the Donje Tlamino and Surlica-Dukat ELs suggest it is unlikely that Žuti Kamen would share any project infrastructure with these two ELs. Therefore, the Žuti Kamen EL is not considered part of the Tlamino Project.

Table 4.1. Details of Medgold’s Exploration Licences.

Licence Name	Licence No.	Holder	Received	End of Y3	Commodities	Area (km^2.)
Donje Tlamino	310-02-01245/2020-02	Medgold Istraživanja d.o.o	12-Oct-2020	12-Oct-2023	Au, Ag, Cu, Pb, Zn, Mo	97.51
Surlica-Dukat	310-02-01245/2020-02	Medgold Istraživanja d.o.o	12-Oct-2020	12-Oct-2023	Au, Ag, Cu, Pb, Zn, Mo	95.12
Žuti Kamen	310-02-034/2017-0	Medgold Istraživanja d.o.o	15-Mar-2019	15-Mar-2022	Au, Ag, Cu, Pb, Zn, Mo	6.15

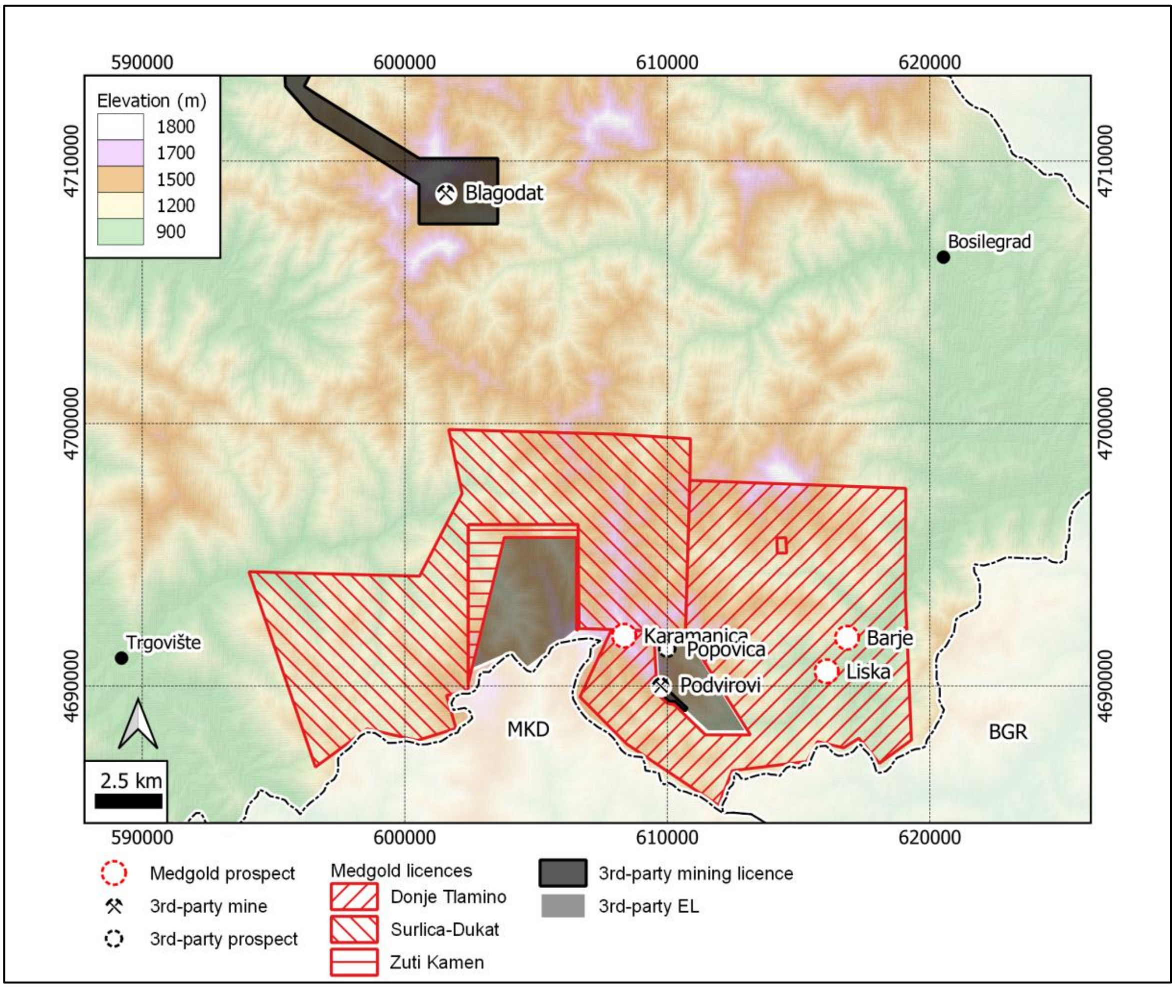


Figure 4.2. Property details.

The ELs grant the holder the exclusive right to explore over a period of up to eight years for Au, Ag, Cu, Pb, Zn and Mo to an unspecified depth below the ground surface and thereafter apply for an exploitation licence upon the discovery of a declared reserve. Exploration activities are regulated by the MME in consultation with the Institute of Environmental Protection and the Institute for the Protection of Cultural Heritage. Exploration is required to be completed under a work programme and budget approved by the MME with the holder obliged to fulfil a minimum of 75% of the approved programme and to submit annual licence reports. Work programmes are generally split to cover year one through three, four through six, and seven through eight.

4.3 Property Ownership

The Donje Tlamino and the Surlica-Dukat ELs are held by Medgold Istraživanja d.o.o., a Serbian registered company wholly owned by Tlamino Mining Limited of Malta, a wholly owned subsidiary of MGold International Limited, also of Malta. MGold International Limited is wholly owned by Medgold Resources Corp of British Columbia, Canada. Medgold Istraživanja d.o.o. was formerly wholly owned by Medgold Resource Limited of England and Wales, itself wholly owned by Medgold

Resources Corp of British Columbia, Canada. The registration of the above change of ownership of Medgold Istraživanja d.o.o. by Tlamino Mining Limited with the Serbian Business Registry is pending.

4.4 Property Agreements

In June 2016, Medgold Istraživanja d.o.o. granted Fortuna the right to enter into an option agreement (the "Original Option") on a selected project in Serbia. In March 2017 Fortuna elected to option the Tlamino Project consisting of the Donje Tlamino and the Surlica-Dukat ELs. Under the terms of the Original Option Fortuna was required to spend US$3.0 million to acquire 51 percent of the Project; this milestone was reached during H2 2019. In January 2020 Medgold, Medgold Istraživanja d.o.o. and Tlamino Mining Limited entered into a new Option Agreement (the "New Agreement") with Fortuna to acquire Fortuna's 51% interest in the Tlamino Project. The Original Option was terminated under the terms of the New Agreement.

The terms of the New Agreement provide Medgold with an exclusive option (the "Option") to purchase Fortuna's 51% interest in the Project for a cash consideration of US$3.468 million. The Option is valid for three years and is exercisable (i) at any time at the election of Medgold prior to the expiry of the term of the Option; or (ii) at the date of completion of a sale by Medgold of a 100% interest in the Project to a third party; or (iii) at the date of completion of a merger between Medgold and a third party, whichever arises soonest.

In the event that Medgold completes a sale of the Project or corporate merger during the term of the Option and receives consideration in excess of US$8.84 million (the "Sale Consideration"), Medgold will pay to Fortuna an asset sale bonus equal to 10.2% of any amount in excess of the Sale Consideration, less all of Medgold's costs related to the sale or corporate merger. No other consideration is due by Medgold to Fortuna under the terms of the Agreement. The monthly Option Fees referred to in the non-binding letter of intent announced on June 18, 2020 have been struck.

Should Medgold not exercise the Option or complete a sale of the Project or corporate merger within the term of the Option Medgold will transfer its undivided 49% interest in the Project to Fortuna for no consideration, such that Fortuna will then hold an undivided 100% interest in the Project.

4.5 Property Encumbrances

Medgold was awarded the ELs by the MME following an application over open ground with no current exploration title. As such, the Property has no encumbrances to any prior holders of exploration or exploitation rights.

Holders of ELs are required to pay the Serbian State an annual fee in the form of a royalty of 10,000 Serbian dinars per square kilometre per year for the right to explore and retain the exploration area. Total royalties for the project ELs of approximately CA$25,400 equivalent per year are payable annually for the proposed year one through three work programmes. A similar annual fee would apply to any optional extension period after year three.

Medgold is required to fulfil a minimum of 75% of its proposed year four through six work programmes for the project ELs. The total value of the programmes is approximately US$147,000.

4.6 Surface Rights

Exploration licences issued by the MME do not automatically confer legal access to the land, and exploration activities must be carried out with permission of the relevant surface landowner or land-user as recorded in the Serbian cadastral register. Surface landowners within the Property include private individuals and the Serbian State forestry body, Srbijašume. Medgold has signed a comprehensive suite of agreements with private landowners to provide access rights for drilling during 2018 and 2019 and has no reason to believe that similar agreements will not be attainable in the future.

4.7 Other Property Factors

Should Medgold fail to meet, or in the case that it exceeds, its work programme commitments by more than 25%, it is required to file an annex to the MME. No additional permits are required for the currently submitted work programmes. AMS knows of no other significant risks or factors, including environmental liabilities that may affect title, access, or the right to perform work on the Property.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

The Property may be accessed via the town of Bosilegrad, which is located approximately 385 km via highway, and sealed road from Belgrade. From Bosilegrad, 20 km of sealed road and two km of graded gravel track leads to the Property boundary. Graded gravel tracks continue to an active mine site at Podvirovi immediately adjacent to the Property and smaller unsurfaced tracks provide access to the main prospects in the Property.

Sealed roads and the graded gravel track to Podvirovi are kept open all year round and are only typically closed for short periods immediately after heavy snowfall during the winter. Other tracks within the Property may be inaccessible due to snow accumulation generally limiting the exploration season to April through November.

5.2 Climate

The Property is subject to a humid continental climate (Köppen-Geiger climate classification Dfb, (Beck, 2018)) modified by altitude effects on the higher ground in the area.

Climate data for Bosilegrad indicate summer temperatures with average highs of 20°C to 25°C and lows of 7°C to 11°C between early June and mid-September. Winter temperatures with average highs of 1° to 5°C and lows of -2°C to -7°C are recorded between late November and early March. Rain falls throughout the year with 30 mm to 50 mm of rainfall per month between early April and late November; winters are dryer with around 10 mm of rainfall per month between December and March. Snowfall occurs from early October to early May peaking at between 10 mm and 15 mm liquid equivalent during December and January.

Areas of higher ground within the Property are up to 10°C colder than at Bosilegrad. Snowfall may occur over a longer period in these areas and drifts of wind-driven snow can remain until May.

5.3 Infrastructure and Local Resources

Medgold has an operational base in the town of Bosilegrad, approximately 22 kms from the Property boundary. Bosilegrad town serves a population of around 8,000 people in the surrounding municipality and hosts an array of services including a healthcare centre, a police station, primary and secondary schools, postal and banking services, a fuel station, a hotel and shops and restaurants. The main industries in the area are forestry and related processing of timber products.

There are no significant population centres within the Property. The main villages, Donje Tlamino and Bistar, are found along the valley of the Golema River, and consist of only a few dwellings, most of which are unoccupied or only used during summer months.

Infrastructure within the Property is moderately developed. Although there are no sealed roads, a well-graded gravel road traverses the Property to the mine site at Podvirovi, and a well-established network of unsurfaced tracks used for timber harvesting may be used to access the higher ground in the Property. Nearby power distribution occurs via 35 kV and lower voltage transmission lines running along the Golema River to Podvirovi.

The mine at Podvirovi provides employment for 120 people (Mineco Limited, 2019), and consists of an underground mining operation. Ore is reportedly transported by a rail system to a pilot beneficiation plant that includes crushing, grinding and floatation circuits.

5.4 Physiography

The terrain surrounding the Barje Deposit is formed of incised river valleys (Figure 5.1), steep slopes, and high broad ridges (Figure 5.2) with elevations ranging from approximately 820 m in the Golema River valley to 1820 m at Golemi Peak. Valley sides and higher, northern-facing slopes are dominated by beech and pine forests. Valley bottoms and flatter areas at mid elevations have been cleared for agriculture, although a significant amount of the cleared land is not currently worked. Higher, south-facing slopes and the ridge tops are unimproved and give host to open grassland and small shrubs.

*Figure 5.1. View of Landscape in the Golema River Valley.
Looking south from the Barje Deposit, June 2019.*

*Figure 5.2: View of Landscape looking to Golemi Peak.
Looking north-west in the Karamanica prospect, August 2019.*

6 History

6.1 Prior Ownership

Parts of the Property are known to have been held under an EL by Dundee Plemeniti Metali d.o.o., which carried out exploration work between approximately 2005 and 2012, though the full details of these ELs are not known. Dundee Plemeniti Metali d.o.o. was renamed as Avala Resources d.o.o. prior to being acquired by Avala Resources Ltd. Of Canada during July 2010[1]. The EL held by Avala then lapsed. Medgold acquired new ELs (Donje Tlamino and Surlica-Dukat) over the same area in 2016, obtaining the Avala Resources data pack in the process. Medgold's ELs were reissued by Serbian authorities in 2020.

6.2 Exploration History

Old, long-abandoned pits and workings over mineralized zones on the Property suggest a history of exploration or mining in the area prior to the 20th Century.

During the 1930s, limited surface and underground exploration was carried out by a private company at Božilovo Ležište. This work included two trenches, one short shaft and three exploratory adits with total underground development of approximately 250 m.

6.2.1 State Exploration

Exploration by Yugoslav State agencies was carried out under multiple campaigns between the 1950's and the 1970's. The following summary has been compiled from archival data of the Serbian Geofund and State exploration reports (Petrović et al, 1978a and 1978b). Location details of the Prospects referenced herein may be found in section 7.3.

1950's

- Mapping and prospecting identified numerous sites of surface mineralization and old exploration/mining pits and trenches. Scout geophysical surveys including ground magnetics and self-potential surveys were carried out on specific sites.
- Exploration at the Liska Prospect included seven drill holes totalling 432.9 m, 346 m of adit development and a 30 m shaft discovering Pb-Zn-Cu mineralization.

[1] Within this report, Dundee Plemeniti Metali d.o.o., Avala Resources d.o.o., and Avala Resources Ltd. are collectively referred to as "Avala".

- Exploration at the Barje Deposit included 133 m of adit development and the observation of enargite, sphalerite, pyrite, chalcopyrite, arsenopyrite, marcasite, galena, gold and tetrahedrite in rock specimens.

1960's

- Continued mapping and prospecting included geochemical and geophysical surveys, trenching and limited scout drilling.
- Continued exploration at the Liska Prospect including three drill holes totalling 257 m.
- Development of one short exploration adit at Božilovo Ležište.

1970's

- Continued exploration at the Liska Prospect included a geophysical induced polarisation survey and 39 drill holes, totalling 4665.5 m.
- Limited exploration drilling at Božilovo Ležište.

6.2.2 Exploration by Avala

Between approximately 2005 and 2012, exploration work was conducted within the area of the Property by Avala. Work included multiple stages of stream sediment, soil and rock sampling from which geochemical anomalies were identified at the Karamanica Prospect and Barje Deposit. The work culminated in the drilling of four exploration drill holes, totalling 831.2 m at the Barje Deposit during 2010 and 2011; the drill holes did not intersect any significant mineralisation. The Avala data was purchased by Medgold as a data pack and has been reviewed and interpreted by Medgold staff; Avala drill core was inspected by Medgold staff and independent consultant Richard Sillitoe (Sillitoe, 2016).

6.3 Historical Estimates

The following describes historical Mineral Resources estimates which have not been reported in accordance with NI 43-101 or any similar CRIRSCO aligned reporting code and include classifications which have no equivalent in NI 43-101. The basis of the estimates is unknown and have not been verified by a Qualified Person. As such these are presented only as an indication of the type of mineralization which may be present on the Property. The tonnages and grades reported may not be accurate and the underlying economics parameters used to identify quantify potentially economically exploitable material may have significantly changed.

At the Liska Prospect in the Donje Tlamino EL a historical Mineral Resource estimate was reported by Yugoslav State agencies in 1983 based on drilling carried out at the prospect. The “Balance Reserve” was stated as 4,863,000 t at 0.54% Pb and 1% Zn in B+C1 categories from a tabular zone 15 to 40 m thick with lateral extents of 300 by 450 m associated with a flat-lying boundary between schist/gneiss and overlying conglomerate units (SOUR Geozavod-Belgrade, RO Institut za istraživanje mineralnih sirovina, 1983).

Drilling carried out by Medgold during 2019 confirmed the presence of zinc-lead mineralization within the area of the foregoing historical Mineral Resource estimate (Medgold Resources Corp, 2019).

Historical Mineral Resource estimates of the former Yugoslavia were calculated in accordance with the laws and regulations applicable at that time, defined specifically by *“Zakon o jedinstvenom načinu utvrđivanja, evidentiranja i prikupljanja podataka o rezervama mineralnih sirovina i podzemnih voda i o bilansu tih rezervi”* (The Law on the Uniform Method of Establishing, Recording and Gathering Data on Reserves of Mineral Raw Materials and Underground Water and Their Balancing), Službeni list SFR Jugoslavije, br. 53/1977; and *“Pravilnik o klasifikaciji i kategorizaciji rezervi čvrstih mineralnih sirovina i vođenju evidencije o njima”* (The Book of Regulations on Classification and Categorization of Reserves of Solid Mineral Raw Materials and Keeping a File on Them), Službeni list SFR Jugoslavije br. 53/1979.

7 Geological Setting and Mineralization

7.1 Regional Geology

The Property is situated within the western Tethyan orogenic belt, a complicated zone of tectonic units and magmatic activity related to collisional plate tectonic processes active from the mid-Paleozoic through to the modern day (Richards, 2016).

A north-south trending ophiolite belt to the west of the Property (Figure 7.1) marks the suture of the Vardar Ocean, closure of which occurred from the Late Jurassic to the Paleogene. During this compressional event, Serbo-Macedonian units were thrust over Struma units (Figure 7.2) during Alpine nappe-stacking in the Early Cretaceous (Schmid, et al., 2019). In the Late Cretaceous, subduction is suggested to have stalled either due to trench-retreat and slab-rollback caused by orogenic collapse (von Quadt, Moritz, Peytcheva, & Heinrich, 2005), or due to a transpressional or transtensional structural regime caused by oblique subduction (Chambefort & Moritz, 2006). These structural regimes produced a belt of calc-alkaline magmatism in the Late Cretaceous (Figure 7.3) that is responsible for the formation of significant porphyry-epithermal deposits containing copper ± gold, including Bor and Majdanpek in Serbia and Elatsite, Assarel and Chelopech in Bulgaria (Richards, 2016).

Final closure of the Vardar ocean occurred in the Eocene (Sharp & Robertson, 2006), after which subduction transferred to the Hellenic trench causing extension of the crust in the Aegean and a return to collisional tectonics along the Balkans. Late orogenic extension is proposed to have caused magmatism in the Eocene-Oligocene with associated porphyry and epithermal mineralization possibly linked to the exhumation of the Rhodope as a metamorphic core complex (Marchev, et al., 2005). A return to collisional, or possibly post-subduction, tectonics in the Miocene caused magmatism in a belt to the south of the Eocene-Oligocene magmatism with associated mineralization such as the Pt-Pd-Au enriched copper porphyry at Skouries in northern Greece (Richards, 2016).

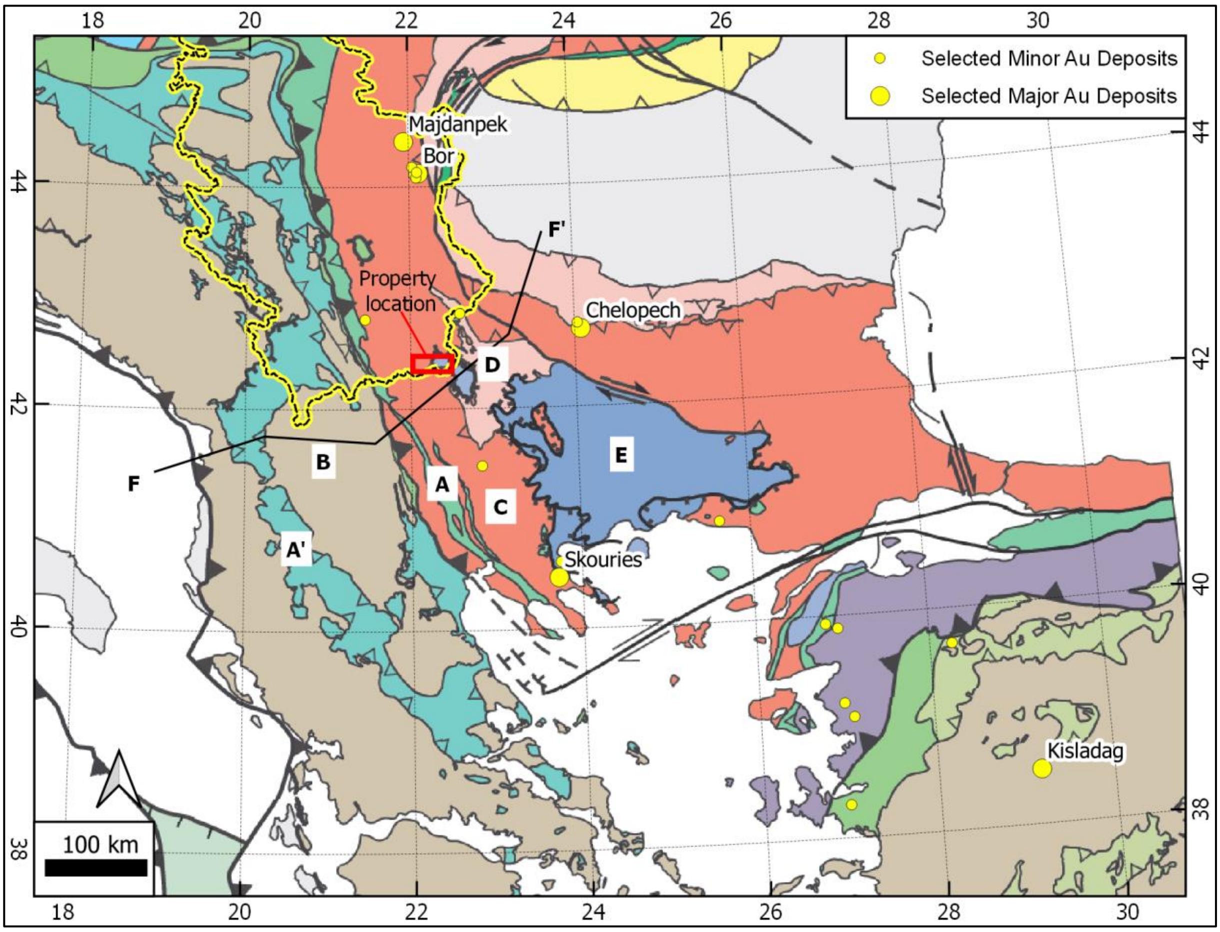


Figure 7.1. Tectonic Boundaries and Selected Major Gold Deposits of SE Europe and NW Turkey.
From Schmid et al (2019). The Property location is outlined in red with Serbia outlined in yellow. A and A' – East and West Vardar ophiolites. B – Adria derived units. C – Serbo-Macedonian units, Europe derived. D – Struma units, Europe derived. E – Rhodope crystalline basement. F-F' line of section in Figure 7.2.

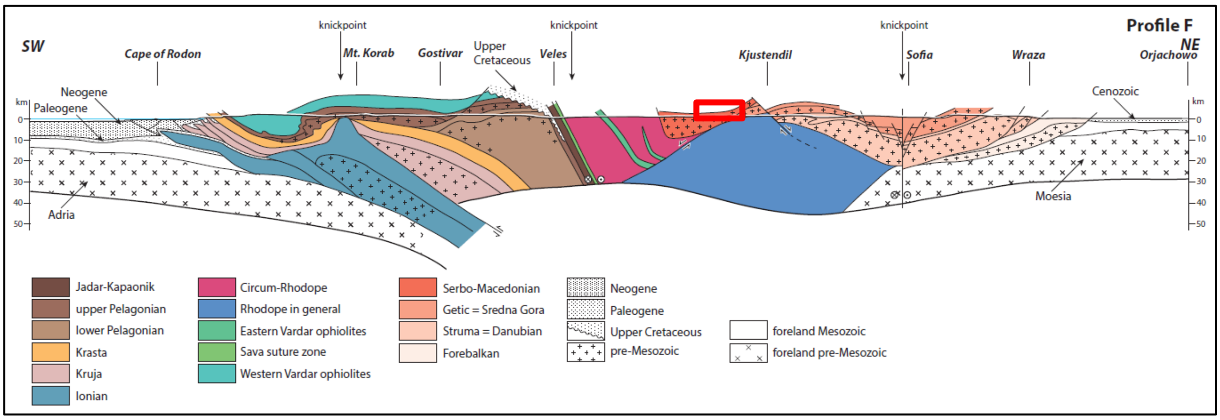


Figure 7.2. Section from WSW to ENE through the region.
From Schmid et al (2019). Red box indicates the off-section location of the Property.

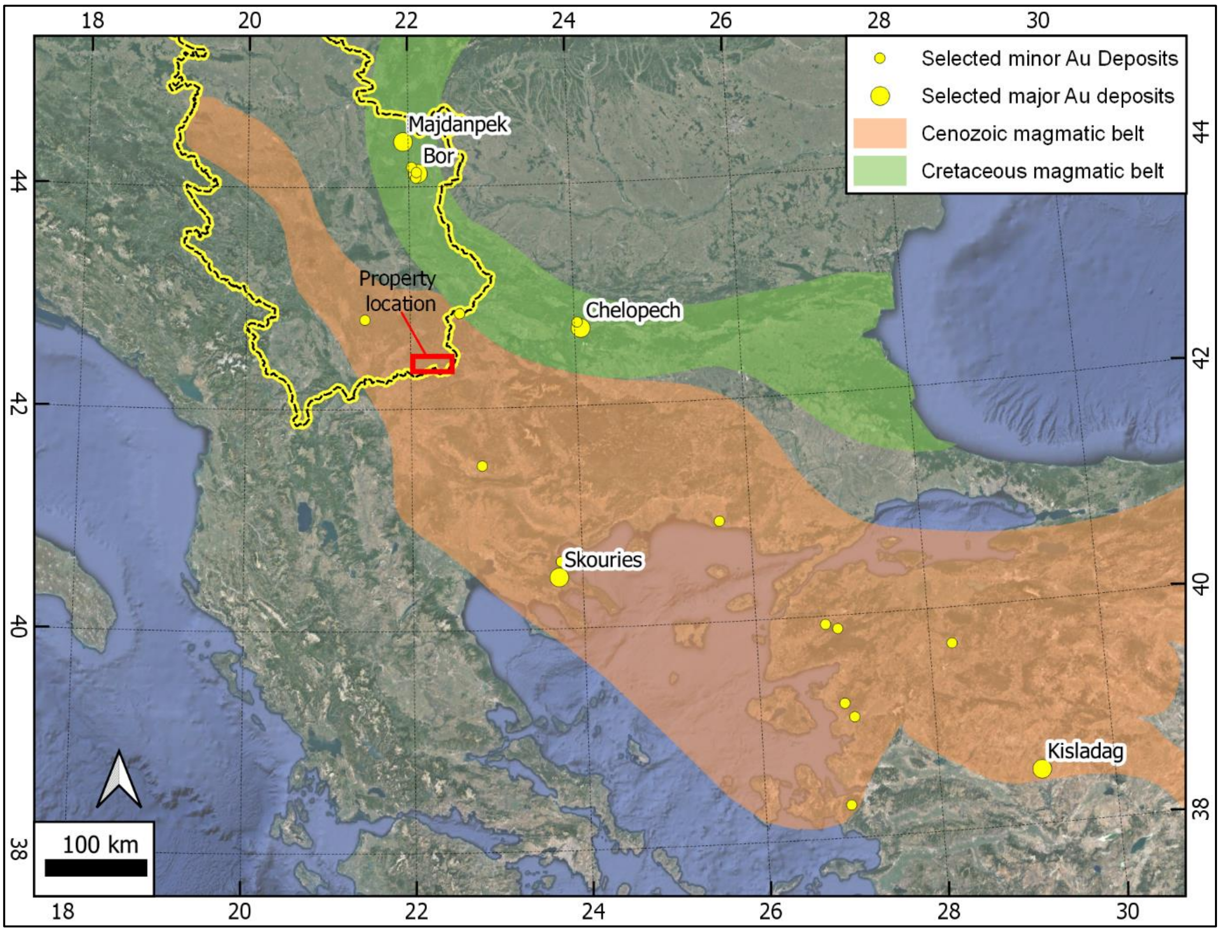


Figure 7.3. Magmatic Belts and Selected Major Gold Deposits of SE Europe and NW Turkey. Modified after Baker (2019)

7.2 Local and Property Geology

The geology of the Property (Figure 7.4) can be roughly split into high metamorphic-grade rocks in the north and east, and low metamorphic-grade rocks in the south and west. Numerous older intrusions represented by orthogneiss cut both metamorphic sequences but are limited in the area of the Property to the higher metamorphic grade rocks. Relatively minor areas of sediments and volcanoclastic sequences are preserved in the Property; these are deposited unconformably on the lower-grade metamorphic sequence. The metamorphic and sedimentary sequences are cut by porphyritic dykes and sills and, in the western part of the Property, by volcanic plugs. Quaternary deposits occur in the Property and include unconsolidated talus on the higher ground and alluvial sediments along valley floors. The geology of the Property is described in more detail below.

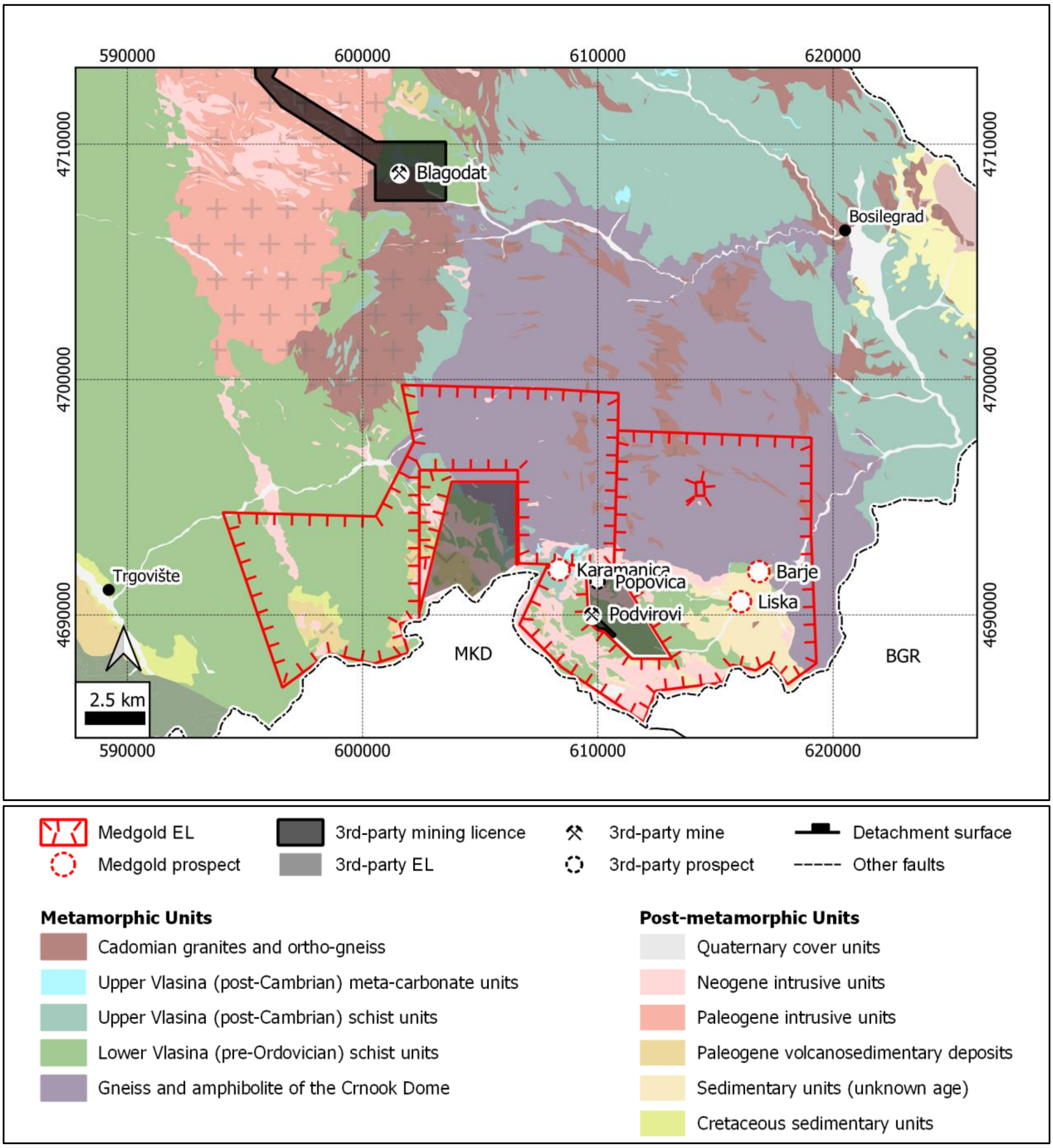


Figure 7.4. Geology of the Property and surrounding areas.

7.2.1 The Crnook Basement

The Crnook Dome in the north and east of the Property is a tectonically exposed window of metamorphic crystalline basement exhumed during middle Eocene to Oligocene (Kounov, Seward, & Bernoulli, 2004). Rocks of the dome are dominated by lower-amphibole facies gneiss and amphibolite forming the basement units in the broader Property area.

Pale-coloured gneissic intrusions are found in the Crnook dome and outside of the Property in the Vlasina Units to the west. These have been dated as Ediacaran to early Cambrian and are thought to represent a magmatic-arc complex of the Cadomian Orogeny (Antić, et al., 2015).

7.2.2 Lower and Upper Vlasina Units

The rocks of the Crnook dome are separated from the Vlasina units by a shallow angle detachment structure (Figure 7.4). The latter include sub-greenschist to greenschist facies schists, marbles, phyllites and quartzites suspected to be formed from protoliths of seafloor sedimentary sequences with minor periods of volcanic input.

7.2.3 Sedimentary and volcano-sedimentary sequences

Two main areas of non-metamorphosed sedimentary units are found within the Property.

In the southwest a marl unit rests, unconformably in part, on the Vlasina Units. State geological maps mark this unit as being Cretaceous in age. Lying unconformably over this unit and the Vlasina units are other isolated areas of tuffs and agglomerates with features of a subaerial eruption environment e.g., accretionary lapilli; state geological maps mark these volcanosedimentary units as Paleogene.

In the east of the Property and dominating the surface of the Liska Prospect, a sequence of conglomerate with sub-dominant sandstone is present. On state geological maps, these are marked as Eocene in age, and sit unconformably on top of the Vlasina Units; a previous interpretation suggests the units are a late graben-fill. Drilling by Medgold during 2018 at the Barje Deposit indicates that the conglomerates are, at least partially, bounded by shallow fault structures – both below and above the unit, causing the conglomerates to sit between a lower and upper block of schist. The shallow fault structures are speculated to be sub-parallel thrusts or rotated normal faults. As such, these conglomerates are likely to be of an older age than previously assumed, possibly being deposited prior to Alpine nappe-stacking during the Cretaceous.

7.2.4 Intrusions

Throughout the Property, felsic porphyry dykes, sills, and rare volcanic plugs are present. The dykes and sills trend approximately north-south in the western part of the Property with a gradual change to an east-west trend in the area of the Barje Deposit. State geology maps indicate the intrusions as being Late Tertiary quartz latites. The intrusions are formed of a fine- to medium-grained groundmass containing variable amounts of plagioclase feldspar ± quartz, biotite or hornblende ± minor amounts of magnetite. Phenocryst content varies from intrusions that are non-porphyritic to intrusions with plagioclase mega-crysts to larger than 20 millimetres across. Major oxide geochemistry carried out by Medgold on a sill at the Barje Deposit classify the rock type as a calk-alkaline series dacite.

7.2.5 Recent cover units

Recent cover units include alluvial sediments and terraces occurring along the floors of the larger river valleys and areas of unconsolidated talus on the higher ground.

7.3 Mineralized Zones on the Property

The Property hosts three identifiable mineralized zones at the Barje Deposit, the Liska Prospect and the Karamanica Prospect. These Prospects are described in detail below.

7.3.1 The Barje Deposit

The Barje Deposit is the most advanced mineralized zone within the Property. It is located on an east-west trending ridge at elevations ranging from approximately 1100 m in the east to 1300 m in the west. Historical prospecting located two main areas of outcropping gold and base metal mineralization at Barje; drilling on the margin of these areas by Avala, targeting steep feeder zones, failed to intersect significant mineralization. Medgold's drilling has confirmed the continuation of mineralization between and to the west of the discovery outcrops in an area of 700 m east-west by 250 m north-south (Medgold Resources Corp, 2019a). Said mineralization is controlled by a hydrothermal breccia, of up to 20 m in thickness, following a structure inclined approximately 18° towards the south. This structure cuts a fault-bounded sequence of schist and conglomerate above a dacite sill intruded along a detachment surface at the top of the Crnook basement (Figure 7.5 and Figure 7.6). While mineralization is strongest in the hydrothermal breccia, a halo of lower-grade mineralization is also found in the overlying rocks. The hydrothermal breccia comprises transported clasts of the local wall-rocks cemented by a matrix of quartz ± calcite/siderite and sulphide minerals, including pyrite, arsenopyrite, sphalerite, galena and more rarely chalcopyrite and tennantite. Grains of electrum up to around 50 µm in size and containing approximately 60% Au and 40% Ag, have been observed microscopically within the higher-grade mineralization. A Mineral Resource Estimate for Barje is presented in section 14 of this report.

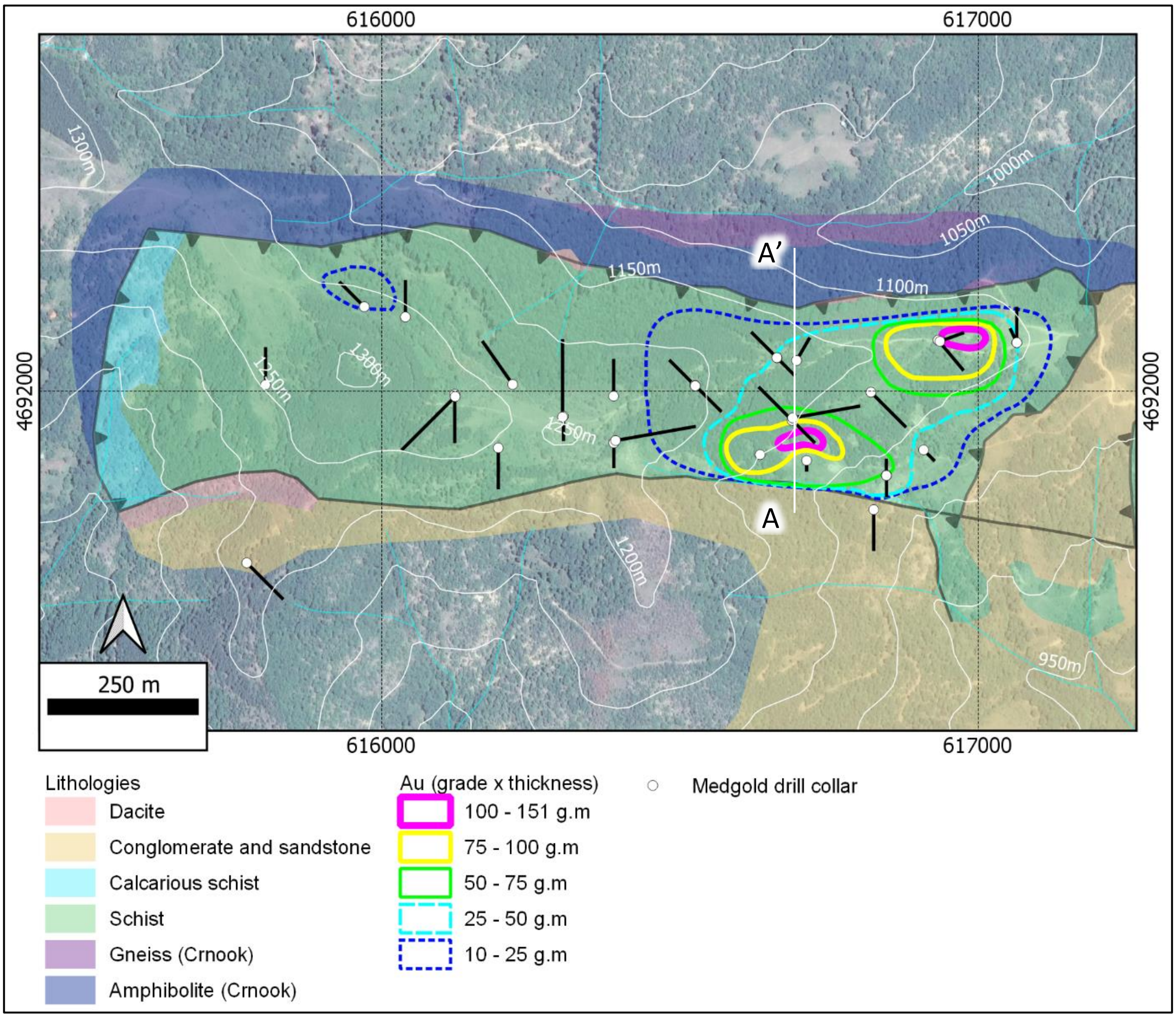


Figure 7.5. Geology, drill holes and grade-thickness of intersected Au mineralization at Barje.
See Figure 7.6 for cross-section A-A’. Thickness is vertical thickness of mineralized zone, internal waste included in calculation.

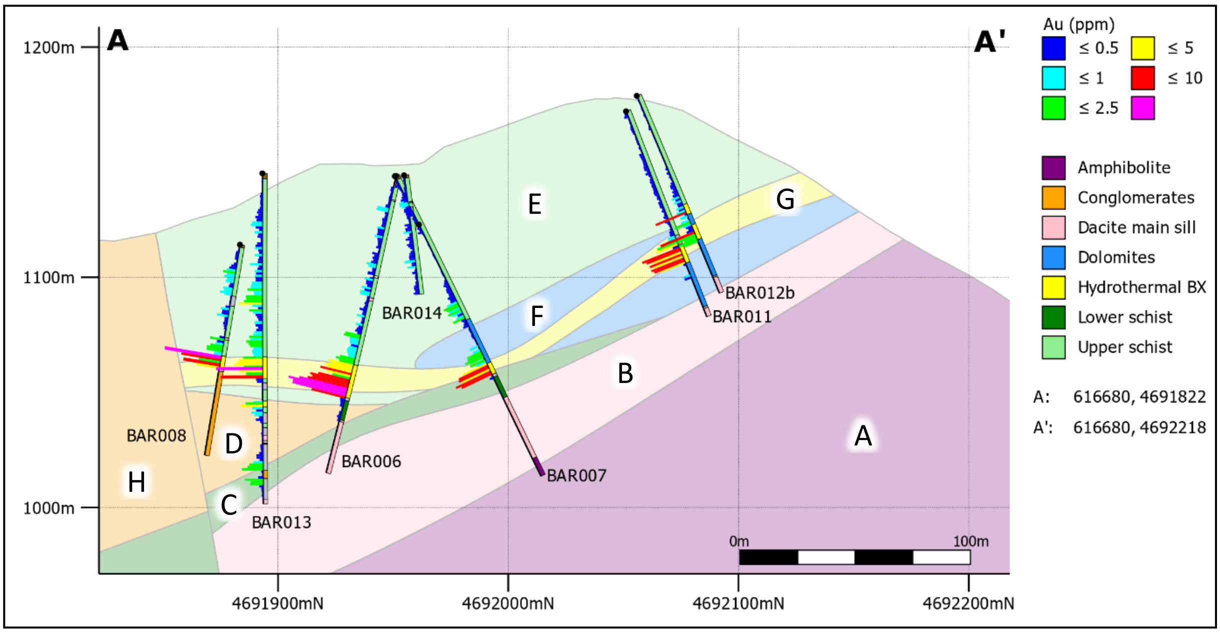


Figure 7.6. Representative Cross-section at Barje.
North-south section at 616688mE looking west. See below for lettering reference. Drill holes projected from up to 50 metres off-section.

The main lithologies at Barje are summarized as follows (letters refer to Figure 7.6, see Figure 7.7 for drill hole references).

Amphibolite/gneiss (A) – The unit is part of the higher-grade metamorphic basement of the Crnook dome and is formed mainly of amphibolite with sub-dominant intervals of gneiss. It outcrops on the northern slope of the ridge at Barje and has been confirmed at depth by drilling. The amphibolite has an obvious metamorphic texture and is dominantly formed of amphiboles, quartz and feldspars with local zones containing garnets. This unit is separated from the overlying units by a detachment structure, now utilized by a dacite sill. The amphibolite has a narrow baked-margin where it is adjacent to the sill but no mineralization or alteration has been observed in the unit.

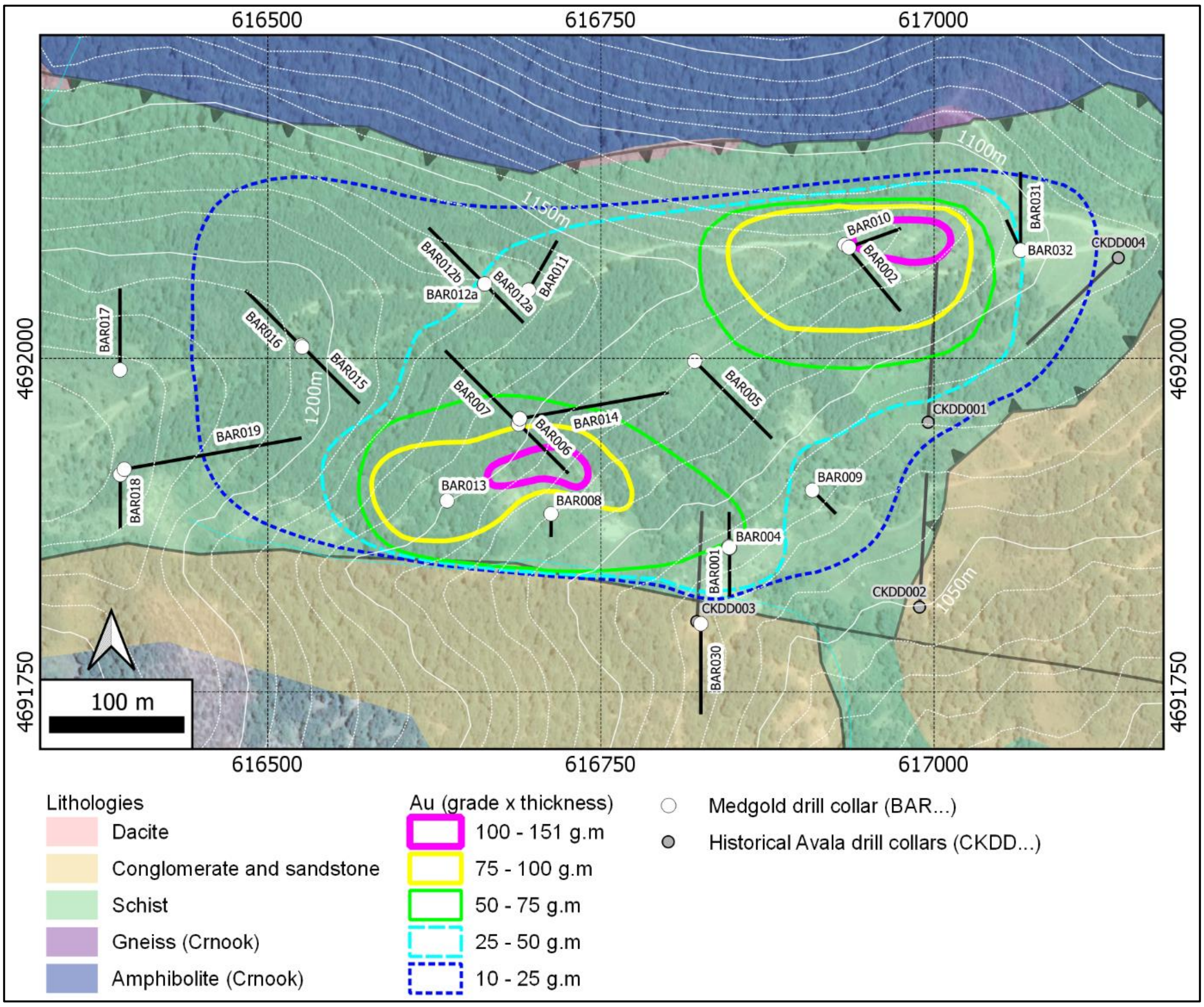


Figure 7.7. Drill hole locations within the main mineralization at Barje.

Dacite intrusions (B) – A 20- to 50-metre thick sill separates the amphibolite from overlying units intruded along the detachment structure at the top of the amphibolite. The sill displays recessive weathering and does not form large exposures however, it was consistently intersected during drilling. It consists of anhedral to subhedral quartz and euhedral biotite and plagioclase phenocrysts in a fine-grained devitrified glass matrix (Figure 7.8). The intrusion is variably altered, with weak to moderate calcite and sericite alteration being dominant. Narrow intervals of the same intrusive unit are seen in the overlying schist but are volumetrically insignificant. Analysis of five specimens of the intrusion from Barje drill core for whole-rock oxide geochemistry shows the unit to be consistent with a calk-alkaline series dacite. The main sill appears to be unmineralized. Drill core samples from the narrow sills in the overlying upper schists (unit E) indicate some weak gold (to 0.2 ppm) and zinc (to 0.2%) mineralization, thought to be an overprint of mineralization related to the hydrothermal breccias (see unit G).

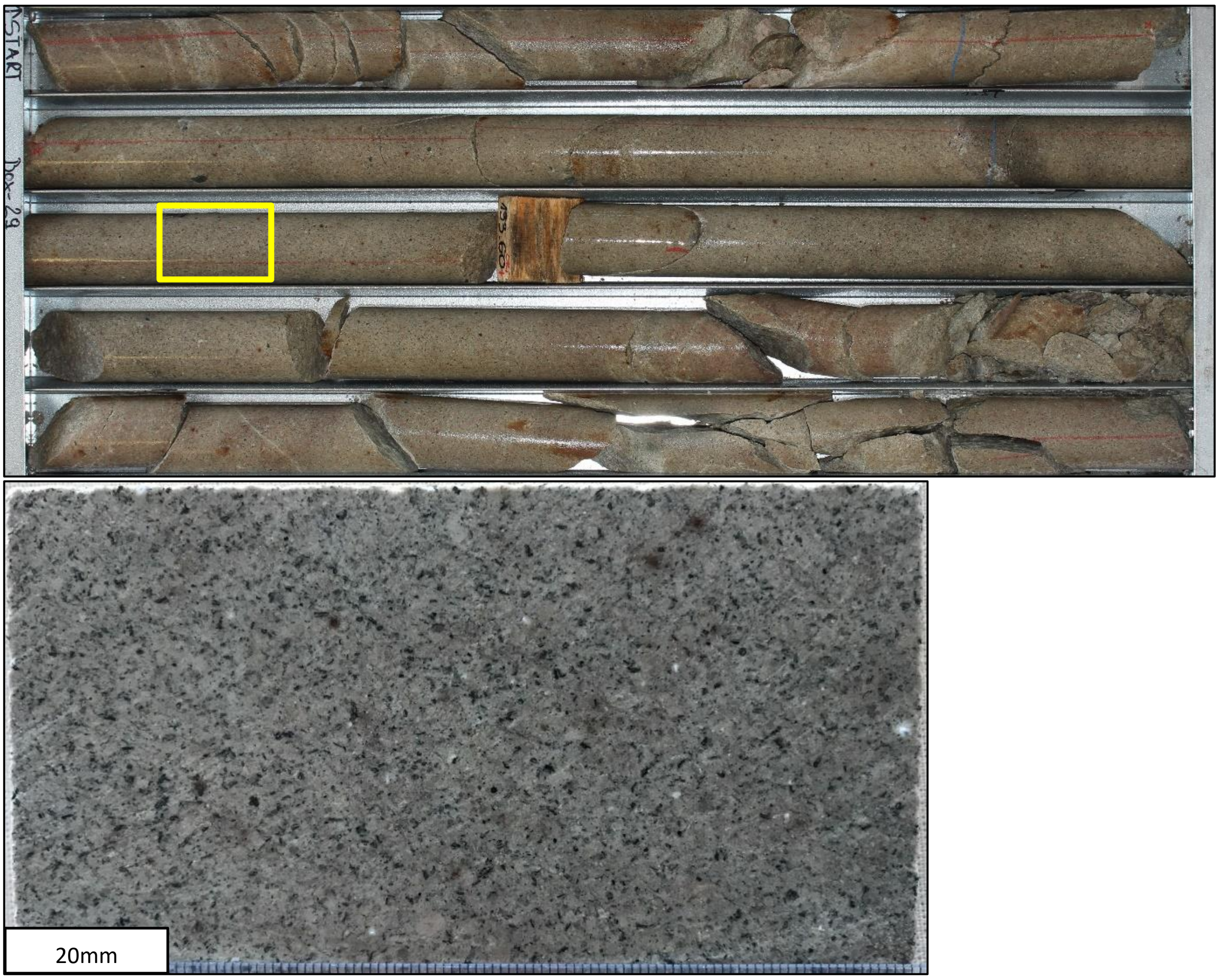


Figure 7.8. Dacite. Drill hole BAR006.
Analysis specimen: 2018XRF01. Study specimen: 2018MC09 (analysis and study specimens taken from same core interval at 133.4 metres outlined in yellow). 0.002 ppm Au, 0.04 ppm Ag, 0.00% As, 0.00% Zn, 0.00% Pb. Core diameter 61.1 mm, box length 1 m.

Lower schists (C) – In the southern and the eastern parts of the drilled area a unit of schist was intersected by drilling, bounded by the dacite sill below and a shallowly inclined fault zone above; these schists have not been identified in outcrop. Their mineralogy is generally quartz-albite-muscovite and they have been weakly affected by sericite and chlorite alteration. The lower schists are weakly mineralized, containing local pyrite, sphalerite and galena. Drill core samples indicate isolated narrow intervals with grades of 0.1 to 0.8 ppm gold 0.4% lead and 3.9% zinc.

Conglomerate (D and F) – Conglomerate outcrop forms minor exposures on the eastern end of the ridge and more common exposures on the southern side of the Barje ridge (Figure 7.9), south of an east-west trending sub-vertical fault known as the Barje fault. The conglomerate was thought by previous workers to be a thin veneer of unconformable Eocene sediments deposited in a graben setting on top of the schist units; however, Medgold's 2018 drilling identified a "wedge" of conglomerate sitting between the lower and upper schists (see unit E) at depth in several holes on the northern side of the Barje fault and recognized that the unit comes to surface north of the Barje fault on the east end of the ridge. This implies that the schists and conglomerate were faulted together after the last metamorphic event in the area. The fault zones bounding the conglomerate wedge are shallow-angle east and south dipping structures; it is not known if these represent low-angle thrusts or steeper-angle normal faults that have since been rotated. The conglomerate on the southern side of the Barje fault is of the same or a similar lithology to the conglomerate seen in the wedge. This southern conglomerate has a vertical thickness of around 120 metres to the immediate south of the main mineralized area. Conglomerates on the north and the south side of the fault are mineralized, mainly by the introduction of disseminated pyrite, when proximal to the hydrothermal breccias (unit G).

*Figure 7.9. Outcropping conglomerates south of the Barje fault.
Note geologists for scale at bottom left of outcrop.*

Upper schists (E) – A poorly outcropping schist unit, seen mainly in track cuts and in drill core, forms most of the ridge at Barje. At and close to the surface in the area of mineralization, the schist is invariably “leached” due to weathering of disseminated sulphides and has a pale colour with blebs and fracture coatings of goethitic iron oxides. In non-weathered and less altered intervals of drill core, the schist is a dark grey-green colour and is seen to contain quartz, plagioclase, chlorite and micas. Approaching zones of mineralization, the schist becomes a paler grey-buff colour due to the presence of sericite (Figure 7.10). Mineralization within the schist is zoned with an increasing amount of hydrothermal infill by quartz ± siderite, pyrite, arsenopyrite, sphalerite and galena along and cross-cutting the metamorphic fabric, eventually forming a matrix to a jigsaw-fit breccia (Figure 7.11) when proximal to the hydrothermal breccias (see unit G).

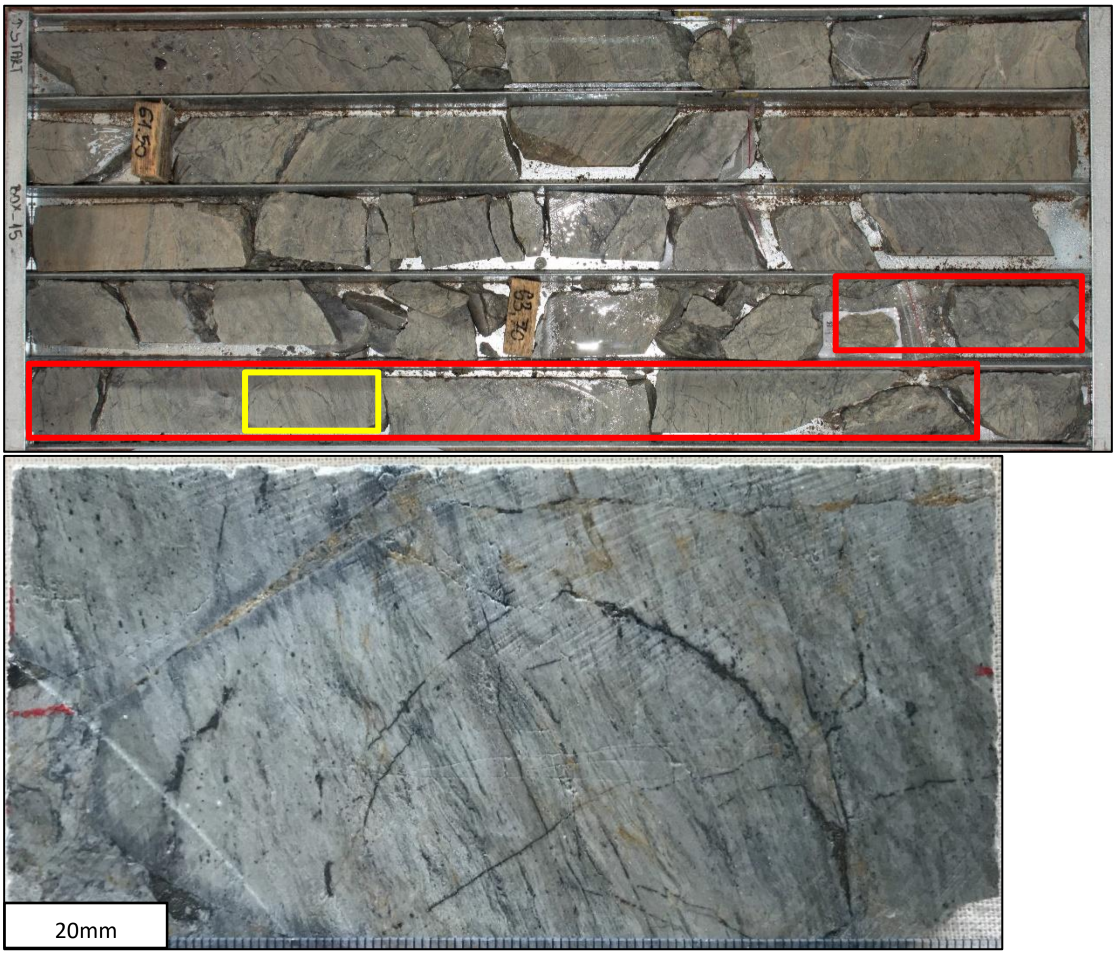


Figure 7.10. Weakly altered and mineralized upper schist. Drill hole BAR007.
Analysis sample: 1000590 (red interval 64.0 – 65.0 metres). Study specimen: 2018GD03 (yellow interval and close-up view below). 0.845 ppm Au, 2.26 ppm Ag, 0.63% As, 0.00% Zn, 0.00% Pb. Core diameter 61.1 mm, box length 1 m.

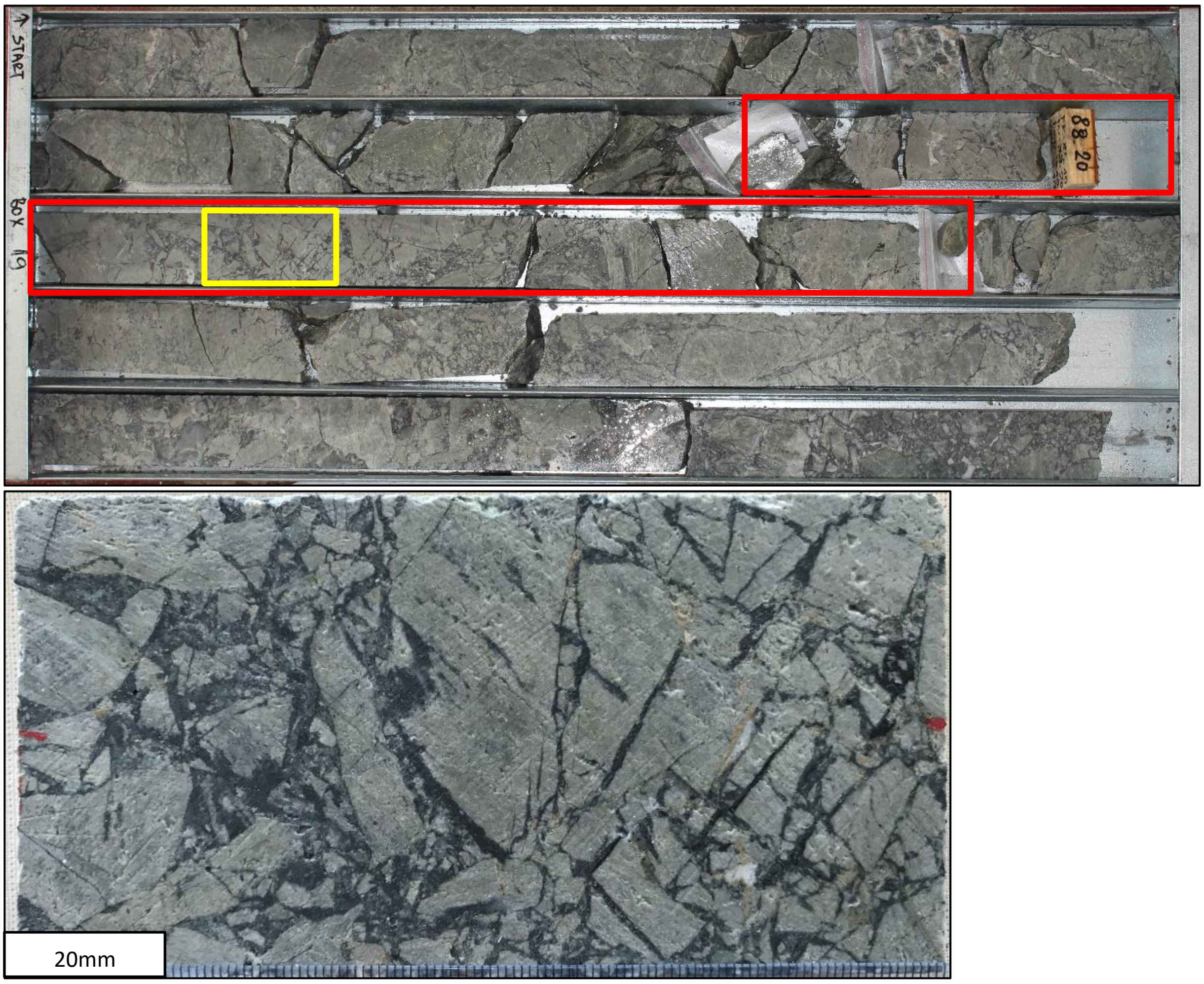


Figure 7.11. Jigsaw-fit strongly altered and mineralized upper schist. Drill hole BAR006.
Analysis sample: 1000490 (red interval, 88.0– 89.0 metres). Study specimen: 2018MC04 (yellow interval and close-up view below). 3.22 ppm Au, 4.42 ppm Ag, 2.01% As, 0.34% Zn, 0.08% Pb. Core diameter 61.1 mm, box length 1 m.

Dolomite (F) – A unit of dolomite forms small outcrops on the northeast end of the Barje ridge and was intersected in several drill holes. The exact relationship between the dolomite, the conglomerate, and the schist units is unknown, although Medgold's current working hypothesis is that the dolomite forms part of the upper schist sequence. In surface outcrops and drill core, the dolomite is strongly fractured (Figure 7.12). Microscopic examination of the dolomite shows it to have undergone multiple stages of brecciation and recrystallization. The dolomite is weakly mineralized with very fine-grained pyrite and arsenopyrite when proximal to the hydrothermal breccias (unit G).

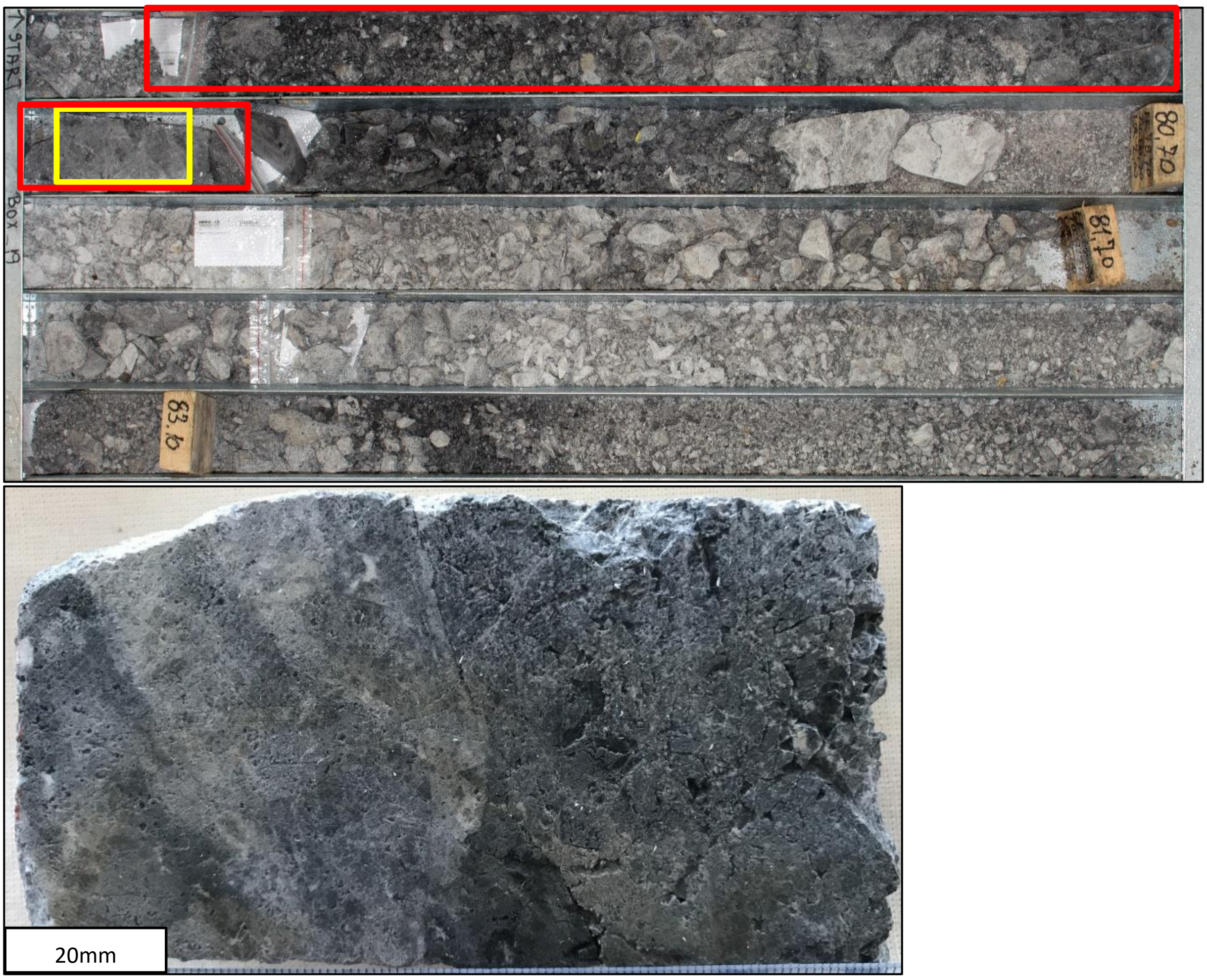


Figure 7.12. Dolomite. Drill hole BAR007.
Analysis sample: 1000607 (red interval 79.0 – 80.0 metres). Study specimen: 2018MC07 (yellow interval and close-up view below). 0.08 ppm Au, 0.16 ppm Ag, 0.02% As, 0.00% Zn, 0.00% Pb. Core diameter 61.1 mm, box length 1 m.

Hydrothermal breccia (G) – At the Barje Deposit, hydrothermal breccia is present in heavily weathered outcrops first recognized during Medgold's 2018 drilling. The breccia forms a flat-lying to shallowly dipping zone that is continuous within an area of 700 m east-west by 250 m north-south, on the north side of the Barje fault. The hydrothermal breccia cuts through the conglomerate, the upper schist and the dolomite units, generally as a single structure of up to 23-metre vertical thickness. Two breccia types were initially defined with logging codes HBX and XXX. HBX was the "default" logging option for the breccia with critical features being the presence of transported clasts cemented by a quartz ± carbonate infill containing sulphide minerals. The XXX logging code was used when sulphide material was seen to form "clasts" in the breccia (Figure 7.13), though microscopic examination suggests that these clasts may be in-situ precipitation of intergrown sulphides rather than true clasts.

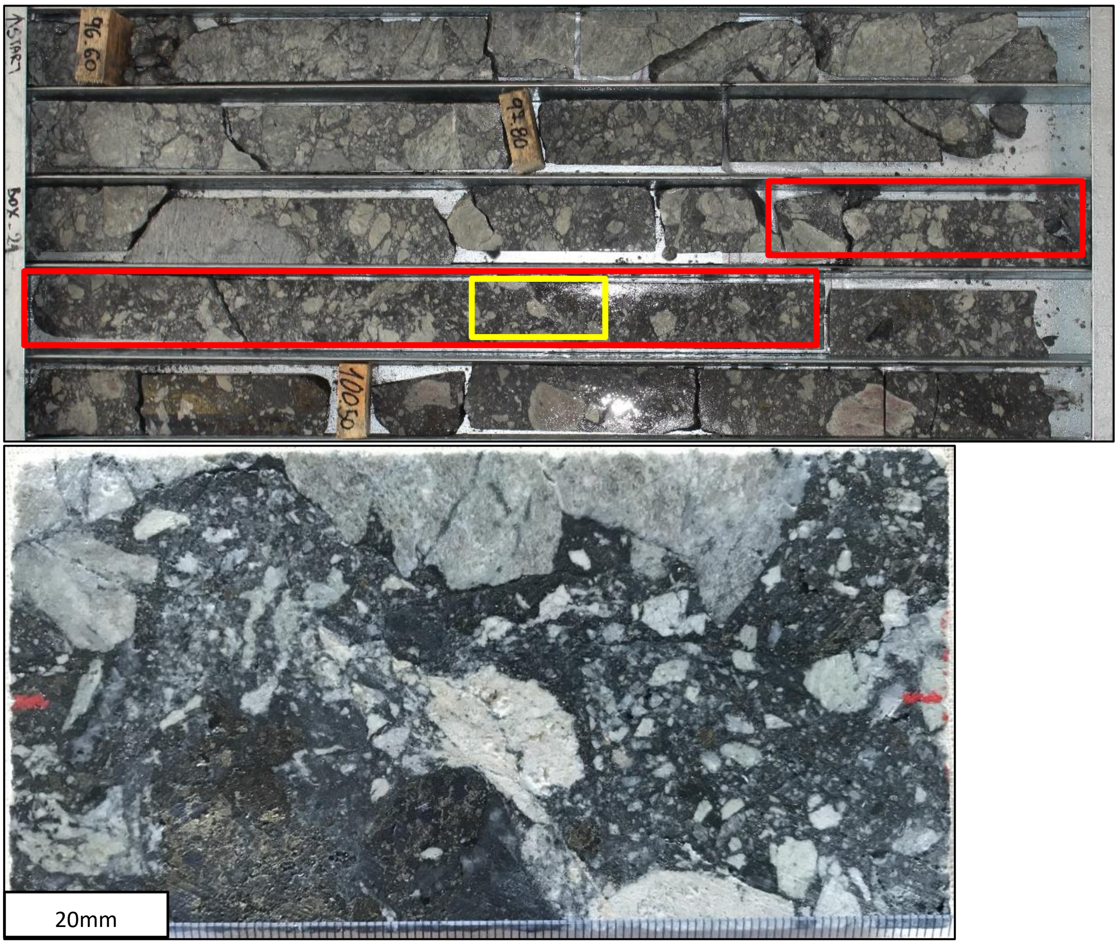


Figure 7.13. Hydrothermal breccia with sulphide "clasts". Drill hole BAR006.
Analysis sample: 1000502 (red interval 99.0 – 100.0 metres). Study specimen: 2018GD02 (yellow interval and close-up view below). 15.9 ppm Au, 33.5 ppm Ag, 3.33% As, 2.09% Zn, 0.74% Pb. Core diameter 61.1 mm, box length 1 m.

See sections 9 (Exploration) and section 10 (Drilling) for further information relating to exploration and drilling at the Barje Deposit.

7.3.2 The Liska Prospect

The Liska Prospect is situated 1500 m southwest of the Barje Deposit at an elevation of approximately 900 m. Lead-zinc mineralization was discovered by Yugoslav State agencies in the 1950s after recognition of a mineralized outcrop at the base of the valley (Figure 7.14).

Figure 7.14. The discovery outcrop at Liska.

The State drilled 50 drill holes, thought to total approximately 5200 m, and developed 346 m of adit and a shallow (30 m) shaft, generating the historical estimate listed in section 6.3 above. During 2019, Medgold carried out drilling confirming and extending the area of mineralization reported by the State. The combined area of mineralization reported from State drilling and Medgold's drilling is approximately 680 m north-south by up to 270 m east-west. A map showing Medgold and State drilling is shown in section 10.

The mineralization is associated with a flat-lying faulted boundary between overlying conglomerate units and underlying schist units; these appear to be similar lithologies to the conglomerates and the lower schist units, respectively, at Barje (see section 7.3.1). Mineralization occurs at and below the boundary between the conglomerates and the schist (Figure 10.5) with thicknesses of up to approximately 50 m. The zone outcrops at surface in the south of the Prospect and continues to depth under higher topography to the north. The current spacing of Medgold's drill holes does not allow geostatistical calculation of continuity within the mineralization; estimates using Medgold and historical State data suggest continuity in the order of 100 m to greater than 250 m north-south and 50 metres to 100 metres east-west.

Mineralization consists of quartz-pyrite veins of up to approximately 4 centimetres in width containing sphalerite and galena. Examples of weak, medium and strong mineralization, intersected during Medgold's drilling in 2019, are shown in Figure 7.15, Figure 7.16, and Figure 7.17.

Figure 7.15. Example of veining with weak pyrite-sphalerite mineralization, Drill hole LIS009.
Analysis sample: 1006146 (red interval 101.0 – 103.0 metres): 0.028 ppm Au, 0.84 ppm Ag, 0.01% As, 0.42% Zn, 0.06% P.; close-up photo outlined in yellow.

Figure 7.16. Example of veining with moderate pyrite-sphalerite-galena mineralization, Drill hole LIS006.
Analysis sample: 1005940 (red interval 127.95 – 128.95 metres): 0.377 ppm Au, 12.55 ppm Ag, 0.04% As, 3.71% Zn, 2.27% Pb; close-up photo outlined in yellow.

Figure 7.17. Example of veining with strong pyrite-sphalerite-galena mineralization, Drill hole LIS007. Analysis sample: 1006062 (red interval 214.6 – 215.6 metres): 0.453 ppm Au, 14.45 ppm Ag, 0.04% As, 9.34% Zn, 1.59% Pb; close-up photo outlined in yellow.

See sections 9 (Exploration) and section 10 (Drilling) for further information relating to exploration and drilling at the Liska Prospect.

7.3.3 The Karamanica Prospect

The Karamanica Prospect covers numerous minor occurrences of gossan, alteration and vein-related mineralization recognized during exploration by Yugoslav State agencies between the 1950s and 1980s. The State exploration work led to the discovery of the Podvirovi occurrence (see section 23, Adjacent Properties, for further information). Rock sampling by Medgold has confirmed the presence of hydrothermal alteration and veining with elevated levels of gold and base-metals at multiple locations within the Prospect (Figure 7.18) but with no significant continuity between elevated values. Scout drilling by Medgold during 2019 intersected only minor lead-zinc mineralization in quartz-carbonate veins associated with a fault zone and the margin of a porphyritic intrusion.

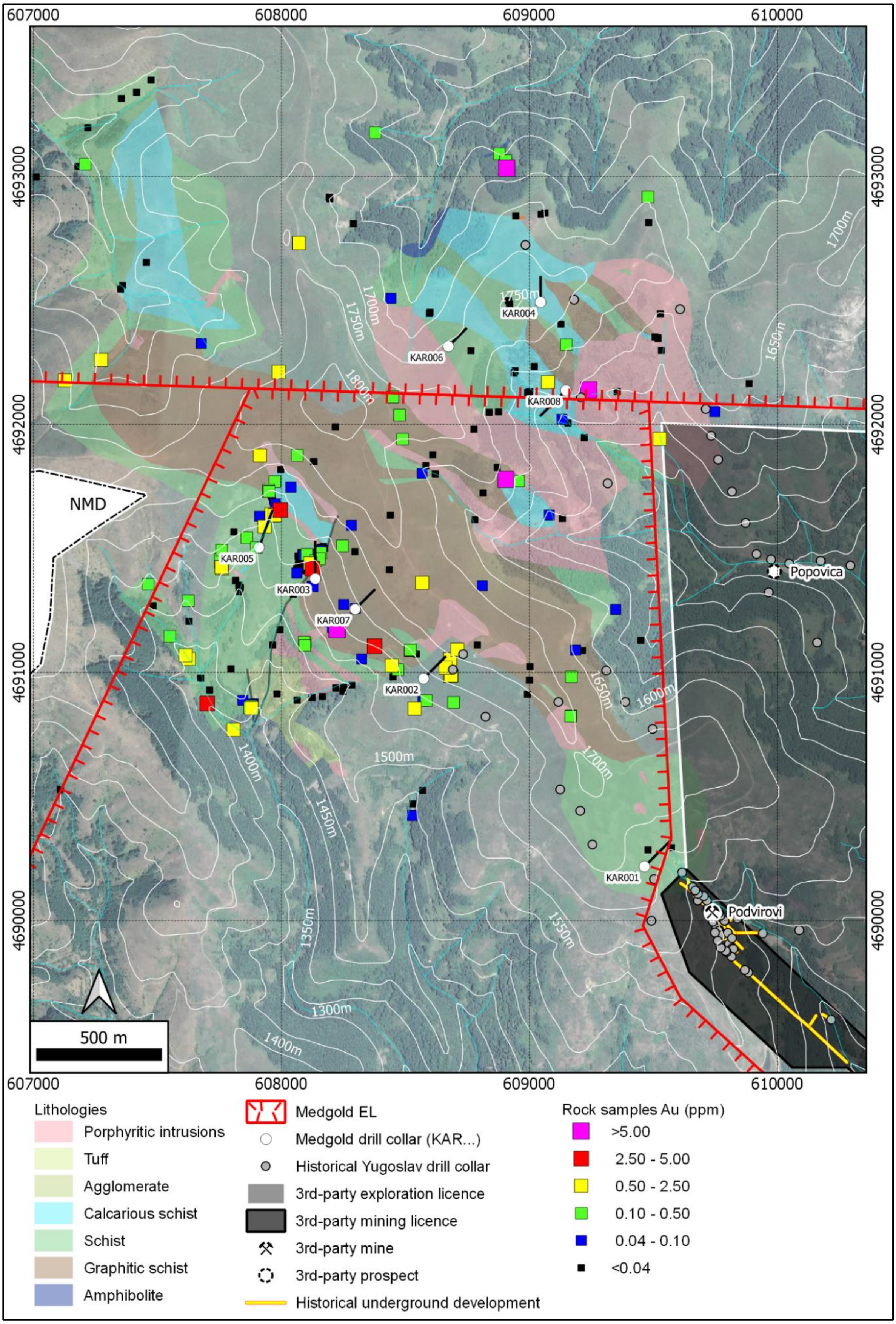


Figure 7.18. Geology, drill holes and rock samples at Karamanica.

8 Deposit Types

Medgold's exploration programmes in southwest Serbia have targeted magmatic-hydrothermal type mineralization related to Cenozoic magmatism in the Western Tethyan Metallogenic Belt ("WTMB", see section 7.1, Regional Geology), with deposit styles including porphyry, epithermal, skarn and carbonate replacement deposits (CRD). Example deposits in similar geological settings within the Greece – Bulgaria – North Macedonia – Serbia segment of the WTMB include Skouries (porphyry, Greece), Olympias and Piavista (CRD, Greece), Perama hill and Sappes (high-sulphidation epithermal, Greece), Krumovgrad (low-sulphidation epithermal, Bulgaria), Illovitza (porphyry, North Macedonia), Bucim (porphyry, North Macedonia), Plavica (high-sulphidation epithermal, North Macedonia) and Tulare (porphyry, Serbia) (Baker, 2019).

Erosion levels within the district surrounding the Property have exposed sub-volcanic porphyritic intrusions including sills, dykes and volcanic necks; only minor areas of eruptive volcanic rocks associated with the magmatism remain and no lithocaps are observed. Exploration by Medgold within the district has focused on porphyry, CRD and sub-epithermal vein target types (Figure 8.1). Potentially analogous mineralization styles from adjacent properties (section 23) include the third-party mining operations at Podvirovi (sub-epithermal vein deposit) and Blagodat (CRD).

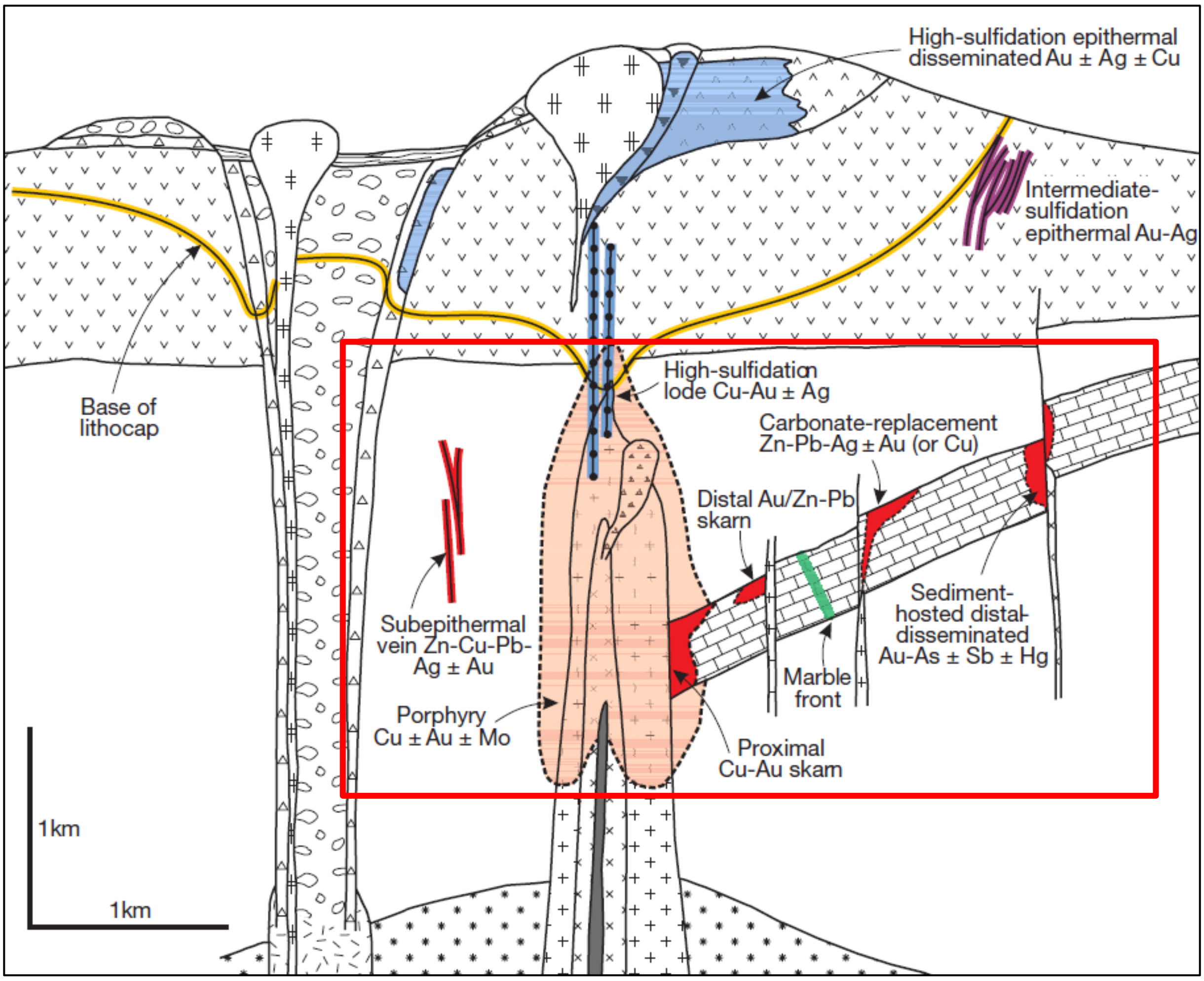


Figure 8.1: Possible target types in the Property.
Modified from Sillitoe, 2010. Potential environments of ore deposition within the Property are outlined by the red box.

9 Exploration

9.1 Desktop Study of exploration work performed by previous operators

Medgold does not have access to complete records of exploration performed by Yugoslav State agencies during the 1950s to 1980s. Much of what is known about the exploration work from this period has been compiled by Medgold from summary reports and not original data sources. The Qualified Person has not been able to verify the State exploration data.

Avala carried out exploration work on the Property from approximately 2005 to 2012. Medgold purchased a data package from Dundee Precious Metals for surface samples collected by Avala that contain merged field and analysis data exported from an acQuire database. Sample locations, field procedures and QAQC data for the Avala sample dataset have not been verified by the Qualified Person.

9.1.1 Stream sediment geochemistry - Avala

Stream Sediment samples were collected by Avala throughout the Property in sampling campaigns between 2005 and 2011. Figure 9.1 shows sample locations and results for gold analysis.

Field procedures for Avala’s stream sediment samples are not known. As such, the sampling method and the sample quality cannot be verified.

Full laboratory procedures for Avala’s stream sediment samples are also unknown. The samples were prepared to a <180 micron size fraction and analysed for gold and a multielement ICP suite. No QAQC data have been reviewed for the analytical results from this work.

Results clearly highlight source areas at the Karamanica Prospect and Barje Deposit producing downstream sediment anomalies for gold (Figure 9.1); associated pathfinders, such as arsenic and antimony were also anomalous. The third-party prospects of Podvirovi and Popovica fall in separate drainages and are not considered to have influenced the results.

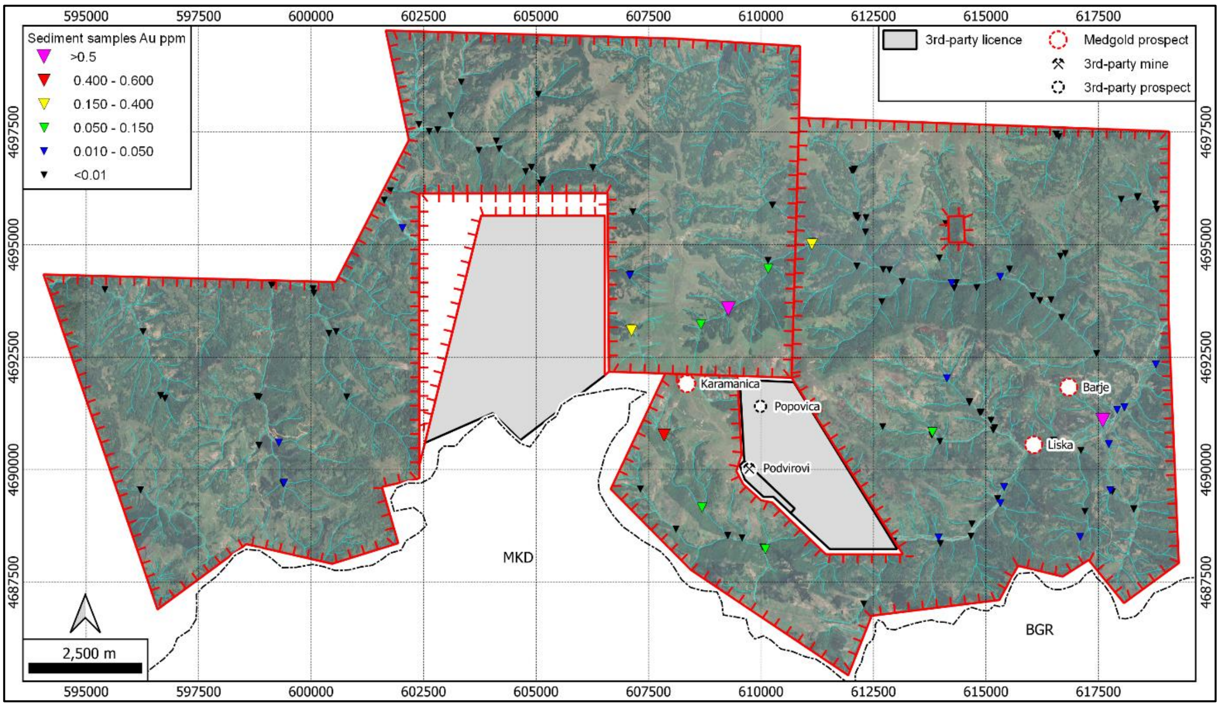


Figure 9.1. Results of analysis for stream sediment samples collected by Avala: Gold.

9.1.2 Soil geochemistry - Avala

Soil samples were collected by Avala with a focus on the Barje Deposit and the Karamanica and Liska Prospects in sampling campaigns performed in 2006 and 2007. Figure 9.2 shows sample locations and results for gold analysis.

Field procedures for Avala's soil samples are not known except that samples targeted B horizon material. As such, the sampling method and the sample quality cannot be verified.

Full laboratory procedures for Avala's soil samples are not known. Samples were analysed by multiple methods for gold and a multielement ICP suite. No QAQC data have been reviewed for the analytical results from this work. Results clearly highlight areas at Karamanica and Barje producing soil anomalies for gold (Figure 9.2); with anomalous pathfinder elements including silver, arsenic, antimony and lead. Data shows that several different elemental associations exist in the area, suggesting the possibility of different mineralization styles.

Additional soil samples were collected by Medgold between 2016 and 2019, discussed in section 9.2.1 below.

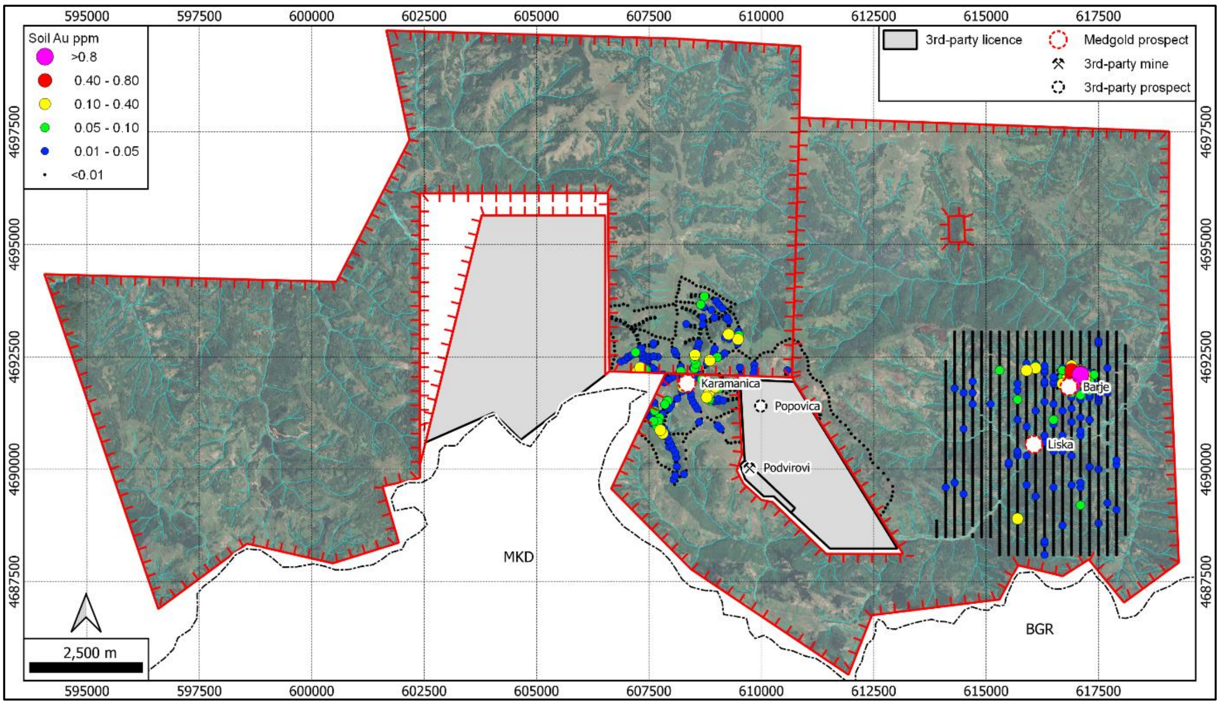


Figure 9.2. Results of analysis for soil samples collected by Avala: Gold.

9.1.3 Lithogeochemistry by Avala

Rock samples with a focus on the Karamanica Prospect and Barje Deposit were collected by Avala in sampling campaigns between 2005 and 2011. See Figure 9.2 for rock sample locations and results for gold from the general area of the Property and Figure 9.2 for rock sample locations, including trenches and outcrop channel sampling at Barje.

Field procedures are not known for Avala's rock samples. As such the sampling method and the sample quality cannot be verified. Trenches were excavated mechanically but it is believed that both trench samples and channel samples were collected by means of "continual chip" methodology as opposed to being cut as a true channel sample.

Full laboratory procedures are not known for Avala's rock samples. Samples were generally analysed by a fire-assay method for gold and a multielement ICP suite. No QAQC data have been reviewed for the analytical results of this work.

Results for rock samples collected at Karamanica (Figure 9.3) show isolated elevated gold values with little apparent continuity. Results at Barje (Figure 9.4) show significantly higher gold values with some wide zones in trench samples of good continuity.

Additional rock samples were collected by Medgold from 2016 to 2019 and are discussed in section 9.2.2 below.

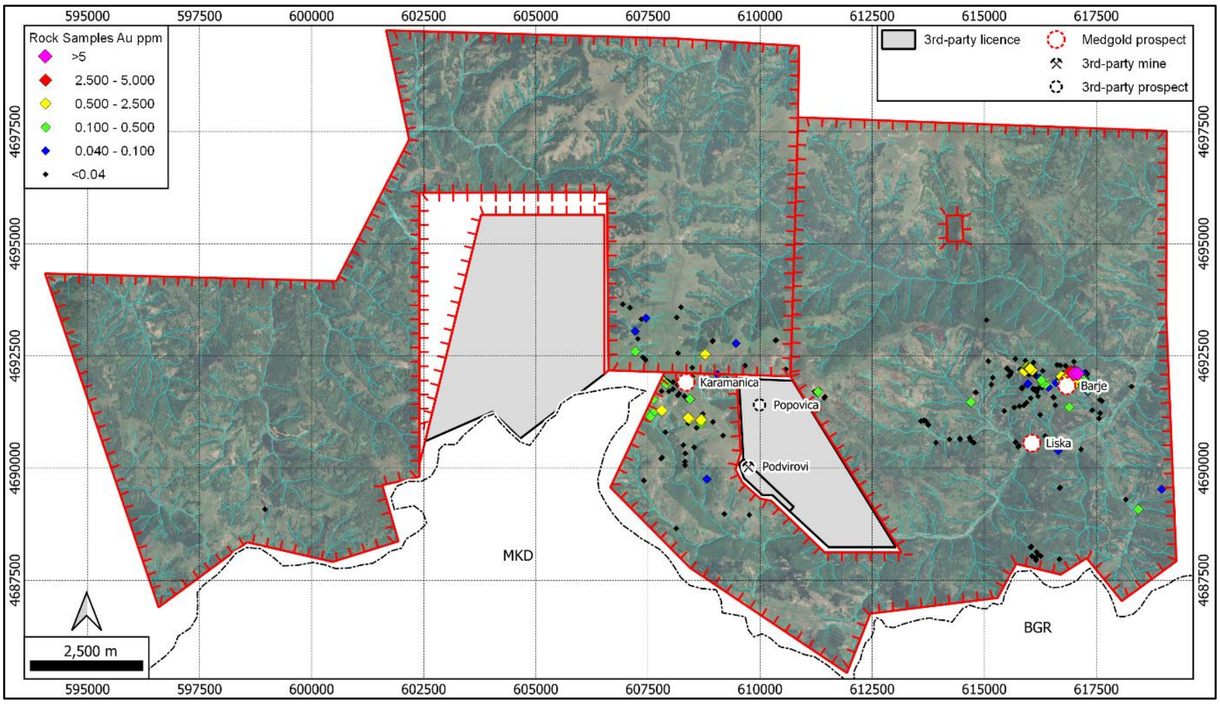


Figure 9.3. Results of analysis for rock samples collected by Avala within the Property: Gold.

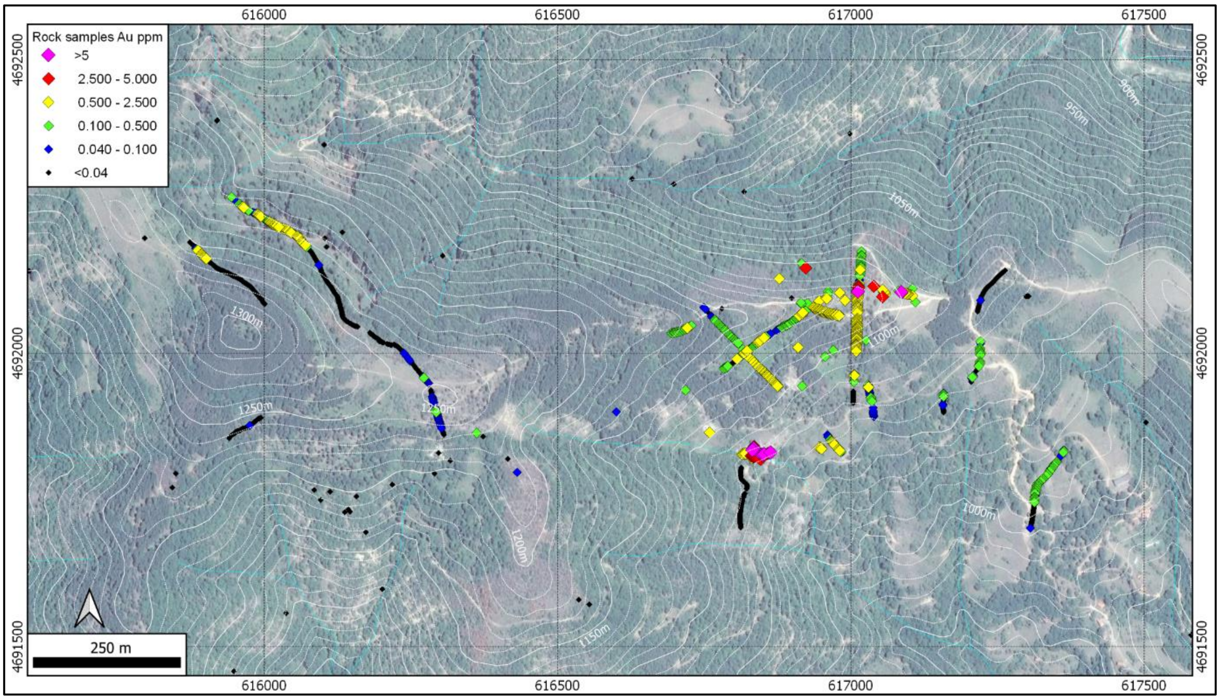


Figure 9.4. Results of analysis for rock samples collected by Avala at the Barje Deposit: Gold.

9.1.4 Drilling – Avala

Avala drilled four diamond drill holes at the Barje Deposit (Figure 7.5), none of which intersected any significant mineralization. It is likely that Avala targeted their drilling to hit a conceptual sub-vertical structure below the discovery outcrops. Due to the location of the drill sites, all Avala's holes appear to have drilled beneath the zone of flat- to shallow-dipping mineralization at Barje.

9.2 Exploration work performed by Medgold

9.2.1 Soil geochemistry

Soil samples were collected by Medgold between 2016 and 2019 with four aims, specifically to better define anomalies identified by Avala's ridge-and-spur soil sample work at the Karamanica Prospect, to confirm soil geochemistry levels in Avala's data at the Barje Deposit, to extend the historical soil sample coverage surrounding the Barje and Liska Prospects, and to continue soil sampling along a structural corridor related to the detachment surface between the amphibolite basement and the overlying schist in the north-eastern part of the Property. See Figure 9.5 for sample locations and results for gold analysis.

Results from Medgold's soil geochemical sampling programme achieved the following:

- The outline of an area at the Karamanica Prospect of approximately 2000 by 1500 m with values for gold of generally >0.1 ppm. Within this anomaly, locally elevated concentrations of Ag, Cu, Pb, Zn, As and Sb form NW-SE linear features.
- Samples from the centre of the Barje Deposit confirmed similar metal concentrations to those recorded by Avala.
- Extension of sampling to the east, south and west of the Avala samples surrounding Barje and Liska indicated only weak and/or small anomalies.
- Ridge-and-spur sampling in the area of the detachment structure in the northeast of the Property indicated only weak and/or small anomalies.

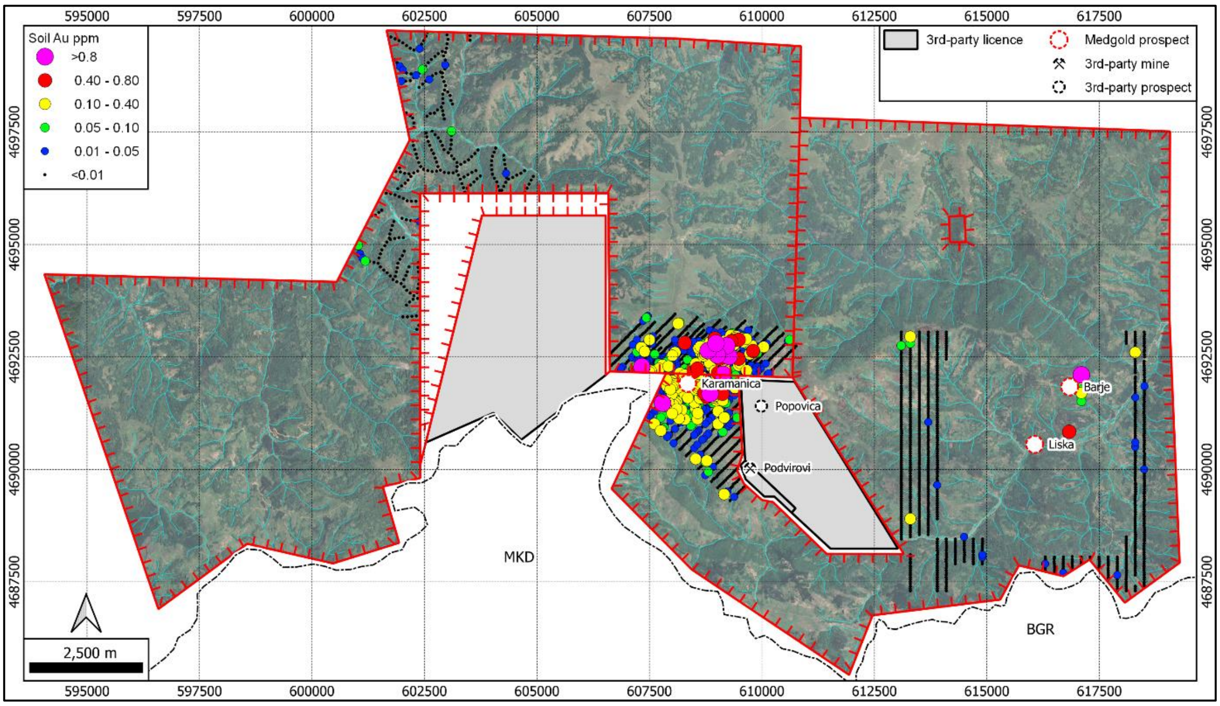


Figure 9.5. Results of analysis for soil samples collected by Medgold: Gold.

9.2.2 Lithogeochemistry

Rock samples were collected by Medgold between 2016 and 2019. Medgold's samples can be grouped by those samples collected during prospecting and mapping, and those samples taken as lines of chips and channel samples at Barje and Liska (Figure 9.6).

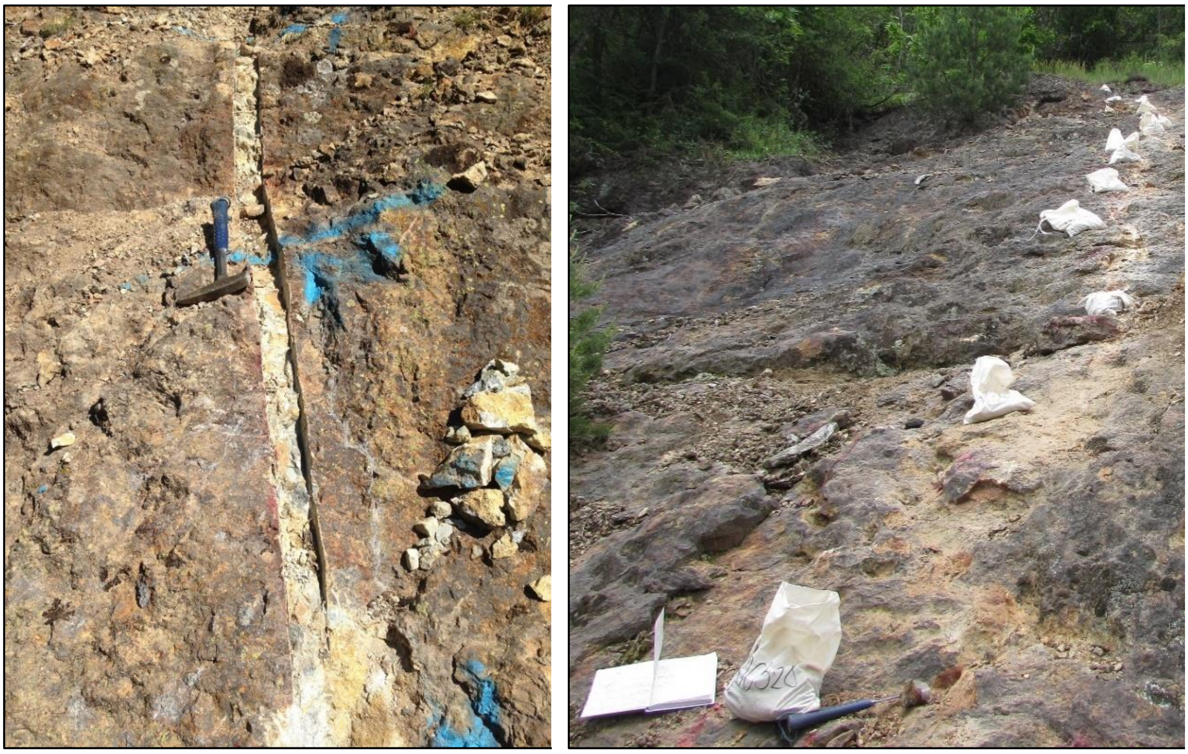


Figure 9.6.Example of channel samples and chip sample lines.
Part of a channel sample at Barje (right) and a line of chip samples at Liska (left). Photos from Medgold.

Significant results from Medgold's rock sampling include the following:

- Samples collected outside the Prospects of Karamanica, Barje and Liska were limited and returned no significant gold values (Figure 9.7).
- Many elevated values for gold were returned from rock samples at the Karamanica Prospect.
- Samples from cut channels on the main discovery outcrop at Barje (Figure 9.8).confirmed high gold, silver and base-metal values including a line of adjacent, 1 m length, cut channel samples over 84 m with an average gold grade of 5.6 g/t and 105 g/t silver (Medgold Resources Corp, 2017b).
- Chip sampling across the Liska discovery outcrop returned weakly elevated gold up to 0.5 g/t with only two samples returning higher grades of 0.53 and 0.74 g/t gold.

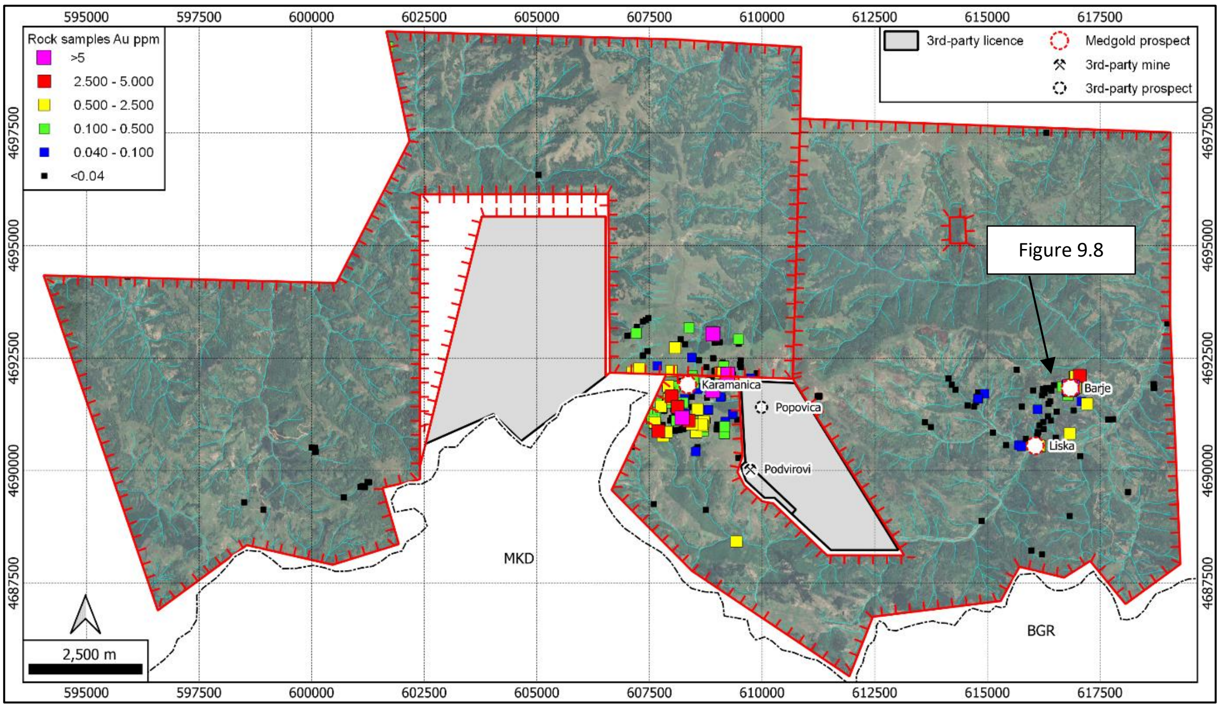


Figure 9.7. Results of analysis for rock samples collected by Medgold within the Property: Gold.

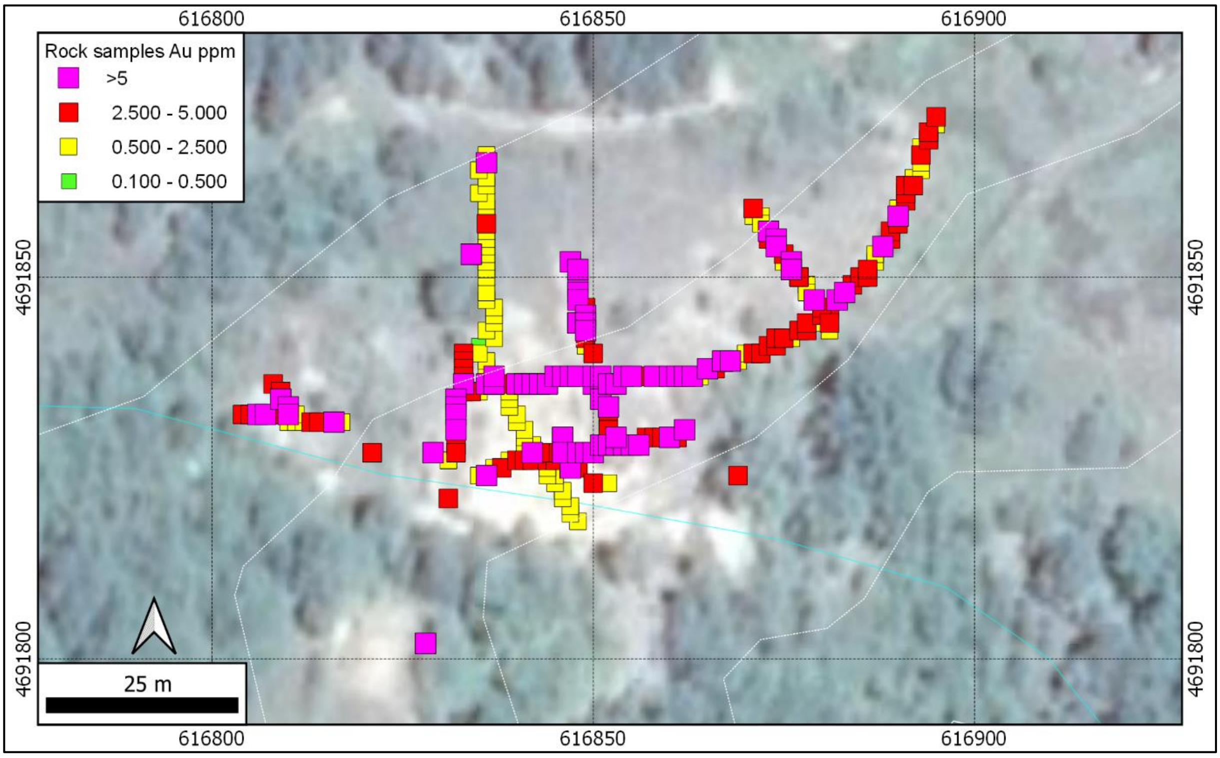


Figure 9.8. Results of analysis for rock samples collected by Medgold at the Barje Deposit: Gold.

9.2.3 Geophysics

Two geophysical campaigns were carried out in the Property by Medgold. In 2017, an Induced Polarization (IP) survey was carried out over the Barje Deposit and Liska Prospect. In 2018, additional lines were added to extend the IP coverage at Liska and Barje and a survey was carried out over the Karamanica Prospect.

9.2.3.1 2017 Barje-Liska IP survey:

Geophysical data in 2017 were collected by Géophysique TMC of Val-d'Or, Canada. Survey parameters are summarized as follows:

- 35.625 line-km along 22 lines with 100 m line separation.
- Pole-dipole array with electrode spacings of 25 or 50 m with 10 dipoles read.
- Locations surveyed by real-time a Differential Global Positioning System (DGPS).
- TX-II GDD transmitter injecting a bipolar current waveform with an 8-second period with a 50% duty cycle. Transmitter power of 3.6 kw with average injected current of 1.6 A.
- IRIS Instruments ElrecPro Instruments time domain receiver.

Data plots were provided by Géophysique TMC as inverted 2D pseudo-sections and slices of a 3D inversion shown at a series of fixed depths below surface. Results for the Barje-Liska survey are shown as a slice at 100 m below surface for chargeability and resistivity in Figure 9.9 and Figure 9.10. The results outlined the area of the historical State drilling at Liska to have high chargeability (>15 mV/V) with corresponding high resistivity. A chargeability high is present in the north of the survey area to the west of the Barje mineralization. This does not however directly coincide with areas of high resistivity.

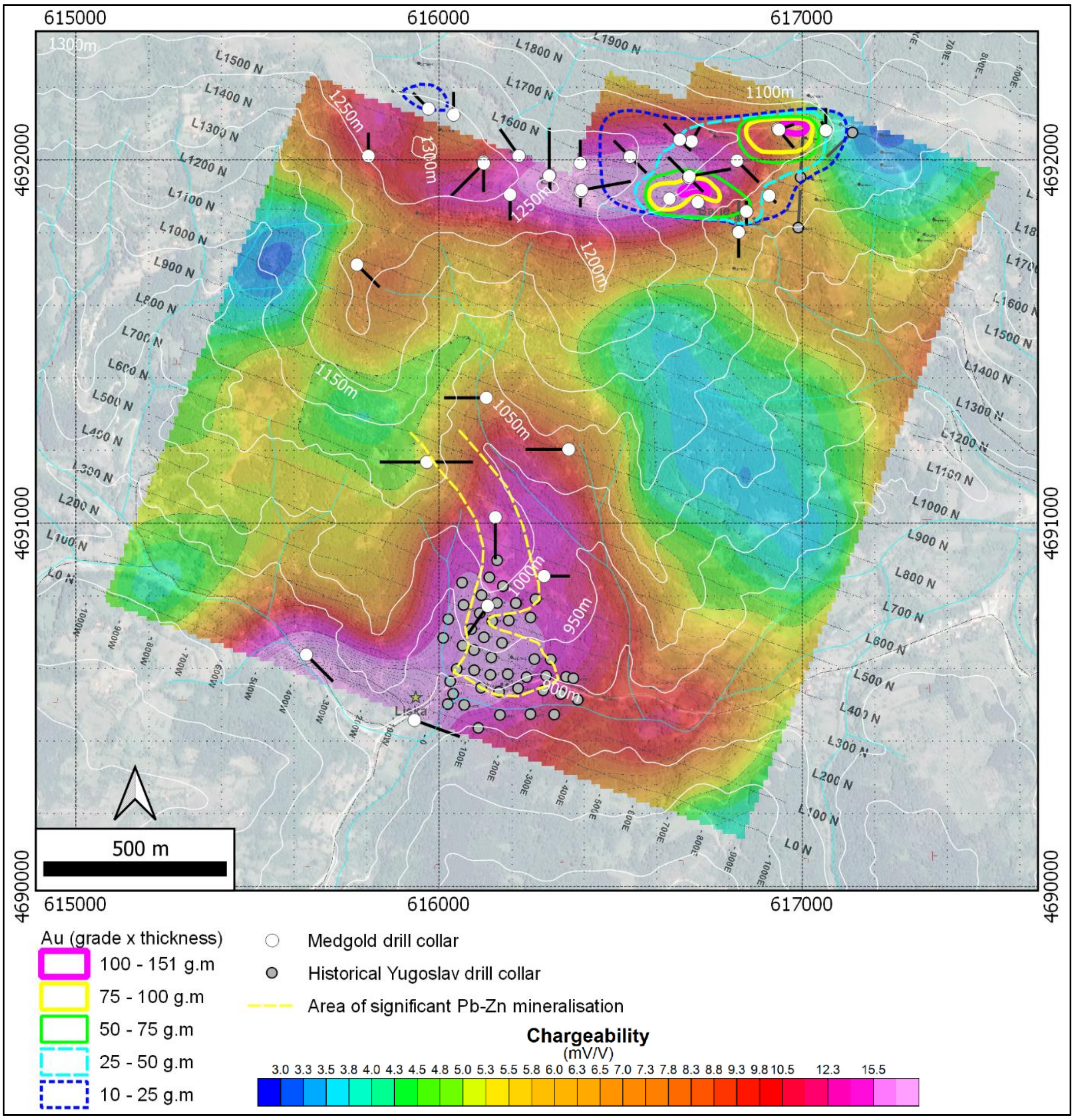


Figure 9.9. 2017 IP survey results at Barje-Liska; chargeability 100 metres below surface.
Also shown are the historical drill locations and Medgold drill locations and summary outlines from the later 2018 and 2019 drill programmes.

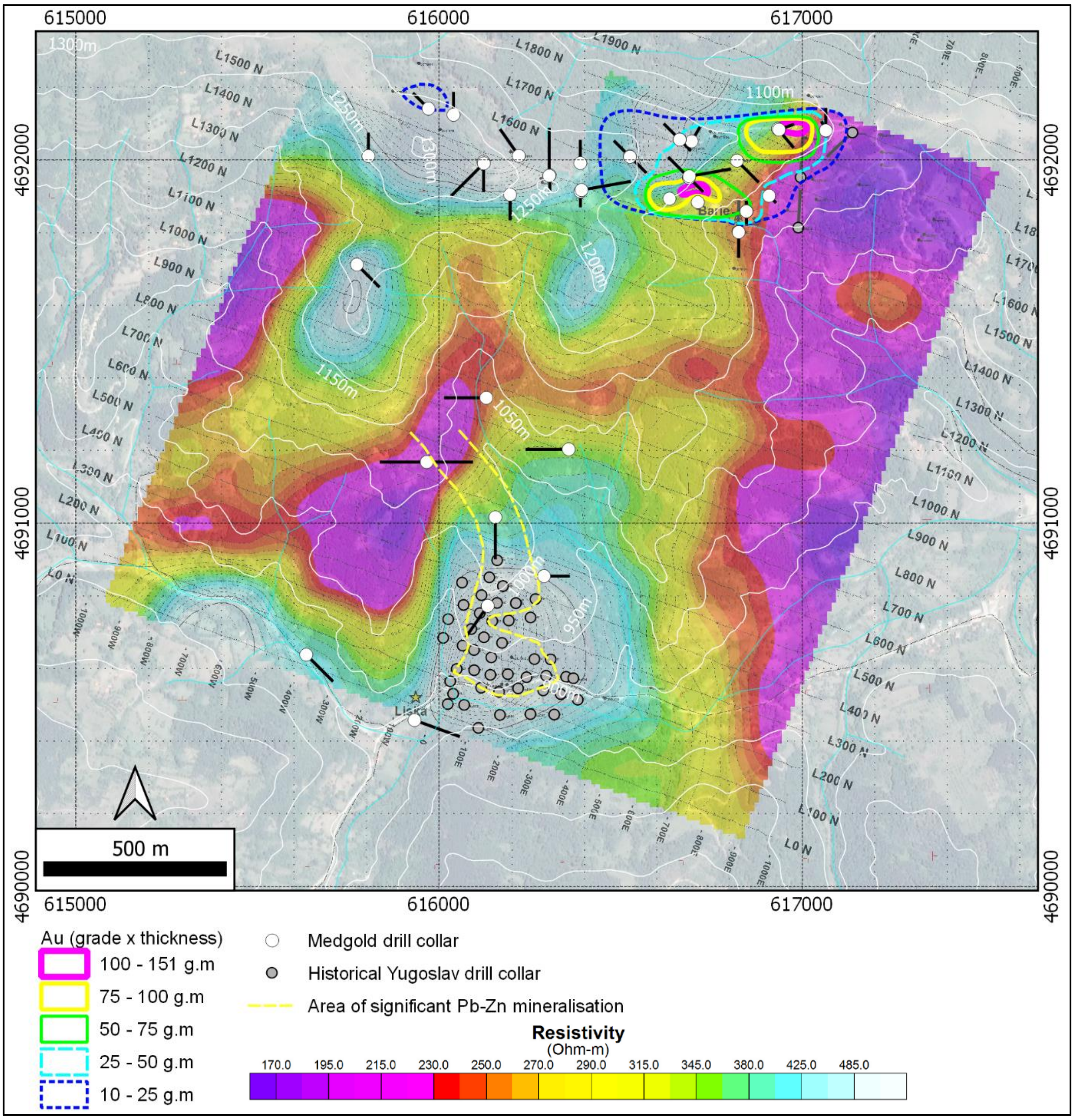


Figure 9.10. 2017 IP survey results at Barje-Liska; resistivity 100 metres below surface.
Also shown are the historical drill locations and Medgold drill locations and summary outlines from the later 2018 and 2019 drill programmes.

9.2.3.2 2018 IP surveys

Geophysical data in 2018 were collected by Enerson Mühendislik Sond. Mad. Pet. Jeo. Mak. Elek. San. Tic. Ltd. Şti. ("Enerson") of Ankara, Turkey. Survey parameters are summarized as follows:

- 35.65 line-km along 12 lines with 200 m line separation at Karamanica.
- 6.75 line-km along 4 lines with 100 m line separation at Liska.
- 3.75 line-km along 3 lines with 100 m line separation at a geochemistry anomaly located approximately 2 km west of Barje.

- Modified pole-dipole array with electrodes at 25, 25, 50, 100, 100, 100, 200, 200 m spacing and the number of dipoles read being 8. Arrays moved in steps of 100 m.
- Locations marked by survey pegs and surveyed by DGPS after data collection.
- IRIS Instruments VIP10000 transmitter injecting a bipolar current waveform with an 8-second period with a 50% duty cycle. Injected voltage of up to 3200 V at 1 to 5 A.
- IRIS Instruments ElrecPro Instruments time domain receiver.

Data plots were provided by Enerson as inverted 2D sections and slices of a 3D inversion shown at a series of fixed depths below surface.

Results from the Karamanica survey are shown as a slice at 100 m below surface for chargeability and resistivity in Figure 9.11 and Figure 9.12 The data show multiple northwest – southeast features of high chargeability (20 to >30 mV/V), which partly coincide with parallel to sub-parallel zones of low resistivity. Of note is the spatial relationship between rock samples with higher gold grades and the strongest and longest NW-SE feature of chargeability.

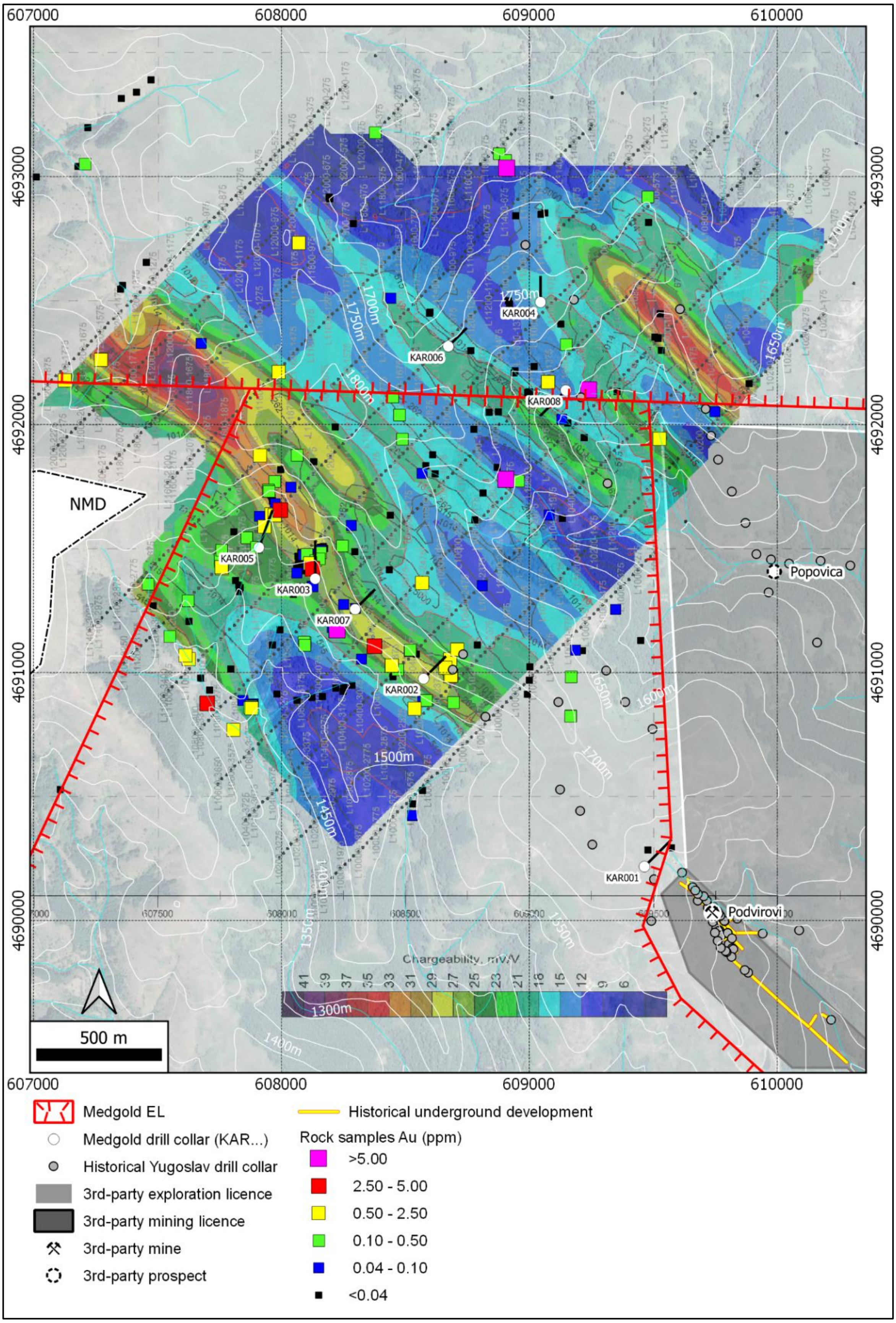


Figure 9.11. 2017 IP survey results at Karamanica; chargeability 100 metres below surface.
Also shown are Medgold drill locations from 2018 and 2019 and Medgold rock sample results for gold.

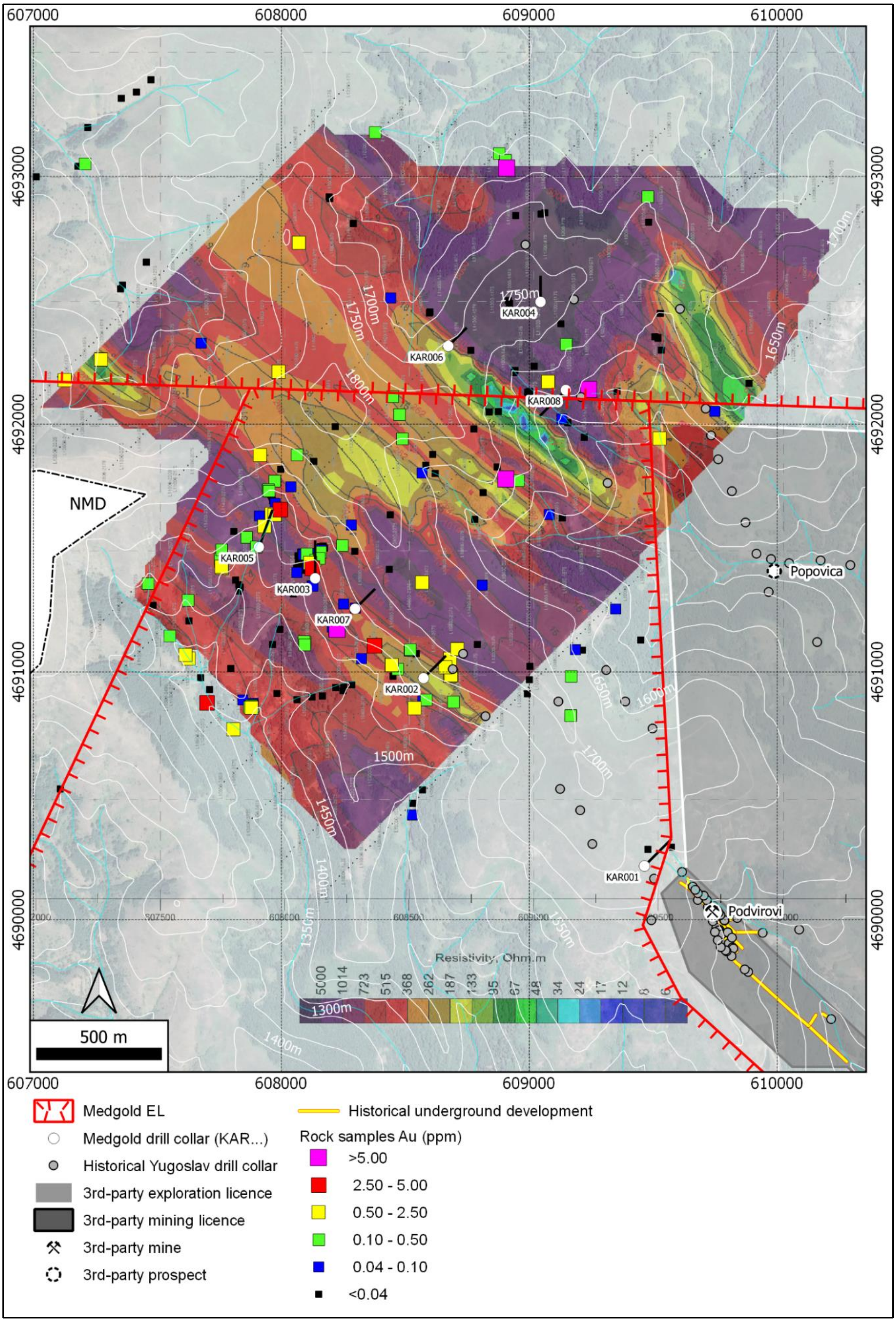


Figure 9.12. 2017 IP survey results at Karamanica, resistivity 100 metres below surface.
Also shown are Medgold drill locations from 2018 and 2019 and Medgold rock sample results for gold.

Results from the extension to the 2017 Barje-Liska survey are shown as a slice at 100 m below surface for chargeability and resistivity in Figure 9.13 and Figure 9.14. The data from the 2018 extension show a clear extension of the chargeability high located southwest of the Liska Prospect with values above 10 mV/V. The resistivity high seen at Liska is present within the high chargeability of the 2018 data but is not so apparent due to the colour ranges used to present the 2018 survey results.

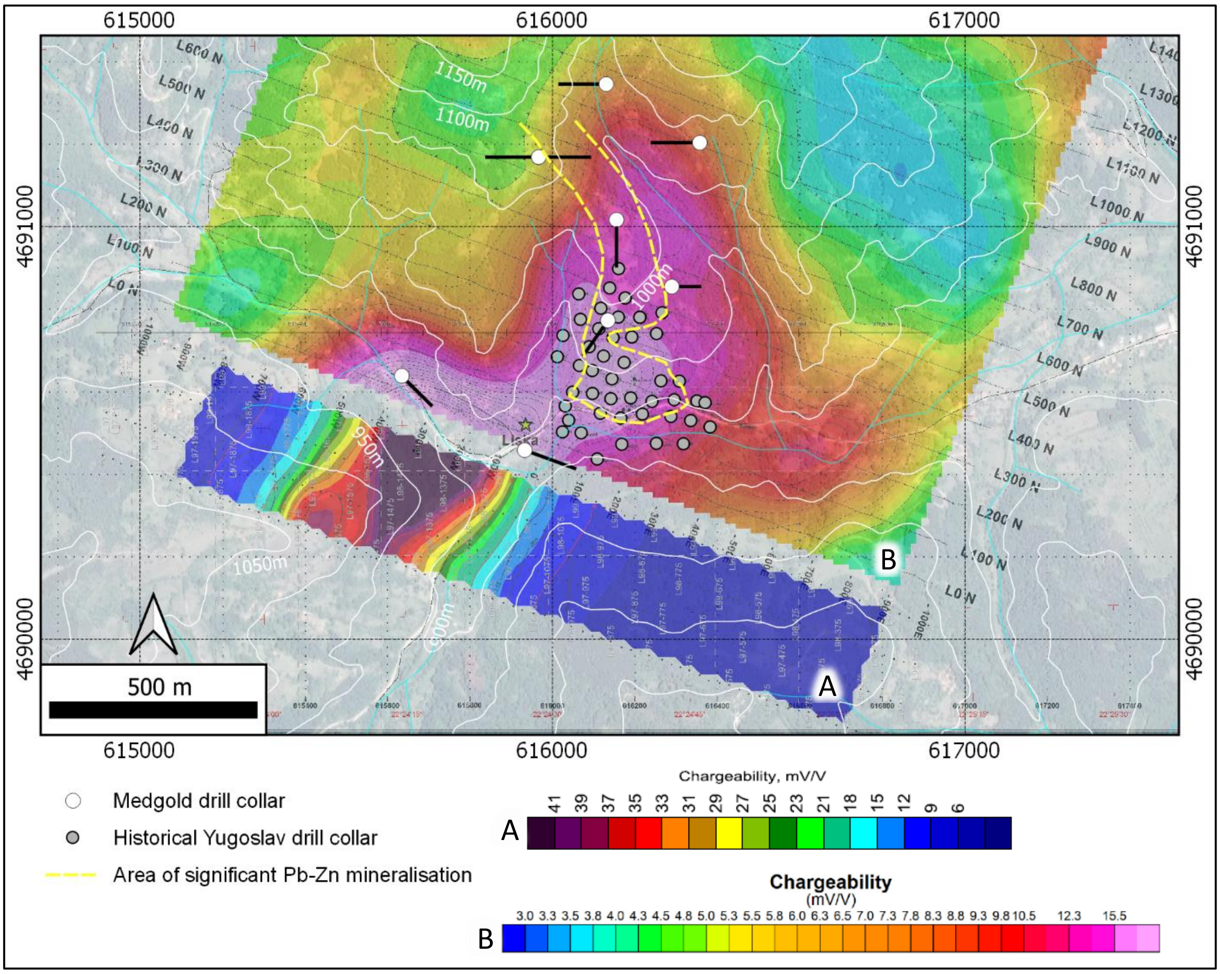


Figure 9.13. 2018 extension of IP survey at Liska; chargeability 100 metres below surface.
Also shown are historical drill locations, results of the 2017 IP survey, Medgold's 2019 drill locations and summary outlines. Note that the resistivity scale for the 2018 data (A) is inverted and colour values are not the same as for the 2017 data (B).

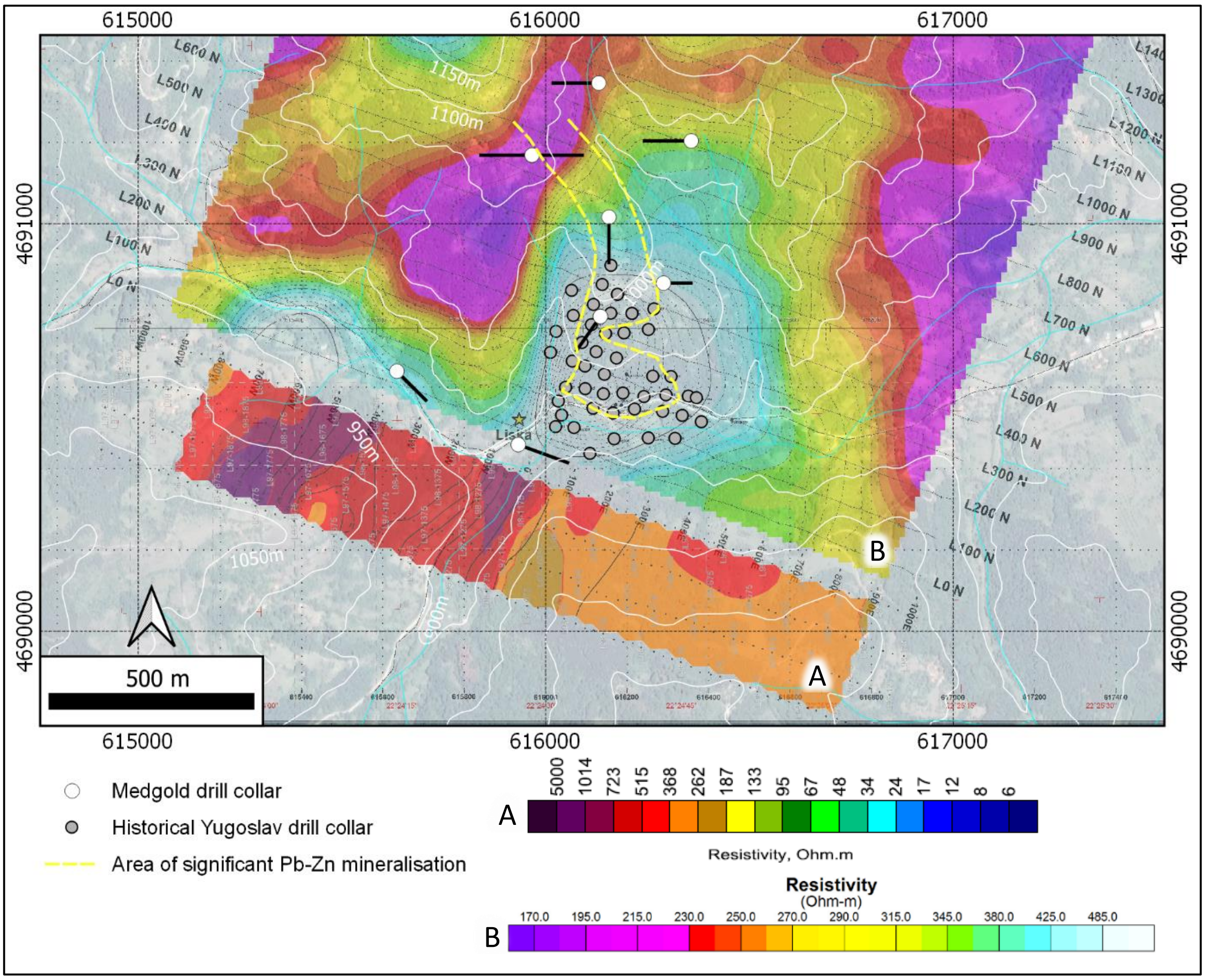


Figure 9.14. 2018 extension of IP survey at Barje-Liska.; resistivity 100 metres below surface.
Also shown are historical drill locations, results of the 2017 IP survey, Medgold's 2019 drill locations and summary outlines.
Note that the resistivity scale for the 2018 data (A) is inverted and colour values are not the same as for the 2017 data (B).

Three scout lines of IP carried out approximately 2 km west of the Barje mineralization (over a geochemical anomaly) were hampered by steep terrain. Due to the 200 m line separation, the limited number of lines, and some missing data points due to the terrain, processing of the results was carried out only in 2D. Inverted 2D sections for the lines are show in Figure 9.16 to Figure 9.18. Results show a near-surface chargeability zone on lines L14500E and L14700E which extends to depth on line L14900E. Near-surface resistivity values are highly variable possibly due to changing contact resistance of each receiving electrode caused by the variable ground conditions (deep to shallow or no soil cover).

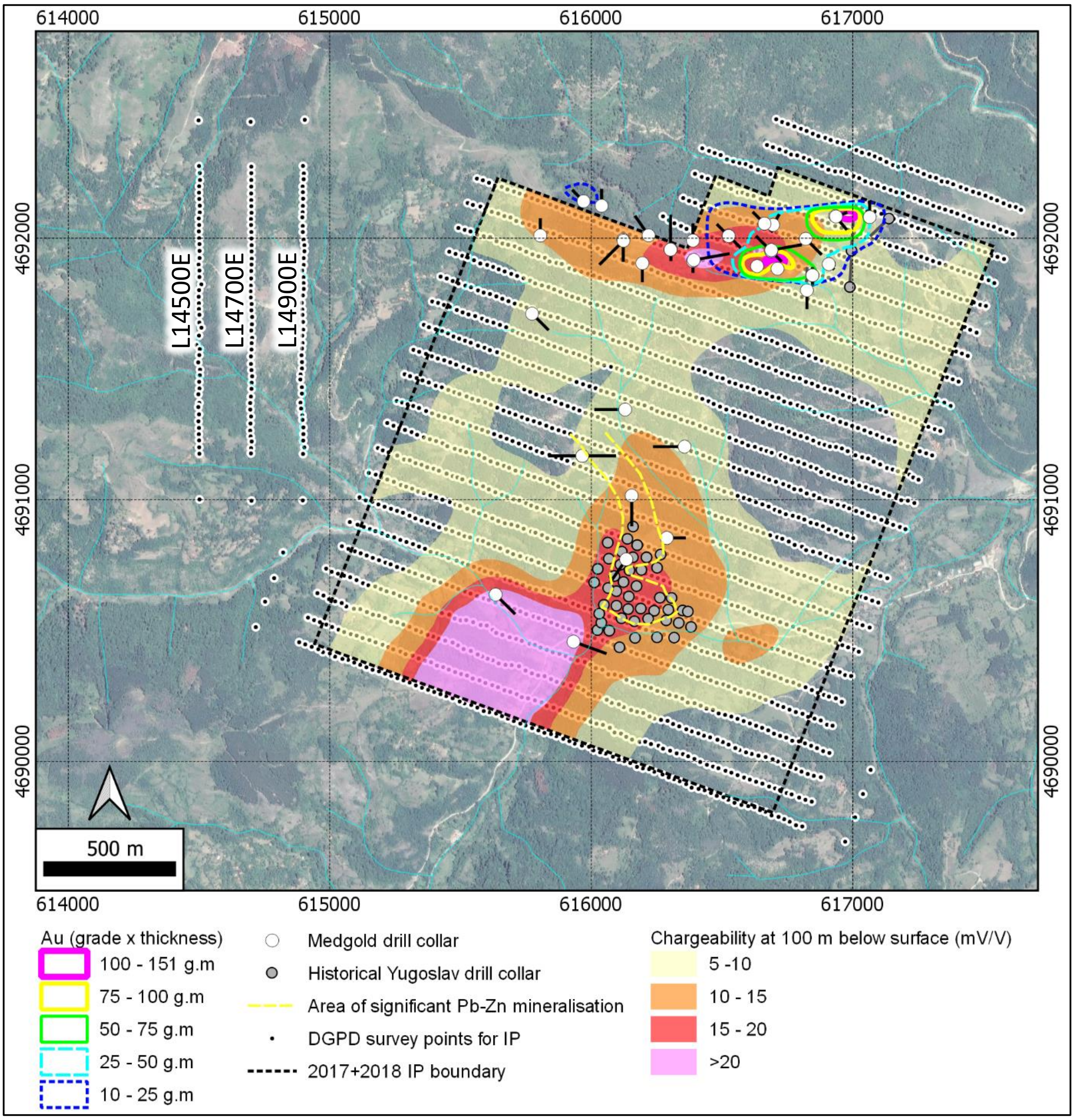


Figure 9.15. Summary of 2017 and 2018 IP chargeability results at Barje-Liska.
Also showing the location of IP survey lines at Barje West during 2018.

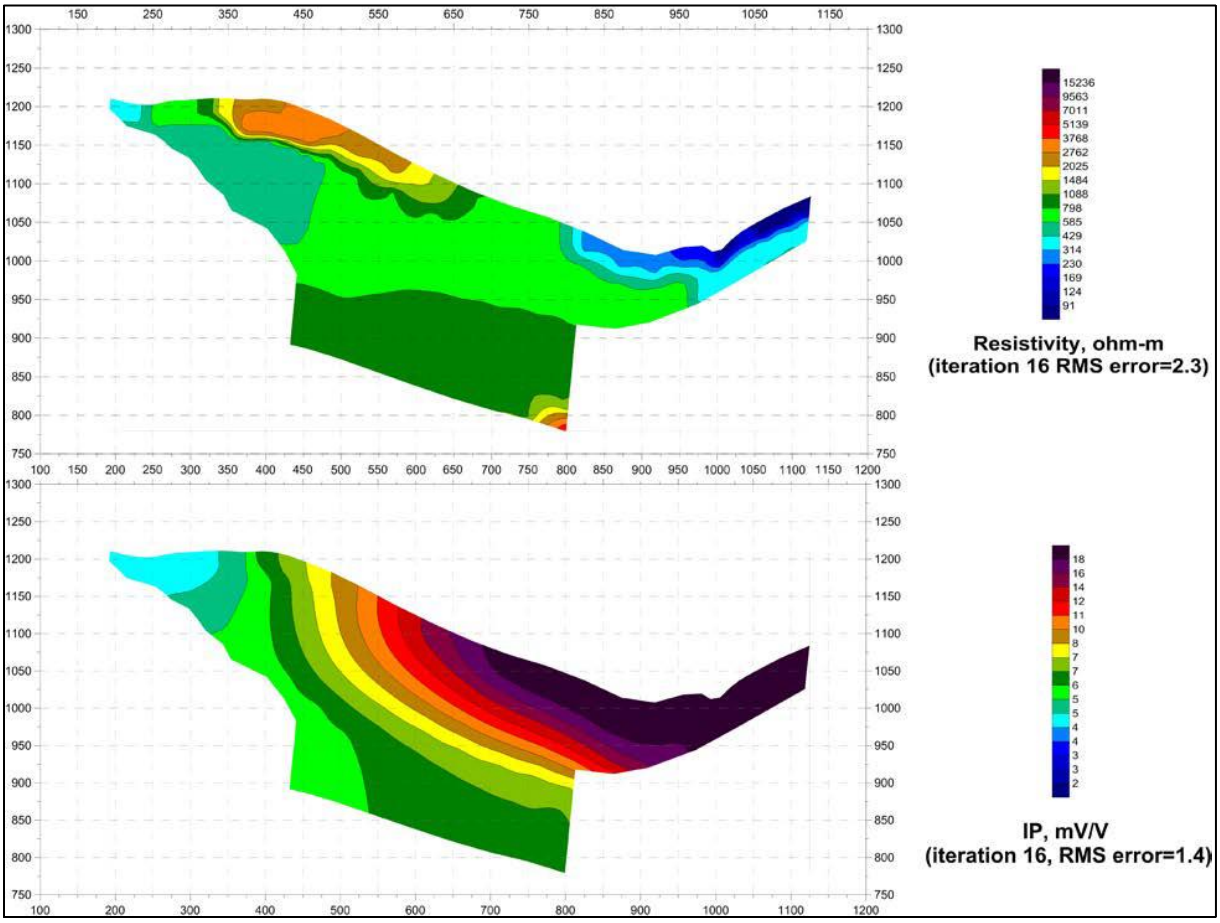


Figure 9.16. 2017 IP chargeability and resistivity pseudo-section at Barje west. Line L14500E.

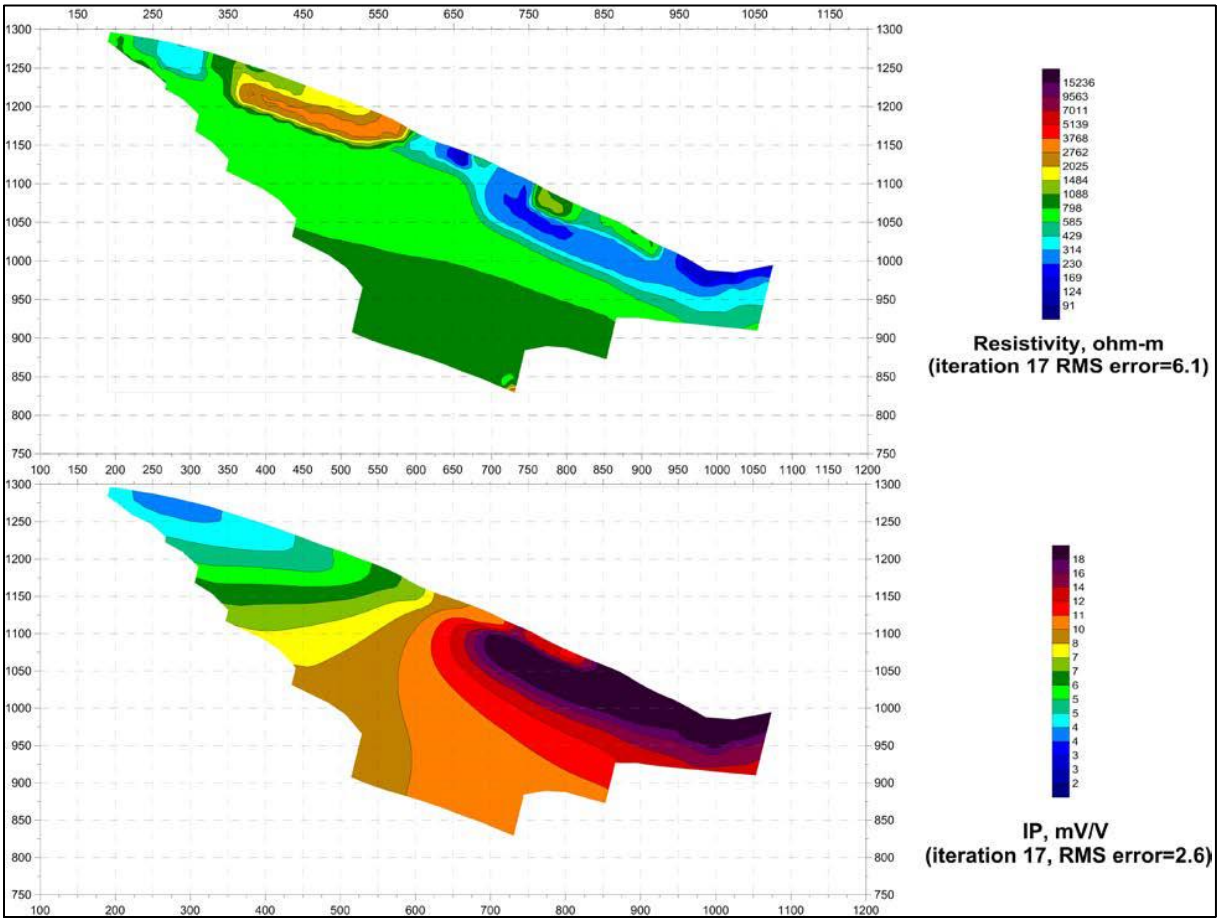


Figure 9.17. 2017 IP chargeability and resistivity pseudo-section at Barje west. Line L14700E.

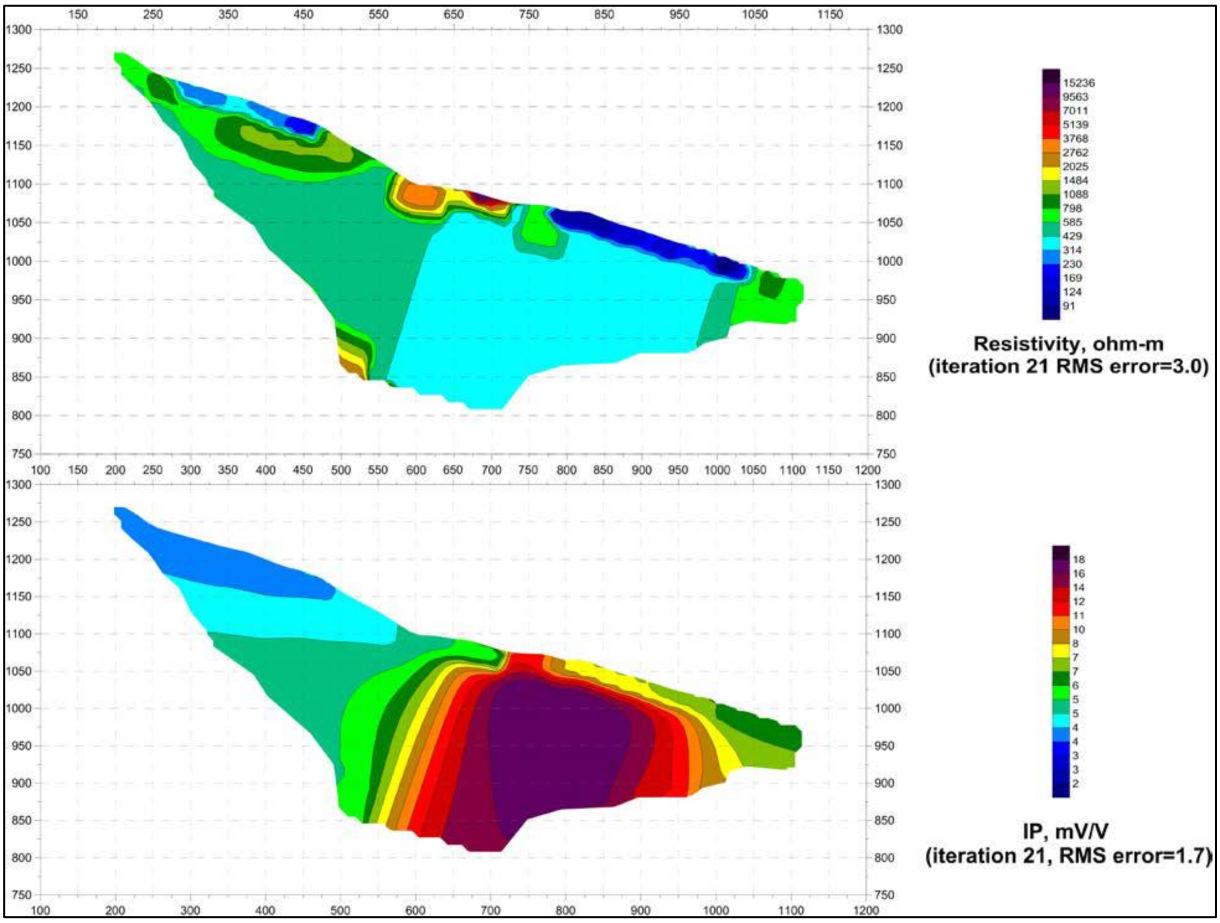


Figure 9.18. 2017 IP chargeability and resistivity pseudo-section at Barje west. Line L14900E.

9.2.4 Conclusions Regarding Exploration work performed by Medgold

The Qualified Person did not observe any fieldwork or sampling during their site visit to the Property as no active work was being conducted at that time.

From discussion with Medgold geologists and consultants and observations made during the site visit, the Qualified Person has no reason to doubt the exploration work presented in Section 9, carried out by both Avala and Medgold, was systematic and representative in nature and carried out to normal industry practices with no significant bias. As such, the data generated is suitable for use in early-stage exploration projects to allow identification of drill targets for follow-up as described in Section 10.

10 Drilling

10.1 Drilling details

Medgold carried out drilling at the Barje Deposit and Liska and Karamanica prospects within the Property in multiple phases between May 2018 and October 2019 as shown in Table 10.1. Drill hole groupings are shown in Figure 10.1. Examples of drilling activities are shown in Figure 10.2.

Table 10.1. Summary of drilling on the Property by Medgold.

Phase	Year	No. of holes	Total metres	Average recovery	Total samples
Barje phase 1	2018	7	734.1	89%	580
Barje phase 2	2018	23	3903.1	92%	2726
Barje phase 3	2019	3	354.3	93%	132
Barje - total drilling		**33**	**4991.5**	**92%**	**3438**
Liska phase 1	2018	2	401.0	94%	272
Liska phase 2	2019	8	1738.4	98%	443
Liska - total drilling		**10**	**2139.4**	**97%**	**715**
Karamanica phase 1	2019	8	1996.5	98%	337
Property – total drilling		**55**	**9127.4**	**94%**	**4490**

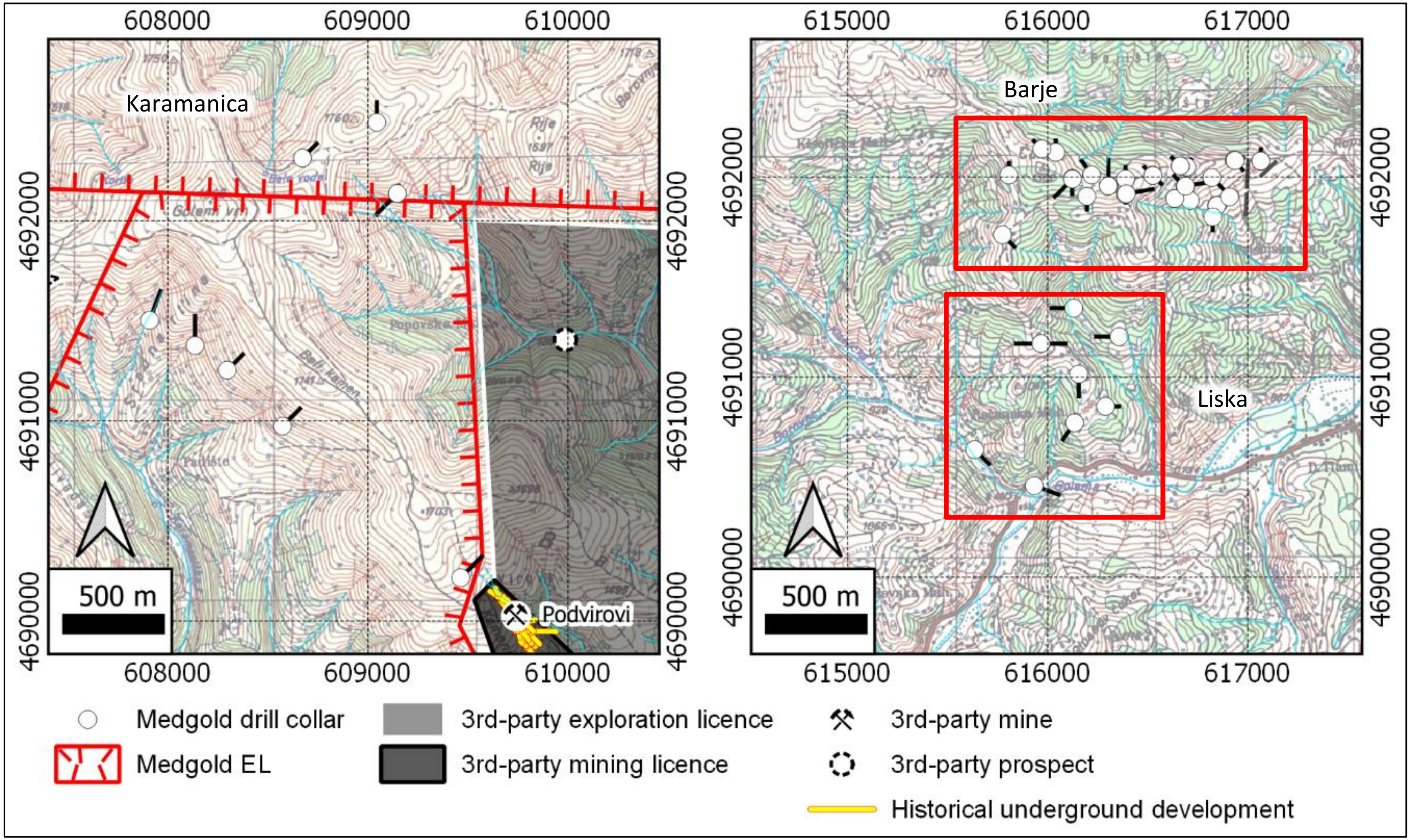


Figure 10.1. Drill holes by Prospect.
Drill locations at Karamanica (left) and Barje and Liska (right)


Figure 10.2. Examples of drilling on the Property.
BAR001 rig set-up during Barje phase 1 (left) and KAR001 drilling during Karamanica phase 1 (right)

All drilling was diamond core drilling carried out by Serbian contractors using track-mounted Atlas Copco CS-14 and Coretech CSD1300G drill rigs. Holes were collared using PQ3 diameter tooling giving nominal 83 mm diameter core, before stepping down to HQ3 tooling (61 mm diameter core) when into more competent ground conditions. Planned inclinations for drill holes varied from -90° to -45°. End of hole depths varied from 49.1 m to 311.0 m, details of all drill holes can be seen in Table 10.2.

Table 10.2. Drill hole details. Collar coordinates are rounded to the nearest meter.

Drill hole	Easting	Northing	Elevation (m)	Azimuth (°)	Inclination (°)	Length (m)	Phase
BAR001	616847	4691858	1082	180	-50	54.90	Barje phase 1
BAR002	616933	4692085	1131	140	-50	99.30	Barje phase 1
BAR003	616846	4691859	1082	0	-60	49.10	Barje phase 1
BAR004	616846	4691858	1082	0	-90	100.20	Barje phase 1
BAR005	616820	4691998	1130	135	-50	125.60	Barje phase 1
BAR006	616688	4691951	1144	135	-70	151.60	Barje phase 1
BAR007	616688	4691951	1144	315	-60	153.40	Barje phase 1
BAR008	616713	4691883	1114	180	-80	92.80	Barje phase 2
BAR009	616909	4691901	1081	135	-65	56.10	Barje phase 2
BAR010	616936	4692083	1131	70	-60	80.50	Barje phase 2
BAR011	616696	4692051	1172	30	-65	98.50	Barje phase 2
BAR012a	616663	4692056	1179	315	-60	80.50	Barje phase 2
BAR012b	616663	4692056	1179	130	-60	116.50	Barje phase 2
BAR013	616635	4691893	1145	0	-90	143.60	Barje phase 2
BAR014	616689	4691955	1144	80	-50	174.60	Barje phase 2
BAR015	616525	4692010	1206	135	-70	176.70	Barje phase 2
BAR016	616526	4692009	1206	315	-65	139.30	Barje phase 2
BAR017	616389	4691991	1232	0	-70	174.70	Barje phase 2
BAR018	616389	4691913	1234	180	-80	227.90	Barje phase 2
BAR019	616392	4691917	1234	80	-60	267.60	Barje phase 2
BAR020	616305	4691958	1248	180	-80	224.70	Barje phase 2
BAR021	616303	4691956	1249	0	-50	199.10	Barje phase 2
BAR022	616220	4692011	1242	325	-65	203.70	Barje phase 2
BAR023	616196	4691904	1242	180	-75	257.70	Barje phase 2
BAR024	616123	4691994	1262	180	-74	286.90	Barje phase 2
BAR025	616123	4691991	1262	225	-60	248.70	Barje phase 2
BAR026	615805	4692010	1246	0	-70	176.30	Barje phase 2

Drill hole	Easting	Northing	Elevation (m)	Azimuth (°)	Inclination (°)	Length (m)	Phase
BAR027	615774	4691712	1208	135	-70	245.70	Barje phase 2
BAR028	616040	4692124	1263	0	-60	118.10	Barje phase 2
BAR029	615971	4692141	1264	315	-60	112.90	Barje phase 2
BAR030	616823	4691799	1077	180	-70	195.00	Barje phase 3
BAR031	617061	4692083	1106	0	-50	89.30	Barje phase 3
BAR032	617061	4692081	1077	335	-70	70.00	Barje phase 3
LIS001	615634	4690638	910	135	-60	200.60	Liska phase 1
LIS002	615932	4690457	894	110	-50	200.40	Liska phase 1
LIS003	616133	4690769	992	0	-90	179.50	Liska phase 2
LIS004	616287	4690856	994	90	-60	134.50	Liska phase 2
LIS005	616134	4690769	992	225	-60	203.50	Liska phase 2
LIS006	616155	4691014	1031	180	-60	223.50	Liska phase 2
LIS007	615965	4691179	1059	90	-60	247.00	Liska phase 2
LIS008	616129	4691344	1032	270	-60	224.20	Liska phase 2
LIS009	616351	4691205	1029	270	-60	225.20	Liska phase 2
LIS010	615963	4691177	1059	270	-60	301.00	Liska phase 2
KAR001	609472	4690235	1618	45	-60	299.50	Karamanica phase 1
KAR002	608582	4690968	1632	45	-60	245.90	Karamanica phase 1
KAR003	608139	4691381	1586	0	-60	296.50	Karamanica phase 1
KAR004	609043	4692494	1734	0	-60	200.30	Karamanica phase 1
KAR005	607908	4691504	1555	20	-60	311.00	Karamanica phase 1
KAR006	608673	4692305	1710	45	-60	201.00	Karamanica phase 1
KAR007	608299	4691262	1638	45	-60	218.00	Karamanica phase 1
KAR008	609157	4692138	1615	225	-50	224.30	Karamanica phase 1

All drill holes were collared at pre-planned sites marked by Medgold's geologists, who checked the drill azimuth and inclination during rig set-up at every site. Down hole surveys to record the azimuth and inclination of each drill hole were carried out by the drill contractors on nominal 20 m down-hole spacings for Barje phase 1 holes and nominal 30 m spacing for all other holes. Survey tools used were a Devico Devitool for Barje phase 1 holes and a Reflex EZ-TRAC for all other holes.

Holes were capped and marked by a cement block after drilling. Collar locations were recorded by a certified Serbian surveyor using DGPS at the end of each phase of drilling.

10.2 Core handling and logging

A documented system of core handling procedures was used at all times during the drilling programmes.

Drill core was received from the drillers after core extraction into galvanized metal core boxes. Core boxes were photographed at the drill site by Medgold's on-site personnel before being transported to Medgold's core-yard in Bosilegrad where core boxes were laid onto logging racks. Core blocks recording individual drill run intervals were checked before recovery and RQD measurements were made. Meter marks were put onto all core trays before a high-resolution photograph was taken. Logging and sample definition were then performed by Medgold's geologists, after which sample intervals and numbers were marked onto the core boxes. The core was then cut and sampled (see Section 11) before a final high-resolution photo was taken of the core boxes.

Logging data during 2018 was captured using paper forms before entry into Excel sheets and import into a Geospark database developed by Geospark Consulting Inc of British Columbia. Logging data during 2019 was captured directly into WiFi enabled tablets linked to an MX Discover cloud-hosted database maintained by Geosoft Inc of Ontario.

10.3 Summary of results

Significant intersections reported by Medgold (Medgold Resources Corp, 2019a) (Medgold Resources Corp, 2019b) (Medgold Resources Corp, 2019c) are shown in Table 10.3.

Table 10.3. Significant intersections of within drill core.
Length weighted average applied; internal waste included in calculation. See sections 10.3.1, 10.3.2 and 10.3.3 below for the relationship between the interval length and the true thickness of . Results are as reported in Medgold News Releases.

Drill Hole	From (m)	To (m)	Interval (m)	Interval inclination (°)	True thickness (m)*	Au g/t	Ag g/t	As%	Pb%	Zn%	Pb-Zn combined%
BAR001	2.38	33.20	30.82	-50	23.7	2.06	55	0.67	0.17	0.37	0.54
BAR002	13.35	48.00	34.65	-52	27.3	3.11	28	0.76	0.13	0.30	0.43
Inc.	42.00	44.00	2.00	-53	1.6	23.88	340	3.29	1.08	2.61	3.69
BAR003	2.00	28.10	26.10	-59	22.3	2.44	219	0.92	0.11	0.19	0.30
inc.	3.60	9.60	6.00	-58	5.1	4.20	754	0.88	0.27	0.09	0.36
BAR004	2.20	24.30	22.10	-90	22.1	1.83	109	0.61	0.17	0.34	0.51
BAR005	1.60	102.40	100.80	-52	78.9	0.52	5	0.26	0.07	0.27	0.34
inc.	82.15	102.40	20.25	-51	15.8	1.08	11	0.05	0.25	1.14	1.39
BAR006	74.00	104.00	30.00	-68	27.8	5.45	25	1.47	0.25	0.76	1.01
inc.	85.00	95.00	10.00	-68	9.3	2.73	10	1.22	0.09	0.24	0.33
and	95.00	104.00	9.00	-67	8.3	14.17	58	2.70	0.69	2.11	2.80
BAR007	53.00	77.00	24.00	-58	20.4	0.52	4	0.44	0.01	0.02	0.03
and	89.50	101.60	12.10	-58	10.3	3.40	12	1.37	0.11	0.30	0.41
BAR008	41.45	54.80	13.35	-81	13.2	5.06	109	0.74	1.44	2.85	4.29
inc.	48.95	54.80	5.85	-83	5.8	10.35	484	0.50	5.03	10.67	15.70
BAR009	0.00	28.00	28.00	-66	25.5	0.86	7	0.52	0.02	0.08	0.10
BAR010	19.80	58.00	38.20	-61	33.5	3.98	158	0.85	0.21	0.45	0.66
inc.	48.15	55.00	6.85	-61	6.0	13.49	788	1.36	1.01	2.08	3.09
BAR011	62.00	72.75	10.75	-66	9.8	4.76	33	2.93	0.16	0.51	0.67
BAR012b	57.80	73.00	15.20	-61	13.3	1.68	10	1.55	0.14	0.35	0.49
BAR013	70.50	89.10	18.60	-90	18.6	3.09	22	1.31	0.40	0.87	1.27
BAR014	87.00	120.00	33.00	-51	25.5	1.14	10	0.45	0.05	0.12	0.17
inc.	117.70	120.00	2.30	-52	1.8	7.46	111	1.32	0.58	1.21	1.79
BAR015	135.00	143.20	8.20	-70	7.7	2.29	19	1.62	0.40	1.06	1.46
BAR016	101.00	120.90	19.90	-65	18.1	1.01	11	0.48	0.03	0.10	0.13
BAR017	No significant intersection of mineralization										
BAR018	No significant intersection of mineralization										
BAR019	154.90	167.90	13.00	-62	11.5	0.74	5	0.58	0.10	0.26	0.36
and	201.00	208.60	7.60	-62	6.7	0.70	4	0.63	0.05	0.16	0.21
BAR020	No significant intersection of mineralization										
BAR021	No significant intersection of mineralization										
BAR022	No significant intersection of mineralization										
BAR023	No significant intersection of mineralization										
BAR024	174.00	175.10	1.10	-90	1.1	1.64	4	0.39	0.00	0.00	0.00
BAR025	180.65	181.50	0.85	-55	0.7	1.53	2	0.02	0.00	0.00	0.00

Drill Hole	From (m)	To (m)	Interval (m)	Interval inclination (°)	True thickness (m)*	Au g/t	Ag g/t	As%	Pb%	Zn%	Pb-Zn combined%
BAR026	No significant intersection of mineralization										
BAR027	No significant intersection of mineralization										
BAR028	22.00	28.00	6.00	-60	5.2	1.45	1	0.03	0.00	0.00	0.00
BAR029	2.00	21.00	19.00	-61	16.6	1.34	1	0.06	0.00	0.00	0.00
BAR030	No significant intersection of mineralization										
BAR031	0.00	32.90	29.40	-52	23.2	0.70	5	0.15	0.10	0.20	0.30
inc.	15.80	27.80	8.50	-52	6.7	1.49	10	0.08	0.24	0.65	0.89
BAR032	0.00	15.50	14.50	-71	13.7	2.13	7	0.22	0.13	0.27	0.40
inc.	10.00	12.00	2.00	-71	1.9	7.35	23	0.19	0.70	1.36	2.06
LIS001	No significant intersection of mineralization										
LIS002	No significant intersection of mineralization										
LIS003	76.70	108.70	32.00	-90	32.0	0.12	2	0.01	0.41	0.60	1.01
LIS004	No significant intersection of mineralization										
LIS005	90.90	150.25	60.05	-60	52.0	0.13	2	0.01	0.24	0.59	0.83
LIS006	120.60	152.95	32.35	-61	28.3	0.25	7	0.02	1.18	2.30	3.48
and	172.60	223.50	50.90	-59	43.6	0.03	3	0.00	0.12	0.97	1.10
LIS007	198.60	228.60	30.00	-58	25.4	0.11	8	0.02	1.12	2.83	3.95
inc.	212.60	217.60	5.00	-57	4.2	0.24	15	0.02	1.89	6.32	8.21
LIS008	No significant intersection of mineralization										
LIS009	No significant intersection of mineralization										
LIS010	No significant intersection of mineralization										
KAR001	124.80	132.80	8.00	-61	6.0	0.29	3	0.42	0.03	0.10	0.13
KAR002	102.70	104.00	1.30	-59	1.0	0.41	10	0.08	0.11	0.60	0.71
and	241.00	243.00	2.00	-57	1.6	0.90	0	0.00	0.01	0.05	0.05
KAR003	78.00	80.00	2.00	-59	1.6	0.35	35	0.66	0.10	0.16	0.26
KAR004	No significant intersection of mineralization										
KAR005	58.20	68.90	4.00	-60	3.1	0.36	41	0.02	0.12	0.10	0.22
KAR006	No significant intersection of mineralization										
KAR007	45.00	47.00	2.00	-58	1.6	0.31	5	0.09	0.12	0.86	0.97
KAR008	No significant intersection of mineralization										

10.3.1 Barje

At the Barje Deposit, drilling intersected high-grade gold ± silver, lead and zinc controlled by a hydrothermal breccia following a flat lying to gently inclined structure (Figure 10.3). Mineralization is strongest within the structure but also forms a lower-grade halo of mineralization in the rocks above the structure. For more details of the mineralization and host rocks see section 7.3.1.

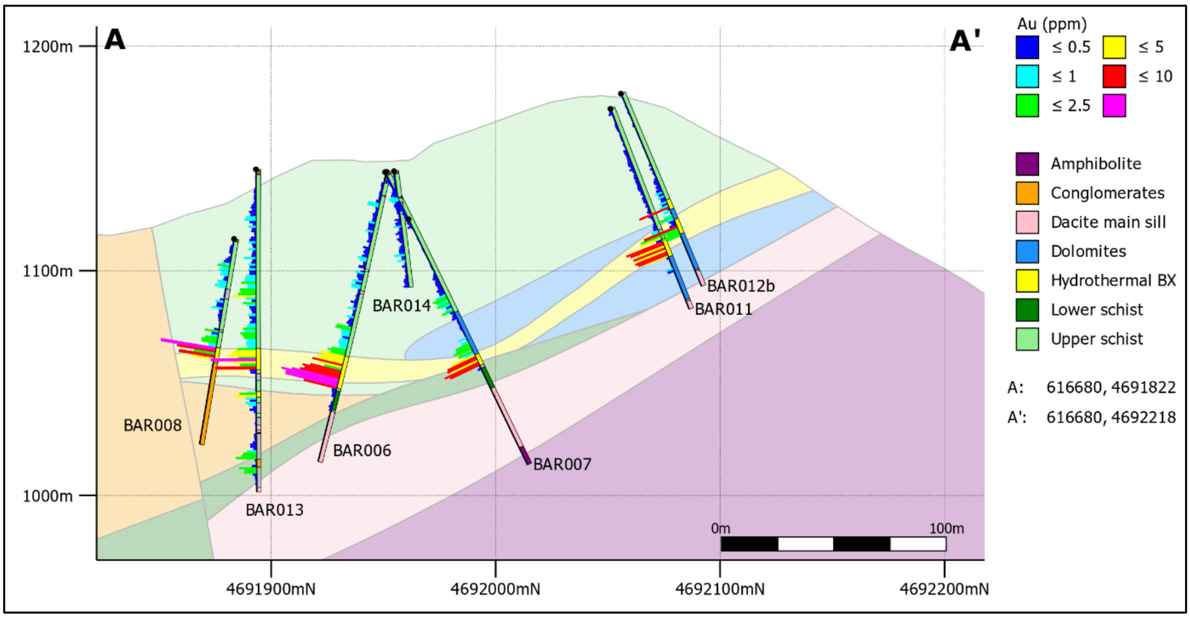


Figure 10.3. Representative Cross-section at Barje.
North-south section at 616688mE looking west. Drill holes projected from up to 50 metres off-section. See Figure 10.4 for cross section location.

The true thickness of mineralization reported in Table 10.3 for holes at Barje is calculated using the surveyed inclination of the drill hole and assumes that the mineralization is flat-lying, thus providing an estimate of the vertical thickness of the interval. At this stage of exploration, this is considered a reasonable approximation for the thickness of the interval. Using these calculated thicknesses a "grade times thickness" quantum was calculated for gold in the reported intersections, summed per drill hole, and contoured to outline the area of significant mineralization (Figure 10.4); this shows the reported intervals to form an area between and to the west of the discovery outcrops of approximately 700 m east-west by 250 m north-south (Medgold Resources Corp, 2019a).

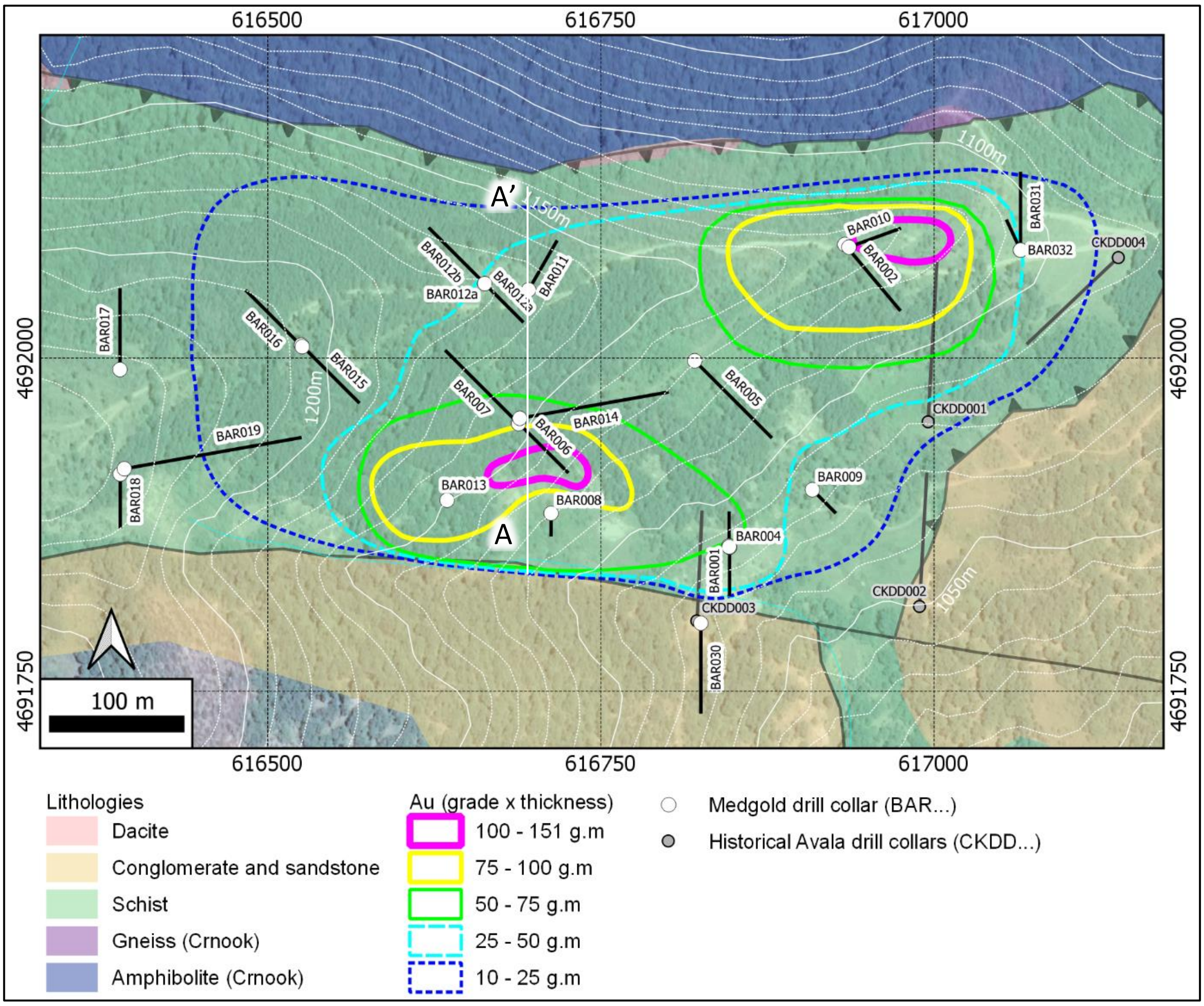


Figure 10.4. Drill hole locations within the main mineralization at Barje.
Thickness is vertical thickness of mineralized zone, internal waste included in calculation.

The area of mineralization at Barje is limited by the outcropping barren amphibolite to the north and has been truncated by the east-west trending Barje fault to the south. To the east the mineralization comes to surface and has been lost due to erosion. The structure is believed to continue to the west of the mineralized area, but only low-grade gold mineralization has been seen in the drilling to date, e.g., BAR024 and BAR025.

10.3.2 Liska

At the Liska Prospect, drilling intersected a flat-lying zone of lead-zinc mineralization hosted within the upper part of a schist unit (Figure 10.5). The schist and the mineralization are bounded by a flat-lying fault separating the schist from an overlying conglomerate. For more details of the mineralization and host rocks see Section 7.3.2.

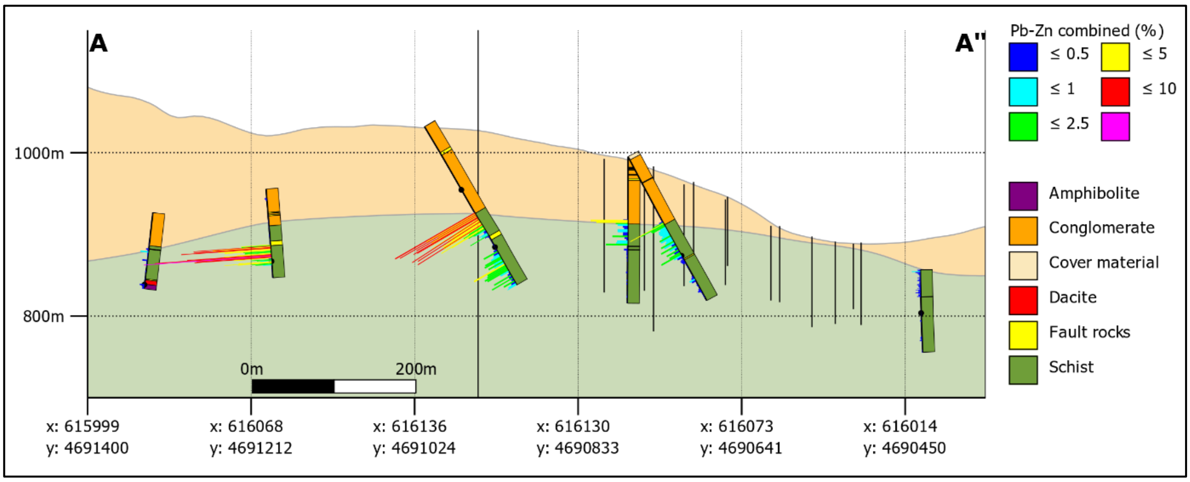


Figure 10.5. Representative Cross-section at Liska.
Fence section from A to A" looking east. Drill holes projected from up to 50 metres off-section. Historic drill holes shown as black traces. See Figure 10.6 for section location.

The true thickness of mineralization reported in Table 10.3 for holes at Liska is calculated using the surveyed inclination of the drill hole and assumes that the mineralization is flat-lying, thus providing an estimate of the vertical thickness of the interval. At this stage of exploration, it is thought that this is a reasonable approximation for the thickness of the mineralization. The inferred limits of mineralization at Liska using data from Medgold's drilling and historical (non-verified) data from State exploration are shown in Figure 10.6.

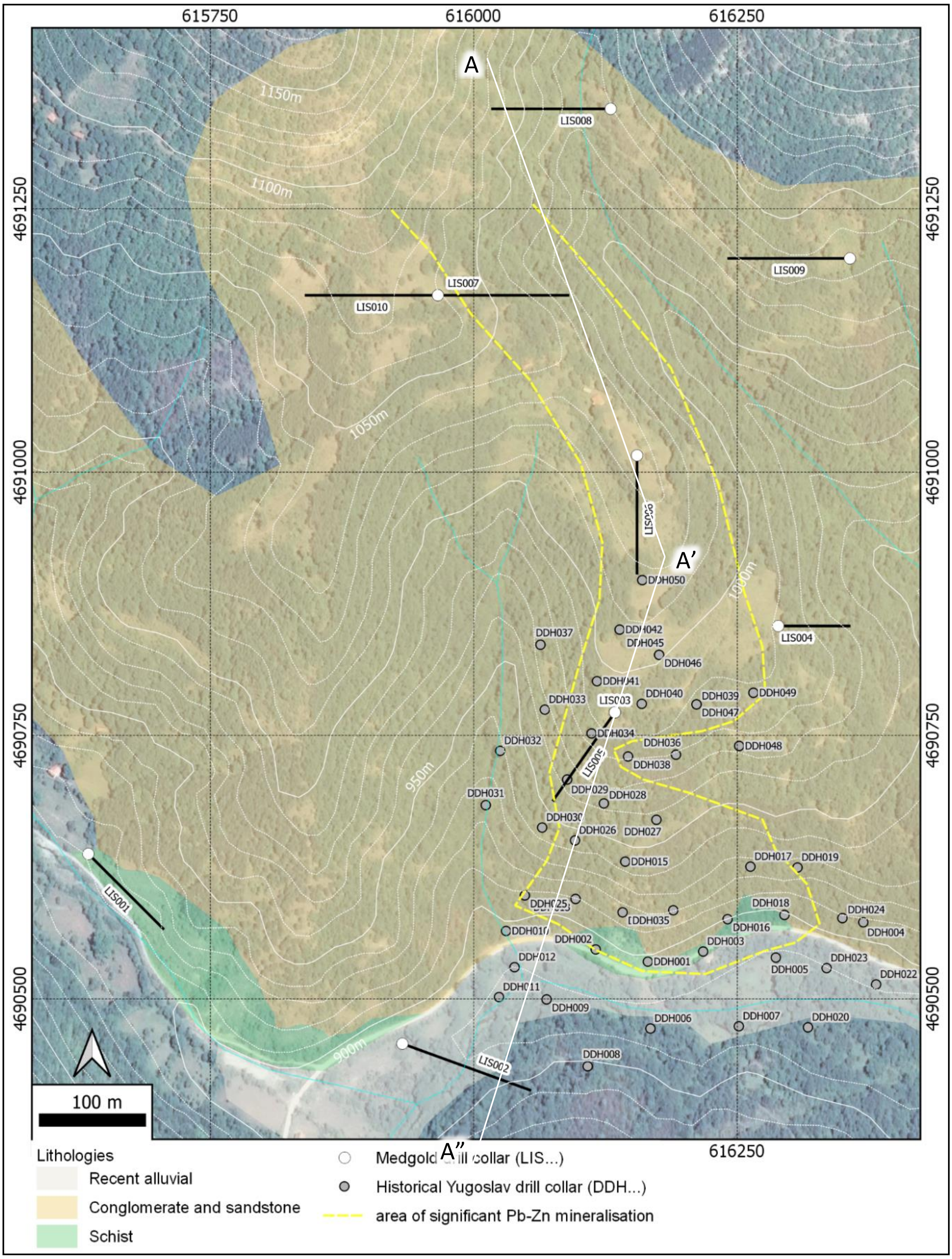


Figure 10.6. Geology, drill holes and outline the area of mineralization at Liska.
See Figure 10.5 for cross-section A-A'-A"

The area of mineralization at the Liska Prospect appears to be confined to a north-south trending corridor, though the mineralization may continue to the north-west from hole LIS007 under an increasing thickness of conglomerates. Historical (non-verified) drilling in the valley south of the mineralization suggests that mineralization does not continue in that direction or has been off-set by an inferred east-west fault in the main river valley.

10.3.3 Karamanica

Drilling at the Karamanica Prospect targeted combinations of structural, geochemical and geophysical targets. Only weak mineralization was intersected associated variously with fault zones, dark carbonaceous schists, and the margins of porphyritic intrusions.

The true thickness of mineralization reported in Table 10.3 for holes at the Karamanica Prospect is calculated using the surveyed inclination of the drill hole and assumes that the mineralization is hosted by fault structures inclined at 70° to the horizontal, thus providing an estimate of the true thickness of the interval.

Several geochemical and geophysical targets remain un-drilled on the Prospect.

11 Sample Preparation, Analyses and Security

11.1 Medgold Soil Samples

Field procedures as deployed by Medgold for sample collection are summarized as follows:

- Sample sites were pre-planned in the office and then located in the field using handheld GPS with an accuracy of generally better than ±7 m.
- Samples weighing 1 to 1.5 kg were collected from B horizon material after removal of the A horizon.
- Sample information including the data, the sampler, the location as recorded by GPS in the field, and site characteristics were recorded in a sample book with predefined sample numbers and tear-off sample number tags.
- Samples were placed in a sample bag with the sample number tag. The bag was sealed, and the sample number was marked on the bag with a permanent marker.
- All sampling was carried out by geologists or field technicians under a geologists' supervision.

Soil samples were grouped into dispatches and driven directly to the labs of ALS Laboratory Services D.O.O. in Bor, Serbia, by Medgold or by a contracted private courier.

Samples were prepared by dry-screened to 180 µm after which a 25 g aliquot of the <180 µm fraction was sent by ALS to their laboratory in Loughrea, Republic of Ireland. In the Loughrea laboratory, the aliquot was analysed by a gold and multielement ICP-MS analysis package (ALS code AuME-TL43) after an aqua regia digestion.

Once sample results were received by Medgold, QAQC data generated by ALS was reviewed before analysis data were merged with sample data and stored in a postgreSQL database system.

11.2 Medgold Rock Samples

Field procedures for sample collection as deployed by Medgold are summarized as follows:

- Sample sites were selected in the field with locations recorded using handheld GPS with an accuracy of generally better than ±7 m. For channel samples at Barje, coordinates were reported from a post-sampling DGPS survey.
- Samples collected during prospecting and mapping were taken from outcrop, subcrop or float to sample individual features of lithology, alteration, or mineralization. Samples taken along chip or channel lines were collected from outcrop to represent the material within the sample interval.
- Mass of the collected sample was generally between 2 and 7 kg.
- Sample information including the data, the sampler, the location (either as recorded by GPS in the field or by reference to the tape-and-compass measurements), and geological characteristics were recorded in a sample book with predefined sample numbers and tear-off sample number tags.
- Samples were placed in a sample bag with the sample number tag. The bag was sealed, and the sample number was marked on the bag with a permanent marker.
- All sampling was carried out by geologists or field technicians under a geologists' supervision.

Rock samples were grouped into dispatches and driven directly to the labs of ALS Laboratory Services D.O.O. in Bor, Serbia, by Medgold or by a contracted private courier.

Samples were prepared by crushing to 70% <2 mm in size before a 250 g split was taken for pulverising to 85% passing 75 µm. A split of the fine fraction material was sent by ALS to their laboratory in Loughrea, Republic of Ireland. In the Loughrea laboratory, the material was split into a 30 g aliquot for analysis by fire assay with an ICP-AES finish (ALS code Au-ICP21) using a gravimetric finish for samples of over 10 g/t gold. Samples were also analysed for a multi-element ICP-AES/MS analysis package; during 2019 this was done using ALS code MS-ME41 following an aqua regia digestion; prior to 2019 the package used was ALS code MS-ME61 following a 4-acid digestion.

Once sample results were received by Medgold, QAQC data generated by ALS was reviewed before analysis data were merged with sample data and stored in a postgreSQL database system.

11.3 Medgold Drill Samples

11.3.1 Sample Preparation Prior to Dispatch.

All work on Medgold's drill core prior to sample dispatch was carried out by Medgold's geologists and technicians at the Company's core yard in Bosilegrad. After sample definition (see Section 10.2), drill core to be sampled was cut using a conventional core saw with a water-cooled rotating blade. One-half of the core was taken for sample and one half returned to the core box. Sample number tickets were placed both in the core box and the sample bag for each sample. Low-density polyethylene samples bags of 350 µm thickness were used to avoid the loss of fine material from the samples; sample bags were sealed with a cable tie.

Samples were packed into polyweave sacks sealed with a cable tie and tape. Sample dispatches were driven directly to the labs of ALS Laboratory Services D.O.O. in Bor, Serbia, by Medgold or by a contracted private courier.

11.3.2 Bulk Density Measurements.

Measurements were made using a traditional water-displacement method with the sample mass recorded while in air and while suspended in water. As sample porosity was relatively minor, the samples were not wax coated prior to measurement. The bulk density of a reference specimen was measured at the start of each set of core measurements as a control sample.

During drilling in 2018 and 2019, a bulk density measurement was made at a nominal 20 m spacing along every drill core, producing 590 bulk density values. A Terraplus KT-20S/C magnetic susceptibility meter with digital density scale attachment was used to record and calculate bulk density values using a built-in bulk density routine.

An additional 133 bulk density measurements were made using archived core from within the main zone of mineralization at Barje during November 2019. A Radweg PS2100.R2 precision balance with under-pan weighing was used to record and calculate bulk density values using a built-in bulk density routine.

11.3.3 Laboratory Sample Preparation and Analysis.

Preparation and analysis of drill samples was carried out under contract by ALS Laboratory Services D.O.O. (ALS), a member of the global ALS Group. ALS is independent to Medgold and, apart from a commercial contract for sample preparation and analysis, the two companies have no other relationship.

Sample preparation was completed at the ALS sample preparation facility in Bor, Serbia. Samples were prepared by crushing to 70% passing 2 mm and a rotary splitting device was used to separate a 1 kg sub-sample and a coarse reject. The sub-sample was pulverized to 85% passing 75 µm. A split of the -75 µm pulp was sent by ALS to their laboratory in Loughrea, Republic of Ireland, for analysis.

The ALS Loughrea laboratory performed the following analysis on each received pulp:

- Gold by 30 g fire assay with an ICP-AES finish (ALS code Au-ICP21)
- Multi-element ICP-AES/MS analysis package (ALS code MS-ME61) following a 4-acid digestion
- Over-grade gold (>10 ppm) by 30 g fire assay with a gravimetric finish (ALS code Au-GRA21)
- Over-grade silver (>100 ppm), copper (>1%), lead (>1%), zinc (>1%), arsenic (>1%), bismuth (>1%), molybdenum (>1%) or sulphur (>10%) by ICP-AES/MS over-grade package (ALS code OG62)
- Over-grade silver (>1500 ppm) by 30 g fire assay with a gravimetric finish (ALS code Ag-GRA21)

Sample coarse rejects and pulp material not forwarded to the ALS Loughrea laboratory were either stored at ALS' laboratory in Bor or returned to Medgold.

11.3.4 QC procedures

Medgold's QAQC procedures include controls on sampling, the insertion of control material into sample sequences, predefined client templates for sample preparation and analysis, monitoring of control material against industry standard control gates, and re-assaying of selected samples by an umpire laboratory.

Control material was inserted into sample sequences before dispatch from the core yard. Certified reference material (CRM), a coarse-crushed granite blank and ¼ -core duplicates were inserted at a target ration of 2 CRMs, 1 blank and 1 duplicate per 32 primary core samples. The laboratory may become aware of samples that contain control material, but the values of the material will be unknown to the laboratory. The CRMs used during the 2018 and 2019 drilling programmes were supplied by CDN Resource Laboratories Ltd of British Columbia; a range of CRMs were used and inserted into sample sequences with an attempt to match the expected grades of the samples based on mineral logging and, in later holes, comparison with assay results from previous core. The following CRMs were used during the programme:

Table 11.1. Certified reference material used during 2018 and 2019 drilling.
Grades indicated are certified mean values for the primary analysis method.

CRM	Au ppm	Ag ppm	Pb%	Zn%	In use
CDN-ME-1709	0.178	11.8	0.053	0.194	2018
CDN-ME-1410	0.542	69	0.248	3.682	2018
CDN-ME-1406	0.678	57.1	0.4	2.2	2019
CDN-ME-1606	1.069	116	1.7	0.6	2019
CDN-ME-1505	1.29	360	1.87	0.72	2018
CDN-ME-1308	1.4	45.7	0.541	0.429	2018
CDN-ME-1607	3.33	150	1.72	0.56	2018
CDN-ME-1807	7.88	327	2.3	2.3	2019
CDN-ME-1402	13.9	131	2.48	15.23	2018, 2019

Sample assay data during 2018 was imported from laboratory certificates into a Geospark database developed by Geospark Consulting Inc of British Columbia. Sample assay data during 2019 was imported from laboratory certificates into a MX Discover cloud-hosted database maintained by Geosoft Inc of Ontario. Assays for control CRMs and blanks were reviewed using inbuilt reports within each database and data were released for use only if it had passed QAQC criteria.

11.4 AMS Comments

The Qualified Person is satisfied that the Sample Preparation, Analysis, and the Security of samples is fit for purpose. A robust documentation process along with suitable quality control procedures is in place.

12 Data Verification

The following section describes steps taken by the Qualified Person for Geology and Resources to verify the data presented in this report and which was used for the estimation of Mineral Resources reported herein.

12.1 Site Visit by AMS

A site visit was conducted to the Tlamino Licences between November 11 and November 14, 2019, by Mr Richard Siddle (Senior Consultant Geologist, AMS) who is the Qualified Person (Geology and Resources) for the study. The purpose of the visit was to inspect the Property, core processing procedures and to confirm the presence of mineralization. Verification activities are described in the following sections. Due to restrictions relating to the COVID-19 pandemic in force during the study, no other QP was able to visit the site in support of the PEA.

12.2 Drilling and Drill Core Inspection

At the time of the site visit, no drilling activities were underway and, as such, it was not possible to inspect drilling activities. The Qualified Person discussed core handling and logging procedure with the geological team and inspected drill core with comparison to geological logs made by Medgold geologists. Core handling procedures were found to be good, with daily inspections of the drill rig by geologists, photography, and quick logging of the core at the drill rig before transport to the core processing facility. Core logging was found to be accurate and capture an appropriate level of detail. Zones of mineralization were clearly visible within the drill core.

12.3 Collar Location Verification

The Qualified Person collected non-differential GPS locations of drill collars from Medgold drilling. The results are presented in Table 12.1. Medgold complete collar location survey using differential GPS which is much more accurate than the handheld device used by the Qualified Person. The offset distances between the AMS and Medgold collars are considered to be within the accuracy of the non-differential GPS and the collar locations are considered to be suitably accurate.

Table 12.1: Qualified Person's collar location check coordinates.

	AMS			Medgold			
Collar	**EAST**	**NORTH**		**EAST**	**NORTH**		**Offset / m**
BAR003	616849.12	4691861.34		616846.23	4691859.03		3.70
BAR005	616817.68	4692003.43		616820.41	4691997.93		6.14
BAR009	616906.59	4691901.84		616908.74	4691901.14		2.26
BAR010	616933.88	4692087.78		616936.19	4692083.22		5.11
BAR013	616634.49	4691896.08		616634.53	4691893.23		2.85
BAR014	616685.61	4691961.57		616689.08	4691954.73		7.67
BAR015	616520.41	4692015.35		616525.16	4692009.74		7.35
BAR019	616393.53	4691914.17		616392.28	4691916.87		2.97
BAR031	617060.55	4692082.68		617061.39	4692083.02		0.91

12.4 Laboratory Inspection

The Qualified Person has not inspected the ALS laboratory used in routine assay nor the ALS laboratory used in metallurgical test work.

12.5 Database Verification

Logging data during 2018 was captured using paper forms before entry into Excel sheets and import into a Geospark database developed by Geospark Consulting Inc of British Columbia. Logging data during 2019 was captured directly into WiFi enabled tablets linked to an MX Discover cloud-hosted database maintained by Geosoft Inc of Ontario.

The Qualified Person discussed the database management system with Medgold's geologists during the site visit. At the time, the two database sets had not been combined into a single version, and this was done without delay at the request of the Qualified Person ahead of the Mineral Resource estimation. The Qualified Person verified logging by comparison of strip logs to drill core, compared a selection of original laboratory assay reports to the database and completed drill hole validation in Micromine (see section 14.3). The exploration database was found to be robust and fit for purpose.

12.6 Quality Control

Medgold has in inserted routine quality control samples into their sample stream, blank material tests for contamination in the laboratory which is most commonly introduced at the sample preparation stage. Certified Reference Materials (CRMs) are highly homogenous powdered materials with an estimated concentration of certain elements to within a reported within standard deviation and test for the accuracy of the analysis. Field duplicates are duplicate samples of remaining drill core, taken from the same interval as the original and test the precision (reproducibility) of the sampling method. Umpire samples were also submitted to a different laboratory to test for analytical bias.

The number of quality control samples is shown in Table 12.2 and are discussed in the following section.

Table 12.2: Summary of Quality Control Samples.

QC Type	Number	% of Exploration database
Blank	155	3
CRM	307	6
Field Duplicate	153	3
Umpire	24	0.5

12.6.1 Field Duplicates

Comparison of field duplicate pairs was found to show excellent precision for a gold deposit (Figure 12.1). However, it should be noted that the proportion of samples taken from potentially economic grade mineralization is relatively low (around 11%) and future drill programmes should target field duplicates towards mineralized core.

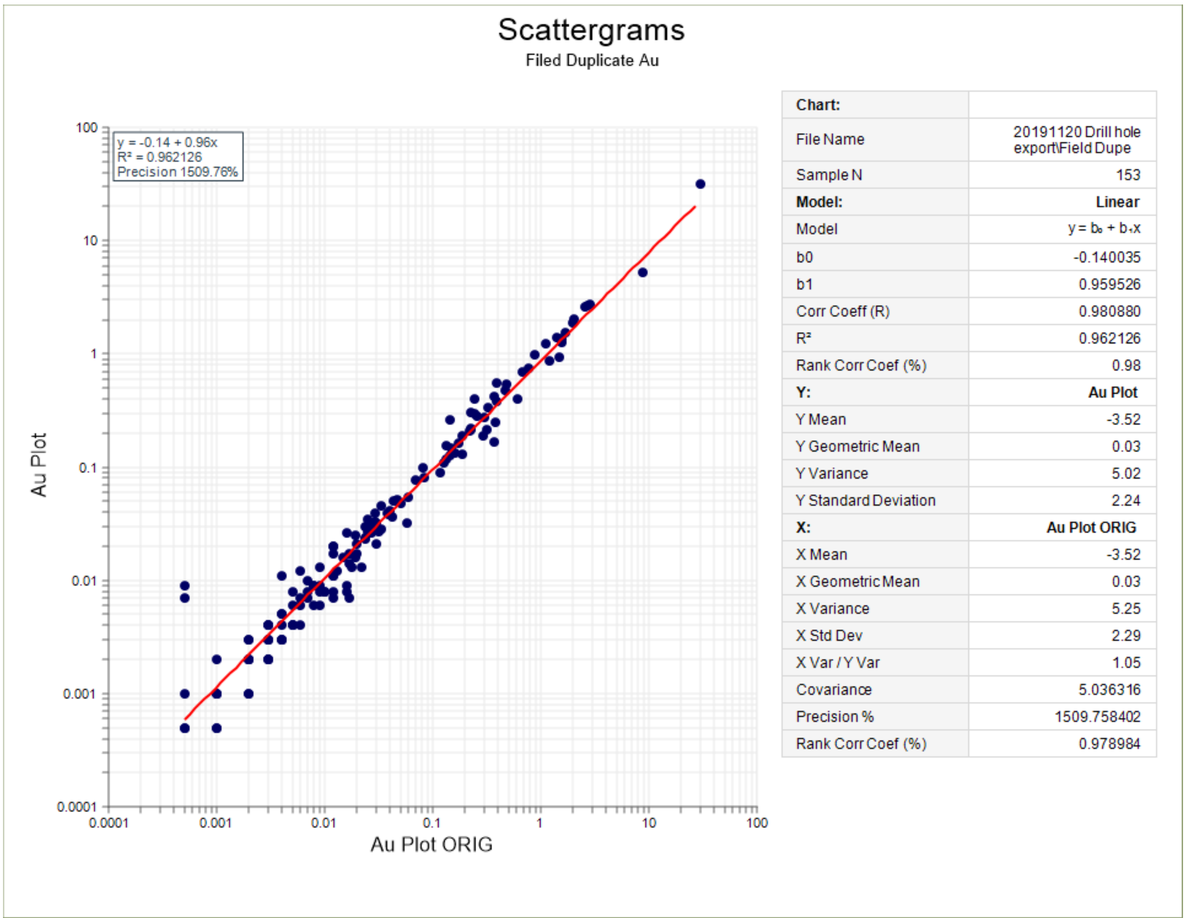


Figure 12.1: Field Duplicates Au.

12.6.2 Certified Reference Materials

CRMs used by Medgold are shown in Table 12.3 along with summary analysis of Au. Medgold conducted routine analysis of CRMs on receipt of results from the laboratory. Failures (values outside the action limit of 3 Standard Deviations (SD) or sequential values outside the warning limit of 2 SD) from areas of mineralized drill core were addressed with the laboratory and re-analysis requested; the CRMs were also re-analysed until acceptable results were achieved. Some CRM failures remaining in the database are from areas of un-mineralized drill core, the absence of mineralization was verified by visual inspection of the drill core.

Analysis of Au and Ag CRM results as well as for Pb and Zn was completed by Medgold. The Qualified Person has inspected this analysis as well as completing his own analysis for Au and Ag using

Micromine software. Table 12.3 shows the results of the analysis for Au. Example control charts are shown in Figure 12.2 and Figure 12.3. The data are considered to be suitably accurate and tested over a suitable range of grades.

Table 12.3: Summary of Certified Reference Materials Au analysis.

CRM	**CDN-ME-1308**	**CDN-ME-1807**	**CDN-ME-1709**	**CDN-ME-1607**	**CDN-ME-1606**	**CDN-ME-1505**	**CDN-ME-1410**	**CDN-ME-1406**	**CDN-ME-1402**
Field Name	Au Plot	Au Plot	Au Plot	Au Plot	Au Plot	Au Plot	Au Plot	Au Plot	Au Plot
Ref Mean	1.400	7.880	0.180	3.330	1.070	1.290	0.540	0.680	13.900
Ref Std Dev	0.050	0.210	0.010	0.140	0.050	0.060	0.020	0.030	0.400
Count	46.000	8.000	48.000	25.000	28.000	42.000	48.000	32.000	30.000
Median	1.410	8.105	0.178	3.370	1.060	1.288	0.554	0.679	13.925
Mode	1.378	8.232	0.173	3.342	1.059	1.236	0.564	0.675	13.850
Mean	1.410	7.996	0.178	3.392	1.068	1.286	0.557	0.684	13.950
Std Dev	0.067	0.300	0.008	0.150	0.041	0.044	0.032	0.028	0.374
Minimum	1.285	7.580	0.158	3.120	1.005	1.220	0.498	0.632	12.700
Maximum	1.555	8.330	0.197	3.730	1.165	1.375	0.643	0.742	14.550
Expected to +1SD	13 (28.26%)	1 (12.50%)	16 (33.33%)	11 (44.00%)	7 (25.00%)	14 (33.33%)	12 (25.00%)	10 (31.25%)	9 (30.00%)
Expected	1 (2.17%)	0 (0.00%)	5 (10.42%)	0 (0.00%)	0 (0.00%)	2 (4.76%)	3 (6.25%)	1 (3.13%)	2 (6.67%)
Expected to -1SD	12 (26.09%)	1 (12.50%)	15 (31.25%)	5 (20.00%)	13 (46.43%)	13 (30.95%)	13 (27.08%)	12 (37.50%)	10 (33.33%)
Inside 1SD	26 (56.52%)	2 (25.00%)	36 (75.00%)	16 (64.00%)	20 (71.43%)	29 (69.05%)	28 (58.33%)	23 (71.88%)	21 (70.00%)
+1SD to +2SD	5 (10.87%)	3 (37.50%)	3 (6.25%)	4 (16.00%)	3 (10.71%)	5 (11.90%)	9 (18.75%)	3 (9.38%)	6 (20.00%)
-1SD to -2SD	6 (13.04%)	2 (25.00%)	5 (10.42%)	2 (8.00%)	4 (14.29%)	8 (19.05%)	4 (8.33%)	3 (9.38%)	2 (6.67%)
1SD to 2SD	11 (23.91%)	5 (62.50%)	8 (16.67%)	6 (24.00%)	7 (25.00%)	13 (30.95%)	13 (27.08%)	6 (18.75%)	8 (26.67%)
+2SD to +3SD	5 (10.87%)	1 (12.50%)	2 (4.17%)	3 (12.00%)	1 (3.57%)	0 (0.00%)	4 (8.33%)	3 (9.38%)	0 (0.00%)
-2SD to -3SD	3 (6.52%)	0 (0.00%)	2 (4.17%)	0 (0.00%)	0 (0.00%)	0 (0.00%)	0 (0.00%)	0 (0.00%)	0 (0.00%)
2SD to 3SD	8 (17.39%)	1 (12.50%)	4 (8.33%)	3 (12.00%)	1 (3.57%)	0 (0.00%)	4 (8.33%)	3 (9.38%)	0 (0.00%)
+3SD	1 (2.17%)	0 (0.00%)	0 (0.00%)	0 (0.00%)	0 (0.00%)	0 (0.00%)	3 (6.25%)	0 (0.00%)	0 (0.00%)
-3SD	0 (0.00%)	0 (0.00%)	0 (0.00%)	0 (0.00%)	0 (0.00%)	0 (0.00%)	0 (0.00%)	0 (0.00%)	1 (3.33%)
Outside 3SD	1 (2.17%)	0 (0.00%)	0 (0.00%)	0 (0.00%)	0 (0.00%)	0 (0.00%)	3 (6.25%)	0 (0.00%)	1 (3.33%)

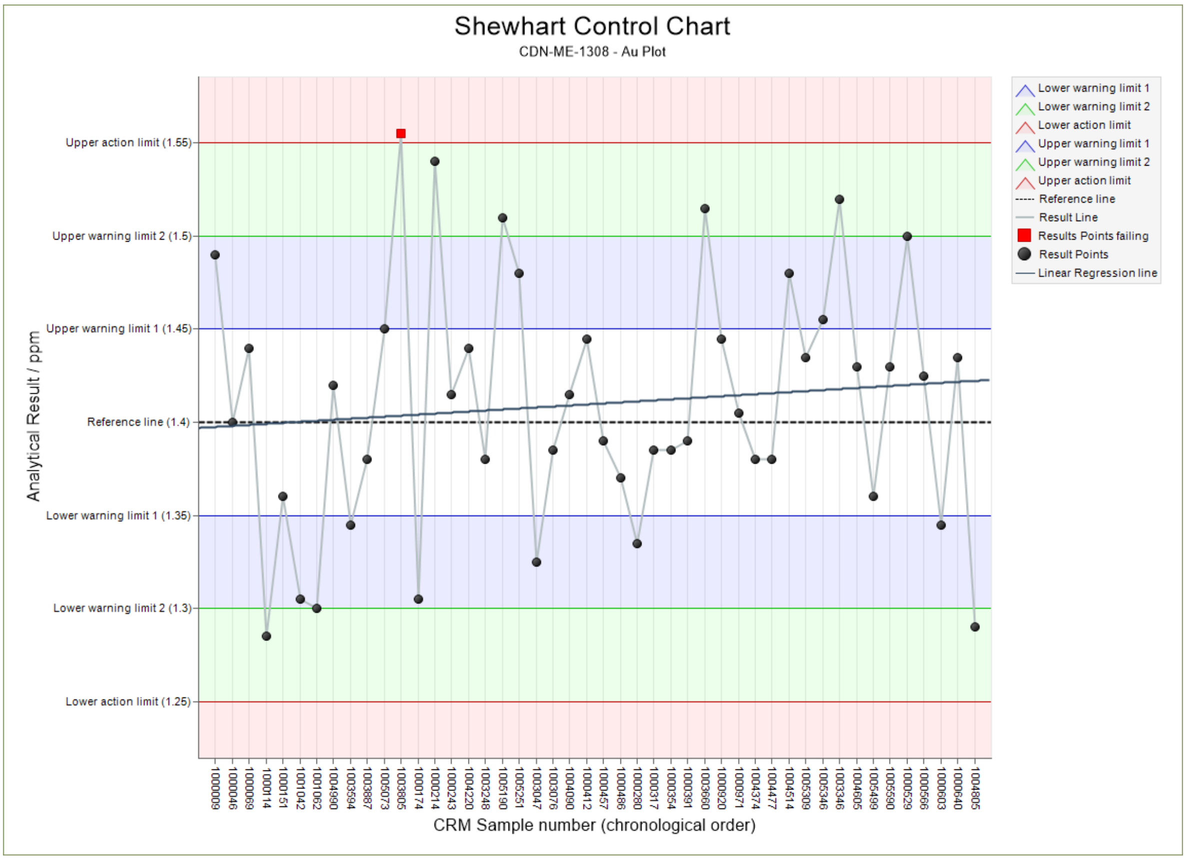


Figure 12.2: CRM CDN-ME-1308 Au Analysis with 3 standard deviation action limits.

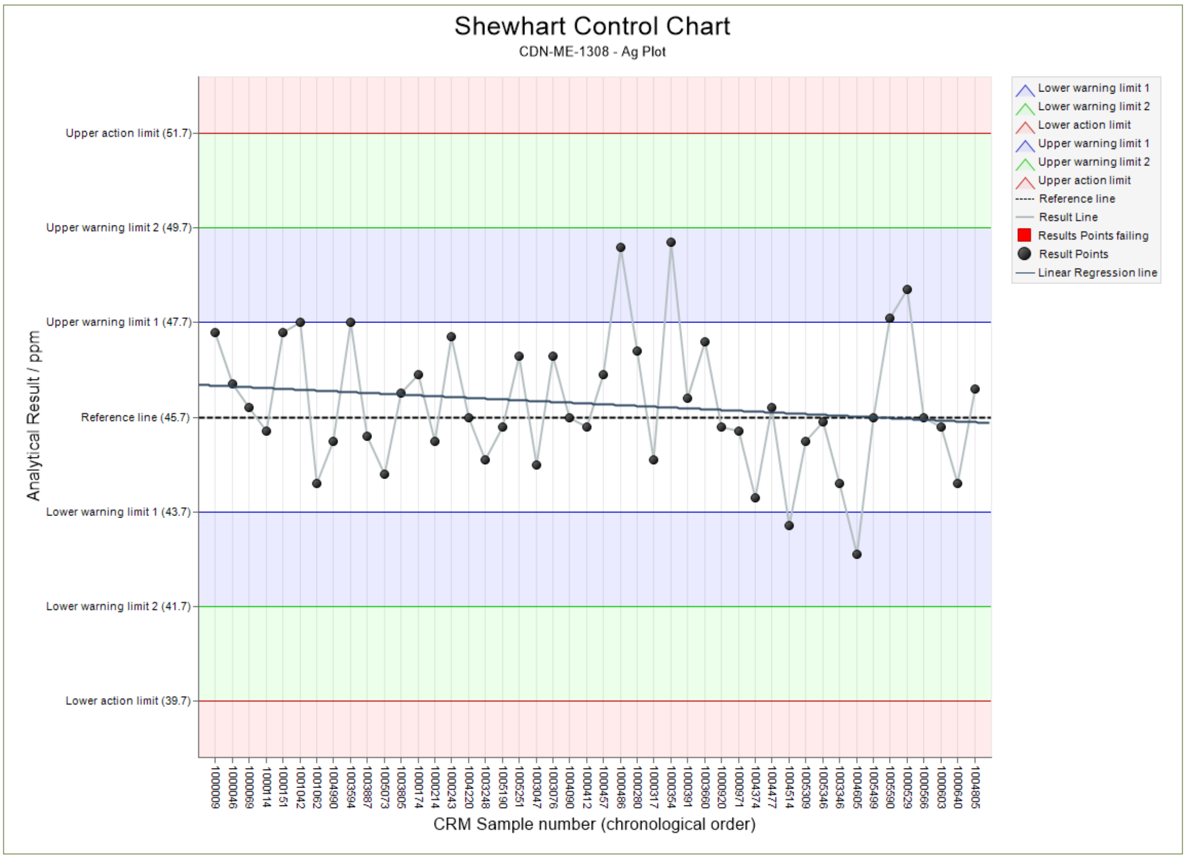


Figure 12.3: CRM CDN-ME-1308 Ag Analysis with 3 standard deviation action limits.

12.6.3 Blanks

Analysis of blank reference materials showed no signs of contamination.

12.6.4 Verification (Umpire) Samples

The Qualified Person did not collect or submit any samples for independent assay verification. Medgold has submitted 24 pulp and 10 CRMs for umpire lab verification to SGS Bulgaria Ltd., Sofia, Bulgaria, an affiliated company of SGS S.A., Switzerland. The Qualified Person has inspected the original laboratory certificates for the original and umpire assays as a means of verification. The reproducibility between the two laboratories was found to be excellent with no evidence of bias (Figure 12.4 and Figure 12.5). It should be noted that initially some of the CRMs submitted to the umpire lab (SGS) failed and a number of re-assays were completed before finalization. No umpire analysis has been completed for the 2019 drill programme as the results of the 2018 analysis were found to be highly satisfactory.

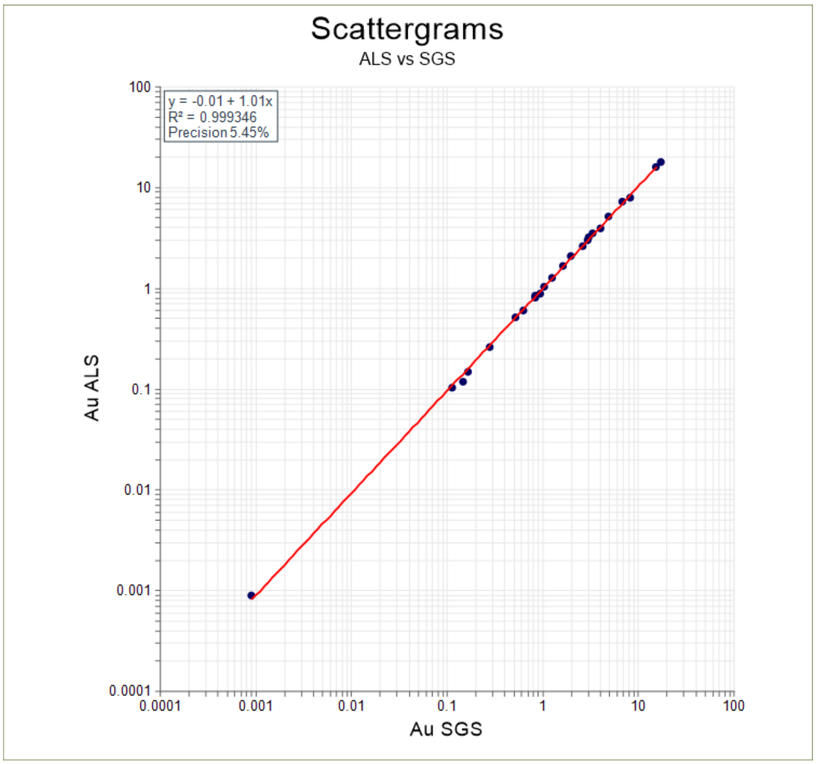


Figure 12.4: ALS vs SGS Au umpire analysis.

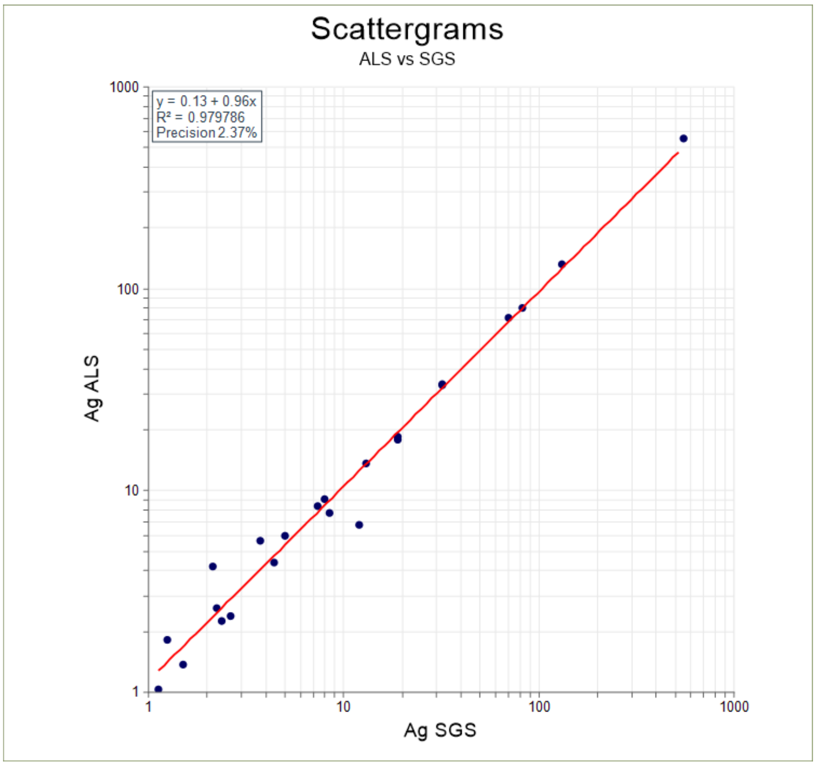


Figure 12.5: ALS vs SGS Ag umpire analysis.

12.7 Sample Recovery and Grade

The Qualified Person inspected drill core recovery data versus sample assay grades to determine if a relationship between sample recovery and grade exists. As shown in Figure 12.6 no relationship was identified.

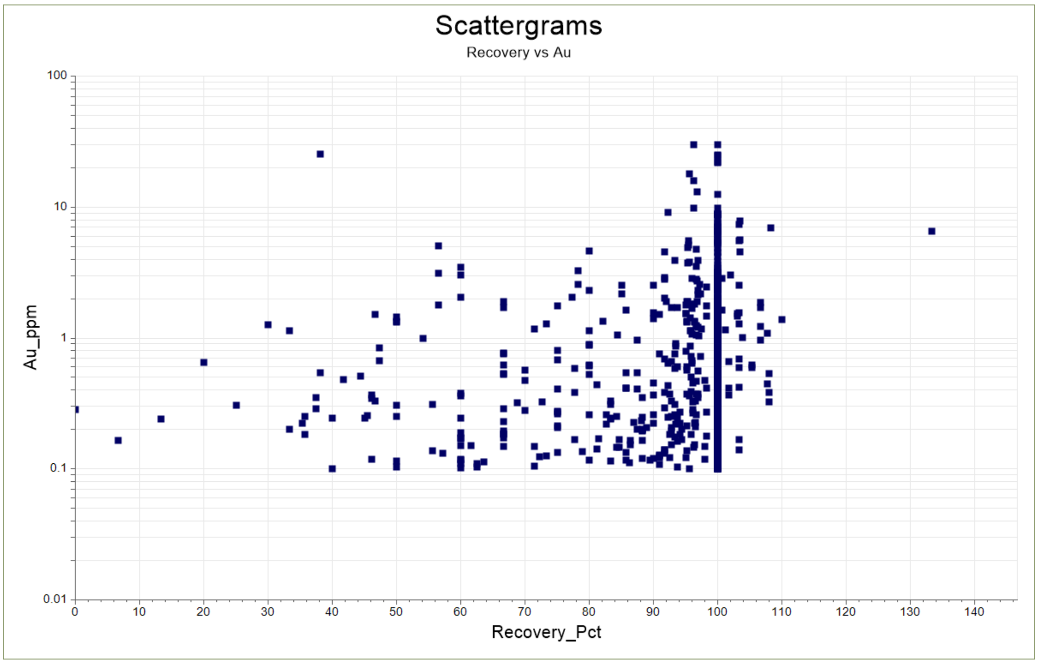


Figure 12.6: Drill hole sample recovery vs Au grade.

12.8 AMS Comments

No issues have been identified by the Qualified Person for Geology and Resources with regards to the exploration database, location control and data collection methods, the assay data are considered within acceptable limits of accuracy and precision for use in Mineral Resource estimation.

13 Mineral Processing and Metallurgical Testing

Two programmes of metallurgical sampling and testing have been completed on material from the Barje Deposit. An initial metallurgical test work programme with the objective of determining precious and base metal recoveries as well as arsenic deportment was completed by Woods Process Services LLC (WPS), of Denver, USA on drill core material in 2019. A second programme comprising mineralogical examination and bench-scale gravity concentration, flotation and leaching tests was completed at ALS (Kamloops, Canada) during 2020.

13.1 Initial Programme

The results of the initial programme of tests were summarized in a Technical Memorandum compiled by Jeff Woods (WPS) dated November 1, 2019 (Woods, 2019) and reported previously in the earlier NI 43-101 report on the Barje Deposit by Addison Mining Services Ltd (Siddle, 2020).

13.1.1 Sample Selection and Preparation

Metallurgical sampling was undertaken on composites generated from selected coarse drill core reject samples from the HBX and Triple X ("XXX") breccia zones totalling 50.39 kg. All samples were taken from unoxidized/unweathered breccia material containing - on a visual basis - pyrite, arsenopyrite, sphalerite, galena and more rarely chalcopyrite and tennantite. Grains of electrum up to approximately 50 µm in size, containing approximately 60% gold and 40% silver, were observed microscopically within higher-grade zones of mineralization. Eleven HBX subsamples across a range of grades were used to make up the HBX composite, and five Triple X subsamples were used to make up the higher-grade XXX composite.

The resulting combined Au and Ag grades of the composites submitted for testing, based on back calculation of the metallurgical results, are shown in Table 13.1.

Table 13.1: Calculated Au and Ag grades of the composites submitted for testing.

Sample	Calculated head grades	
	Au, g/t	Ag, g/t
HBX	2.05	16.90
XXX	9.71	108.40

No material from the lower grade domain or partially oxidized material was tested, and the resulting grades for Au and Ag were slightly higher than those estimated in the Mineral Resource estimate described. They were, however, considered to be suitable for an initial indication of metallurgical response.

Initial metallurgical testing was used to determine baseline metallurgical performance, including recoveries and reagent consumptions for a range of likely extraction routes. Baseline tests include preliminary cyanidation, bulk sulphide flotation, sequential flotation and diagnostic leaching. No comminution test work was undertaken.

13.1.2 Baseline cyanidation test work

Baseline cyanidation work targeting gold and silver values only was undertaken using standard bottle roll tests on sub-samples of the HBX and XXX composites under the following conditions:

- Feed particle size F_{80}: 74 µm
- Slurry solids content: 40% by weight
- Slurry pH: 10.5 to 11.0 with hydrated lime
- NaCN concentration: 1 g/L maintained throughout the test
- Test duration: 48 hrs
- Sampling at 6, 24 and 48 hours

Neither of the materials responded well to cyanidation. Diagnostic leaching of the HBX and XXX composites indicated little of the gold to be cyanide soluble, with a high proportion contained in either arsenopyrite or pyrite. Additionally, the bottle roll tests suggested that a high proportion of the silver is potentially occluded in galena, inhibiting cyanide recovery of Ag as well. Reagent consumption was moderate to high indicating poor process economics for this route.

Cyanide leaching was not recommended as a treatment route for the Barje breccias.

13.1.3 Sequential gravity/flotation testing

Sequential gravity and flotation testing were also performed and targeted production of separate Pb and Zn concentrates as well as a treatable Au Ag concentrate by gravity.

For the HBX composite, gravity testing resulted in recoveries of 16.5% Au and 4.6% Ag at 48.99 g/t Au and 19.3 g/t Ag into a 0.6% concentrate by mass. For the XXX sample, recoveries of 10.0% Au and 4.4% Ag at 66.59 g/t Au and 30.20 g/t Ag into a 1.5% concentrate by mass was achieved. While neither result is particularly high, the presence of gravity recoverable gold and silver in the form of native gold or electrum was indicated and gravity concentration was recommended for inclusion in future testing for flowsheet development.

Results of the sequential float tests suggest Pb and Zn recoveries to separate concentrates are poor and the concentrates produced are unlikely to be marketable.

13.1.4 Bulk flotation testing

Baseline bulk sulphide flotation tests were considered as a means of producing a bulk polymetallic concentrate which could then be sold for toll treatment. Tests were conducted on sample splits from each of the composites. Test conditions used were as follows:

- Feed particle size F_{80} 74 µm
- Slurry solids content: 40% by weight
- Slurry pH: natural 8.2
- Collectors:
  - Potassium amyl xanthate (PAX): 100 g/t
  - Cytec AP404: 100 g/t
  - Cytec 3418A: 100 g/t
- Frother: MIBC
- Conditioning time: 3 minutes
- Flotation time: 9 minutes

Summary results from the test work are shown in Table 13.2.

Table 13.2: Baseline Bulk Sulphide Rougher Flotation Test: Recoveries and Concentrate Grades.

	HBX	XXX	Combined
Concentrate			
Au g/t	16.83	36.48	26.66
Ag g/t	124	379	251.5
Cu%	0.08	0.15	0.12
Pb%	0.57	2.69	1.63
Zn%	1.2	1.2	1.2
As%	5.85	12.1	8.98
Recovery to concentrate*			
Au%	88.2	90.5	89.35
Ag%	88.2	96.4	92.3

Gold recoveries to concentrate for the baseline rougher test were 88.2% and 90.5% for the HBX and XXX composites, respectively, with rougher concentrate grades of 16.8 g/t and 36.5 g/t respectively.

Silver recovery for the HBX composite was 88.2% with a concentrate grade of 124 g/t. The XXX composite test resulted in 96.4% silver recovery with a concentrate Ag grade of 379 g/t. The recoveries and rougher concentrate grades were higher than were expected from unoptimized test conditions.

Copper, lead, and zinc recoveries for the HBX composite were 93.2%, 96.5% and 74.2% respectively, with 95.3%, 91.6% and 91.4% respectively for the XXX composite. However, concentrate grades were too low to be of economic interest for the project. Arsenic grades of 5.86% for HBX and 12.10% for XXX were reported for the bulk concentrate.

13.2 2020 Programme

A second programme of mineralogical examination and metallurgical testing was performed during 2020. Samples from the fresh (unweathered) High Grade Breccia ("HG_BX"), Low Grade Schist ("LG_Sch"), and the partially oxidized component of these two material types ("OX") were delivered to ALS Metallurgy facility in Kamloops, BC, Canada in the form of coarse drill core rejects. All sub-samples of each material type were combined to form a single composite for that type.

The HG_BX material can be to be considered a composite of the HBX and XXX rock types described in section 13.1 together with other hydrothermal breccias containing sulphide mineralization with similar chemical and mineralogical characteristics. Material from the LG_Sch and OX zones have not previously been subject to metallurgical testing.

The test work had the objective of producing concentrates that are suitable for toll treatment by pressure oxidation, Albion process, or roasting. Optimal concentrate target grades of 45–50 g/t gold and less than 15% arsenic were set during flowsheet development.

13.2.1 Chemical and Mineral Composition

Duplicate sub-samples from each composite were extracted and analysed. The results are shown in Table 1.13. The gold and silver grades for HG_BX and LG_Sch are extremely close to the resource block model, OX gold grades are slightly higher in the resource block model and lower in silver, however the difference is considered to be within acceptable limits. Other elements are within close limits.

Table 13.3: Chemical Composition of Samples.

		Metallurgical Samples			Resource Block Model		
Element	Unit	HG_BX	LG_Sch	OX	HG_BX	LG_Sch	OX
Gold	g/t	3.90	1.01	1.29	3.9	1.1	1.7
Silver	g/t	68	11	46	65	8.4	29
Copper	%	0.03	0.01	0.01	0.03	0.01	0.01
Lead	%	0.32	0.07	0.08	0.36	0.05	0.08
Zinc	%	0.66	0.15	0.04	0.83	0.12	0.18
Arsenic	%	1.2	0.86	0.6	1.26	0.73	0.73
Iron	%	3.4	2.4	3.1	4.70	3.19	3.91
Sulphur	%	3.17	1.56	0.19	3.76	1.58	0.21

Sub-samples of each composite were ground to a P_{80} of 80 µm and subjected to mineralogical examination by QEMSCAN Particle Mineral Analysis protocols and trace element examination by TESCAN Trace Mineral Analysis protocols. Detailed results are recorded in the laboratory report (ALS, 2020) and the mineralogical composition as determined by QEMSCAN is summarised in Table 13.4.

Table 13.4: Mineral Composition of Samples.

Mineral	**Unit**	**HG_BX**	**LG_Sch**	**OX**
Copper/silver sulphide	%	0.05	0.05	0.03
Galena	%	0.31	0.1	<0.01
Sphalerite	%	1.1	0.23	0.01
Pyrite	%	4.3	2.3	0.2
Arsenopyrite	%	2.5	1.7	0.2
Quartz	%	50.0	49.9	47.1
Feldspars	%	18.5	14.7	17.3
Micas	%	17.1	21.2	24.2
Calcium carbonate	%	1.8	4.2	0.2
Other non-sulphide gangue	%	4.4	5.4	10.7

Liberation of arsenopyrite and pyrite ranged between 55% and 64%, the liberation in the HG_BX and LG_Sch Fresh materials being higher than in the OX sample. Between 16% and 35% of the arsenopyrite and pyrite was measured in binary form with non-sulphide gangue minerals and was highest for the OX composite.

Sphalerite and galena liberations for the HG_BX and LG_Sch Fresh composites ranged between 52% and 61%. Between 16% and 22% of the sphalerite and 6% of the galena was measured in binary form with non-sulphide gangue minerals.

Photomicrographs of the three rock types are shown in Figure 13.1.

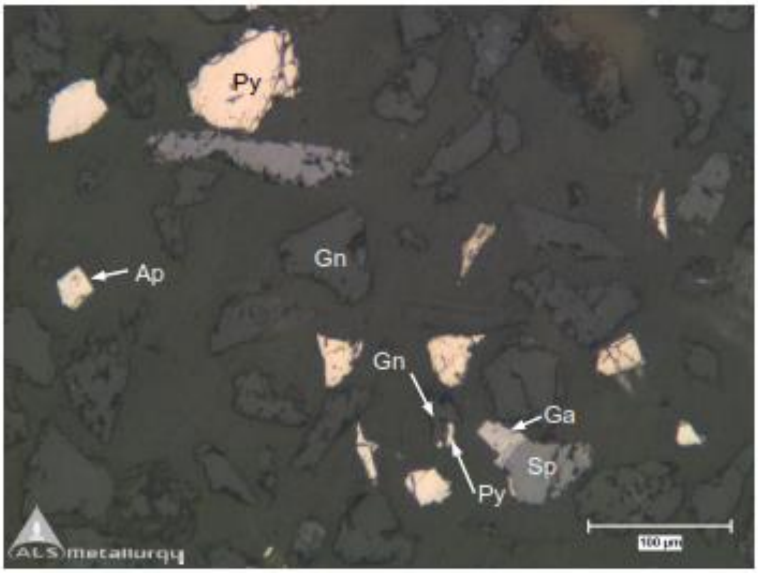
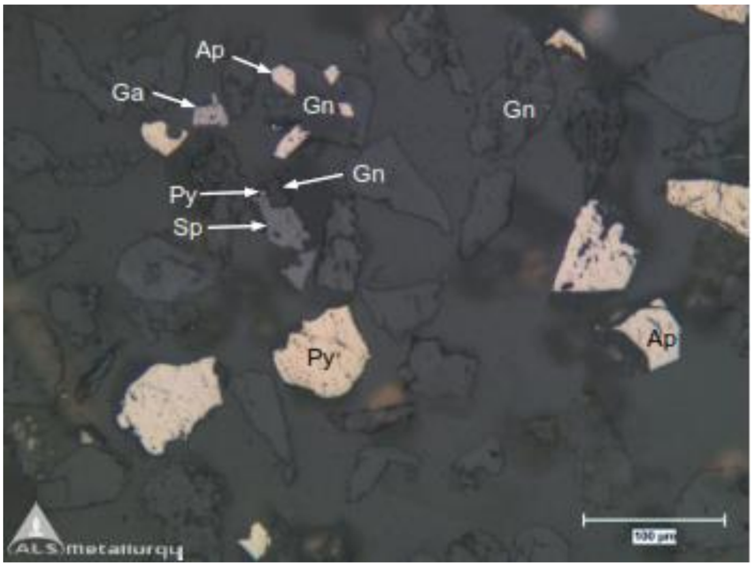
<table>
<tr><th colspan="2">Py = pyrite, Ap = arsenopyrite, Ga = galena, Sp = sphalerite, Gn = gangue</th></tr>
<tr><td>HG_BX</td><td>
</td></tr>
<tr><td>LG_Sch</td><td>
</td></tr>
<tr><td>OX</td><td>
</td></tr>
</table>

Figure 13.1: Photomicrographs of HG_BX, LG_Sch and OX Samples.

X-ray spectra indicated that many of the gold occurrences contained a percentage as silver, and that this varied widely. For one of the gold occurrences for the LG_Sch composite, tellurium was also measured.

The percentage of gold measured as liberated gold ranges between 35% for the LG_Sch composite and 53% for the HG_BX composite. The unliberated gold in these composites was generally associated with sulphide minerals such as sphalerite, pyrite, galena, and arsenopyrite. Although gold in binary forms with arsenopyrite was not measured, arsenopyrite was measured for gold occurrences within multiphase assemblages. Gold occurrences were relatively fine, with projected diameters between 2 µm and 10 µm. Between 92% and 100% of the gold occurrences observed were either liberated or associated with particles with high sulphide mineral particle surface exposures, which would indicate high potential for recovery in bulk flotation.

The number of gold occurrences detected for each composite was low, the highest being seen in the HG_BX Fresh composite. Estimated gold grades based on the gold occurrences detected by TESCAN were less than 0.1 g/t for each composite, indicating that less than 10% of the gold contained in the samples was detected by TESCAN and that a high percentage of the gold was not detected because it is either refractory or sub-microscopic. If the unseen gold was present within sulphide minerals, this would be considered recoverable to a bulk flotation concentrate.

For the OX composite, the gold was measured primarily as liberated gold grains. About 21% of the gold was measured in binary form with non-sulphide gangue as inclusions, and as such would not be expected to be recoverable in bulk sulphide flotation.

13.2.2 Gravity Concentration

Gravity concentration by a laboratory Knelson Concentrator was tested on a sample of each material type that had been ground to a P_{80} of 80 µm. The Knelson concentrate was hand panned, the results of which are shown in Table 13.5.

Table 13.5: Results of Gravity Concentration Tests (Pan concentrate).

Sample	Mass recovery %	Au grade g/t	Au recovery %	Ag grade g/t	Ag recovery %
HG_BX	1.2	65.1	19.5	432	7.4
LG_Sch	0.8	33.1	21.7	136	9.9
OX	0.5	20.2	6.3	992	11.2

Whilst the recovery of gold from HG_BX and LG_Sch was moderate, the recovery from OX was poor and the grades of all three gravity concentrates were low, indicating that gravity concentration has limited application to these material types. These findings are in line with the TESCAN results in that only a small portion of the gold is present as discrete particles.

13.2.3 Rougher Flotation

The results of the previous programme of test work showed that while good recovery had been achieved during bulk sulphide flotation, economic lead and zinc concentrates were not feasible, requiring an increase in concentrate grade to achieve marketable concentrates. This programme therefore aimed to develop the bulk sulphide flotation process, increasing the grade of gold in a bulk concentrate while limiting resultant decreases in recovery. To achieve this the secondary collector addition that had been used in the first programme was not applied. The grind was kept at approximately the same size since the recovery in the previous tests was good and this size was also indicated by the mineralogical analysis.

Samples of each composite were milled and batch rougher tests were performed. The test conditions were:

- Grind: P_{80} of 80 µm
- pH: natural (7.1 – 7.7)
- Collector: potassium amyl xanthate (PAX)
- Frother: methyl isobutyl carbinol (MIBC)
- Flotation time: 10 minutes.

The collector addition rate was varied according to the amount of sulphide minerals known to be present in each rock type:

- HG_BX 160 g/t in four separate doses at 2-minute intervals
- LG_Sch 120 g/t in four separate doses at 2-minute intervals
- OX 40 g/t in four separate doses at 2-minute intervals.

The results of the three rougher tests are summarised in Table 13.6.

Table 13.6: Results of Rougher Tests.

Sample	HG_BX	LG_Sch	OX
Mass pull to concentrate, %	11.1	8.4	2.1
Rougher concentrate Au grade, g/t	32.8	12.2	1.04
Au recovery to rougher concentrate, %	91.1	87.5	28.7
Rougher concentrate Ag grade, g/t	540	103	1 299
Ag recovery to rougher concentrate, %	90.6	90.4	63.6

The recovery of gold to the rougher concentrate of the OX test was low and no further flotation testing was performed on this material type.

The kinetics of the LG_Sch flotation test were observed to be slower than the test on HG_BX, particularly in respect of the zinc flotation, and single tests of each of copper sulphate (added at the grinding stage at 100 g/t) to activate sphalerite and Aero 208 (added to rougher flotation at 60 g/t) to promote gold flotation from the LG_Sch were made. These results are summarised in Table 13.7. The grades and recoveries of gold and silver to the bulk rougher concentrate were unaffected by these changes. Addition of the Aero 208 collector appeared to improve lead recovery and the copper sulphate improved zinc recovery in the bulk roughers. Since there was no benefit to gold and silver recovery from these additional reagents but the operating cost would increase if they were implemented and no further trials of these reagents were conducted.

Table 13.7: Results of Rougher Tests on LG_Sch Using Copper Sulphate and 208.

Sample	PAX only	PAX + $CuSO_4$	PAX + 208
Mass pull to concentrate, %	8.4	7.9	7.9
Rougher concentrate Au grade, g/t	12.2	12.8	12.8
Au recovery to rougher concentrate, %	87.5	86.8	86.6
Rougher concentrate Ag grade, g/t	103	100	100
Ag recovery to rougher concentrate, %	90.4	89.6	89.6

A single test was run on HG_BX composite material to investigate whether galena could be depressed without negatively affecting the flotation of gold and silver. Sodium metabisulphite (SMBS) was used as the depressant and added to the grinding stage at 1000 g/t and PAX addition was reduced to 40 g/t. The results of this test are presented with the results of the rougher test using PAX only for comparison in Table 13.8, from which it may be seen that the grade of lead in the rougher concentrate actually increased and its recovery was barely affected. The recovery of gold and silver was reduced while the gold grade was unaffected, and the silver grade increased slightly.

Should the lead grade of the final concentrate require to be reduced, alternative lead depressants could be investigated in future test work. The lower PAX addition appears to have affected the recovery of gold and silver only slightly and future test work should investigate the concentrate grade and recovery achievable at lower doses of collector.

Table 13.8: Results of Rougher Test on HG_BX Using SMBS.

Sample	PAX only	PAX + SMBS
Mass pull to concentrate, %	11.1	9.9
Rougher concentrate Au grade, g/t	32.8	32.9
Au recovery to rougher concentrate, %	91.1	84.9
Rougher concentrate Ag grade, g/t	540	591
Ag recovery to rougher concentrate, %	90.6	89.0
Rougher concentrate Pb grade, %	2.11	2.46
Pb recovery to rougher concentrate, %	81.4	81.8

13.2.4 Cleaner Flotation

Cleaner tests were performed on sub-samples of the HG_BX and LG_Sch composites. Bulk rougher concentrates were subjected to regrind and two stages of cleaning, as shown in Figure 13.2 below. The rougher flotation conditions were as used for the rougher tests and rougher concentrates were reground prior to cleaner flotation. The results are summarised in Table 13.9.

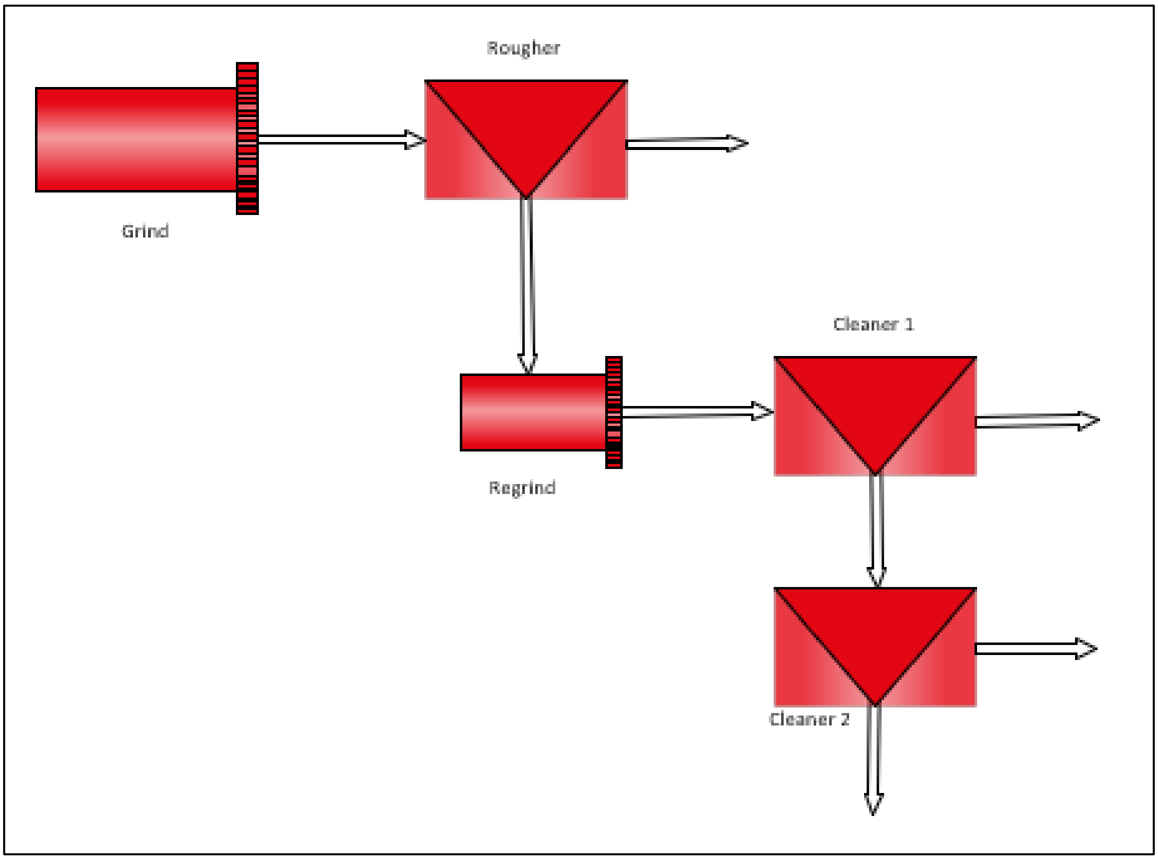


Figure 13.2: Flowsheet of Cleaner Flotation Tests.

Table 13.9: Results of Cleaner Tests.
** Recovery calculated from rougher feed.*

	HG_BX	LG_Sch
Recovery in rougher		
Au%	91.6	88.0
Ag%	93.4	90.5
Cleaner 1 concentrate		
Au g/t	45.9	22.0
Ag g/t	770	199
Cu%	2.81	0.22
Pb%	2.81	1.66
Zn%	8.17	2.75
As%	11.3	16.2
S%	35.3	30.4
Recovery to Cleaner 1 concentrate*		
Au%	85.8	76.5
Ag%	88.8	82.7
Cleaner 2 conc		
Au g/t	48.9	24.4
Ag g/t	824	228
Cu%	0.29	0.24
Pb%	3.02	1.94
Zn%	8.80	3.12
As%	11.8	18.0
S%	37.7	34.6
Recovery to Cleaner 2 concentrate*		
Au%	83.4	71.2
Ag%	86.8	79.2

13.2.5 Gravity and Flotation

A single test of gravity concentration followed by rougher and cleaner flotation, as shown in Figure 13.3, was performed on a sub-sample of the HG_BX composite. The results are presented in Table 13.10 with the results of the cleaner test without gravity concentration for comparison.

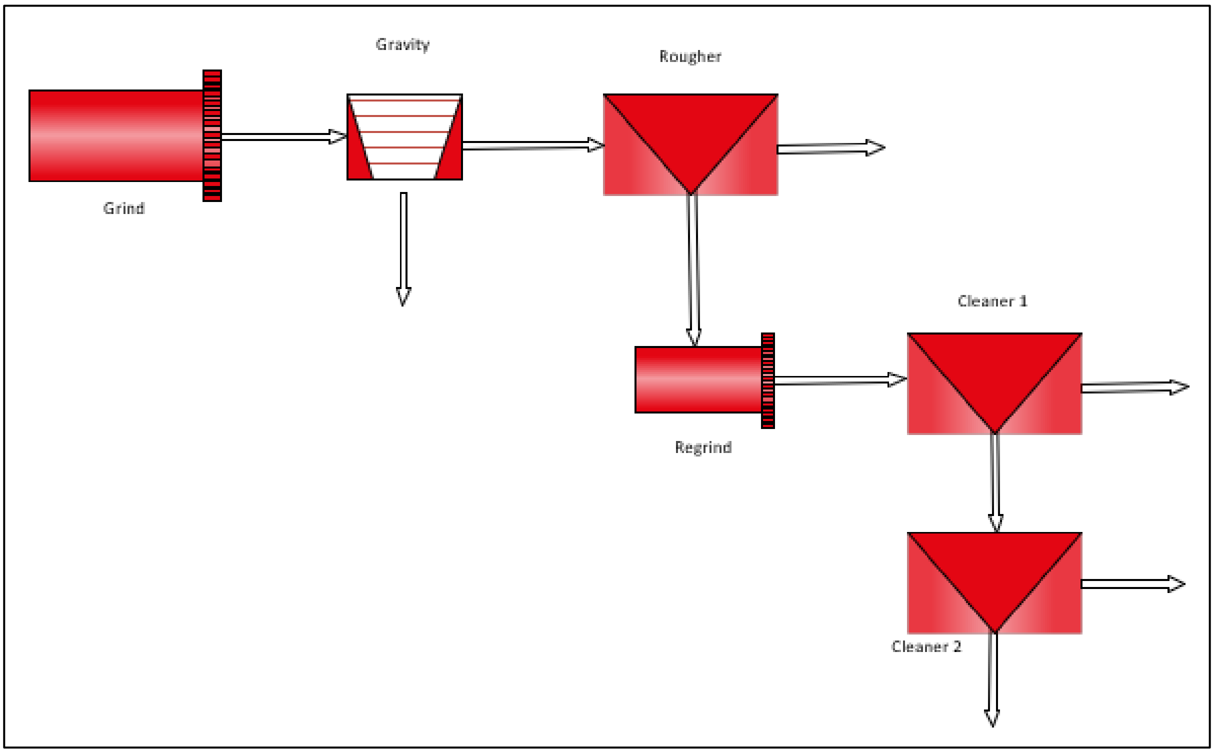


Figure 13.3: Flowsheet of Gravity + Cleaner Flotation Test.

Table 13.10: Results of Gravity Concentration and Cleaner Flotation.
** Recovery calculated from gravity feed.*

	Gravity + Cleaner	Cleaner only
Gravity concentrate		
Au g/t	72.9	N/A
Ag g/t	400	
Au recovery %	25.6	
Ag recovery %	9.3	
Cleaner 2 conc		
Au g/t	45.1	48.9
Ag g/t	930	824
Cu%	0.37	0.29
Pb%	3.31	3.02
Zn%	10.4	8.80
As%	10.1	11.8
S%	37.0	37.7
Recovery to Cleaner 2 concentrate*		
Au%	55.9	83.4
Ag%	76.5	86.8
Combined products		
Au g/t	51.2	48.9
Ag g/t	813	824
Cu%	0.31	0.29
Pb%	3.36	3.02
Zn%	8.5	8.80
As%	12.5	11.8
S%	37.1	37.7
Au recovery %	81.4	83.4
Ag recovery %	85.8	86.8

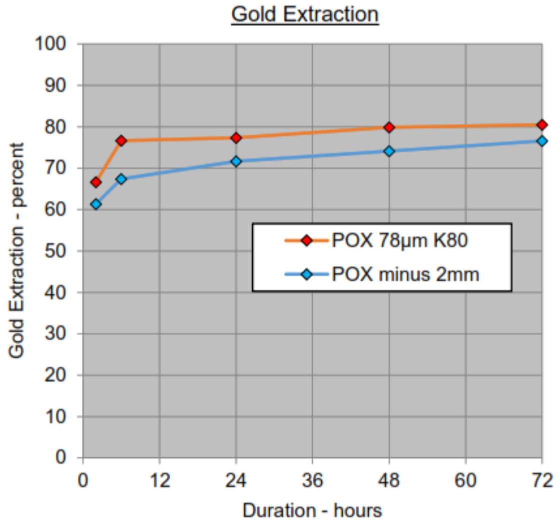
There was no discernible difference between using gravity concentration followed by rougher and cleaner flotation and using flotation alone.

13.2.6 Cyanide Leaching

Due to the poor flotation response of the OX composite material, cyanide leaching tests were also performed.

Tests were performed on the OX material in the state as it had been delivered to the laboratory and after grinding to a P_{80} of 78 µm. Bottle roll tests were conducted for 72 hours at pH 11 with a sodium cyanide concentration of 1000 ppm. Oxygen was also provided.

Figure 13.4 shows the leach extraction curves for gold and silver. The leaching of gold showed markedly superior kinetics to that of silver and grinding finer was beneficial to both kinetics and final extraction, as would be expected. The leach extractions after 72 hours were as shown in Table 13.11.



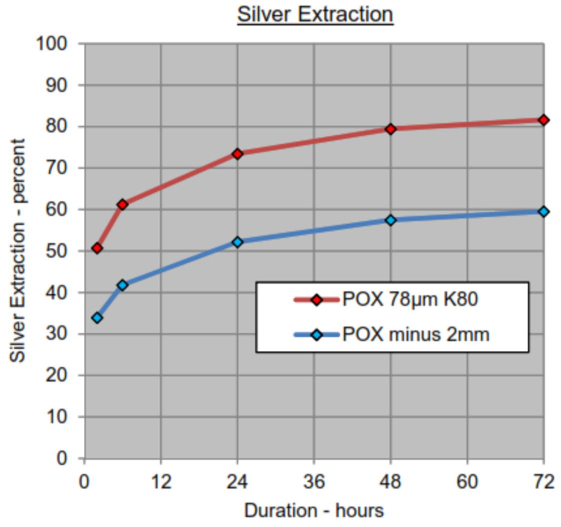


Figure 13.4: Cyanide Leach Extraction Curves.

Table 13.11: Results of Cyanide Leaching of OX Material.

Grind size	Gold			Silver		
	Recalc feed g/t	Leach tail g/t	Extraction %	Recalc feed g/t	Leach tail g/t	Extraction %
-2 mm	1.62	0.38	76.6	39	15.7	59.5
80% -78 µm	1.59	0.31	80.5	39	7.2	81.6

14 Mineral Resource Estimates

An initial Mineral Resource estimate of Au and Ag was completed by AMS in Q4 2019 with an effective date of January 13, 2020 for the Barje Deposit. The Mineral Resource estimate has been updated following metallurgical test work undertaken in support of this study as described in section 13 and to incorporate variation in commodity prices.

The updated Mineral Resource estimate has an effective date of January 07, 2021 and supersedes the previous initial Mineral Resource estimate, there has been no material change to the Mineral Resource estimate in terms of tonnage, grade and contained metal.

The only alteration to the block model used in Mineral Resource estimation was the refinement of the weathering surface used to represent the base of partially oxidized material as described in 14.4 below, individual block grades remain unchanged from the previous estimate as does much of the methodology which is described in the following subsections.

No Mineral Resource estimates were completed for other Prospects on the Tlamino Project.

14.1 Software Used

The Mineral Resource estimate was completed using Micromine software version 2020, Services Pack 1 with some data handling completed in Microsoft Excel. Updates to the Mineral Resource were completed in Micromine 2021.

14.2 Input Data Summary

The Mineral Resource estimate for the Barje Deposit utilized data from 33 diamond drill holes collected by Medgold during the 2018 and 2019 drill programmes. Four Avala drill holes (CKDD001 to 004) did not intercept significant mineralization but were used to restrict the extents of the model and inform its geometry. The total length of drilling used was 5822.7 m, including the Avala drill holes. This includes 4263 assay values over 4745.7 m, of which 587 assay values fall within the mineralized models over a length of 595.2 m. All drill core was logged for geology, core recovery and rock quality designation. The extents of the drill hole collars are shown in Table 14.1.

Table 14.1: Extents of drill collars used in Mineral Resource estimate.

Field	Records	Minimum	Maximum
X	37	615773.82	617138.391
Y	37	4691711.732	4692141.271
Z_real	37	1050.216	1264.21
Hole Depth, m	37	49.1	286.9

A digital terrain model was used as a surface constraint for the Mineral Resource estimate. The DTM was generated from a purchased data set of contours for the Serbian 1:25,000 topographic maps. When compared to DGPS elevations from geophysical surveys, the mean elevation difference between the DTM and geophysics is +2 metres with a standard deviation of 3.9 metres.

14.3 Data Validation

The drill hole data were validated using Micromine's drill hole database validation tools. The database was checked for errors such as overlapping intervals, intervals beyond drill hole collar depth, missing intervals, missing drill holes and large deviations in drill hole surveys. The drill hole traces were also visually inspected. A small number of minor errors were detected including an incorrect collar elevation in drill hole BAR032 and an erroneous drill hole survey at depth 120.1 metres in drill hole BAR020, which was removed.

The drill hole database is considered by the Qualified Person to be robust and fit for purpose in Mineral Resource estimation.

14.4 Domain Interpretation and Modelling

Mineralization at the Barje Deposit is predominantly held in a shallow-lying, hydrothermal breccia unit and in an overlying, weakly brecciated/veined schist. The mineralization is truncated to the southern edge by the steeply dipping, east-west striking, Barje Fault; to the northern edge by the topography where the unit sub-crops; and by the moderately shallow (35°) basement fault.

A restricting wireframe solid was created using the confining fault model surfaces and the digital terrain model. Two mineral domain solid models were generated using implicit modelling within Micromine to represent the higher-grade mineralization held within the hydrothermal breccia units (HG_BX) and the overlying lower grade mineralization held within the schist (LG_Sch). A combination of geology and Au assay grades were used in the interpretation of the mineralized domains, together with grade composites with a minimum grade of 0.3 g/t Au over an interval of 2 metres. Up to 2 metres of internal waste below this minimum grade was accepted. Additional control points were used to guide the Qualified Person's interpretation of the mineralization and isolated volumes were removed following the site visit and discussion with Medgold geologists.

Silver mineralization is associated with the Au mineralization, with higher grades of Ag being associated with the HG_BX domain (Figure 14.1). As a strong geological control is present Ag was treated as an accessory phase inside the Au domains.

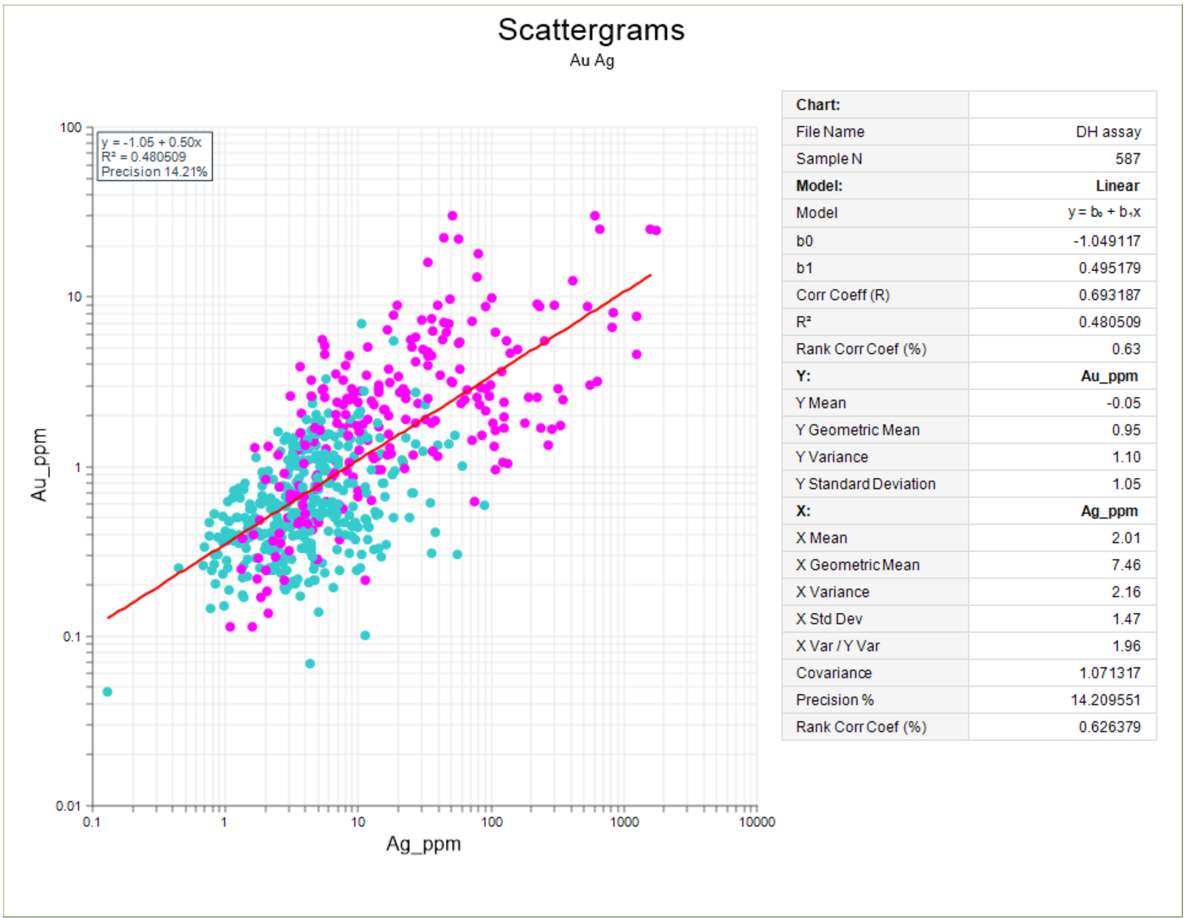


Figure 14.1: Au Ag relationship.
Points in pink are from the high-grade breccia domain, green points fall within the low-grade schist domain.

A surface representing the base of partially oxidized rock was generated. This was not used for domaining of assay data as strong oxidation is rarely observed within the mineralized drill core with no sign of supergene enrichment of Au. The surface was assigned to the block model to allow separate reporting of partially oxidized (OX) and fresh material (HG_BX and LG_Sch). As part of the January 2020 study this surface was generated using logging codes with some consideration of sulphur assay values, in this October 2020 update the surface was updated prioritizing low sulphur to base metal ratios in drill core assays over qualitative logging, relation to the depth below topography was also considered in areas of sparse data. The updated surface was generated by the Qualified Person for Geology and Resources (Mr Siddle) in collaboration with the Medgold's Consultant Exploration Manager Mr Sant.

The resultant wireframes are shown in Figure 14.2 and Figure 14.3.

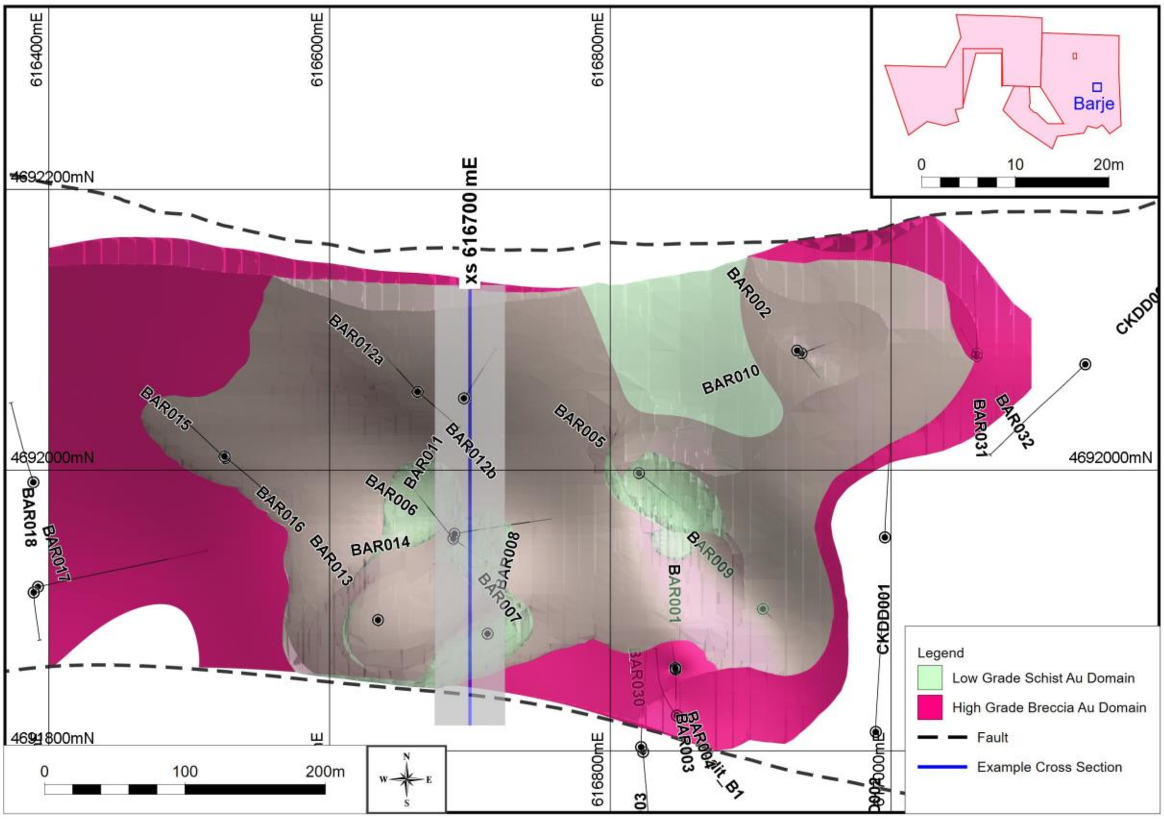


Figure 14.2: Plan view of mineralized wireframes used in resource estimation.
An example cross section line which is used throughout this section of the report is also shown.

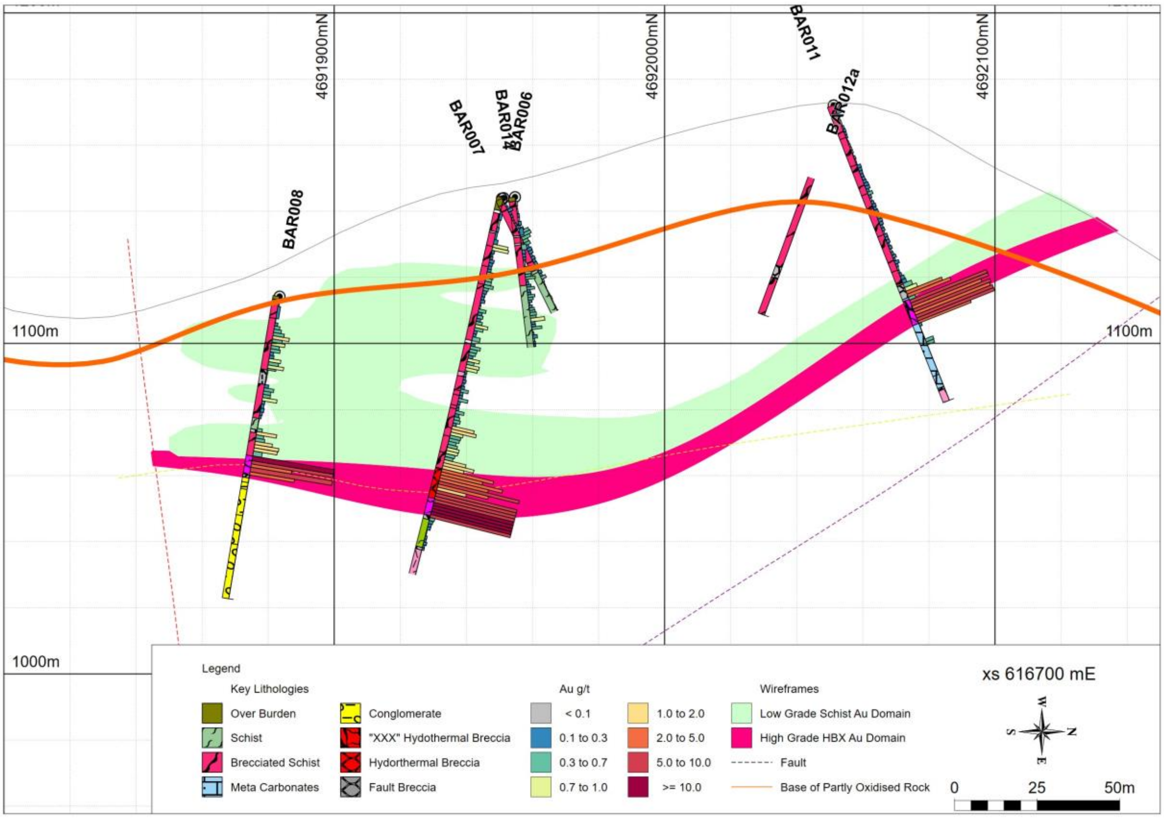


Figure 14.3: Example cross section 616700 mE.
Showing mineralized wireframes used in resource estimation.

14.5 Domain Statistics and Compositing

The mineralization wireframes were used to domain the Au and Ag assay data. Once a domain was assigned to each assay value the data were composited to 2 metre intervals for use in geostatistical analysis and grade interpolation. Length-weighted averaging was applied, a minimum composite length of 1 metre was accepted with residual lengths added to last qualifying composite. Composites were not allowed to cross domain boundaries.

Inspection of Au and Ag grades in histograms (Figure 14.4) and consideration of the 3D locations of the highest grade samples showed that there was no need to apply top cutting or grade capping to the input data used for Mineral Resource estimation.

Domain statistics for the combined domains are shown in Table 14.2. Of particular note is the low Coefficient of Variation for the combined domains of 1.66, suggestive of relatively low variability for Au within the combined populations (Au Coefficient of Variation values in excess of 3 or more are not uncommon in Au deposits). The length weighted mean of the assay data and the mean of the composite data compare favourably.

Table 14.2: Assays and composite statistics for Au and Ag for the combined mineral domains.

	Assays		**Composites**	
Field Name	Au_ppm	Ag_ppm	Au_ppm	Ag_ppm
Minimum	0.05	0.13	0.14	0.79
Maximum	30.00	1750.00	26.15	1226.90
No of Points	587.00	587.00	303.00	303.00
Sum	1080.90	22269.23	558.17	11374.96
Mean	1.84	37.94	1.84	37.54
Variance	11.11	21209.87	9.21	16169.57
Std Dev	3.33	145.64	3.04	127.16
Weighted Mean	1.85	37.81	1.84	37.50
Weighted Variance	11.48	21062.58	9.26	16284.09
Weighted Std. Dev.	3.39	145.13	3.04	127.61
Coeff. of Variation	1.81	3.84	1.65	3.39

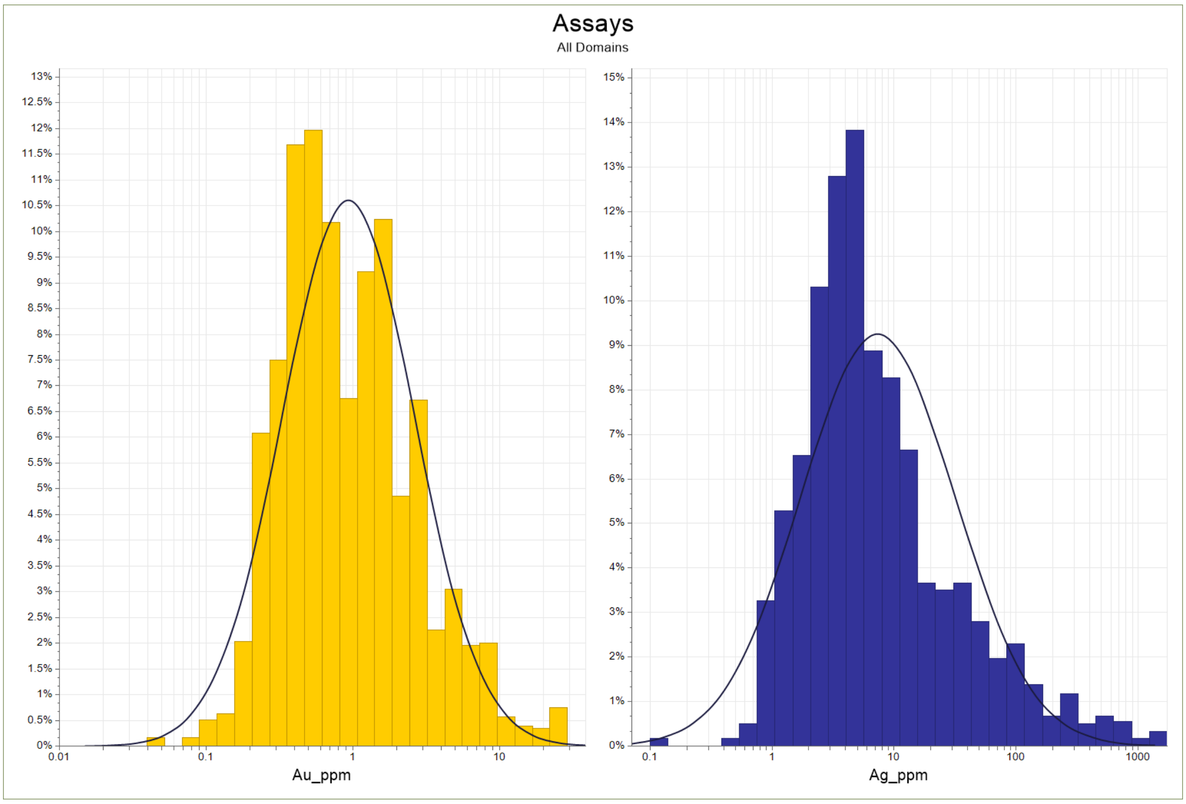


Figure 14.4: Histograms for Au and Ag assays for the combined mineral domains.

14.6 Model Unfolding

The deposit shows some weak folding and variable dip, particularly to the base of the high breccia domain. To improve variography and grade mapping the model the base of the high-grade domain was used to generate a structural trend in Micromine to unfold and stretch the model. While the impact on geostatistical analysis, given the sparse data, is considered to be minimal, the unfolding results in a significant improvement in grade mapping during grade interpolation which would not be achieved by dynamic anisotropy alone. The resultant structural tend is shown in Figure 14.5.

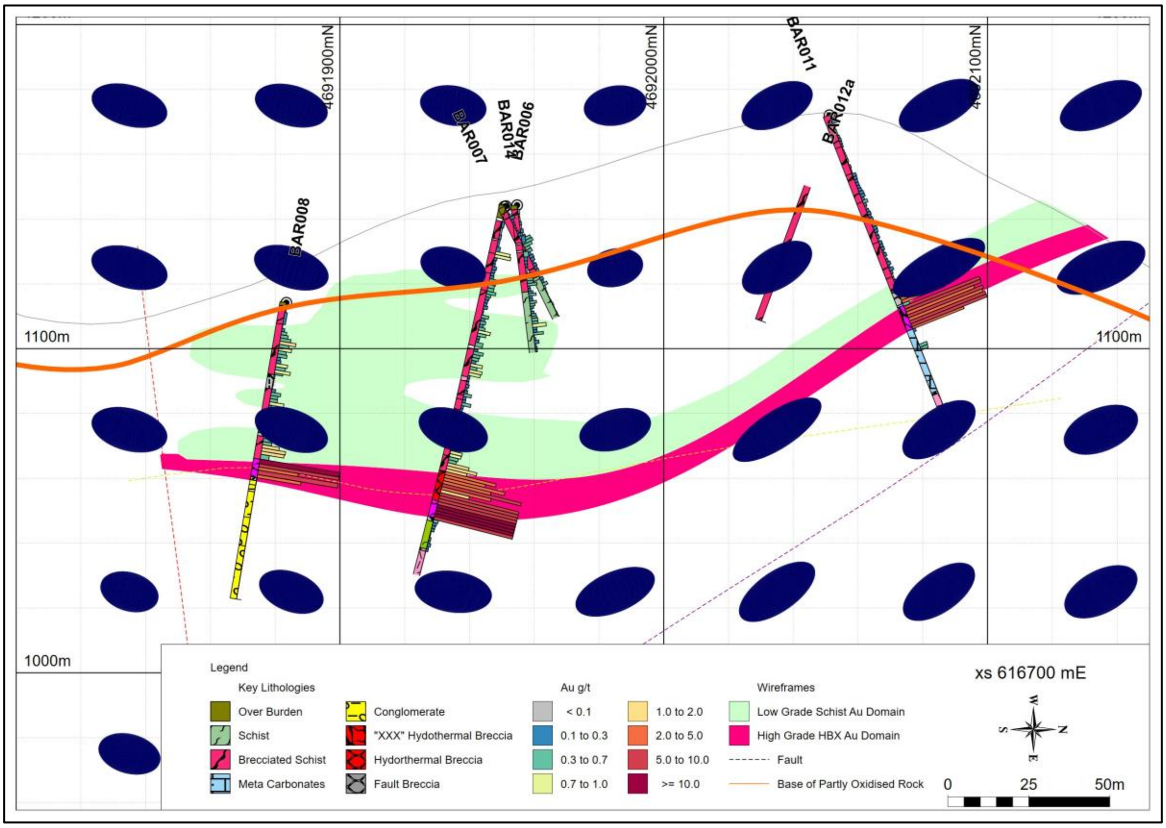


Figure 14.5: Structural trend used to unfold the model.

14.7 Geostatistics

Variography was conducted on the Au value of the composite data in unfolded space for each domain in isolation. The Au variograms were used to interpolate all elements. The data are currently relatively sparse and experimental semi-variograms are not well-defined. The median indicator variograms were generated for both domains and two component models fitted (Table 14.3, Figure 14.6 and Figure 14.7).

Table 14.3: Variogram Parameters.

Axis	Azi	Plunge	Nugget	Component 1 Range	Component 1 Partial Sill	Component 2 Range	Component 2 Partial Sill
HG_BX Domain							
1	227.64	0.06	0.08	100	0.01	140	0.16
2	317.64	0.31	0.08	104.7	0.01	160	0.16
3	127.53	89.68	0.08	5	0.01	10	0.16
LG_Sch Domain							
1	227.64	0.06	0.06	92	0.11	267.9	0.08
2	317.64	0.31	0.06	89.2	0.11	241.2	0.08
3	127.53	89.68	0.06	21.07	0.11	29.67	0.08

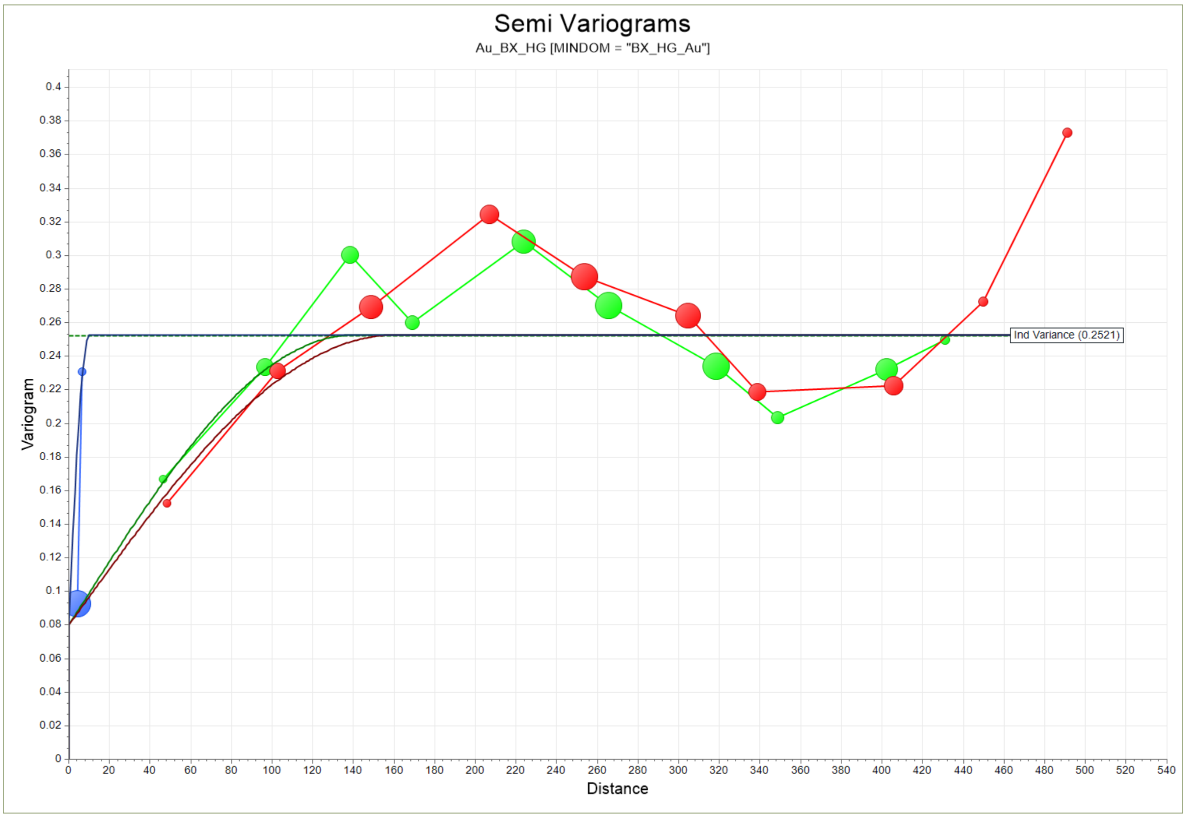


Figure 14.6: Median Indicator Directional Semi-Variograms for Au in HG_BX Domain.

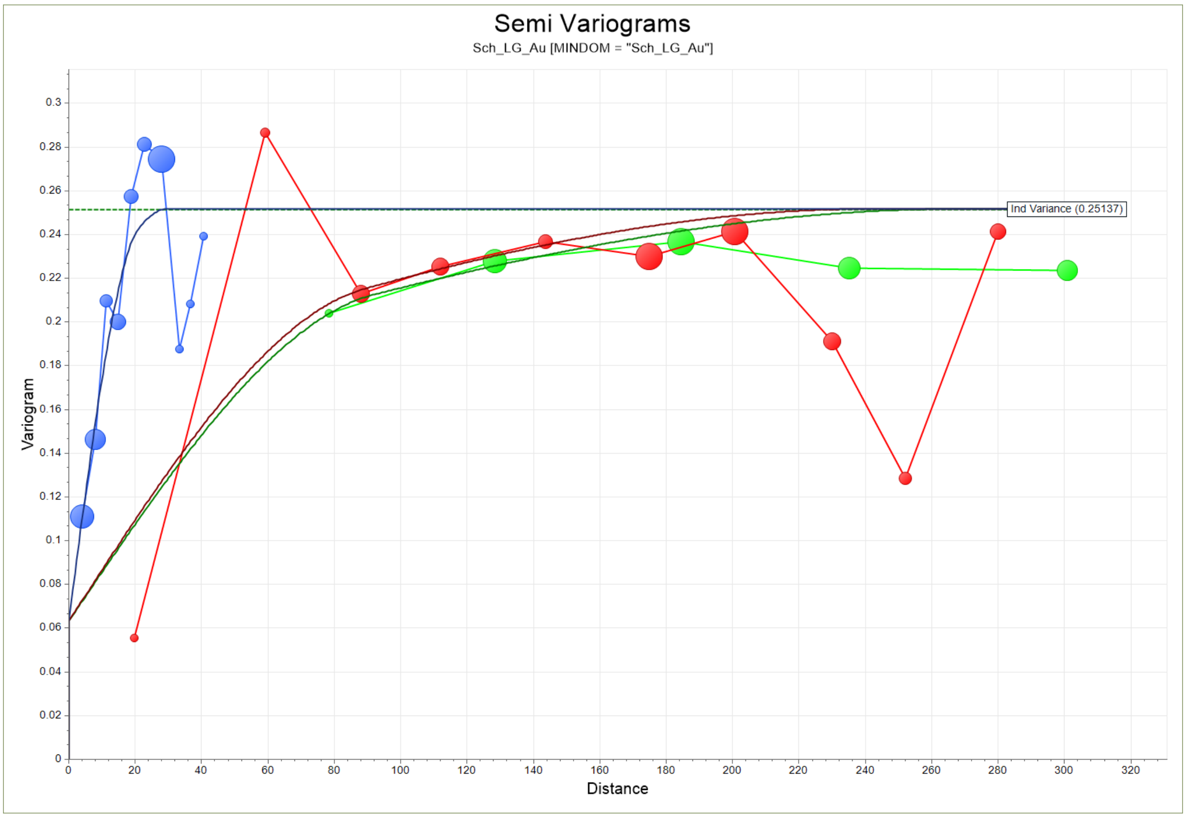


Figure 14.7: Median Indicator Directional Semi-Variograms for Au in LG_Sch Domain.

14.8 Block Modelling and Grade Interpolation

A block model was created and restricted to the mineralized wireframes and the topographic surface. The block model dimensions are shown in Table 14.4. Sub-blocking was applied, with up to 5 x 5 x 2 block divisions in the East, North and Z directions respectively giving a minimum sub block size of 2 meters in each direction.

Table 14.4: Block Model Dimensions.

	Min Centre	Block Size m	Max Centre	Blocks
East	616400	10	617090	70
North	4691790	10	4692210	43
Z	1028	4	1180	39

Unfolded space coordinates were generated for each cell and the model interpolated in unfolded space. Grade interpolation was completed using Ordinary Kriging and the Au variograms for the respective domains. Hard boundaries were used for each domain. A multiple pass kriging neighbourhood strategy was employed and can be described as follows.

- Single sector search ellipsoid
- Maximum of 2 composite samples per drill hole
- Maximum of 10 samples total
- High grade domain search ellipsoid 20 m vertical thickness by 60 m horizontally, increasing to 160 m horizontally for pass 2
- Low grade domain search ellipsoid 40 m vertical thickness by 60 m horizontally, increasing to 200 m horizontally for pass 2
- Block discretization; 5 x 5 x 2 m equal division in the East, North and Z directions respectively.

The following elements were interpolated into the block model using the above parameters: Au, Ag, As, Cu, Pb, Zn, Fe, and S. The selected Kriging neighbourhood was validated by visual inspection of Kriging statistics, grade mapping and use of Micromine's Quantitative Kriging Neighbourhood Analysis tools.

14.9 Bulk Density

During the site visit, a strong relationship was observed between the density of samples and the abundance of sulphide mineralization in the dill core. Medgold has collected 397 bulk density determinations of which 152 are within the mineralized wireframe, including 90 in the lower grade domain and 62 in the higher-grade domain.

A broad linear relationship has been determined from the combined major sulphide mineral concentrations and bulk density within the fresh rock.

The formula 2.57 + 0.02(As%+Cu%+Fe%+S%+Pb%+Zn%) was used to estimate bulk density within the block model on a cell-by-cell basis for fresh rock. The regression formula was recalculated following the update to the base of partially oxidized material surface and did not change.

Within the partially oxidized rock, a lower default density of 2.54 g cm^3 was used. Box and whisker plots (Figure 14.9) show that density within individual domains is close and, as such, the higher and lower grade domains no not need to be treated separately at this stage.

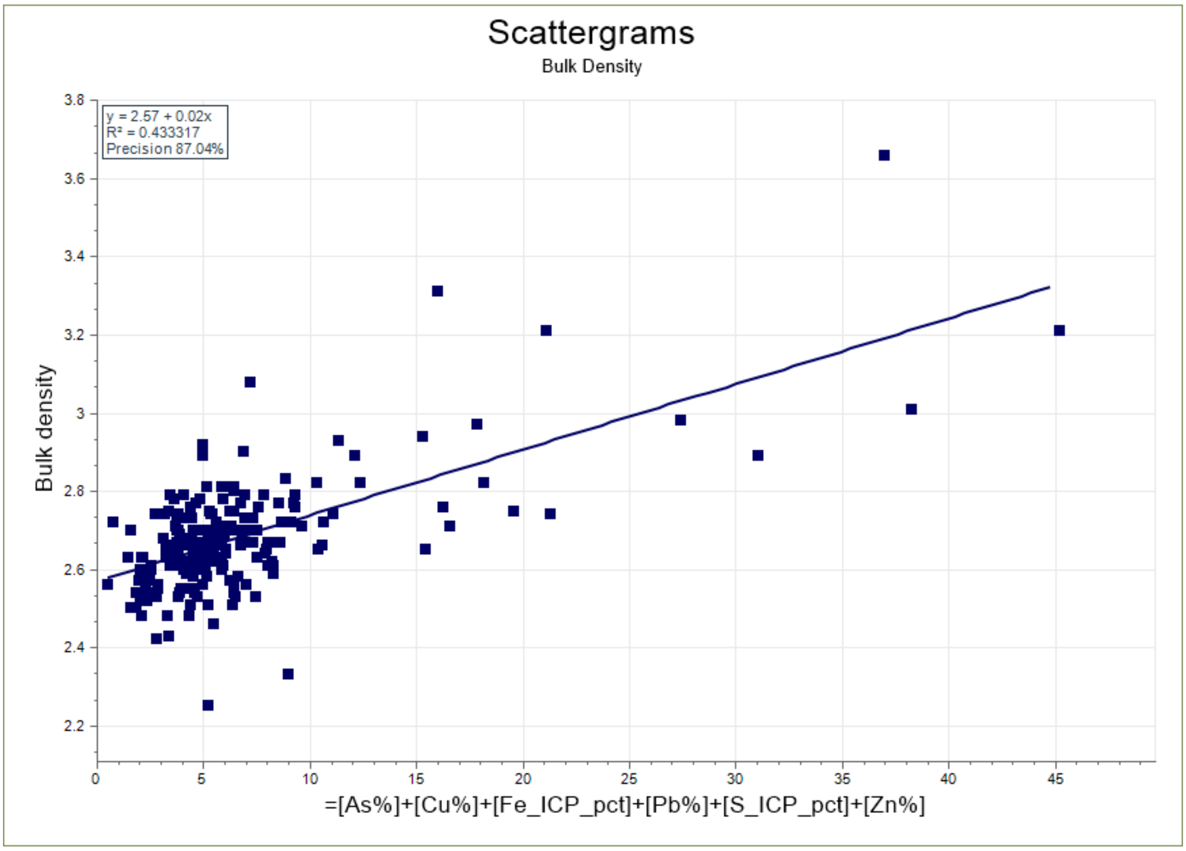


Figure 14.8: Bulk Density vs Sulphide Element Concentrations.

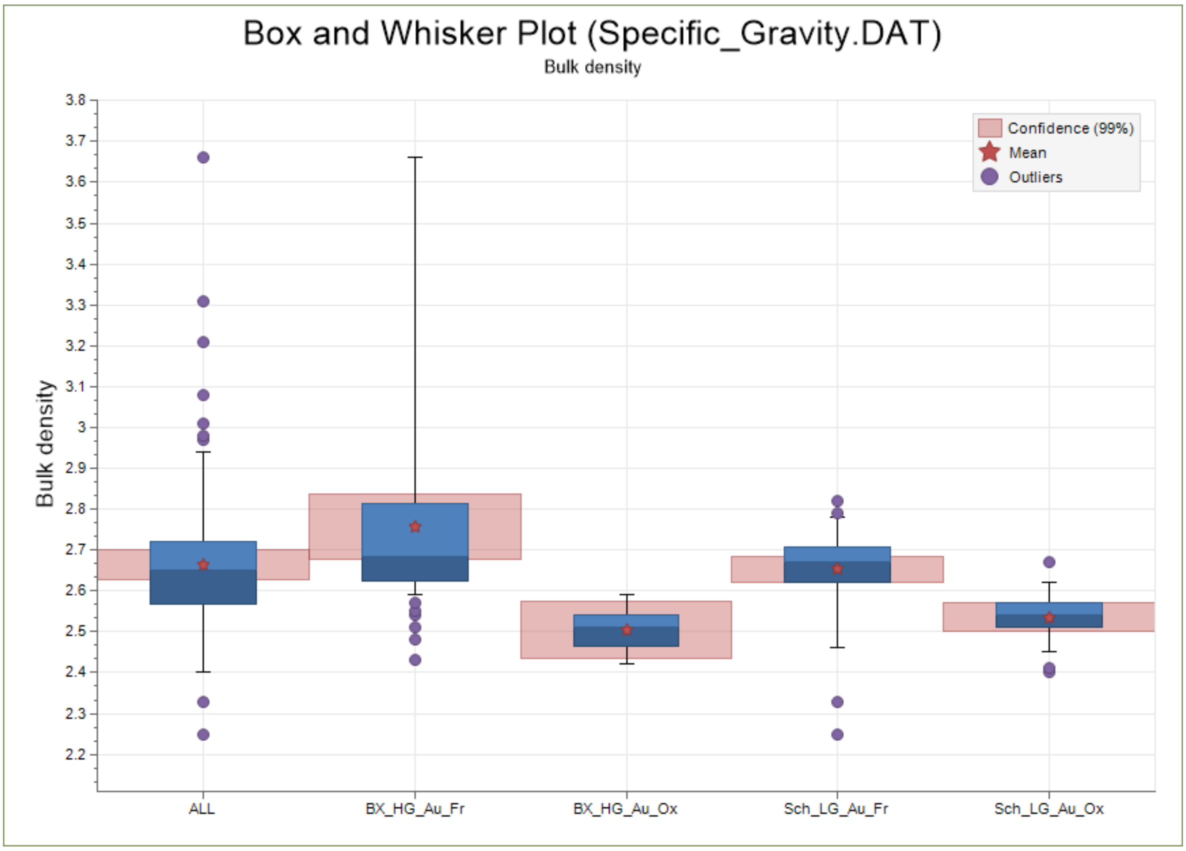


Figure 14.9: Box and Whisker plots for Bulk Density by mineral domain and weathering horizon.

14.10 Model Validation

The model was validated, globally, by inspection of the input and output statistics and, locally, by comparison of the model and composite data in 3D space. The global mean of the input composite data compares favourably with the volume-weighted mean of the block model for Au (Table 14.5). The Ag mean is lower in the block model than in the composite data; inspection of Q-Q plots (Figure 14.11) and histograms (Figure 14.12 and Figure 14.13) shows that the high grade tail of the Ag in the block model is diluted, possibly indicative of some over-smoothing. The QQ plots and histograms for Au show a good reproduction of the distribution with some smoothing and squeezing of the tails towards the mean, as was expected.

The number of input data are relatively small and the drill spacing is sparse, which accounts for some of the over-smoothing in Ag. The statistical global model validation is considered acceptable and suitable for reporting of Mineral Resources to appropriate levels of classification.

Table 14.5: Comparison of Composite vs Block Model Statistics.

	Composites		**Block Model**	
Field Name	Au_ppm	Ag_ppm	Au_ppm	Ag_ppm
Minimum	0.14	0.79	0.15	0.79
Maximum	26.15	1226.90	21.59	1226.90
No of Points	303.00	303.00	33827.00	33827.00
Sum	558.17	11374.96	65306.01	852778.87
Mean	1.84	37.54	1.93	25.21
Variance	9.21	16169.57	4.35	4454.56
Std Dev	3.04	127.16	2.09	66.74
Weighted Mean	1.84	37.50	1.86	27.01
Weighted Variance	9.26	16284.09	5.42	6226.71
Weighted Std. Dev.	3.04	127.61	2.33	78.91
Coeff. of Variation	1.65	3.39	1.08	2.65

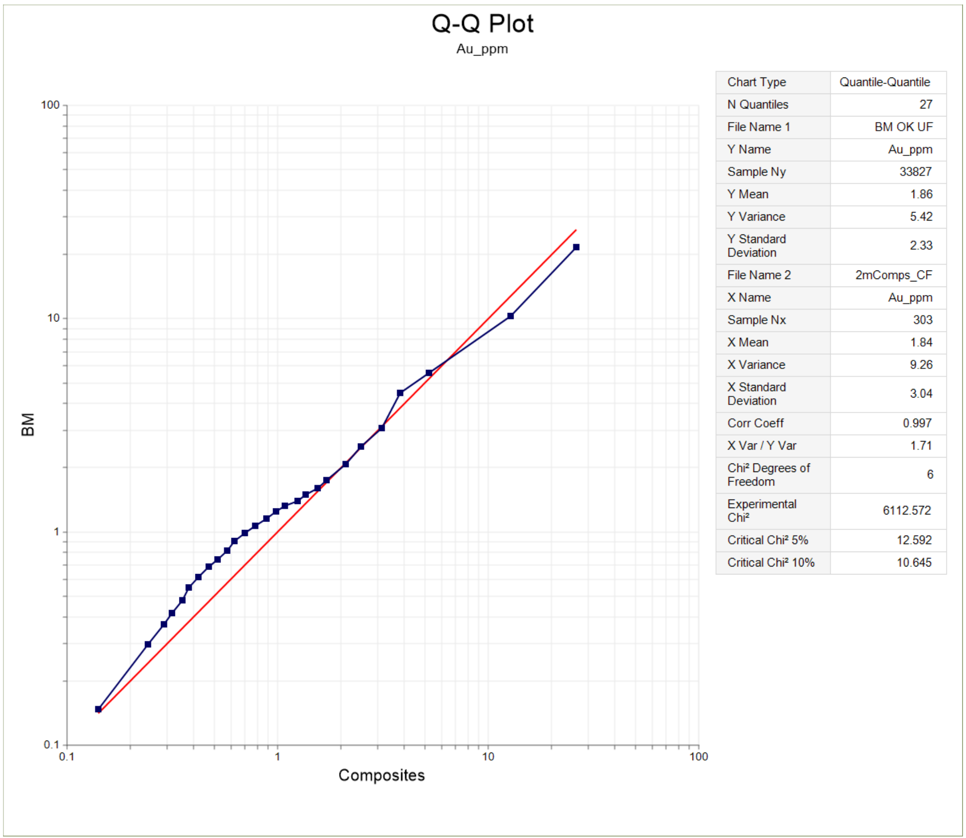


Figure 14.10: Q-Q Plot for Au values in Block model vs Composites.

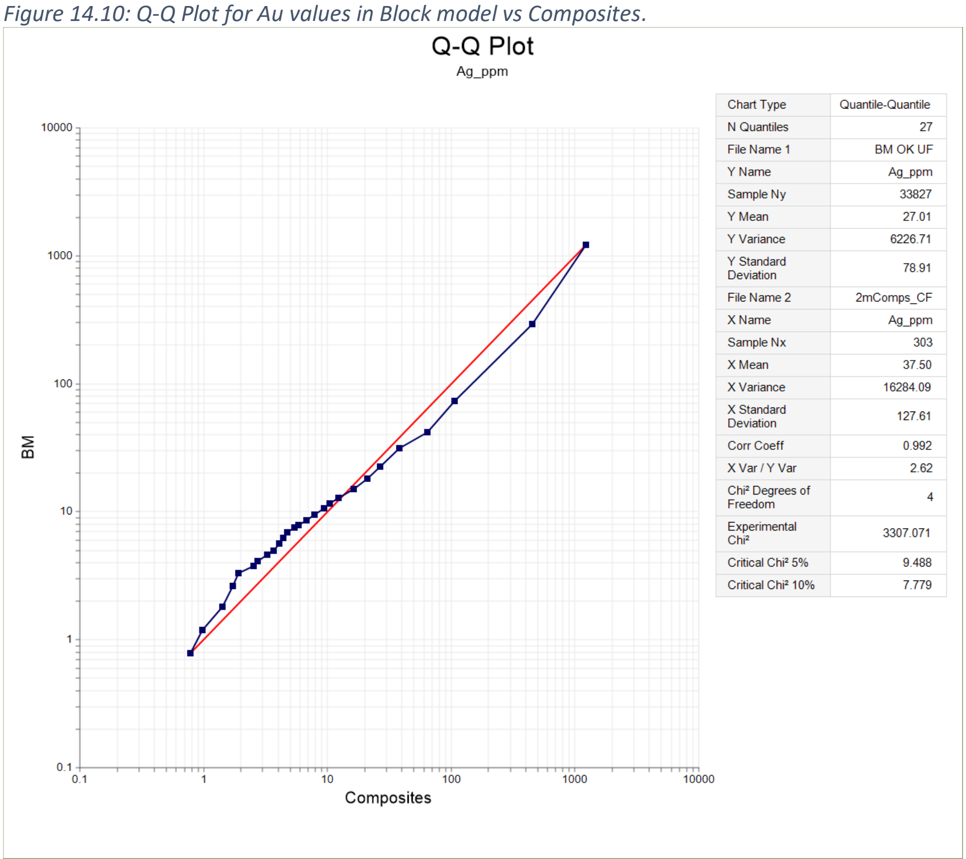


Figure 14.11: Q-Q Plot for Ag values in Block model vs Composites.

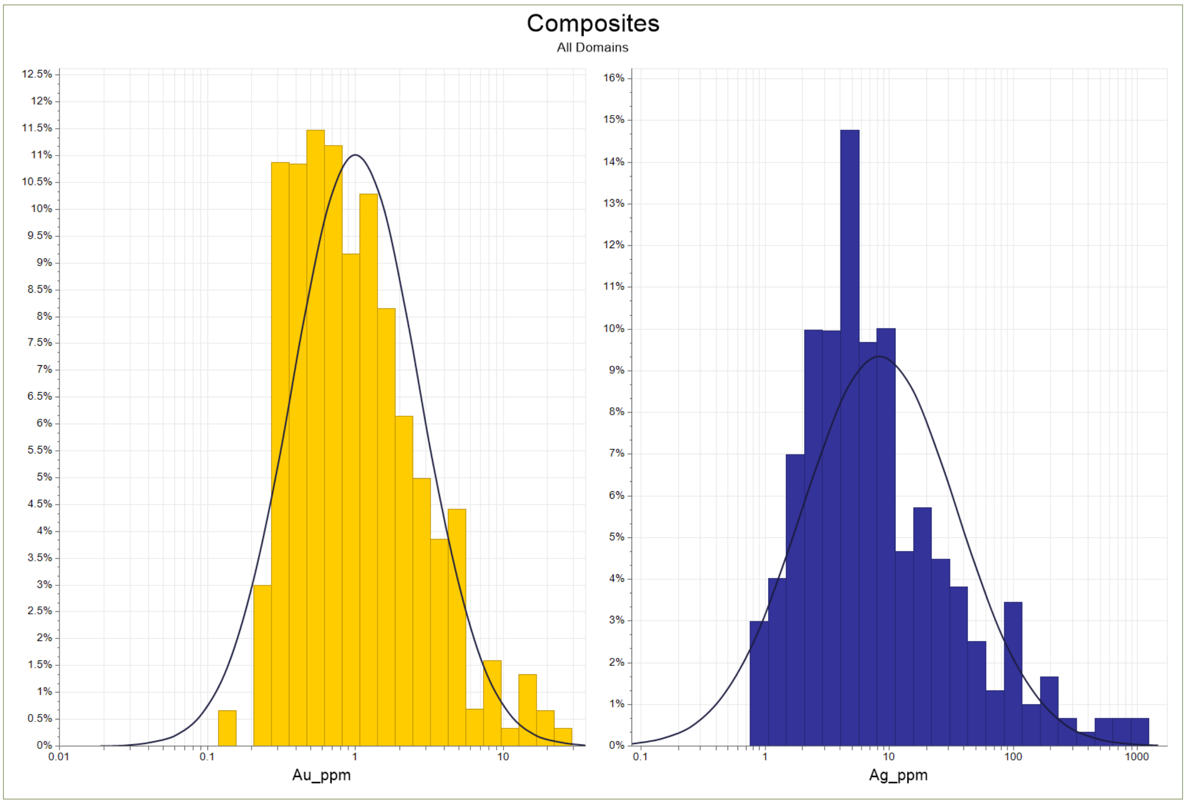


Figure 14.12: Histograms of Au and Ag Composite Data.

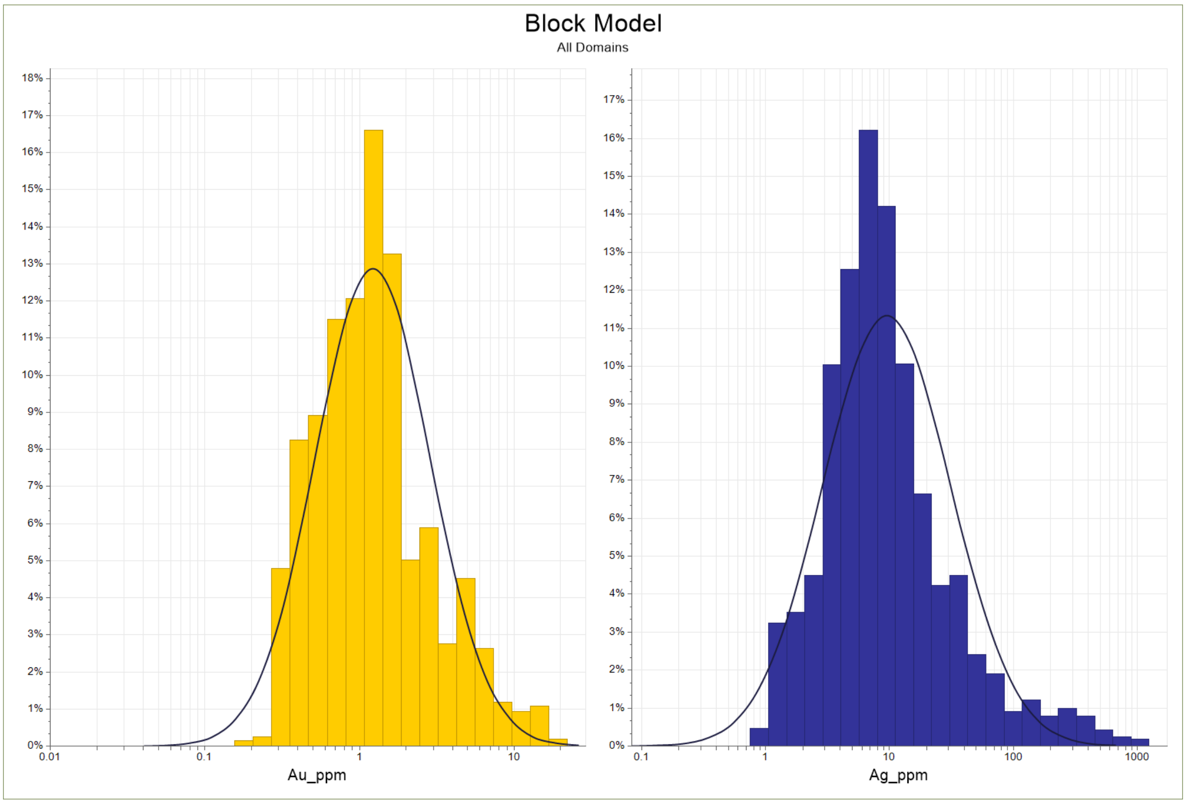


Figure 14.13: Histogram of Au and Ag Block Model Data.

The Block Model was inspected in cross section and in 3D space; in cross section, grades were found to be preserved and map extremely well between points of observation, low-grade areas and high-grade areas were preserved extremely well with minimal smearing of high grades.

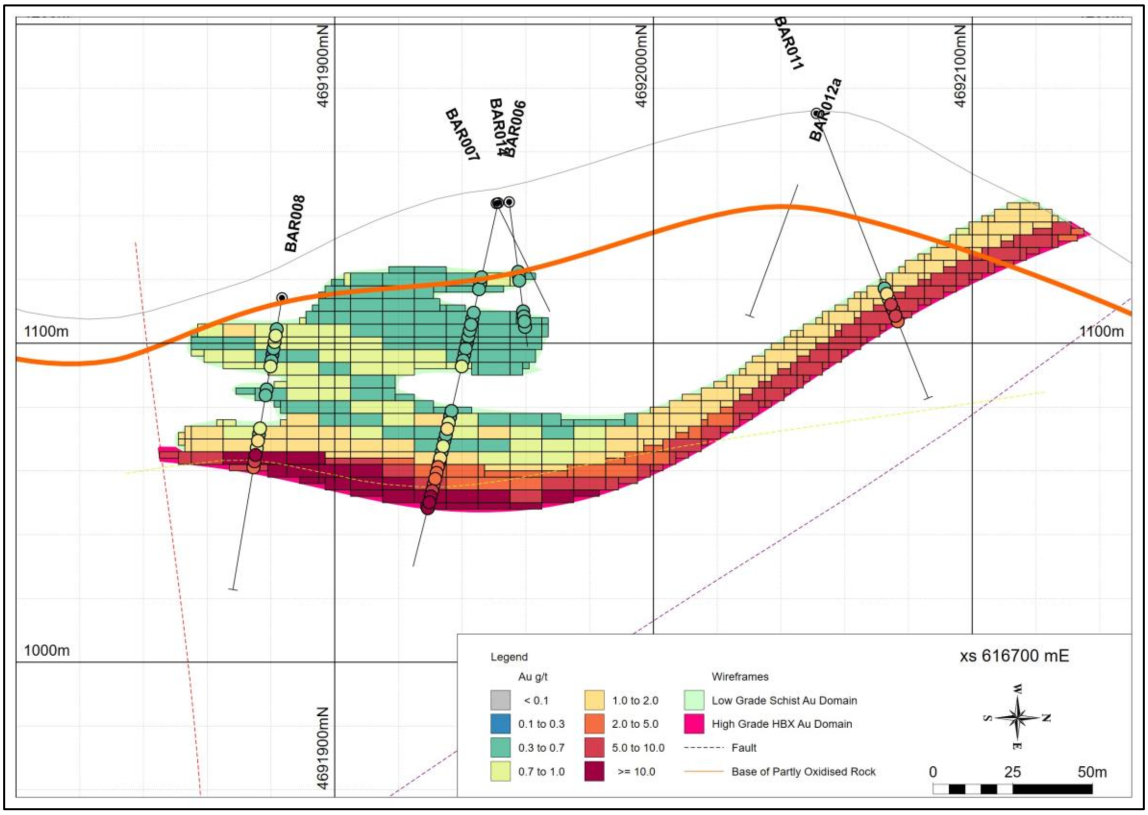


Figure 14.14: Grade preservation between composite samples and block model.
Example cross section 616700 mE. Composite data and block model both colour coded to Au grade.

14.11 Resource Classification

In the resource classification of the Barje Mineral Resource estimate, the Qualified Person has considered the following factors:

- Data spacing is relatively sparse, of the order 80 to 100 m and that in places, data have been extrapolated up to 160 m.
- The assumed geological continuity of the high-grade hydrothermal breccia domain is good, showing a laterally extensive system which is closed off by faults and drilling. There is some degree of variability in the thickness of this unit. The grade continuity of the unit appears to be very good with variogram ranges in excess of 100 m, although experimental variograms are poorly formed due to lack of data and the grade continuity has not been confirmed or modelled at shorter ranges.

- The assumed geological continuity of the low-grade schist domain is good close to the contact with the hydrothermal breccia, showing a laterally extensive system immediately overlying the high-grade domain. However, beyond 10-15 m above this contact, the geological continuity of the mineralization is less apparent and different interpretations, for example, a steep vein system, may result in a differing estimation of tonnages. As with the high-grade breccia domain, grade continuity of the low grade domain appears to be very good with variogram ranges in excess of 100 metres, although experimental variograms are poorly formed due to lack of data and the grade continuity has not been confirmed or modelled at shorter ranges.
- The input data are of a high quality with good quality control, which the Qualified Person has reviewed. Survey and location control of samples and geological contacts is excellent. Logging is excellent.
- A robust Bulk Density database has been collected.
- The digital terrain model is currently fit for purpose however it shows some discrepancies when compared to DGPS data. The resolution of DTM may restrict the classification of resources close (within ~6 metres of the surface).
- Preliminary metallurgical test work has been completed which gives an indication of the likely recoveries achievable and likely process route.

It is the opinion of the Qualified Person that, given the relatively sparse drilling at Barje, the resource classification be restricted to the Inferred category. Inferred Mineral Resources are defined by CIM as follows:

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

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

Despite the above, a Preliminary Economic Assessment that is based on Inferred Mineral Resources may be disclosed if the disclosure states with equal prominence that the preliminary economic assessment is preliminary in nature, that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the preliminary economic assessment will be realized.

14.12 Metal Equivalent and Reasonable Prospect of Economic Extraction

The cut-off determination is based on the assumption that sale of a Au concentrate is the most likely product route for the unweathered HG_BX and LG_Sch material types, partially oxidized material (OX) is amenable to leaching as shown by testing described in section 13.2.6 - Cyanide Leaching.

An Au price of US$1500/oz and an Ag price of US$16.5/oz were used as the base case for estimation of metal equivalents and used in cut of grade determination. Taking into account the individual metallurgical recoveries of Au and Ag for each material type, a value for the conversion of Ag to Au equivalent was determined as follows:

$$Ag\ Value\ \times Ag\ Recovery \div Au\ Value\ \times Au\ Recovery$$

No other elements are considered to have a reasonable prospect of economic extraction. Metal equivalent conversion values for silver to gold are given in Table 14.6 for each material type.

Table 14.6:Metal equivalent factors for conversion to AuEq

	High Grade Breccia		Low Grade Schist		Partially Oxidized	
	Au	Ag	Au	Ag	Au	Ag
Base Case Metal Price US$/oz	1500	16.5	1500	16.5	1500	16.5
Process Recovery Factor	0.858	0.843	0.765	0.827	0.8	0.8
Metal Equivalent Factor	1	0.011	1	0.012	1	0.011

For each cell in the block model, an Au equivalent grade was calculated by multiplying the Ag grade by the metal equivalent factor and adding to the Au grade.

Preliminary mining analysis completed as part of this study (section 16) showed that open pit mining is the most likely extraction method for mining with floatation to a bulk concentrate for mineral processing. The preliminary economic assessment did not consider oxide material due to only cursory leach testing results being available. The oxide material is still considered by the Qualified Person for Resources as having a reasonable prospect of economic extraction.

To identify material that has a reasonable prospect of economic extraction for Mineral Resource reporting, a pit optimisation was completed using the same parameters described in the base case mining study in section 16 (including pit slope parameters), but also including oxide material as a payable material type based on the assumption of leaching and the production of a doré bar. Input parameters for the pit optimisation are given in Table 14.7. The pit optimisation was completed using the same regularized 5 x 5 x 2.5 m (E x W x Z) block model used in the PEA study which accounted for material mixing, dilution and mining loss within the individual block model cells. As such no dilution or mining loss was applied in the pit optimisation algorithm. A more detailed description of the pit optimisation process is given in section 16.3- Pit Optimisation.

The ultimate pit shell differed only slightly from pit shell used in the PEA at the edges of the pit near surface where it widened to include additional oxide material that had been treated as waste in the PEA pit optimisations described in section 16.

Table 14.7: Pit optimisation parameters for Mineral Resource reporting.
**Transport costs of selling Au doré considered to be within typical metal price fluctuations and not applied to partially oxidized material.*

	High Grade Breccia		Low Grade Schist		Partially Oxidized	
	Au	Ag	Au	Ag	Au	Ag
Base Case Metal Price US$/oz	1500	16.5	1500	16.5	1500	16.5
Payability	75	75	60	60	98	98
Royalty Gross %	5	5	5	5	5	5
Process Recovery %	85.8	84.3	76.5	82.7	80	80
Mining Cost US$/t	2.3	2.3	2.3	2.3	2.3	2.3
Process Cost US$/t	10	10	10	10	12	12
GNA US$/t ROM	4	4	4	4	4	4
Transport US$/t ROM	2	2	2	2	0*	0*

Material falling within the ultimate pit shell was considered for economic potential through application of an AuEq cut-off grade. The variability of the gold price was considered when assessing cut off grades which have a reasonable prospect of economic extraction. The three-year trailing average gold price from the London Bullion Market Association (LBMA) to October 2020 was US$1466/oz with a monthly average high of US$1970/oz in August 2020 and monthly average low of US$1199/oz in September 2018, the two-year trailing average was US$1550/oz to the same date and the one-year trailing average was US$1700/oz. The LBMA annual forecast survey published February 3, 2020, had an average low of US$1443.9, average high or US$1688.6 and overall average of US$1558.8 (LBMA, 2020).

Based on the above, Au prices ranging from US$1300/oz to US$1700/oz were considered against break even Au equivalent cut-off grades; the Ag price and the input parameters described in Table 14.7 were not varied. Table 14.8 shows the effect of gold price on estimated break-even cut-off grade. Based on the break-even cut-off grades at the selected gold prices, high, low, and base cut-off values were selected for reporting to show sensitivity of cut-off grade to metal process. The gold prices used, and the resultant cut off grades presented are all considered by the Qualified Person for Resources suitable for identifying material which has a reasonable prospect of economic extraction.

Table 14.8: Break even cut-off grade sensitivity to Au price with High, Low and Base cases selected for reporting.

	Cut-off grade Au g/t		
Au Price US$	High Grade Breccia	Low Grade Schist	Partially Oxidized
1300	0.66	0.88	0.54
1400	0.61	0.82	0.5
1500	0.57	0.76	0.47
1600	0.53	0.71	0.44
1700	0.5	0.67	0.41
Case			
High cut-off	0.7	0.9	0.6
Base cut-off	0.6	0.8	0.5
Low cut-off	0.5	0.7	0.4

14.13 Block Modelling Results

The grade tonnage estimates for the Barje Deposit are presented in Table 14.9. Figures are reported for each material type as well as the total resource after the application of the base case cut-off grade for each material type. All figures presented are considered by the Qualified Person for Resources to have a reasonable prospect of economic extraction. The base case cut-off grades selected for reporting of the Mineral Resource are highlighted and are as follows; high grade breccia, 0.6 g/t AuEq; low grade schist, 0.8 g/t AuEq and partially oxidized material, 0.5 g/t AuEq.

The grade tonnage curves for all material are shown in Figure 14.15 to Figure 14.17 with resultant wireframes and block model shown in Figure 14.18 to Figure 14.20. All resources are of the Inferred category.

Table 14.9: Grade Tonnage Tables for Inferred Resources at Barje.
Numbers are rounded to reflect the fact that an estimate has been made and as such figures may not total. Metallurgical recoveries are incorporated into calculation of Au Equivalent. All ounces are troy ounces and tonnes are metric tonnes (dry basis).
**Cut off values selected for resource reporting.*

Cut-off grade AuEq	Tonnes	Density	AuEq g/t	AuEq oz	Au g/t	Contained Au oz	Ag g/t	Contained Ag oz
High Grade Breccia								
2	2,200,000	2.8	6.0	430,000	5.1	360,000	88	6,300,000
1.5	2,700,000	2.8	5.2	460,000	4.4	380,000	75	6,600,000
1.25	2,800,000	2.8	5.1	460,000	4.3	390,000	72	6,600,000
1	3,000,000	2.8	4.8	470,000	4.1	400,000	68	6,600,000
0.9	3,100,000	2.8	4.8	470,000	4.0	400,000	68	6,600,000
0.8	3,100,000	2.8	4.7	470,000	4.0	400,000	67	6,700,000
0.7	3,100,000	2.8	4.7	470,000	4.0	400,000	66	6,700,000
0.6*	**3,200,000**	**2.8**	**4.7**	**470,000**	**3.9**	**400,000**	**65**	**6,700,000**
0.5	3,200,000	2.8	4.6	470,000	3.9	400,000	65	6,700,000
Low Grade Schist								
2	13,000	2.8	2.1	890	1.9	830	11	4,800
1.5	470,000	2.7	1.7	26,000	1.6	24,000	9.6	140,000
1.25	1,000,000	2.7	1.5	51,000	1.4	48,000	8.9	300,000
1	1,700,000	2.7	1.4	76,000	1.3	71,000	8.6	480,000
0.9	2,000,000	2.7	1.3	86,000	1.2	79,000	8.5	560,000
0.8*	**2,400,000**	**2.7**	**1.2**	**96,000**	**1.1**	**88,000**	**8.4**	**650,000**
0.7	2,700,000	2.7	1.2	100,000	1.1	94,000	8.3	720,000
Partially Oxidized Material								
2	390,000	2.5	4.4	55,000	3.4	43,000	90	1,100,000
1.5	810,000	2.5	3.0	78,000	2.4	64,000	50	1,300,000
1.25	1,000,000	2.5	2.6	89,000	2.2	74,000	40	1,300,000
1	1,200,000	2.5	2.4	96,000	2.0	80,000	35	1,400,000
0.9	1,300,000	2.5	2.4	96,000	2.0	81,000	34	1,400,000
0.8	1,300,000	2.5	2.3	98,000	1.9	82,000	33	1,400,000
0.7	1,400,000	2.5	2.2	100,000	1.9	84,000	31	1,400,000
0.6	1,500,000	2.5	2.2	100,000	1.8	85,000	31	1,400,000
0.5*	**1,500,000**	**2.5**	**2.1**	**100,000**	**1.7**	**87,000**	**29**	**1,400,000**
0.4	1,600,000	2.5	2.0	100,000	1.7	88,000	28	1,500,000
Total								
Variable	**7,100,000**	**2.7**	**2.9**	**670,000**	**2.5**	**570,000**	**38**	**8,800,000**

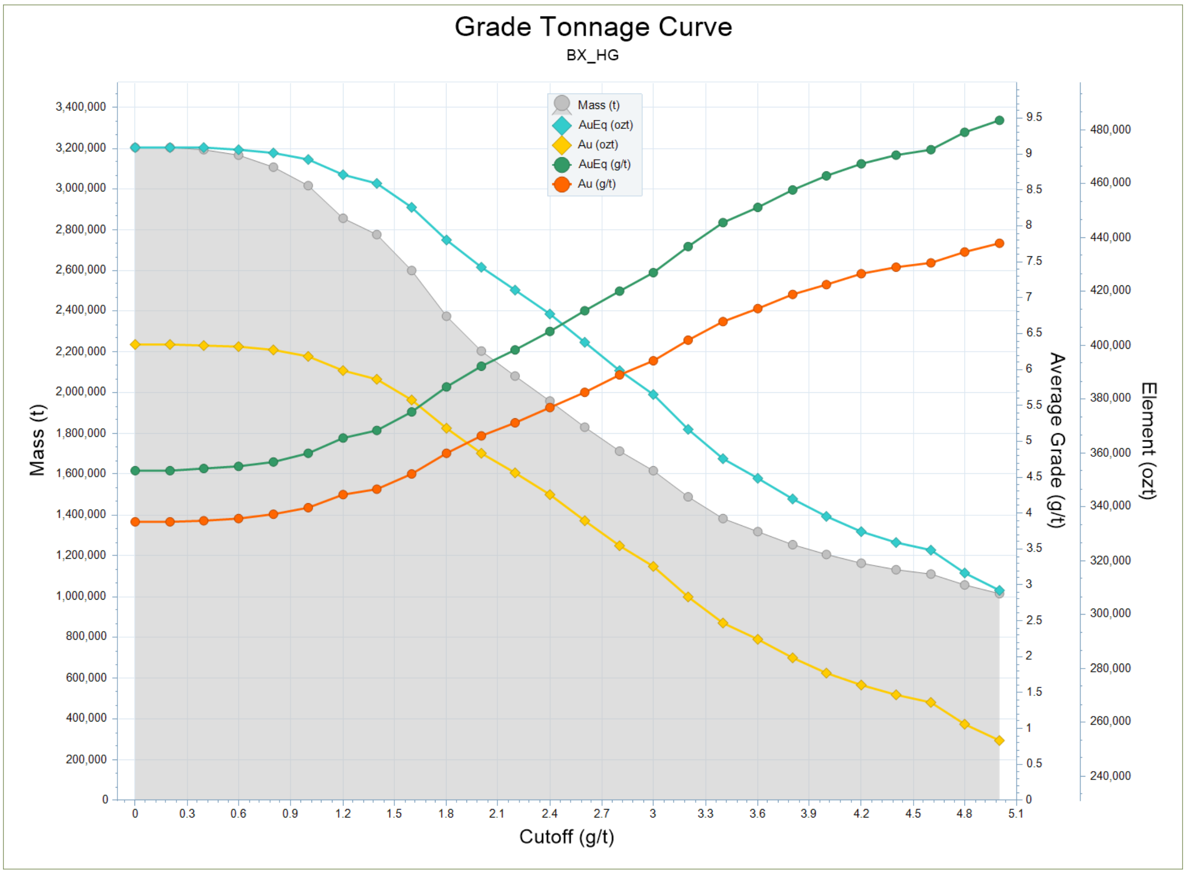


Figure 14.15: Grade tonnage curves for the Barje Inferred block model, HG_BX domain.

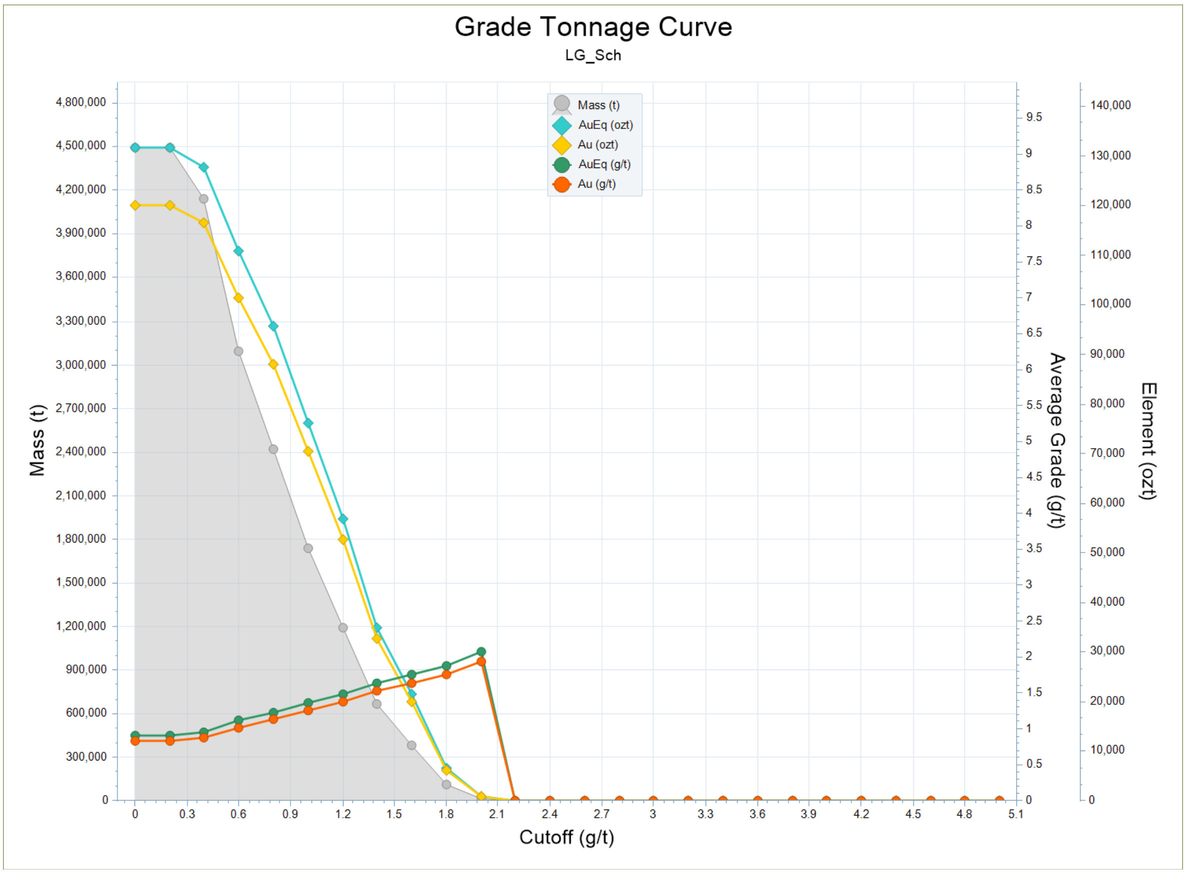


Figure 14.16: Grade tonnage curves for the Barje Inferred block model, LG_Sch domain.

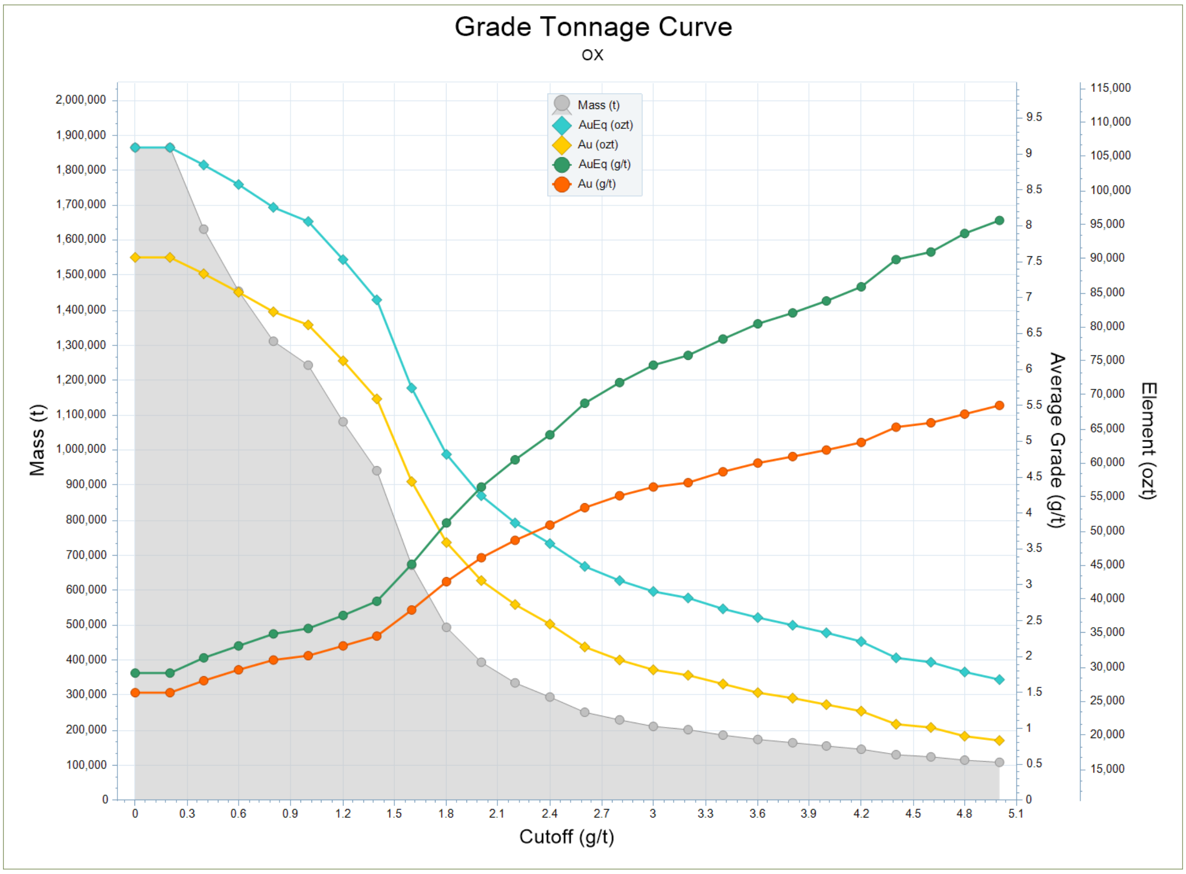


Figure 14.17: Grade tonnage curves for the Barje Inferred block model, OX domain.

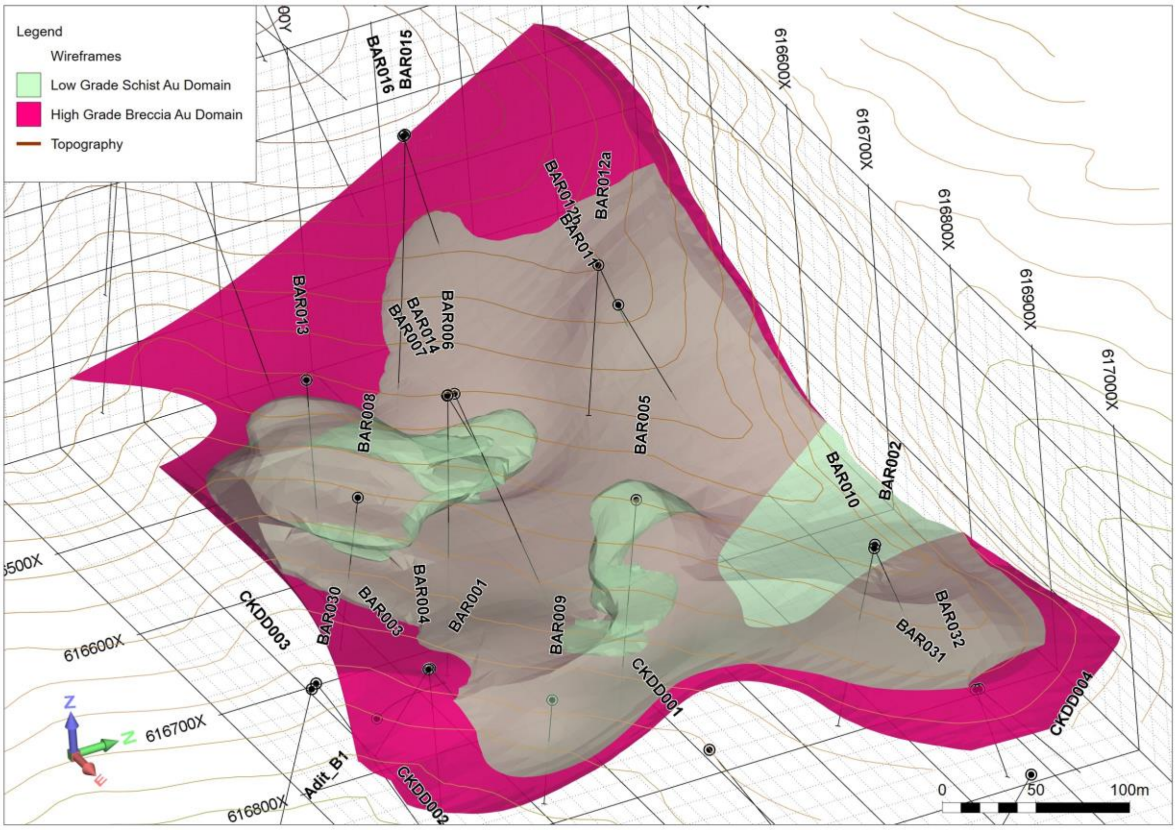


Figure 14.18: Wireframe mineralization model looking North West.

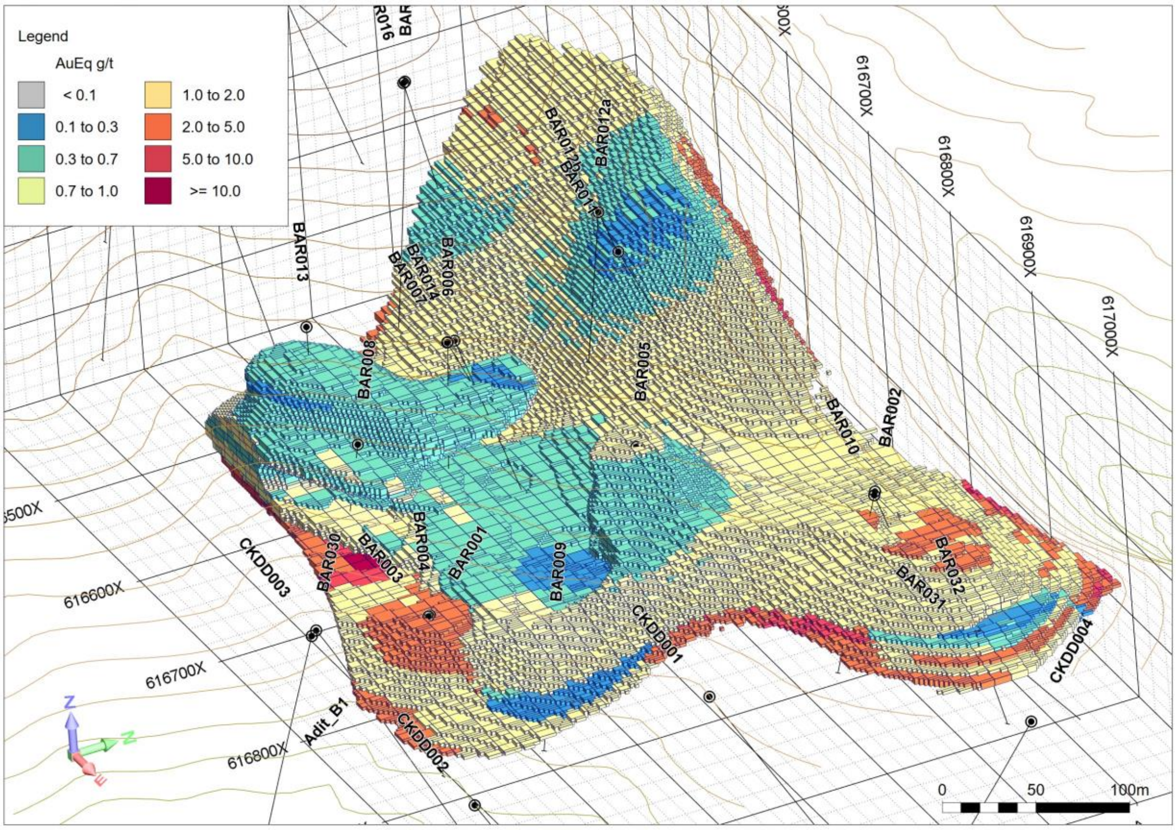


Figure 14.19: Block Model looking North West; no cut-off applied.
Blocks falling below the cut-off grade do not qualify as a Mineral Resource.

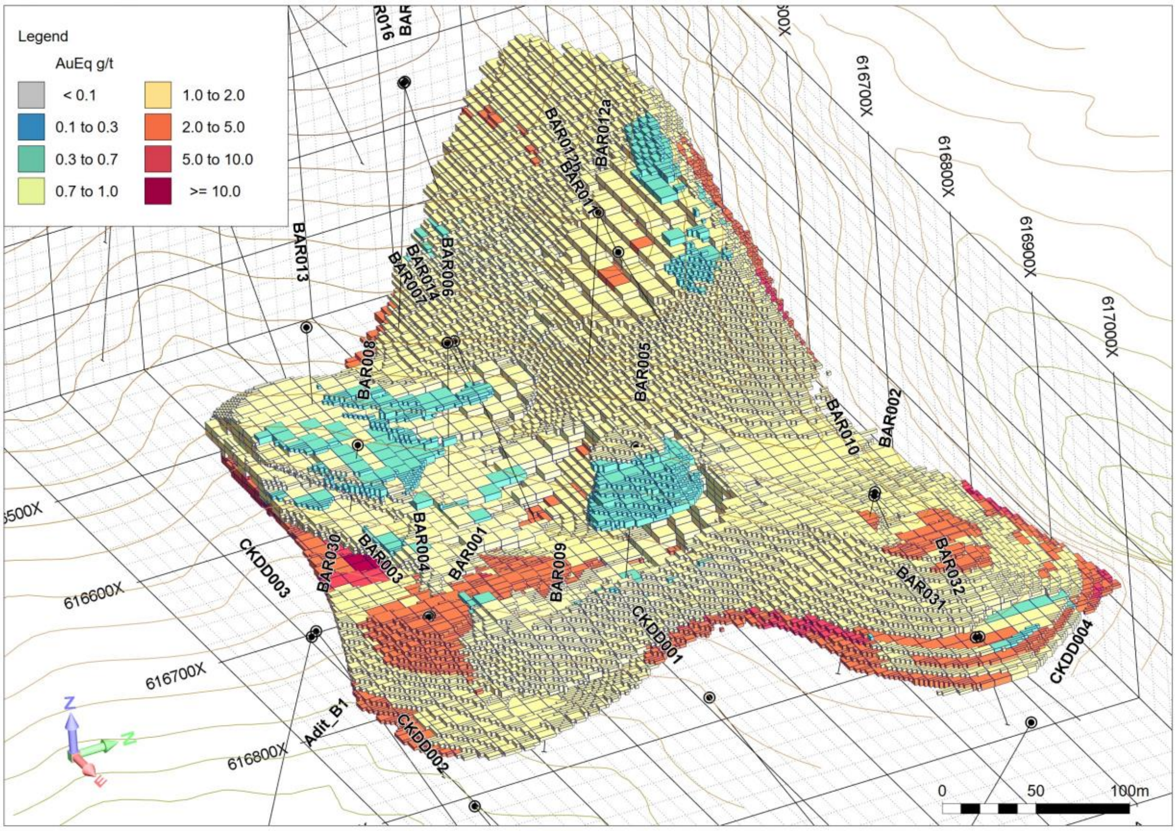


Figure 14.20: Block Model looking North West; AuEq cut-off applied by material type.
Blocks shown form the Mineral Resource.

14.14 Resource Statement

The estimated Mineral Resource, reported in accordance with NI 43-101 and the CIM Definition Standards above cut-off grades of 0.6 g/t AuEq for high grade breccia, 0.8 g/t AuEq for low grade schist, and 0.5 g/t AuEq for partially oxidized material is approximately 7.1 Mt at 2.5 g/t Au and 38 g/t Ag in the Inferred category, and containing 570,000 oz of Au and 8.8 Moz of Ag. This equates to approximately 2.9 g/t AuEq or 670,000 oz AuEq. It is the opinion of the Qualified Person that all elements included in the Au Equivalent calculation (gold and silver) have a reasonable prospect of being recovered and sold. The updated Mineral Resource estimate has an effective date of January 07, 2021 and supersedes the previous initial Mineral Resource estimate, there has been no material change to the Mineral Resource estimate in terms of tonnage, grade and contained metal.

Table 14.10: Inferred Resources for the Barje Deposit by material type.

Tonnes	Density	AuEq g/t	Contained AuEq oz	Au g/t	Contained Au oz	Ag g/t	Contained Ag oz
Total							
7,100,000	2.7	2.9	670,000	2.5	570,000	38	8,800,000
High Grade Breccia							
3,200,000	2.8	4.7	470,000	3.9	400,000	65	6,700,000
Low Grade Schist							
2,400,000	2.7	1.2	96,000	1.1	88,000	8.4	650,000
Partially Oxidized Material							
1,500,000	2.5	2.1	100,000	1.7	87,000	29	1,400,000

The Qualified Person for Resources, Mr Richard Siddle, MSc, MAIG, has reviewed the available geological and assay data, quality control data and has completed a site visit to the Barje Deposit in November 2019. Mr Siddle has completed the Mineral Resource estimate and has been an employee of Addison Mining Services since November 2014.

No estimates of Mineral Reserves have been completed. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

The Mineral Resource extends from surface to approximately 110 m below surface, it is laterally extensive over an area of approximately 600 m from east to west and approximately 350 m north to south. The thickness of resource mineralization ranges from approximately 10 to 40 m with some isolated thinner areas. It is closed by bounding faults to the north and south and by drilling to the east and west, there is some possibility of identifying additional mineralization by infill drilling in areas where the model has interpreted to pinch and data are sparse, and in the northwest corner of the deposit.

Additional drilling is required to increase the confidence in the Mineral Resources, as the level of information increased resource quantities may increase or decrease.

14.15 Comparison to Previous Estimates

The previous Mineral Resource estimate of Au and Ag was completed by AMS in Q4 2019 with an effective date of January 07, 2021 for the Barje Deposit. The Mineral Resource, as reported above a cut-off grade of 0.7g/t AuEq, was approximately 7.1 Mt at 2.5 g/t Au and 38 g/t Ag in the Inferred category, containing 570,000 oz Au and 8.6 Moz Ag equating to approximately 3 g/t AuEq for 680,000 oz AuEq. The updated Mineral Resource does not reflect a material change in terms of tonnage, grade and contained metal.

A small reduction in total AuEq ounces is seen due to the greater increase in Au price relative to silver and is within rounding error of the estimate. Adjustment to the base of partially oxidized rock model has resulted in an additional 2 Mt of resource rock assigned to the partially oxidized material largely from the low-grade schist domain. This equates to approximately 21 koz of Au and 200 koz of Ag. All changes are within the typical rounding error and uncertainty that is expected for an Inferred resource estimate.

15 Mineral Reserve Estimates

There are no Mineral Reserves for the Barje Project and Tlamino Project Licences to which this report relates. The remainder of this page is left intentionally blank.

16 Mining Methods

A summary of the mining study undertaken in support of the PEA is presented in the following subsections of the report. All costs used in the mining study are in US$.

16.1 Proposed Mining Method

The Barje Deposit is relatively thick, flat-lying and situated beneath shallow to medium-depth overburden. Initially both open pit, as well as underground methods with higher selectivity were considered, however open pit methods were chosen on account of the overall low stripping ratio and generally low RQD of the rock mass. Mining by open pit methods using hydraulic excavators and wheel loaders loading articulated dump trucks (ATD) for haulage of both waste and potentially economic material is therefore proposed. Mining activities at Barje will include free-digging of the weathered zones plus drilling and blasting of fresh rock, with loading, hauling and dumping, plus mining support activities. Pre-mining, removal and stockpiling of topsoil is assumed.

16.2 Parameters Relevant to Mine Design

16.2.1 Geotechnical Characterization

A preliminary geotechnical characterization was undertaken, including logging of resource drill holes geotechnically for RQD, plus visual observation of cores for validation. RQD is generally low to moderate, with values ranging from 20% – 70%, although higher RQD was determined for the calcareous schists. The proposed open pit was split into a number of sectors for optimisation by the Geotechnical Consultant based on the dip direction of the various pit walls (Middindi Consulting, 2020). The slope geometry is shown in Figure 16.1 and the parameters are summarised in Table 16.1.

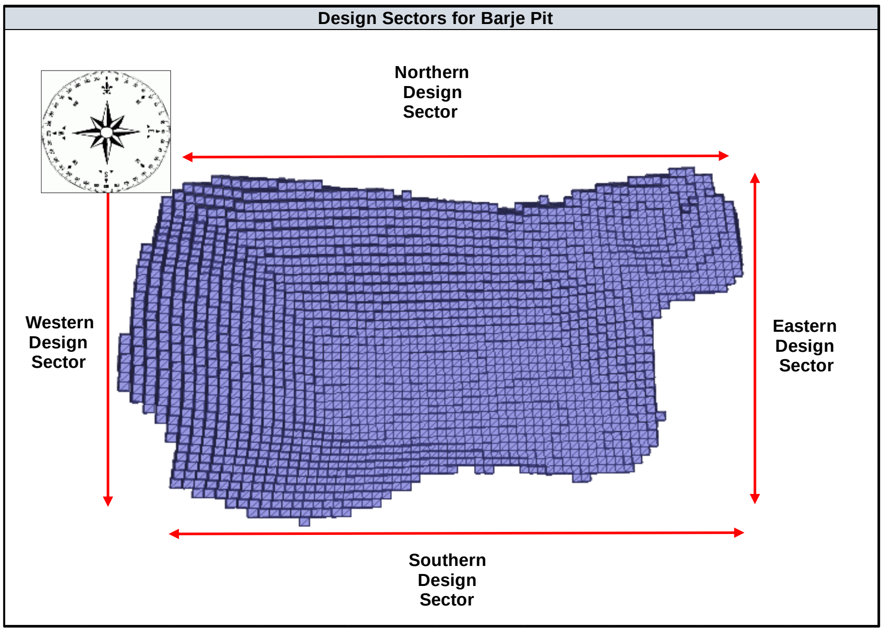


Figure 16.1: Geotechnical Design Sectors for the Barje Open Pit.

Table 16.1: Recommended Geotechnical Parameters (Middindi, 2020)
1. Overall Slope Angles (OSA) include catch benches and are based on the maximum face height.

Pit Wall	Zone Number	Dip Direction	Max Slope Height (m)	Max Stack Height (m)	Bench Batter Angle (°)	Safety Berm Width (m)	Catch Berm Width (m)	Overall Slope Angle[1] (°)
North	1	180	100	50	63	6.5	13	39
East	2	270	40	-	50	6.5	0	41
South	3	0	110	60	67	6.5	13	38
West	4	90	170	60	68	6.5	13	37

16.2.2 Hydrogeology and Pit Dewatering

There has been no testing or modelling of pit hydrogeology at this time, however it is assumed that the slopes are well drained due to the topography and the fractured nature of the rock mass. Further work is required to confirm this.

Pit inflows will likely be dominated by precipitation at the Barje site, as well as groundwater seepage from the relatively fractured surrounding rock mass and geological structures.

The pit water management systems will include perimeter dikes and diversion ditches, in-pit water collection ditches, and in-pit pumps and collection systems to transfer water from the open pits to discharge points, for settling, and potentially treatment prior to discharge.

16.3 Pit Optimisation

16.3.1 Optimisation Process

Pit optimisation was undertaken using Datamine Studio NPVS ("NPVS") and SimSched DBS ("DBS") software packages.

This two-stage optimisation process was adopted to establish the maximum economic pit limits and the optimum mining sequence within said limits, as it was recognised that the standard Lerch Grossman optimisation methodology of determining the pit limit and mining sequence based on pit shells cannot adequately account for blending constraints or stockpiling. These constraints were recognized as important for this project due to the significant difference in processing recovery and concentrate grade between the HG_BX and the LG_Sch.

Ideally, the optimised mine sequence should focus on HG_BX units in the early periods and delay mining of the LG_Sch for as long as possible. This was not entirely feasible as LG_Sch tends to overlay HG_BX and the Lerch Grossman optimisation will be driven by block value and waste stripping ratio, rather than blending requirements. However, as the LG_Sch is higher in the geological sequence it will naturally have a lower stripping ratio and tend to be mined first even though it has a lower grade than the HG_BX. Subsequent application of a blending constraint in NPVS to the Lerch Grossman mining sequence reduces the project value due to the need to stockpile increasing amounts of LG_Sch.

However, by applying a blending constraint to LG_Sch in the optimisation stage of the second DBS it was possible to search for alternative mining sequences that maximise value whilst stockpiling the Schist. This led to a higher overall project value compared to the standard Lerch Grossman solution.

The output from DBS is a series of period surfaces (annual in this case) that represent the optimal mining sequence. These surfaces were used to manually develop the pit stages, which were then imported back into NPVS to produce a hybrid solution that includes the optimised DBS mining sequence as user defined pushbacks and the optimised NPVS pit limit. This pushback sequence was then re-scheduled in NPVS and the mineral inventory reported.

16.3.2 Mineral Resource Model

The Mineral Resource block model was prepared in Micromine software for pit optimisation though addition of country rock waste blocks to extend beyond the limits of the pit optimisation, the dominant rock type was written to the block model and the mean bulk density values, as estimated from exploration drilling, were applied to each rock type. The bulk density values for the mineral domains were preserved. The block model was then regularized to 5 x 5 x 2.5 m (E x W x Z) before being exported from Micromine (*.dat) to Datamine (*.dm) format.

The block size was selected to represent the minimum Selective Mining Unit ("SMU") with the chosen mining equipment. The block size in the Z direction is particularly important for Barje as the mining method needs to allow for mining to the contacts between the LG_Sch and the HG_BX, as well as between the HG_BX and waste on the footwall contact.

The Block Model dimensions for the Regularised (Planning) model are listed in Table 16.2.

Table 16.2: Block Model Dimensions for the Planning Model

Parameter	X Direction (E)	Y Direction (N)	Z Direction (RL)
Model Origin	616,147.5	4,691,597.5	1008.75
Block Size, m	5.0	5.0	2.5
Number of Blocks	211	141	101

The fields that were included in the Resource model are listed in Table 16.3.

Table 16.3: Block Attribute Fields in the Planning Model

Field	Units	Description
Au_ppm	ppm	Gold grade
Ag_ppm	ppm	Silver grade
Bulk_Den	t/m^3	Bulk Density
BF_DTM	Factor	Proportion of block below topography
BF_MIN	Factor	Proportion of block that is mineralised
BF_HG	Factor	Proportion of mineralised block that is HG_BX
BF_LG	Factor	Proportion of mineralised block that is LG_Sch
MINDOM	Text	Dominant mineral material type
REDOX	Text	Oxidation state
ZONE	Number	Geotech Zone number (1 to 4)
ORE	Text	Rock type code
ROCK	Text	Rock type code
NSR	US$	Net Smelter return

The main products in the concentrate derived from the HG_BX and the LG_SCH are Au and Ag. While the Resource model also includes grades for a number of other elements, such as Cu, Pb, Zn and Fe, these elements have not been assigned any economic value in the pit optimisation. Similarly, the associated concentrate penalties for the deleterious elements (e.g., As) have not been explicitly modelled in determining payability for Au and Ag.

Because the Resource model has been regularised to a constant block size it was necessary to record the percentage of the block that is below topography ("BF_DTM").

It was also necessary to keep track of the percentages of Schist ("BF_LG") and Breccia ("BF_HG") in a block as this will determine the classification of dominant material type ("MINDOM"), which controls the process route and the calculation of process recovery and Payability.

To help track the quantity of mixed material present two additional fields were inserted in the Planning model (MAT and ROCK) which are based on MINDOM, REDOX and BF_LG as shown in Table 16.4.

Table 16.4: Rock Codes used in the Planning Model.

Mineral Type	MINDOM	REDOX	MAT	% BF_LG	ROCK
Low Grade Schist	LG_Sch	Ox	LG_SchOx	< 20 > 80	LGOx
				≥ 20 ≤ 80	LGOxMx
		Fr	LG_Sch	< 20 > 80	LG_Sch
				≥ 20 ≤ 80	LG_SchMx
High Grade Breccia	HG_BX	Ox	HG_BXOx	< 20 > 80	HGOx
				≥ 20 ≤ 80	HGOxMx
		Fr	HG_BX	< 20 > 80	HG_BX
				≥ 20 ≤ 80	HG_BXMx

This means that if the block contains between 20% and 80% Schist it is classed as mixed material and the recovered metal and payable metal are calculated from the weighted average based on the ratio of HG_BX to LG_Sch.

Additional fields for Geotech Zone ("Zone") and Net Smelter Return ("NSR") were also added to the Planning model for the purposes of pit optimisation.

16.3.3 Topographic Data

The most recent fully validated topographic data ("DTM") was used during construction of the block model. The DTM was generated from a purchased data set of contours for the Serbian 1:25000 topographic maps.

16.3.4 Optimisation Constraints

The pit optimisation in NPVS was unconstrained by mining boundaries or other physical boundaries. To generate the pit shells the mining rate was set to 600 Ktpa and a discount rate of 8% was applied.

The pit optimisation in DBS also assumed no physical boundaries, a mining rate of 600 Ktpa and a discount rate of 8%. In addition, a constraint was set on the blending to limit the percentage of the lower-grade LG_Sch to HG_BX in the first 4 years to less than 25%.

This allows DBS to search for a mining sequence that attempts to maximise project value in terms of discounted cashflow ("DCF") as well as satisfying the objective of delaying the processing of the lower grade material. This was important in the case where it was not possible to process a blended feed of HG_BX and LG_Sch, and where batch processing was instead required.

16.3.5 Optimisation Parameters

The optimisation parameters typically consist of:

- Overall Slope Angles by Sector (Azimuth range)
- Mining Cost and Processing costs
- Mining Modifying Factors (Mineralized Material Loss and Waste Dilution)
- Metallurgical factors (Process Recovery and Payability)
- Concentrate Transport costs
- Other Financial Parameters
- Other Constraints

16.3.5.1 Mining and Processing costs

The mining cost was estimated at 2.3 US$/t mined based on benchmarking against other contract mining operations in the Balkans region and cost data obtained by Medgold. It was assumed that a significant proportion of the deposit can be mined with no or minimal blasting due to the low rock strength. If required, the fresh material will be ripped or lightly blasted with a low powder factor to increase productivity.

The processing cost was assumed to be between 10.0 US$/t and 14.5 US$/t processed. Initial optimisation studies in DBS used the higher value of 14.5 US$/t and it was subsequently reduced to 10.0 US$/t during the PEA study.

16.3.5.2 Pit Slopes

The Overall Slope Angles ("OSA") are shown in Table 16.5 have been adapted from the values shown in Table 16.5 to take account of the orientation of the pit walls, the variable height of the various pit walls, the position of the ramp systems, and the inclusion of catch benches to limit the stack height.

Table 16.5: Overall Slope Angles used for Pit optimisation.
Note: The OSA have been increased by approximately 3 degrees compared to Table 16.1

Pit Wall	Zone Number	Azimuth From	Azimuth To	Overall Slope Angle (°)
North	1	315	45	42
East	2	45	135	44
South	3	135	225	42
West	4	225	315	40

16.3.5.3 Modifying Factors

The modifying factors for mining recovery and waste dilution were assumed to be accounted for through regularization of the resource block model to a standard block size that represents the selective mining unit. No further factors have been applied.

The selected block size was 5 m x 5 m x 2.5 m, and was shown to give similar tonnage and grade factors to a 5 m x 5 m x5 m model (Table 16.6). The tonnage and grade factors when compared to the un-regularised model are of the order of +5% to +6% in tonnage and -6% to -9% for grade. These factors are considered reasonable for a flat lying deposit with a height of the mineralised section of around 20 m to 30 m.

Table 16.6: Tonnage and Grade Factors

	Sub Block Model			**5 x 5 x 2.5 Model**			**5 x 5 x 5 Model**		
Au Cut-off	Inventory	Au	Ag	Tonnage Change	Au Grade Change	Ag Grade Change	Tonnage Change	Au Grade Change	Ag Grade Change
(g/t)	(tonnes)	(g/t)	(g/t)						
0.30	1,360,000	0.41	4.15	2.1%	-1.8%	-2.3%	2.4%	-3.2%	2.7%
0.50	1,030,000	0.59	6.04	17.7%	0.5%	0.4%	20.8%	2.0%	-0.8%
0.70	895,000	0.79	7.22	11.1%	0.3%	0.7%	10.7%	1.8%	-1.8%
0.90	868,000	1.01	7.24	10.6%	-0.6%	-0.8%	13.1%	5.9%	12.8%
1.10	5,030,000	3.13	49.66	1.7%	-5.1%	-8.0%	2.0%	-4.3%	-6.7%
Total	9,180,000	2.02	29.89	5.3%	-6.5%	-9.0%	6.1%	-6.3%	-8.6%

The smaller block size in the Z direction was chosen to maximise the resource recovery when mining to the contacts between the low and high-grade zones and between the mineralised zones and waste. This supports an expectation that during operations, the benches will generally be split into 5 m flitches with further subdivision to 2.5 m when mining at the contacts.

Close grade control at these contacts, aided by the noted colour changes between the various lithologies, will be required. With precision digging and careful dozing it is expected that mining to the required level of precision will be possible.

The variation in tonnage and grade factors with cut-off grade shows that there are two distinct populations (high and low grade) and that the effects of ore loss and waste dilution are most evident in grade range 0.5 to 0.9 g/t Au. This is mainly since the Schists overlay the Breccias and the bulk of dilution tends to take place between the two mineralised zones. This suggests that modifying factors will have limited impact on estimates of the tonnage of high-grade material but can still have a significant impact on expected volumes of lower grade material, which is likely to have a cut-off grade of nearer to 0.9 g/t Au.

16.3.5.4 Process Recovery Factors

The metallurgical factors are specified by dominant ROCK code (HG_BX or LG_Sch) and are calculated as a weighted average where there is a mix of materials (e.g., HG_BXMx and LG_SchMx) in a block.

Table 16.7: Metallurgical Factors

ROCK	Units	Au Recovery	Ag Recovery
HG_BX	%	85.8	84.3
LG_Sch	%	76.5	82.7

It was assumed that the oxidised material (HGOx, HGOxMx, LGOx and LGOxMx) would be stockpiled for the future. No economic value was therefore assigned to this material in the pit optimisation.

16.3.5.5 Payability Factors

Payability factors were assigned on the assumption that there will be two separate concentrate streams, one from processing HG_BX (and mixed material) and one from processing LG_Sch (and mixed material). The payability factors take into account all downstream costs, including refining; factors for the dominant rock types are shown in Table 16.8.

Table 16.8: Payability Factors

ROCK	Units	Au Concentrate	Ag Concentrate
HG_BX/HG_BXMx	%	75.0	75.0
LG_Sch/LG_SchMx	%	60.0	60.0

16.3.5.6 Transport Costs

An allowance of 2.0 US$/t Run of Mine (ROM) was allowed for concentrate transport costs to the port.

16.3.5.7 Other Financial Parameters

Other financial parameters included the long-term metal price forecast for Au and Ag (sections 14.12, 19.2), General and Administration (G&A) costs, and Royalties.

Table 16.9: Metal prices, G&A and Royalty used in pit optimisation.

Parameters	Units	Au Concentrate	Ag Concentrate
Metal Price	US$/oz(troy)	1,500	16.5
G&A	US$/t ROM	4.0	4.0
Royalty	%	5.0	5.0

16.3.5.8 Other Constraints

The pit optimisation was constrained by a maximum plant throughput of 600 Ktpa and a blending constraint on LG_Sch was applied in DBS. No constraints were applied to maximum mining rate.

16.3.6 Optimisation Results

16.3.6.1 Direct Block Scheduler (DBS)

The pit optimisation was run in DBS by first processing the block model to calculate the NSR using the optimisation parameters listed in Section 16.3.5. The OSA was also included in the input file for DBS.

The DBS schedule optimisation was constrained by the ratio of Schist (LGFr and LGFrMx) to Breccia (HGFr and HGFrMx) to limit the LG_Sch processed in the first 4 years to less than 25%. This was achieved by stockpiling up to 1 Mt of low-grade material over the active mining life of 6 years, with two years of subsequent stockpile reclaim. The resulting mining volumes, with smoothed plant feed, are shown in Table 16.2.

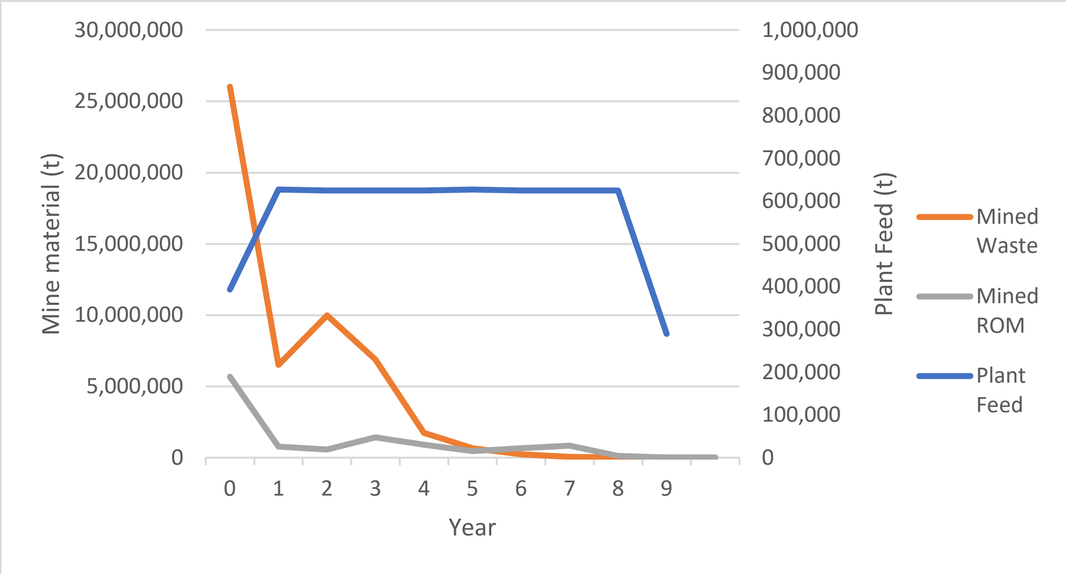


Figure 16.2: ROM production tonnes by period.

Mined grades for HG_BX, LG_Sch, and blended plant feed, for both Au and Ag is shown in Table 16.3.

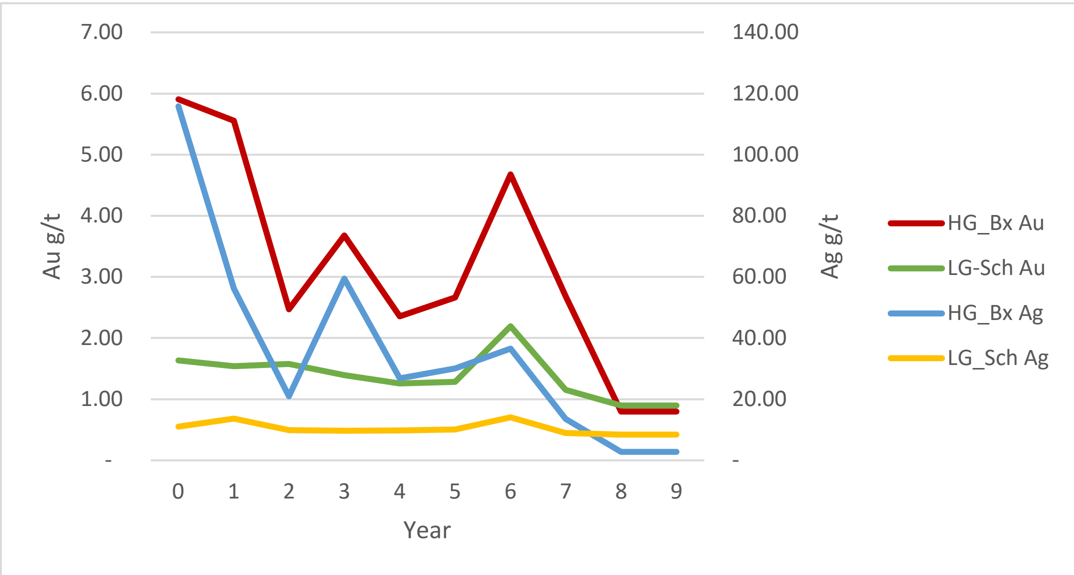


Figure 16.3: Processed grade by period.

The mining sequence is shown in Figure 16.4 for all mined blocks, with the colour coding by period mined. Mining will commence in two distinct areas. Firstly, to the east where the waste stripping ratio is relatively low, and it is possible to access the HG_BX material close to surface. However, in this area the deposit pinches out to the east and does not extend in depth more than 50 m below surface.

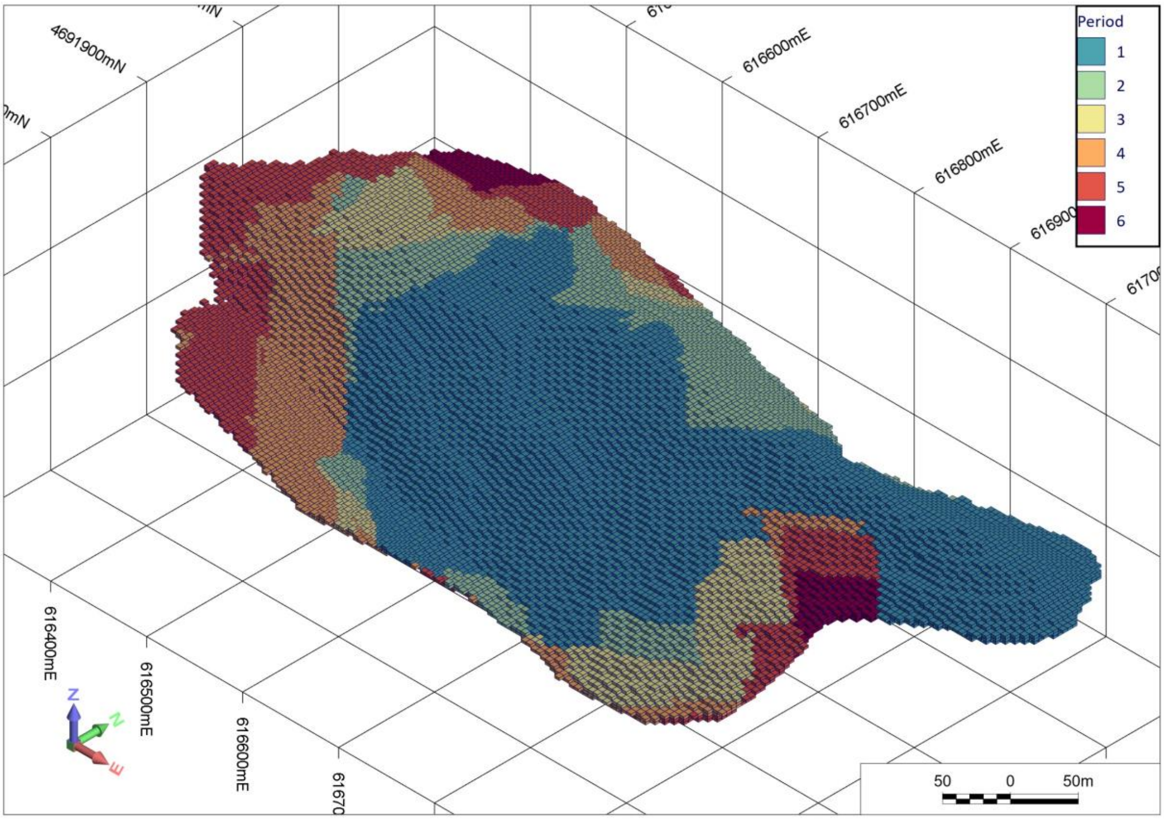


Figure 16.4: Block Mining Sequence generated by DBS.

The next main area of mining will be in the central portion where the topography again helps as there is a low stripping ratio and high-grade ROM can be accessed quickly by expanding the pit from the south. This logically means that the pit exit will be to the south and the plant should be located to the south east of the pit. Low grade material can also be stockpiled near this pit exit in the valley that runs to the south east (Figure 16.5).

Waste will be transported to the north of the pit to fill the valley to the north. The waste haul for most of the benches will be relatively short and can take advantage of the fact that haul routes can be established that follow the contours of the hill. This greatly simplifies the ramp systems as there is limited need to establish a permanent ramp system for either plant feed or waste, other than when the pit is well developed, and mining is below the pit rim elevation at the southern pit exit.

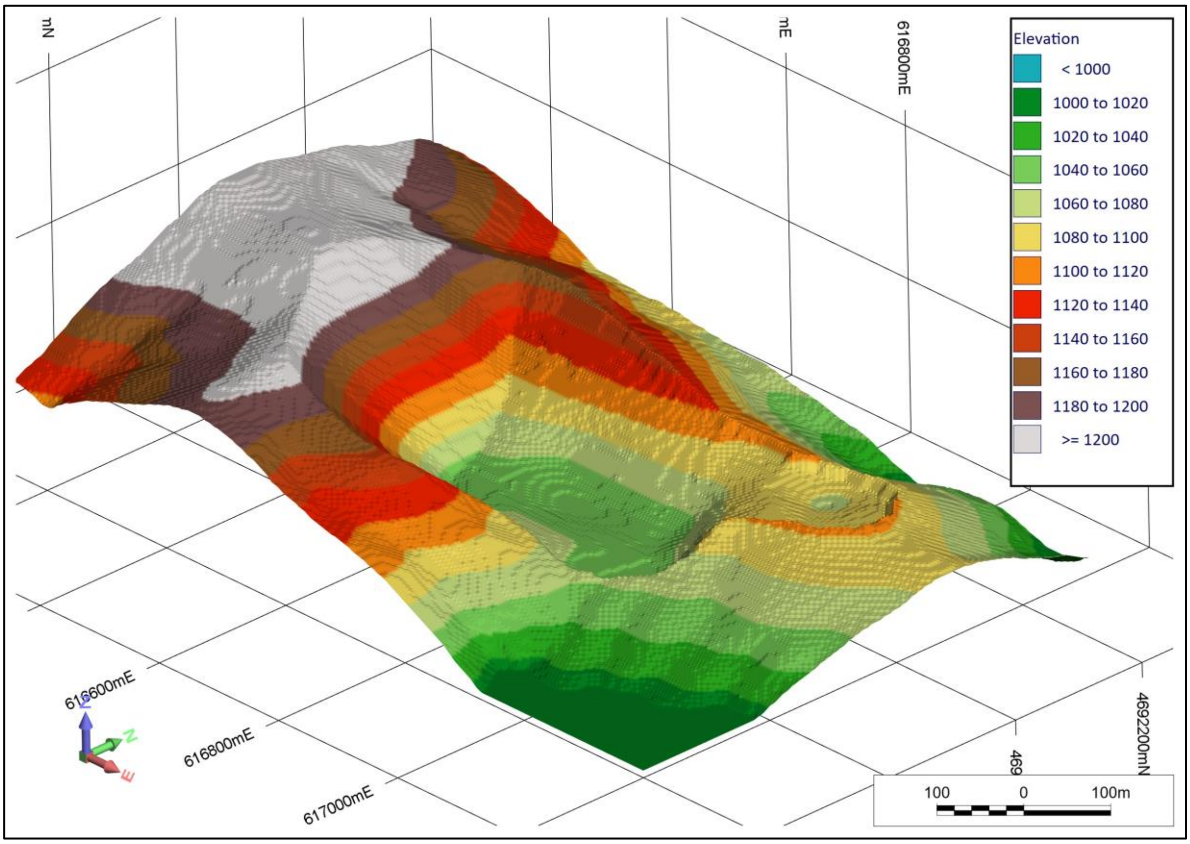


Figure 16.5: DBS Optimised Pit Limit

16.3.6.2 NPV Scheduler

The pit optimisation was run in Datamine's NPV Scheduler as a check on DBS. In this case the optimisation was run with a range of input parameters in order to test the sensitivity of the pit limit and mineral inventory.

- Metal Price (± 10% and ± 20%)
- Process Cost (± 10% and 20%)
- Mining Cost (± 10% and 20%)
- LG Payability (40%, 50% and 60%)
- Pit Slope angle (± 3°)

The sensitivity analysis (Table 16.10) demonstrates that the pit size (i.e., total rock) or mineral inventory is not particularly sensitive to Price, Process Cost, Mining Cost or OSA, with the total rock mined ranging from 30 Mt to 33 Mt and total mineral inventory ranging from 5.7 Mt to 6.2 Mt.

The key driver on pit size and mineral inventory appears to be the estimate of payability (40% to 60%) with a range in total rock mined between 25 Mt to 33 Mt and a mineral inventory between 3.9 Mt to 6.2 Mt. This is not entirely surprising as changing the payability factor from 60% to 40% would result in a 33% change in revenue and a similar effect would be seen by changing either price or process recovery by this amount.

Table 16.10: Sensitivity Analysis

Scenario	Rock	Waste	High Grade Breccia			Low Grade Schist		
	(Mt)	(Mt)	(Mt)	(Au g/t)	(Ag g/t)	(Mt)	(Au g/t)	(Ag g/t)
Base Case	31.7	25.6	3.38	3.42	53.9	2.53	1.17	9.5
Au Price – 10%	30.2	24.3	3.45	3.50	55.5	2.51	1.17	9.6
Au Price – 20%	28.6	22.9	3.28	3.61	57.7	2.46	1.17	9.6
Process Cost + 10%	31.4	25.7	3.52	3.46	54.7	2.20	1.23	9.8
Process Cost + 20%	31.0	25.6	3.46	3.50	55.5	1.95	1.28	10.1
Mining Cost + 10%	30.9	24.8	3.54	3.44	54.4	2.52	1.17	9.5
Mining Cost + 20%	30.2	24.2	3.48	3.47	55.1	2.52	1.17	9.5
LG Payability 40%	29.5	24.2	3.35	3.56	56.8	1.94	1.28	10.1
LG Payability 50%	24.9	21.0	2.95	3.83	62.6	0.95	1.52	12.6
Pit Slopes + 3°	31.0	24.8	3.65	3.38	53.2	2.53	1.17	9.5
Pit Slopes - 3°	33.1	27.0	3.57	3.42	54.1	2.53	1.17	9.5

The detailed results for the Base Case are shown in Table 16.11. At a Price Factor of 1.0 the total mineral inventory is 6.1 Mt @ 2.49 g/t Au and 35.6 g/t Ag. Of this 3.4 Mt @ 3.53 g/t Au and 65.1 g/t Ag is high grade Breccia and 2.5 Mt @ 1.17 g/t and 9.6 g/t Ag is lower grade Schist. The overall waste to ROM strip ratio is 4.6:1.

Table 16.11: NPVS Pit Optimisation Results

Price	Rock	HG_BX			LG_Sch			Strip Ratio	Cum Rock	Cum ROM	DCF Increment
Factor	tonnes	tonnes	Au g/t	Ag g/t	tonnes	Au g/t	Ag g/t	(t/t)	kt	kt	US/t
8%	1,010,000	317,000	6.75	256.1	103,000	1.71	10	1.4	1,010	420	194
10%	73,900	23,500	6.09	226.1	11,400	1.63	9.81	1.1	1,080	455	155
12%	44,300	16,000	4.64	179.3	3,690	2.01	24.7	1.2	1,120	475	138
14%	49,400	14,700	4.17	150.3	9,400	1.61	9.91	1.1	1,170	499	93
16%	24,000	8,720	3.68	98.4	2,500	1.67	13.76	1.1	1,200	510	92
18%	20,900	5,010	3.6	140.9	1,850	1.59	9.38	2	1,220	517	91
20%	34,100	12,000	2.52	100.7	2,170	1.91	20.84	1.4	1,250	531	71
22%	157,000	24,600	2.61	196.3	11,700	1.57	32.45	3.3	1,410	567	80
24%	4,040,000	257,000	6.5	98.7	273,000	0.94	15.62	6.6	5,450	1,100	77
26%	1,390,000	162,000	4.09	65.9	84,900	0.98	13.07	4.6	6,840	1,340	62
28%	5,690,000	538,000	4.76	35.4	486,000	1.22	10.71	4.5	12,500	2,370	49
30%	753,000	145,000	2.83	36.7	63,000	1.16	9.76	2.6	13,300	2,580	36
32%	511,000	94,700	2.68	35.2	43,500	1.14	8.45	2.7	13,800	2,720	32
34%	530,000	79,100	2.87	27.5	39,900	1.15	9.97	3.5	14,300	2,830	31
36%	410,000	80,900	2.42	22.8	24,800	1.16	11.86	2.9	14,700	2,940	28
38%	485,000	75,100	2.65	20.3	45,300	1.14	8.59	3	15,200	3,060	25
40%	616,000	101,000	2.41	18.9	61,900	1.16	9.21	2.8	15,800	3,220	22
42%	693,000	76,300	2.87	17.9	67,300	1.15	7.06	3.8	16,500	3,370	21
44%	2,180,000	188,000	3.18	16.3	167,000	1.24	7.21	5.1	18,700	3,720	22
46%	899,000	98,600	2.62	15.8	71,900	1.15	7.21	4.3	19,600	3,890	18
48%	571,000	51,900	2.76	18.6	43,300	1.22	7.4	5	20,200	3,990	18
50%	184,000	36,200	1.8	12.4	23,900	0.95	8.4	2.1	20,400	4,050	12
52%	565,000	53,500	2.51	14.5	39,100	1.13	7.37	5.1	20,900	4,140	15
54%	690,000	62,700	2.43	16.7	38,100	1.16	6.4	5.8	21,600	4,240	15
56%	677,000	111,000	1.72	9.9	83,900	1.05	8.01	2.5	22,300	4,440	9

Price	Rock	HG_BX			LG_Sch			Strip Ratio	Cum Rock	Cum ROM	DCF Increment
Factor	tonnes	tonnes	Au g/t	Ag g/t	tonnes	Au g/t	Ag g/t	(t/t)	kt	kt	US/t
62%	734,000	97,600	1.7	13.1	40,000	1.11	7.24	4.3	23,800	4,730	9
64%	537,000	70,300	1.6	13	48,600	1.24	7.1	3.5	24,300	4,850	8
66%	547,000	51,900	1.87	13.6	51,700	1.11	7.19	4.3	24,800	4,950	7
68%	273,000	37,600	1.42	11.4	39,200	1.12	7.45	2.6	25,100	5,030	6
70%	89,700	15,200	1.25	8.4	29,200	1.02	7.12	1	25,200	5,070	4
72%	1,070,000	80,400	1.98	16.7	36,600	1.16	6.91	8.1	26,300	5,190	7
74%	134,000	22,600	1.12	7.3	57,300	1.02	6.54	0.7	26,400	5,270	3
76%	1,380,000	94,800	1.68	14.5	253,000	1.17	7.24	3	27,800	5,620	4
78%	799,000	68,100	1.66	14.9	31,000	1.15	6.68	7.1	28,600	5,720	5
80%	46,900	11,100	0.81	8.1	14,800	1.15	6.78	0.8	8,630	5,740	3
82%	427,000	38,900	1.45	13.4	24,900	1.21	7.52	5.7	29,100	5,810	4
84%	507,000	48,100	1.47	13.3	10,100	1.09	6.85	7.7	29,600	5,860	3
86%	397,000	37,000	1.45	13.3	9,550	1.12	7.53	7.5	9,970	5,910	3
88%	17,800	5,390	0.78	7.1	3,670	0.99	6.07	1	30,000	5,920	1
90%	248,000	35,800	1.11	10.7	3,010	0.94	6.94	5.4	30,200	5,960	2
92%	457,000	41,700	1.39	12.8	4,010	1.04	7.25	9	30,700	6,000	2
94%	41,500	8,940	0.84	7.6	3,500	0.98	6.44	2.3	30,700	6,020	1
96%	705,000	51,300	1.55	13.5	5,350	1.03	7.24	11.4	31,400	6,070	1
98%	152,000	22,200	0.99	9.8	3,010	0.96	7.36	5	31,600	6,100	0
100%	102,000	10,000	1.23	10.4	1,670	0.87	5.54	7.7	31,700	6,110	0
Total	31,700,000	3,580,000	3.42	53.9	2,530,000	1.17	9.54	4.2			

As can be seen in Table 16.11 the incremental value in terms of DCF rapidly reduces beyond a Price Factor of 60%, at which point, the total mineral inventory is 4.6 Mt. This pit shell closely equates to the pit limit obtained with DBS; it is further evident from Table 16.11 that there are a number of breakpoints (e.g., Pit 7, 11 or 22) at which the pit could be split into stages.

However, although the overall pit limit generated by NPVS is valid, the mining sequence generated by the LG shells is not necessarily optimal when blending constraints on processing the LG_Sch and LGFrMx are taken into account. In this case it can be shown that the DBS mining sequence is better able to limit the amount of LG_Sch mined in the early periods when compared to the solution produced with NPVS only (Figure 16.6).

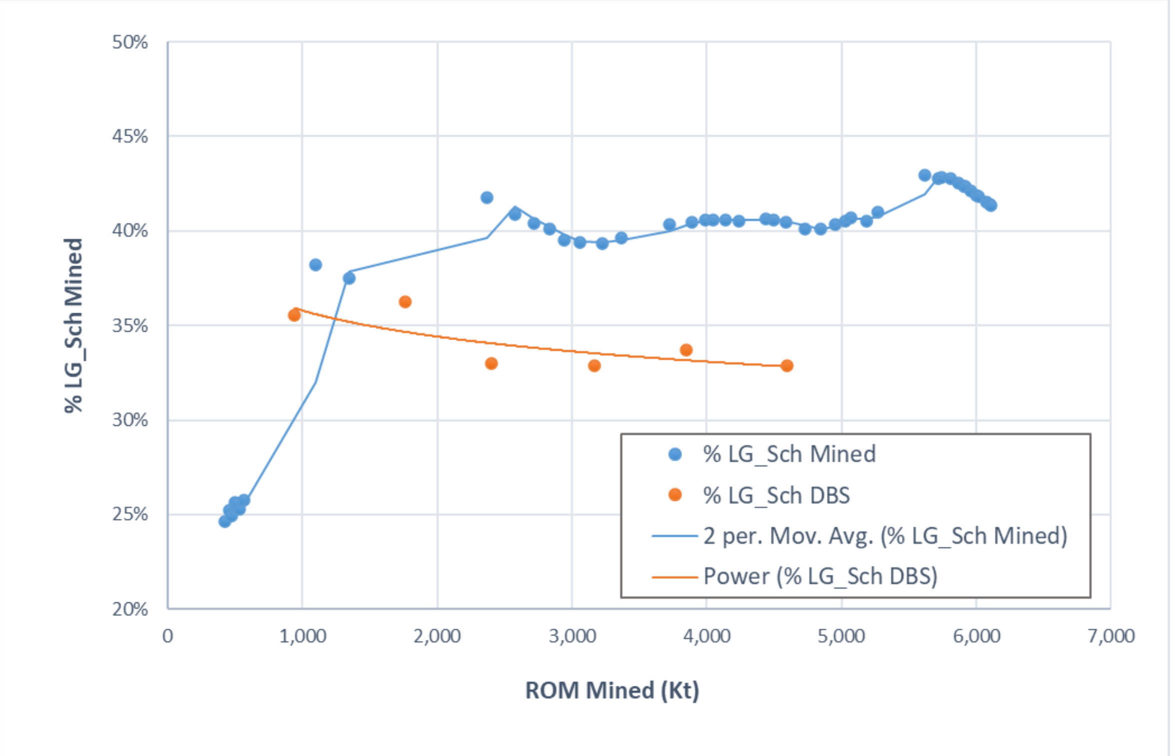


Figure 16.6: Comparison of Percentage of LG Schist Mined with Total tonnes of ROM Mined (NPVS and DBS)

Although the target of 25% of plant feed cannot be met without stockpiling at least some of the LG_Sch, it is evident in the NPVS schedule that a large amount of LG_Sch will need to be stockpiled, and this is significantly less with the DBS-developed mining sequence. It is for this reason that the DBS mining sequence for the first 4.6 Mt was chosen to represent the initial mining sequence, whilst the NPVS ultimate pit limit (Price Factor = 1.0) was selected to maximise resource recovery and extend mine life as much as possible (whilst still meeting economic criteria on profitability).

16.4 Pit Design

Using the annual surfaces generated by DBS a series of pit expansions (Pit Stages or Pushbacks) were created to follow the general DBS mining sequence and account for the minimum mining width (35m) and other practicalities of mine planning.

The pit limit was selected from the NPVS analysis (Price Factor = 1.0) and was divided up into 4 Stages with at least 6 months of plant feed in each stage. This ensured that the vertical advance rate in each stage could be kept to below 90 m per year, which is regarded as feasible with the selected mining equipment. The general layout of the stages is shown in Figure 16.7 to Figure 16.10

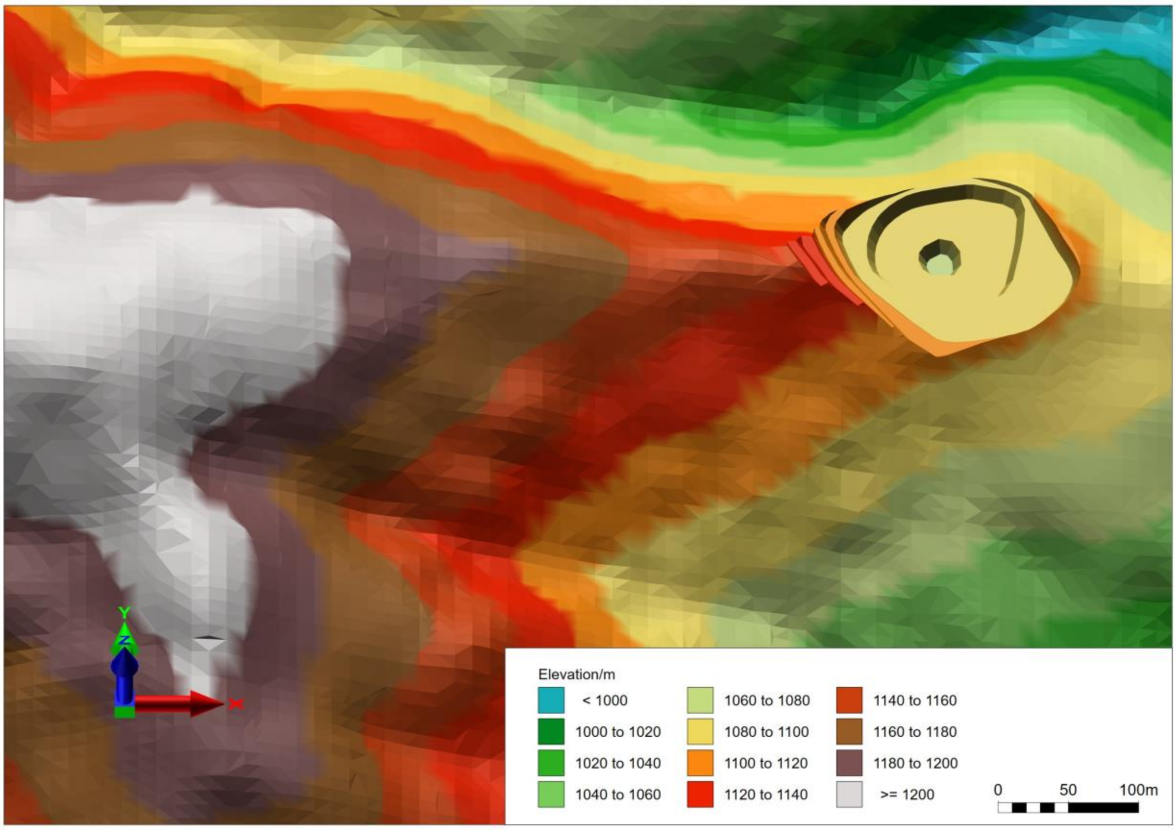


Figure 16.7: Development of Pit Stage Designs Pushback 1

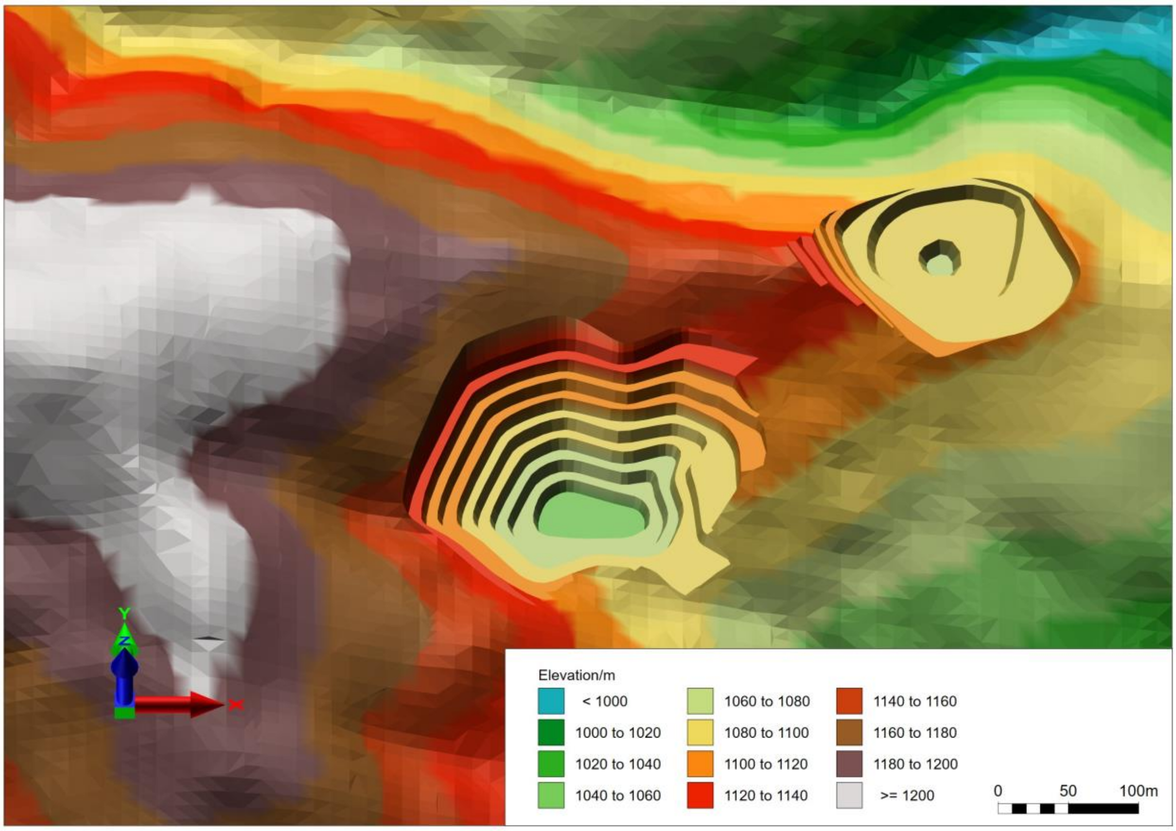


Figure 16.8: Development of Pit Stage Designs Pushback 2

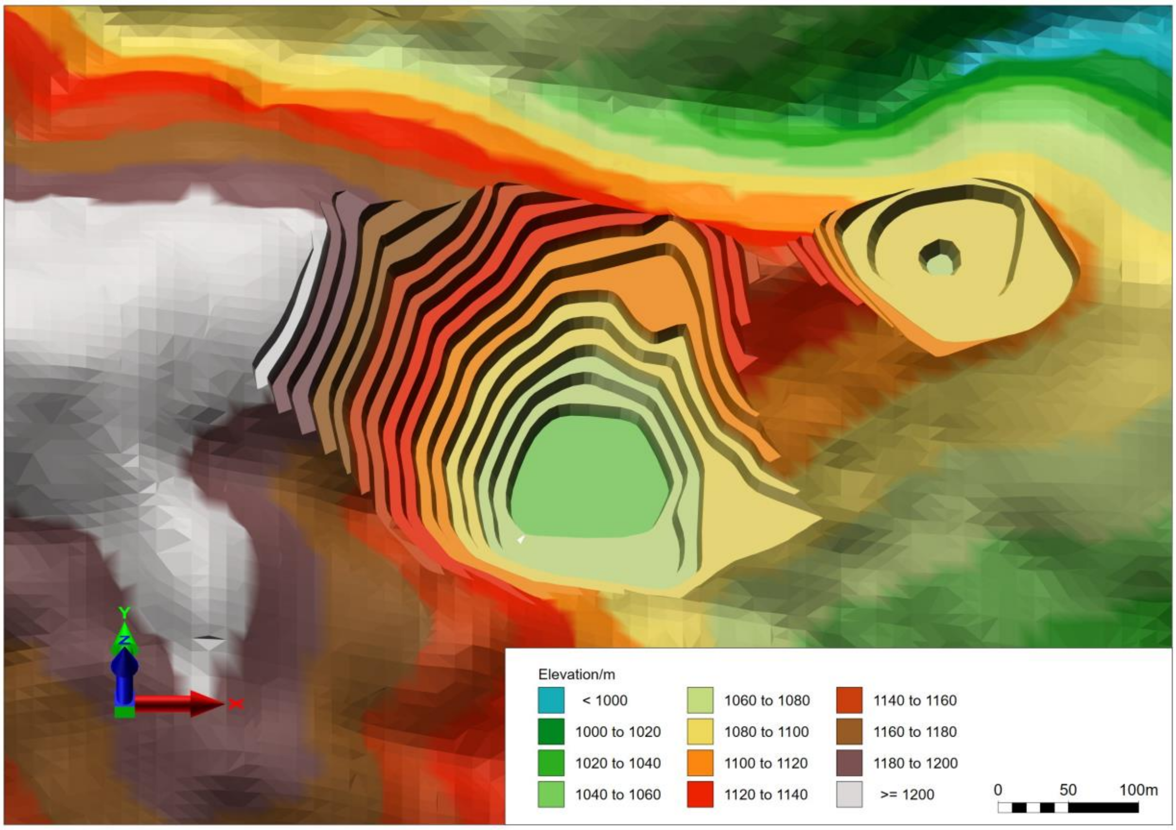


Figure 16.9: Development of Pit Stage Designs Pushback 3

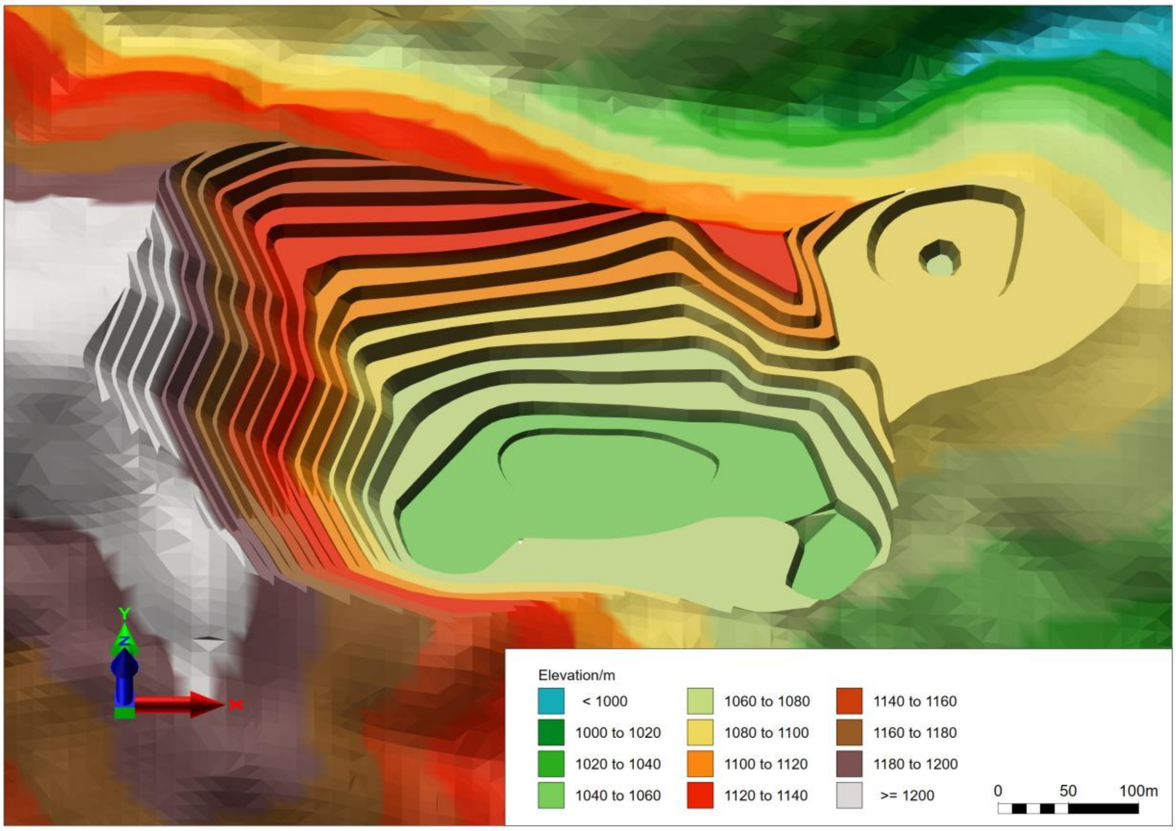


Figure 16.10: Development of Pit Stage Designs Pushback 4

The resulting mineral inventory within each pit stage is summarised in Table 16.12.

Table 16.12: Mineral Inventory by Pit Stage
Note: All blocks with a negative value have been removed from the mineral inventory as these blocks will no longer be economic if stockpiled and reclaimed later. This has reduced the mineral inventory from 6.1 Mt to 5.7 Mt.

Pit	Waste	HG_BX + Mx			LG_Sch + Mx		
Stage	**tonnes**	**tonnes**	**Au g/t**	**Ag g/t**	**tonnes**	**Au g/t**	**Ag g/t**
Pit 1	1,500,000	330,000	5.4	188.8	130,000	1.7	10.2
Pit 2	3,400,000	85,000	9.7	136.6	340,000	0.9	12.8
Pit 3	5,800,000	220,000	4.5	40.5	410,000	1.2	9.9
Pit 4	15,000,000	3,000,000	2.9	40.6	1,700,000	1.2	8.8
Total	26,000,000	3,600,000	3.4	56.6	2,500,000	1.2	9.5

The distribution of unweathered mineralised blocks in relation to the pit stages is shown in Figure 16.12 to Figure 16.17 as a selection of example East-West and North-South cross sections through the deposit. The location of each cross section is shown in Figure 16.11. The initial pit stage targets the high-grade zone to the east (primarily HG_BX), which has a low stripping ratio. Mining then switches to the central portion of the orebody where again there is a high-grade zone.

Subsequent pit expansions (Stages 3 and 4) take the pit to the final pit limit. The extent of the final stage is primarily driven by a thinning of the orebody to the west, in combination with an increasing stripping ratio as the topography generally rises to the west. The mining sequence is also driven by the surface of the mineralisation rising to the north; however, this material is generally of lower grade and would not be mined early on unless the payability of this material was nearer to that of the high grade.

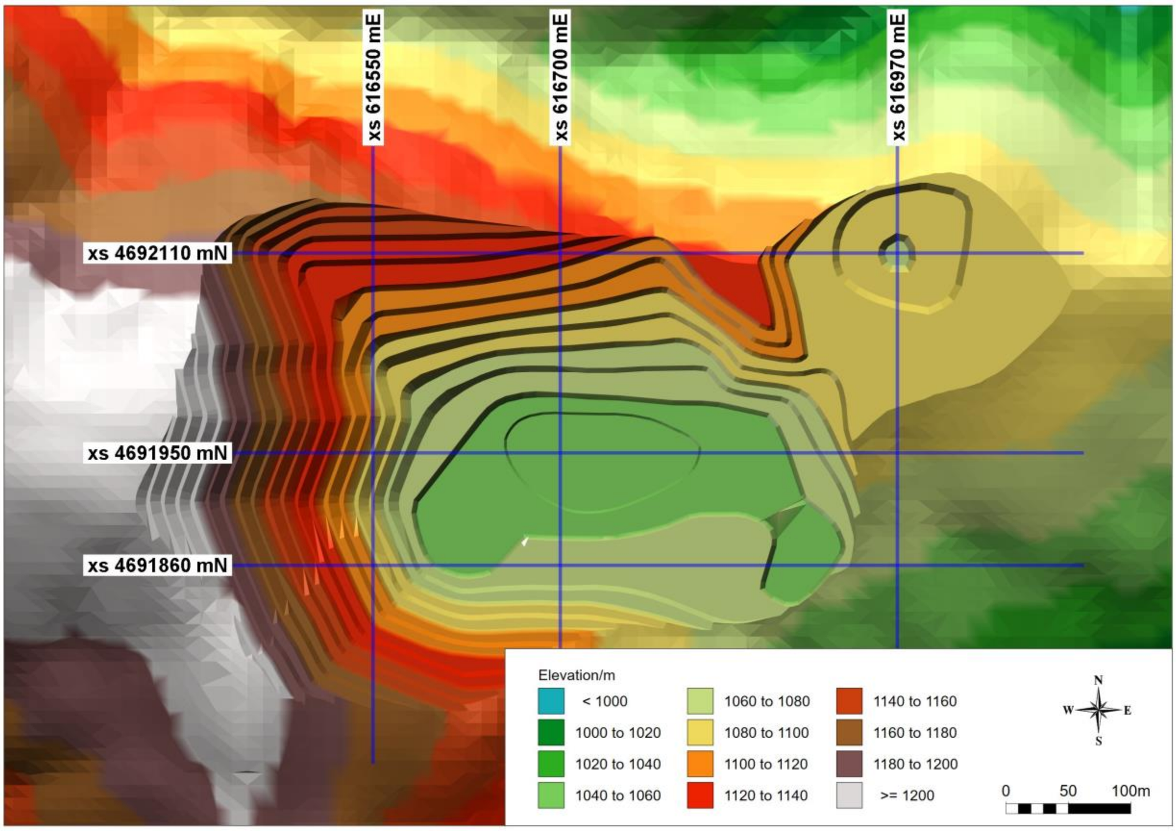


Figure 16.11: Location of example cross sections through conceptual pit design.

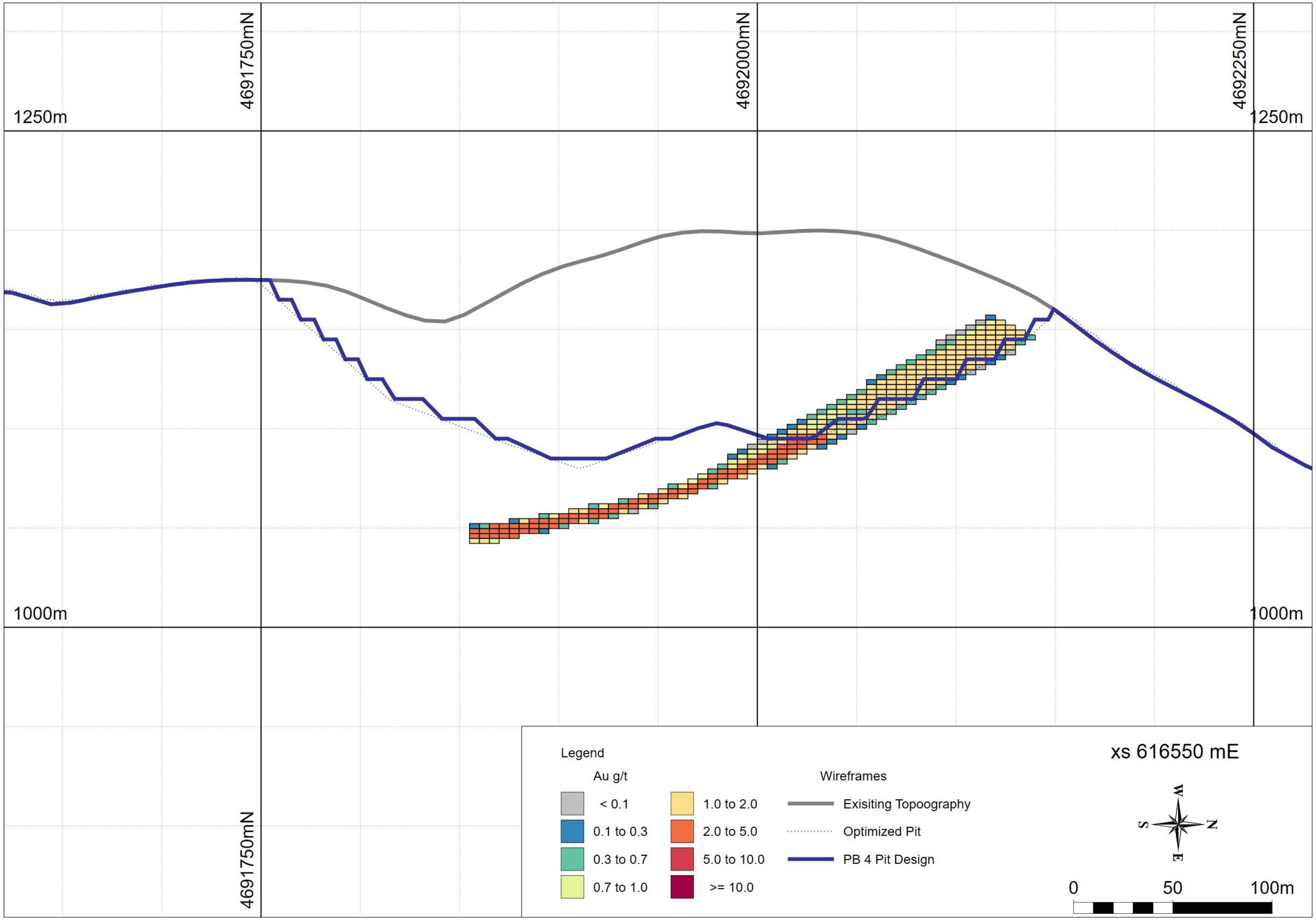


Figure 16.12: Example cross section through conceptual pit with block model. 616550 mE

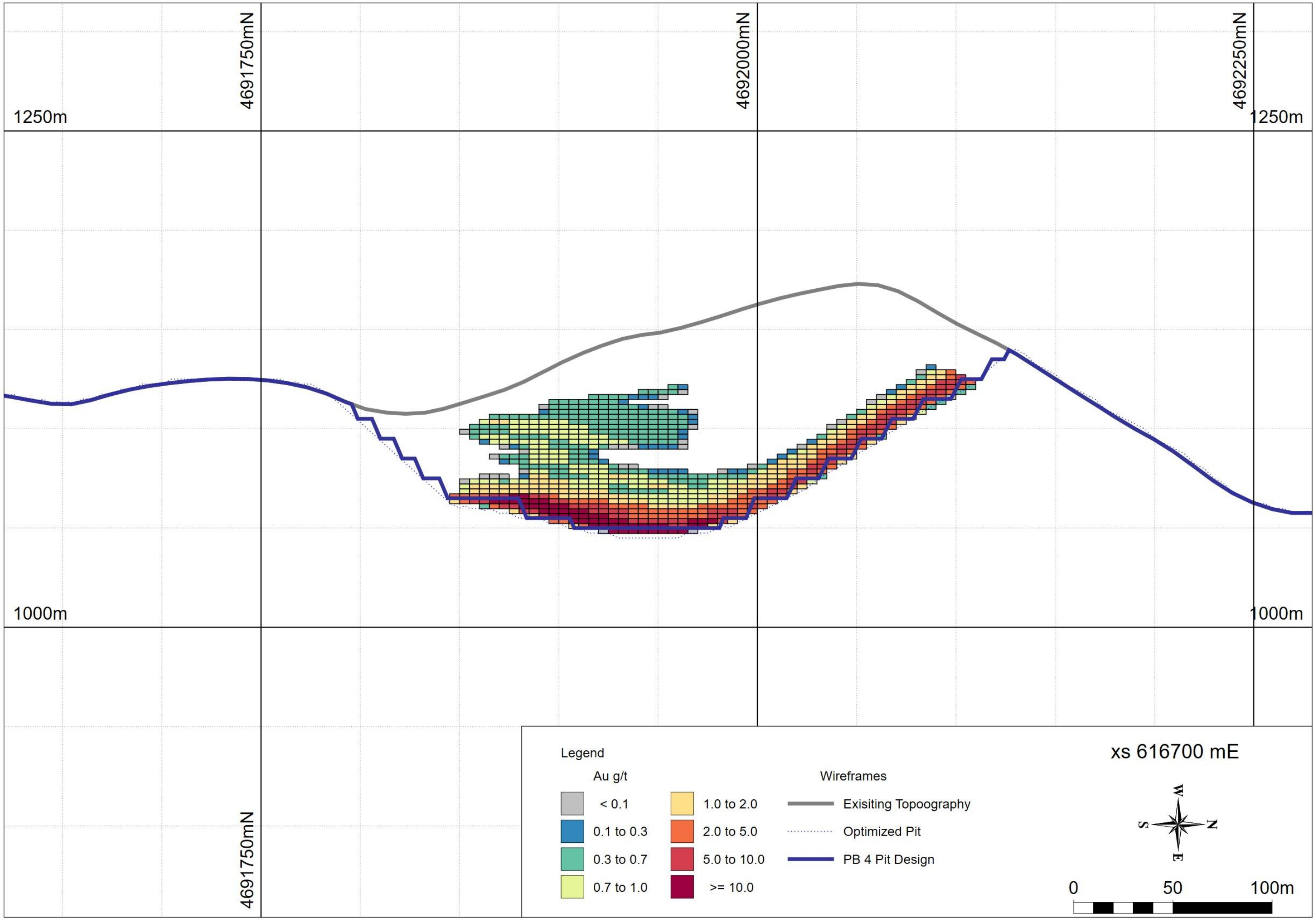


Figure 16.13: Example cross section through conceptual pit with block model. 616700 mE

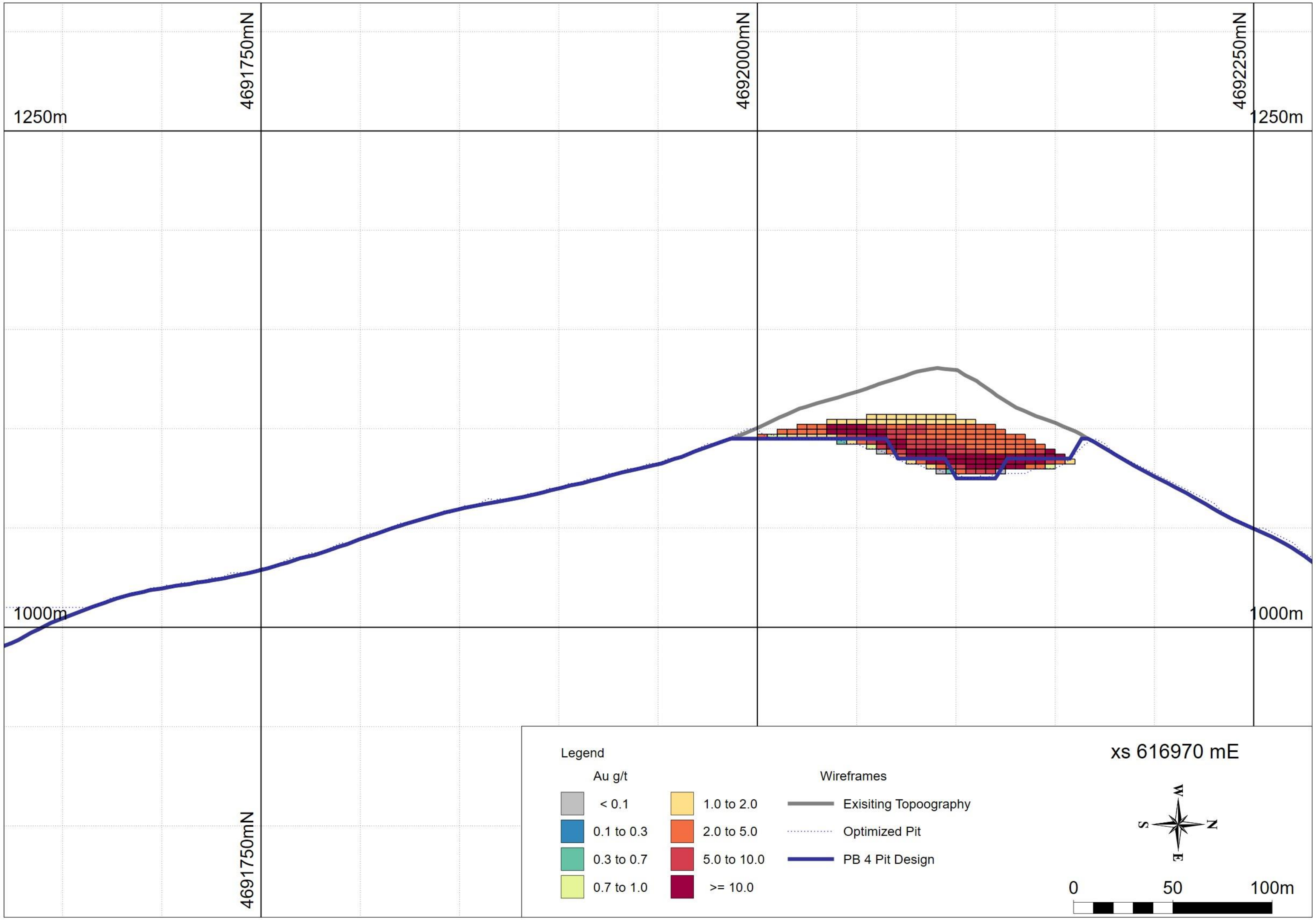


Figure 16.14: Example cross section through conceptual pit with block model. 616970 mE

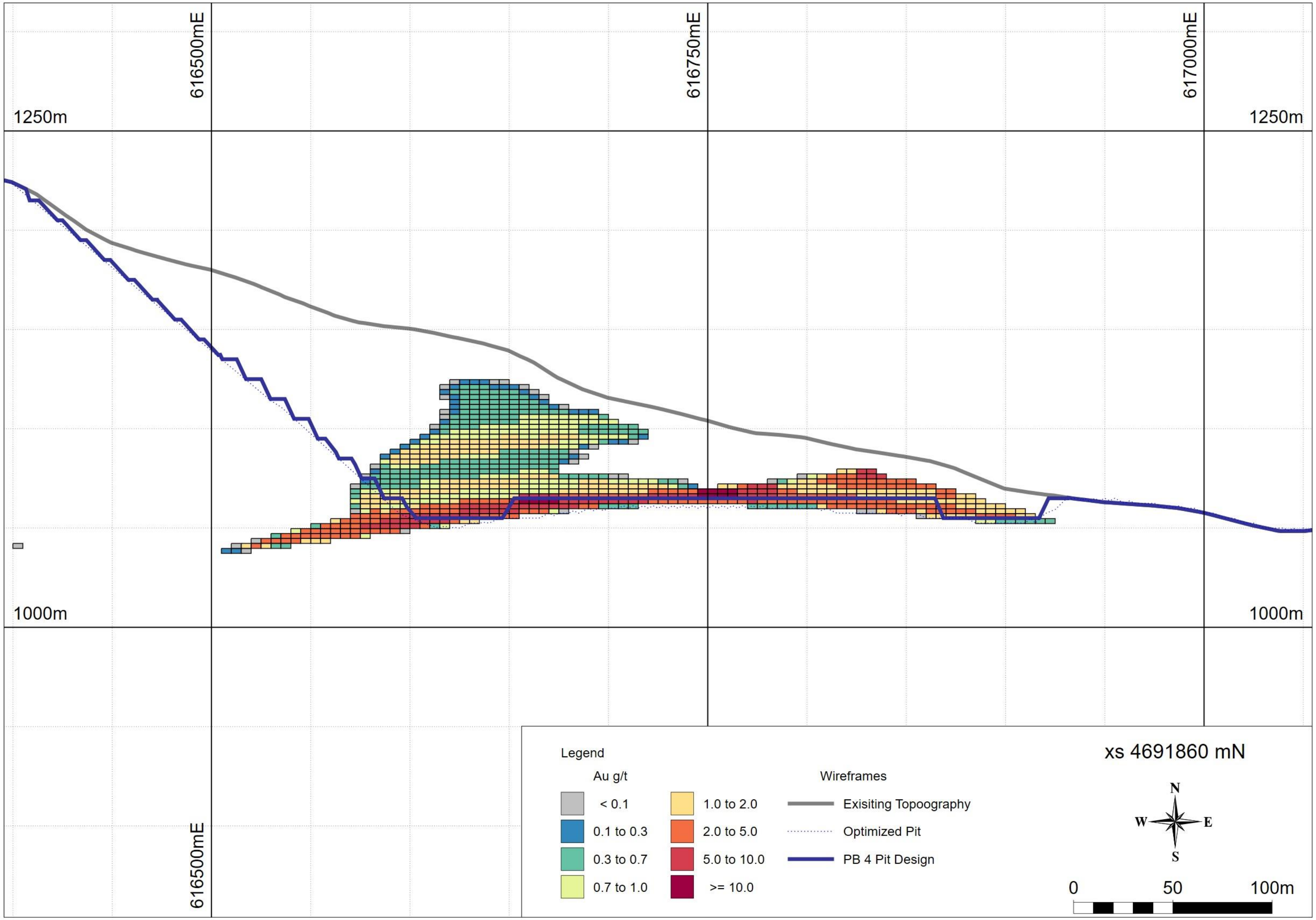


Figure 16.15: Example cross section through conceptual pit with block model. 4691860 mN

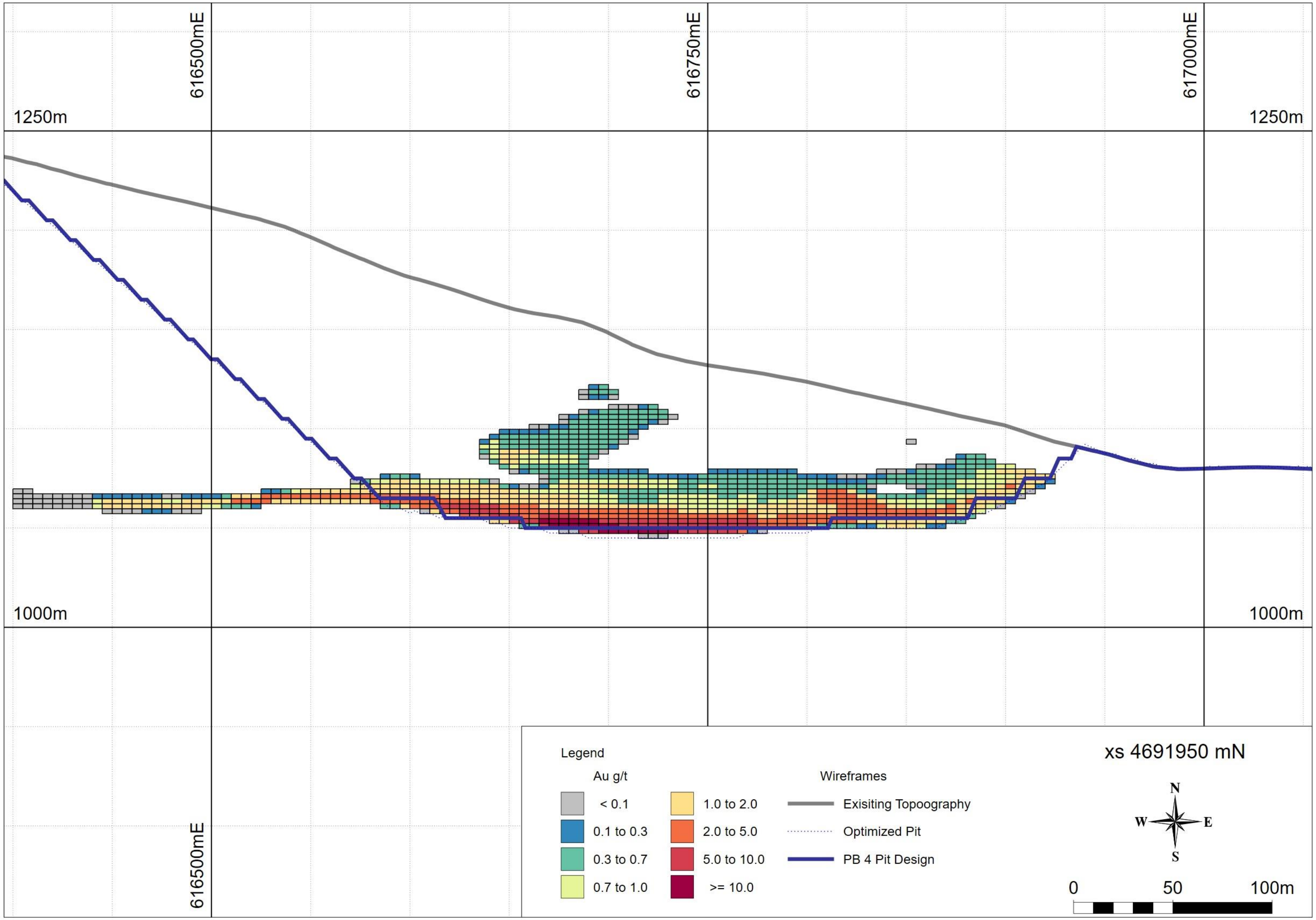


Figure 16.16: Example cross section through conceptual pit with block model. 4691950 mN

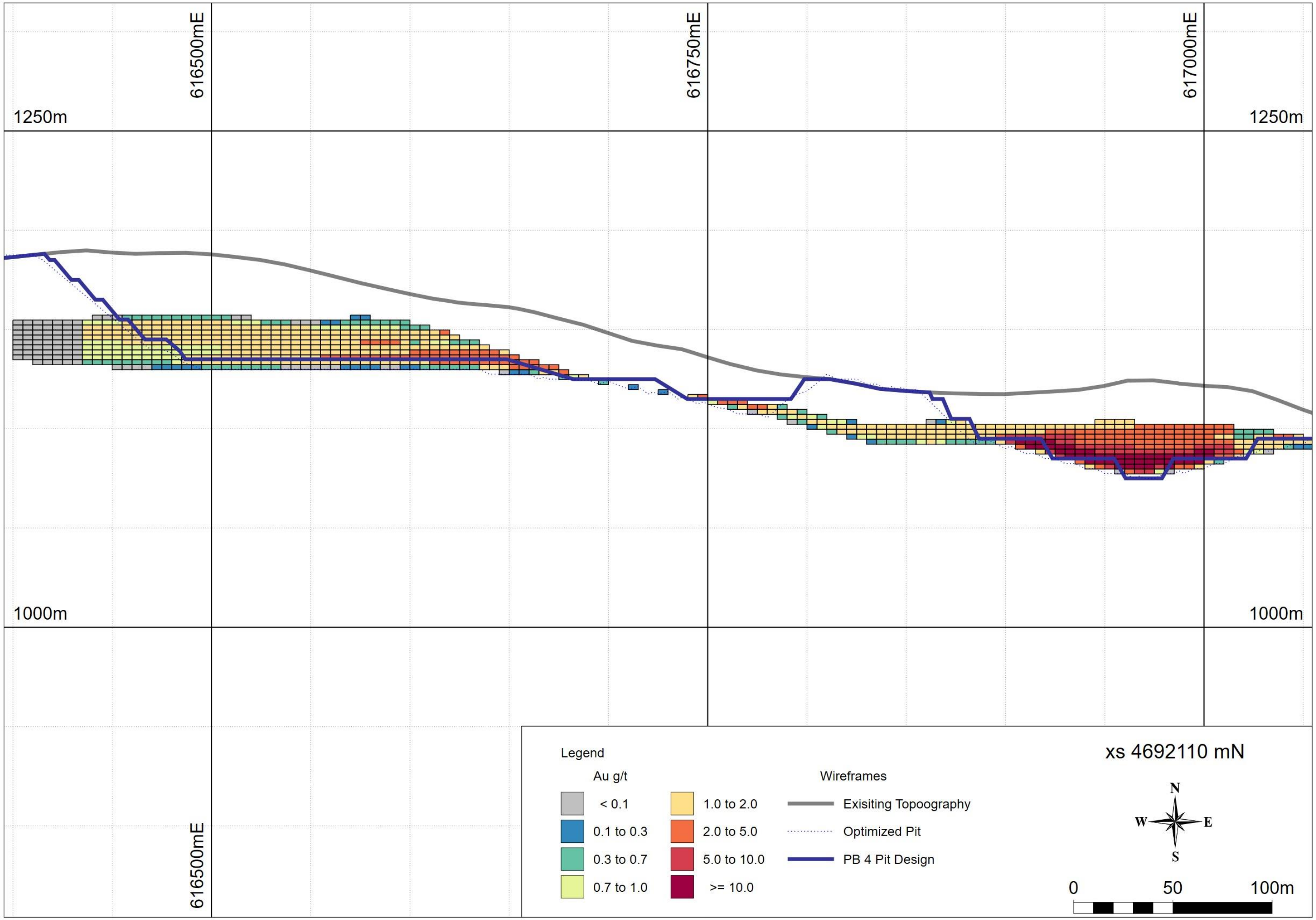


Figure 16.17: Example cross section through conceptual pit with block model. 4692110 mN

16.5 Mine Scheduling

The mine schedule is based on the mineral inventory shown in Table 16.12. The main constraints observed were as follows:

- Plant ramp up at 50% capacity for the first 3 months.
- Average Plant throughput of 600 Ktpa after commissioning.
- Percentage of LG_Sch to be minimised in the early periods.
- Vertical advance rate per stage limited to 90 m/annum.
- Mine capacity limited to 29,000 t/month.

Due to the relatively short mine life the pit stages were scheduled in periods of 3 months to ensure that the mine capacity was smoothed out as much as possible and that the stockpile levels were controlled to provide some smoothing of the grade profile. The annualised schedule is summarised in Table 16.13.

Table 16.13: Annualised Mine schedule.
Note: Scheduling Periods are 12 months with a Pre-strip period of 3 months included in Period -1

Mining Summary	Units	Total	Year									
			0	1	2	3	4	5	6	7	8	9
Total Rock	t	31,700,000	7,270,000	10,600,000	8,300,000	2,620,000	1,090,000	878,000	876,000	121,000	0	0
Total Waste	t	26,000,000	6,500,000	9,980,000	6,880,000	1,730,000	650,000	227,000	51,600	932	0	0
Total ROM	t	5,690,000	766,000	569,000	1,420,000	892,000	443,000	652,000	824,000	120,000	0	0
Plant Feed (All)	t	5,690,000	393,000	627,000	625,000	625,000	625,000	627,000	625,000	625,000	625,000	289,000
	oz Au	480,000	55,000	80,000	45,000	55,000	36,000	44,000	94,000	43,000	17,000	8,000
	oz Ag	7,100,000	1,500,000	1,200,000	470,000	1,300,000	630,000	670,000	740,000	340,000	160,000	75,000
	g/t Au	2.6	4.4	4.0	2.3	2.7	1.8	2.2	4.7	2.1	0.86	0.86
	g/t Ag	39	120	62	23	63	32	33	37	17	8.1	8.1
Plant Feed (HG_BX)	t	3,570,000	251,000	379,000	479,000	369,000	317,000	418,000	619,000	401,000	231,000	107,000
	oz Au	390,000	48,000	68,000	38,000	44,000	24,000	36,000	93,000	35,000	5,900	2,700
	oz Ag	6,400,000	940,000	680,000	320,000	710,000	270,000	400,000	730,000	170,000	21,000	9,500
	g/t Au	2.6	4.4	4.0	2.3	2.7	1.8	2.2	4.7	2.1	0.86	0.86
	g/t Ag	39	120	62	23	63	32	33	37	17	8.1	8.1
Plant Feed (LG_Sch)	t	2,110,000	142,000	248,000	146,000	256,000	308,000	208,000	5,870	224,000	394,000	182,000
	oz Au	85,000	7,400	12,000	7,400	11,000	12,000	8,600	410	8,300	11,000	5,300
	oz Ag	670,000	50,000	110,000	46,000	79,000	96,000	68,000	2,700	65,000	110,000	49,000
	g/t Au	2.6	4.4	4.0	2.3	2.7	1.8	2.2	4.7	2.1	0.86	0.86
	g/t Ag	39	120	62	23	63	32	33	37	17	8.1	8.1
Stockpile IN	t	2,430,000	568,000	259,000	804,000	337,000	70,700	144,000	250,000	0	0	0
Stockpile OUT	t	2,430,000	196,000	317,000	8,780	70,000	253,000	119,000	50,800	505,000	625,000	289,000
Cum Stocks	t		373,000	315,000	1,110,000	1,380,000	1,190,000	1,220,000	1,420,000	914,094	289,094	0

It should be noted that although the total rock movement appears to peak in Period 1 this is due to the schedule commencing mid-way through Period 0 and there being a relatively short period of peak mining rate of 30 months (excluding pre-strip) with a rapid decline in production rate from Period 4 onwards. The total mine life is almost 8 years with 2 years of stockpile reclaim at the end.

16.6 Waste Rock Storage Facility and Stockpiles

A total waste rock capacity of 26 Mt will be required. This can be stored within the valley to the north of the pit and will entail a relatively short haul from the upper benches of the stages by developing haul routes to the north around the contour of the hill.

The initial toe of the Waste Rock Storage Facility (WRSF) will need to be established with a compacted foundation keyed into the bedrock. The WRSF can then be developed by backfilling the valley from east to west with lifts of 10 m and face angle of 25°. Catch berms will be left at 10 m intervals in order that a final profile angle of less than 18° can be obtained by dozing down the faces during rehabilitation.

It should be possible to progressively rehabilitate the WRSF over time with surface topsoil that has been stockpiled separately for this purpose. It may also be possible to consider in-pit storage of waste once Stage 1 has been mined out, and further in-pit storage could take place once Stage 3 is mined out.

A water diversion system will be required to divert surface run-off away from the WRSF as the catchment area at the head of the valley is substantial.

The low-grade stockpile (mainly LG_Sch) will be built up during the life of the mine as part of the envisaged approach to blending of high and low grade ROM material, and it is estimated that the maximum capacity will be around 1.5 Mt.

The low-grade stockpile should be placed as close as possible to the plant to reduce costs. There is an area to the south east of the pit, next to the southern pit exit, that would provide sufficient capacity for this material. The stockpile has been designed with a face angle of 35° and berm width of 5 m on each 10 m lift; the designed capacity, as shown in Figure 16.18, is in excess of 2 Mt.

The general layout of the proposed WRSF and low-grade stockpile is shown, relative to the open pit, in Figure 16.18 . The WRSF has been designed such that the centre of gravity of the open pit and the WRSF are similar, thereby reducing the Effective Haul Distance. It is also possible that the WRSF could be moved further down the valley in the latter stages of the mine development, but this poses additional considerations due to the steepness of the terrain and the ability to stabilise the toe of the WRSF.

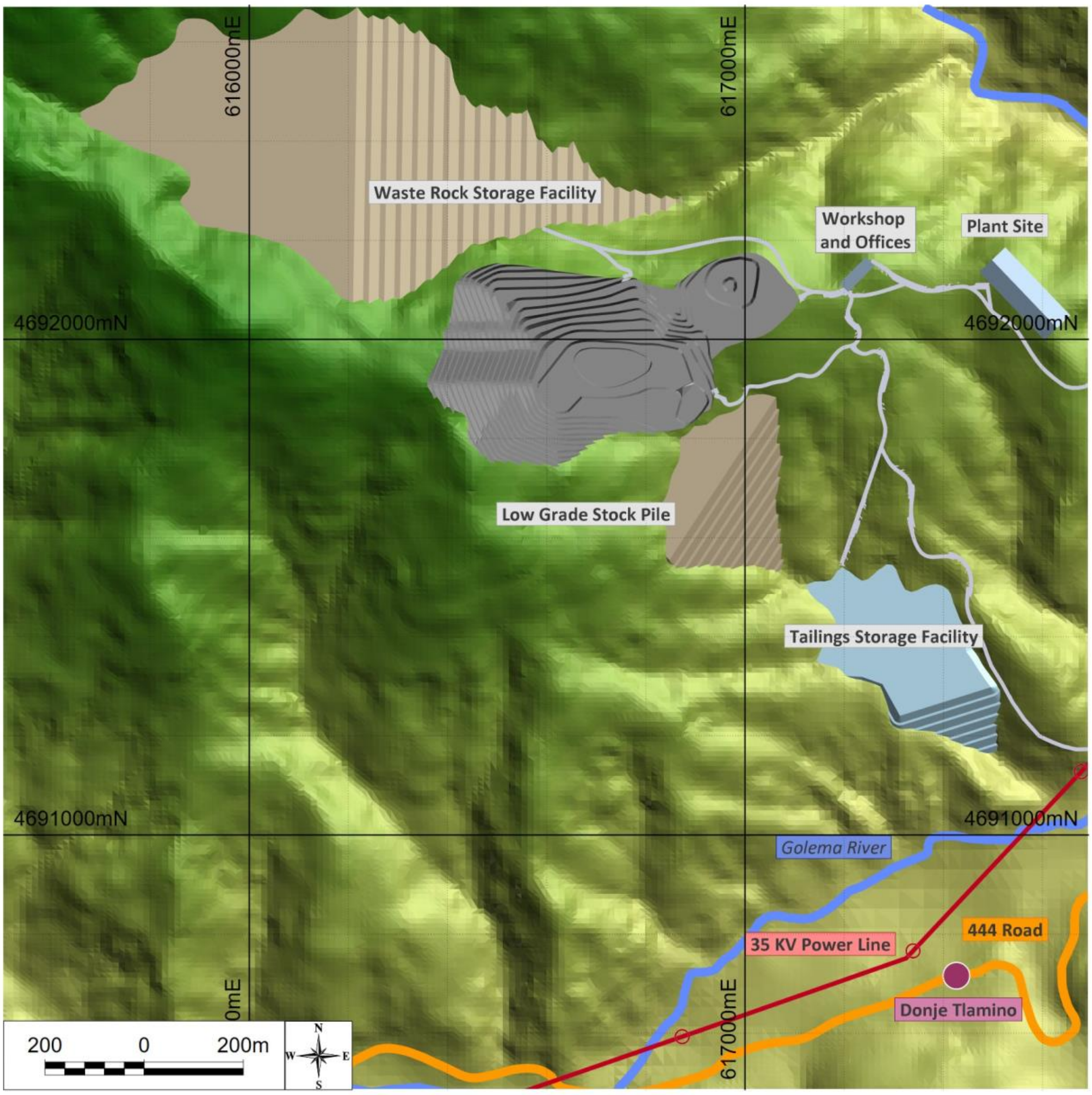


Figure 16.18: Plan View of Pit Limit and location of the WRSF and LG Stockpile

16.7 Mining Fleet Requirements

16.7.1 Fleet Selection

It is expected that the mine will be operated by a mining contractor and the final selection of equipment will be left up to the contractor. There is however a need to ensure that certain areas of the mine can be mined selectively with 5 m, or even 2.5 m flitches, implying that a relatively small hydraulic excavator will be required for this task. Conversely the short mine life (< 8 years) and stripping ratio of around 4.6:1 requires a relatively high peak mining rate of approximately 29,000 tpd to be maintained for at least 2 to 3 years. It is recommended that a mix of two to three smaller excavators (3 - 5 m^3 bucket) and one front end loader is used to give some flexibility and allow for reclaiming from a ROM stockpile at times.

16.7.2 Drilling and Blasting

Based on the relatively low RQD rating assessed for the deposit it is expected that the majority of material can be mined without the need to blast. If blasting is required in the fresh material it will be designed for a 10 m bench height with a holed diameter of 105 mm. A typical blast pattern for a relatively low powder factor is expected to be around 4 m x 5 m. In the areas of selective mining choke blasting should be used where possible to minimise blast movement and reduce waste dilution.

16.7.3 Load and Haul

Based on an average cycle time of 20 minutes for waste and 15 minutes for potentially economic material it is expected that the maximum haul fleet requirement will be 18 x 40 tonne articulated dump trucks. The expected haulage times at Barje are relatively short due to the flat-lying nature of the deposit and limited pit depth allowing the majority of the haul routes to be flat or downhill.

The required loading rate is of the order of 500 tph based on:

- 365 days per year
- 24 hours per day (2 shifts of 12 hours)
- 10 days lost to public holidays and weather
- 65% overall utilisation
- Peak mining rate of 29,000 tpd
- Two pit stages mined at any one time

It is assumed that a typical small to medium hydraulic excavator (3 to 5m^3 bucket) will load an articulated dump truck in 4 passes with a total loading time of around 3.7 minutes, giving an annual production rate of > 2.5 Mtpa. The front end loader will be used for stockpile reclaim as well as clean-up of the pit floor so that precision mining to the waste contacts is possible.

The hydraulic excavators will be used mainly for waste mining on the upper benches and should be suited to a 10 m bench height mined on two 5 m flitches. The excavator needs to have a sufficient breakout force to ensure that the bulk of the mineralization can be mined without blasting, but the fresh material may require some ripping or light blasting.

16.7.4 Support Equipment

The primary mining operations will be supported by a fleet of support equipment consisting of track dozers, graders, water trucks, as well as maintenance and service vehicles. A list of major and support equipment is provided in Table 16.14.

16.7.5 Fleet Requirements

The fleet requirements in Table 16.14 have been estimated based on the mine schedule listed in Table 16.13.

Table 16.14: Fleet Requirements

Equipment	Size/Model	Scheduling Period (Years)										
		0	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
Excavator	$5m^3$	3	3	3	1	1	1	1	1			
Wheel Loader	$5m^3$	1	1	1	1	1	1	1	1	1	1	
Haul Truck	40 t ATD	14	18	15	6	4	4	4	3	2	2	
Backhoe		1	1	1	1	1	1	1	1			
Track Dozer	D9	1	1	1	1	1	1	1	1			
Grader	12' Blade	1	1	1	1	1	1	1	1	1	1	
Rubber Tyre Dozer		1	1	1	1	1	1	1	1	1	1	
Water truck	30,000 l	1	1	1	1	1	1	1	1	1	1	
Fuel & Lube		1	1	1	1	1	1	1	1	1	1	
Service truck		1	1	1	1	1	1	1	1	1	1	
Crane	Grove 40 t	1	1	1	1	1	1	1	1	1	1	
Forklift		1	1	1	1	1	1	1	1	1	1	
Welding truck		1	1	1	1	1	1	1	1	1	1	
Personnel van		1	1	1	1	1	1	1	1	1	1	
Pickup truck		4	6	6	6	6	4	4	3	2	2	
Lighting Plants		4	6	6	6	6	6	4	4	4	2	

16.8 Mine Labour

Mine labour has been estimated with two shifts per day covered by a 4-crew roster, the majority of whom will be provided by the mining contractor. In addition, there will be a small management team and technical services will be required to manage the mining contractor. Workforce requirements are outlined in Table 16.15.

Table 16.15: Manpower Requirements

Role	Contractor/ Owner	Scheduling Period (Years)										
		0	1	2	3	4	5	6	7	8	9	10
Mine Manager	Both	2	2	2	2	2	2	2	2	2	2	
Mine Supervisors	Contractor	8	8	8	8	8	8	8	8	8	8	
Operators	Contractor	148	180	168	124	116	108	100	92	68	60	
Admin staff	Both	8	8	8	8	8	8	8	4	4	2	
Chief Surveyor	Contractor	1	1	1	1	1	1	1	1			
Surveyors	Contractor	4	4	4	4	4	4	4	2			
Samplers	Contractor	4	4	4	4	4	4	4	2			
Maint Mngr	Contractor	2	2	2	2	2	2	2	2	2	2	
Mechanics	Contractor	32	32	32	32	24	16	8	8	8	8	
Tyre Bay	Contractor	8	8	8	8	8	8	4	4	4	4	
Chief Geologist	Owner	1	1	1	1	1	1	1	1			
Geologists	Owner	8	8	8	8	8	8	8	4	2	2	
Tech Services Mngr	Owner	1	1	1	1	1	1	1	1			
Engineers	Both	8	8	8	8	8	8	8	4			
Total		235	267	255	211	195	179	159	135	98	98	

17 Recovery Methods

Laboratory testing for this PEA has demonstrated that bulk sulphide flotation can produce a concentrate with gold and silver contents at grades that are commercially attractive to international markets, as such, bulk sulphide flotation was adopted for this assessment.

17.1 Flowsheet Selection

No comminution testing has been performed, however ball-milling with feed prepared by three-stage crushing and screening is assumed as this represents a simple and robust option for this material type. Alternative comminution methods should be investigated during any subsequent test work programme.

Gravity concentration by Knelson concentrator was tested in the laboratory though no increase in recovery or concentrate grade was achieved and therefore gravity concentration is not included in the proposed flowsheet.

The laboratory programme used a simple flotation scheme to produce concentrates that met the desired specifications. A rougher stage followed by two stages of cleaning are sufficient to produce acceptable concentrate.

Concentrate will be filtered for transportation to customers, and a thickener and pressure filter are provided for this purpose.

Tailings are deposited in a dry-stack Tailings Storage Facility ("TSF"); thickening and pressure filtration have been used to improve the geotechnical properties and reduce environmental impacts. Filtered tailings are hauled to the TSF.

17.2 Process Description

A schematic flowsheet is shown in Figure 17.1.

ROM material is hauled by trucks and tipped on a storage and blending stockpile. Stockpiled ROM is the reclaimed by front-end loader and tipped into a bin. A vibrating grizzly feeder ("VGF") extracts ROM from the bin and scalps coarse material which is fed to a jaw crusher. Undersize from the VGF joins the jaw crusher discharge and is conveyed to a double deck vibrating screen. Oversize from the top deck is conveyed to a secondary cone crusher and oversize from the lower deck is conveyed to a tertiary cone crusher. The cone crushers are adjacent to the jaw crusher and located above the conveyor which collects the VGF undersize, and the combined crusher discharges are returned to the vibrating screen. A nominal screen undersize P_{80} of 10 mm has been assumed.

Undersize from the vibrating screen is the final product from the crushing circuit and is conveyed to a storage bin.

Crusher product is extracted from the bin by feeders which discharge onto a conveyor which delivers the material to a ball mill. This mill operates in closed circuit with hydrocyclones, the cyclone underflow returning to the mill and the overflow advancing to flotation. A mill circuit product that has P_{80} of 80 µm is required, as used in the flotation test work.

Overflow from the mill cyclones enters an agitated tank in which the slurry is conditioned with flotation reagents. Based on the laboratory test work, potassium amyl xanthate (PAX) is used as a sulphide collector and methyl isobutyl carbinol (MIBC) as a frother.

Concentrate from the rougher cells is reground (P_{80} of 30 µm) before it passes to cleaner cells for upgrading. The rougher concentrate is pumped to a hydro cyclone, the underflow of which is fed to a regrind ball mill while the overflow advances to cleaner flotation. Concentrate slurry from the cleaner cells passes to a recleaner cell. The recleaner tailing enters the first cleaner cell with the rougher concentrate. The cleaner tailing is returned to the rougher. In the laboratory tests the cleaner tailing did not return to the rougher as only single batch tests were performed and, while it is assumed for this PEA that returning the cleaner tail is not detrimental to rougher flotation and will therefore have a positive effect on recovery, this should be confirmed through locked cycle tests. PAX and MIBC are added to the cleaner stages as required.

Recleaner concentrate is the final product. Water is first separated in a conventional thickener; the thickened slurry being stored in an agitated tank before further dewatering in a pressure filter. Filter cake is stored and blended in a shed before transport off site by road. The water recovered by the thickener and filter is returned to the mill circuit.

The rougher flotation tailing is densified in a high-rate thickener, the underflow being stored in an agitated tank before final dewatering in a pressure filter. The water recovered by the thickener and filter is returned to the mill circuit. The filter cake is trucked to the TSF.

High-grade breccia and low-grade schist will be processed separately to maximise revenue from the concentrate produced from the high-grade material. Low-grade material from the mine will be stockpiled and processed in campaigns. The laboratory test work shows that the same grind size and flotation parameters are applicable to both material types and can result in commercially viable products.

Some minor modifications to the configuration of the flotation circuit may be required to accommodate the reduced quantity of rougher concentrate and lower flows circulating in the cleaner circuit but these should be easily implemented. Optimisation of the milling and flotation requirements for each rock type will confirm the extent of any adjustments to be made to the circuit.

The low-grade material constitutes 37% of the total feed to the plant over the life of the mine and ranges from 0.9% to 63.1% annually, the highest fraction occurring in each of the last two years of the mine life. Use of a run-of-mine stockpile will allow the variation in annual tonnage of the two feed types to be reduced and campaigns of treating low-grade feedstocks that are around three months in duration are envisaged.

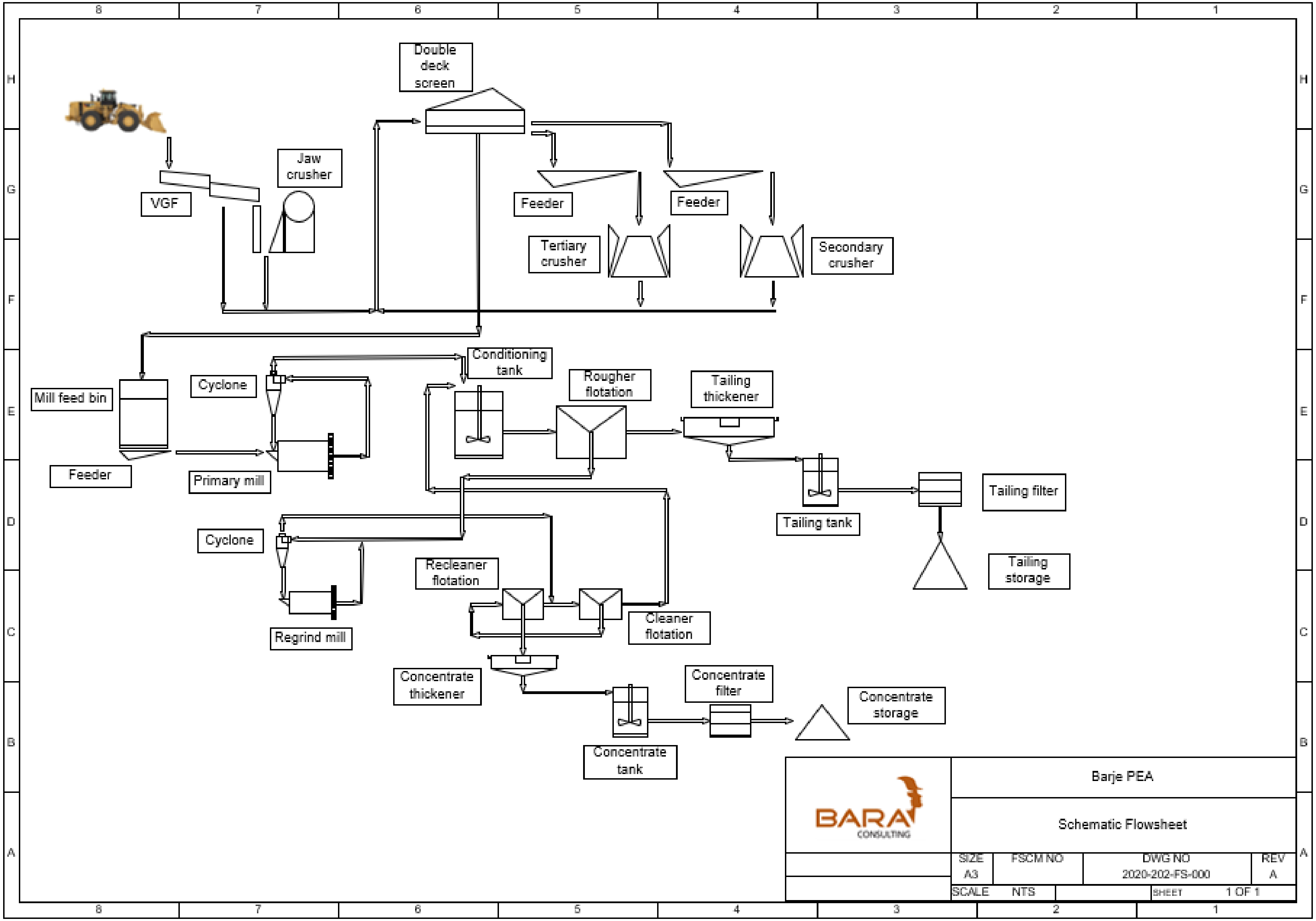


Figure 17.1: Schematic Flowsheet.

17.3 Process Design Criteria

The process design criteria that have been used are listed Table 17.1. Where laboratory tests have provided data, these have been used; parameters not available from laboratory work have been estimated from similar processes or assumed.

Table 17.1: Process Design Criteria

Parameter	Value	Unit	Data type
Annual tonnage	625,000	t/y	Assumption
Calendar time	8,760	h/y	Calculation
Available time	8,712	h/y	Calculation
Utilisation - crushing plant	75	%	Typical
Operating time - crushing plant	6,534	h/y	Calculation
Utilisation - wet plant	93	%	Typical
Operating time - wet plant	8,102	h/y	Calculation
Feed rate - crushing	96	t/h	Calculation
Feed rate - milling	77	t/h	Calculation
Feed composition			
Gold	3.90	g/t	Testwork
Silver	68	g/t	Testwork
Copper	0.026	%	Testwork
Lead	0.32	%	Testwork
Zinc	0.66	%	Testwork
Arsenic	1.20	%	Testwork
Sulphur (total)	3.17	%	Testwork
Sulphur (sulphide)	3.14	%	Testwork
Carbon (total)	0.45	%	Testwork
Carbon (organic)	0.02	%	Testwork
Feed size			
d100	600	mm	Assumption
d80	250	mm	Assumption
Crusher product size (d80)	10	mm	Assumption
Mill discharge solids content	70	% solids	Typical
Cyclone overflow density	35	% solids	Assumption
Mass pull to rougher concentrate	11.7	%	Test work
Regrind milling density	65	% solids	Typical
Mass pull to Cleaner 1 concentrate	7.7	%	Test work
Mass pull to Cleaner 2 concentrate	7.0	%	Test work
Recovery to final concentrate			
Gold	83.4	%	Test work
Silver	82.4	%	Test work
Concentrate thickener underflow solids	70	% solids	Assumption
Moisture content filter cake	12.0	% moisture	Assumption
Tailing thickener underflow solids	65	% solids	Assumption

17.4 Preliminary Plant Layout

A preliminary layout has been developed for the Barje process plant, details of which are presented in Figure 17.2.

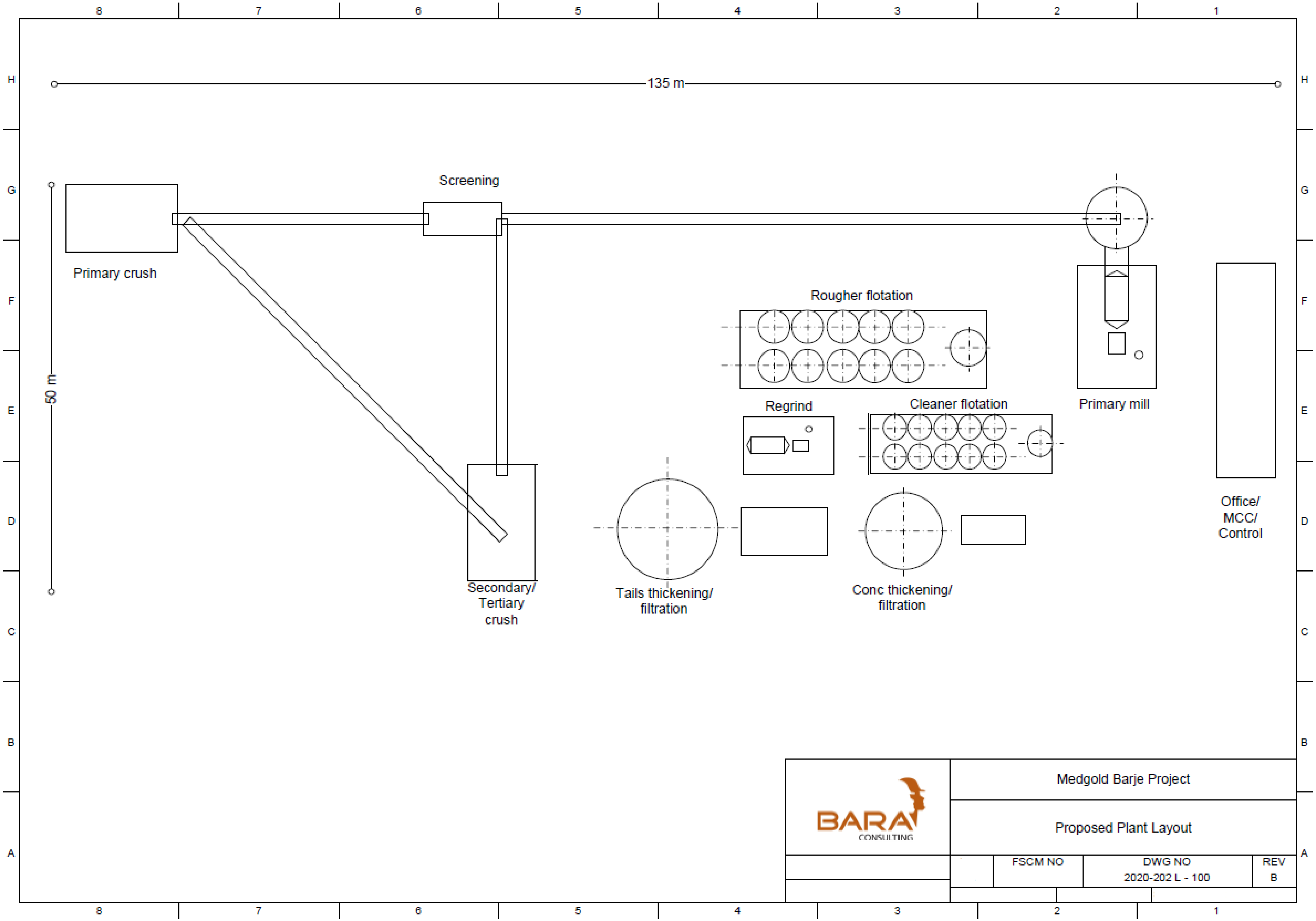


Figure 17.2: Barje Proposed Process Plant Layout

18 Project Infrastructure

18.1 Site Description

Barje is located 20 km south from the town of Bosilegrad, via the 444 sealed road. From route 444, existing gravel tracks lead to the site. Roads are kept open all year round and are only closed for short periods immediately after any heavy snowfall during the winter. The 444 route is also serviced by existing 35 kV powerlines running parallel to the road, from Bosilegrad to the settlement of Podvirovi to the west. The site is flanked by existing rivers, a non-perennial drainage to the north, and the perennial, transboundary Golema river to the south.

18.2 Project Infrastructure

Site infrastructure suitable for a 600,000 tpa ROM open pit mine is planned and costed in the budget. New haulage-standard roads will be established from the existing gravel road junction with route 444, up to the site from the south, passing the pit and stockpile sites, and through to the ROM tip at the plant site. Additional roads to the WRSF site and the TSF location will also be established. Main power will be teed from the existing 35 kV powerline along the 444 road to Podvirovi with a 35 kV/1000 V transformation station and sub-station established near the plant site. A backup power line at 35 kV is planned per Serbian mining regulations. Electricity will be reticulated to other main consumers at 1000 V.

The main mobile equipment workshop, mine office and changehouse will be located near the pit ramp exit. Structures will comprise steel framework with brick walling, sheeted, insulated roofs and standard finish for interiors. Communications and control will be by pervasive WiFi. Pit dewatering will be by semi-permanent submersible pump stations delivering mine water to a common settling pond on surface prior to discharge.

Plant site infrastructure will include a main office, workshop and store with the plant motor control centre and control room located above. A small laboratory for assay/metal accounting and QA/QC will be provided, along with a weighbridge for concentrate accounting. Structures again will be steel framed with brick walling, sheeted insulated roofs and standard fittings.

Fresh water will be sourced from groundwater and pumped to the freshwater tank for use. Potable water will be treated by reverse osmosis prior to distribution. Fresh water will also be pumped to the process water tank for process water make-up. Brown water from changehouses and washrooms in the offices will be routed to a packaged bio-disc sewage system for treatment prior to discharge.

Non-contact surface water will be routed around terraces and structures by suitable berms and culverts for discharge to pre-existing drainage channels. Contact surface water together with excess mine water pumped from underground will be collected, settled, and treated if required prior to discharge to pre-existing site drainage.

Tailings disposal will be by dry-stack method, with the filter plant located at the plant site and dry tailings cake loaded by FEL and transported by ADT to the TSF for shaping and compaction. Water recirculation will be direct from the tailings thickener and filter to the process water tank for re-use.

A schematic of the overall planned surface layout of the project is shown in Figure 18.1.

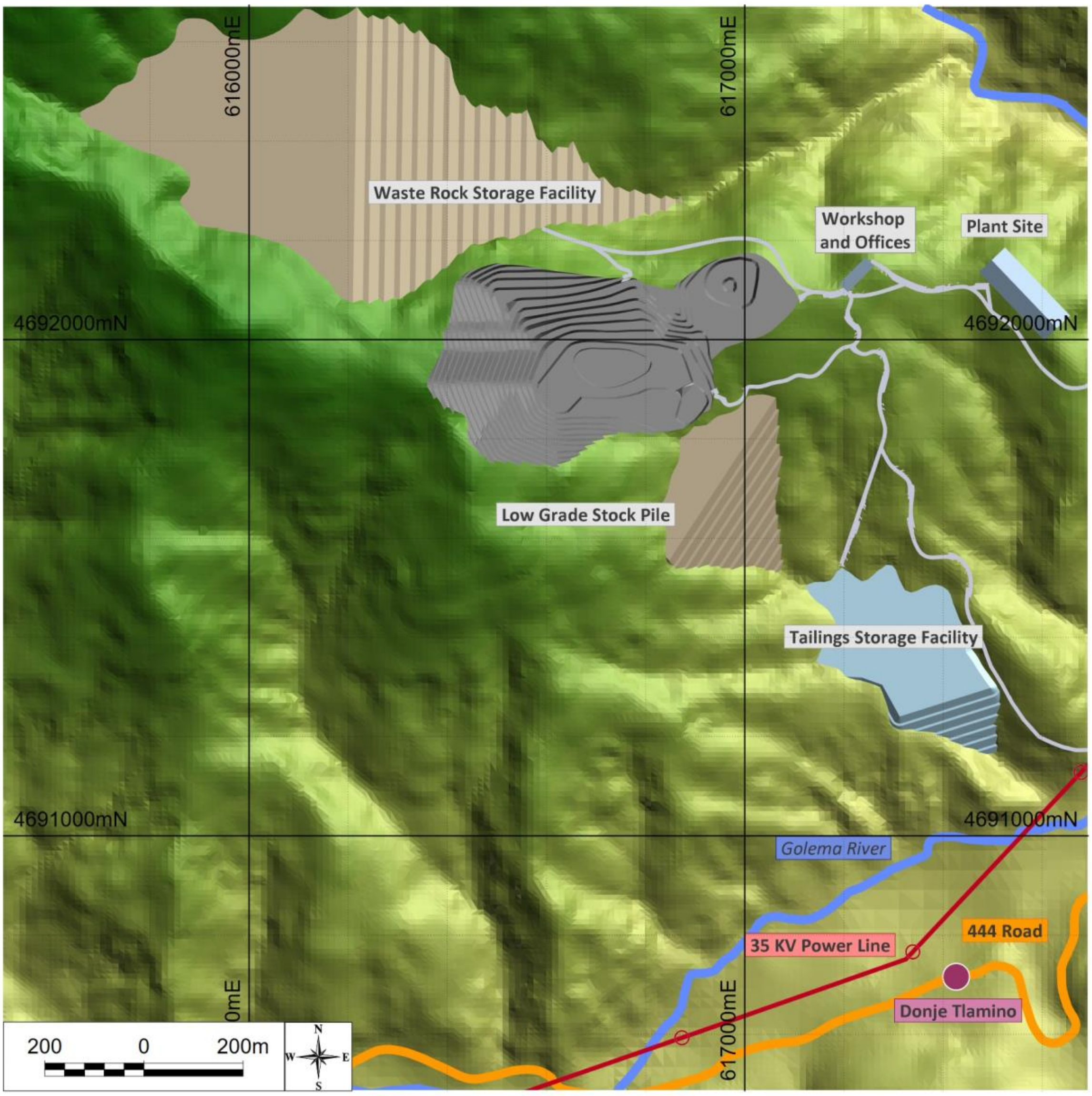


Figure 18.1: Barje Overall Site Layout

19 Market Studies and Contracts

19.1 Concentrates and Offtake

No formal market study was completed for the technical report. Potential concentrate offtake options, and terms for offtake, were however assessed following extensive discussions with gold concentrate traders. As a result of these discussion the decision was taken to produce two separate products, a high grade and a low-grade concentrate, with the majority of Barje revenues derived from a sought after and readily marketable higher grade gold (HG) concentrate at 45-50 g/t Au and <12% As.

Likely offtakers for a refractory Au-Ag concentrate from Barje would be Chinese roasters able to treat and refine such concentrates while accommodating levels of arsenic and other deleterious elements pertinent to the Barje concentrate. Concentrate would be filtered, stored in bulk bags, loaded on superlink trucks and shipped to a suitable port, either Thessaloniki in Greece or Burgas in Bulgaria for shipping via container to China. Terms, CIF, for the HG were assessed at 75% gross, with road transport costs of between US$20/t and US$34/t and sea freight costs of US$45/t to US$60/t. Gross payables for the LG concentrate at 25-30 g/t Au and 14% As were, conservatively, assessed at 40% of ruling prices.

As of the Effective Date of this report, no concentrate sales contracts that are material to Medgold have been entered into. Contracts with respect to transportation and the sale of the HG and LG concentrates will require negotiation in due course.

19.2 Market Analysis

The 2020 survey of gold analysts by the London Bullion Market Association was reviewed (LBMA, 2020). Analysis by Bank of America suggests an average gold price in 2020 of US$1695/oz. Goldman Sachs were more bullish with a forecast of US$1800/oz over the next 12 months. The 2020 survey suggests the gold price will continue to improve over the period, with predictions ranging between the most bullish analyst forecasting an average of US$1,755 and the most bearish analyst surveyed an annual average of US$1,398 (LBMA, 2020).

Surveys are corroborated by actual gold data for the period. The 2-year trailing average gold price (October 2018-October 2020) is US$1550/oz, with a 1-year trailing price of US$1700/oz (Figure 19.1). The long term gold price of US$1500/oz used in the study is therefore considered reasonable.

An assessment of long terms silver prices was also made. The 3-year price history for silver is presented in Figure 19.1. A long-term silver price of US$16.50/oz was assessed, which is considered reasonable to conservative, considering current spot prices for the metal.

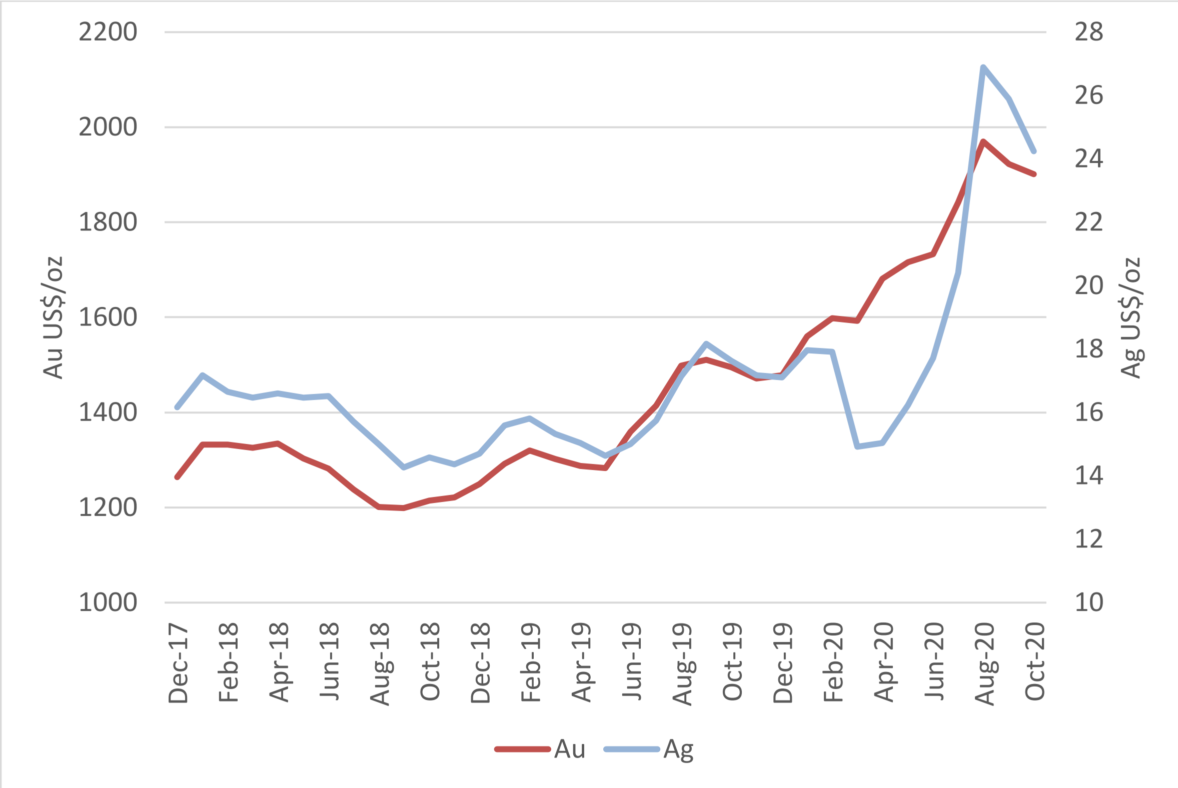


Figure 19.1: Gold and silver price, 2018-2020
Source: London Bullion Market Association

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Introduction

A PEA-level study should include initial evaluation of the environmental and social setting of the project site, noting potentially significant conditions or permitting issues. This section therefore describes the Serbian legal and permitting context, current understanding of site settings and proposed further studies, together with initial identification of potential issues and impacts.

20.2 Legal and Permitting

The legal framework for mining in Serbia was updated with a new Mining Law in 2015 which has increased the efficiency of the permitting process. Although well-defined it is still an in depth and complex process. The framework is aligned with EU regulations and as such has a formal ESIA procedure that requires a minimum of 12 months baseline data collection.

The permitting process for greenfield mines in Serbia requires approximately 20 different approvals and multiple steps in the various procedures. While many of these steps can be made in parallel, it is critical to understand the required timings and to keep a licensing register with application submission dates, authorising bodies and expiry dates where appropriate.

The Serbian EIA permitting process includes three rounds of submission and public consultation, each with associated environmental authority decision points and corresponding opportunities for appeal. The whole ESIA process can take more than 18 months to complete, after the full 12 months of baseline data collection. This needs to be accommodated within the timeframe of Project phases of scoping, Pre-Feasibility and Feasibility studies.

A Certificate of Reserves is a prerequisite to obtain the main permits required for mining in Serbia, namely:

- Exploitation Field Permit
- Mine Works and Facilities Construction Permit
- Approval for Use Permit (which allows production to commence)

The Certificate of Reserves and Exploitation Field Permit both require Scoping responses with Terms of Reference and conditions for the ESIA-Cultural Heritage and Water-approval studies. All such approvals, including final ESIA following evaluation and public consultation, must be obtained before an application for the Mine Works and Facilities Construction Permits may be made. Approvals that are required for the Mine Works and Facilities Construction Permits are:

- Final Environmental Impact Assessment
- Water Management Plan (for supply, usage, treatment, and discharge)
- Protection of Cultural Heritage
- Agricultural Land Usage
- Nature Conservation
- Spatial Planning Compliance (local and regional plans)
- Spatial Plan for Special Purposes (support services and infrastructure)

It is not yet clear whether mining at the Project is compatible with current regional and local spatial plans, and it may be necessary to submit a Spatial Plan for Special Purposes for approval.

All required land acquisition must be completed for Project affected areas prior to submission of the Mine Works and Facilities Construction Permit application and 'proof of ownership' or the 'right to use' for all areas for a minimum period of ten years must be included.

The ESIA study will have to investigate the consequences from 'cumulative impacts', taking account of any existing environmental effects from historic and neighbouring mining operations, as well as 'transboundary impacts', given the Project streams and rivers are tributaries to the Dragovištica River which flows into Bulgaria (Figure 20.1), joining the Struma River which eventually discharges first to the dammed Kerkini Lake in northern Greece and then to the Aegean Sea.

A closure plan and cost estimate must accompany the application for the Mine Building Permit together with a bank guarantee, bill of exchange or corporate guarantee equal at least 30% of estimated closure cost.

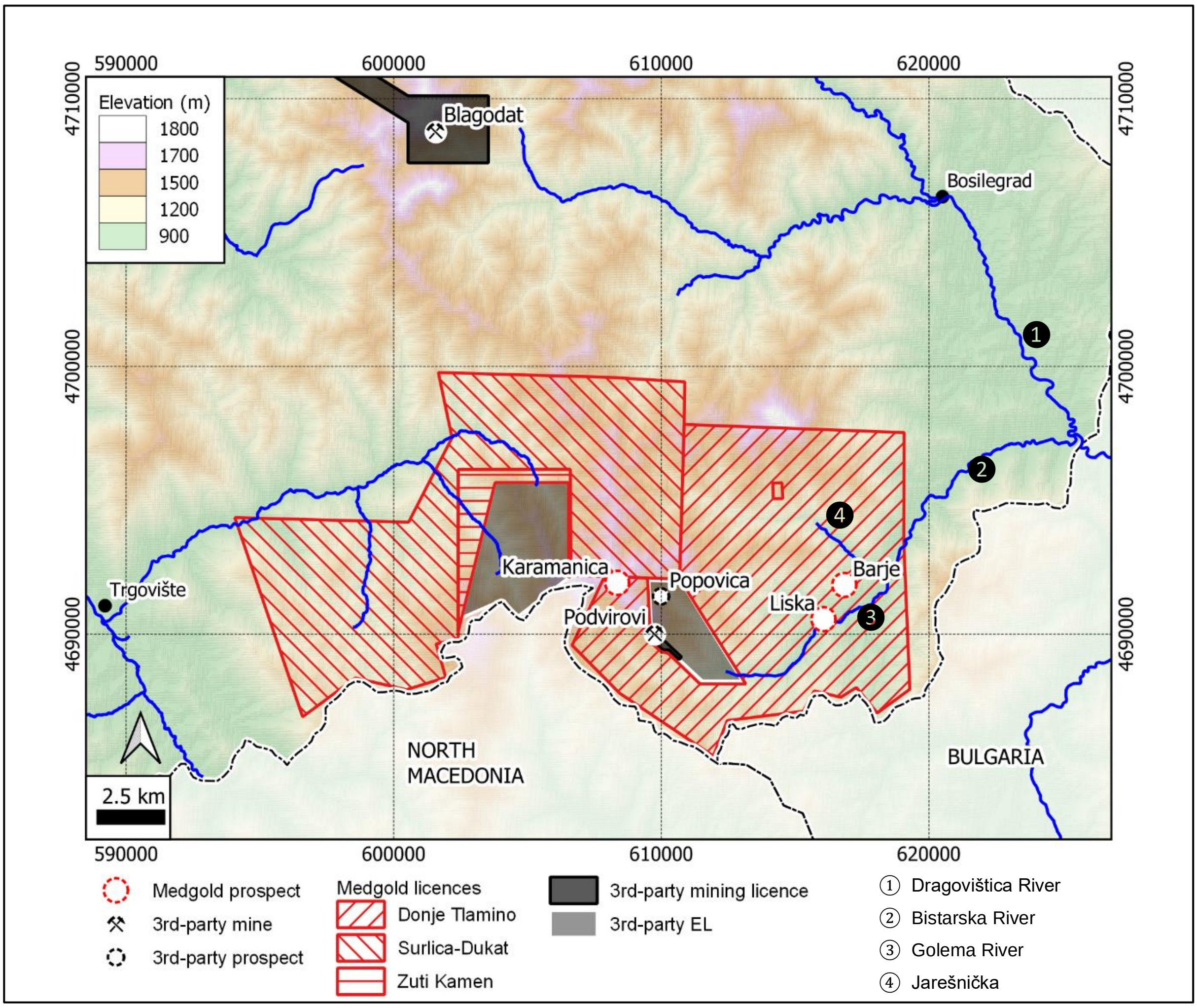


Figure 20.1: Project location with respect to international borders and major rivers.

20.3 Environmental and Social Settings

Some initial baseline studies have started at the Barje site, with a surface and groundwater sampling exercise completed in 2017 by Medgold, together with desktop investigations and compilation of existing and publicly available data. Additional physical data is available from baseline studies undertaken in the adjacent property at the Mineco Podvirovi deposit in 2016. This is less than 5 km from the Medgold Barje Deposit and is therefore relevant in the absence of specific data collection at site at this early stage. The Podvirovi baseline work concentrated on surface water quality, air quality and noise monitoring. Summarization of the available data and information is provided in the following sub sections.

20.3.1 Climate

The Hydrometeorological Service of Serbia has a good network of weather stations across the country including 15 automatic stations providing live streamed data on temperature, pressure, humidity and wind speed and direction. However, in the Project location, where 90% of the area is above 700 m above sea level, the local climate is significantly affected by topography and a site weather station will need to be established to accumulate local data for input to water balance calculations and dispersion modelling.

Generally, the Project area is influenced by Mediterranean climate from the south and EuroSiberian from the northeast, with hot, dry summers and long, cold and snowy winters with strong winds and severe frosts. Average maximum temperature is 27°C in the summer and mean winter minimum of -2°C. Average annual precipitation is 626.7 mm, mostly in the autumn and as snow in winter. The predominant wind throughout the year is from the northeast, but not generally high strength.

20.3.2 Ambient air quality

Given the rural context of the project, ambient air quality is likely to be generally good, influenced by agricultural and forestry activities, fires and summer dust from road traffic. Neighbouring quarry and mining activity could also add to dust and exhaust emissions.

Baseline air quality investigations undertaken in 2016 the adjacent property, less than 5km away from the Project, collected data from a single location over a 7 day sampling period in springtime. The monitoring device collected solid particle dust on 47 mm diameter filter paper for 24 hour periods. Gas emission monitoring was not included, but the collected dust was analysed for a full range of chemical and heavy metal elements. This study found that none of the daily results exceeded Serbian national Air Quality Maximum Allowable Concentration (MAC) standards for either total suspended particles; or for contained Pb, Cd, As and Ni, all of which were below detection limit (Table 20.1).

Table 20.1: Air Quality Test Results - Mineco Potvirovi, 2016

Tested Parameters	MAC µg/m^3	Results µg/m^3
Total suspended particles	120 (24hr period))	11- 67 (7 day range)
Pb	1	<0.2
As	6	<2
Cd	5	<2
Ni	20	<5

20.3.3 Noise

Existing noise sources in the area are likely from traffic, farming and forestry activity (tree felling), quarry and mining activity, dog barking and other farm/wild animal noise.

Noise level testing was also undertaken at the neighbouring property during the 2016 study, with measurement of noise levels performed in March, at one measuring point. This measured noise ranging 27 – 39dB (A), well below authorized per day levels.

20.3.4 Vibrations and Seismicity

Seismic data for the region and area has been gathered from publicly available information. Serbia – and particularly the Pčinja District in the SE - has a general earthquake hazard classification of “medium” according to the Seismological Survey of Serbia. This means that there is a 10% chance of potentially damaging earthquake shaking in the project area in a 50-year period. The area is described as unstable terrain with a VIII Mercelli intensity score, raising to IX in the east part of the municipality (Bosilegradska kotlina). This equates to magnitude 6 – 6.9. Based on this, potential earthquake impact will need to be considered in all phases of the Project, especially during design and construction, as well as blasting operations.

20.3.5 Geology and Geochemistry

The Project is in the western Tethyan orogenic belt, a complex zone associated with tectonic plate collision and magmatic activity. The geology of the Project area is composed of high metamorphic-grade rocks in the north and east, and low metamorphic-grade rocks in the south and west. Older intrusions cut both metamorphic sequences but mostly in the higher metamorphic grade rocks. Minor areas of sediments and volcanoclastic sequences are deposited unconformably on the low grade metamorphics and all sequences are cut by porphyritic dykes and sills and volcanic plugs. There are overlying quaternary deposits of unconsolidated talus on the higher ground and alluvial sediments along valley floors.

The Barje Deposit is the most advanced of three mineralized zones within the Property, where mineralization is associated with a hydrothermal breccia, and found also in a lower-grade halo in the overlying rocks. The breccia contains clasts of the local wallrocks cemented by a matrix of quartz ± calcite/siderite and sulphide minerals, including pyrite, arsenopyrite, sphalerite, galena and more rarely chalcopyrite and tennantite. Gold is found associated with sulphide minerals in disseminated sulphides and sulphide breccia fill in HG_BX and LG_Sch. There are also partially weathered and oxidized components of these two materials in a third recognised type, OX, where gold is observed as liberated grains or associated with non-sulphide gangue.

In initial plans materials to be extracted from a Barje pit are expected to include approximately 26.5 Mt waste rock that will largely be from the upper schist unit and generally contain <0.5% pyrite; 2.1 Mt of LG_Sch material with pyrite averaging 1.5%; and 3.6 Mt HG_BX material with over 1.5% pyrite and other sulphides. Simple flotation is planned for both the HG_BX and LG_Sch material, with the HG_BX processed directly from excavation in the pit, but LG_Sch possibly campaign batched into the plant feed during early production years and towards the end of mine life. The OX material did not respond favourably to test flotation and is presently expected to be stockpiled. Various mineralized- and waste-rock types will be exposed in the pit walls at different locations and times over the LOM; and flotation tailings are likely to contain remnant sulphides. All these sources have the potential to release metal contaminants, especially As, even under non-acid/alkaline conditions.

Chemical composition of the three material types as well as tailings derived from the high- and low-grade materials were determined and are shown in Table 20.2. Sulphur content assayed between 0.2 and 3.2%, with the lowest sulphur content measured for the OX composite. Sulphur content is predominantly as sulphide minerals for the HG_BX Fresh and LG_Sch Fresh composites, principally as iron sulphide minerals given the relatively low content of copper, lead, and zinc. Arsenic is elevated at 0.6% to 1.2%. Carbon is low in all samples suggesting limited effective carbonate content for neutralising potential. The rougher tailings would form the majority of material in the tailings storage facility (TSF) while the HG_BX and LG_Sch represent materials that would have to be stockpiled at various times prior to processing.

Table 20.2: Chemical Analysis of Mineralised Feed and Tailings from Metallurgical testing

Element	HG_BX	LG_Sch	OX	HG_BX tails	LG_Sch tails
Cu	0.026	0.01	0.006	0.003	0.001
Pb	0.32	0.07	0.08	0.06	0.02
Zn	0.66	0.15	0.04	0.04	0.02
Fe	3.4	2.4	3.1	1.4	1.5
As	1.2	0.86	0.6	0.16	0.12
S (T)	3.17	1.56	0.19	0.2	0.1
S (S-)	3.14	1.52	0.13	-	-
C	0.45	0.9	0.12	0.45	0.89
TOC	0.02	0.04	0.02	-	-

Mineral composition from QEMSCAN Particle Mineral Analysis showed that pyrite and arsenopyrite were the predominant sulphide minerals in the HG_BX Fresh and LG_Sch Fresh composites, with lesser sphalerite and galena. Pyrite and arsenopyrite measured only about 0.2% for OX material with only trace galena and sphalerite. Arsenic occurs almost exclusively as arsenopyrite for the HG_BX and LG_Sch material, whereas most of the arsenic in the OX sample was in non-sulphide mineral forms. Gangue minerals were primarily quartz, feldspars and micas.

As the Project progresses, further specific geochemical data for Acid Rock Drainage and Metal Leaching (ARD/ML) risk assessment will be gathered from core log evidence of sulphides, especially pyrite, carbonate veining (calcite/dolomite) and any additional mineralogy/chemistry from future metallurgical tests. A programme of geochemical testing including Acid Base Accounting (ABA), Net Acid Generating (NAG) and Humidity Cell Test (HCT) testing would need to be undertaken as part of the PFS investigations to inform the mine design, site layouts and proposed construction and management systems. It is also recommended that field trials are set up as soon as possible to start gathering long-term practical field data.

Measurements of non-ionizing radiation levels (RF band 100 kHz - 3 GHz) undertaken at another deposit "Lisina" in the municipality of Bosilegrad, showed that the maximum measured values were far less than those recommended by the International Commission on Non-Ionizing Radiation Protection (ICNIRP). Concentration values of natural radionuclides are characteristic of the soil and no different from other localities in Serbia. Concentration values for radionuclides in water were below the detection limit, as well as those obtained from plant cultures, except values for potassium. Ambient dose of gamma radiation is the same as across Serbia.

20.3.6 Terrain and Landscape

The terrain of the Project area is composed of deeply incised river valleys with steep slopes and high broad ridge tops, with elevations ranging from around 820 m in the Golema River valley to the high point of the Golemi Peak at 1,820 m. The valley sides and higher, north-facing slopes are dominated by beech and pine forests. South-facing slopes and the ridge tops are mostly open grassland and unimproved pasture with small shrubs. Valley floors and flatter areas at mid elevations have been cleared for agriculture although a significant amount of the cleared land is not currently used for that purpose.

20.3.7 Soils and land-use

Data have been collected from publicly available sources on soils, fertility and land capability. The purpose of this exercise is to characterize the soil resource in terms of Ecosystem service contribution and capacity for land-uses. Baseline studies will undertake site soil mapping and pits to expose profiles to provide input for topsoil stripping requirements on the Project footprint and stockpile resources for progressive and final rehabilitation. Aerial photography will provide detailed site land-use coverage.

Soils in the area are podzols, reddish-, brown forest- and mountain black soils and deposits of recent alluvial deposits in the lower parts along the river. The alluvial deposits are easier to work and is used for much of the good agricultural land. The majority of the area Bosilegrad municipality is made up of relatively low-quality land with fertility Category V-VIII soil/land mostly forested and for growing fodder crops. Farmed hill ground is used for some potato and oat production, but most is pasture and livestock or is forested. Terrains below 800 m have favourable conditions for commercial fruit production. Agricultural land covers over 62% of the total area of the municipality, composed of arable land; increasingly left fallow for meadows and pasture, especially on higher hills; and 3% orchards and market gardens. Forest makes up 36% of the total land area, mostly beech and pine, and the municipality of Bosilegrad has important forest resources, with mountain beech forest on acidic brown soils.

Most of the arable land is held as individual farms for food production, with less than 30% owned by companies and cooperatives. Farmland in the higher hills is increasingly converting to pastures for livestock. Average size of plots per household is 1-5 ha for the town of Bosilegrad and the settlements of Rajčilovci and Radičevci in the lower hills below 800m; and 20-50 ha in mountain settlements areas, over 800 m. Modern agrotechnical measures are poorly applied and production cereal yields are low. State and cooperative forests cover 34% of the area under forest, belonging to the South Moravian forest area, managed by the Bosilegrad Forest Administration and part of the Vranje Forestry Farm. Private forests accounts for the remaining 66%. Wood processing capacities account for most economic activity in the region, including quality wood resources as well as firewood. However, the industry is underdeveloped and presents significant potential for economic development.

20.3.8 Surface water hydrology

The Property includes catchment areas for the Dragovištica River, which flows southeast into Bulgaria to the Struma river, and the Tripušnica River which flows west into North Macedonia to the Pčinja River. Both the Struma and Pčinja rivers eventually discharge to the Aegean Sea after passing through northern Greece. Most of the rivers and streams of the municipality of Bosilegrad are in the Dragovištica catchment and all watercourses in and surrounding the Barje Deposit drain to this catchment (Figure 20.2).

The Barje Deposit is on an east-west trending ridge between two valleys. The area to the south of the ridge has been proposed to contain stockpiles of low-grade material and a dry-stack tailings storage facility; there are several springs and small streams in this valley that flow into the Golema River. The valley to the north is proposed to contain the waste rock storage facility; springs and small streams in this valley flow into the Jarešnička River which in turn joins the Golema River at Bistar. The Golema watercourse continues northeast as the Bistarska River before joining the Dragovištica River at Ribarce and turning south across the Serbian-Bulgarian border.

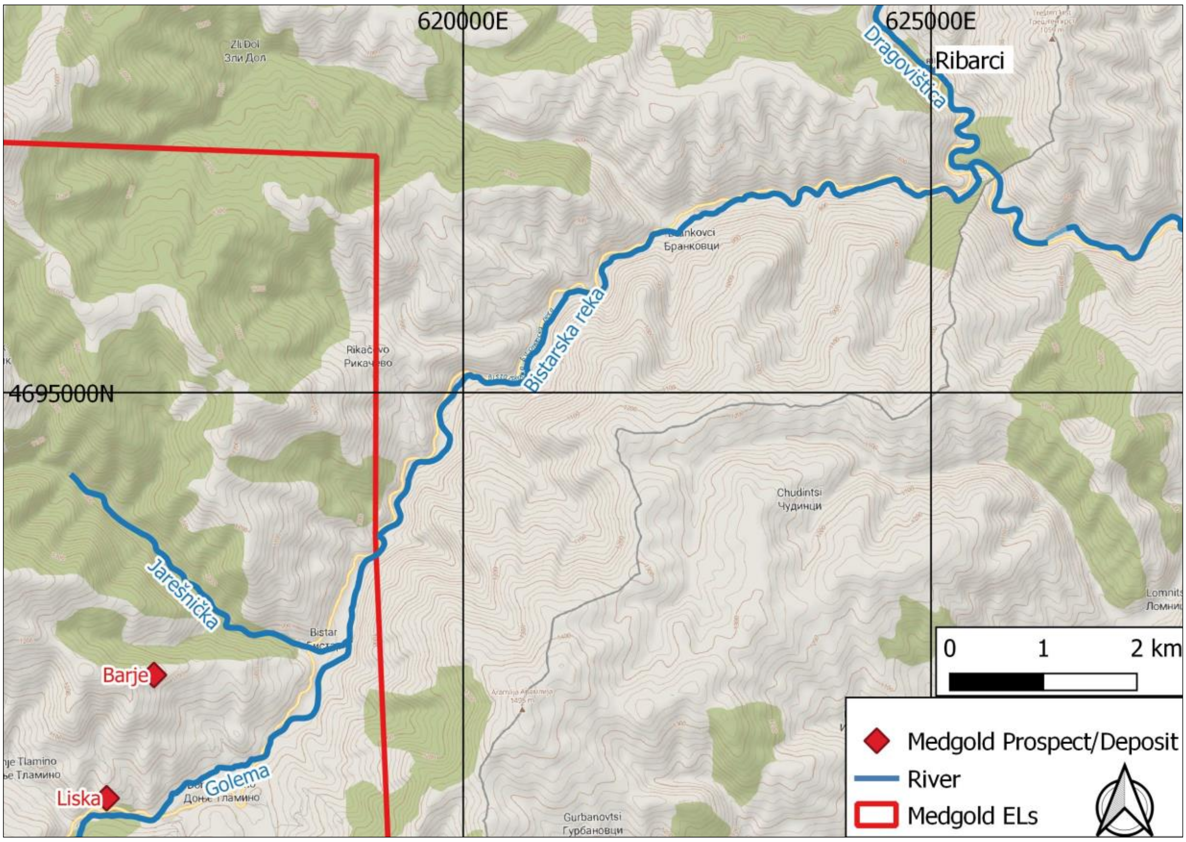


Figure 20.2: Major waterways in the Barje area.

The Republic Hydrometeorological Service of Serbia ("RHSS") runs a network of automated hydrological stations with real time data on surface water levels, as well as gathering river flow measurements, temperature, TSS, ice events and water quality monitoring. The closest stations to the Project site are at Ribarci on the Dragovištica River (Figure 20.2). These will provide a good record of historic data starting in the mid-1960s.

20.3.9 Hydrogeology

There are no RHSS groundwater stations in the area, with almost all official stations associated with the catchment and aquifer sources for the Danube and tributaries to the north. Public/academic data on regional and local aquifers, together with physical site evidence of springs, borehole records of water intersections and water levels in local wells will be collated.

20.3.10 Surface and groundwater quality

Water sampling undertaken in March 2016 on the adjacent property determined the surface water quality of the Bezimeni stream, located 6km west and upstream from the Medgold Barje Project area, at two measuring points (Figure 20.3). The sampling programme was financed by Bosil Metal doo., an investor in the Podvirovi mine. These results included neutral pH; MnO_2 4 and 4.6 mg/l; NH_4 0.14 and 0.19 mg/l; NO_3 0.011 and 0.007 mg/l; NO_2 3.6 and 2.7 mg/l; Cl 5 and 8 mg/l; Fe 0.51 and

0.51 mg/l; SO_4 66.6 and 43.7 mg/l. All other analytes were below detection limits and all results were below water quality standard limits.

Medgold undertook a water quality sampling programme in June 2017 focused on drainages around the Barje Deposit, the Liska Prospect and the main drainage from Mineco's Podvirovi mine. A total of thirteen surface water and nine groundwater samples (five wells and four springs) were sampled (Figure 20.3). This was an effective initial site water quality study that will set the ambient conditions for the Barje ESIA scoping study. Only arsenic showed ambient levels above regulatory limits in a few of the Project area sites.

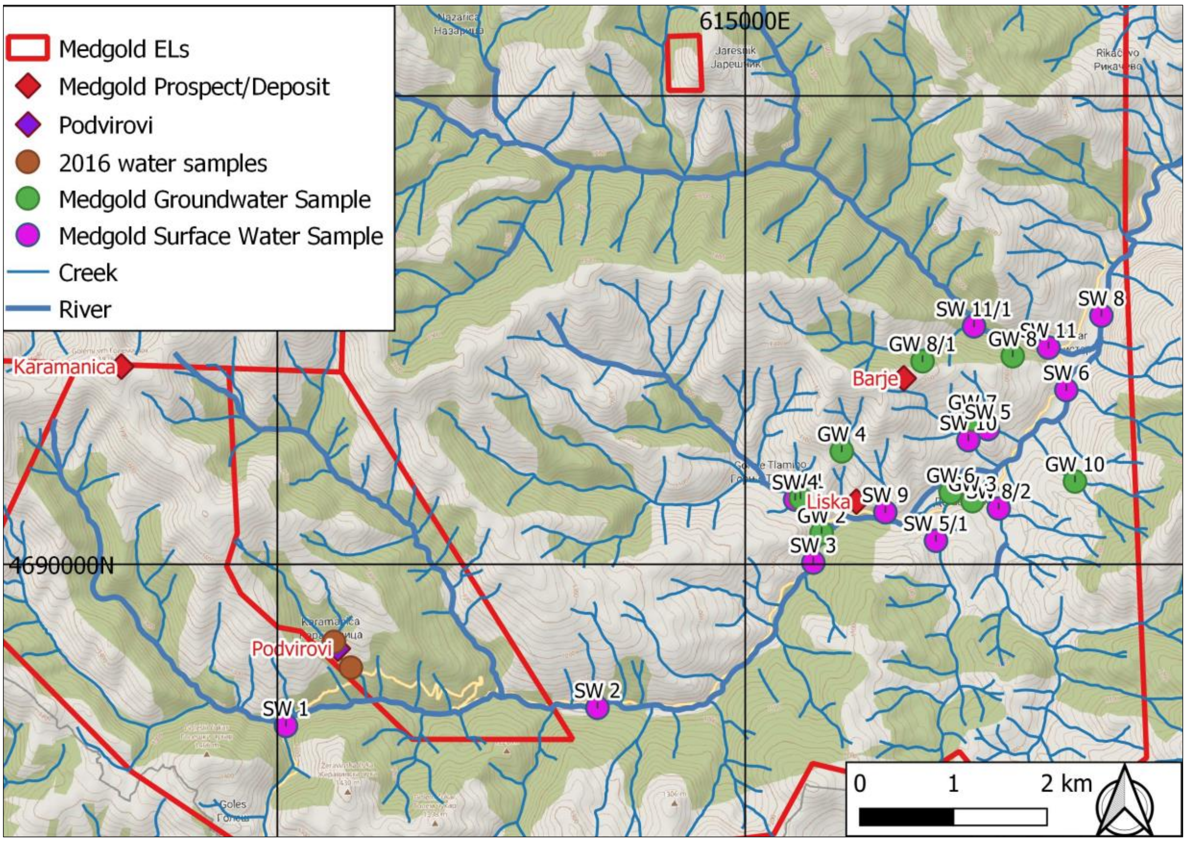


Figure 20.3: Location map of Medgold 2017 water sampling program.

Coordinates and a brief location description of the were recorded, and photos taken for each sample. In situ measurements included water temperature, pH, electrical conductivity, dissolved O_2 concentration, O_2 saturation, and turbidity. Samples of surface and groundwater were sent in refrigerators for physico-chemical, bacteriological and radiological analyses together with sediment samples from 6 of the surface streams.

Surface water temperature ranged from 8.7°C to 16.5°C reflecting ambient temperature, influenced by the time of day of measurement and general weather conditions. The groundwater temperature ranged from 7.0°C to 15.3°C. Electrical conductivity is high due to dissolved mineralization. All surface water was well aerated with a dissolved oxygen content of 8.5 mg/l to 10.52 mg/l and oxygen saturation 79% to 96%. In groundwater, oxygen content ranged from 5.02 mg/l to 10.66 mg/l. All sampled water was circum-neutral pH (pH 6.5- 8.5); surface water samples all reported above pH7, including one alkaline sample at pH 8.59; five of the nine groundwater samples were below pH7 (weakly acidic).

Concentrations of bicarbonate in surface waters was generally higher than groundwater from 48.8 mg/l to 232.4 mg/l. Measured sulphate concentrations ranged from 7.5 mg/l to 54.9 mg/l in surface waters and from 5.0 mg/l to 23.2 mg/l in groundwater. Groundwater belongs to the calcium bicarbonate type of groundwater according to the Piper diagram.

Of the potential contaminants, chemical analyses showed low Cu, Ni, Cd, Se and Sb in all water samples; and below limits for Fe, Mn, Cr and Pb. Barium and Sr had some elevated values, but there are no national limits set in legislation for these elements. Arsenic was found above the national limit of 10 µg/l in one of the streams draining from the Barje Deposit area at 87.1 µg/l and from two site groundwater samples (54.5 µg/l and 403.6 µg/l respectively).

Sediment samples collected from six of the watercourses were sent for chemical analysis. Concentrations of As in all samples except one exceed the limit value prescribed by the Regulation on limit values of pollutants in surface and groundwater and sediment ("Official Gazette of RS", No. 50/2012). High concentrations of As (169.8 and 508.6 mg/kg) were found in 2 sediment samples from the proposed TSF valley, (SW5 and SW10) along with an elevated Zn result in one sample (1717 mg/kg compared to a regulation limit of 430 mg/kg). Two sites upstream from the Barje site (SW4 and SW9) had concentrations of Ni that exceed the limit (possibly from adjacent mining activities).

20.3.11 Biodiversity

The high mountain areas of Serbia, which includes most of the municipality of Bosilegrad, represent one of the 6 centres of European and one of 158 centres of world biodiversity. The subject area includes: areas of exceptional national and international importance from the aspect of bird protection (IBA-Important Bird Areas), NATURA 2000 network of protected natural areas, Emerald network, preliminary IPA-areas (Important Plant Areas) - of special importance for plants , PBA areas (Prime Butterfly Areas in Serbia) - selected areas for diurnal butterflies, as well as Ramsar sites - internationally important wetlands, of equivalent priority for protection and management.

20.3.12 Vegetation

Initial desktop study of the existing vegetation in the area has shown the importance of the forest resources. This identifies the main flora communities and eco-systems. The Project landscape has an altitude range of 700 m to 1753 m with many communities - forest, meadow, peat near springs and numerous wetland and aquatic ecosystems, including hygrophilous-gley and organogenic-wetland peatlands. Research on the flora of this area to date has shown over 500 plant taxa, classified into 260 genera and 70 families, including some endemic plants in the Bistrica basin, alpine and sub-alpine vegetation and some rare Allium plant species. Analysis of plant communities reveals the presence of 40 flora associations. The Rudina mountain, approximately 50 km to the north of Barje, includes one plant species, Allium paczoskianum L., which, while not protected by law, is listed in the Red Book of Flora of Serbia. On the same mountain there is also Helichrysum plicatum, a species listed amongst strictly protected wild species of plants, animals, and fungi and protected by law, an environmental impact assessment must contain mitigation measures to protect such listed species should they be identified within the vicinity the Barje prospect.

20.3.13 Fauna

No information has yet been gathered on fauna in the Project area and local surveys for mammals (especially bats), birds, and insects will need to be undertaken. There is some publicly available data on wildlife regionally and in the local area, from academic papers and conservation websites. A compilation of data on amphibian and reptiles of the area indicate that 12 species of amphibians may be found in the Bosilegrad region, out of 21 species found in the Republic of Serbia. For reptiles, the data indicate that 11 species are present in the region, out of 23 found in Serbia. As this survey was undertaken in summer only, the list is unlikely to be exhaustive and other species of amphibians are expected in the upper reaches of the local rivers. Affiliation of the Dragovištica and Bistarska rivers to the Aegean basin suggests the importance of aquatic biodiversity.

20.3.14 Protected Areas

There are several large Protected areas to the north of the Project area, but partially in the municipality of Bosilegrad. There are three spatial units, part of which belong to the municipality of Bosilegrad, that include protected natural assets, the Vardenik mountain area; Besna kobila mountain with flora and faunal richness; and areas of Dukat mountain, 5 km north of the Project, including Golemi peak, that stand out for the Crimean pine population, rich flora and birds. In the vicinity of the Project there are three protected natural assets, a special nature reserve "Jarešnik"; and natural "Black pine" areas in Petkovska mahala and in Crnoštica.

The most important area is the strict nature reserve "Jarešnik", declared as such in 1961, located on Dukat mountain, 5 km north of the Project (Figure 20.4). The site was named after the village of Jarešnik and is protected for the black pine forest (Pinus nigra Subsp. Pallasiana), the last in the region. The forest stand is at an altitude of 1275 to 1350 m and is exposed to the southwest. On the Dukat mountain, there were individual black pine trees near the village of Crnostica and elsewhere.

These mosaic stands in the area are remnants of forest that was, in the past, widespread in Eastern Serbia. The forest is under protection of the municipality of Bosilegrad, but obligations under the Law on Nature Protection are not fulfilled and the facility is not marked. Destruction of this forest began in the period when this area was part of Bulgaria, when the state distributed pine trees to the inhabitants of Jarešnik, instigating the mass felling of pine trees. The current status of this protected forest will have to be determined and measures taken to prevent any adverse impacts from the Project.

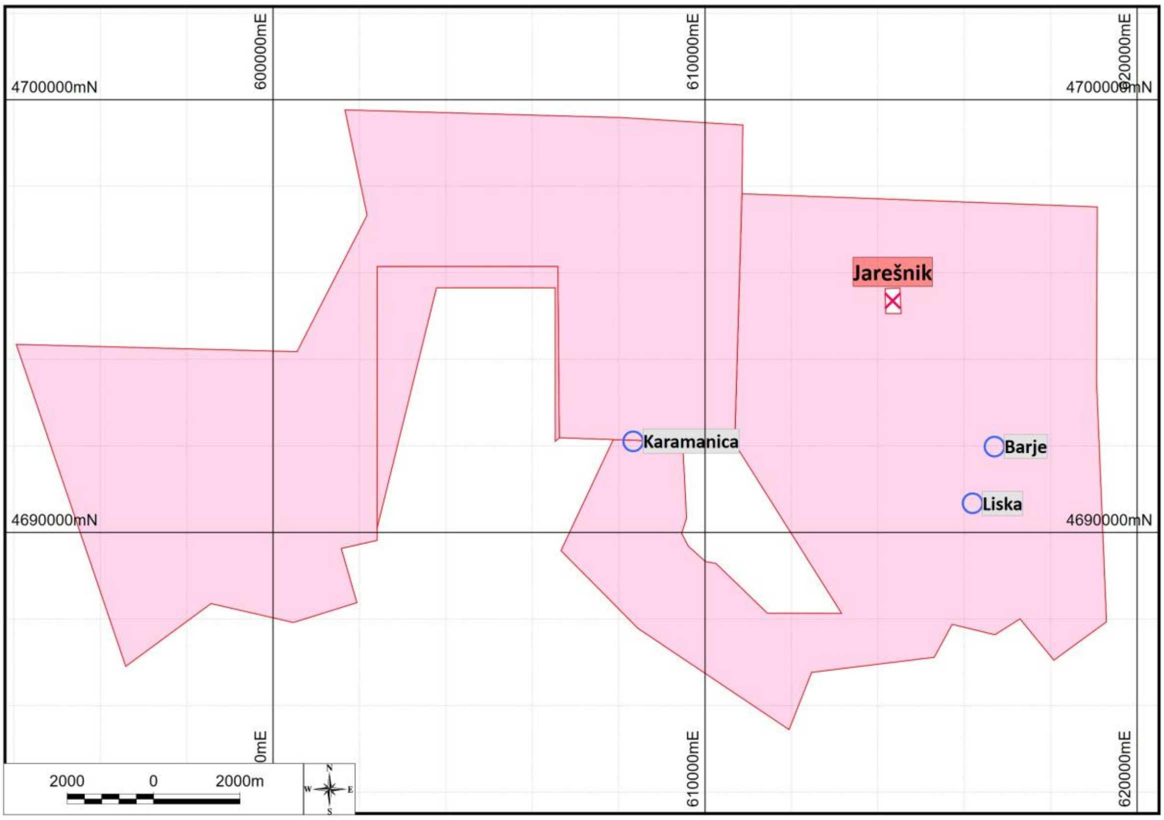


Figure 20.4: Location of Jarešnik protected area.

20.3.15 Socio-Economics

National, regional, and local statistics and demographics have been obtained from public records with available population census data and local municipal reports. The municipality town is Bosilegrad, approximately 22 kilometres north of the Property. Bosilegrad serves a population of around 8,000 people in the municipality and hosts an array of services including a healthcare centre, a police station, primary and secondary schools, postal and banking services, a fuel station, a hotel and shops and restaurants. Approximately 2,530 people (32% of the municipality population) live in Bosilegrad town, while the remaining population are scattered across approximately 35 small rural settlements and individual farms and properties. Settlements are organized in 38 cadastral units, with seven local offices. There has been a significant decrease in the municipal population over the last 60 years, but the proportion of people living in the town has increased with the on-going migration from rural settlements, similar to the trend of urbanization across Serbia.

Emigration away from the municipality has been predominantly amongst the young, leading to a significantly aging population, particularly in the rural settlements; and a reduction in the number of people per household. Despite a uniform gender structure, natural regeneration of the population is unlikely. The majority (71%) of the inhabitants of the municipality are of Bulgarian ethnicity, with 13% Serbs and the remainder people from Macedonia, Montenegro and Roma. Educational achievement is poor with only 25% of the population completing secondary school and less than 4% with a higher education. Health services in the municipality is through the Health Centre in Bosilegrad and primary level health care in seven rural settlements and an ambulance service. Administration and socio-political organizations are concentrated in the city area, including the Municipal Assembly, the Court, post office, banks and customs. Six villages settlements have local offices and three have post offices. Bosilegrad provides most of the sporting and cultural facilities. There is one church in the town and a further 30 religious buildings in the area of 25 villages.

Economic decline began in the 1990s and there has been a period of stagnation for the last 20 years with lack of development, modernisation or investment in the area. The municipality lags significantly behind in all economic indicators compared to the rest of Serbia and unemployment is high. Of the 1,417 employees in the municipality, nearly 60%, are in non-economic services activities including education, health and administration. Manufacturing; retail and wholesale; transport and storage; electricity, energy, gas and water services; agriculture, forestry and water management; construction; hotels and restaurants and mining and quarrying make up the rest. While the largest share of non-employed activity is in agriculture, hunting and forestry, the amount of people in agriculture has declined significantly since the 1970s.

The main neighbour villages to the Barje Deposit area, Donje Tlamino and Bistar, are located in the valley of the Golema River, and consist of only a few dwellings, most of which are unoccupied or only used during summer months. While there are no villages within the Project area, there are a few farms that may need to be relocated, land purchased, and affected people compensated. The main industry in the area is forestry and related processing of timber products, as well as new mining activity.

20.3.16 Archaeology and Cultural Heritage

Initial archaeological and cultural heritage study has included desktop research of available data from the databases of Local Institute for Monument Protection in Niš and of the Central Institute for Monument Protection in Belgrade; as well as analysis of satellite images from Google Earth, Bingand GeoSrbija databases to detect under/above-ground anomalies that may signal possible presence of unregistered remains of archaeological/cultural heritage sites and monuments in the study area.

There are 21 registered cultural monuments in the whole Pčinja region and one spatial cultural-historical unit, all of them far outside of the Project site area, with the closest over 80 km from site. However, the fact that there is not a single cultural-historical site/monument recorded by Central Registry in the area should be taken with caution, as there has been little research and the region is one of the most archaeologically under-investigated zones in Serbia.

The law on the protection of culture states that all unknown, unlisted, or unregistered sites and monuments have the status of previous/prior protection and have a rank equal to registered monument for up to 1 year after discovery. For example, should a discovery be made, Medgold must notify the Cultural Monument Institute, after notification a period of one year is available for the institute to declare protection and initiate protective measures, should no action have been taken by the institute at the end of the one year period no case for protection is necessary. There are two ongoing projects of the Ministry of Culture and Information to collect all unpublished data on position and characteristics of archaeological sites in Serbia with a network of cultural institutions involved in collecting data. Preliminary results show that there are at least 20 times more archaeological sites than those that have been documented in the Central Registry.

Satellite images of the Project area shows no clear signs of archaeological or other structural remains either beneath- or above-ground structure ruins. The highest potential for presence of invisible, and non-registered archaeological sites is in the valleys of small rivers (such as the Golema Reka, Mala Reka, Strov Dol, Pojište and Klisura) and on spacious flattened plateaus beneath 1000 m elevation.

The geographical and historical position of Donje Tlamino indicates that several types of sites could be expected, including:

- Palaeolithic sites of the first modern humans in the rock shelters and caves in the mountains;
- Prehistoric mine workings could be expected in areas with occurrence of near surface copper ore; and historical mine shafts on ores of copper, iron, silver and lead;
- Iron Age and Medieval hill forts or refuge sites on flat mountain plateaus;
- Individual burial sites and graveyards of any period;
- Enclosures of stockbreeding communities from any period; and

- Individual hoards from Hellenistic and Roman times.

The Regional Institute for Protection of Cultural Monuments in Niš is responsible for issuing the permissions that are required for exploration rights in the Medgold Resources area in Donje Tlamino.

20.4 Stakeholder Engagement

Medgold has an established operational base in the town of Bosilegrad and has been engaging with the municipality and local communities since the start of exploration activities. The Issuer has a Community Policy that states:

> *Medgold believes that a social "license to operate" is essential in developing our exploration projects. We are committed to engaging transparently with the communities who may be affected by our exploration activities and we will keep local communities regularly informed about our work-plans.*
>
> *We will develop a set of guidelines defining the company's internal requirements for community engagement and we will make these a core principle of the way we work.*
>
> *We will provide simple methods for community stakeholders to contact us. We will record and respond to community concerns in a timely and fair manner.*

The Medgold Community and Stakeholder Engagement Guideline outlines commitment to engaging with the communities and stakeholders in the Bosilegrad area and at national level, using appropriate, effective and inclusive practices during project exploration and development phases. Medgold has initiated a stakeholder mapping exercise to identify all the groups that may be affected by Project. While Medgold pursues open disclosure of activities, the Project is still in exploration phase, and as such this is subject to limitations. Medgold does however meet with affected local stakeholders that may be impacted by Project prior to drilling programmes at least once per year to understand the expectations, needs and priorities, as well as whenever community members express an interest to meet.

Medgold information disclosure is supported by the following:

- The Medgold Resources website (English and Serbian versions)
- Community meetings and presentations
- Meetings with local and national government officials
- Press Releases and Media Relations
- Conference presentations, and
- Legal mandatory engagement with Ministry of Mines and Energy.

Medgold has established a Grievance Mechanism based on international expectations. In summary, on receipt the grievance is recorded in a register, and is then screened and assessed to determine the appropriate action, with the proposed action communicated to the complainant. If the proposed action does not resolve the issue, further re-assessment is undertaken. A timeframe for these procedures should be defined, acknowledging that flexibility is required to address special circumstances that may require independent 3rd party involvement. Grievances may be submitted to Medgold by letter, by e-mail, or by telephone to Medgold's Community Relations advisor. All such contacts are readily available to the stakeholders.

20.5 Community Assistance

Medgold has also established a Community Assistance Programme, which is aimed at communities affected by the Project. This programme includes the preferential local hiring of labour and services when suitable capabilities are present, assistance to locally registered charitable organizations, and community development in partnership with local stakeholders.

20.6 Closure

A closure plan and cost estimate must accompany the application for the Mine Building Permit along with a bank guarantee, bill of exchange or corporate guarantee equal at least 30% of estimated closure cost. At this conceptual stage of the Project, it is assumed that the LOM assets will include an open pit, waste rock storage facility and a tailings storage facility that will require rehabilitation. Offices, processing plant, workshops and other infrastructures will have to be demolished and footprints scarified and revegetated. Closure costs of similar sized mines in Europe and Africa are around US$2-4M, but costs will largely depend on post-closure land-use and the level of degradation and contamination requiring remediation and/or long-term management.

20.7 Project E&S Risks

At this early stage of Project development, the risks to and from the proposed mining of the Barje Deposit are as follows:

- Positive approvals and timely permitting.
- Early quantification of potential geochemical issues to inform Project design.
- ARD acid generating potential is not yet known, but high pyrite content suggests this may be an issue.
- Arsenic levels are naturally high in the rock material and is already identified in water and sediment sampling and analysis.

- No detailed studies have yet been undertaken on fauna or flora of the area and field surveys will be essential to identify species of particular conservation concern.
- The Jarešnik black Pine forest area and other similar forests and tree copses will need to be investigated to establish whether they are likely to be impacted by the Project development.
- Early investigation of the farms and households that will be directly affected by the Project footprint and exclusion zone will input to the required Resettlement Action Plan.
- Special attention will have to be given to the potential cumulative impacts with other current- and future mining activities in the area, and further to potential Transboundary impacts to neighbouring Bulgaria.

If the Project were to proceed it would undoubtedly make a positive impact on the existing social problems of unemployment, rural- and municipal de-population and lack of investment and community development.

21 Capital and Operating Costs

Scoping level estimates (±30%) of capital and operating costs relating to the Project have been developed in the course of this study.

21.1 Capital Costs

Capital costs for mine development, mine infrastructure, process plant, and surface infrastructure including mine offices, control, plant building, common workshop and stores, changehouse, water, powerline and substation, and earthworks including tailings, roads and platforms are projected. Estimates have been made based on current designs and quotes from recent, similar projects by Bara Consulting. Details are presented in Table 21.1. Estimates for closure were also assessed during the ESIA review process.

Table 21.1: Capital Cost Estimates, 625,000 tpa Barje Au-Ag Project

	DESCRIPTION	**QTY**	**UNIT**	**TOTAL COST US$**
1	MINE DEVELOPMENT			
	Site prep/pre-strip/site establishment	3,252,382	t	7,480,478
2	PROCESS PLANT			
	Flotation plant	650,000	tpa	34,599,124
3	SURFACE INFRASTRUCTURE			
	Surface infrastructure and buildings	1	lot	5,000,000
	TSF	2,236,306	m^3	4,000,000
	Closure	1	lot	5,000,000
4	INDIRECT COSTS			
	Exploration, Engineering, Permitting	15	%	8,411,940
5	CONTINGENCY			
	Contingency	15	%	9,673,731
6	TOTAL			74,166,973

Mine capital includes the initial development of haul roads, and site preparation for the plant terrace, as well as a WRSF, LG stockpiles and TSF. Plant capital provides for the design and construction of a 600,000 tpa flotation plant including crushing, grinding, froth flotation, concentrate and tailings handling facilities including filtration of tailings for dry stacking. Infrastructure includes for mine support infrastructure, plant infrastructure, dry stack tailings storage facility, power (including backup 35kV line), water and internal roads.

Additional capital amounts are allowed for closure costs per the 2020 assessment of US$5M.

21.2 Operating Costs

A high-level breakdown of operating costs is provided. Estimates have been made based on current designs and quotes from recent similar projects by Bara Consulting. Mine operating costs include ore mining and waste mining at US$2.30/t, plus a stockpile reclaim cost for LG material of US$1/t equating to US$0.50/ROM tonne.

Process costs include crushing, grinding, flotation, concentrate handling and tailings handling (including filtration) for 600,000 tpa flotation feed. Process cost estimates are based on in country power costs of US$0.11/kWh, water costs of US$0.70/$m^3$, plus current estimates of reagent costs and grinding media. Applicable labour rates for local and expatriate labour were also used. G&A includes on mine administration and general costs. Concentrate transport is costed for delivery of concentrate CIF to customers in China. Details are presented in Table 21.2 below.

Table 21.2: Operating Cost Estimates, 600,000 tpa Barje Au-Ag Project

	DESCRIPTION	QTY/Mt	UNIT	COST US$ / UNIT	TOTAL COST US$M
1	**MINING**				
	Mining cost – ROM	5.69	t	2.80	15.9
	Mining cost – Waste	26.5	t	2.30	61.0
2	**PROCESSING**				
	Processing HG	3.57	t	11.50	41.0
	Processing LG	2,11	t	11.50	24.3
	Conc Transport (/ROM t)	5.69	t	3.24	8.42
3	**GENERAL AND ADMIN**				
	G&A (/ROM t)	5.69	t	5.80	33.0

22 Economic Analysis

The economic analysis presented in this report is preliminary in nature and is based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is therefore no certainty that the preliminary economic assessment presented here will be realized.

Economics are based on the mining schedule as presented in Section 16 - Mining Methods. A mining cost of US$2.30/t is applied, with flotation costs at US$11.50/t including tailings handling. G&A is applied at US$5.80/t. Mining capex is estimated at US$7.5M, with surface capital including process plant and tailings at US$33M plus Closure cost at US$5M per the breakdown in Table 22.1. A long-term Au price of US$1500/oz and a silver price of US$16.50/oz as per LMBA data were used. Metal payability is 75% for HG and 40% for LG, both net of treatment charges. A summary is presented in Table 21.1, with details presented in Appendix 1, Financial Model.

Table 22.1: Barje Project Key Financial Metrics

Revenue	**458**	**US$M**
Operating Cost	181	US$M
Project Capital Cost	74	US$M
Free Cashflow	153	US$M
LOM C1 Cash Cost	464	US$ / oz
LOM AISC	522	US$ / oz
Pre-Tax Project NPV8	101	US$M
Post-Tax Project NPV8	86	US$M
Pre-Tax Project IRR	49	%
Post-Tax Project IRR	46	%
Operating Margin	61	%
Peak Funding Requirement	37	US$M
Payback Period	2.0	years

The NPV of the project, at a discount rate of 8%, is US$101M with an IRR of 49%, and the operating margin, describing an extremely robust project, is 61%. Upfront capital is US$64M, plus US$10M contingency, with peak funding of US$37M and a payback of 2 years. Life of Mine C1 cash costs of US$464/oz, and LOM AISC of US$522/oz, would place the project - if operating - in the current lowest quartile cost of global gold production.

22.1 Sensitivity

Sensitivity analysis of key capital and operating cost parameters, and gold price indicates significant upside potential to the project are shown in Figure 22.1. The Project was demonstrated to be most sensitive to variance in gold price, and least sensitive to variances in capital cost. Specific post-tax NPV and IRR sensitivity ranges are presented in Table 22.2.

Table 22.2: NPV and IRR sensitivities, Barje Prospect

Variance	Gold Price US$/oz	NPV (8%)	IRR	Capital Cost (US$M)	NPV (8%)	IRR	Operating Cost US$/t	NPV (8%)	IRR
-30%	1050	10	12	52	102	72	24	118	63
-25%	1125	23	18	56	99	66	26	112	60
-20%	1200	36	23	59	97	61	27	107	57
-15%	1275	48	29	63	94	57	29	102	54
-10%	1350	61	34	67	91	53	31	96	51
-5%	1425	73	40	70	88	49	32	91	49
0%	1500	86	46	74	86	46	34	86	46
5%	1575	98	52	78	83	43	36	80	43
10%	1650	110	57	81	80	40	37	75	40
15%	1725	123	63	85	77	38	39	69	38
20%	1800	135	69	89	74	36	41	64	35
25%	1875	147	76	93	71	34	43	59	32
30%	1950	160	82	96	69	32	44	53	30

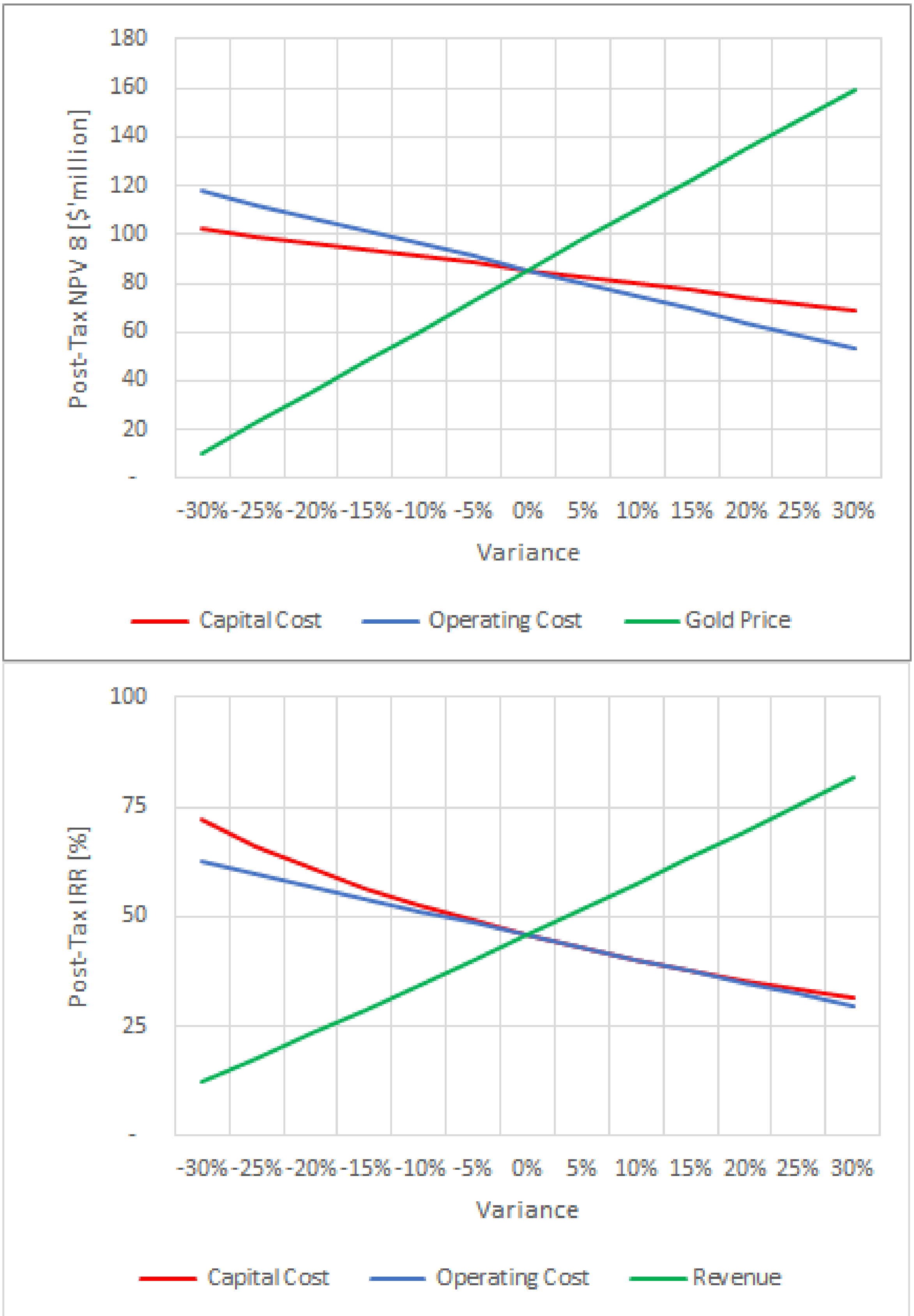


Figure 22.1: Post-Tax NPV and IRR Sensitivity, Barje Deposit

23 Adjacent Properties

Adjacent to the Property are a 6.6 km^2 EL and a 0.33 km^2 EL for lead and zinc (Figure 4.2) held by a Serbian subsidiary of UK-registered Mineco Limited (Mineco), a privately held company unrelated to Medgold.

Mineco appear to have completed a locally compliant mineral inventory estimate for approval by the Serbian Government in 2014 and commenced detailed mine planning and development works during 2015 and 2016 to access mineralization at Podvirovi and Popovica, linking these areas with a 1300 metre underground development drive and commissioning a metallurgical pilot plant for the trial production of concentrates during 2017 (Mineco Limited, 2019).

Former Yugoslav State agencies carried out drilling and underground development at the Podvirovi and Popovica Prospects during the 1950s and 1960s although full details of this exploration work are not known.

The qualified person for this Technical Report has been unable to verify the information relating to the adjacent properties and this information is not necessarily indicative of mineralization on the Property that is the subject of this Technical Report.

24 Other Relevant Data and Information

All relevant information and data are included in the appropriate sections of the report.

25 Interpretations and Conclusions

25.1 Preliminary Economic Analysis

A preliminary economic analysis has been undertaken for the Barje Deposit of the Tlamino Project. Economic analysis considers revenue based on the preliminary mining schedule presented in section 16, metal recoveries as presented in section 13, market factors as presented in section 19 and capital and operating costs as presented in section 21. Estimates and analysis are to scoping level (+-30%).

Capital costs for mine development, mine infrastructure, process plant, and surface infrastructure including dry stack tailings were estimated. These estimates have been made on the basis of current designs and quotes from recent, similar projects by Bara Consulting. Estimates for closure were also assessed during the ESIA review process. The total estimated capital cost for the Project is US$64M, plus US$10M contingency, with no material sustaining capital assessed on account of the nature of the pit and short life of mine.

Operating costs were also estimated to an appropriate level of accuracy for the study, based on mining and process designs developed. These estimates have been made on the basis on current designs and quotes from recent similar projects by Bara Consulting. Operating costs estimated mining (including stockpile reclaim cost for LG material), processing (including tailings disposal), G&A and concentrate transport cost. A summary is presented in Table 25.1 below.

Table 25.1: Operating Cost Estimates, 600,000 tpa Barje Au-Ag Project

DESCRIPTION	UNIT	COST / UNIT (US$)
MINING		
Mining cost - ROM	t	2.80
Mining cost - Waste	t	2.30
PROCESSING		
Processing	t	11.50
Conc. Transport (per ROM t)	t	3.24
General and Admin	t	5.80

A preliminary economic analysis has been completed based on these cost estimates in the context of revenue estimates based on the mining schedule presented, Au and Ag price estimates, and assessments of metal payabilities based on assessment of likely offtake customers and locations.

The NPV of the project, at a discount rate of 8%, is US$101M with an IRR of 49%, and the operating margin, describing an extremely robust project, is 61%. Upfront capital costs are US$64M, plus US$10M contingency, with peak funding of US$37M and a payback of 2 years. Life of Mine C1 cash costs of US$464/oz, and LOM AISC of US$522/oz, would place the project - if operating - in the current lowest quartile cost of global gold production. Note that this economic analysis is preliminary in nature and is based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is therefore no certainty that the preliminary economic assessment presented here will be realized. Key project financial metrics are presented in Table 25.2.

Table 25.2: Barje Project Key Financial Metrics

Revenue	**458**	**US$M**
Operating Cost	181	US$M
Project Capital Cost	74	US$M
Free Cashflow	153	US$M
LOM C1 Cash Cost	464	US$ / oz
LOM AISC	522	US$ / oz
Pre-Tax Project NPV8	101	US$M
Post-Tax Project NPV8	86	US$M
Pre-Tax Project IRR	49	%
Post-Tax Project IRR	46	%
Operating Margin	61	%
Peak Funding Requirement	37	US$M
Payback Period	2.0	years

25.2 Mineral Resources

The estimated Mineral Resource, reported in accordance with NI 43-101 and the CIM Definition Standards above cut-off grades of 0.6 g/t AuEq for high grade breccia, 0.8 g/t AuEq for low grade schist, and 0.5 g/t AuEq for partially oxidized material is approximately 7.1 Mt at 2.5 g/t Au and 38 g/t Ag in the Inferred category, and containing 570,000 oz of Au and 8.8 Moz of Ag. This equates to approximately 2.9 g/t AuEq or 670,000 oz AuEq. It is the opinion of the Company and Qualified Person that all elements included in the Au Equivalent calculation (gold and silver) have a reasonable prospect of being recovered and sold. The updated Mineral Resource estimate has an effective date of January 07, 2021, and supersedes the previous initial Mineral Resource estimate, there has been no material change to the Mineral Resource estimate in terms of tonnage, grade and contained metal. See Table 25.3 for further details relating to the Mineral Resource Estimate.

No estimates of Mineral Reserves have been completed. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.

The Mineral Resources extend from surface to approximately 110 m below surface, it is laterally extensive over an area of approximately 600 m from east to west and approximately 350 m north to south. The thickness of resource mineralization ranges from approximately 10 to 40 m with some isolated thinner areas. It is closed by bounding faults to the north and south and by drilling to the east and west. There is some possibility of identifying additional mineralization by infill drilling in areas where the model has interpreted to pinch and data are sparse, and in the north-west corner of the deposit.

Additional drilling is required to increase the confidence in the Mineral Resources, and as the level of such information increases, Mineral Resources may increase or decrease.

The Wireframe models used in Mineral Resource estimation are shown in Figure 25.1, the block mode is shown in Figure 25.2 with blocks above cut-off grade which qualify as a Mineral Resource.

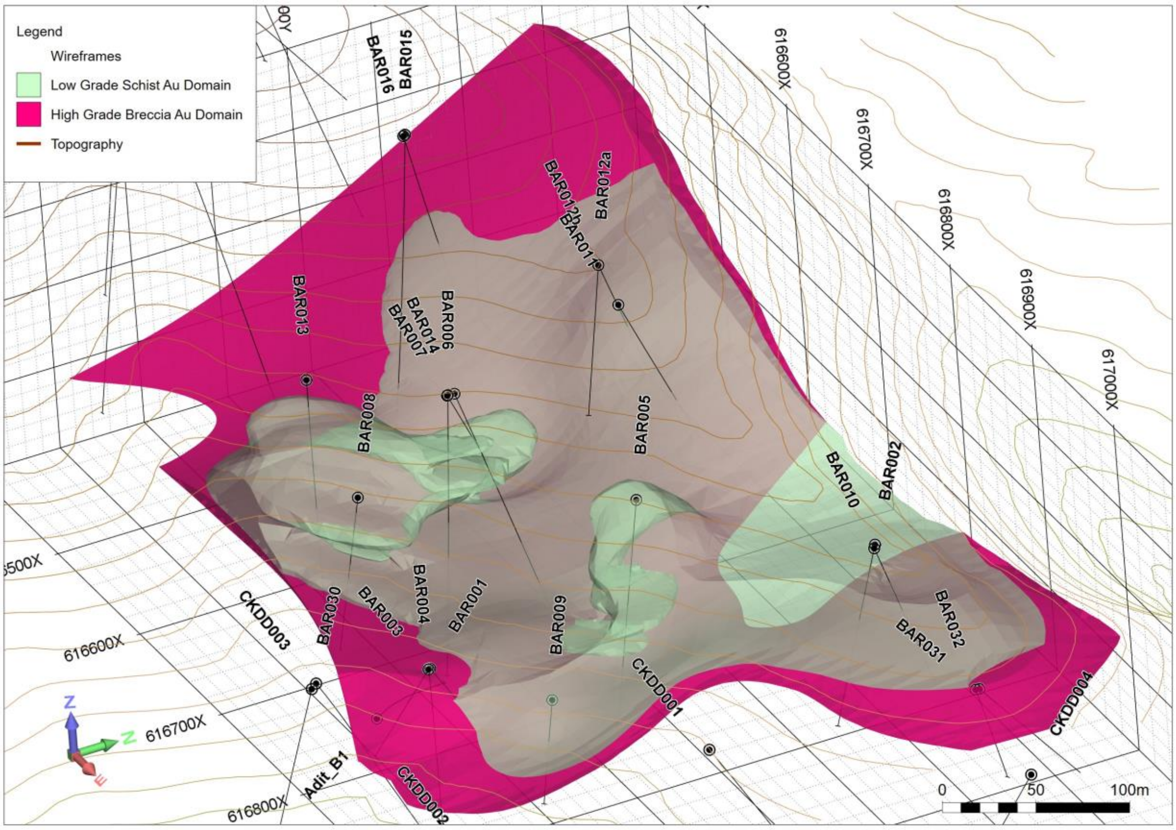


Figure 25.1: Wireframe mineralization model looking North West.

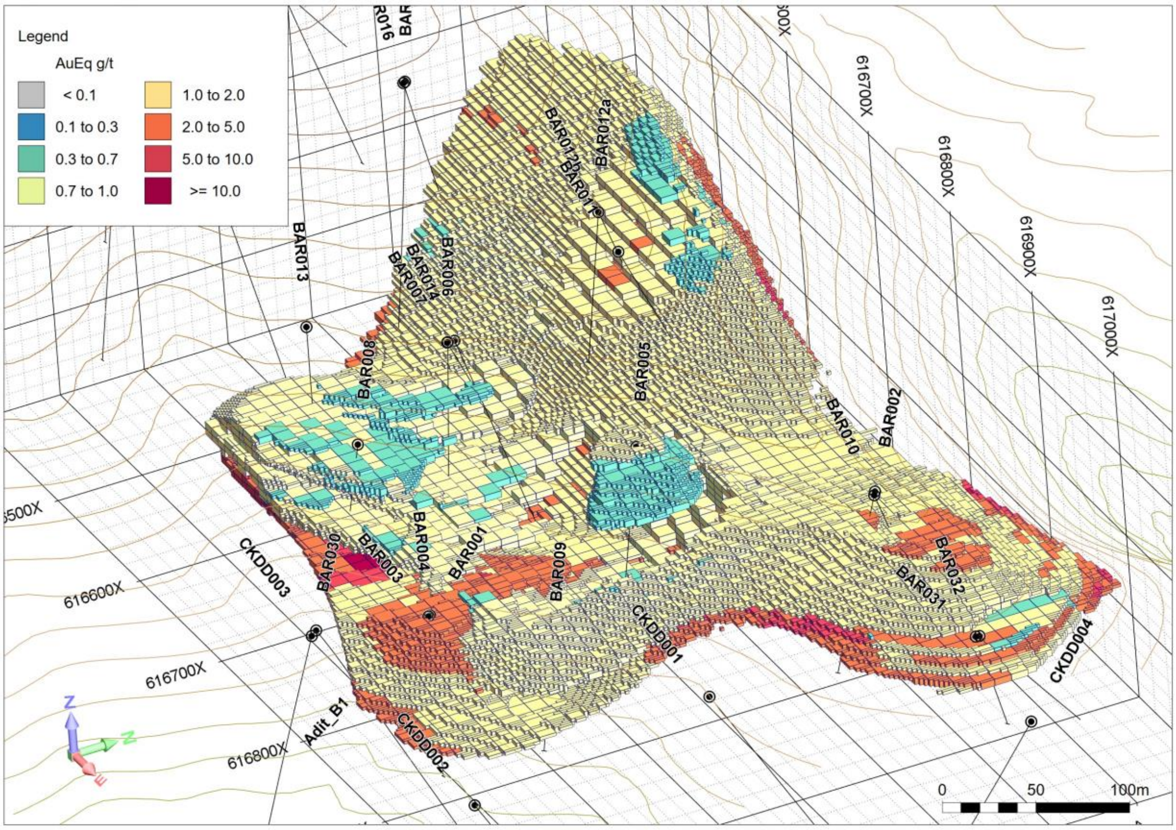


Figure 25.2: Block Model looking North West; 0.7 g/t AuEq cut-off applied.
Blocks shown form the Mineral Resource.

Table 25.3: Inferred Mineral Resource Estimate for the Barje Deposit

Tonnes	Density	AuEq g/t	AuEq oz	Au g/t	Contained Au oz	Ag g/t	Contained Ag oz
Total Inferred Resources							
7,100,000	2.7	2.9	670,000	2.5	570,000	38	8,800,000
Including							
High Grade Breccia							
3,200,000	2.8	4.7	470,000	3.9	400,000	65	6,700,000
Low Grade Schist							
2,400,000	2.7	1.2	96,000	1.1	88,000	8.4	650,000
Partially Oxidized Material							
1,500,000	2.5	2.1	100,000	1.7	87,000	29	1,400,000

Notes to the Mineral Resource Estimate:

13. The independent Qualified Person for the Mineral Resource Estimate, as defined by NI 43-101, is Mr. Richard Siddle, MSc, MAIG, of Addison Mining Services Ltd since November 2014. The effective date of the Mineral Resource Estimate is January 07, 2021.
14. These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The quantity and grade of reported Inferred Resources in this Mineral Resource Estimate are uncertain in nature and there has been insufficient exploration to define these Inferred Resources as Indicated or Measured, however it is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
15. Mineral Resources reported in the above table are presented as undiluted and in-situ for an open-pit scenario and are considered to have reasonable prospects for economic extraction. The Mineral Resources constrained by open pit optimisation.

16. Break even cut-off grades were estimated for each material type of 0.6 g/t, 0.8 g/t and 0.5 g/t AuEq for the High Grade Breccia, Low Grade Schist and Partially Oxidized materials respectively, these cut-off grades were used in Resource Reporting. The cut-off grades were calculated on the basis of the following assumptions: a gold price of US$1500/oz, a silver price of US$16.5/oz, mining costs of US$2.3/t, processing costs including tailings disposal of US$10/t for sulphide rock and US$12/t for oxide, G&A costs of US$4/ROMt and transport costs of US$2/ROMt.
17. Per metallurgical test work completed to date, recovery to concentrate after flotation of 85.8% for gold and 84.3% for silver were used for the High Grade Breccia material with 75% payability. For the Low Grade Schist recoveries used were 76.5% for gold and 82.7% for silver with 60% payability. For the Partially Oxidized material 80% recovery via leaching for gold and silver was assumed with 98% payability. 5% gross royalty was applied to both metals.
18. Geological and block models for the Mineral Resource Estimate used data from 33 surface drillholes performed by Medgold in 2018 and 2019; data from four drillholes completed by Avala Resources Ltd., a prior operator, were used to constrain the model though they did not intercept significant mineralization. The drill database was validated prior to resource estimation and QA/QC checks were made using industry-standard control charts for blanks, core duplicates and commercial certified reference material inserted into assay batches by Medgold and by comparison of umpire assays performed at a second laboratory. No QA/QC was possible on the data relating to the drilling by Avala.
19. The geological model as applied to the Mineral Resource Estimate comprises two mineralized domains, a shallowly inclined high-grade hydrothermal breccia unit and a lower-grade schist unit immediately overlying the hydrothermal breccia. Individual wireframes were created for each domain. Weathering domains of fresh and partially oxidized material were defined within the two mineralised domains.
20. The block model was prepared using Micromine version 2020, Services Pack 1, A 10 m x 10 m x 4 m block model was created with sub-blocks of minimum 2 m x 2 m x 2 m on domain boundaries. Grade estimation from drillhole data was carried out for Au, Ag, As, Cu, Pb, Zn, Fe, S using Ordinary Kriging and was validated by comparison of input and output statistics, kriging neighbourhood analysis and by inspection of the assay data and block model in cross section. A gold equivalent (AuEq) grade was calculated for each block using the formula AuEq = ((Ag g/t) x 0.011)) + (Au g/t) for the High Grade Breccia and Partially Oxidized materials and AuEq = ((Ag g/t) x 0.012)) + (Au g/t) for the Low Grade Schist.
21. Bulk density values were calculated for each block of the model based on a broad linear relationship observed between 152 measured bulk density values within the mineralized domains and the assayed values of As, Cu, Fe, S, Pb and Zn. Blocks within the partially oxidized material were assigned a single bulk density value of 2.54 g/ cm^3.
22. Estimates in the above table have been rounded to two significant figures.
23. CIM Definition Standards for Mineral Resources have been followed.
24. The independent Qualified Person for Resources is not aware of any additional known environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues that could materially affect the Mineral Resource Estimate.

25.3 Data Quality and Verification

The Qualified Person for Geology and Resources, Mr Richard Siddle MSc, MAIG, has reviewed the drilling data collected by Medgold and considers it reliable for use in Mineral Resource estimation. The data collection practices are considered in line with current industry best practices. Regional exploration data are considered suitable for the generation of follow up drill targets. Mr Siddle completed a site visit to the Tlamino Project in November 2019. Due to restrictions relating to the COVID-19 pandemic in force during the study, no other QP was able to visit the site in support of the PEA.

25.4 Mineral Processing and Metallurgical Testing

A recent programme of laboratory testing has investigated the recovery of gold and silver from fresh HG_BX, LG_Sch and weathered OX material types by a variety of means, work which builds on preliminary testing by Medgold in 2019. The mineralogical work has shown the gold to be fine-grained and generally not present as free gold. When ground to a P_{80} of 80 µm the liberation of sulphides was favourable, with approximately 60% of the sulphide grains occurring being liberated and a further 25% to 30% occurring as binary particles in association with other sulphides.

The HG_BX Fresh and LG_Sch Fresh materials showed highly positive responses to rougher and cleaner flotation, the HG_BX material yielding a concentrate grading 48.9 g/t Au and 824 g/t Ag at recoveries of 83.4% and 86.8% respectively. The concentrate from flotation of LG_Sch material graded 24.4 g/t Au and 228 g/t Ag and contained 71.2% of the gold and 79.2% of the silver. The OX material displayed a poor response to flotation, only 28.7% of the gold and 63.6% of the silver being recovered to a rougher concentrate.

Use of additional reagents to improve the kinetics of flotation from the LG_Sch material and to depress lead showed no material differences from the baseline flotation tests.

Gravity concentration was not beneficial, either on its own or in combination with flotation.

In bottle roll tests, cyanide leaching of the OX extracted 80.5% of the gold and 81.6% of the silver after grinding to a P_{80} of 78 µm and 72 hours of leaching. A flowsheet was developed based on the results of the testwork and costed to scoping levels of accuracy. Mass and metal balances developed in tested were used in discounted cashflow analysis.

Table 25.4: Summary of Mineral Processing Parameters used in the PEA base case.

Parameter	Units	Value
Flotation Throughput	Ktpa	600
Au Recovery HG_BX	%	85.8
Ag Recovery HG_BX	%	84.3
Au Recovery LG_Sch	%	76.5
Ag Recovery LG_Sch	%	84.3
Mass pull	%	5
Au grade HG conc	g/t	49
Ag grade HG conc	g/t	824
Au grade LG conc	g/t	24
Ag grade LG conc	g/t	238
Flotation Process Costs - OPEX	US$/processed t	11.50
G&A	US$/processed t	5.80
Concentrate Transport Cost	US$/t processed t	3.24

25.5 Mining

A relatively detailed pit design with associated stage design and schedule has been developed from the Barje geological and resource model. Additional detail was required on account of several factors including the unusually shallow flat lying nature of the deposit, low RQD in the wall materials, as well as the low payability of LG_Sch zones overlying the higher grade HG_BX zones with higher payability. The resulting pit design describes a low-cost operation with relatively low stripping ratios, a small, cost effective mining fleet and modest labour charges, producing high initial grades of both Au and Ag during the critical capital payback period. Total mining inventory is 5.69 Mt at 2.62 g/t Au and 38.9 g/t Ag overall. A total of 26 Mt of waste is generated over the Life of Mine, however adequate storage capacity has been found in valleys to the East and North of the proposed pit. Detailed geotechnical and hydrogeological work will be required for subsequent phases; however, designs and cost estimates are considered reasonable and relatively low risk considering reasonable assumptions for both geotechnical and hydrological factors made.

25.6 Environmental and Social

Compilation of desktop data and initial baseline studies have been initiated at the Project to inform the Scoping level Environmental and Social Impact Assessment. These early studies have identified potential risks to and from proposed mining of the Barje Deposit, including obtaining approvals and timely permitting; possible geochemical issues from ARD acid generation and metal leaching, especially of arsenic; biodiversity impacts to fauna and/or flora species of conservation concern, particularly the identified Jarešnik black Pine forest area and other similar forests; social impacts to Project Affected People especially those that will require resettlement, buy-out and compensation; potential cumulative effects from other current- and future mining activities in the area; and potential Transboundary impacts to neighbouring Bulgaria. If the Project were to proceed it would undoubtedly make a positive impact on the existing social problems of unemployment, rural- and municipal de-population and lack of investment and community development.

25.7 Exploration Potential

Potential exists for a laterally faulted offset of the Barje mineralization to be present on the EL. Additional geological mapping and interpretation may assist in exploring for a faulted continuation of the mineralization. Drilling at Liska has identified the presence of mineralization, however, the metal grades identified are not considered to be economically significant, or where potentially economic, are currently interpreted to be isolated with a lack of demonstrated continuity. At Karamanica, only weak mineralization was intersected associated variously with fault zones, dark carbonaceous schists, and the margins of porphyritic intrusions. Several geochemical and geophysical targets remain un-drilled on the Prospect and further exploration is warranted.

26 Recommendations

Recommendations include commencing infill drilling work relevant to support potential next-step conversion of Inferred to Indicated Resources, and field programmes in support of a potential Preliminary Feasibility Study (PFS) on the Barje Deposit. This work would include additional geotechnical and hydrogeological investigation, additional metallurgical testing and commencement of environmental base line studies including air, water, soil, fauna and flora.

26.1 Geology and Resources

In regard to increasing confidence in Mineral Resources at the Barje Deposit, infill drilling on approximately 60 m centres is recommended as shown in Figure 26.1. Targeting should focus on pierce point spacing of the mineralized zone rather than collar spacing with optimum dip and azimuths carefully planned to account for the variable geometry of the deposit. Some potential for additional resources exists as indicated in Figure 26.1. Approximately 3200 m of drilling is recommended; results of the drilling should be periodically re-evaluated during the programme to test confidence and appropriateness of drill spacing. Dedicated drillholes for geotechnical and hydrogeological purposes would ideally be completed while the drill rig is on site to avoid remobilisation costs.

While quality control procedures employed in previous Medgold drill programmes have been robust it is however recommended that additional core duplicate samples be targeted towards zones of visible mineralization. Continued and routine bulk density determinations together with routine logging, sampling and quality control procedures is recommended. Structural analysis from orientated core has provided limited useful information to date and is not considered a necessity for geological interpretation, however some drillholes should be orientated to support full geotechnical logging. A topographic survey of increased accuracy is also recommended over the Prospect area.

Potential exists for a laterally faulted offset of the Barje mineralization to be present on the EL. Additional geological mapping and interpretation may assist in exploring for a faulted continuation of the mineralization. Drilling to date at Karamanica Prospect has been limited to targets identified mainly by interpretation of geochemical and geophysical datasets. A reassessment of the Karamanica Prospect, considering the additional information gained by drilling in 2019, may lead to additional targets on the Prospect which, if present, would require additional drill testing.

Additional mineralogy and paragenetic studies may also be useful in understanding the genesis of the deposit which may bolster metallurgical studies and improve the understanding of the larger mineralized systems in the area.

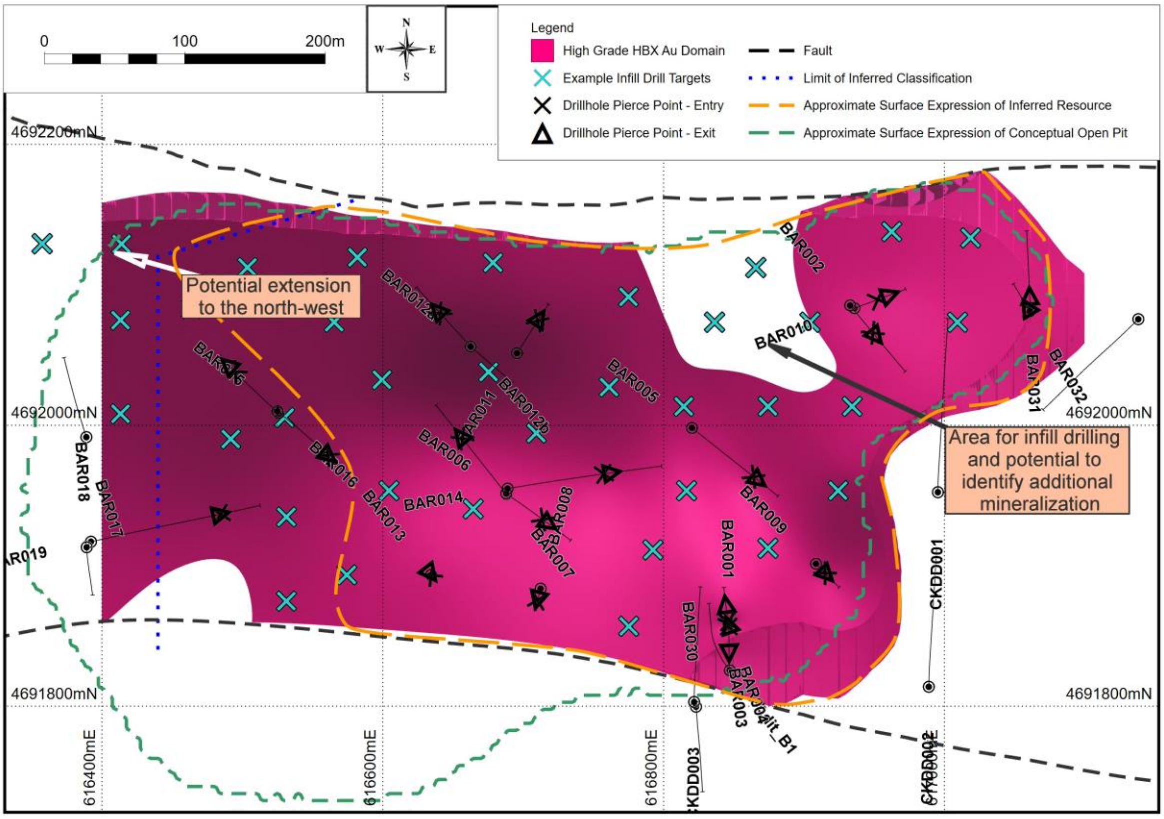


Figure 26.1: Recommended additional drill locations at Barje.

26.2 Mineral Processing and Metallurgical Testing

Further metallurgical testing is required to support a PFS. Standardised comminution tests are required to provide data to determine the most effective methods for primary comminution and flotation concentrate regrinding and to enable preliminary sizing of equipment. Additional flotation test work should be completed on samples of the HG_BX and LG_Sch material types to evaluate grade and metal concentration vs. recovery relationships. Testwork on one or more composite samples of both material types is also recommended to evaluate the effects of blending and material mixing during mining on metallurgical response.

Further testing of the OX material by cyanide and cyanide-free leaching (such as thiosulphate) warrants further investigation, as does scout testing of biohydrometallurgical and oxidizing leach processes for all material types.

26.3 Mining

No immediate mining specific studies are required for the subsequent recommended work programmes, however information and data that is collected in said work programmes should be incorporated into any future mining studies. This includes further work relating to analysis of the slope stability using geotechnical data from the orientated drill core in combination with hydrogeological data to account for the water table and other hydrogeological parameters.

Metallurgical modelling of the various material types should be improved provided appropriate further metallurgical testwork has been completed. Consideration should be given to grade vs. recovery across both main material types and mixed material types. Blending the HG_BX and LG_Sch material types should also be evaluated with respect to final concentrate grade, following which a re-evaluation of mining cut-off grades and the mine schedule should be undertaken; detailed scheduling in months is required to confirm the pre-strip requirements. Formal offtake studies should also be pursued which may also impact the mine schedule and cut-off grades.

26.4 Environmental and Social

PFS reporting requires evaluation of project impacts based on results of the baseline studies for the initial permitting process. The baseline studies may not have a full 12 months of monitoring data at the time of PFS, but at least initial field surveys should be completed. It is therefore recommended to commence ESIA study programmes and baseline data gathering including air, water, soil, fauna and flora studies, urbanization mapping and community consultation, plus development of an impact management programme.

26.5 Indicative Budget for Further Work.

An indicative Phase 1 budget for an exploration programme with the intention of converting the majority of Inferred Mineral Resources to Indicated Mineral Resources is presented in Table 26.1.; allowances are included for regional exploration drilling as well as dedicated geotechnical and hydrogeological drillholes and consulting fees to complete a Mineral Resource update. Based on favourable results from the Phase 1 it is advised to proceed with Phase 2 recommendations which include additional scout metallurgical testwork and commencing environmental baseline data collection, an indicative budget for which is outlined in Table 26.2. It may be practical to undertake some components of Phase 1 and 2 in parallel, for example initiating low-cost environmental baseline data collection during the next field season.

Table 26.1:Phase 1 indicative costs for additional drilling and MRE update.
Exchange rates; EUR1 = CAD1.5 or USD1.2

Item	Units	Unit Cost	Sub-totals		
			EUR	CAD	USD
Infill diamond drilling	3,200 m	75	240,000	360,000	288,000
Target testing diamond drilling	2,000 m	75	150,000	225,000	180,000
Geotech and Hydro Drilling	600 m	100	60,000	90,000	72,000
Assays of drill core	4,500	55	247,500	371,000	297,000
Staffing – core yard technicians	3 months	6,000	18,000	27,000	22,000
Staffing – geology and professional	6 months	25,000	150,000	225,000	180,000
Overheads, vehicles, core yard, rental	6 months	10,000	60,000	90,000	72,000
Land access and groundworks	1	30,000	30,000	45,000	36,000
Consulting and MRE update	1	50,000	50,000	75,000	60,000
		Sub-total	1,006,000	1,509,000	1,207,000
		Contingency (10%)	101,000	152,000	121,000
		Total	1,107,000	1,661,000	1,328,000

Table 26.2: Phase 2 indicative costs for additional metallurgical testwork and commencement of environmental baseline data collection.
Exchange rates as per Table 26.1.

Item	Units	Unit Cost	Sub-totals		
			EUR	CAD	USD
Additional Scout Metallurgical Tests	1	50,000	50,000	75,000	60,000
Commence EBLS Data collection	12 months	5,000	60,000	90,000	72,000
Consulting and Advisory	1	50,000	50,000	75,000	60,000
		Sub-total	160,000	240,000	192,000
		Contingency (10%)	16,000	24,000	19,000
		Total	176,000	264,000	211,000

27 References

ALS. (2020). *Metallurgical Testing, Barje Project, KM6180, Serbia, Work Performed on Behalf of Medgold Resources.* Kamloops: ALS Metallurgy .

Antić, M., Peytcheva, I., von Quadt, A., Kounov, K., Trivić, B., Serafimovski, T., . . . Wetzel, A. (2015). Pre-Alpine evolution of a segment of the North-Gondwanan margin: geochronological and geochemical evidence from the central Serbo-Macedonian. *Gondwana Res., 36*, 523-544.

Baker, T. (2019). Gold ± Copper Endowment and Deposit Diversity in the Western Tethyan Magmatic Belt, Southeast Europe: Implications for Exploration. *Economic Geology, 114*(7).

Beck, H. N. (2018). Present and future Köppen-Geiger climate classification maps at 1-km resolution. *Scientific Data*, doi:10.1038/sdata.2018.214.

Chambefort, I., & Moritz, R. (2006). Late Cretaceous structural control and Alpine overprint of the high-sulfidation Cu–Au epithermal Chelopech deposit, Srednogorie belt, Bulgaria. *Mineralia Deposita, 41*, 259-280.

Kounov, A., Seward, D., & Bernoulli, D. (2004). Thermotectonic evolution of an extensional dome: the Cenozoic Osogovo-Lisets core complex (Kraishte zone, western Bulgaria). *Int J Earth Sci, 93*, 1008-1024.

LBMA. (2020). *Precious Metal Forcasts 2020*. Retrieved from London Bullion Market Association: http://www.lbma.org.uk/downloads/Forecast-2020.pdf

Marchev, P., Kaiser-Rohrmeier, M., Heinrich, C., Ovtcharova, M., von Quadt, A., & Raicheva, R. (2005). Hydrothermal ore deposits related to post-orogenic extensional magmatism and core complex formation: The Rhodope Massif of Bulgaria and Greece. *Ore Geology Reviews, 27*, 53-89.

Medgold Resources Corp. (2016, June 13). *Medgold secures $1.5-Million investment and strategic alliance for Serbian Exploration.* Retrieved from Medgold Resources: https://medgoldresources.com/medgold-secures-1-5-million-investment-and-strategic-alliance-for-serbian-exploration/

Medgold Resources Corp. (2017, March 7). *Medgold options the Tlamino Project to Fortuna Silver Mines.* Retrieved from Medgold Resources: https://medgoldresources.com/medgold-options-the-tlamino-project-to-fortuna-silver-mines/

Medgold Resources Corp. (2017b, July 18). *Medgold Extends Channel Results at Tlamino, Returning 84 m at 5.6 g/t Au, and Identifies Large Near-Surface Geophysical Anomaly*. Retrieved from https://medgoldresources.com/: https://medgoldresources.com/

Medgold Resources Corp. (2019, September 18). *Medgold Announces Start of Exploration Drilling at its Karamanica Prospect, Tlamino Project, Serbia*. Retrieved from Medgold Resources: https://medgoldresources.com/medgold-announces-start-of-exploration-drilling-at-its-karamanica-prospect-tlamino-project-serbia/

Medgold Resources Corp. (2019a, March 21). *Medgold Announces 2019 Drilling Program at its Tlamino Project, Serbia and Reports Final Results of 2018 Program*. Retrieved from https://medgoldresources.com/medgold-announces-2019-drilling-program-at-its-tlamino-project-serbia-and-reports-final-results-of-2018-program/

Medgold Resources Corp. (2019b, September 18). *Medgold Announces Start of Exploration Drilling at its Karamanica Prospect, Tlamino Project, Serbia*. Retrieved from Medgold Resources: https://medgoldresources.com/medgold-announces-start-of-exploration-drilling-at-its-karamanica-prospect-tlamino-project-serbia/

Medgold Resources Corp. (2019c, November 28). *Medgold Announces Results of Metallurgical Testwork and Final Drilling at the Tlamino Project, Serbia*. Retrieved from Medgold Resources: https://medgoldresources.com/medgold-announces-results-of-metallurgical-testwork-and-final-drilling-at-the-tlamino-project-serbia/

Medgold Resources Corp. (2020, 01 30). *Medgold Announces Maiden Mineral Resource Estimate And Initial Metallurgical Testwork Results From The Tlamino Project, Serbia*. Retrieved from https://medgoldresources.com/medgold-announces-maiden-mineral-resource-estimate-and-initial-metallurgical-testwork-results-from-the-tlamino-project-serbia/

Medgold Resources Corp. (2021, January 07). *Medgold Signs Option Agreement With Fortuna Silver Mines To Acquire Full Ownership And Management Control Of The Tlamino Project, Serbia*. Retrieved from https://medgoldresources.com/medgold-signs-option-agreement-with-fortuna-silver-mines-to-acquire-full-ownership-and-management-control-of-the-tlamino-project-serbia/

Mineco Limited. (2019, November 16). *Bosilegrad Mine*. Retrieved from Mineco Web Site: https://www.minecogroup.com/bosilegrad

Petrović, M., & et al. (1978a). *Godišnji izveštaji o rezultatima istraživanja na području Karamanice (Annual reports on the results of research in the Karamanica area).* Belgrade: FSD Republičkog druš. fonda za geol. istr. (Republic of Serbia, Geofund Institute).

Petrović, M., & et al. (1978b). *Godišnji izveštaji o rezultatima istraživanja na Liski (Annual Report on the Results of Research at Liski).* Belgrade: FSD Republički društveni fond za geol. istr. (Rebuplic of Serbia, Geofund Institute).

Richards, J. (Ed.). (2016). *Special Publication Number 19. Tectonics and Metallogeny of the Tethyan Orogenic Belt.* Society for Economic Geology.

Sant, T. (2019). *Consultants Report on the Medgold Tlamino Project: History, Geology, Exploration and Drilling.* Exploration Direction Ltd.

Schmid, S. M., Fügenschuh, B., Kounov, A., Matenco, L., Nievergelt, P., Oberhänsli, R., . . . van Hinsbergen, D. J. (2019). Tectonic units of the Alpine Collision Zone Between Eastern Alps and Western Turkey - In Press. *Gondwana Research*. Retrieved from https://doi.org/10.1016/j.gr.2019.07.005

Sharp, I. R., & Robertson, H. (2006). Tectonic-sedimentary evolution of the western margin of the Mesozoic Vardar Ocean: evidence from the Pelagonian and Almopias zones, northern Greece. In *Special Publications, 260* (pp. 373-412). London: Geological Society.

Siddle, R. J. (2020). *Initial Gold-Silver Mineral Resource Estimate and NI43-101 Technical Report for the Medgold Barje Prospect and Tlamino Project Licences, Serbia.* Addison Mining Services Ltd.

Sillitoe, R. (2016). *Geology and Exploration of Medgold's Precious and Base Metal Prospects in Serbia.*

SOUR Geozavod-Belgrade, RO Institut za istraživanje mineralnih sirovina. (1983). *Tehničko-ekonomska ekspertiza mogućnosti eksploatacije rudne strukture Podvirovi-Božilovo ležište i tehničko-ekonomska ekspertiza mogućnosti eksploatacije rudnih ležišta u području Karamanice.*

von Quadt, Moritz, R., Peytcheva, I., & Heinrich, C. A. (2005). Geochronology and geodynamics of Late Cretaceous magmatism and Cu–Au mineralization in the Panagyurishte region of the Apuseni–Banat–Timok–Srednogorie belt, Bulgaria. *Ore Geol. Rev., 27*, 465-468.

Woods, J. (2019). *Technical Memorandum, Fortuna Met Review.* Denver: Woods Process Services LLC.

28 Illustrations.

Additional cross sections through the Barje Prospect are provided below.

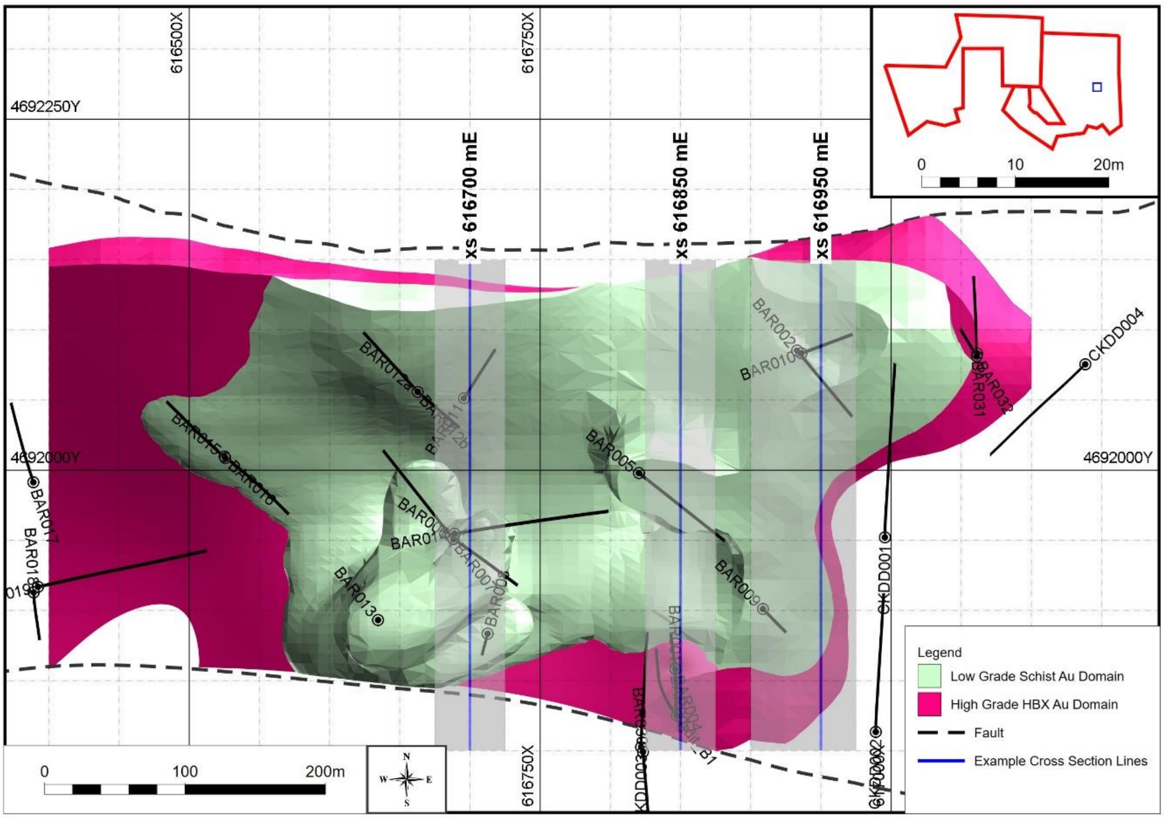


Figure 28.1: Barje Cross Section Plan View.

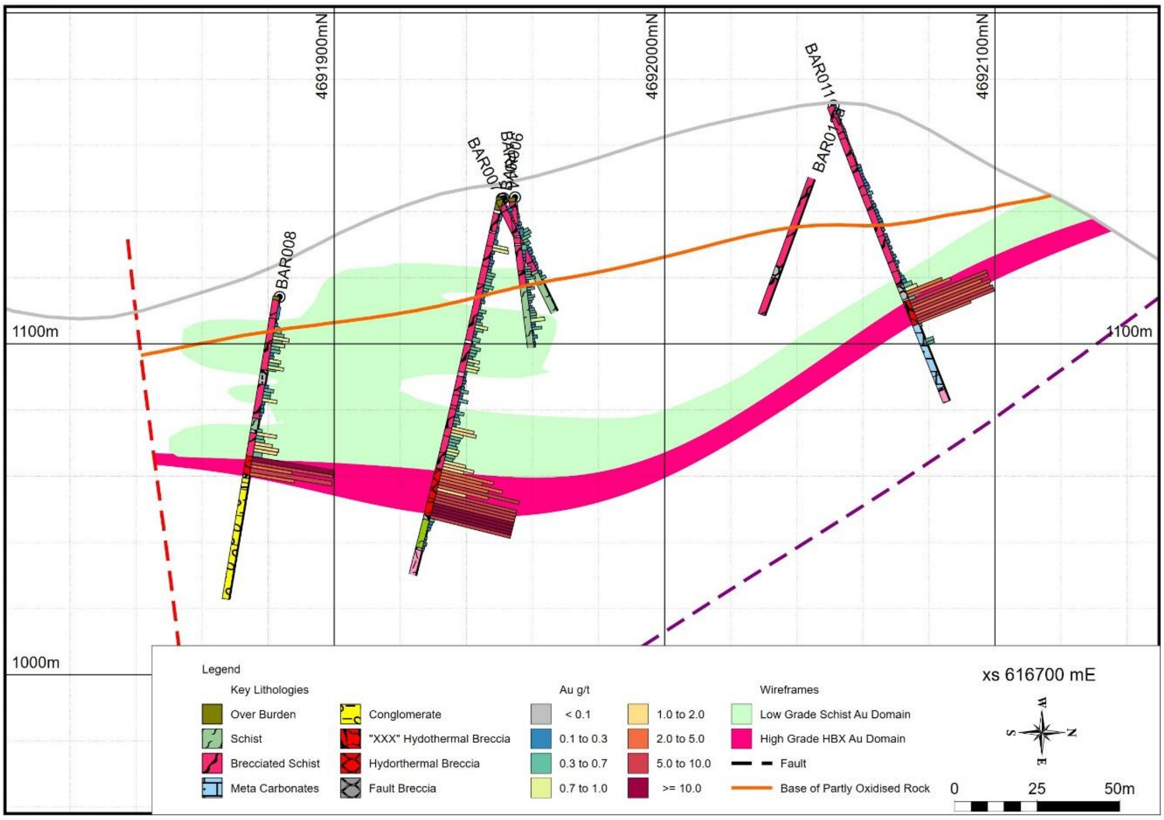


Figure 28.2: Cross Section 616700 mE Wireframes.

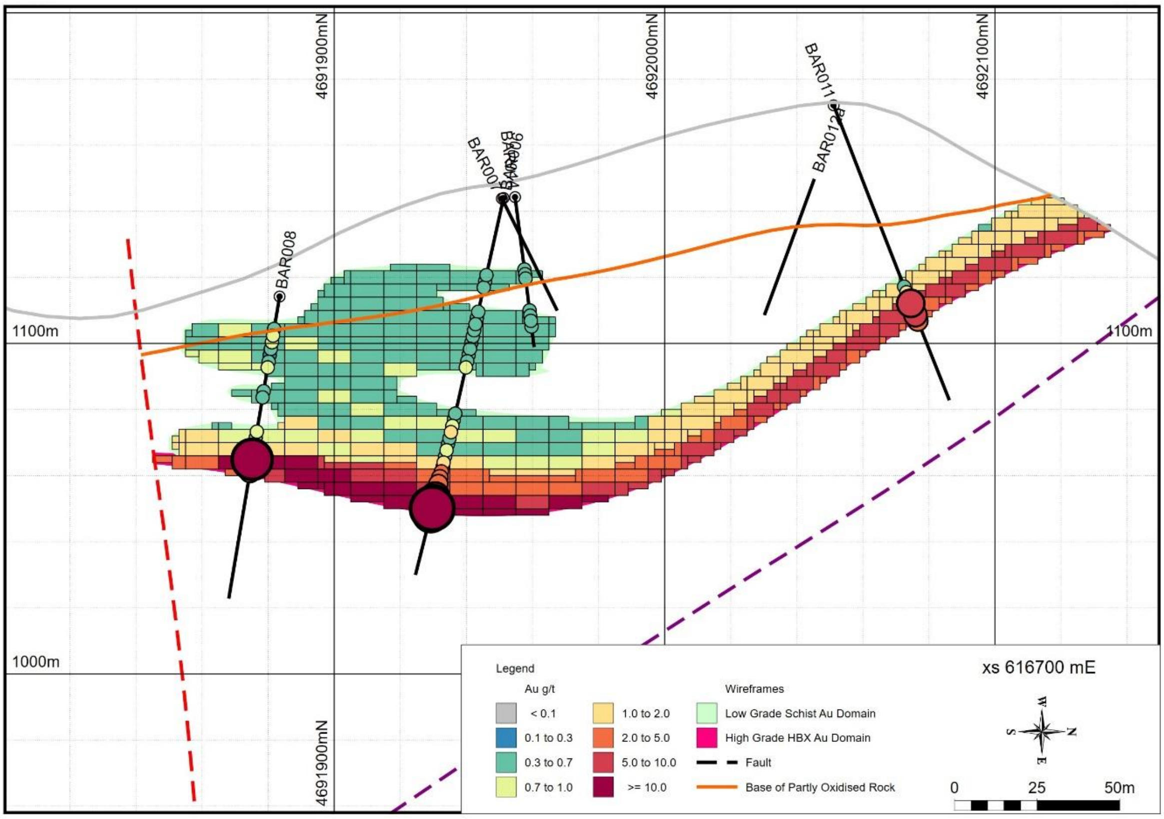


Figure 28.3: Cross Section 616700 mE Block Model.

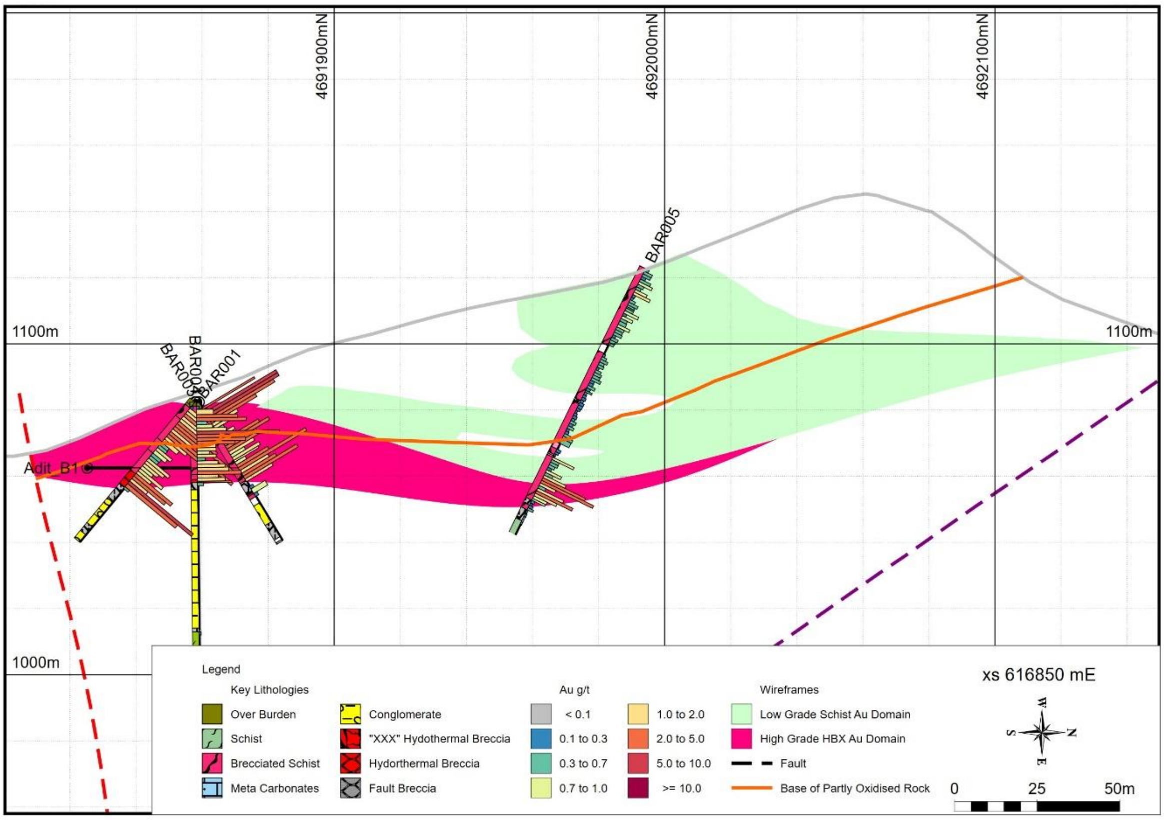


Figure 28.4: Cross Section 616850 mE Wireframes.

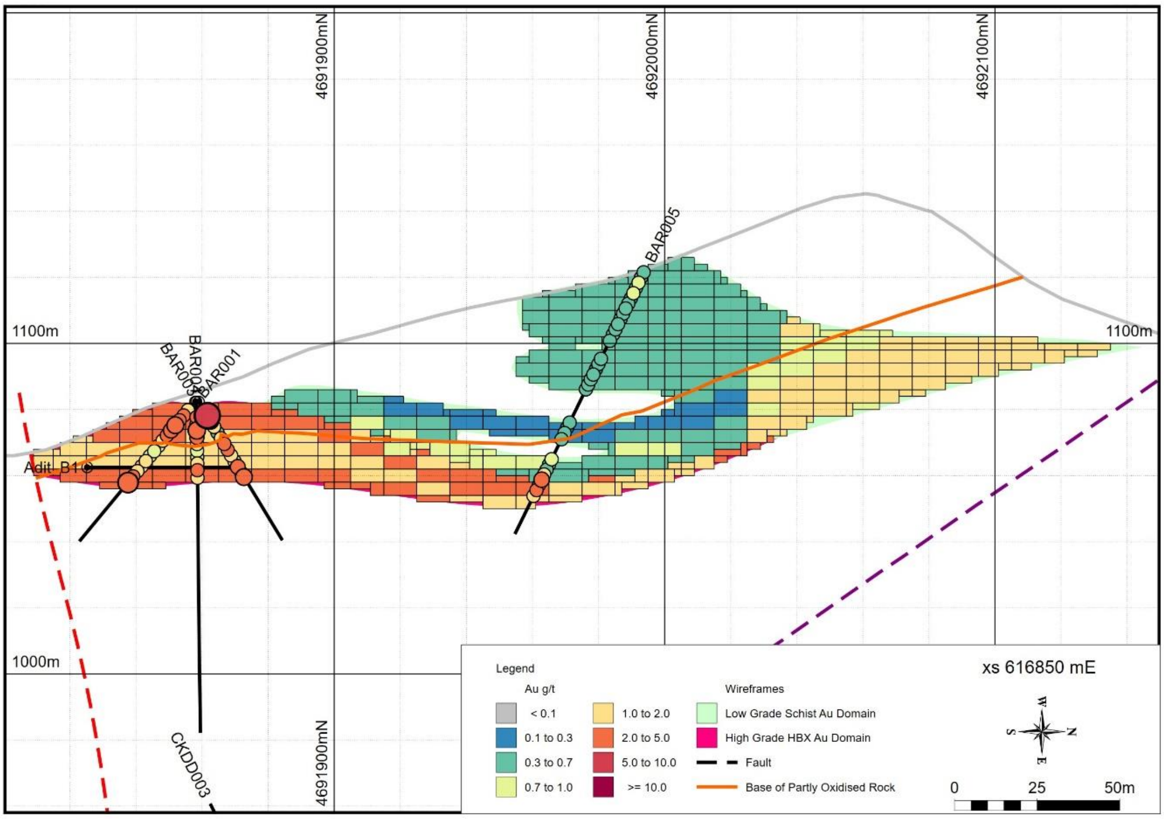


Figure 28.5: Cross Section 616850 mE Block Model.

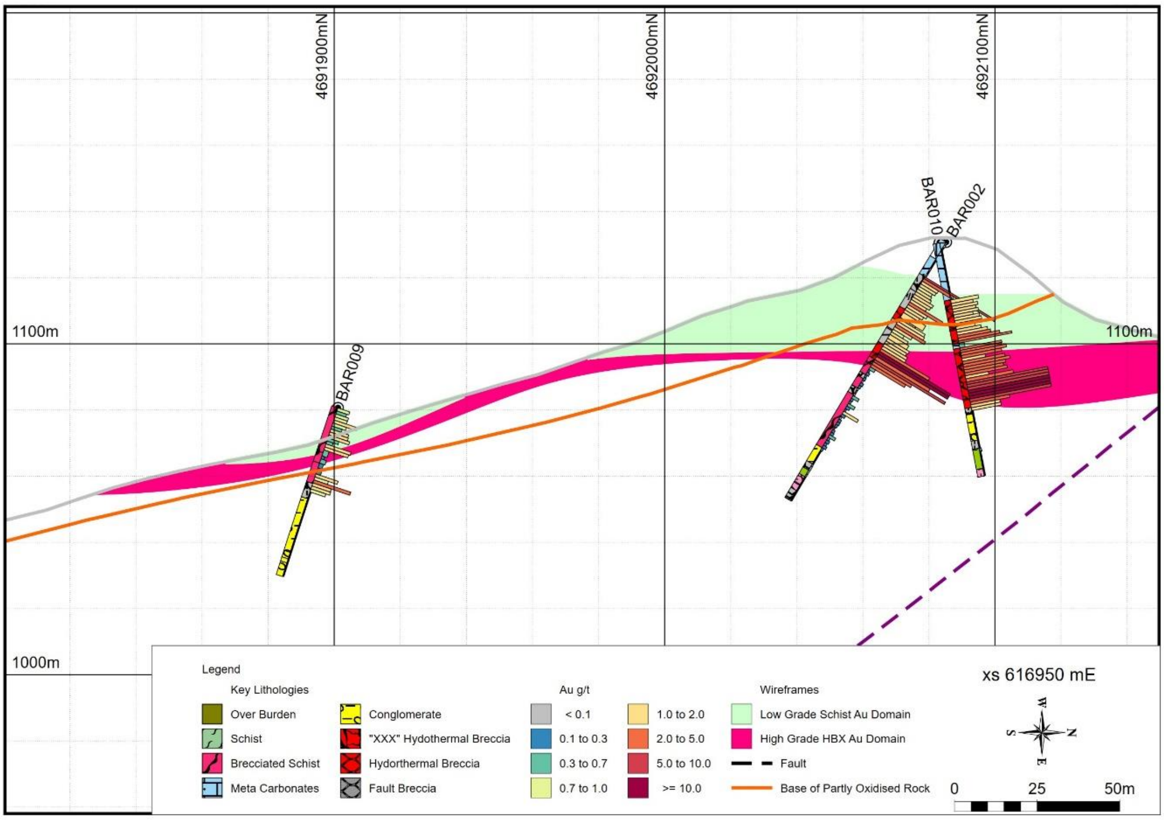


Figure 28.6: Cross Section 616950 mE Wireframes.

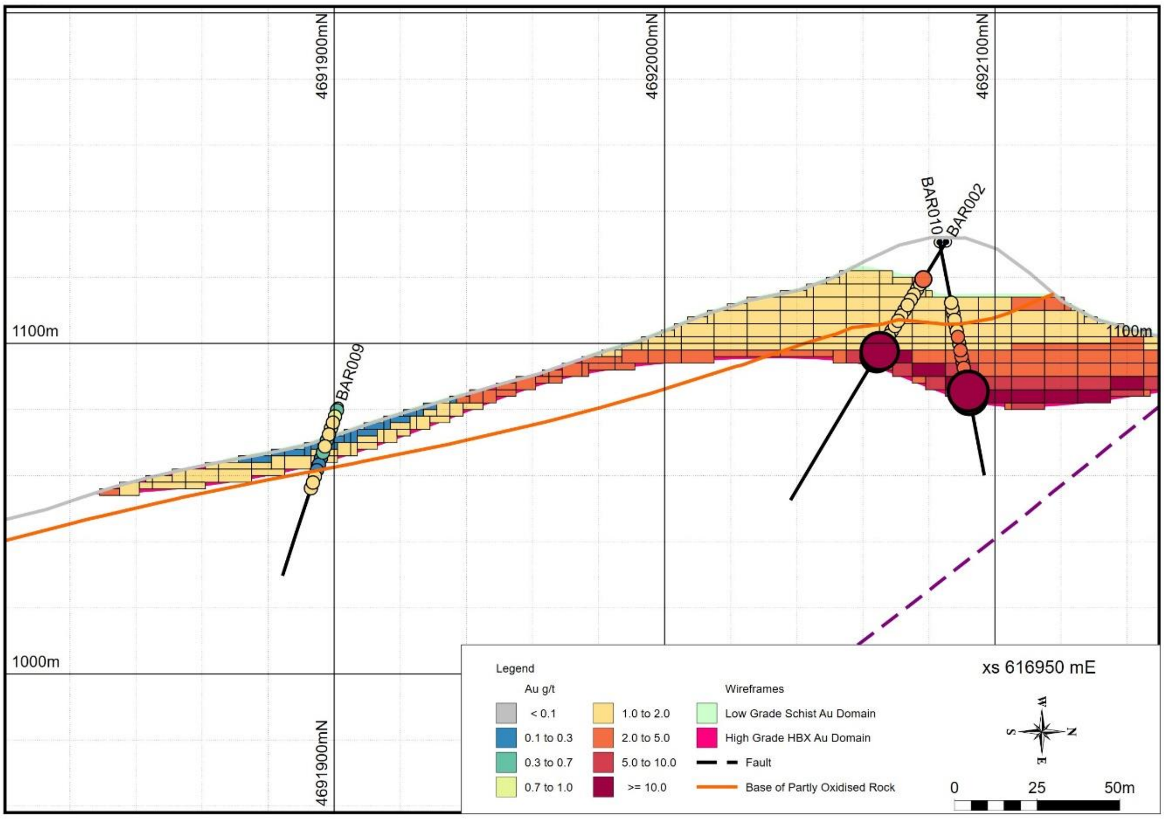


Figure 28.7: Cross Section 616950 mE Block Model.