Ore Reserves Report
for
Azerbaijan International Mining Company
(Ugur Gold Deposit)

September 2017

1 Executive Summary

Datamine International was requested by Azerbaijan International Mining Company (AIMC) [Anglo Asian Mining plc], to estimate the mineral reserves for the newly defined Ugur mineral deposit located in Gedabay area in the Republic of Azerbaijan. The estimation was completed in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (The Joint Ore Reserves Committee, 2012).

The mineral reserve estimate is based on the latest Mineral Resource estimate (August 2017) which takes into consideration geological exploration information from year 2016 and 2017, including results from exploration drilling campaigns.

A total of 11,042 samples were taken from 11,127 metres of drilling that resulted from 141 reverse circulation (RC), Diamond Core (DD) and Geotechnical drillholes.

The main objectives of the drilling campaign were to increase the level of geological knowledge and to provide confidence in the quantity (tonnage) and quality (gold, silver and other elements) estimates while focusing on continuity.

The Resource Estimate was classified into Measured, Indicated and Inferred mineral resources based on a cut- off grade greater than or equal to 0.2 grammes per tonne (g/t) of gold (Au). The results for the resource estimate are summarised in the Table 1-1 below;

Table 1-1 Mineral Resources Statement

Mineral Resource	Tonnage (millions)	Gold Grade (g/t)	Silver Grade (g/t)	Gold (ounces)	Silver (ounces)*
Measured	4.12	1.2	6.3	164,000	841,000
Indicated	0.34	0.8	3.9	8,000	44,000
Measure and Indicated	*4.46*	*1.2*	*6.2*	*172,000*	*884,000*
Inferred	2.50	0.3	2.1	27,000	165,000
Total	6.96	0.9	4.7	199,000	1,049,000

The Reserve estimate assumes a direct correlation between Proven and Probable, and Measured and Indicated, and excludes Inferred Resources, where the economic portion of the measured resource is converted to proven reserves and the economic portion of the indicated resource is converted to probable reserves. It is also assumed that all material types with gold grade greater than 0.3g/t can be processed through a combination of process routes that include Heap Leach (HL) and Agitation Leach Plant (AGL). Both theses processing facilities are available at the Anglo Asian Mining Gedabek contract area.

The other key difference between the Resource and Reserve estimate is that the Reserve is based on a fixed cut-off grade as the material is directed to the most appropriate processing method according to the economic criteria for the contained metals (gold and silver) and processing recovery.

It is also assumed that with heap leaching or agitation leaching, silver is recovered by the Sulphidisation-Acidification-Recycle-Thickening (SART) process (for which a processing plant is also located at the Gedabek contract area).

The assumed parameters for the various processing methods are shown in Table 1-2 .

Table 1-2-Metallurgical recovery factors.

Processes	Recovery	
	Au%	Ag%
Agitation Leach Plant (AGL)	90%	66%
Heap Leach (Dore) Crushed ore (HLC)	70%	7%
Heap Leach ROM (Dore) (ROM)	40%	7%

Further to assessing that the deposit is best mined by open pit method, and based on the optimised economic open pit limit, an open pit design was prepared using an overall pit slope angle of 38.0° and slope parameters recommended by the CQA International Ltd (CQA) (environmental and geotechnical engineering company). The resulting Reserve estimate is shown in Table 1-3.

Table 1-3 Mineral Reserves

Mineral Reserves	Tonnage (millions)	Gold Grade (g/t)	Silver Grade (g/t)	Gold (ounces)	Silver (ounces)
Proved	3.37	1.3	7.2	142,000	779,000
Probable	0.22	0.8	4.1	5,000	29,000
Proved and probable	3.59	1.3	7.0	147,000	808,000

Note: The calculation of recovered metal includes modifying factors for tonnage and grade that are based on historical recoveries for the process methods selected.

In addition to the ore Reserve of 3.6Mt, there is only 93kt within the selected open pit limit that is classified as Inferred resources within the geological model, the other resources being classified as measured and indicated resources.

Should the metal prices increase, then the geometry of a possible pit expansion would lend itself to further pushbacks ('pushback' represents an area that can be mined in a single continuous operation as defined within the ultimate pit) as there would be sufficient access width on all benches for continued mining.

Using the selected pit design, a number of pushbacks have been created which, when scheduled allowed to increase the AGL process material, and increased the production rate to 360kt per year from the beginning year 2018.

The Life-of-Mine (LOM) Schedule demonstrated that a practical blend of materials can be achieved that will meet the constraints on maximum plant capacity. Besides evaluating the economics as part of the pit optimisation, the LOM schedule has been evaluated using AIMC's own financial model. This confirms that the selected pit is economic and is in line with the valuation (before tax) produced by the computerised open pit optimiser (NPV Scheduler).

It is concluded that the Ore Reserve for the Ugur open pit is 3.59Mt, with a contained metal content of 147,000 ounces of gold and 808,000 ounces of silver.

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Tables

Figures

2 Acronyms and Abbreviations

List of Abbreviations/Acronyms is below:

Abbreviation/Acronym	Meaning
.dm file	A Datamine format file
Ag	Silver
AIMC	Azerbaijan International Mining Company
AGL	Agitation Leach Plant
Au	Gold
Cu	Copper
DI	Datamine International Ltd.
g/t	gram per tonne
G&A	General and Administration
HL	Heap Leach
HLC	Heap Leach –Crushed
HLROM	Heap leach- ROM
Kt	kilo
lb	pound
LOM	Life of Mine
Mt	million tonnes
NPVS	Datamine NPV Scheduler
NSR	Net Smelter Return
OSA	Overall Slope Angle
oz	ounce
SART	Sulphidisation - Acidification-Recycle -Thickening
t	tonne
tpa	tonne per annum
TSF (TMF)	Tailings Storage Facility (Tailings Management Facility)

1 Introduction

Datamine International Ltd, here after called DI (utilising the Datamine software products; Studio RM, Studio OP and NPV Scheduler) were requested to prepare an Ore Reserve Statement for the Ugur mineral deposit, by Azerbaijan International Mining Company Limited (Anglo Asian Mining). The aim was to carry out an estimation of the mineral ore reserves of the Ugur mineral deposit located in the Republic of Azerbaijan.

The estimate is the first reserve estimate of the deposit since its discovery in 2016. The estimation was completed in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (The Joint Ore Reserves Committee, 2012).

The mineral resources estimation and reserves calculation was carried out by taking into consideration all exploration data since the discovery of the deposit. This report has been prepared taking into consideration the guideline of the JORC Table 1 as shown in Appendix I.

The “Ugur” deposit is located within the locally defined Ugur exploration area. The Ugur gold deposit was discovered in 2016 by the Gedabek Exploration Group of Anglo Asian Mining who worked on the regional area of Ugur from 2014 year.

Historical work on the area included regional mapping and large-scale regional geophysical programmes (magnetic and gravity) by Soviet geologists (however, the Ugur deposit itself was not discovered during this period of exploration).

The mine is primarily focused on the extraction of gold, with bi- products of silver. The agreed process flow algorithm utilised in scheduling the production is shown in Figure 1-1 in terms of the cut-off grades.

The general layout of the mine infrastructure for the proposed Ugur mine is shown in Figure 1-2 and consists of:

- Ugur final pit limit
- Leach Pads
- Ugur Waste dump
- Main access roads

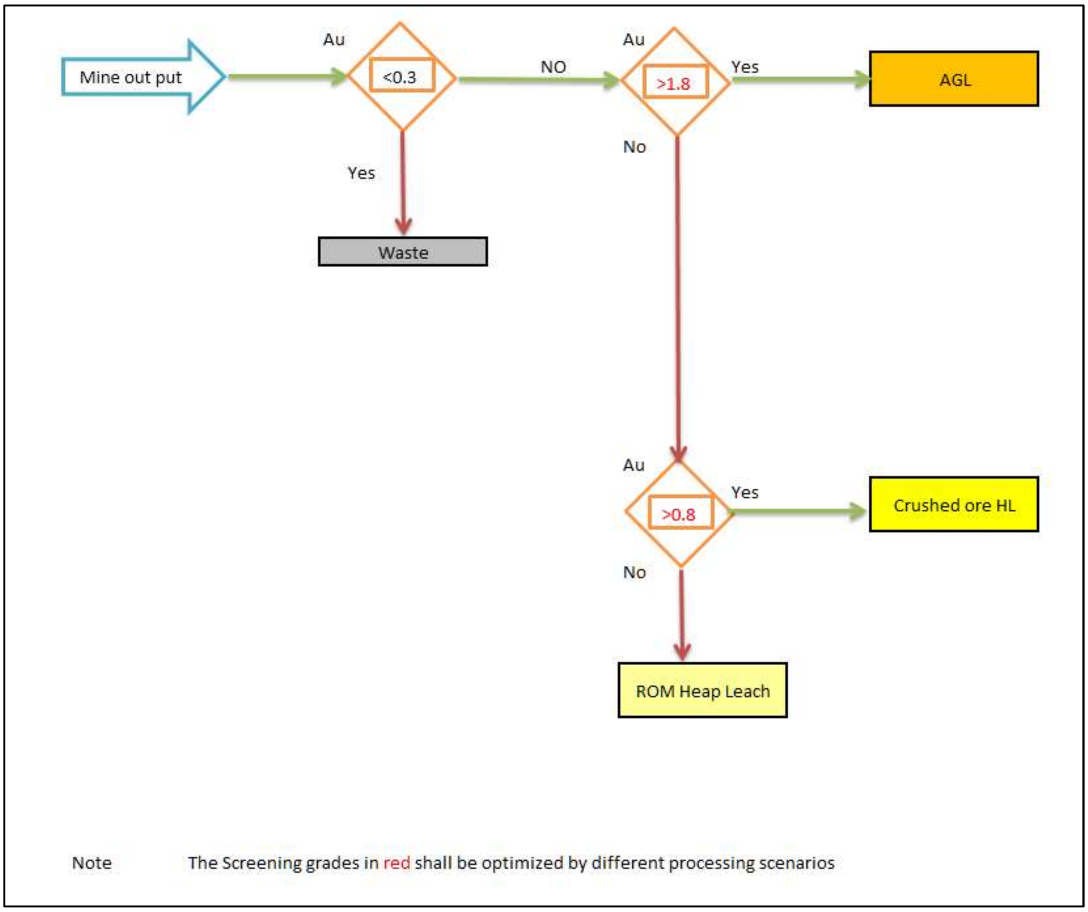


Figure 1-1-Simplified process configuration for Ugur

A full description of the location of the property and the geological setting is given in DI's Mineral Resources report of September 2017 and is not repeated here.

For the purposes of the pit optimisation, the minimum cut-off grade for HL was calculated on the basis of the NSR (in Datamine software) such that the cut-off between ore and waste is where the NSR is greater than or equal to processing plus fixed costs.

A report was prepared by the geotechnical consultant company, CQA International Limited, regarding the slope parameters that were used for pit design. This resulted in a report being issued with a recommended overall slope angle (OSA) of 38°.

During the audit, the Competent Person also confirmed with AIMC the modifying factors to be used for:

- Tonnage and grade estimates
- Process recoveries
- Metal prices
- Mining and processing costs
- Fixed costs.

The main objective of this report is to document the procedure used to determine the Ore Reserve and ensure that this estimate follows the guidelines set down by JORC (2012).

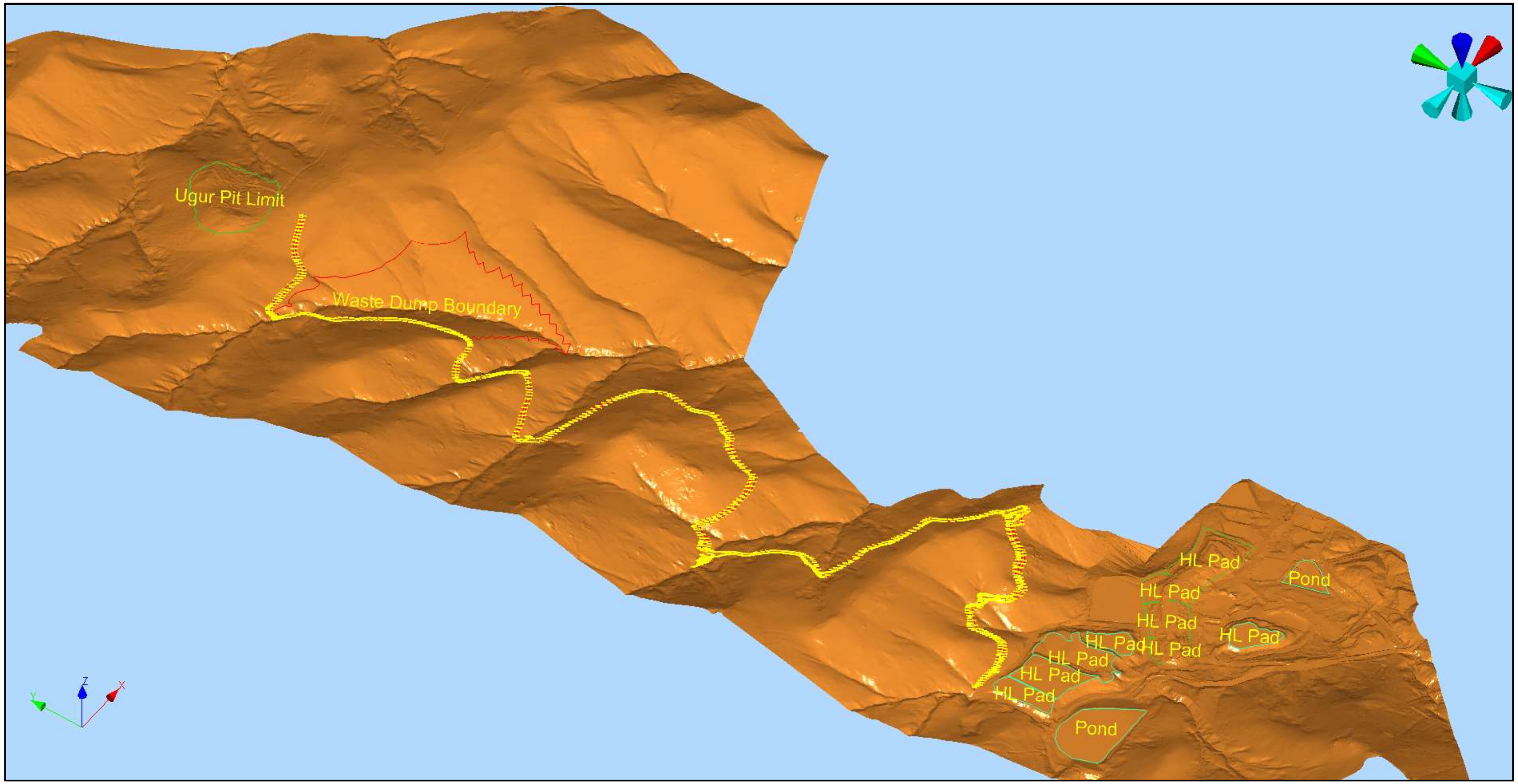


Figure 1-2-General location plan for Ugur mine

1-1 Qualification of Consultants

Kayvan Samadani is a mining engineer with more 21 years' experience in the mining industry in areas of strategic mine planning, mid to short term scheduling and planning, QA-QC study for exploration sample analysis, project management of a 2.5 million USD consulting project, including resource estimation, geotechnical drilling supervision, geotechnical study, strategic mine planning, health-safety-environment (HSE) and short-term scheduling. Kayvan is also head of the mine planning department of a consulting company and also senior mining consultant for various software solution implementation (Datamine products).

Kayvan has experience in several commodities, including and more extensive with gold, copper, iron ore, cement and talc.

Kayvan has got Bachelor degree and Master degree in Mining Engineering and is a Professional Member of the Institution of Materials, Minerals and Mining (MIMMM).

Kayvan joined Datamine International Ltd in January 2009 and in his current role at Datamine International Limited, he has taken responsibility of a consulting project and works as a senior mining consultant.

Kayvan is an experienced user of mining software solutions such as NPV Scheduler, Studio OP (Open Pit Engineering) and Sirovision and also is familiar with other software solutions such as enhanced production scheduler (EPS), Ore Controller, 5DPlanner underground, DataBlast and "3d Dig" software. He is also an experienced user of general software such as Microsoft (MS) Excel, MS Word and MS Power Point.

1-2 Qualification of Competent Person

Stephen Westhead is a geologist who earned an extractive industries Doctorate (PhD) in "Structural Controls on Mineralisation", a Master's degree (MSc) in "Mineral Exploration and Mining Geology", a European Union Certificate in "Environmental Technology" and a Honours Bachelor Degree (BSc) in "Applied Geology".

In 1989, Stephen started his career in the mining sector as a Geologist with Anglesey Mining working at the Parys Mountain property in Wales. Following completion of a PhD in 1993, worked in India for five years as a consultant geologist focusing on cement and base metals sectors. For the final year in India, was a founder member of Fluor Daniel India (Pvt) Ltd working in resource analysis for the group mining and metals division, infrastructure and project development.

In 1997, Stephen moved to work in Central Asia for a period of 10 years, working in Tajikistan, Uzbekistan, Kyrgyzstan and Kazakhstan. The positions held included Project Geologist, Country Chief Geologist, Subsidiary mining company Director, Group Chief Geologist, and General Director. The focus of this period was gold, silver and base metals projects, including resources and reserves management, project development and production.

In 2006, Stephen worked in Ukraine, Eastern Europe, and Kazakhstan as Group Chief Geologist and Project Manager, again focusing on gold and silver commodities. In 2009, Stephen joined the Polyus Gold Group as Group Project Manager and subsequently as Technical Adviser to the Managing Director of the group's largest business production unit, covering exploration and mining geology, mining, material handling and processing.

In April 2016, Stephen consulted to Azerbaijan International Mining Company (Anglo Asian Mining plc), and joined the group in May 2016 as Director of Geology. Subsequently in January 2017, became Director of Geology and Mining (current position).

Stephen has expertise heading project management from exploration stages to construction and mine production. Has been part of teams that have taken projects through feasibility study, raised finance, constructed mines/plants and brought into production.

Professional accreditations include being a Chartered Geologist (CGeol) and Fellow of The Geological Society (FGS), Professional Member of the Institution of Materials, Minerals and Mining (MIMMM), Fellow of the Society of Economic Geologists (FSEG) and Member of the Institute of Directors (MIoD). Recently awarded the Institute of Directors Certificate in Company Direction (August 2017), with awards in; The Role of the Director and the Board, Finance for Non-Financial Directors; The Director's Role in Strategy and Marketing, and Leadership for Directors.

1-3 Site visits

Datamine International company developed and audited the Ugur Mineral Resource block model. Datamine engineer, Kayvan Samadani, worked on the reserves and was able to verify work ethics and procedure. During the period from discovery to reserve estimation, the Datamine consultant carried out 6 trips to Gedabek that comprised 42 on site days.

Datamine have been involved with other mining projects of the company within the same licence area as Ugur and as such are familiar with the processing methods available, value chain of the mining and cost structure. The data has been audited and considered robust for Mineral Resource estimates.

Internal company and external reviews of the Mineral Resources yield estimates that are consistent with the Mineral Resource results. The methods used include sectional estimation, and three-dimensional modelling utilising both geostatistical and inverse distance methodologies. All results showed good correlation.

The Competent Person (CP), Stephen Westhead is an employee of the company and as such has been actively in a position to be fully aware of all stages of the exploration and project development. The CP has worked very closely with the independent resource and reserve estimation staff of Datamine, both on site and remotely, to ensure knowledge transfer of the geological situation, to allow geological "credibility" to the modelling process. Extensive visits have been carried out by two staff of Datamine over the last year and have been fully aware of the Ugur project development. All aspects of the data collection and data management has been observed.

2 Resource Model

The filename of the resource model used for this reserve estimation process was “final2_grade_mod_ugu_20170711.dm”, which was issued by DI in August 2017. The Resource Statement for the Ugur deposit is shown in Table 2-1.

Table 2-1-Resource statement for the Ugur deposit (August 2017)

Au Cut off=0.2 g/t					
Mineral Resources	Tonnage	Gold Grade	Silver Grade	Gold	Silver
	Mt	(g/t)	(g/t)	(K oz)	(K oz)
Measured	4.12	1.2	6.3	164	841
Indicated	0.34	0.8	3.9	8	44
Measured + Indicated	4.46	1.2	6.2	172	884
Inferred	2.50	0.3	2.1	27	165
Total	6.96	0.9	4.7	199	1049

3 Modifying Factors

The modifying factors to be used in the Reserve estimate are summarised in Table 3-1 . These follow the guidelines set out in the JORC code.

Table 3-1-Modifying factors used to determine the ore reserve

<table>
<tr><th>Criteria</th><th>Commentary</th></tr>
<tr><td><i>Mineral Resource estimate for conversion to Ore Reserves</i></td><td>
A JORC resource estimate comprising Measured, Indicated and Inferred Resources has been made for the Ugur Deposit (as tabulated below):
<table>
<tr><th>Mineral Resources</th><th>Tonnage (Mt)</th><th>Gold Grade (g/t)</th><th>Silver Grade (g/t)</th></tr>
<tr><td>Measured</td><td>4.12</td><td>1.2</td><td>6.3</td></tr>
<tr><td>Indicated</td><td>0.34</td><td>0.8</td><td>3.9</td></tr>
<tr><td><i>Measured+Indicated</i></td><td><i>4.46</i></td><td><i>1.2</i></td><td><i>6.2</i></td></tr>
<tr><td>Inferred</td><td>2.50</td><td>0.3</td><td>2.1</td></tr>
<tr><td>Total</td><td>6.96</td><td>0.9</td><td>4.7</td></tr>
</table>
The contained metal in ounces of gold and silver is presented below:
<table>
<tr><th>Mineral Resources</th><th>Gold ('000 ounces)</th><th>Silver ('000 ounces)</th></tr>
<tr><td>Measured</td><td>164</td><td>841</td></tr>
<tr><td>Indicated</td><td>8</td><td>44</td></tr>
<tr><td><i>Measured+Indicated</i></td><td><i>172</i></td><td><i>884</i></td></tr>
<tr><td>Inferred</td><td>27</td><td>165</td></tr>
<tr><td>Total</td><td>199</td><td>1,049</td></tr>
</table>
The relative % of contained metal shows a very high % of Measured Resource and Indicated Resource that can be tested for Reserve estimation.
<table>
<tr><th>Mineral Resources</th><th>% gold ounces</th><th>% silver ounces</th></tr>
<tr><td>Measured</td><td>82%</td><td>80%</td></tr>
<tr><td>Indicated</td><td>4%</td><td>4%</td></tr>
<tr><td><i>Measured+Indicated</i></td><td>87%</td><td>84%</td></tr>
<tr><td>Inferred</td><td>13%</td><td>16%</td></tr>
<tr><td>Total</td><td>100%</td><td>100%</td></tr>
</table>
The Ore Reserve statement is inclusive (not additional to) of the Resource statement.
</td></tr>
</table>

Criteria	Commentary
Site visits	• The Competent Person is an employee of the company and as such has been actively in a position to be fully aware of all stages of the exploration and project development including the estimation of Mineral resources and Ore Reserves. The Competent Person has worked very closely with the independent resource and reserve estimation staff of Datamine company, both on site and remotely, to ensure knowledge transfer of the geological situation, to allow geological "credibility" to the modelling process. Extensive visits have been carried out by two staff of Datamine (one of whom estimated the resources and one estimate the reserves) over the last year and have been fully aware of the Ugur project development. All aspects of the data collection and data management has been observed.
Study status	• Study undertaken to enable Mineral Resources to be converted to Ore Reserves are considered as being Feasibility level. The ore will be mined utilising the current mining fleet and will be processed in the current processing facilities of the Company which operates two other mines in the same licence/contract area. The Ugur deposit is considered to part of the same geological terrain. • A technically achievable mine plan that is economically viable has been designed taking into consideration the JORC resources and modifying factors.
Cut-off parameters	• Financial factors included in the cut-off grade estimates are process and overhead costs, mining dilution, payable gold and silver price, and processing recovery and used in the basis for cut-off grade calculation. • The ore from Ugur can be processed by three different available processing methods within the Gedabek contract area, namely agitation leach (AGL), heap leach of crushed material (HLC) and heap leach of blasted material or run-of-mine (ROM). • The acceptable gold head grade in grammes per tonne gold for AGL, HLC and ROM is 1.8g/t, 0.8g/t and 0.47g/t respectively. • Further to the gold cut-off grade calculations, after long term scheduling the mill cut-off grade resulted in 0.3g/t gold.
Mining factors or assumptions	• On establishing the modifying factors, the Mineral Reserve has been optimised using the Datamine NPV Scheduler ® software. This resulted in the economic open pit shell and contained mineable material in that pit shell. Subsequently, this was further optimised in the mine design process, using Datamine Studio OP ® software, where bench toe and crest, catch benches and haul road layout was designed. The final mineable material comprised the Ore Reserves. • The mining method selected is by open pit method given the orebody geometry and the position relative to topographic surface. The central part of the orebody is exposed at surface, and over the remaining 70% surface area of the orebody there is a top soil cover

Criteria	Commentary
	varying in thickness between zero and 50 centimetres. Access to the orebody is from surface. The open pit mining method is considered appropriate, and will comprise conventional truck and shovel. • Pit slope angles have been determined based on independent geotechnical investigation taking into account geological structure, rock type and design orientation parameters. The overall pit slope angle is 38 degrees containing an average bench angle of 58 degree. • Based on the geotechnical findings further to the independent report by CQA, the overall pit slope angle is maximum 38 degrees, berm width 6 metres and after each 5 benches (50 metre height), a catch bench of 10 metre width should be considered for the open pit design. • Mining dilution used in the Datamine NPV Scheduler software for reserve estimation is 5%. • Ore mining recovery factor used in the Datamine NPV Scheduler software for reserve estimation is 95%. • A minimum mining width of 20m has been used. • The total tonnage of inferred material in the final pit design was 87,100 tonnes which represents about 2.37% of total ore tonnage in the pit and contains 0.76% (1,134 ounces) of contained gold in the pit. • The inferred material was excluded from economic model in NPV Scheduler so it had no impact on the total reserve. • Infrastructure required for the open pit mining method include haul road access (completed to the mine area), offices for geology/mining department, mining workshop, fuel storage, weighbridge and medical/HSEC facilities. Explosives will be transported from another mine operating within the contract area.
Metallurgical factors or assumptions	• The proposed metallurgical processes are well tested being processing facilities of current mining operations in the contract area. The processing facilities include agitation leach by conventional methods, crushed heap leach, and run-of-mine dump leach. AGL process comprises comminution (crushing and grinding), Knelsen concentration, thickening, agitation leaching, resin-in-pulp extraction, and elution and electrowinning to produce gold dorè. The final product will be shipped off site for refining. Tails from the process will be transferred via gravity pipeline to the existing tailings management facility (TMF) that has enough capacity to manage the ore from the Ugur deposit. • Metallurgical testwork has been conducted in the form of bottle roll testing and column leach tests. The amount of testwork is considered representative of the processing technology to be employed. • Deleterious elements were not detected in analytical tests and assaying utilised for the resource estimate. • No pilot scale testwork has been conducted. However, given the

Criteria	Commentary
	nature of the ore type and its close relationship with existing ore bodies being processed, the metallurgical testwork carried out is considered representative of the orebody as a whole. • The ore reserve estimation has been based on the appropriate mineralogy to meet the specification.
Environmental	• Previous ESIA (Environmental Social Impact Assessment) has been carried out by Amec Foster Wheeler (2012) and TexEkoMarkazMMC (2012) (submitted to Government authorities). The Ugur deposit is located within the Gedabek Contract Area for which the ESIA is valid, hence the most recent ESIA is applicable to Ugur. Processing and tailings storage reported in the ESIA is the same as will be utilised for Ugur ores. • Environmental and geotechnical consultants, CQA International Ltd of the UK (CQA), have on-site representation, and carried out both geotechnical and environmental assessments of the Ugur mine area. Baseline environmental monitoring has been carried out on receptors downstream of the mine site, due to an additional catchment being located in the vicinity of the Ugur mine. • The waste rock has a low potential for acid rock drainage due to the absence of sulphide bearing mineralisation. Watercourses downstream of stockpiles will be monitored on a routine basis for pH and heavy metals. • A topsoil management plan is in place, that has been reviewed by a CQA consultant deemed in accordance with the storage principles of the Ministry of Ecology and Natural Resources of the Republic of Azerbaijan and European Union (EU) guidelines. Topsoil removal will take place in August 2017, and be stockpiled in a dedicated location with specific design parameters. Stockpiling of materials will be carried out following the soil management plan. • A stockpile area for waste rock has been identified following condemnation drilling verifying the absence of mineralisation beneath the proposed stockpile. The top soil at the planned site will be removed, and the hill terraced to “key” in the waste dump for maximum stability. • The tailings management facility (TMF) has the capability for the additional storage requirements for Ugur process waste. The design and operations of the TMF have been reviewed by CQA along with a visit by the Ministry of Ecology and Natural Resources of the Republic of Azerbaijan. Regular environmental monitoring is carried out at the TMF, along with monitoring all receptors associated with the TMF. • All approvals for conducting the mining fall under the management “PSA” agreement.
Infrastructure	• Infrastructure is considered excellent to the deposit. The deposit is located within the Company’s contract/licence area with extraction

Criteria	Commentary
	rights according to the Government contract. Ore can be processed at the Company’s current facilities, with ore being delivered by truck from the mine to processing via the newly constructed haul road over a distance of about 6 kilometres. Land availability for the mine and associated infrastructure is approved. Offices and mechanical workshop buildings are available within the company and will be relocated to Ugur. Power for the offices and weighbridge will be initially via diesel generators, although solar power is also under consideration. Labour is readily available as the operation is relatively small and only additional mine site labour will be required. G&A and process labour are part of the existing company compliment of staff. Regarding accommodation, canteen facilities and associated services, the Ugur deposit can be considered a “satellite” deposit to the current mining operations and will be serviced by the current infrastructure.
Costs	• Project capital costs are “minimal” given that no processing facilities or manpower camps are required. The costs in relations to the facilities already referenced above are based on actual quotations and capital construction experience at the licence area and sustaining capital projects are based on operational experience locally. • Operating costs are estimated based on current mining and processing operations within the licence area, as the processing will be carried out at the same plants, and the mining contract and haulage costs are the same as current contracts. • No allowances have been made for deleterious elements. • Commodity pricing is based on forecasts by reputable market analysts. • Local Azeri exchange rates are pegged to the United States $. The source of exchange rates used in the study is the Central Bank of the Republic of Azerbaijan. • Transportation charges are based on current contracts that will be extended to include haulage of ore from Ugur deposit to the processing facilities. All other transport costs will be per the current contracts for the operating mines. • Treatment and refining costs are based on current contracts, as the ore will be treated in the operating processing plants and refined under the current agreement. • Royalties have been considered as part of the cost structure for the company to operate under the Government Contract. • The estimated operating costs per tonne used in NPV Scheduler are:

Criteria	Commentary
	Parameters used in NPV Scheduler **Processing cost (includes G&A)** *per tonne of ore* AGL $ 29.22 HL Crushed $ 6.37 HLROM $ 5.22 **Other costs** Total G&A $ 3.22 Mining cost $ 1.75 Haulage cost (per tonne km) Manat 0.1
Revenue factors	• Revenue is based on the US$ gold price and US$ silver price. • The price of gold in the reserve model is $1250 per troy ounce and the price of silver in the reserve model is $18.66 per troy ounce.
Market assessment	• The market for gold and silver is well established. The metal price is fixed externally to the Company, however, the Company has reviewed a number of metal forecast documents from reputable analysts and is comfortable with the market supply and demand situation. • A specific study of customer and competitor analysis has not been completed as part of this project. • Price and volume forecasts have been studied in reports from reputable analysts, based on metal supply and demand, US$ forecasts and global economics. • Industrial minerals do not form part of this study.
Economic	• Prices for gold and silver used in NPV Scheduler are: Gold: $40.19 per gramme Silver: $0.55 per gramme • Processing Recovery (for gold / silver) % Agitation Leach 90% / 66% Crushed Heap Leach 70% / 7% Run-of-mine (ROM) 40% / 7% • Costs used in NPVS are show below: **Parameters used in NPV Scheduler** **Processing cost (includes G&A)** *per tonne of ore* AGL $ 29.22 HL Crushed $ 6.37 HLROM $ 5.22 **Other costs** Total G&A $ 3.22 Mining cost $ 1.75 Haulage cost (per tonne km) Manat 0.1 Selling Cost %0.05 of revenue of Gold

Criteria	Commentary
	Selling Cost %0 of revenue of Silver • Sensitivity analysis has been used at a range of gold prices.
Social	• To the best of the Competent Person's knowledge, agreements with key stakeholders and matters leading to social licence to operate are valid and in place.
Other	• There are no material naturally occurring risks associated with the Ore Reserves. • Anglo Asian Mining plc is currently compliant with all legal and regulatory agreements, and marketing arrangements. • The project is located within a current contract area that is managed under a "PSA" production sharing agreement. • The PSA grants the Company a number of periods to exploit defined licence areas, known as Contract Areas, agreed on the initial signing with the Azerbaijan Ministry of Ecology and Natural Resources ('MENR'). The exploration period allowed for the early exploration of the Contract Areas to assess prospectivity can be extended. • A 'development and production period' commences on the date that the Company issues a notice of discovery, which runs for 15 years with two extensions of five years each at the option of the Company. Full management control of mining in the Contract Areas rests with Anglo Asian. • Under the PSA, Anglo Asian is not subject to currency exchange restrictions and all imports and exports are free of tax or other restriction. In addition, MENR is to use its best endeavours to make available all necessary land, its own facilities and equipment and to assist with infrastructure. • The PSA is valid for the forecast life of mine.
Classification	• Measured Mineral Resources have been converted to Proved Reserves after applying the modifying factors. • Indicated Mineral Resources have been converted to Probable Ore Reserves after applying modifying factor. • The resultant Ore Reserves are appropriate given the level of understanding of the deposit geology and reflects the Competent Person's view of the deposit. • The inferred material was excluded from economic model in NPV Scheduler so it had no impact on the total reserve, and no Probable Ore Reserves have been derived from Measured Mineral Resources.
Audits or reviews	• Datamine company developed and audited the Mineral Resource and Mineral Reserve block models. Two Datamine engineers worked on the resources and reserves and were able to verify work and procedure. • Datamine have been involved with other mining projects of the company within the same licence area as Ugur and as such are

<table>
<tr><th>Criteria</th><th>Commentary</th></tr>
<tr><td></td><td>
familiar with the processing methods available, value chain of the mining and cost structure. The data has been audited and considered robust for Ore Reserve estimates.
• Internal company and external reviews of the Ore Reserves yield estimates that are consistent with the Ore Reserve results. The in-situ Ore Reserves classified by process type is presented below:
<table>
<tr><th>Ore Reserves (Process & Class)</th><th>Tonnage (Tonnes)</th><th>Gold Grade (g/t)</th><th>Silver Grade (g/t)</th><th>Gold ('000 ounces)</th><th>Silver ('000 ounces)</th></tr>
<tr><td>Proved-AGL</td><td>1,604,200</td><td>1.94</td><td>10.26</td><td>99.99</td><td>529.06</td></tr>
<tr><td>Proved-HLC</td><td>1,261,813</td><td>0.84</td><td>4.95</td><td>34.22</td><td>200.74</td></tr>
<tr><td>Proved-ROM</td><td>504,400</td><td>0.48</td><td>3.05</td><td>7.85</td><td>49.45</td></tr>
<tr><td>Total Proven</td><td>3,370,413</td><td>1.31</td><td>7.19</td><td>142.06</td><td>779.25</td></tr>
<tr><td>Probable-AGL</td><td>23,238</td><td>1.42</td><td>5.12</td><td>1.06</td><td>3.83</td></tr>
<tr><td>Probable-HLC</td><td>120,413</td><td>0.80</td><td>4.56</td><td>3.12</td><td>17.65</td></tr>
<tr><td>Probable-ROM</td><td>71,988</td><td>0.47</td><td>3.10</td><td>1.09</td><td>7.16</td></tr>
<tr><td>Total Probable</td><td>215,639</td><td>0.76</td><td>4.13</td><td>5.27</td><td>28.64</td></tr>
<tr><td>Proved+Probable</td><td>3,586,052</td><td>1.28</td><td>7.01</td><td>147.33</td><td>807.89</td></tr>
</table>
• The reference point for the Ore Reserves is where the ore is delivered to the processing plant.
• The amount of waste material calculated inside the pit shell is about 3.05 million tonnes, resulting in a strip ratio (ore:waste) of 1:0.83.
</td></tr>
<tr><td><i>Discussion of relative accuracy/ confidence</i></td><td>
• The Ore Reserves has been completed feasibility standard with the data being generated from a tightly spaced drilling grid, thus confidence in the resultant figures is considered high.
• Extraction of ore from the Ugur deposit commenced in August 2017, and processing of the ores will commence in September 2017. As on date of this report, top soil pre-strip has commenced.
• Mining costs and haulage costs will be as per the current contracts in place being utilised at other mines in the contract area.
• Project capital is well managed, and certain infrastructure facilities are available from with the Anglo Asian Mining group, thus minimising capital requirements.
• The global Mineral Resource estimates have been estimated by using a sectional (polygonal) method, and by 3D modelling using both inverse distance and kriging methods. All results are within 5% of each other.
• The Modifying Factors for mining, processing, metallurgical, infrastructure, economic, gold price, legal, environmental, social and governmental factors as referenced above have been applied to the pit design and Ore Reserves calculation on a global scale and data reflects the global assumptions.
• No mine production data is available at this stage for reconciliation and/or comparative purposes.
</td></tr>
</table>

The details of the key modifying factors are discussed in more detail in the following sections of the report.

3-1 Modifying factors used to determine the ore reserve

On establishing the modifying factors, the Mineral Reserves has been optimised using the Datamine NPV Scheduler ® software. This resulted in the economic open pit shell and contained mineable material in that pit shell. Subsequently, this was further optimised in the mine design process, using Datamine Studio OP ® software, where bench toe and crest, catch benches and haul road layout was designed. The final mineable material comprised the Ore Reserves.

3-2 Mining method

The mining method selected is by open pit method given the orebody geometry and the position relative to topographic surface. The central part of the orebody is exposed at surface, and over the remaining 70% surface area of the orebody there is a top soil cover varying in thickness between zero and 50 centimetres. Access to the orebody is from surface. The open pit mining method is considered appropriate, and will comprise conventional truck and shovel.

3-3 Geotechnical Parameters

Pit slope angles have been determined based on independent geotechnical investigation, taking into account geological structure, rock type and design orientation parameters. The overall pit slope angle is 38 degrees containing an average bench angle of 58 degree.

Based on the geotechnical findings, further to the independent report by CQA, the overall pit slope angle is maximum 38 degrees, berm width 6 metres and after each 5 benches (50 metre height), a catch bench of 10 metre width should be considered for the open pit design.

Based on the 10 metre bench height, the slope parameters are shown in Table 3-2. (extract from Table 4 of the CQA geotechnical report).

Table 3-2- Recommended Geotechnical Parameters

Parameter	Value	Units	Comments
Overall slope angle	38	degrees	Max wall height between geotech berms
Average bench angle	58	metres	Toe to crest
Maximum local bench angle	70	metres	Toe to crest
Bench height	10	metres	
Normal bench width	6	metres	Safety berm/ Berm on each bench
Catch bench width	10	metres	Catch bench every 50 m

Figure 3-1 shows a schematic view of five 10m benches.

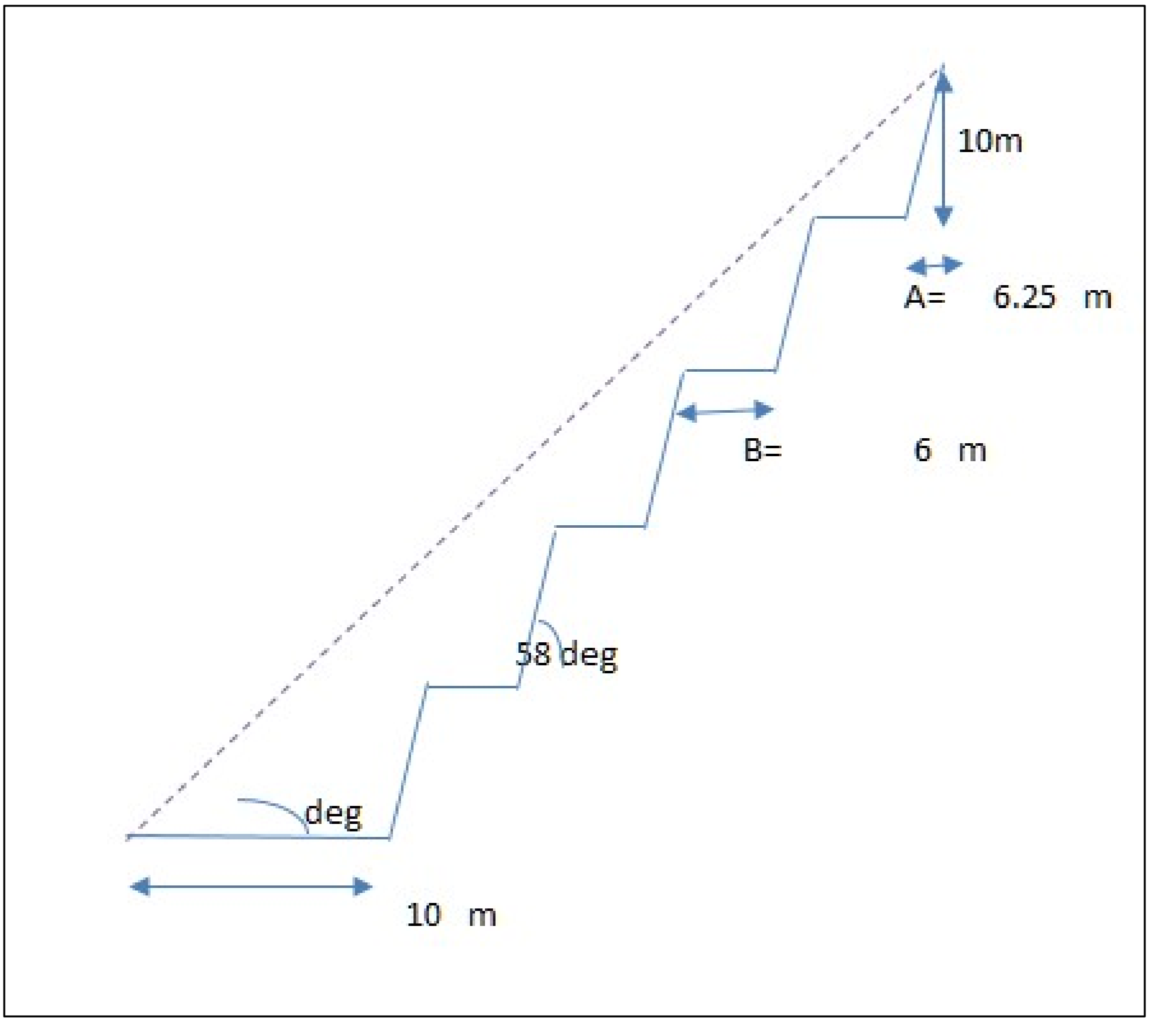


Figure 3-1-Schematic view of one stack of pit wall

These parameters have been applied to all walls irrespective of orientation. The CQA report is referenced in Appendix II.

3-4 Mining Recovery and Ore Dilution

The low grade nature of the deposit, in conjunction with the complex geological setting, makes it very difficult to apply global factors for mining recovery and ore dilution and as the extraction operation has not started at the time of commencement of preparation of this report, hence the correlation between the geological model and actual production was not possible.

A 5% factor for mining dilution and a 95% factor for ore mining recovery were used in the Datamine NPV Scheduler software for reserve estimation.

3-5 Minimum mining width

With consideration of the size of the resource and mining equipment, a minimum mining width of 20 metres has been used. The same width was applied to distances between contiguous pushbacks.

3-6 Inferred Resource implication

The inferred classified material was excluded from economic model in NPV Scheduler so it had no impact on the total reserve. During the pit optimisation, it was determined that the total tonnage of inferred material in the final pit design was 87,100 tonnes which represents about 2.37% of total ore tonnage in the pit and contains 0.76% (1,134 ounces) of contained gold in the pit.

3-7 Infrastructure

Infrastructure is considered excellent to the deposit. The deposit is located within the Company’s contract/licence area with extraction rights according to the Government contract. Ore can be processed at the Company’s current facilities, with ore being delivered by truck from the mine to processing via the newly constructed haul road over a distance of about 6 kilometres. Land availability for the mine and associated infrastructure is approved. Offices and mechanical workshop buildings are available within the company and will be relocated to Ugur. Power for the offices and weighbridge will be initially via diesel generators, although solar power is also under consideration. Labour is readily available as the operation is relatively small and only additional mine site labour will be required. General and administration (G&A) and process labour are part of the existing company compliment of staff. Regarding accommodation, canteen facilities and associated services, the Ugur deposit can be considered a “satellite” deposit to the current mining operations and will be serviced by the current infrastructure.

3-8 Metallurgical Factors

AIMC currently operates an agitated leach plant, a flotation plant, a crushed heap leach facility, and a run-of-mine dump leach facility at the Gedabek mine site. As such, the basis for assumptions and predictions of processing routes and type of “ores” suitable for each process available are well understood. The current processing method at Gedabek and other adjustment to feed Ugur ore, is detailed in this section.

3-8-1 Gedabek Processing Methods

The process recoveries for the various process routes are specified for gold and silver and copper. For HL and ALP, it is assumed that the SART process is used.

The SART process, developed jointly by SGS Lakefield and Teck Corporation, can remove the metallurgical interference of leachable copper and silver, and regenerate cyanide so that it can be recycled to the gold operation. The claimed benefits (SGS, 2007) are:

- The revenues received from the sale of the copper sulphide precipitate will likely exceed the operating costs of the SART process. Therefore, the process can add value to a project to the extent that copper leaches naturally from the ore during gold leaching.
- The cyanide associated with the copper complex is recycled as free cyanide, available for further leaching. The cost of this cyanide is part of the overall cost of the SART process, which is more than covered by revenues from copper sales, i.e. this represents a source of zero cost cyanide.
- The alternative to SART would be to process the HL liquor either periodically during the operating life of the mine or at the end of the project, with a cyanide detoxification process such as the SO2/air process. The significant costs associated with this process will be avoided by incorporation of the SART process.
- If the SART process is installed ahead of the gold recovery process, the removal of copper and most of the silver in SART significantly simplifies the gold recovery operation, whether it is by adsorption of gold on carbon or resin, or cementation on zinc powder.

It is interesting to note that SGS (2007) reported that the recovery of gold from heap leaching is relatively insensitive to grade in the range 0.76 g/t to 3.73 g/t Au. A recovery of 70% was achieved with half-inch particle size, and this increased to 90% with a particle size of 75 microns.

3-8-2 Ugur Processing Method

A Metallurgical testwork programme has been carried out to assess the amenability of the Ugur mineralisation to cyanidation and leaching processes by current Gedabek AGL plant and Heap Leach process. The results showed a high level of amenability. The mineralisation is an “oxide” type, that is relatively soft, and requires comparatively low levels of processing reagents for recovery.

The metallurgical testwork was carried out on samples with a mean of a range of gold grades; 3.6g/t, 2.5g/t, 1.5g/t and 1.0g/t. The results for a 48 hour bottle roll test showed high gold recovery and low cyanide usage (see below).

Table 3-3- Leaching Recovery

Leaching, %	
Au	Ag
88.5	82.8
85.7	62.0
95.0	60.5
83.8	73.2

The process costs and overall recoveries used (including the contribution from SART) are shown in Table 3-4 .

Table 3-4-Metallurgical recovery factors

Processes	Recovery	
	Au%	Ag%
Agitation Plant	90%	66%
Heap Leach (Dore) Crushed ore	70%	7%
Heap Leach ROM (Dore)	40%	7%

No metallurgical factors assumptions have been used in mineral resource estimate.

3-9 Financial Parameters

The financial parameters used to determine the NSR and block values are shown in Table 3-5.

Table 3-5- Financial parameters

Parameter	Units	Value
Total G&A	US$/t ore	3.22
AGL Cost	US$/t	26.00
HLC cost	US$/t	3.15
HLROM Cost	US$/t	2.00
Mining cost	US$/t mined	1.75

The AGL cost of 26.00$/t is when material from Ugur is mixed with other sources, however recent operation results shows that the AGL cost is less than 26.00$/t if just material from Ugur is processed. This also applies on G&A cost and the recent result from operation shows that it can be reduced to 2.00$/t. Although reducing the processing cost will improve the economics of the open pit operation, but as it is shown in Section 4, the selected optimum pit is based on 64% of the base price. Hence, whilst the reduced processing cost and G&A can be used in AIMC financial models, it is unlikely to have any significant impact on the final pit shell.

The selling price is deduced from the market price to determine the NSR. The values used are specified by process route and product shown in Table 3-6.

It should be noted that the cyanide used in HL is regenerated cyanide from the SART process and that, therefore, there is no cyanide cost in HL processing cost.

Table 3-6-Selling costs

Processes	Selling price - Net of refining and	
	Au	Ag
Agitation Plant (Dore)	99.95% Payable	100% Payable
Heap Leach Crushed Ore	99.95% Payable	100% Payable
Heap Leach ROM (Dore)	99.95% Payable	100%Payable (SART)

3-10 Cut-off parameters

Financial factors included in the cut-off grade estimates and calculations included process and overhead costs, mining dilution, payable gold and silver price, and processing recovery.

The ore from Ugur can be processed by three different available processing methods within the Gedabek contract area, namely agitation leach (AGL), heap leach of crushed material (HLC) and heap leach of blasted material or run-of-mine (ROM). The acceptable gold head grade in grammes per tonne gold for AGL, HLC and ROM is 1.8 g/t, 0.8 g/t and 0.47 g/t respectively.

Further to the gold cut-off grade calculations, after long term scheduling the mill cut-off grade resulted in 0.3 g/t gold.

It can be shown that the theoretical cut-off grades for AGL, HLC and HLROM are 0.81 g/t Au, 0.23 g/t Au and 0.32 g/t Au respectively. Based on the assumed grades of 7 g/t for silver, cut-off grades for AGL, HLC and HLROM are 0.76 g/t Au, 0.23 g/t Au and 0.32 g/t respectively (see Table 3-7).

Table 3-7-Calculated Cut-offs

Processes	Calculated Cut-offs	
	Au Cut-off	assumed grades of 7 g/t for silver
AGL	0.81	0.76
HLC	0.23	0.22
HLROM	0.32	0.32

3-11 Environmental factors

Previous ESIA (Environmental Social Impact Assessment) has been carried out by Amec Foster Wheeler (2012) and TexEkoMarkazMMC (2012) (submitted to Government authorities). The Ugur deposit is located within the Gedabek Contract Area for which the ESIA is valid, hence the most recent ESIA is applicable to Ugur. Processing and tailings storage reported in the ESIA is the same as will be utilised for Ugur ores.

Environmental and geotechnical consultants, CQA International Ltd of the UK (CQA), have on-site representation, and carried out both geotechnical and environmental assessments of the Ugur mine area. Baseline environmental monitoring has been carried out on receptors downstream of the mine site, due to an additional catchment being located in the vicinity of the Ugur mine.

The waste rock has a low potential for acid rock drainage due to the absence of sulphide bearing mineralisation. Watercourses downstream of stockpiles will be monitored on a routine basis for pH and heavy metals.

A topsoil management plan is in place, that has been reviewed by a CQA consultant deemed in accordance with the storage principles of the Ministry of Ecology and Natural Resources of the Republic of Azerbaijan and European Union (EU) guidelines. Topsoil removal took place in August 2017, and be stockpiled in a dedicated location with specific design parameters. Stockpiling of materials will be carried out following the soil management plan.

A stockpile area for waste rock has been identified following condemnation drilling verifying the absence of mineralisation beneath the proposed stockpile. The top soil at the planned site will be removed, and the hill terraced to “key” in the waste dump for maximum stability.

The tailings management facility (TMF) has the capability for the additional storage requirements for Ugur process waste. The design and operations of the TMF have been reviewed by CQA along with a visit by the Ministry of Ecology and Natural Resources of the Republic of Azerbaijan. Regular environmental monitoring is carried out at the TMF, along with monitoring all receptors associated with the TMF.

All approvals for conducting the mining fall under the management “PSA” agreement.

3-12 Legal Tenure

Legal tenure of the Ugur deposit is held by AIMC under the production sharing agreement (PSA).

There are no material naturally occurring risks associated with the Ore Reserves.

Anglo Asian Mining plc is currently compliant with all legal and regulatory agreements, and marketing arrangements.

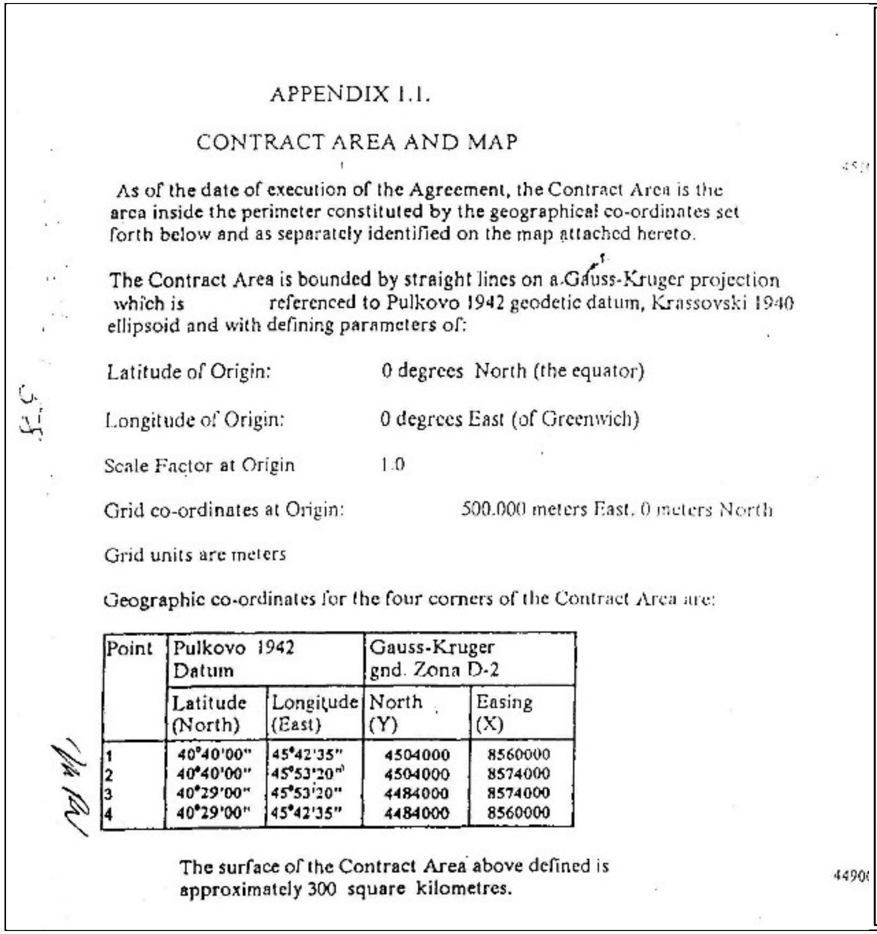
The project is located within a current contract area that is managed under a “PSA” production sharing agreement (Figure 3-2).

The PSA grants the Company a number of periods to exploit defined licence areas, known as Contract Areas, agreed on the initial signing with the Azerbaijan Ministry of Ecology and Natural Resources ('MENR'). The exploration period allowed for the early exploration of the Contract Areas to assess prospectively can be extended.

A 'development and production period' commences on the date that the Company issues a notice of discovery, which runs for 15 years with two extensions of five years each at the option of the Company. Full management control of mining in the Contract Areas rests with Anglo Asian Mining.

Under the PSA, Anglo Asian Mining is not subject to currency exchange restrictions and all imports and exports are free of tax or other restriction. In addition, MENR is to use its best endeavours to make available all necessary land, its own facilities and equipment and to assist with infrastructure.

The PSA is valid for the forecast life of mine.



APPENDIX 1.1.

CONTRACT AREA AND MAP

As of the date of execution of the Agreement, the Contract Area is the area inside the perimeter constituted by the geographical co-ordinates set forth below and as separately identified on the map attached hereto.

The Contract Area is bounded by straight lines on a Gauss-Kruger projection which is referenced to Pulkovo 1942 geodetic datum, Krassovski 1940 ellipsoid and with defining parameters of:

Latitude of Origin:	0 degrees North (the equator)
Longitude of Origin:	0 degrees East (of Greenwich)
Scale Factor at Origin	1.0
Grid co-ordinates at Origin:	500.000 meters East. 0 meters North

Grid units are meters

Geographic co-ordinates for the four corners of the Contract Area are:

Point	Pulkovo 1942 Datum		Gauss-Kruger gnd. Zona D-2	
	Latitude (North)	Longitude (East)	North (Y)	Easing (X)
1	40°40'00"	45°42'35"	4504000	8560000
2	40°40'00"	45°53'20"	4504000	8574000
3	40°29'00"	45°53'20"	4484000	8574000
4	40°29'00"	45°42'35"	4484000	8560000

The surface of the Contract Area above defined is approximately 300 square kilometres.

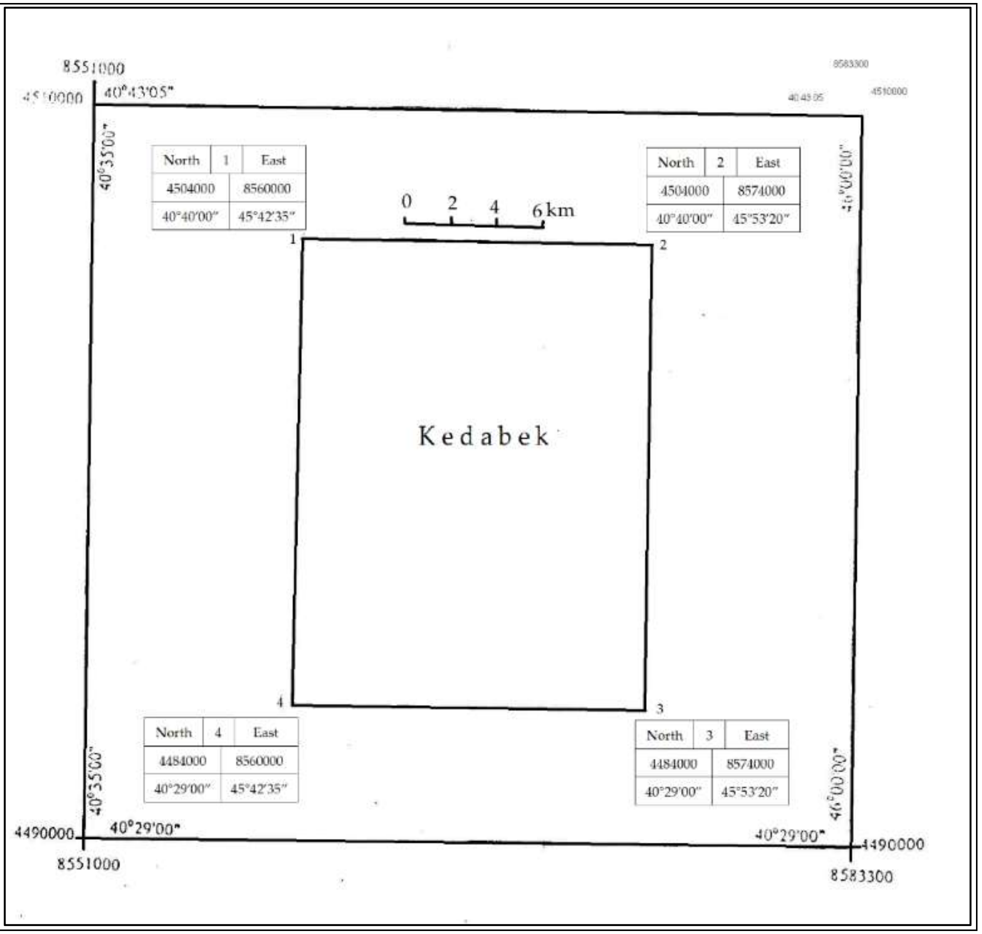


Figure 3-2- Location coordinates of the Gedabek Contract Area

4 Pit Optimisation

The open pit optimisation was run using software “NPVS”, which uses the standard Lerchs-Grossman algorithm to determine the pit limit and incremental pit shell. The latter are used as a guide for selecting the pit limit and as the basis for the creation of a sequence of pushbacks within the pit limit.

The main input parameters to NPVS are:

- Product prices (gold and silver)
- Selling prices
- Mining cost
- Process cost (by process route)
- Process recovery (by process route)
- Slope parameters.

NPVS was set up so that the rock types are further subdivided into Measured, Indicated and Inferred for reporting purposes. When determining the pit limit and Reserves, the grades for the Inferred material are given a value of zero as they cannot be included in the valuation. However, it is useful to report these values as they represent a potential ore source should it be possible to reclassify them in the future.

The parameters used on the optimisation are discussed in Section 5 and a summary of the results from the pit optimisation are discussed below.

4-1 Results

The pit optimisation was run with an increment of 1% for Price Factor so as to determine if there was a logical breakpoint at which to select the pit limit. Note that at a Price Factor of 100%, the metal prices will be equal to the assumed prices presented in Section 5 of this report.

It can be seen in Figure 4-1 that at Pit 51 (64% Price Factor) the cumulative NPVs is flattened and the increase in NPV over this increment is relatively small as more than 98% of the final value has already been achieved this while 88% of ore in ultimate pit LG is mined.

As shown in Figure 4-2 the ultimate pit could, therefore, be treated as a potential expansion for the future if prices rise and also further exploration shows higher grade gold in the boundary between Pit 51 and ultimate pit.

Overall the pit optimisation is “well behaved” and provides a good framework from which a detailed mine design can be produced. This mine design will take into account the detailed geotechnical parameters of batters and berms as well as ensuring that there is access space to develop the mine.

Pit 51 has been selected as a suitable point from which the mine design can commence. This does not preclude the opportunity to further expand the pit whilst ensuring that the project value has been maximised within the practical constraints such as fleet capacity. The ore reserve should, therefore, be based on Pit 51 and not one of the larger shells.

Figure 4-1- Plot of cumulative ore tonnage and NPV versus pit shell number

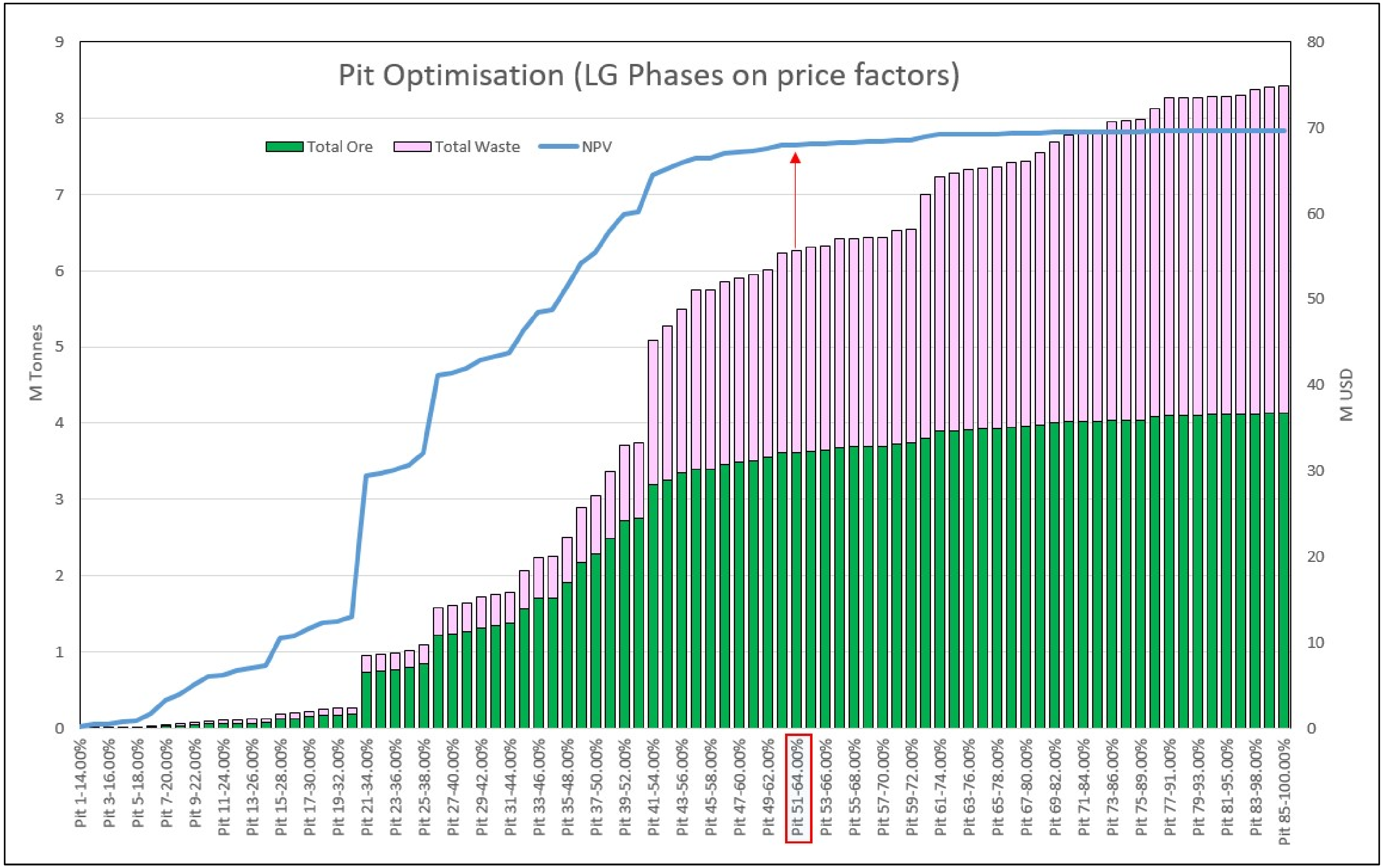
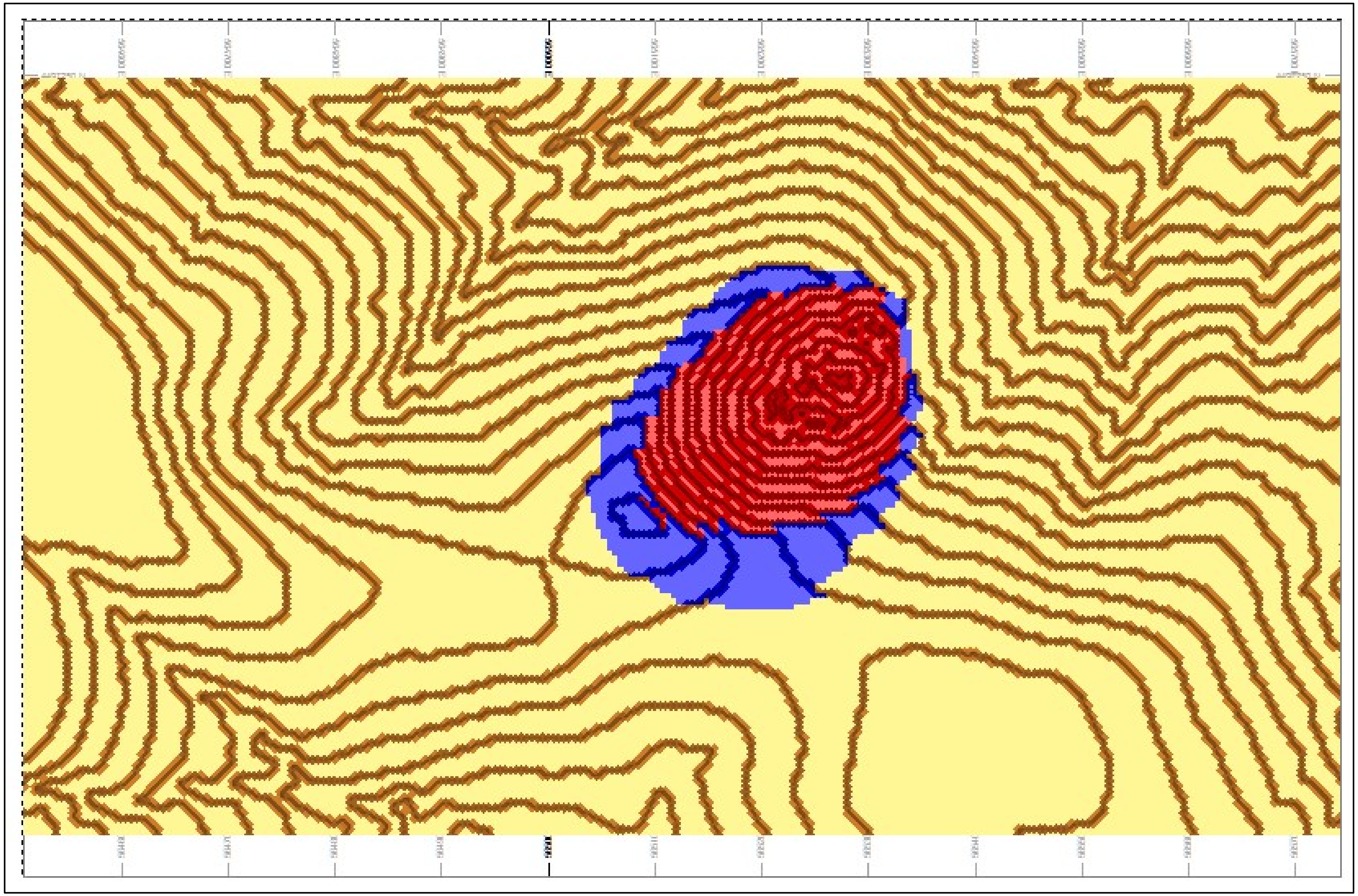


Figure 4-2-Optimised pit limit shown in red with potential expansion in blue



5 Mine Design

Based on the selected pit limit described in Section 4, mine designs were prepared for:

- Final pit limit
- Interim pit stages/pushbacks

The pit limit and pushbacks are designed according to the geotechnical parameters discussed in Section 3. It should be noted that the total tonnage within the pit limit will vary slightly from that shown in the optimisation due to the batter angle and smoothing of the wall to avoid potential geotechnical issues with “noses” etc.

The designs are discussed below along with the resulting reserves.

5-1 Pit Limit Design

The final pit wall has been designed to include a 10 metre wide catch bench every five benches (50 metres). This bench acts as a haul road for the 30 tonne trucks, so that they can exit either side of the slope and link to the roads to the waste dumps or crusher, as shown in Figure 5-1.

The access to the pit can be made from the south or south west of the pit. Due to the limited width of each pushback, the pit wall has to be mined bench by bench from the top down but during the operation the final ramp access of each pushback needs to be maintained to make sure material can be transported to AGL, HL or waste dump.

Some internal ramps will be required to make sure that there’s an easy access to pit exit points. The ramp system of Ugur pit can be seen in Figure 5-1.

Besides determining the optimal extent of the open pit, an important aspect of the mine design is the distribution of material types within this pit. This is shown in Figure 5-2 to Figure 5-6 in terms of the assigned process route (AGL, HLC and HLROM). The mine sequence is constrained by the capacity of the Agitation Plant and the crusher which is designated for material feeding the heap leach.

This is discussed in more detail in Section 8.

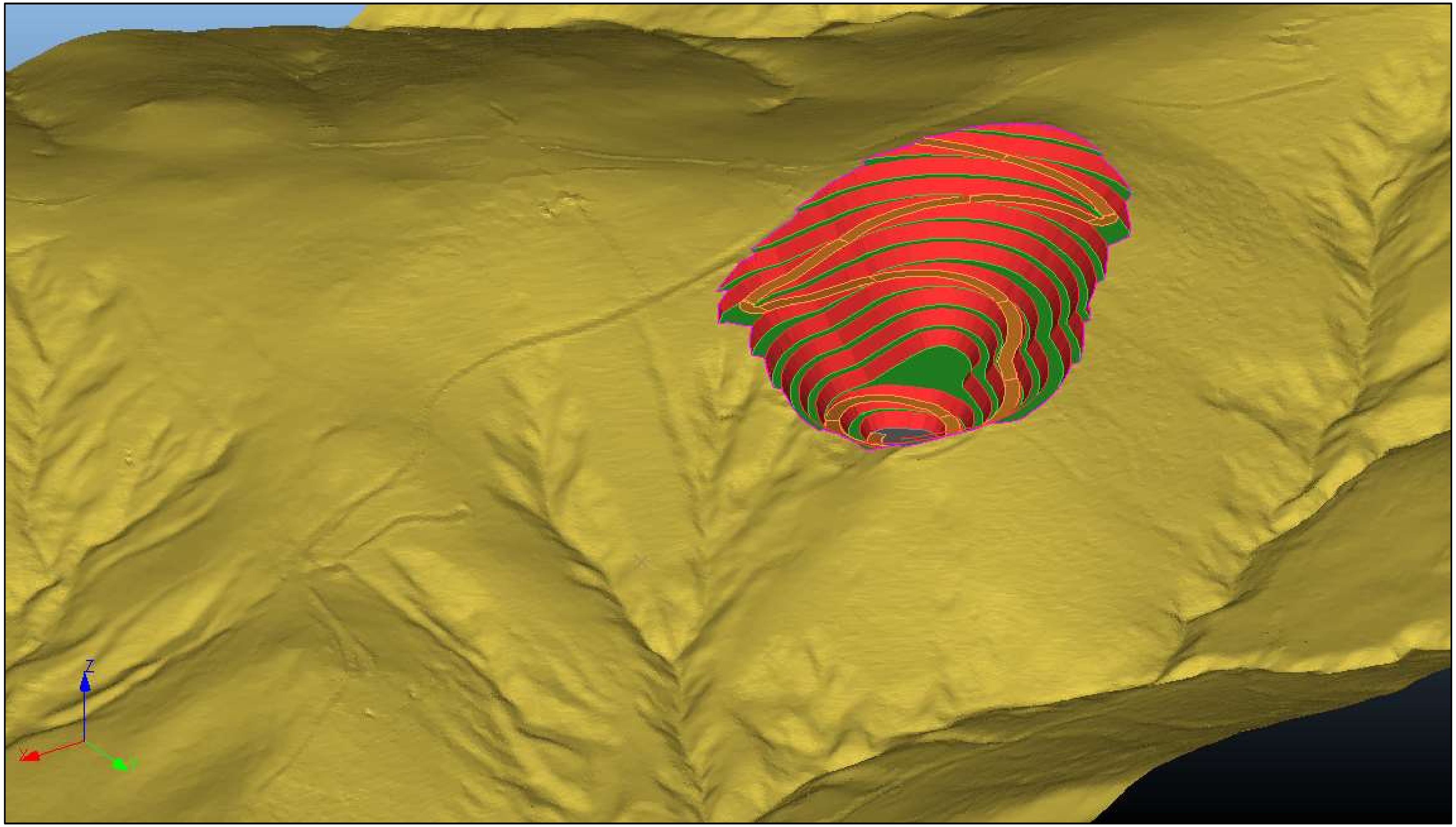


Figure 5-1-Final Pit limit design based on LG Pit 51

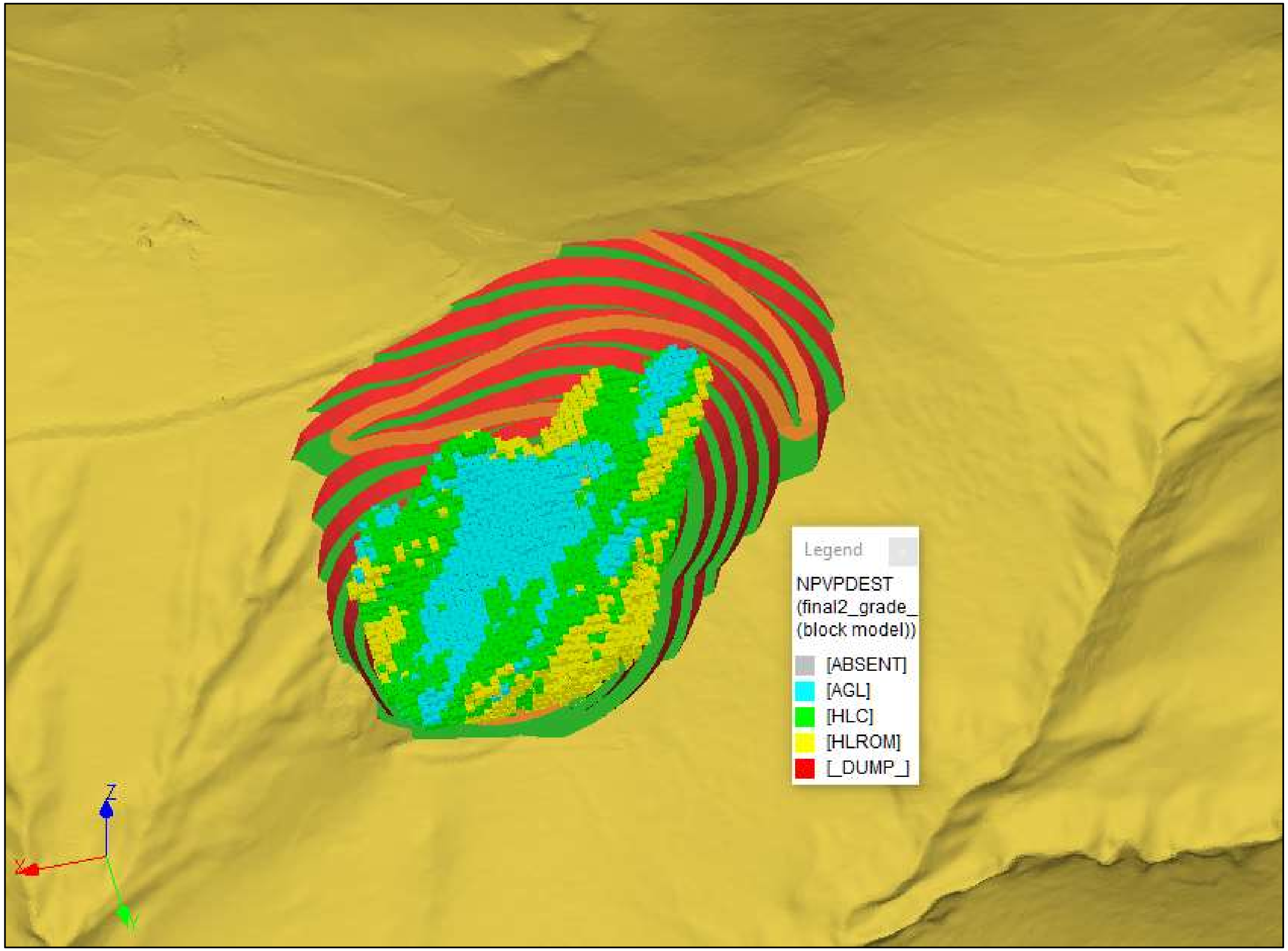


Figure 5-2- Distribution of material types with the final pit limit

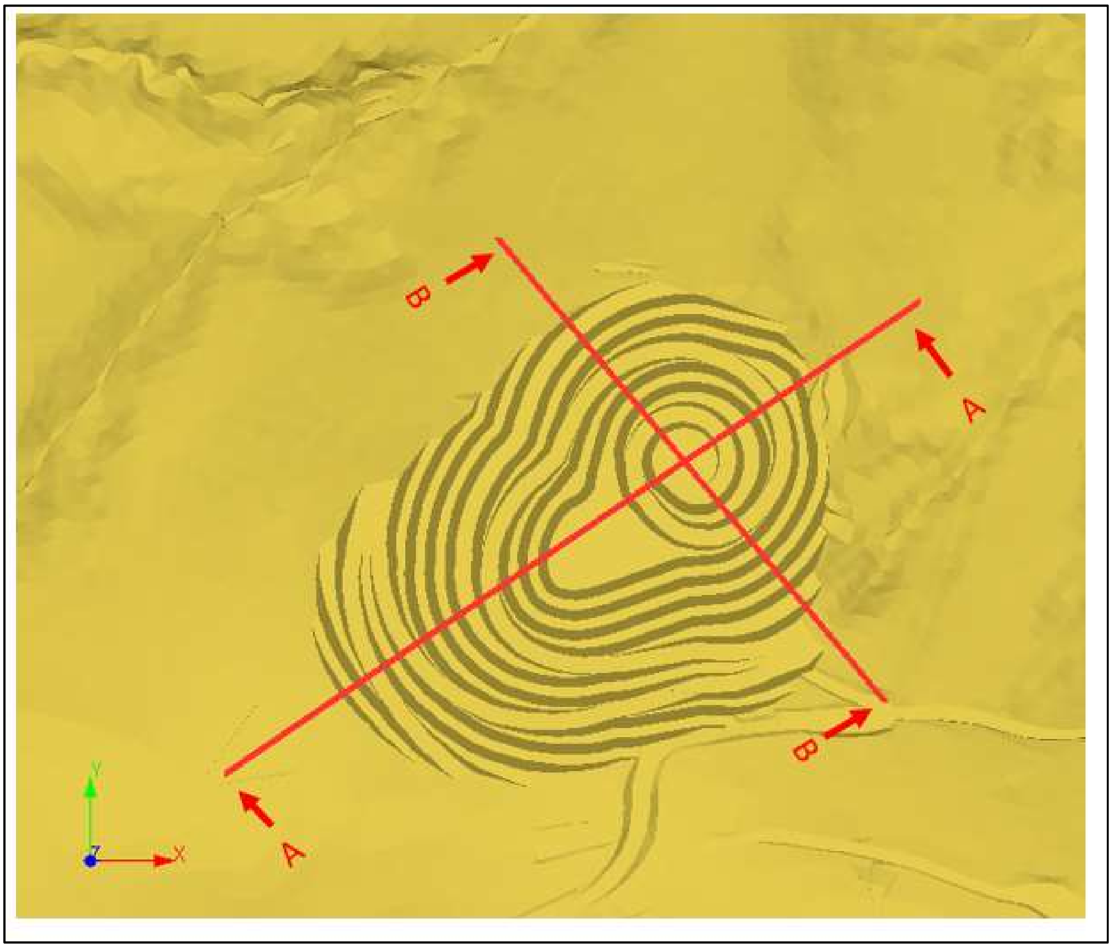


Figure 5-3-Final Pit cross sections

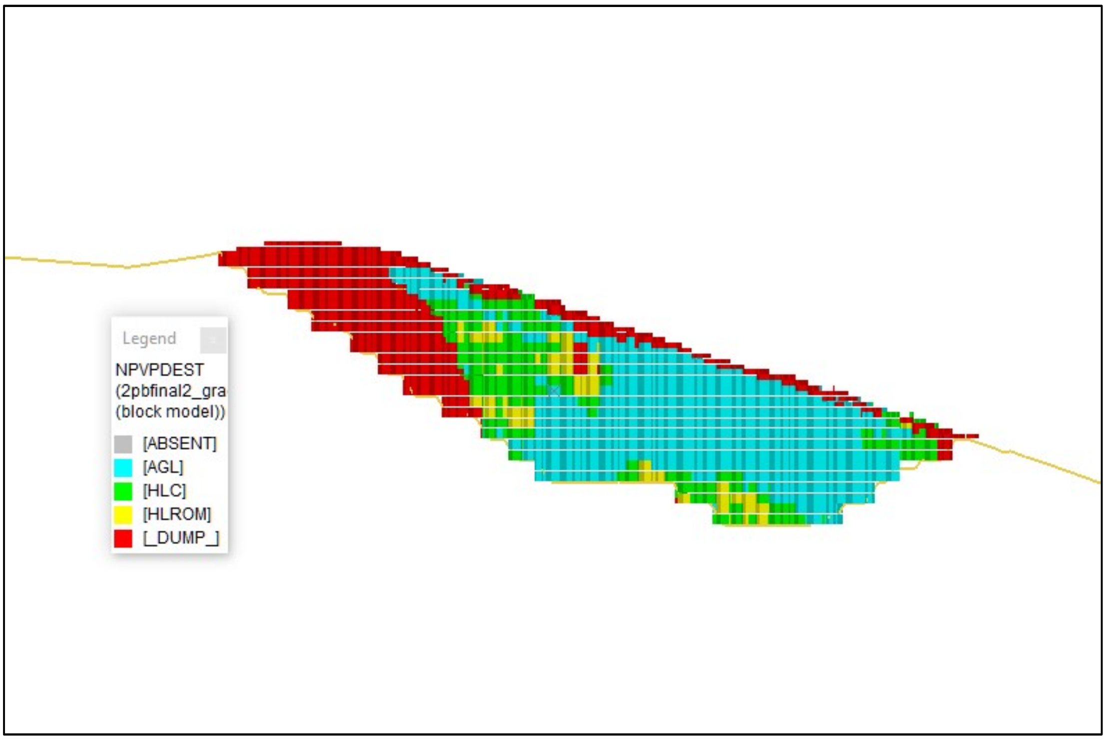


Figure 5-4-Section A-A

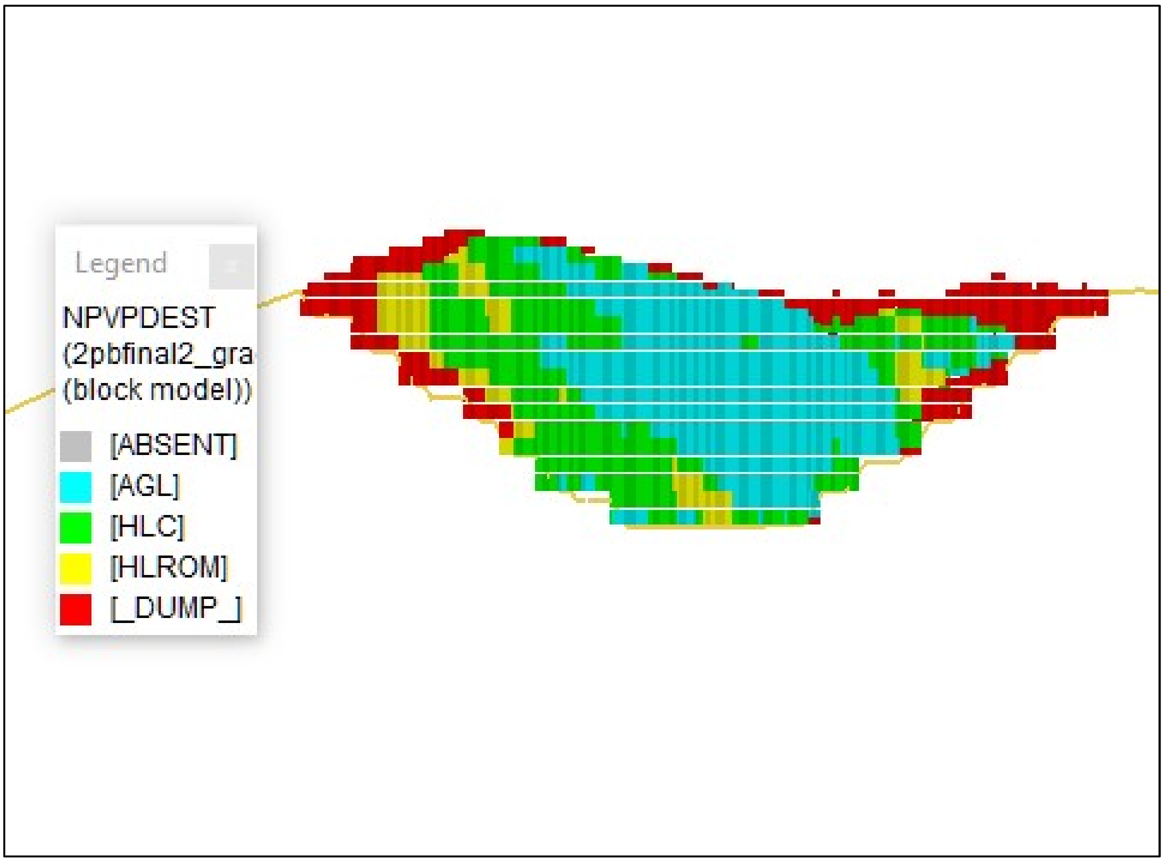


Figure 5-5-Section B-B

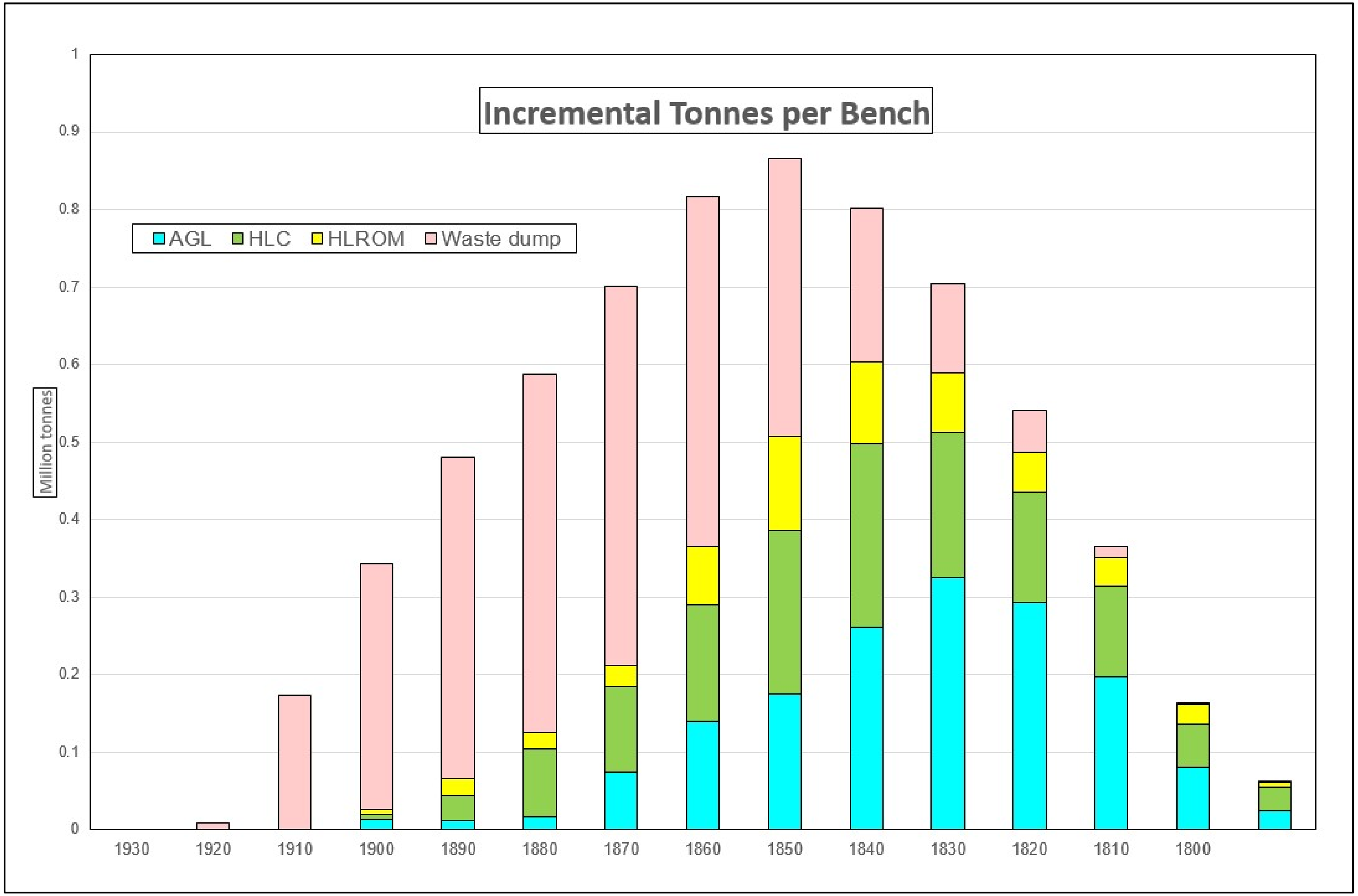


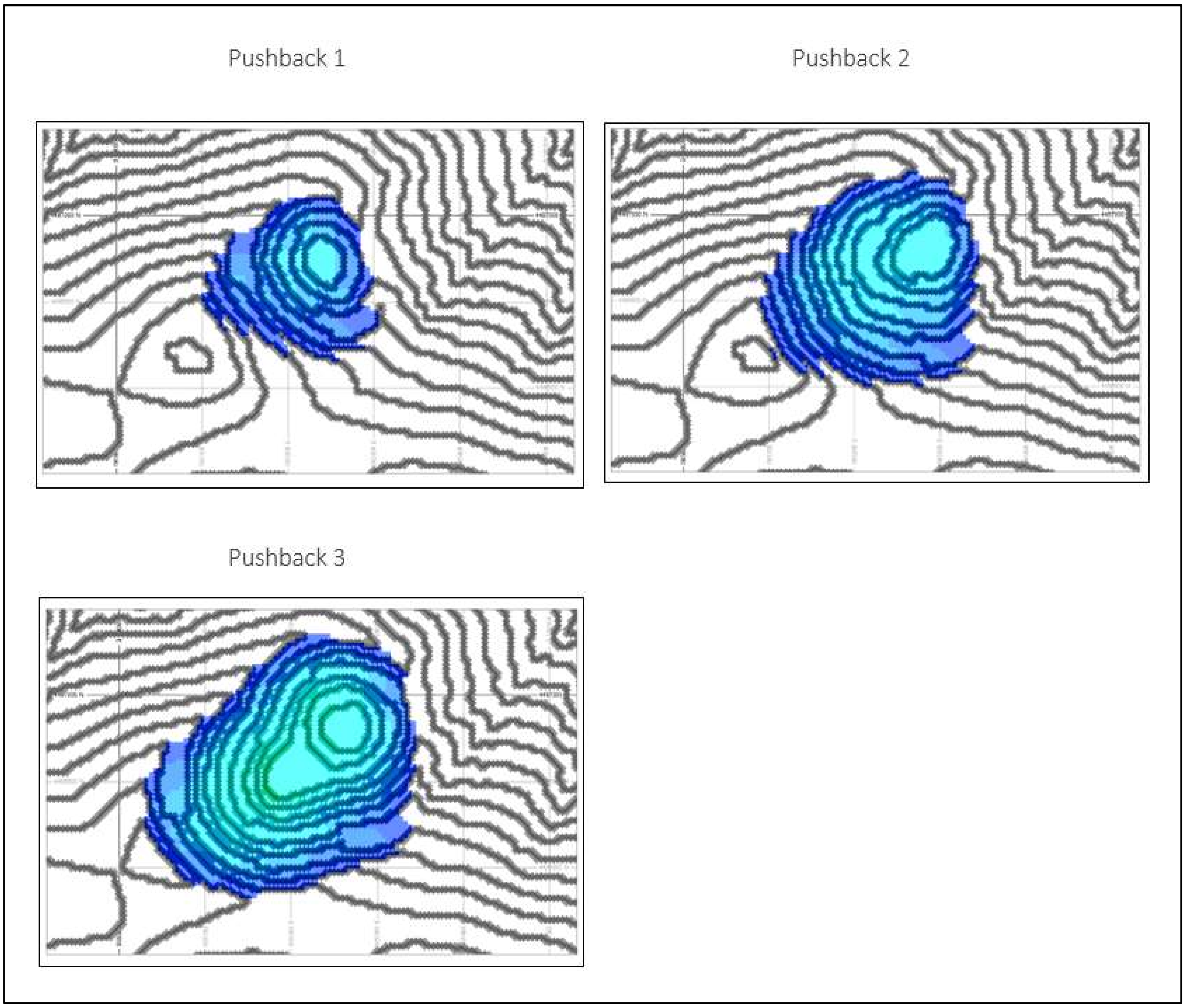
Figure 5-6- Distribution of material types, classified by processing route

5-2 Pushback Design

The main constraints on the design of the pushbacks are:

- Slope design parameters (Table 3-2)
- Bench access to pit exits at all times
- Minimum bench width for equipment (20 metres)
- Maximum bench sinking rate (12 benches per year)
- Blending to plant feed requirements

Three pushbacks were designed to accommodate all production requirements and physical constraints. The sequence of pushbacks is shown in Figure 5-7 and the reserves by pushback are shown in Figure 5-8.



Pushback 1

Pushback 2

Pushback 3

Figure 5-7-Pushback sequence

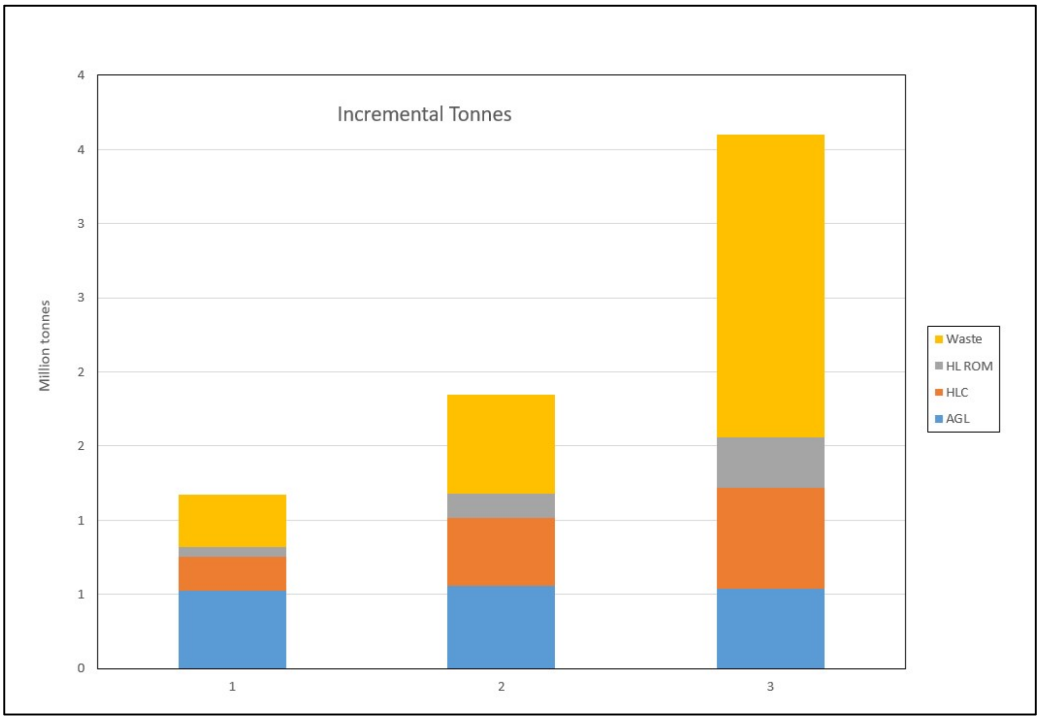


Figure 5-8- Tonnage by pushback (incremental)

It is evident that the distribution of material types suitable for AGL by pushback is even over the life of mine, however, the material type suitable for HLC and HLROM will increase through successive pushbacks (Figure 5-8). As it's shown in Section 6 that this will not be a major issue when scheduling. It can also be seen that the waste stripping ratio is 0.43 and 0.56 in the first and second pushback respectively while this will be increased up to 1.3 in the last pushback, where the Au grade of material is decreased. In general, the scheduling of the pushbacks behaves well.

5-3 Reserves

It is concluded that the forecast mineable material following a pushback design option for the Ugur open pit is 3.561 Mt, with a contained metal content of 4,552.4 kg of Au (146,360 oz) of Au, 25,057.0 of Ag (805600 oz).

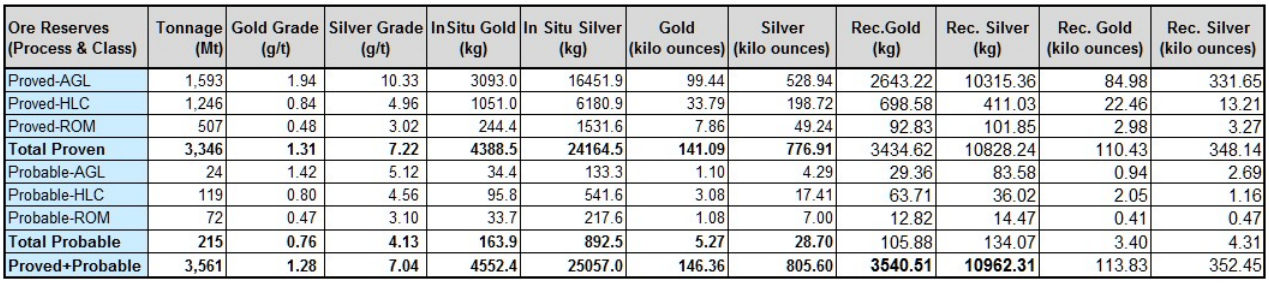
The Reserves are summarised in Table 5-1 in terms of the Reserve categories of Proven and Probable, where Proven and Probable relate directly to Measured and Indicated resource classes.

The total waste (including mineralised material that is uneconomic) within the pit is 3.06 Mt, giving a total rock tonnage of 6.62 Mt and an average waste stripping ratio of 0.86. This is relatively low for an open pit. Consideration must be given to the low grade nature of the deposit and the need to mine to a very low cut-off grade.

The potential for expanding the reserves lies with:

- Upgrading the Inferred Resource (additional 0.568 Mt, Au Cut-off=0.3g/t)
- Expanding the pit beyond Pit Shell 51 (additional 0.508 Mt, Au Cut-off=0.3g/t)

Table 5-1-Reserve summary (following detailed pushback design)

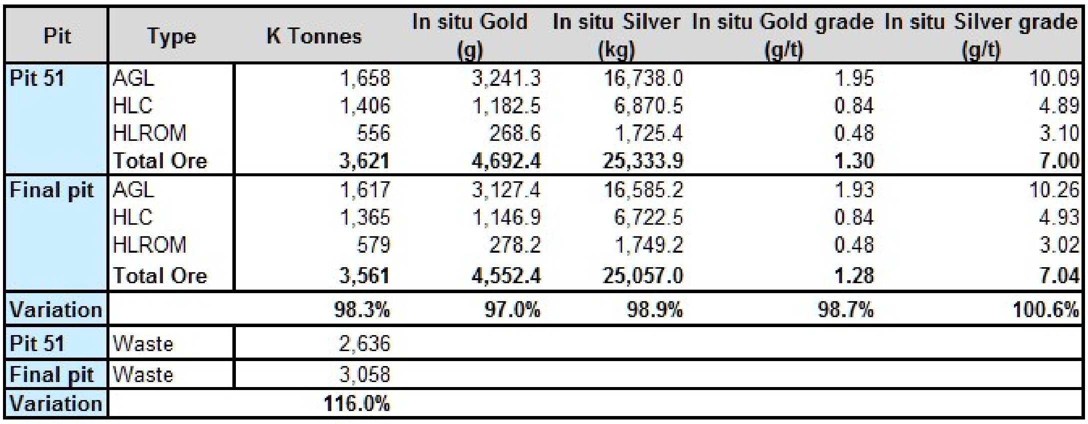


Ore Reserves (Process & Class)	Tonnage (Mt)	Gold Grade (g/t)	Silver Grade (g/t)	In Situ Gold (kg)	In Situ Silver (kg)	Gold (kilo ounces)	Silver (kilo ounces)	Rec.Gold (kg)	Rec. Silver (kg)	Rec. Gold (kilo ounces)	Rec. Silver (kilo ounces)
Proved-AGL	1,593	1.94	10.33	3093.0	16451.9	99.44	528.94	2643.22	10315.36	84.98	331.65
Proved-HLC	1,246	0.84	4.96	1051.0	6180.9	33.79	198.72	698.58	411.03	22.46	13.21
Proved-ROM	507	0.48	3.02	244.4	1531.6	7.86	49.24	92.83	101.85	2.98	3.27
Total Proven	**3,346**	**1.31**	**7.22**	**4388.5**	**24164.5**	**141.09**	**776.91**	3434.62	10828.24	110.43	348.14
Probable-AGL	24	1.42	5.12	34.4	133.3	1.10	4.29	29.36	83.58	0.94	2.69
Probable-HLC	119	0.80	4.56	95.8	541.6	3.08	17.41	63.71	36.02	2.05	1.16
Probable-ROM	72	0.47	3.10	33.7	217.6	1.08	7.00	12.82	14.47	0.41	0.47
Total Probable	**215**	**0.76**	**4.13**	**163.9**	**892.5**	**5.27**	**28.70**	105.88	134.07	3.40	4.31
Proved+Probable	**3,561**	**1.28**	**7.04**	**4552.4**	**25057.0**	**146.36**	**805.60**	**3540.51**	**10962.31**	113.83	352.45

Note: tonnes (dry) and grades shown are after applying reconciliation factors.

A comparison of the reserves for the selected pit limit (Pit Shell 51) and the detailed pit design in Table 5-1 shows that there is less than a maximum 3% variance in the total ore within the pit (see Table 5-2). The pit design has, therefore, followed the guidelines provided by the pit optimisation with minimal loss of ore as a result of imposing practical mining constraints on the design which has 16% more waste tonnage.

Table 5-2-Reserves comparison (pit design versus optimised pit limit - Pit Shell 51)



Pit	Type	K Tonnes	In situ Gold (g)	In situ Silver (kg)	In situ Gold grade (g/t)	In situ Silver grade (g/t)
Pit 51	AGL	1,658	3,241.3	16,738.0	1.95	10.09
	HLC	1,406	1,182.5	6,870.5	0.84	4.89
	HLROM	556	268.6	1,725.4	0.48	3.10
	Total Ore	**3,621**	**4,692.4**	**25,333.9**	**1.30**	**7.00**
Final pit	AGL	1,617	3,127.4	16,585.2	1.93	10.26
	HLC	1,365	1,146.9	6,722.5	0.84	4.93
	HLROM	579	278.2	1,749.2	0.48	3.02
	Total Ore	**3,561**	**4,552.4**	**25,057.0**	**1.28**	**7.04**
Variation		**98.3%**	**97.0%**	**98.9%**	**98.7%**	**100.6%**
Pit 51	Waste	2,636				
Final pit	Waste	3,058				
Variation		**116.0%**				

Notes:

1. Process routes are AGL and HL (Crushed and ROM)
2. Tonnes and grades are based on in situ rock and the ore mining recovery and dilution were used in NPVS for only optimisation purposes.
3. Pit optimisation only uses the overall slope angle and is based on the parent cell size.
4. Pit design uses the slope design parameters, including batter angle, berms and maximum stack height.

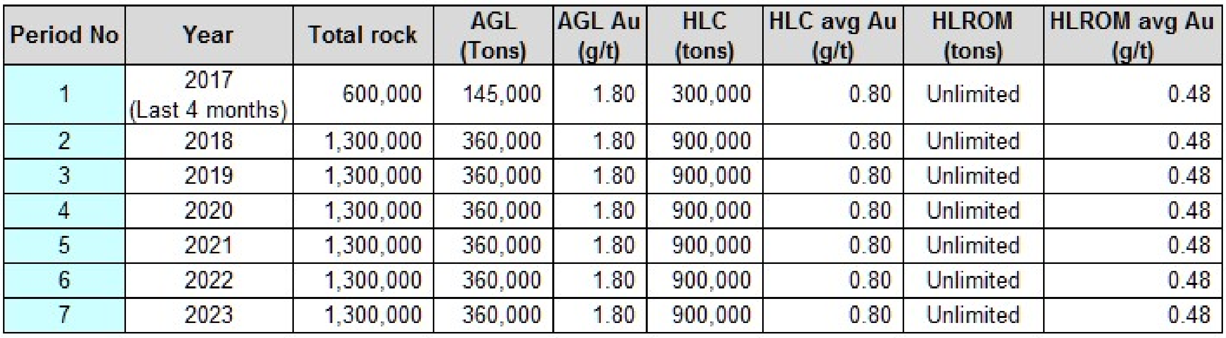
6 Scheduling

Using the pit design and pushback sequence described in Section 5, a life-of-mine (LOM) schedule was created in order to demonstrate that an acceptable mining sequence can be achieved, whilst honouring the various constraints.

6-1 Plant Production

The destination for each ore block depends a set of constraints for Au in Figure 6-1. The main constraints imposed on the schedule are shown in Table 6-1 :

Table 6-1- Ugur Mine Production Constraints



Period No	Year	Total rock	AGL (Tons)	AGL Au (g/t)	HLC (tons)	HLC avg Au (g/t)	HLROM (tons)	HLROM avg Au (g/t)
1	2017 (Last 4 months)	600,000	145,000	1.80	300,000	0.80	Unlimited	0.48
2	2018	1,300,000	360,000	1.80	900,000	0.80	Unlimited	0.48
3	2019	1,300,000	360,000	1.80	900,000	0.80	Unlimited	0.48
4	2020	1,300,000	360,000	1.80	900,000	0.80	Unlimited	0.48
5	2021	1,300,000	360,000	1.80	900,000	0.80	Unlimited	0.48
6	2022	1,300,000	360,000	1.80	900,000	0.80	Unlimited	0.48
7	2023	1,300,000	360,000	1.80	900,000	0.80	Unlimited	0.48

The maximum capacity of the agitation plant (AGL), is 600,000 tonnes per annum, however just 360,000 t/a is planned to be fed by material from Ugur open pit. The remaining capacity will be used for feeding material from other production material sources.

6-2 Mine Production

Considering all production constraints and using the designed pushbacks, a life of mine schedule was created in NPVS.

The variation in mined grade (in-situ) by material type is shown in Figure 6-1. This shows that the gold grade going to AGL or HL is relatively well behaved and does not pose many issues for grade control. The nature of the Ugur resource is the main reason for decreasing the Au grade within subsequent years. The Au grade in the last year of production can be increased by blending with other production material resources. It may be possible to reduce the grade fluctuations with further detailed scheduling but at this level of study it is considered preferable to manage the fluctuations in the mix of materials by stockpiling management so as to simplify the mining sequence.

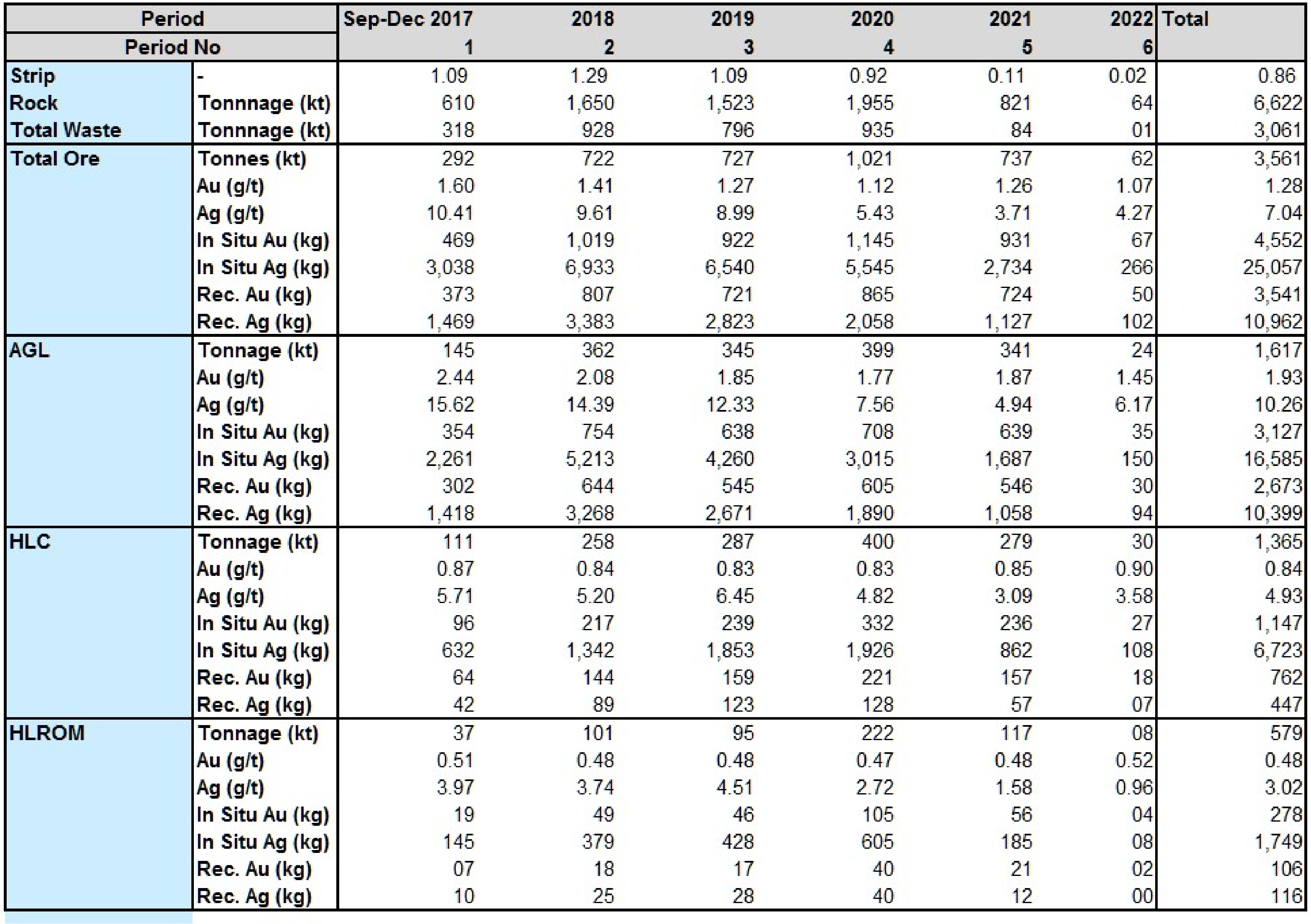
The total material movement indicates that the required fleet capacity peaks at 1.96 Mt of rock in 2020 and then gradually declines (see Table 6-2).

In order to reach the head grade targets in Table 6-1, different scenarios using Datamine NPVS, the minimum gold cut-off grade of AGL, HLC and HLROM were set to 1.1, 0.6 and 0.3 g/t Au respectively.

The three pushbacks design has been relatively successful in meeting the constraints of blending whilst avoiding excessive advance stripping.

Schematic view of Ugur annual plans are illustrated in Figure 6-2.

Table 6-2-Total material movement from the pit



Period		**Sep-Dec 2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**Total**
Period No		**1**	**2**	**3**	**4**	**5**	**6**	
Strip	-	1.09	1.29	1.09	0.92	0.11	0.02	0.86
Rock	**Tonnnage (kt)**	610	1,650	1,523	1,955	821	64	6,622
Total Waste	**Tonnnage (kt)**	318	928	796	935	84	01	3,061
Total Ore	**Tonnes (kt)**	292	722	727	1,021	737	62	3,561
	Au (g/t)	1.60	1.41	1.27	1.12	1.26	1.07	1.28
	Ag (g/t)	10.41	9.61	8.99	5.43	3.71	4.27	7.04
	In Situ Au (kg)	469	1,019	922	1,145	931	67	4,552
	In Situ Ag (kg)	3,038	6,933	6,540	5,545	2,734	266	25,057
	Rec. Au (kg)	373	807	721	865	724	50	3,541
	Rec. Ag (kg)	1,469	3,383	2,823	2,058	1,127	102	10,962
AGL	**Tonnage (kt)**	145	362	345	399	341	24	1,617
	Au (g/t)	2.44	2.08	1.85	1.77	1.87	1.45	1.93
	Ag (g/t)	15.62	14.39	12.33	7.56	4.94	6.17	10.26
	In Situ Au (kg)	354	754	638	708	639	35	3,127
	In Situ Ag (kg)	2,261	5,213	4,260	3,015	1,687	150	16,585
	Rec. Au (kg)	302	644	545	605	546	30	2,673
	Rec. Ag (kg)	1,418	3,268	2,671	1,890	1,058	94	10,399
HLC	**Tonnage (kt)**	111	258	287	400	279	30	1,365
	Au (g/t)	0.87	0.84	0.83	0.83	0.85	0.90	0.84
	Ag (g/t)	5.71	5.20	6.45	4.82	3.09	3.58	4.93
	In Situ Au (kg)	96	217	239	332	236	27	1,147
	In Situ Ag (kg)	632	1,342	1,853	1,926	862	108	6,723
	Rec. Au (kg)	64	144	159	221	157	18	762
	Rec. Ag (kg)	42	89	123	128	57	07	447
HLROM	**Tonnage (kt)**	37	101	95	222	117	08	579
	Au (g/t)	0.51	0.48	0.48	0.47	0.48	0.52	0.48
	Ag (g/t)	3.97	3.74	4.51	2.72	1.58	0.96	3.02
	In Situ Au (kg)	19	49	46	105	56	04	278
	In Situ Ag (kg)	145	379	428	605	185	08	1,749
	Rec. Au (kg)	07	18	17	40	21	02	106
	Rec. Ag (kg)	10	25	28	40	12	00	116

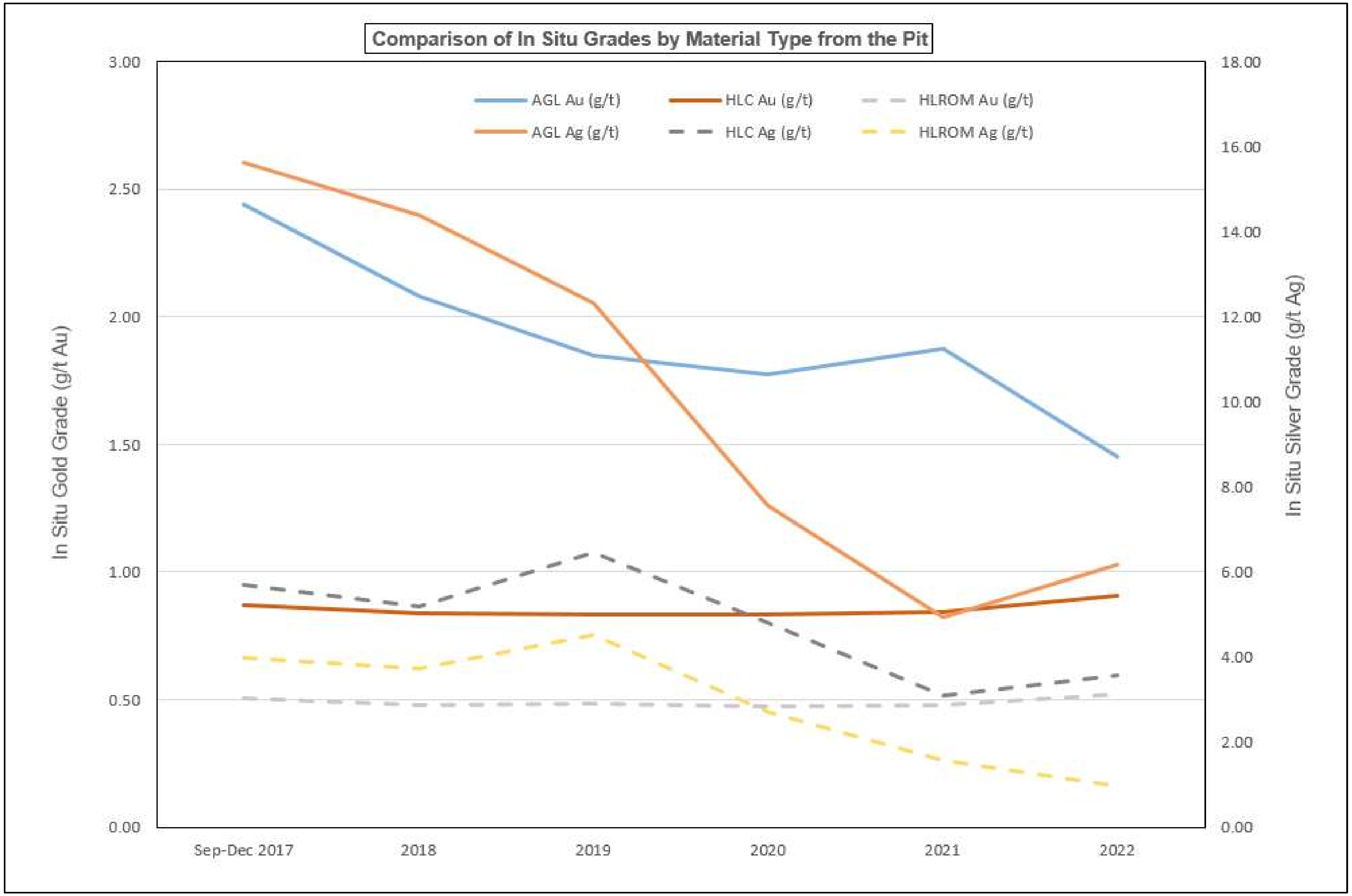


Figure 6-1- Comparison of in-situ grades by material type from the pit

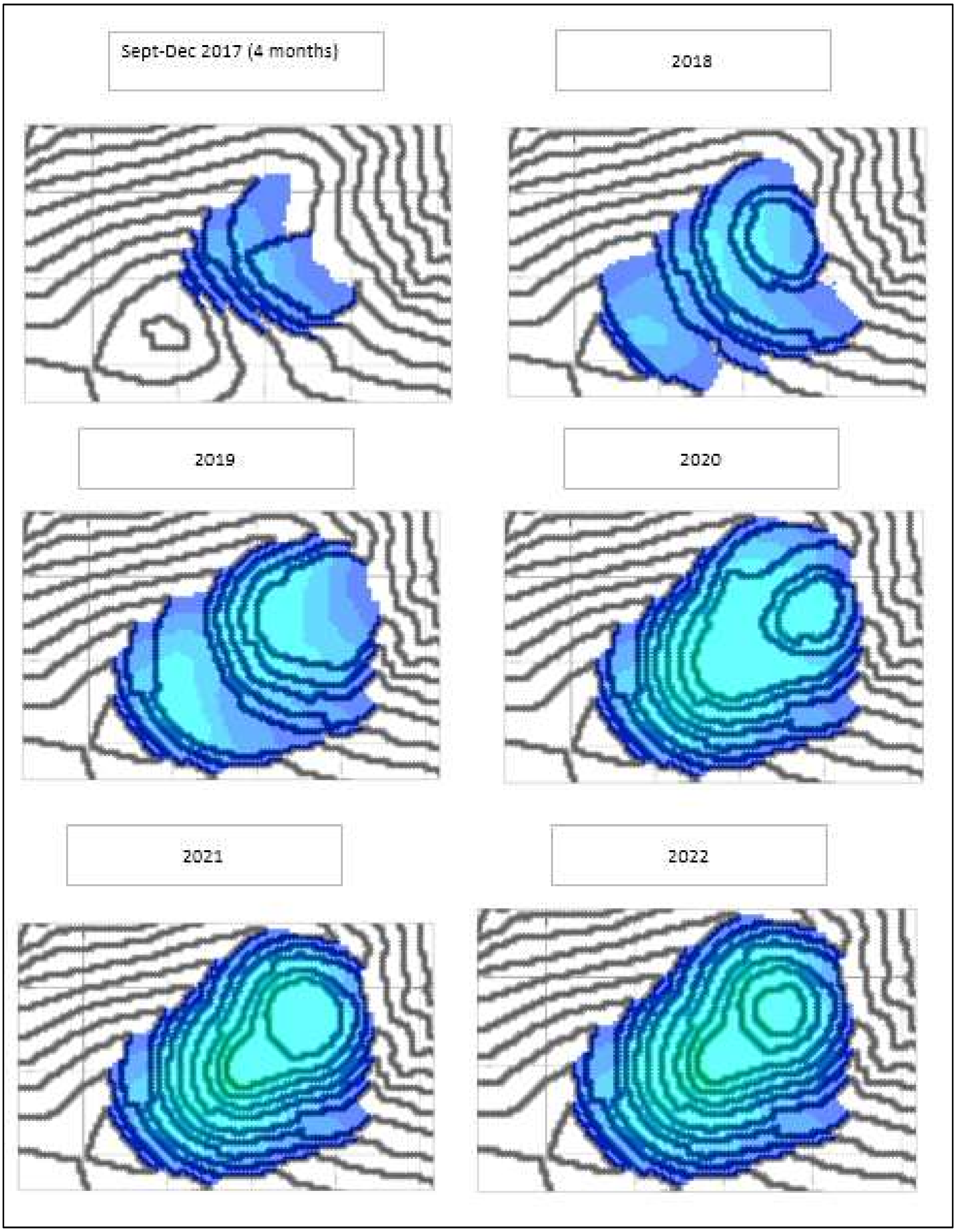


Figure 6-2- Life of Mine Plans

7 Conclusions and Recommendations

In order to refine the mining recovery and dilution, it’s recommended that the correlation between the geological model and actual production on a bench-by-bench basis will be investigated during the ore production.

It is concluded that the Ore Reserve for the Ugur open pit is 3.59Mt, with a contained metal content of 147,000 ounces of Au and 808,000 ounces of Ag. Following the initial pushback design options, a total of 3.56Mt, with a contained metal content of 4,552.4 kg of Au (146,360 oz) of Au, 25,057.0 of Ag (805600 oz) has been scheduled.

The selected pit was defined at a Price Factor of 64%, which was selected on the basis of maximising NPV. There is potential to expand the pit beyond the selected pit limit (Pit Shell 51) but this would involve a more information about the resource, which will be generated during the mining and grade control processes.

As regards the open pit, Datamine recommends that:

- Reconciliation studies are undertaken to improve the model for short term planning
- Infill drilling over several benches is used to optimise grade control
- Slopes are monitored to give advance warning of potential failure
- Detailed scheduling is undertaken to:
    - Refine the mining sequence
    - Avoid grade excursions where possible
    - Optimise the usage of the plants
    - Establish cycle times and haul truck requirements
    - Optimise the waste dumping strategy.

The may result in opportunities to improve the schedule as production information is gathered.

8 References

- "Ugur Oxidised Ore Body, Gedabek, Pit slope Stability Assessment" report issued by CQA International Limited, 27 July 2017.

- "Updated Mineral Reserve Statement for AIMC-Gedabek Mineral Deposit" report by CAE Mining, November 2014.

- Flotation testing of Gedabek Copper-Gold ore, Optimet Laboratories, South Australia, Optimet Report P0184, 8 February 2007, 19pp.

- An investigation into the recovery of gold from the Gedabek deposit, SGS Lakefield Research Limited, Project 11367-002 Report 1, 31 May 2007, 152pp.

- Scouting Leachbox and Flotation test work on Gedabek sulphide ore samples, Maelgwyn Mineral Services Africa, Report No 11/10, 2 December 2011, 27pp.

- Closure and Rehabilitation Management Plan, AMEC Earth and Environmental UK Ltd, December 2012, 62pp.

- Environmental Materiality Assessment, Daniel Limpitlaw on behalf of Datamine, 7 September 2014, 23pp.

- Gedabek Exploration Report, 2016, Gedabek Exploration Group, December 2016.

9 Compliance Statement

The information in the report that relates to exploration results, minerals resources and ore reserves is based on information compiled by Dr Stephen Westhead, who is a full-time employee of Anglo Asian Mining with the position of Director of Geology & Mining.

Stephen Westhead is a senior extractive industries professional with over 28 years of experience, who has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the ‘Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves’.

Stephen Westhead has sufficient experience, relevant to the style of mineralisation and type of deposit under consideration and to the activity that he is undertaking, to qualify as a “competent person” as defined by the AIM rules. Stephen Westhead has reviewed the reserves included in this report.

The information in this report that relates to Exploration Targets, Exploration Results, Mineral Resources or Ore Reserves is based on information compiled by Dr Stephen Westhead, a Competent Person who is a Member or Fellow of a ‘Recognised Professional Organisation’ (RPO) included in a list that is posted on the ASX website from time to time (Chartered Geologist and Fellow of the Geological Society and Professional Member of the Institute of Materials, Minerals and Mining), Fellow of the Society of Economic Geologists (FSEG) and Member of the Institute of Directors (MIoD). Dr Stephen Westhead is a full-time employee of Anglo Asian Mining plc (Azerbaijan International Mining Company).

Stephen Westhead consents to the inclusion in the report of the matters based on his information in the form and context in which it appears.

Dr Stephen J. Westhead
Competent Person
Director of Geology and Mining, Azerbaijan International Mining Company
(Anglo Asian Mining)

Appendix I- JORC Table 1 Checklist

The following table provides a summary of assessment and reporting criteria used at the Ugur deposit for the reporting of exploration results, Mineral Resources and Ore Reserves in accordance with the JORC Table 1 checklist in The Australasian Code for the Reporting of Exploration Results, Mineral Resources and Ore Reserves (The JORC Code, 2012 Edition).

JORC Code, 2012 Edition – Table 1 report: Ugur Deposit (Anglo Asian Mining plc)

Mineral Resource and Ore Reserve statement date: 14 August 2017

Section 1 Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.* • *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* • *Aspects of the determination of mineralisation that are Material to the Public Report.* • *In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.*	• Full core was split longitudinally 50% using a rock diamond saw and half-core samples were taken at typically 100 centimetre intervals or to rock contacts if present in the core run for both mineralisation and wall rock. The drill core was rotated prior to cutting to maximise structure to core axis of the cut core. • Reverse Circulation (RC) drill samples were collected via a cyclone system in calico sample bags following on site splitting using a standard riffle "Jones" splitter attached to the RC drill rig cyclone, and into plastic chip trays for every one metre interval. • To ensure representative sampling, diamond drill core was marked considering mineralisation and alteration intensity, after ensuring correct core run marking with regards recovery. • RC samples were routinely weighed to ensure sample is representative of the metre run. • Sampling of drill core and RC cutting were systematic and unbiased. • RC samples varies from 3kg to 6kg, the smaller weight sample related to losses where water was present. The average sample size was 4.7kg, and pulverised to produce a 50g sample for routine Atomic Absorption analysis and check fire assaying.

Criteria	JORC Code explanation	Commentary
		• Handheld XRF (model THERMO Niton XL3t) was used to assist with mineral identification during field mapping and core logging procedures.
Drilling techniques	• *Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	• Both diamond core drilling and reverse circulation (RC) drilling were completed. • Upper levels of core drilling from collar to an average depth of 35metres at PQ (85.0 mm) core single barrel wireline, stepping down to HQ (63.5mm) when necessary. • Diamond Core Drilling with HQ (63.5mm) core single tube barrel, stepping down to NQ (47.6mm) core barrel when necessary. • Diamond Core drilling with NQ (47.6mm) core single tube barrel • The proportions of PQ:HQ:NQ drilling were 17:60:23 percentage. • Oriented drill coring was not used. • Reverse Circulation drilling using 133 millimetre diameter face sampling drill bit. • Downhole surveying was carried out on 92% of core drillholes utilising Reflex EZ-TRAC equipment at a downhole interval of every 9 metres. • Drilling penetration speeds were also noted to assist with rock hardness indications.
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.* • *Measures taken to maximise sample recovery and ensure representative nature of the samples.* • *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	• Core recovery (TCR – total core recovery) was recorded at site, verified at the core logging facility and subsequently entered into the database. The average core recovery was 93%. Recovery measurements were poor in fractured and faulted rocks, however the contract drill crew maximised capability with use of drill muds and reduced core runs to ensure best recovery. In these zones where oxidised friable mineralisation was present, average recovery was

Criteria	JORC Code explanation	Commentary
		86%. • RC recovery was periodically checked by weighing the sample per metre for RC drill cuttings and compared to theoretical weight. • Geological information was passed to the drilling crews to make the drillers aware of areas of geological complexity, to maximise recovery of sample through the technical management of drilling (downward pressures, rotation speeds, water flushing, use of clays). • Zones of faulting and presence of water resulted in variable weights of RC sample, suggesting losses of fines. Historical drilling at adjacent deposits with similar situations tended to underestimate the in-situ gold grades. • There is no direct relationship between recovery and grade variation, however in core drilling, losses of fines is believed to result in lower gold grades due to washout of fines in fracture zones. This is likely to result in an underestimation of grade, which will be checked during production.
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.* • *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.* • *The total length and percentage of the relevant intersections logged.*	• Drill core was logged in detail for lithology, alteration, mineralisation, geological structure, and oxidation state by Anglo Asian Mining geologists, utilising logging codes and data sheets as supervised by the competent person. • RC cuttings were logged for lithology, alteration, mineralisation, and oxidation state. • Logging was considered sufficient to support Mineral Resource estimation, mining studies and metallurgical studies. • Rock Quality Designation (RQD) logs were produced for all core drilling for geotechnical purposes. Fracture intensity and fragmentation proportion analysis was also used for geotechnical information.

Criteria	JORC Code explanation	Commentary
		• Additionally, two "geotechnical" core drillholes were targeted and drilled to pass through mineralisation into wall rocks of the "planned" backwall to the open pit. This ensured geotechnical data collected related to open pit design work. • Point load testing and unconfined compressive strength (UCS) tests were conducted on all major rock (mineralised and wall rock) types. This data was utilised in establishing the open pit design parameters. • Independent geotechnical studies have been completed by the environmental engineering company, CQA International Limited (CQA), to assess rock mass strength and structural geological relationships for mine design parameters. • Logging was both quantitative and qualitative in nature. All core was photographed in the core boxes to show the core box number, core run markers and a scale, and all RC chip trays were photographed. • 100% of the core drilling was logged with a total of 6,354.75 metres of core and 100% of RC drilling with a total of 4,608.00 metres, that is included in the resource model.
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.* • *If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.* • *For all sample types, the nature, quality and appropriateness of the sample preparation technique.* • *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.* • *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.* • *Whether sample sizes are appropriate to the grain*	• Full core was split longitudinally 50% using a rock diamond saw and half-core samples were taken at typically 100 centimentre intervals or to rock contacts if present in the core run for both mineralisation and wall rock. The drill core was rotated prior to cutting to maximise structure to core axis of the cut core. • Half core was taken for sampling for assaying, and one half remains in the core box as reference material. • Reverse Circulation (RC) drill samples were collected in calico sample bags following on site splitting using a standard riffle "Jones" splitter, and into plastic chip trays for every one metre interval. • Where RC samples were wet, the total sample was collected for

Criteria	JORC Code explanation	Commentary
	size of the material being sampled.	drying at the laboratory, following which, sample splitting took place. Primary duplicates have also been retained as reference material. • RC field sampling equipment was regularly cleaned to reduce the chance of sample contamination by previous samples, on a metre basis by compressed air. • Both core and RC samples were prepared according best practice, with initial geological control of the half core or RC samples, followed by crushing and grinding at the laboratory sample preparation facility that is routinely managed for contamination and cleanliness control. Sampling practice is considered as appropriate for Mineral Resource Estimation. • Sample preparation at the laboratory is subject to the following procedure. ➢ After receiving samples at the laboratory from the geology department, all samples are cross referenced with the sample order list. ➢ All samples are dried in the oven at 105-110 degree centigrade temperature ➢ First stage sample crushing to -25mm size ➢ Second stage sample crushing to -10mm size. ➢ Third stage sample crushing to -2mm size. ➢ After crushing the samples are split and 200-250 gramme sample taken. ➢ A 75 micron sized prepared pulp is produced that is subsequently sent for assay preparation. • Quality control procedures were used for all sub-sampling preparation. This included geological control over the core cutting,

Criteria	JORC Code explanation	Commentary
		and sampling to ensure representativeness of the geological interval. • 127 Field duplicates of the reverse circulation (RC) samples were collected, representing 2.6% of the total RC metres drilled. • Sample sizes are considered appropriate to the grain size of the material and style of mineralisation being sampled, by maximising the sample size, hence the total absence of any BQ drill core.
Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.* • *For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.* • *Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	• Laboratory procedures and assaying and analysis methods are industry standard. They are well documented and supervised by a dedicated laboratory team. The techniques of Atomic Absorption and Fire Assay were utilised, and as such both partial and total techniques were employed. • Handheld XRF (model THERMO Niton XL3t) was used to assist with mineral identification during field mapping and core logging procedures. • Commencement of drilling was 23/09/2016 and completion was 15/07/2017 being 295 days, during which period 4,928 RC samples and 6,447 core drill samples (a total of 11,375 samples) were taken. A total of 1,740 QA/QC samples were measured, equivalent to 15.3%. • QA/QC procedures included the use of field duplicates of RC samples, blanks, certified standards or certified reference material (CRMs) from OREAS (Ore Research & Exploration Pty Ltd Assay Standards, Australia), in addition to the laboratory control that comprised pulp duplicates, check samples, and replicate samples. This QA/QC system allowed for the monitoring of precision and accuracy of assaying for the Ugur deposit. • The quality of the QA/QC is considered adequate for resource and reserve estimation purposes.

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
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Pulp duplicates analysis showed the largest error in waste or very high grade samples (see below), Note: with silver classified by gold grade:

<p>Pulp Duplicates for gold and silver</p>
<table>
<tr><th>Gold Grade Range g/t</th><th>Au (1) Average g/t Au</th><th>Au (2) Average g/t Au</th><th>Ag (1) Average g/t Ag</th><th>Ag (2) Average g/t Ag</th></tr>
<tr><td>Average</td><td>1.46</td><td>1.48</td><td>1.86</td><td>1.77</td></tr>
<tr><td>0.0 to ≤0.3</td><td>0.10</td><td>0.21</td><td>1.86</td><td>1.77</td></tr>
<tr><td>0.3 to ≤1.0</td><td>0.64</td><td>0.69</td><td>4.51</td><td>4.33</td></tr>
<tr><td>1.0 to ≤2.0</td><td>1.44</td><td>1.44</td><td>8.10</td><td>7.93</td></tr>
<tr><td>2.0 to ≤5.0</td><td>2.82</td><td>2.74</td><td>13.62</td><td>13.52</td></tr>
<tr><td>5.0 to ≤20.0</td><td>7.27</td><td>7.23</td><td>32.09</td><td>29.91</td></tr>
</table>

External check assay was carried out by ALS Minerals (OMAC) based in Ireland. The following analytical work was conducted for each sample:

Sample login / pulverize split to 85% < 75 micron / pulverizing QC test / Received sample weight
Ore grade for Gold 30g AA finish
35 Element Aqua Regia ICP-AES analysis (to include the following elements: Ag, Al, As, B, Ba, Be, Bi, Ca, Cd, Co, Cr, Fe, Ga, Hg, K, La, Mg, Mn, Mo, Ni, P, Pb, S, Sb, Sc, Sr, Th, Ti, Tl, U, V, W, Zn.

Comparison of average gold grades between the on-site laboratory and ALS shows a general bias towards the on-site laboratory under-estimating grade with the exception of high grade (as presented

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<tr><td></td><td></td><td>below):
<table>
<tr><th>Gold Grade Range</th><th>AAZ Average g/t Au</th><th>ALS Average g/t Au</th></tr>
<tr><td>Average</td><td>0.83</td><td>0.90</td></tr>
<tr><td>0.0 to≤0.3</td><td>0.08</td><td>0.08</td></tr>
<tr><td>0.3 to ≤1.0</td><td>0.60</td><td>0.70</td></tr>
<tr><td>1.0 to ≤2.0</td><td>1.31</td><td>1.36</td></tr>
<tr><td>2.0 to ≤5.0</td><td>2.97</td><td>3.76</td></tr>
<tr><td>5.0 to ≤20.0</td><td>12.21</td><td>11.16</td></tr>
</table>
Based on QA/QC work, and instances of poor repeatability, it is recommended to carry out thorough QA/QC of all samples during the extraction process and assess laboratory capacities.</td></tr>
<tr><td><i>Verification of sampling and assaying</i></td><td><i>The verification of significant intersections by either independent or alternative company personnel.</i>
<i>The use of twinned holes.</i>
<i>Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.</i>
<i>Discuss any adjustment to assay data.</i></td><td>Significant intersections were verified by a number of company personnel within the management structure of the Exploration Department. Intersections were defined by the exploration geologists, and subsequently verified by the Exploration Manager. Further, independent verification was carried out as part of the due diligence for resource estimation. Assay intersection were cross validated with drill core visual intersections.
An initial programme of RC drilling was followed up by a core drilling programme where two drillholes were twinned and validated the presence of mineralisation. Reverse circulation drilling as compared with the core showed a positive grade bias of up to 10%. It is suspected that losses may have occurred during the core drilling process especially in very strongly oxidised mineralised zones due to drilling fluid interaction.</td></tr>
</table>

Criteria	JORC Code explanation	Commentary
		• Data entry is supervised by a data manager, and verification and checking procedures are in place. The format of the data is appropriate for direct import into “Datamine”® software. All data is stored in electronic databases within the geology department and backed up to the secure company electronic server that has limited and restricted access. Four main files are created relating to “collar”, “survey”, “assay” and “geology”. Laboratory data is loaded electronically by the laboratory department and validated by the geology department. Any outlier assays are re-assayed. • Independent validation of the database was made as part of the resource model generation process, where all data was checked for errors, missing data, misspelling, interval validation, and management of zero versus no data entries. • All databases were considered accurate for the Mineral Resource Estimate. • No adjustments were made to the assay data.
Location of data points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.* • *Specification of the grid system used.* • *Quality and adequacy of topographic control.*	• The exploration area was initially surveyed by high resolution drone survey. Five topographic base stations were installed and accurately surveyed using high precision GPS, that was subsequently tied into the local mine grid using ground based total station surveying (LEICA TS02) equipment. All trench, drill holes collars were then surveyed using total station survey equipment. • Downhole surveying was carried out on 92% of core drillholes utilising Reflex EZ-TRAC equipment at a downhole interval of every 9 metres. • The grid system used is Universal Transverse Mercator (UTM)84WGS zone 38T (Azerbaijan) • The adequacy of topographic control is adequate for the purposes of

Criteria	JORC Code explanation	Commentary
		resource and reserve modelling (having been validated by both aerial and ground based survey techniques), with a contour interval of 2m metres.
Data spacing and distribution	- *Data spacing for reporting of Exploration Results.* - *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.* - *Whether sample compositing has been applied.*	- Drill hole spacing carried out was from 20 metres over the main mineralised zone to 45 metres on the periphery of the resource. - The data spacing and distribution (20 x 20 metre grid) over the mineralised zones is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied. The depth and spacing is considered appropriate for defining geological and grade continuity as required for a JORC Mineral Resource estimate. - No physical sample compositing has been applied for assay purposes, however for some metallurgical tests, 4 to 5 metre composites were applied.
Orientation of data in relation to geological structure	- *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.* - *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	- Detailed surface mapping and subsequent drilling has provided the characteristics of the deposit. The orientation of the drill grid to NNE was designed to maximise the geological interpretation in terms of true contact orientations. - The Ugur gold deposit is considered as a high sulphidation gold deposit located in rocks ranging from Bajocian (Mid-Jurassic) to Tithonian (Upper-Jurassic) in age. The gold mineralisation is hosted by Upper Bajocian age sub-volcanic rocks, that comprise rhyo-dacitic breccias. These rocks have been intruded into a sub-volcanic sequence that was subsequently subjected to strong hydrothermal alteration. - The Ugur primary mineralisation is hosted in acidic volcanic rocks, that consists of haematite-barite-quartz-kaolin veins-veinlets and

Criteria	JORC Code explanation	Commentary
		breccia, pyritic stock-stockwork and quartz-sulphide veins. The central surface expression of the mineralisation exhibit accumulations of hydrous ferric oxides cementing breccias of silica with vein-veinlets barite-haematite mineralisation. • The deposit was emplaced at the intersection of NW, NE, N and E trending structural systems regionally controlled by a first order NW transcurrent fault structure. The fault dips between 70º to 80° to the north-west. The faults of the central zone control the hydrothermal metasomatic alteration and gold mineralisation. • Given the geological understanding and the application of the drilling grid orientation, grid spacing and vertical drilling, no orientation based sample bias has been identified in the data which resulted in unbiased sampling of structures considering the deposit type.
Sample security	• *The measures taken to ensure sample security.*	• Regarding drill core: at the drilling site which was supervised by a geologist, the drill core is placed into wooden core boxes that are sized specifically for the drill core diameter. Once the box is full, a wooden lid is fixed to the box to ensure not spillage. Core box number, drill hole number and from/to metres are written on both the box and the lid. The core is then transported to the core storage area and logging facility, where it is received and logged into a data sheet. Core logging, cutting, and sampling takes place at the secure core management area. The core samples are bagged with labels both in the bag and on the bag, and data recorded on a sample sheet. The samples are transferred to the laboratory where they are registered as received, for laboratory sample preparation works and assaying. Hence, a chain of custody procedure has been followed from core collection to assaying and storage of reference material. • Reverse Circulation samples are bagged at the drill site and sample

Criteria	JORC Code explanation	Commentary
		numbers recorded on the bags. Batches of 10 metre samples are boxed for transport to the logging facility where the geological study and sample preparation for transfer to the laboratory take place. • All samples received at the core facility are logged in and registered with the completion of an "act". The act is signed by the drilling team supervisor and core facility supervisor (responsible person). All core is photographed, subjected to geotechnical logging, geological logging, samples interval determinations, bulk density, core cutting, and sample preparation (size 3-5 centimetre). • Daily, all samples are weighed and Laboratory order prepared which is signed by the core facility supervisor prior to release to the laboratory. On receipt at the laboratory, the responsible person countersigns the order. • After assaying all reject duplicate samples are received from laboratory to core facility (recorded on a signed act). All reject samples are placed into boxes referencing the sample identities and stored in the core facility. • In the event of external assaying, Anglo Asian Mining utilised ALS-OMAC in Ireland. Samples selected for external assay are recorded on a data sheet, and sealed in appropriate boxes for shipping by air freight. Communication between the geological department of the Company and ALS monitor the shipment, customs clearance, and receipt of samples. Results are sent electronically by ALS and loaded to the Company database for study.
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	• Reviews on sampling and assaying techniques were conducted for all data internally and externally as part of the resource and reserve estimation validation procedure. No concerns were raised as to the procedures or the data results. All procedures were considered

Criteria	JORC Code explanation	Commentary
		industry standard and well conducted. QA/QC tolerance concerns of some of batches of assaying has been raised.

Section 2 Reporting of Exploration Results

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.* • *The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	• The project is located within a current contract area that is managed under a "PSA" production sharing agreement. • The PSA grants the Company a number of periods to exploit defined licence areas, known as Contract Areas, agreed on the initial signing with the Azerbaijan Ministry of Ecology and Natural Resources ('MENR'). The exploration period allowed for the early exploration of the Contract Areas to assess prospectivity can be extended. • A 'development and production period' commences on the date that the Company issues a notice of discovery, which runs for 15 years with two extensions of five years each at the option of the Company. Full management control of mining in the Contract Areas rests with Anglo Asian Mining. • Under the PSA, Anglo Asian is not subject to currency exchange restrictions and all imports and exports are free of tax or other restriction. In addition, MENR is to use its best endeavours to make available all necessary land, its own facilities and equipment and to assist with infrastructure. • The deposit is not located in any national park. • At the time of reporting no known impediments to obtaining a licence to operate in the area exist and the contract (licence) area agreement is in good standing.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	• The "Ugur" deposit, renamed the "Reza" deposit (named for company exploration identification proposes) is located within the locally defined Ugur area. The Reza gold deposit was discovered in 2016 by the Gedabek Exploration Group of Anglo Asian Mining who worked on the regional area of Ugur from 2014 year. • Historical work on the area included regional mapping and large-scale

Criteria	JORC Code explanation	Commentary
		regional geophysical programmes (magnetic and gravity) by Soviet geologists. • Prior to the drill programme for resource estimate, Anglo Asian Mining carried out the following work: ➢ Stream sediment sampling; 7 samples (2014), 16 samples (2016), ➢ Stream Grab sampling; 37 samples (2016) ➢ Geological mapping; 90 000m2 1:10 000 (2014-2015), 35 000m2 1:1 000 (2016) ➢ Outcrop sampling; 1,460 samples (2016) ➢ Trenching & shallow pits; 610 samples (2016)
Geology	• *Deposit type, geological setting and style of mineralisation.*	• The Ugur gold deposit is located in Gedabek Ore District of the Lesser Caucasus in NW of Azerbaijan, 48 kilometres East of the city of Ganja, and 4.7 kilometres north west of Gedabek open-pit gold copper mine. • The exploration "centre" of the project is the outcrop, independently located on Google Earth at Latitude 40°37'13.10"N and Longitude 45°46'15.34"E. The known gold mineralisation has an estimated north-south strike length of 400 m and a total area of approximately 20 hectares or 0.2 km². The deposit was found based on gold-silver assays of surface outcrop rock chip samples over an area of 2.5 kilometres North-South by 2 kilometres East-West, with the Ugur gold deposit located on the central part. • Secondary quartzites were formed under the influence of Atabek-Slavyanka plagiogranite intrusion with exposures observed to the north from the gold mineralisation area. The area in tectonic attitude is confined to Gyzyldjadag fault of North-eastern sub-latitudinal strike 080° with a vertical dip. • Rocks in the alteration zone area crumpled, argillic altered, brecciated, with strong limonite and haematite alteration, where crystalline haematite is observed. Intensive barite and barite-

Criteria	JORC Code explanation	Commentary
		hematite vein and veinlets, also gossan zones are present in outcrop. The main mineralisation zones have been sampled in three trenches with a total length of 270 metres (trenches #1, #2 and #3) and received positive results for gold and silver. About 550 samples from main outcrops #1 and #2 were taken. • The main mineralised zone comprises secondary quartzites with vein-veinlets barite-haematite mineralisation over which remain accumulations of hydrous ferric oxides cementing breccias of quartz and quartzites. Erosion surfaces exhibit “reddish mass” being an oxidation product of stock and stockwork haematite ores. • A Lithological-structural map of the Gedabek Ore District is presented in Mineral Resource Report. • The Ugur gold deposit is considered as a high sulphidation gold deposit located in rocks ranging from Bajocian (Mid-Jurassic) to Tithonian (Up-Jurassic) in age. The gold mineralisation is hosted by an Upper Bajocian age sub-volcanic rocks, that comprise rhyo-dacitic breccias. These rocks have been intruded into a sub-volcanic sequence that was subsequently subjected to strong hydrothermal alteration. • The Ugur primary mineralisation is hosted in acidic volcanic rocks, that consists of haematite-barite-quartz-kaolin veins-veinlets and breccia, pyritic stock-stockwork and quartz-sulphide veins. The central surface expression of the mineralisation exhibit accumulations of hydrous ferric oxides cementing breccias of silica with vein-veinlets barite-haematite mineralisation. • The deposit was emplaced at the intersection of NW, NE, N and E trending structural systems regionally controlled by a first order NW transcurrent fault structure. The fault dips between 70º to 80° to the north-west. The faults of the central zone control the hydrothermal metasomatic alteration and gold mineralisation.

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr>
<td>Drill hole Information</td>
<td>

A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:

easting and northing of the drill hole collar
elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar
dip and azimuth of the hole
down hole length and interception depth
hole length.

If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.

</td>
<td>

A summary of the type and metres of drilling completed is shown below:

<table>
<tr><th>Type of drill-hole</th><th>Type</th><th>Start date</th><th>Finish date</th><th>Number of drill holes</th><th>Length (metres)</th></tr>
<tr><td>Reverse circulation</td><td>Reverse circulation</td><td>23-Sep-16</td><td>14-Nov-16</td><td>56</td><td>1,842</td></tr>
<tr><td>Core</td><td>Diamond</td><td>04-Oct-16</td><td>25-Jun-17</td><td>50</td><td>6,355</td></tr>
<tr><td>Geotechnical</td><td>Diamond</td><td>16-Apr-17</td><td>27-Apr-17</td><td>2</td><td>164</td></tr>
<tr><td>Reverse circulation</td><td>Reverse circulation</td><td>19-Mar-17</td><td>09-Jul-17</td><td>33</td><td>2,766</td></tr>
<tr><td>TOTAL DRILLING</td><td></td><td></td><td></td><td>141</td><td>11,127</td></tr>
</table>

Coordinates and RL of the drill collars and depth to end of drill hole are presented below:

DD drillholes are diamond core drillholes
RC drillhole are reverse circulation drillholes

<table>
<tr><th>hole_id</th><th>x</th><th>y</th><th>z</th><th>max_ depth</th><th>hole_type</th></tr>
<tr><td>GTDD01</td><td>565173.423</td><td>4496827.437</td><td>1,907.03</td><td>76.5</td><td>DD</td></tr>
<tr><td>GTDD02</td><td>565238.685</td><td>4496871.059</td><td>1,886.89</td><td>87.25</td><td>DD</td></tr>
<tr><td>RGRC01</td><td>565226.845</td><td>4496897.396</td><td>1,885.51</td><td>84</td><td>RC</td></tr>
<tr><td>RGRC02</td><td>565188.902</td><td>4496909.231</td><td>1,896.80</td><td>82</td><td>RC</td></tr>
<tr><td>RGRC03</td><td>565199.867</td><td>4496885.521</td><td>1,895.03</td><td>120</td><td>RC</td></tr>
<tr><td>RGRC04</td><td>565175.67</td><td>4496873.255</td><td>1,902.65</td><td>102</td><td>RC</td></tr>
<tr><td>RGRC05</td><td>565212.099</td><td>4496857.204</td><td>1,895.82</td><td>111</td><td>RC</td></tr>
<tr><td>RGRC06</td><td>565187.095</td><td>4496847.644</td><td>1,902.83</td><td>90</td><td>RC</td></tr>
<tr><td>RGRC07</td><td>565201.521</td><td>4496946.672</td><td>1,888.80</td><td>113</td><td>RC</td></tr>
<tr><td>RGRC08</td><td>565227.273</td><td>4496960.209</td><td>1,879.43</td><td>80</td><td>RC</td></tr>
<tr><td>RGRC09</td><td>565240.567</td><td>4496934.862</td><td>1,874.38</td><td>81</td><td>RC</td></tr>
</table>
</td>
</tr>
</table>

Criteria	JORC Code explanation	Commentary					
		RGRC10	565264.685	4496947.009	1,867.17	54	RC
		RGRC11	565278.053	4496920.284	1,866.33	68	RC
		RGRC12	565254.444	4496916.843	1,872.04	69	RC
		RGRC13	565264.942	4496883.101	1,878.37	85	RC
		RGRC14	565233.946	4496857.616	1,890.95	90	RC
		RGRC15	565250.738	4496845.542	1,890.14	90	RC
		RGRC16	565176.519	4496936.975	1,899.80	96	RC
		RGRC17	565218.676	4496980.836	1,883.64	75	RC
		RGRC18	565151.143	4496921.426	1,910.78	104	RC
		RGRC19	565161.878	4496899.425	1,907.12	87	RC
		RGRC20	565254.44	4496973.287	1,870.34	75	RC
		RGRC21	565208.569	4496903.57	1,890.33	91	RC
		RGRC22	565218.709	4496942.994	1,884.00	76	RC
		RGRC23	565195.134	4496866.565	1,898.72	79	RC
		RGRC24	565182.973	4496953.864	1,896.01	120	RC
		RGRC25	565196.049	4496992.538	1,887.24	78	RC
		RGRC26	565278.344	4496984.036	1,862.07	61	RC
		RGRC27	565239.013	4496998.68	1,878.43	80	RC
		RGRC28	565310.589	4496952.217	1,854.41	103	RC
		RGRC29	565302.606	4496935.639	1,859.11	90	RC
		RGRC30	565297.765	4496915.477	1,863.32	48	RC
		RGRC31	565284.229	4496897.028	1,869.72	75	RC
		RGRC32	565282.66	4496876	1,875.81	80	RC
		RGRC33	565313.5	4497022.3	1,849.60	103	RC
		RGRC34	565165	4496956.9	1,900.40	120	RC
		RGRC35	565179.9	4497001.7	1,884.40	100	RC
		RGRC36	565140.1	4496950.3	1,899.00	100	RC
		RGRC37	565157.4	4496879.7	1,908.00	106	RC
		UGDD01	565277.6	4496960.5	1,863.00	285.5	DD
		UGDD02	565214.3	4496923.1	1,887.90	401.3	DD
		UGDD03	565293.8	4496996.2	1,857.20	138.5	DD

Criteria	JORC Code explanation	Commentary					
		UGDD04	565260.1	4496900.9	1,875.10	123.5	DD
		UGDD05	565241.1	4496828.3	1,895.20	139	DD
		UGDD06	565220.8	4496877.3	1,890.40	133.35	DD
		UGDD07	565228.2	4496919.9	1,883.00	130	DD
		UGDD08	565242.7	4496955.5	1,874.00	124	DD
		UGDD09	565196.9	4496931.4	1,891.90	126.2	DD
		UGDD10	565179.6	4496888.9	1,901.70	122.15	DD
		UGDD11	565729	4496925.5	1,820.70	151.5	DD
		UGDD12	565166.9	4496852.5	1,908.00	125	DD
		UGDD13	565611	4496922.5	1,827.40	151	DD
		UGDD14	565163.6	4496937	1,905.20	132	DD
		UGDD15	565771.7	4497040	1,803.80	250	DD
		UGDD16	565147.4	4496903.4	1,912.40	134	DD
		UGDD17	565130.3	4496869.2	1,919.70	110	DD
		UGDD18	565220.2	4497005.4	1,883.00	125.4	DD
		UGDD19	565253.1	4496998.2	1,873.30	117	DD
		UGDD20	565249.9	4496873.2	1,884.10	125	DD
		UGDD21	565207.6	4496970.2	1,885.90	104.5	DD
		UGDD22	565269.9	4497031	1,867.20	136	DD
		UGDD23	565299.8	4496844.4	1,880.50	117	DD
		UGDD24	565236.6	4497043.7	1,869.20	134	DD
		UGDD25	565305.5	4496888.3	1,870.80	120	DD
		UGDD26	565324.4	4496926.9	1,854.10	135	DD
		UGDD27	565284.6	4496933	1,863.70	124	DD
		UGDD28	565311	4496997.8	1,849.70	119.3	DD
		UGDD29	565313.6	4497059	1,846.70	130	DD
		UGDD30	565297.8	4496975.7	1,854.60	126	DD
		UGDD31	565210	4496841.9	1,898.40	109	DD
		UGDD32	565171.9	4496986.5	1,890.20	113	DD
		UGDD33	565335.4	4496965	1,842.90	122	DD
		UGDD34	565119.6	4496957.4	1,898.90	133	DD

Criteria	JORC Code explanation	Commentary					
		UGDD35	565109.4	4496919.1	1,916.20	130.5	DD
		UGDD36	565351.1	4497001.9	1,833.40	122.5	DD
		UGDD37	565115.5	4496831.8	1,930.20	103.5	DD
		UGDD38	565197.6	4497052.6	1,866.50	122	DD
		UGDD39	565094.7	4496884.3	1,925.80	126.5	DD
		UGDD40	565075.5	4496842.1	1,932.60	150	DD
		UGDD41	565087	4496754.7	1,913.90	121.5	DD
		UGDD42	565115.4	4496878.5	1,924.90	80	DD
		UGDD43	565130.9	4496909.9	1,916.40	61.75	DD
		UGDD44	565188.4	4496977.7	1,890.00	61.8	DD
		UGDD45	565194.1	4497020.6	1,878.80	71	DD
		UGDD46	565228.9	4497023.8	1,878.00	70	DD
		UGDD47	565262.6	4497016.5	1,870.40	71.5	DD
		UGDD48	565298.4	4497007.9	1,856.90	67	DD
		UGDD49	565167.4	4496915.1	1,905.60	61	DD
		UGDD50	565140.6	4496999.8	1,882.60	67	DD
		UGRC01	565169.7	4496819.6	1,908.80	33	RC
		UGRC02	565146.5	4496867.7	1,913.20	34	RC
		UGRC03	565305.8	4496888.9	1,871.10	34	RC
		UGRC04	565275.6	4496958.6	1,863.30	27	RC
		UGRC05	565309.2	4496928.2	1,858.00	13	RC
		UGRC06	565343	4496922.9	1,850.30	32	RC
		UGRC07	565320.4	4496969.7	1,847.30	34	RC
		UGRC08	565347.6	4497022.1	1,833.50	31	RC
		UGRC09	565336.7	4497000.4	1,837.60	22	RC
		UGRC10	565266.6	4496930	1,867.20	34	RC
		UGRC11	565290.5	4496997.6	1,857.70	34	RC
		UGRC12	565267.4	4497018.4	1,869.10	34	RC
		UGRC13	565234.9	4496976.4	1,877.70	34	RC
		UGRC14	565212.8	4496921.8	1,888.00	34	RC
		UGRC15	565222.6	4497010.3	1,882.50	34	RC

Criteria	JORC Code explanation	Commentary					
		UGRC16	565184.4	4496970.7	1,892.90	34	RC
		UGRC17	565204.8	4496869.1	1,896.00	34	RC
		UGRC18	565244.7	4496887.2	1,882.10	34	RC
		UGRC19	565090.1	4496843.9	1,931.90	34	RC
		UGRC20	565163.8	4496916.4	1,905.70	30	RC
		UGRC21	565240.9	4497048	1,867.10	34	RC
		UGRC22	565284.2	4497058.9	1,854.60	34	RC
		UGRC23	565295.5	4496849	1,880.00	34	RC
		UGRC24	565106.9	4496906.4	1,921.20	34	RC
		UGRC25	565140.8	4496976.5	1,891.60	25	RC
		UGRC25A	565144.7	4496977.5	1,891.60	34	RC
		UGRC26	565173.9	4497025.1	1,875.20	31	RC
		UGRC27	565229.9	4496839.6	1,895.30	34	RC
		UGRC28	565355	4496609.9	1,921.10	34	RC
		UGRC29	565303.1	4496611.9	1,915.40	34	RC
		UGRC30	565318.5	4496657.4	1,915.50	34	RC
		UGRC31	565190.3	4496748.9	1,906.10	34	RC
		UGRC32	565209.5	4496795.3	1,904.00	34	RC
		UGRC33	565147.3	4496776.7	1,914.30	34	RC
		UGRC34	565126.2	4496745	1,909.80	34	RC
		UGRC35	565057	4496754	1,915.30	34	RC
		UGRC36	565104.5	4496793.5	1,923.80	34	RC
		UGRC37	565058.9	4496793.8	1,923.90	34	RC
		UGRC38	565027.4	4496748.3	1,918.40	34	RC
		UGRC39	564988.8	4496778.9	1,921.70	34	RC
		UGRC40	565022.2	4496827.5	1,922.50	34	RC
		UGRC41	565045.5	4496870.5	1,922.00	34	RC
		UGRC42	565057.2	4496912.7	1,913.60	34	RC
		UGRC43	564979	4496851.6	1,912.40	34	RC
		UGRC44	564948.3	4496808.5	1,919.60	34	RC
		UGRC45	564909.6	4496841.8	1,912.60	34	RC

Criteria	JORC Code explanation	Commentary

UGRC46	564883.7	4496797.6	1,925.90	34	RC
UGRC47	564921.3	4496775.2	1,926.50	34	RC
UGRC48	564852.4	4496758.8	1,929.80	34	RC
UGRC49	564810.6	4496782.7	1,932.90	34	RC
UGRC50	564840.8	4496824.2	1,921.10	34	RC
UGRC51	564765.9	4496810.9	1,933.80	34	RC
UGRC52	564743.3	4496771.6	1,942.90	34	RC
UGRC53	565702.2	4497046.2	1,785.40	34	RC
UGRC54	565794.7	4497051	1,803.50	34	RC
UGRC55	565770.8	4497019.6	1,807.90	34	RC

- Regarding dip and azimuth data of the core drill holes, all drill holes were vertical. The largest variation of all drill holes was 3.2 degrees off the vertical confirmed by downhole surveying.
- Intercept information has been previously provided in regulatory announcements (see section "substantive exploration data" below).
- The diameter of the drill core for each drill hole is presented below:

hole_id	from	to	length	diameter
UGDD01	0.00	127.00	127.00	HQ
UGDD01	127.00	285.50	158.50	NQ
UGDD02	0.00	72.50	72.50	PQ
UGDD02	72.50	184.00	111.50	HQ
UGDD02	184.00	401.30	217.30	NQ
UGDD03	0.00	42.00	42.00	PQ
UGDD03	42.00	138.50	96.50	HQ
UGDD04	0.00	40.00	40.00	PQ
UGDD04	40.00	123.50	83.50	HQ
UGDD05	0.00	42.00	42.00	PQ
UGDD05	42.00	139.00	97.00	HQ
UGDD06	0.00	43.00	43.00	PQ

Criteria	JORC Code explanation	Commentary				
		UGDD06	43.00	133.35	90.35	HQ
		UGDD07	0.00	60.15	60.15	PQ
		UGDD07	60.15	130.00	69.85	HQ
		UGDD08	0.00	70.00	70.00	PQ
		UGDD08	70.00	124.00	54.00	HQ
		UGDD09	0.00	49.00	49.00	PQ
		UGDD09	49.00	126.20	77.20	HQ
		UGDD10	0.00	63.00	63.00	PQ
		UGDD10	63.00	122.15	59.15	HQ
		UGDD11	0.00	65.00	65.00	PQ
		UGDD11	65.00	151.50	86.50	HQ
		UGDD12	0.00	57.70	57.70	PQ
		UGDD12	0.00	125.00	125.00	HQ
		UGDD13	0.00	58.00	58.00	PQ
		UGDD13	58.00	151.00	93.00	HQ
		UGDD14	0.00	40.00	40.00	PQ
		UGDD14	40.00	132.00	92.00	HQ
		UGDD15	0.00	60.00	60.00	PQ
		UGDD15	60.00	250.00	190.00	HQ
		UGDD16	0.00	48.00	48.00	PQ
		UGDD16	48.00	134.00	86.00	HQ
		UGDD17	0.00	59.50	59.50	PQ
		UGDD17	59.50	110.00	50.50	HQ
		UGDD18	0.00	35.50	35.50	PQ
		UGDD18	35.50	125.40	89.90	HQ
		UGDD19	0.00	33.00	33.00	PQ
		UGDD19	33.00	117.00	84.00	HQ
		UGDD20	0.00	41.50	41.50	PQ
		UGDD20	41.50	125.00	83.50	HQ
		UGDD21	0.00	30.00	30.00	PQ
		UGDD21	30.00	104.50	74.50	HQ

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UGDD22	0.00	37.00	37.00	PQ
UGDD22	37.00	136.00	99.00	HQ
UGDD23	0.00	34.00	34.00	PQ
UGDD23	34.00	117.00	83.00	HQ
UGDD24	0.00	37.00	37.00	PQ
UGDD24	37.00	134.00	97.00	HQ
UGDD25	0.00	16.00	16.00	PQ
UGDD25	16.00	120.00	104.00	HQ
UGDD26	0.00	22.00	22.00	PQ
UGDD26	22.00	135.00	113.00	HQ
UGDD27	0.00	37.00	37.00	PQ
UGDD27	37.00	124.00	87.00	HQ
UGDD28	0.00	24.00	24.00	PQ
UGDD28	24.00	119.30	95.30	HQ
UGDD29	0.00	11.00	11.00	PQ
UGDD29	11.00	130.00	119.00	HQ
UGDD30	0.00	34.00	34.00	PQ
UGDD30	34.00	126.00	92.00	HQ
UGDD31	0.00	14.00	14.00	PQ
UGDD31	14.00	109.00	95.00	HQ
UGDD32	0.00	7.00	7.00	PQ
UGDD32	7.00	113.00	106.00	HQ
UGDD33	0.00	20.50	20.50	PQ
UGDD33	20.50	122.00	101.50	HQ
UGDD34	0.00	20.60	20.60	PQ
UGDD34	20.60	122.00	101.40	HQ
UGDD35	0.00	26.50	26.50	PQ
UGDD35	26.50	130.50	104.00	HQ
UGDD36	0.00	31.00	31.00	PQ
UGDD36	31.00	122.50	91.50	HQ
UGDD37	0.00	27.00	27.00	PQ

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<tr><td>UGDD37</td><td>79.00</td><td>103.50</td><td>24.50</td><td>NQ</td></tr>
<tr><td>UGDD38</td><td>0.00</td><td>9.00</td><td>9.00</td><td>PQ</td></tr>
<tr><td>UGDD38</td><td>9.00</td><td>122.00</td><td>113.00</td><td>HQ</td></tr>
<tr><td>UGDD39</td><td>0.00</td><td>45.00</td><td>45.00</td><td>PQ</td></tr>
<tr><td>UGDD39</td><td>45.00</td><td>126.50</td><td>81.50</td><td>HQ</td></tr>
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<tr><td>UGDD40</td><td>22.00</td><td>150.00</td><td>128.00</td><td>HQ</td></tr>
<tr><td>UGDD41</td><td>0.00</td><td>21.00</td><td>21.00</td><td>PQ</td></tr>
<tr><td>UGDD41</td><td>21.00</td><td>121.50</td><td>100.50</td><td>HQ</td></tr>
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<tr><td>UGDD47</td><td>0.00</td><td>24.00</td><td>24.00</td><td>PQ</td></tr>
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</td></tr>
<tr><td><i>Data aggregation</i></td><td>• <i>In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.</i></td><td>• Drilling results have been reported using intersection intervals based on a gold grade above 0.3 gramme per tonne, and internal waste greater or equal to 1 metre thickness. Grade of both gold and silver</td></tr>
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Criteria	JORC Code explanation	Commentary
methods	• *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* • *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	within the intersections have been state. The results are presented to 2 decimal places. • No data aggregation and no sample compositing was performed. • Drill sample intervals are based on a 1 metre sample interval, unless stated in the table of drill intersections as previously reported (see the section "other substantive exploration data" below). • No metal equivalent values have been reported.
Relationship between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.* • *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* • *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').*	• The relationship between mineralisation widths and intercept lengths in the case of the Ugur deposit is less critical as the mineralisation dominantly forms a broad scale oxide zone. The mineralisation does show trends of grade distribution as determined in the block modelling process. • All intercepts are reported as down-hole lengths. All drilling for the resource and reserve estimate were vertical (see section "Diagrams").
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported. These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	Refer main report
Balanced reporting	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	• All sampled intervals have been previously reported by Anglo Asian Mining via regulated news service (RNS) announcements of the London Stock Exchange (AIM). These data are available on the Anglo Asian Mining website.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	• Previous Anglo Asian Mining announcements on the AIM that report on exploration data of the Ugur deposit include: ➢ 17 October 2016; New Gold Discovery at its Gedabek Licence Area ➢ 16 December 2016; Significant oxide zone drilled at newly discovered Ugur deposit

Criteria	JORC Code explanation	Commentary
		➢ 18 April 2017; Strategy update and Q1 2017 review - Gedabek gold, copper and silver mine, Azerbaijan ➢ 8 May 2017; Ugur Gold Deposit Development & 2017 Strategy Update ➢ 24 July 2017; Ugur Gold Deposit Development and Gedabek Exploration Update • Additional information including photographs of the Ugur area can be viewed on the Anglo Asian Mining website, http://www.angloasianmining.com
Further work	• *The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).* • *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	• No further exploration drilling is planned at the Ugur deposit. Exploration will continue in the Ugur area to test for extensions of the mineralised zones and for other “centres” of mineralisation. Details of this work has not been planned yet. The intent is to initially produce JORC Mineral Resources and Ore Reserves and to bring the deposit into production. • No diagrams to show possible extensions are presented in this document as the work is yet to commence.

Section 3 Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	- *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* - *Data validation procedures used.*	- The Ugur database is stored in Excel ® software. A dedicated database manager has been assigned and checks the data entry against the laboratory report and survey data. - Geological data is entered by a geologist to ensure no confusion over terminology, while laboratory assay data is entered by the data entry staff. - A variety of checks are in place to check against human error of data entry. - All original geological logs, survey data and laboratory results sheets are retained in a secure location. - Independent consultants “Datamine” who carried out the resource estimation also carried out periodic database validation during the period of geological data collection, as well as on completion of the database. - The validation procedures used include random checking of data as compared the original data sheet, validation of position of drillholes in 3D models, and targeting figures deemed “anomalous” following statistical analysis. Hence there are several levels of control.
Site visits	- *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* - *If no site visits have been undertaken indicate why this is the case.*	- The CP is an employee of the company and as such has been actively in a position to be fully aware of all stages of the exploration and project development. The CP has worked very closely with the independent resource and reserve estimation staff of Datamine, both on site and remotely, to ensure knowledge transfer of the geological situation, to allow geological “credibility” to the modelling process. Extensive visits have been carried out by two staff of Datamine over the last year and have been fully aware of the Ugur project development. All aspects of the data collection and data management

Criteria	JORC Code explanation	Commentary
		has been observed.
Geological interpretation	- *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* - *Nature of the data used and of any assumptions made.* - *The effect, if any, of alternative interpretations on Mineral Resource estimation.* - *The use of geology in guiding and controlling Mineral Resource estimation.* - *The factors affecting continuity both of grade and geology.*	- The geological interpretation is considered robust. Geological data collection includes surface mapping, stream sediment and outcrop sampling, RC and core drilling. This has amassed a significant amount of information for the deposit. Various software have been used to model the deposit, including Leapfrog ®, Surpac® and Datamine ®, using dynamic anisotropy to develop the mineralised shells which were subsequently verified. - The geological team have worked in the licence area for many years and the understanding and confidence of the geological interpretation is considered high. - No alternative interpretations of the geology have had any effect on the resource model. - The geology has guided the resource estimation, especially the structural control, where for example faulting has defined "hard" boundaries to mineralisation. The deposit structural orientation was used to control the orientation of the drilling grid and the resource estimation search ellipse orientation. - Grade and geological continuity has been established by the extensive 3D data collection. The deposit is relatively small (300 metres by 200 metres), and the continuity is well understood, especially in relation to structural effects. - A geological interpretation of main mineralised body was completed utilising geological sections typically at spacings of about 20m. These interpretations were used to form a wireframe (solid) in Datamine, that was subsequently used as the main domain/mineralised zones for resource estimation.
Dimensions	- *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	- The footprint of the whole mineralisation is about 300metres by 200 metres, with about 110 metres overall thickness. The main mineralised domain is 230 metres by 170 metres in plan and about

Criteria	JORC Code explanation	Commentary
		100metres thickness.

Estimation and modelling techniques	- *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.*
	- *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.*
	- *The assumptions made regarding recovery of by-products.*
	- *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).*
	- *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.*
	- *Any assumptions behind modelling of selective mining units.*
	- *Any assumptions about correlation between variables.*
	- *Description of how the geological interpretation was used to control the resource estimates.*
	- *Discussion of basis for using or not using grade cutting or capping.*
	- *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*
		- A geological interpretation of main mineralised body was completed utilising geological sections typically at spacings of about 20m. These interpretations were used to form a wireframe (solid) in Datamine, that was subsequently used as the main domain/mineralised zone for resource estimation. Estimation process includes:
		  - Drill holes data were flagged as inside and outside of main zones of mineralisation. Outlier study of gold and silver showed a few samples out of range. A top-cut grade of 16 g/t for gold and 108 g/t for silver was applied for data inside the main mineralised zone.
		  - Drill holes data composited by 2m along the holes.
		  - Variogram analyses of gold data has been carried out using Datamine software. The ranges of variograms at major and semi-major direction are 30 metres and 23 metres. Minor directions show poor continuity and it considered as 10m. The major Azimuth is 040 degrees with 20 degree dip.
		  - Three estimation passes were used; the first search was based upon the variogram ranges in the three principal directions (30x25x10). The second search was 1.5 times and third search was 2 times of first search. Min and Max of samples were 4 and 14 for first and second search and 1 and 14 for third search.
		  - Estimation was carried out using ordinary kriging at the parent block.
		  - More than 90% of blocks inside the main domain/mineral zone are estimated in first search as they fall in the dense drilling area, being the main zone of mineralisation.
		  - The estimated gold block model grades were visually validated against the input drillhole data. Comparisons were carried out against the drillhole data by bench.

Criteria	JORC Code explanation	Commentary
		- The resource estimation was carried out using Datamine Studio RM software. - No previous mining has occurred to allow for check estimates. This will be a requirement on production start-up. - The deposit contains gold and silver mineralisation, with minor copper, and other base metal were tested, and full multi-element analysis was carried out at external laboratories. Results showed no other by-products. - Deleterious non-grade elements were checked and the situation of acid rock drainage (ARD) studies. However, given the extraction dominantly of oxide ores (87% oxide, 3% sulphide, 0.1% transition, 9.9% unclassified within the samples zone) and the processing at a current facility, there are no immediate concerns. Should future mining of the sulphide zone occur or sulphide be released, independent on-site environmental engineers will monitor and recommend mitigation of ARD situations. - A block model was created with parent size of 5x5x5 metres. Sub-blocking is not allowed in X and Y but in Z direction minimum to ½ of block height. This is considered optimum with regards the data spacing and for the planned extraction design, with 5 metre open pit benches in “ore”.
Moisture	- *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	- Tonnage has been estimated on a dry basis
Cut-off parameters	- *The basis of the adopted cut-off grade(s) or quality parameters applied.*	- Continuity of grade was assessed at a range of cut-offs between 0.1 g/t gold and 1.0 g/t gold in 0.1 g/t increments. A tonnage-grade table and graph was prepared based on different cut-off. Following interrogation of data and continuity, the resources area reported above 0.2 g/t gold grade cut-off.
Mining factors or	- *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if*	- Given the geometry of the mineralised zone, the fact the central part is exposed at surface, and a very low forecast waste ratio, an open pit

Criteria	JORC Code explanation	Commentary
assumptions	*applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	mining method is selected. Mining dilution and mining dimensions are referenced in this report. • Other mining factor are not applied at this stage.
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	• The Company currently operates an agitated leach plant, a flotation plant, a crushed heap leach facility, and a run-of-mine dump leach facility. As such, the basis for assumptions and predictions of processing routes and type of "ores" suitable for each process available are well understood. • Metallurgical testwork has been carried out to assess the amenability of the Ugur mineralisation to cyanidation and leaching processes. The results showed a high level of amenability. The mineralisation is an "oxide" type, that is relatively soft, and requires comparatively low levels of processing reagents for recovery. • Metallurgical testwork was carried out on samples with a mean of a range of gold grades; 3.6g/t, 2.5g/t, 1.5g/t and 1.0g/t. The results for a 48 hour bottle roll test showed high gold recovery and low cyanide usage (see below). Leaching, % Au Ag 88.5 82.8 85.7 62.0 95.0 60.5 83.8 73.2

Criteria	JORC Code explanation	Commentary
		- No metallurgical factors assumptions have been used in mineral resource estimate.
Environmental factors or assumptions	- *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	- The Ugur deposit is located within a mining contract area in which the company operates two other mines. As part of the initial start-up, environmental studies and impacts were assessed and reported. This includes the nature of process waste as managed in the tailings management facility (TMF). Other waste products are fully managed under the HSEC team of the company (including disposal of mine equipment waste such as lubricants and oils). - An independent environmental engineering company CQA International Ltd (CQA) has carried out a study of Ugur including installing baseline monitoring systems, and will be integral to the extraction and processing of the ores. - No environmental assumptions have been used in mineral resource estimation.
Bulk density	- *Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.* - *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.* - *Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	- Bulk density measurements have been determined. A total of 538 samples were tested from selected core samples, that comprised both mineralisation and wall rocks. The density was tested by rock type, extent of alteration and depth. The method used was hydrostatic weighing. - Of the 538 samples, 426 density measurement samples are inside mineralisation wireframes. The average density of these samples is 2.62 t/m^3 and has been used for resource calculation. - Density data are considered appropriate for Mineral Resource and Mineral Reserve estimation.
Classification	- *The basis for the classification of the Mineral Resources into varying confidence categories.* - *Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity*	- The Mineral Resource has been classified on the basis of confidence in the continuity of mineralised zones, as assessed by the geological block model based on sample density, drilling density, and confidence in the geological database. Depending on the estimation parameters

Criteria	JORC Code explanation	Commentary
	and distribution of the data). - *Whether the result appropriately reflects the Competent Person's view of the deposit.*	(number of samples per search volume), the resources were classified as Measured, Indicated or Inferred Mineral resources, as defined by the parameters below: ➢ Blocks inside the mineralised zone that capture samples with at least 2 drill holes in first search volume were considered as Measured Resources. ➢ Blocks inside the mineralised zone that capture samples from at least 2 holes data in second search volume are considered as Indicated Resources. ➢ Blocks inside the mineralised zone which fall within with in third search volume are considered as Inferred Resources. ➢ All blocks outside of main central mineralised zone are considered as Inferred. • The results reflect the Competent Person's view of the deposit.
Audits or reviews	- *The results of any audits or reviews of Mineral Resource estimates.*	• Datamine company developed and audited the Mineral Resource block model. Two Datamine engineers worked on the resources and reserves and were able to verify work and procedure. • Datamine have been involved with other mining projects of the company within the same licence area as Ugur and as such are familiar with the processing methods available, value chain of the mining and cost structure. The data has been audited and considered robust for Mineral Resource estimates. • Internal company and external reviews of the Mineral Resources yield estimates that are consistent with the Mineral Resource results. The methods used include sectional estimation, and three-dimensional modelling utilising both geostatistical and inverse distance methodologies. All results showed good correlation. • Recommendations include upgrading laboratory and management systems, and the future implementation of a laboratory information management system. The grade control data produced during mining

Criteria	JORC Code explanation	Commentary
		should be correlated back into the resource model to check for consistency or variation.
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	• Statistical and visual checking of the block model is as expected given the geological data. The mineralisation is tightly constrained geologically, and the level of data acquired and the resource estimation approach is to international best practice. The application of both statistical and geostatistical approaches results in high confidence of the resource resulting in the appropriate relative amounts of Measured, Indicated and Inferred Mineral resources. The periphery of the deposit where sample density was not as high as over main mineralised zone, yielded much of the Inferred category resource. • The drilling grid and sample interval is sufficient to assign Measured and Indicated Mineral Resources. • The Mineral Resource statement relates to a global estimate for the Ugur deposit. • The Ugur deposit has not been previously mined, so no production data is available for comparison. It is recommended that on commencement of extraction of mineralisation, grade control and mining data are used to compare with the Resource model.

Section 4 Estimation and Reporting of Ore Reserves

(Criteria listed in section 1, and where relevant in sections 2 and 3, also apply to this section.)

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr>
<td><i>Mineral Resource estimate for conversion to Ore Reserves</i></td>
<td>

<i>Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve.</i>
<i>Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.</i>

</td>
<td>

Refer to Section 3 (Estimation and Reporting of Mineral Resources)
A JORC resource estimate comprising Measured, Indicated and Inferred Resources has been made for the Ugur Deposit (as tabulated below):

<table>
<tr><th>Mineral Resources</th><th>Tonnage (Mt)</th><th>Gold Grade (g/t)</th><th>Silver Grade (g/t)</th></tr>
<tr><td>Measured</td><td>4.12</td><td>1.2</td><td>6.3</td></tr>
<tr><td>Indicated</td><td>0.34</td><td>0.8</td><td>3.9</td></tr>
<tr><td><i>Measured+Indicated</i></td><td><i>4.46</i></td><td><i>1.2</i></td><td><i>6.2</i></td></tr>
<tr><td>Inferred</td><td>2.50</td><td>0.3</td><td>2.1</td></tr>
<tr><td>Total</td><td>6.96</td><td>0.9</td><td>4.7</td></tr>
</table>

The contained metal in ounces of gold and silver is presented below:

<table>
<tr><th>Mineral Resources</th><th>Gold ('000 ounces)</th><th>Silver ('000 ounces)</th></tr>
<tr><td>Measured</td><td>164</td><td>841</td></tr>
<tr><td>Indicated</td><td>8</td><td>44</td></tr>
<tr><td><i>Measured+Indicated</i></td><td><i>172</i></td><td><i>884</i></td></tr>
<tr><td>Inferred</td><td>27</td><td>165</td></tr>
<tr><td>Total</td><td>199</td><td>1,049</td></tr>
</table>

The relative % of contained metal shows a very high % of Measured Resource and Indicated Resource that can be tested for Reserve estimation.

</td>
</tr>
</table>

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>
<table>
<tr><th>Mineral Resources</th><th>% gold ounces</th><th>% silver ounces</th></tr>
<tr><td>Measured</td><td>82%</td><td>80%</td></tr>
<tr><td>Indicated</td><td>4%</td><td>4%</td></tr>
<tr><td><i>Measured+Indicated</i></td><td>87%</td><td>84%</td></tr>
<tr><td>Inferred</td><td>13%</td><td>16%</td></tr>
<tr><td>Total</td><td>100%</td><td>100%</td></tr>
</table>
The Ore Reserve statement is inclusive (not additional to) of the Resource statement.
</td></tr>
<tr><td><i>Site visits</i></td><td><i>Comment on any site visits undertaken by the Competent Person and the outcome of those visits.</i>
<i>If no site visits have been undertaken indicate why this is the case.</i></td><td>The Competent Person is an employee of the company and as such has been actively in a position to be fully aware of all stages of the exploration and project development including the estimation of Mineral resources and Ore Reserves. The Competent Person has worked very closely with the independent resource and reserve estimation staff of Datamine company, both on site and remotely, to ensure knowledge transfer of the geological situation, to allow geological “credibility” to the modelling process. Extensive visits have been carried out by two staff of Datamine (one of whom estimated the resources and one estimate the reserves) over the last year and have been fully aware of the Ugur project development. All aspects of the data collection and data management has been observed.</td></tr>
<tr><td><i>Study status</i></td><td><i>The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves.</i>
<i>The Code requires that a study to at least Pre-Feasibility Study level has been undertaken to convert Mineral Resources to Ore Reserves. Such studies will have been carried out and will have determined a mine plan that is technically achievable and economically viable, and that</i></td><td>Study undertaken to enable Mineral Resources to be converted to Ore Reserves are considered as being Feasibility level. The ore will be mined utilising the current mining fleet and will be processed in the current processing facilities of the Company which operates two other mines in the same licence/contract area. The Ugur deposit is</td></tr>
</table>

Criteria	JORC Code explanation	Commentary
	material Modifying Factors have been considered.	considered to part of the same geological terrain. - A technically achievable mine plan that is economically viable has been designed taking into consideration the JORC resources and modifying factors.
Cut-off parameters	- *The basis of the cut-off grade(s) or quality parameters applied.*	- Financial factors included in the cut-off grade estimates are process and overhead costs, mining dilution, payable gold and silver price, and processing recovery and used in the basis for cut-off grade calculation. - The ore from Ugur can be processed by three different available processing methods within the Gedabek contract area, namely agitation leach (AGL), heap leach of crushed material (HLC) and heap leach of blasted material or run-of-mine (ROM). - The acceptable gold head grade in grammes per tonne gold for AGL, HLC and ROM is 1.8g/t ,0.8g/t and 0.47g/t respectively. - Further to the gold cut-off grade calculations, after long term scheduling the mill cut-off grade resulted in 0.3g/t gold.
Mining factors or assumptions	- *The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore Reserve (i.e. either by application of appropriate factors by optimisation or by preliminary or detailed design).* - *The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc.* - *The assumptions made regarding geotechnical parameters (eg pit slopes, stope sizes, etc), grade control and pre-production drilling.* - *The major assumptions made and Mineral Resource model used for pit and stope optimisation (if appropriate).* - *The mining dilution factors used.* - *The mining recovery factors used.* - *Any minimum mining widths used.* - *The manner in which Inferred Mineral Resources are*	- On establishing the modifying factors, the Mineral Reserve has been optimised using the Datamine NPV Scheduler® software. This resulted in the economic open pit shell and contained mineable material in that pit shell. Subsequently, this was further optimised in the mine design process, using Datamine Studio OP ® software, where bench toe and crest, catch benches and haul road layout was designed. - The mining method selected is by open pit method given the orebody geometry and the position relative to topographic surface. The central part of the orebody is exposed at surface, and over the remaining 70% surface area of the orebody there is a top soil cover varying in thickness between zero and 50 centimetres. Access to the orebody is from surface. The open pit mining method is considered appropriate, and will comprise conventional truck and shovel.

Criteria	JORC Code explanation	Commentary
	utilised in mining studies and the sensitivity of the outcome to their inclusion. • *The infrastructure requirements of the selected mining methods.*	• Pit slope angles have been determined based on independent geotechnical investigation taking into account geological structure, rock type and design orientation parameters. The overall pit slope angle is 38 degrees containing an average bench angle of 58 degree. • Based on the geotechnical findings further to the independent report by CQA, the overall pit slope angle is maximum 38degrees, berm width 6 metres and after each 5 benches (50 metre height), a catch bench of 10 metre width should be considered for the open pit design. • Mining dilution used in the Datamine NPV Scheduler software for reserve estimation is 5%. • Ore mining recovery factor used in the Datamine NPV Scheduler software for reserve estimation is 95%. • A minimum mining width of 20m has been used. • The total tonnage of inferred material in the final pit design was 87,100 tonnes which represents about 2.37% of total ore tonnage in the pit and contains 0.76% (1,134 ounces) of contained gold in the pit. • The inferred material was excluded from economic model in NPV Scheduler so it had no impact on the total reserve. • Infrastructure required for the open pit mining method include haul road access (completed to the mine area), offices for geology/mining department, mining workshop, fuel storage, weighbridge and medical/HSEC facilities. Explosives will be transported from another mine operating within the contract area.
Metallurgical factors or assumptions	• *The metallurgical process proposed and the appropriateness of that process to the style of mineralisation.* • *Whether the metallurgical process is well-tested technology or novel in nature.* • *The nature, amount and representativeness of metallurgical test work undertaken, the nature of the*	• The proposed metallurgical processes are well tested being processing facilities of current mining operations in the contract area. The processing facilities include agitation leach by conventional methods, crushed heap leach, and run-of-mine dump leach. AGL process comprises comminution (crushing and grinding), Knelsen

Criteria	JORC Code explanation	Commentary
	metallurgical domaining applied and the corresponding metallurgical recovery factors applied. - *Any assumptions or allowances made for deleterious elements.* - *The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole.* - *For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?*	concentration, thickening, agitation leaching, resin-in-pulp extraction, and elution and electrowinning to produce gold dorè. The final product will be shipped off site for refining. Tails from the process will be transferred via gravity pipeline to the existing tailings management facility (TMF) that has enough capacity to manage the ore from the Ugur deposit. - Metallurgical testwork has been conducted in the form of bottle roll testing and column leach tests. The amount of testwork is considered representative of the processing technology to be employed. - Deleterious elements were not detected in analytical tests and assaying utilised for the resource estimate. - No pilot scale testwork has been conducted. However, given the nature of the ore type and its close relationship with existing ore bodies being processed, the metallurgical testwork carried out is considered representative of the orebody as a whole. - The ore reserve estimation has been based on the appropriate mineralogy to meet the specification.
Environmental	- *The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.*	- Previous ESIA (Environmental Social Impact Assessment) has been carried out by Amec Foster Wheeler (2012) and TexEkoMarkazMMC (2012) (submitted to Government authorities). The Ugur deposit is located within the Gedabek Contract Area for which the ESIA is valid, hence the most recent ESIA is applicable to Ugur. Processing and tailings storage reported in the ESIA is the same as will be utilised for Ugur ores. - Environmental and geotechnical consultants, CQA International Ltd of the UK (CQA), have on-site representation, and carried out both geotechnical and environmental assessments of the Ugur mine area. Baseline environmental monitoring has been carried out on receptors downstream of the mine site, due to an additional

Criteria	JORC Code explanation	Commentary
		catchment being located in the vicinity of the Ugur mine. • The waste rock has a low potential for acid rock drainage due to the absence of sulphide bearing mineralisation. Watercourses downstream of stockpiles will be monitored on a routine basis for pH and heavy metals. • A topsoil management plan is in place, that has been reviewed by a CQA consultant deemed in accordance with the storage principles of the Ministry of Ecology and Natural Resources of the Republic of Azerbaijan and European Union (EU) guidelines. Topsoil removal took place in August 2017, and be stockpiled in a dedicated location with specific design parameters. Stockpiling of materials will be carried out following the soil management plan. • A stockpile area for waste rock has been identified following condemnation drilling verifying the absence of mineralisation beneath the proposed stockpile. The top soil at the planned site will be removed, and the hill terraced to "key" in the waste dump for maximum stability. • The tailings management facility (TMF) has the capability for the additional storage requirements for Ugur process waste. The design and operations of the TMF have been reviewed by CQA along with a visit by the Ministry of Ecology and Natural Resources of the Republic of Azerbaijan. Regular environmental monitoring is carried out at the TMF, along with monitoring all receptors associated with the TMF. • All approvals for conducting the mining fall under the management "PSA" agreement.
Infrastructure	• *The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can be provided, or accessed.*	• Infrastructure is considered excellent to the deposit. The deposit is located within the Company's contract/licence area with extraction rights according to the Government contract. Ore can be processed at the Company's current facilities, with ore being delivered by truck

Criteria	JORC Code explanation	Commentary
		from the mine to processing via the newly constructed haul road over a distance of about 6 kilometres. Land availability for the mine and associated infrastructure is approved. Offices and mechanical workshop buildings are available within the company and will be relocated to Ugur. Power for the offices and weighbridge will be initially via diesel generators, although solar power is also under consideration. Labour is readily available as the operation is relatively small and only additional mine site labour will be required. G&A and process labour are part of the existing company compliment of staff. Regarding accommodation, canteen facilities and associated services, the Ugur deposit can be considered a "satellite" deposit to the current mining operations and will be serviced by the current infrastructure.
Costs	• *The derivation of, or assumptions made, regarding projected capital costs in the study.* • *The methodology used to estimate operating costs.* • *Allowances made for the content of deleterious elements.* • *The derivation of assumptions made of metal or commodity price(s), for the principal minerals and co-products.* • *The source of exchange rates used in the study.* • *Derivation of transportation charges.* • *The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc.* • *The allowances made for royalties payable, both Government and private.*	• Project capital costs are "minimal" given that no processing facilities or manpower camps are required. The costs in relations to the facilities already referenced above are based on actual quotations and capital construction experience at the licence area and sustaining capital projects are based on operational experience locally. • Operating costs are estimated based on current mining and processing operations within the licence area, as the processing will be carried out at the same plants, and the mining contract and haulage costs are the same as current contracts. • No allowances have been made for deleterious elements. • Commodity pricing is based on forecasts by reputable market analysts. • Local Azeri exchange rates are pegged to the United States $. The source of exchange rates used in the study is the Central Bank of the Republic of Azerbaijan. • Transportation charges are based on current contracts that will be

Criteria	JORC Code explanation	Commentary
		extended to include haulage of ore from Ugur deposit to the processing facilities. All other transport costs will be per the current contracts for the operating mines. • Treatment and refining costs are based on current contracts, as the ore will be treated in the operating processing plants and refined under the current agreement. • Royalties have been considered as part of the cost structure for the company to operate under the Government Contract. • The estimated operating costs per tonne used in NPV Scheduler are: **Parameters used in NPV Scheduler** **Processing cost (includes G&A)** *per tonne of ore* AGL $ 29.22 HL Crushed $ 6.37 HL_ROM $ 5.22 **Other costs** Total G&A $ 3.22 Mining cost $ 1.75 Haulage cost (per tonne km) Manat 0.1
Revenue factors	• *The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc.* • *The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.*	• Revenue is based on the US$ gold price and US$ silver price. • The price of gold in the reserve model is $1250 per troy ounce and the price of silver in the reserve model is $18.66 per troy ounce.
Market assessment	• *The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future.* • *A customer and competitor analysis along with the identification of likely market windows for the product.* • *Price and volume forecasts and the basis for these forecasts.*	• The market for gold and silver is well established. The metal price is fixed externally to the Company, however, the Company has reviewed a number of metal forecast documents from reputable analysts and is comfortable with the market supply and demand situation. • A specific study of customer and competitor analysis has not been

Criteria	JORC Code explanation	Commentary
	- *For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.*	completed as part of this project. - Price and volume forecasts have been studied in reports from reputable analysts, based on metal supply and demand, US$ forecasts and global economics. - Industrial minerals do not form part of this study.
Economic	- *The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc.* - *NPV ranges and sensitivity to variations in the significant assumptions and inputs.*	- Prices for gold and silver used in NPV Scheduler are: Gold: $40.19 per gramme Silver: $0.55 per gramme - Processing Recovery (for gold / silver) % Agitation Leach 90% / 66% Crushed Heap Leach 70% / 7% Run-of-mine (ROM) 40% / 7% - Costs used in NPV are show below: **Parameters used in NPV Scheduler** **Processing cost (includes G&A)** *per tonne of ore* AGL $ 29.22 HL Crushed $ 6.37 HL_ROM $ 5.22 **Other costs** Total G&A $ 3.22 Mining cost $ 1.75 Haulage cost (per tonne km) Manat 0.1 Selling Cost %0.05 of revenue of Gold Selling Cost %0 of revenue of Silver - Sensitivity analysis has been used at a range of gold prices.
Social	- *The status of agreements with key stakeholders and matters leading to social licence to operate.*	- To the best of the Competent Person’s knowledge, agreements with key stakeholders and matters leading to social licence to operate are

Criteria	JORC Code explanation	Commentary
		valid and in place.
Other	- *To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore Reserves:* - *Any identified material naturally occurring risks.* - *The status of material legal agreements and marketing arrangements.* - *The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.*	- There are no material naturally occurring risk associated with the Ore Reserves. - Anglo Asian Mining plc is currently compliant with all legal and regulatory agreements, and marketing arrangements. - The project is located within a current contract area that is managed under a “PSA” production sharing agreement. - The PSA grants the Company a number of periods to exploit defined licence areas, known as Contract Areas, agreed on the initial signing with the Azerbaijan Ministry of Ecology and Natural Resources ('MENR'). The exploration period allowed for the early exploration of the Contract Areas to assess prospectivity can be extended. - A 'development and production period' commences on the date that the Company issues a notice of discovery, which runs for 15 years with two extensions of five years each at the option of the Company. Full management control of mining in the Contract Areas rests with Anglo Asian. - Under the PSA, Anglo Asian is not subject to currency exchange restrictions and all imports and exports are free of tax or other restriction. In addition, MENR is to use its best endeavours to make available all necessary land, its own facilities and equipment and to assist with infrastructure. - The PSA is valid for the forecast life of mine.
Classification	- *The basis for the classification of the Ore Reserves into varying confidence categories.* - *Whether the result appropriately reflects the Competent Person’s view of the deposit.* - *The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).*	- Measured Mineral Resources have been converted to Proved Reserves after applying the modifying factors. - Indicated Mineral Resources have been converted to Probable Ore Reserves after applying modifying factor. - The resultant Ore Reserves are appropriate given the level of understanding of the deposit geology and reflects the Competent Person’s view of the deposit.

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>The inferred material was excluded from economic model in NPV Scheduler so it had no impact on the total reserve, and no Probable Ore Reserves have been derived from Measured Mineral Resources.</td></tr>
<tr><td>Audits or reviews</td><td>The results of any audits or reviews of Ore Reserve estimates.</td><td>Datamine company developed and audited the Mineral Resource and Mineral Reserve block models. Two Datamine engineers worked on the resources and reserves and were able to verify work and procedure.
Datamine have been involved with other mining projects of the company within the same licence area as Ugur and as such are familiar with the processing methods available, value chain of the mining and cost structure. The data has been audited and considered robust for Ore Reserve estimates.
Internal company and external reviews of the Ore Reserves yield estimates that are consistent with the Ore Reserve results. The in-situ Ore Reserves classified by process type is presented below:
<table>
<tr><th>Ore Reserves (Process & Class)</th><th>Tonnage (Mt)</th><th>Gold Grade (g/t)</th><th>Silver Grade (g/t)</th><th>Gold ('000 ounces)</th><th>Silver ('000 ounces)</th></tr>
<tr><td>Proved-AGL</td><td>1,604,200</td><td>1.94</td><td>10.26</td><td>99.99</td><td>529.06</td></tr>
<tr><td>Proved-HLC</td><td>1,261,813</td><td>0.84</td><td>4.95</td><td>34.22</td><td>200.74</td></tr>
<tr><td>Proved-ROM</td><td>504,400</td><td>0.48</td><td>3.05</td><td>7.85</td><td>49.45</td></tr>
<tr><td>Total Proven</td><td>3,370,413</td><td>1.31</td><td>7.19</td><td>142.06</td><td>779.25</td></tr>
<tr><td>Probable-AGL</td><td>23,238</td><td>1.42</td><td>5.12</td><td>1.06</td><td>3.83</td></tr>
<tr><td>Probable-HLC</td><td>120,413</td><td>0.80</td><td>4.56</td><td>3.12</td><td>17.65</td></tr>
<tr><td>Probable-ROM</td><td>71,988</td><td>0.47</td><td>3.10</td><td>1.09</td><td>7.16</td></tr>
<tr><td>Total Probable</td><td>215,639</td><td>0.76</td><td>4.13</td><td>5.27</td><td>28.64</td></tr>
<tr><td>Proved+Probable</td><td>3,586,052</td><td>1.28</td><td>7.01</td><td>147.33</td><td>807.89</td></tr>
</table>
The reference point for the Ore Reserves is where the ore is delivered to the processing plant.</td></tr>
</table>

Criteria	JORC Code explanation	Commentary
		- The amount of waste material calculated inside the pit shell is about 3.05 million tonnes, resulting in a strip ratio (ore:waste) of 1:0.83.
Discussion of relative accuracy/ confidence	- *Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate.* - *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* - *Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the current study stage.* - *It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	- The Ore Reserve has been completed feasibility standard with the data being generated from a tightly spaced drilling grid, thus confidence in the resultant figures is considered high. - Extraction of ore from the Ugur deposit will commence in August 2017, and processing of the ores will commence in September 2017. As on date of this report, top soil pre-strip has commenced. - Mining costs and haulage costs will be as per the current contracts in place being utilised at other mines in the contract area. - Project capital is well managed, and certain infrastructure facilities are available from with the Anglo Asian Mining group, thus minimising capital requirements. - The global Mineral Resource estimates have been estimated by using a sectional (polygonal) method, and by 3D modelling using both inverse distance and kriging methods. All results are within 5% of each other. - The Modifying Factors for mining, processing, metallurgical, infrastructure, economic, gold price, legal, environmental, social and governmental factors as referenced above have been applied to the pit design and Ore Reserves calculation on a global scale and data reflects the global assumptions. - No mine production data is available at this stage for reconciliation and/or comparative purposes.

Section 5 Estimation and Reporting of Diamonds and Other Gemstones

(Criteria listed in other relevant sections also apply to this section. Additional guidelines are available in the 'Guidelines for the Reporting of Diamond Exploration Results' issued by the Diamond Exploration Best Practices Committee established by the Canadian Institute of Mining, Metallurgy and Petroleum.)

Estimation and Reporting of Diamonds and Other Gemstones in not applicable to this Statement of Resources

GLOSSARY AND OTHER INFORMATION

1. GLOSSARY OF JORC CODE TERMS

The following definitions are extracted from the JORC Code, 2012 Edition

Cut-off grade	The lowest grade, or quality, of mineralised material that qualifies as economically mineable and available in a given deposit. May be defined on the basis of economic evaluation, or on physical or chemical attributes that define an acceptable product specification.
Indicated Mineral Resource	An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade (or quality), densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes, and is sufficient to assume geological and grade (or quality) continuity between points of observation where data and samples are gathered. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Ore Reserve.
Inferred Mineral Resource	An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade (or quality) are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade (or quality) continuity. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to an Ore Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
JORC	JORC stands for Australasian Joint Ore Reserves Committee (JORC). The Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (the JORC Code) is widely accepted as the

	definitive standard for the reporting of a company's resources and reserves. The latest JORC Code is the 2012 Edition.
Measured Mineral Resource	A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade (or quality), densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes, and is sufficient to confirm geological and grade (or quality) continuity between points of observation where data and samples are gathered. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proved Ore Reserve or under certain circumstances to a Probable Ore Reserve
Mineral Reserves or Ore Reserves	An 'Ore Reserve' is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.
Mineral Resource	A 'Mineral Resource' is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade (or quality), and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade (or quality), continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling. Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories.
Modifying Factors	'Modifying Factors' are considerations used to convert Mineral Resources to Ore Reserves. These include, but are not restricted to, mining, processing,

	metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.
Probable Ore Reserve	A 'Probable Ore Reserve' is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Ore Reserve is lower than that applying to a Proved Ore Reserve.
Proved Ore Reserve	A 'Proved Ore Reserve' is the economically mineable part of a Measured Mineral Resource. A Proved Ore Reserve implies a high degree of confidence in the Modifying Factors.

2. SOFTWARE USED IN THE MINERAL RESOURCE AND RESERVES ESTIMATE

"***Datamine Studio RM***" and "***NPV Scheduler***" software was used in the estimate of Mineral Resources and the calculation of Ore Reserves.

"***NPV Scheduler"*** is computer software that uses the Lerch-Grossman algorithm, which is a 3-D algorithm that can be applied to the optimisation of open-pit mine designs. The purpose of optimisation is to produce the most cost effective and most profitable open-pit design from a resource block model to define the reserve.

Appendix II- CQA Report: Pit Slope Stability Assessment

Uğur Oxidised Ore Body, Gedebek

Pit Slope Stability Assessment

Report prepared for: Azerbaijan International Mining Company

Date: 27 July 2017

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Quality management data

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Prepared by	Martin Avery
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Authorised to be issued as a formal report from CQA International Ltd by (name, position)	Andy Hall
Signature	

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Uğur Oxidised Ore Body, Gedebek
Pit Slope Stability Assessment
CQA Reference: 30305, 9 June 2017

Executive Summary

The recently discovered oxidised orebody of the Uğur deposit will be exploited from an open pit, with mining expected to commence in late summer or early autumn of 2017.

A geotechnical assessment of the potential stability issues has been made on the basis of data collected from geological mapping, rockmass classification and the logging of boreholes.

The host rock quality was classified with an RMR of "Good" and an NGI-Q of "Fair". The RMR has been shown to be more representative of ground conditions in the main Gedebek pit. An RMR index, adapted for slope stability studies, classifies the host rock as stable" to "partially stable", with the risk of some toppling or wedge failures.

The analysis of discontinuities in the host rock confirms the risk of some toppling or wedge failures. The level of risk will depend on the orientation of the pit faces, and a sensitivity analyses determined that the least favourable orientations lie between 305° and 005°.

The oxidised ore rockmass quality with an RMR of "Poor", primarily on the basis of core logging and interpretation of core photographs. Some localised failures would be expected on steep slopes behind benches. It was not possible to measure discontinuity data and so a sensitivity analysis of pit face orientations could not be carried out.

With current data, we suggest that the oxidised ore is excavated back to host rock on each bench as soon as possible, to avoid high slopes in this material. The final pit slope design parameters in the host rock are an overall slope angle of 38^{o}, bench height of 10m and bench width of 6m. Every fifth bench should be 10m wide to catch falling rocks. A typical bench batter is 58^{o} but localised steepening to 70^{o} is possible, subject to inspection, if required. These parameters may need to be modified to accommodate the transition between rock types.

The data and assessment may be influenced by two key issues: in oxidised ore body cores were mostly logged from photographs; in the host rock natural exposures were limited and may be biased in favour of better quality rock. Periodic data collection and assessment is recommended during excavations in order to confirm the geotechnical conditions. This should comprise geotechnical inspections of excavated benches in the ore body and host rock, with strength tests carried out on selected samples.

We also suggest that confirmatory stability calculations are undertaken when the preliminary design for the Uğur Pit is prepared and the face orientations are known.

Contents

1. Introduction

1.1 Scope of report

The recently discovered oxidised orebody of the Uğur deposit will be exploited from an open pit, with mining expected to commence in late summer or early autumn of 2017.

This report presents a geotechnical assessment of the rockmass, analysis of slope stability issues and recommendations for the basis of pit slope design.

The scope of work was summarised in CQA's proposal, reference 20436, and was instructed in Call-Off Order № 15 of CQA's service contract with AIMC.

1.2 Approach

CQA's approach to delivering the services involved:

Data collection	Site reconnaissance, geological mapping, determining rockmass parameters, borehole core logging, data extraction from prepared borehole records
Data analysis	Graphical, statistical and stereonet analysis of discontinuity data, rockmass assessment, stability analyses for different failure modes, determination of suitable pit face angles

The data collection and analysis focussed on the oxidised ore body and host rock, with slightly different approaches being required for each material.

The field work for data collection was undertaken by CQA personnel in early March 2017. Much of the data analysis was also carried out on site and in our offices in Baku.

2. Data collection

2.1 Oxidised ore body

Although the oxidised ore body outcrops on the mountainside (at elevations of approximately 1820m - 1900m above datum), there is no visible exposure of the rock due to a thin covering of soil. The top of the ore body was visible in a shallow trial trench, but the rock was weathered and broken-up.

A number of cored boreholes have already been drilled in the ore body for reserve evaluation. Quantitative logging and rockmass characterisation from borehole cores was not possible because the core had, understandably, been sub-sampled for assay purposes. Also, some weaker rock may have deteriorated.

As a result, characterisation of discontinuities within the ore body was not possible. CQA examined the available core samples to obtain a qualitative impression of the rockmass quality. Quantitative data (primarily RQD) were obtained from borehole logs, which had compiled by AIMC's geological team. Additional RQD values were derived by CQA from photographs of the intact cores, which had been taken prior to assay sampling.

A summary of the coordinates, elevation and depth of the boreholes that were used is presented in Appendix A.

Overall, the cores revealed a sequence of igneous and volcanic rock types, with frequent veining and zones of oxidation and alteration. Some sections of core were primarily an aggregate of broken pieces, logged as a breccia. Solid core in less weathered rock displayed discontinuities with a variety of characteristics. Many were relatively rough and undulating, and the estimated spacing was typically 0.2m to 0.6m.

This approach provided much of the data that are necessary for determining rockmass characterisation values, although the orientation of discontinuities could not be defined. The latter need to be determined from rock exposures and it would be beneficial to inspect and record in-situ discontinuity data during excavation works. This would allow the rockmass assessment and stability analysis to be updated as work proceeds.

2.2 Host rock

Geological reconnaissance of the host rock was carried out in a 500m radius from the ore body outcrop. The purpose was to collect data on discontinuities and determine rockmass characteristics. It is likely that the final pit slopes will be formed in, or close to, the host rock, rather than the orebody.

Exposure of the host rock is limited by sporadic and thin soil cover and most exposures occur on the ridges around the ore outcrop and in the stream to the west. Discontinuity measurements were possible at 54 locations within the search radius; and rockmass characterisation was possible at 47 of these locations.

Due to the limited exposure of the host rock, it is possible that the data may have some geometrical bias. Additional measurements from pit slopes during excavations would be prudent to allow the data to be reassessed. Steeply dipping discontinuities may also be missed by vertical boreholes, which could be influential on the results if these were zones of weakness. In response to this observation, AIMC drilled an additional two inclined boreholes to investigate the possibility of steeply dipping zones. The cores also confirmed the characterisation of both the orebody and the host rock.

Photographs of key mapping locations are presented in Appendix B. The rockmass characterisation results are presented in Appendix C. The number of discontinuity (joint) sets was estimated visually. RQD was estimated over a 2m vertical scanline. These parameters, and a number of observations concerning discontinuity characteristics and rock strength, were used to derive the two commonly used indices of rockmass quality: RMR (Rockmass Rating - South African CSIR) and Q (Quality Designation - Norwegian Geotechnical Institute). Refer to Appendix D. Values were also estimated for a modified version of RMR (called SMR), which has been applied to slope stability analyses.

3. Data analysis

3.1 Discontinuity measurements – host rock

The discontinuity measurements were analysed graphically in lower-hemisphere equal area stereonet projections. The orientations of the main joint sets were determined statistically by contouring the poles of all planes and selecting significant concentrations of data points. These data were then compared to different scenarios of pit slope dimensions and rock friction angle in order to determine the likely stability.

Rock slope stability is largely controlled by the relative orientation of discontinuities and the rock face, in particular how many joint intersections are visible (i.e. “daylight”) in the face. Different pit face orientations allow different numbers of intersections to daylight, providing the joints are present, and this can be predicted by the stereonet analyses.

Logging of the inclined boreholes confirmed the presence of steeply dipping joints in the more competent rock horizons, which appear to be similar to the measured joint sets. These were mainly clean joints and so the presence of extensive sub-vertical planes of weakness does not seem likely.

The results are summarised on Figure 1. Six main joint set orientations were defined, with moderate to steep inclinations and a range of dip directions. The joint sets are not well defined and there were considerable variations around the main trends. Discontinuities outside the main sets (i.e. random joints) were observed at 74% of the locations. Random joints influence the stability by increasing possibility variability at each location.

The orientation of discontinuity planes and their intersections were compared to a full range of pit slope orientations at 10° intervals from 0° to 360°, in order to identify potential problems. Faces orientated 305° – 005° are the least favourable geometry, with potentially five to six joint intersections daylighting (leading to a number of geometrically feasible failure modes). Large scale faces with orientations in this range should be avoided if possible.

Faces with orientations of 005° – 305° are likely to be more stable, with one to three possible joint intersections. The likely pit dimensions are in this range. A graph of the number of joint intersections for different face orientations is shown on Figure 2.

Sensitivity analyses were also performed to determine the likely size of failures for different bench slope angles. The results are presented in Figure 3 and show that each 5° increase in slope angle more than doubles the weight of potentially falling rock.

Stability analyses of particular failure modes were undertaken using two approximately orthogonal orientations from the ore body, 060° and 165°, as an estimate of the pit dimensions. Detailed analyses of planar, wedge and toppling failures were modelled for both orientations. Pit slope angles of 38° and 70° were assumed in order to represent the overall slope angle and the individual benches. The results are presented in Appendix E.

The analyses suggest that small block failure, as wedges and toppling, are likely on bench slopes. The risk of large scale failures affecting the overall slope is much lower. The likelihood of these occurring will depend on the occurrence and orientation of joints near cut faces. The risk of such failures could be assessed with greater confidence from geotechnical assessments of benches and cut faces during mining.

3.2 Rockmass assessment – oxidised orebody

The estimated values of RMR for the upper 120m of the ore body range from 33 to 38, with an average of 35. Only 4% of the estimated values were greater than 40. The RMR values estimated from deeper cores (up to 260m maximum) ranged from 33 to 42 with an average of 37. These results suggest that the entire ore body to a depth of 120m can be classified as "poor quality rock".

The RMR values were estimated from photographs. While the results are believed to be representative, some values may have been different if the core could be inspected. Therefore, we suggest that the rockmass assessment and the resulting predictions are checked by periodic geotechnical inspection of the cut benches as mining proceeds.

The variation of RMR with depth is plotted in Figure 4. This appears to show a slight increase in rock quality over the depth range 80-120m, and another increase beyond 160m, possible towards "fair quality rock". The first increase may correspond to the limit of the weathered zone in the ore body. The second increase may be related to the boundary between the oxidised and sulphide zones of the ore body. These predictions can be refined by data collection during excavation.

3.3 Rockmass assessment – host rock

The RMR and Q values of the host rock are presented in Appendix C and are distributed as shown in Table 1. The RMR readings are plotted on a site plan in Figure 5.

Table 1 Host rock quality classification

Classification	RMR	Q
Very good	3 (6%)	-
Good	30 (64%)	1 (2%)
Fair	14 (30%)	32 (68%)
Poor	-	14 (30%)
Very poor	-	-
№ of measurements	47 (100%)	47 (100%)

These results are fairly consistent, although RMR scores are one class higher than the Q values. This relationship was also found in the main Gedebek pit, where RMR values correlated better with actual site conditions. It is also noted that while both methods were designed for tunnelling applications, RMR appears to have gained wider use in other geotechnical engineering applications, such as slopes and foundations.

These results were determined at the ground surface on rock exposures that have resisted weathering and erosion. This may introduce some bias and rock quality between the exposures may be lower due to rock type or faulting. Conversely, rock quality may increase with depth, due to reduced weathering effects. Continued geotechnical monitoring of the pit benches will allow the rockmass classification to be reviewed and revised.

Overall, a classification of “good rock” is considered to be appropriate for the host rock.

Guideline values of SMR were derived from the RMR by applying additional factors concerning the relative orientation of discontinuities to the slope face and the method of excavation. The result classifies slope faces in the host rock as “stable” to “partially stable”, with the risk of some toppling or wedge failures. This is in agreement with the analysis of discontinuity data, thereby increasing confidence levels.

3.4 Assumed physical properties

In order to utilise data from discontinuity measurements and rockmass classification schemes in slope stability assessments, information is also required on the physical properties of the rockmass. A combination of data from borehole core analyses and assumptions based on previous studies in Gedebek was used to derive the values presented in Table 2.

Table 2 Assumed rockmass properties

Property	Assumed value
Bulk unit weight (kN/m^3)	30
Cohesion (kPa)	0
Friction Angle (degrees)	35

3.5 Groundwater conditions

Groundwater was encountered in the RC exploration boreholes. Water levels in the ore body were measured at a depth of 20-30m. The general trend appears to follow the overlying topography, but with less pronounced gradients.

One spring was identified in the nearby valley; and the position corresponds to the extrapolation of groundwater levels identified in the RC boreholes. This stream feeds into a different catchment to the rest of the Gedebek mine and so it would be prudent to set up a monitoring point and obtain baseline water quality data, especially if sulphide ores will be encountered in the new pit.

The presence of a spring suggests that water is able to infiltrate into the rockmass and also flow through it at a sufficient rate to support the outflow. Most flow is likely to occur along discontinuities although there may be some porous storage in the weathered horizons. Flows into the pit are unlikely to be large and will reduce with time. For stability calculations, the joints were assumed to be damp.

3.6 Rock strength

Samples were obtained from four boreholes to ascertain the strength of both the country rock and oxidised ore. Seven point load tests were undertaken on four specimens from borehole GTDD-023. The value of IS(500) was determined, which can be used to estimate shear strength. Subsequently, seven unconfined compressive strength (UCS) tests were performed, with three specimens obtained from borehole GTDD-001 and four specimens taken from borehole GTDD-002 respectively. All testing certificates for the testing are included in Appendix F.

The testing was undertaken to give an understanding of the rockmass strength within the ore body and surrounding host rock. The results are presented in Table 3. The point load test results seem to underestimate the shear strength. This is possibly a result of bias in sample selection in order to satisfy the test requirements.

Table 3 Rock strength test results

BH	Sample	Depth, m	UCS estimated from point load index [20 x Is(50)]	Unconfined compressive strength, MPa
GTDD-023	01	8.7	11.8	35.4
GTDD-023	01	8.7	7.4	31.8
GTDD-023	02	13.5	8.4	28.6
GTDD-023	02	13.5	8.2	35.3
GTDD-023	03	55.6	12.4	32.9
GTDD-023	03	55.6	9.8	42.1
GTDD-023	04	74.6	6.0	25.8
GTDD-001	01	52.1		20.0
GTDD-001	02	69.6		10.0
GTDD-001	03	72.6		25.7
GTDD-002	01	20.9		27.2
GTDD-002	02	37.6		38.0
GTDD-002	03	43.1		3.8
GTDD-002	04	83.5		10.4

The UCS results are probably a better estimate of the strength of larger pieces of intact rock, whilst the point load strengths may be more representative of smaller pieces of rock, possibly affected by weathering or mineralisation.

The results do not show a clear trend in the variation with depth and there is a relatively similar strength throughout the rockmass. The results classify the rockmass as R1-R2, a weak to moderately weak rock. This agrees with the assumptions made in the rockmass analyses described previously.

Some samples produced low strength values, which are 30% or less of the typical values. These are probably a result of alteration of the rock in zones of mineralisation and weathering. The lowest value (3.8 Mpa) was determined on a sample that was taken from an apparent fault zone. The purpose was to specifically analyse rock from the vicinity of such a feature, which may have an effect on stability of cut faces.

The result suggests that the mineralisation and geological structure have an impact on intact rock strength as well as the overall rockmass characteristics. This would possible lead to an overestimate of stability based on the RMA. Therefore, it would be prudent to carry out some additional strength tests on samples taken during excavation, in conjunction with the periodic mapping of faces and rockmass assessment.

4. Parameters for pit slope design

4.1 Oxidised ore body

The oxidised ore rockmass is classified as "poor rock", with the possibility of fault zones, which could lead to stability issues during and after excavation. There is some uncertainty about the rockmass rating and potential failure mechanisms because it was not possible to make in-situ measurements of discontinuities. It would be prudent to assume that all face orientations will have the same properties. This prediction can be refined by the measurement of rockmass properties during excavation of the ore.

We anticipate that the oxidised ore will be fully excavated for processing and that long- term or permanent slopes will not be formed in this deposit. Therefore, benches and pit slopes in the ore body will be short-term features. Rock falls may occur on steep slopes behind benches, but these may not be serious unless they prevent access or create a danger to personnel.

If stability issues are encountered, particularly in the weathered zone of the ore body, it may be necessary to reduce these parameters. Geotechnical monitoring during excavation would allow interim values to be defined.

4.2 Host rock

We anticipate that the main pit faces will be formed in the host rock, after removal of the oxidised ore. On the basis of current data, it would be prudent to excavate each bench back to the host rock as soon as possible, rather than have too many benches open in the oxidised ore body.

The rockmass classification suggests that some toppling or wedge failures may occur in steeper slopes in the host rock behind benches. This is supported by the discontinuity analysis, which predicts that the risk of such failures is highest on slope faces with orientations in the range 305° – 005°. The prediction is confirmed by the site observation of failures in outcrops with these orientations.

The orientation of pit faces will largely be determined by the geometry of the ore body and pit access requirements. However, it would be beneficial to align the faces as close as possible to the most favourable orientations.

The height of the pit faces will vary due to the topography and this will also need to be considered in conjunction with the orientation of the faces.

On the basis of the rockmass assessment and discontinuity data, slope parameters that are suitable for pit design are summarised in Table 4. These parameters include the results of sensitivity analysis of potential wedge and toppling failures. We have assumed that every fifth bench would be widened to catch falling rock.

Table 4 Pit slope parameters

Overall slope angle	38°
Average bench angle	58°
Maximum local bench angle	70°
Bench height	10m
Normal bench width	6m
Catch bench width	10m

The key parameter is the overall slope of 38°. If the bench widths suggested above are used, this will require slopes between benches of 58°. This slope can be modified locally to accommodate different bench widths, up to a maximum of 70°. Any modifications will be subject to inspection and assessment.

The actual slope angle that is used in the host rock will be influenced by the geometry of the contact between the ore body and the host rock. If there is a sharp transition, steeper than the suggested parameters, some host rock will need to be excavated to form pit slopes with adequate stability.

If the transition is gradual and the ore body is defined as an economic boundary, the sub-economic ore will need to be cut back according to suitable parameters. As data were not available to model this scenario, these parameters will need to be confirmed by geotechnical survey when the ore body is exposed on benches. For planning purposes, the same slopes as in the host rock can be assumed, although they may need to be slightly less in practice.

4.3 Additional data collection and assessment

Care was taken to collect all available data for this study and the interpretation is considered to be representative of the ground conditions as they are currently understood.

Nevertheless, prior to excavations being opened there is an unavoidable chance of omission or bias in data collected from isolated data points. These issues are discussed in previous sections and are:

Lower quality host rock may occur in the soil-covered areas between exposures

The rock quality assessment of the oxidised ore was based mainly on photographs and should be confirmed by actual measurements

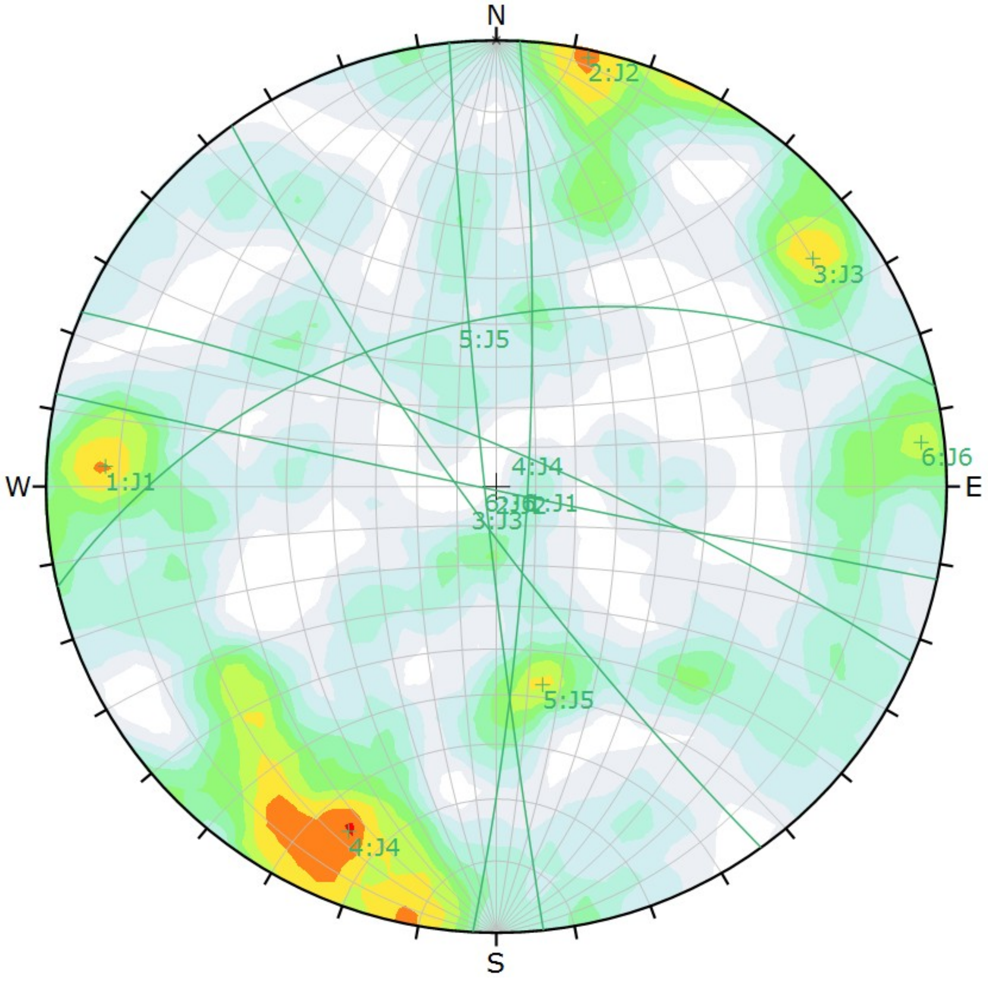
These factors may lead to some divergence between the assumptions made in this study and the actual ground conditions. Therefore, we recommend that the following data collection and assessment activities are incorporated in the development of the mine in order to allow the pit slope parameters to be reviewed. This would lead either to increased confidence in the parameters or the opportunity to make modifications in view of new data, thereby promoting safe operations.

Drilling of some inclined boreholes to intersect likely pit slope locations

Periodic geotechnical inspections of excavated benches in the ore body and host rock, with determination of rockmass assessment ratings

Strength tests on selected samples of rock which are identified during the periodic assessments and which might have an impact on stability

We also suggest that follow-up stability calculations are undertaken when the preliminary design for the Uğur Pit is prepared and the face orientations are known.



Color	Density Concentrations
⬜	0.00 - 0.40
⬜	0.40 - 0.80
🟦	0.80 - 1.20
🟦	1.20 - 1.60
🟩	1.60 - 2.00
🟩	2.00 - 2.40
🟩	2.40 - 2.80
🟨	2.80 - 3.20
🟧	3.20 - 3.60
🟥	3.60 - 4.00

Contour Data	Pole Vectors
Maximum Density	3.67%
Contour Distribution	Fisher
Counting Circle Size	1.0%

Plot Mode	Pole Vectors
Vector Count	184 (184 Entries)
Hemisphere	Lower
Projection	Equal Angle

Figure 1 Discontinuity measurements and main joint sets

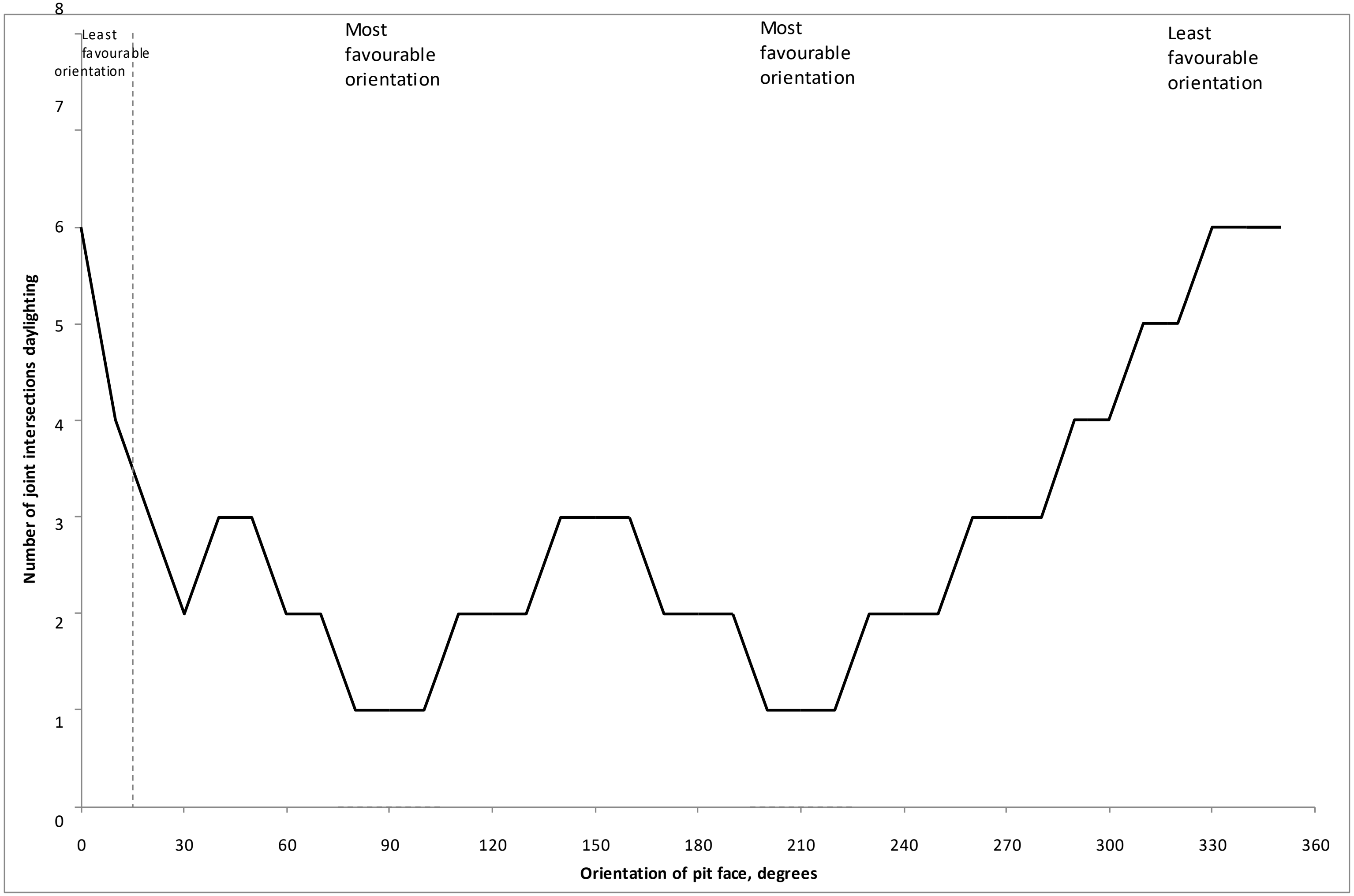


Figure 2 Sensitivity analysis of pit face orientations

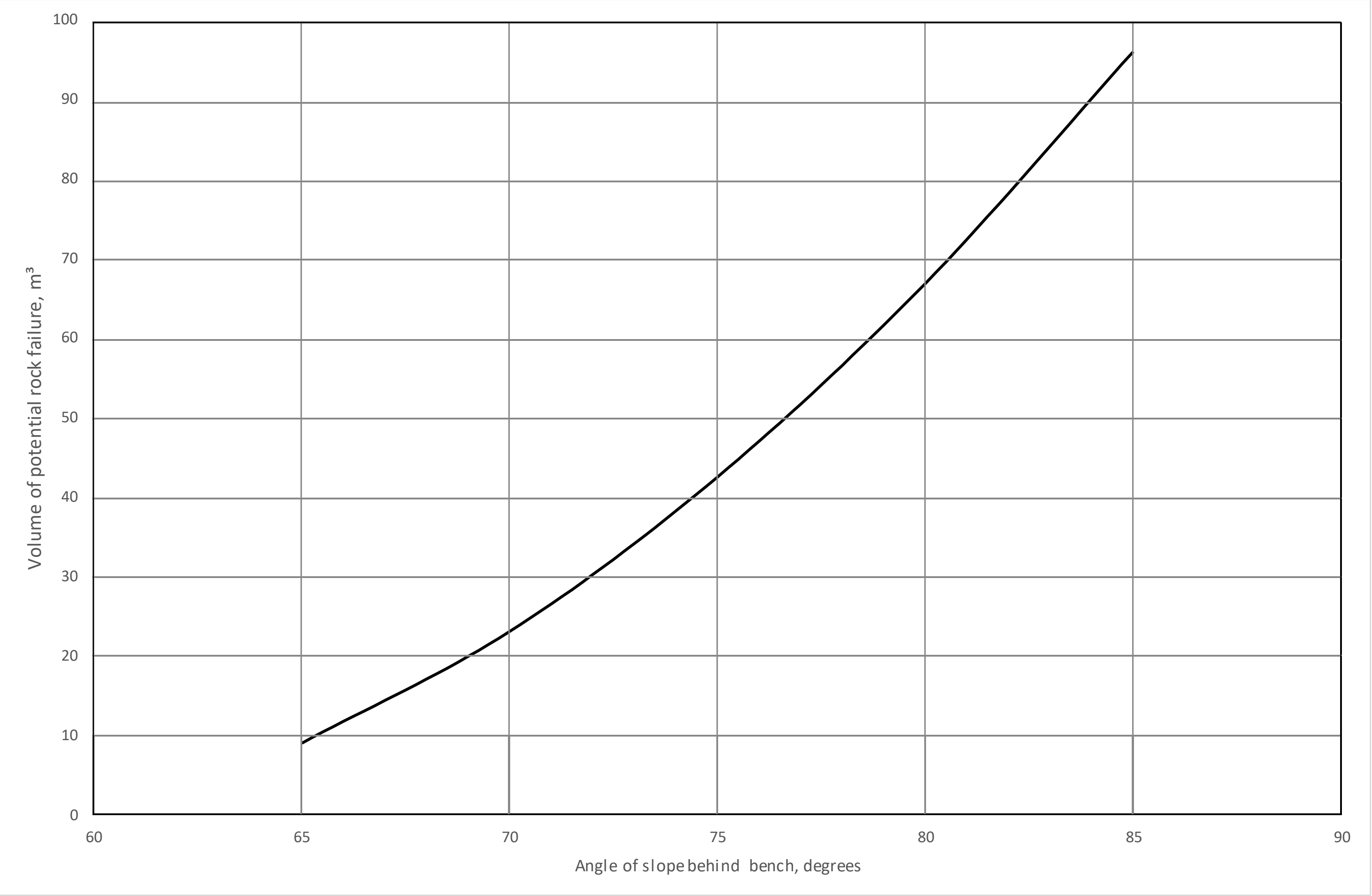


Figure 3 Sensitivity analysis of slope angle and rock fall size

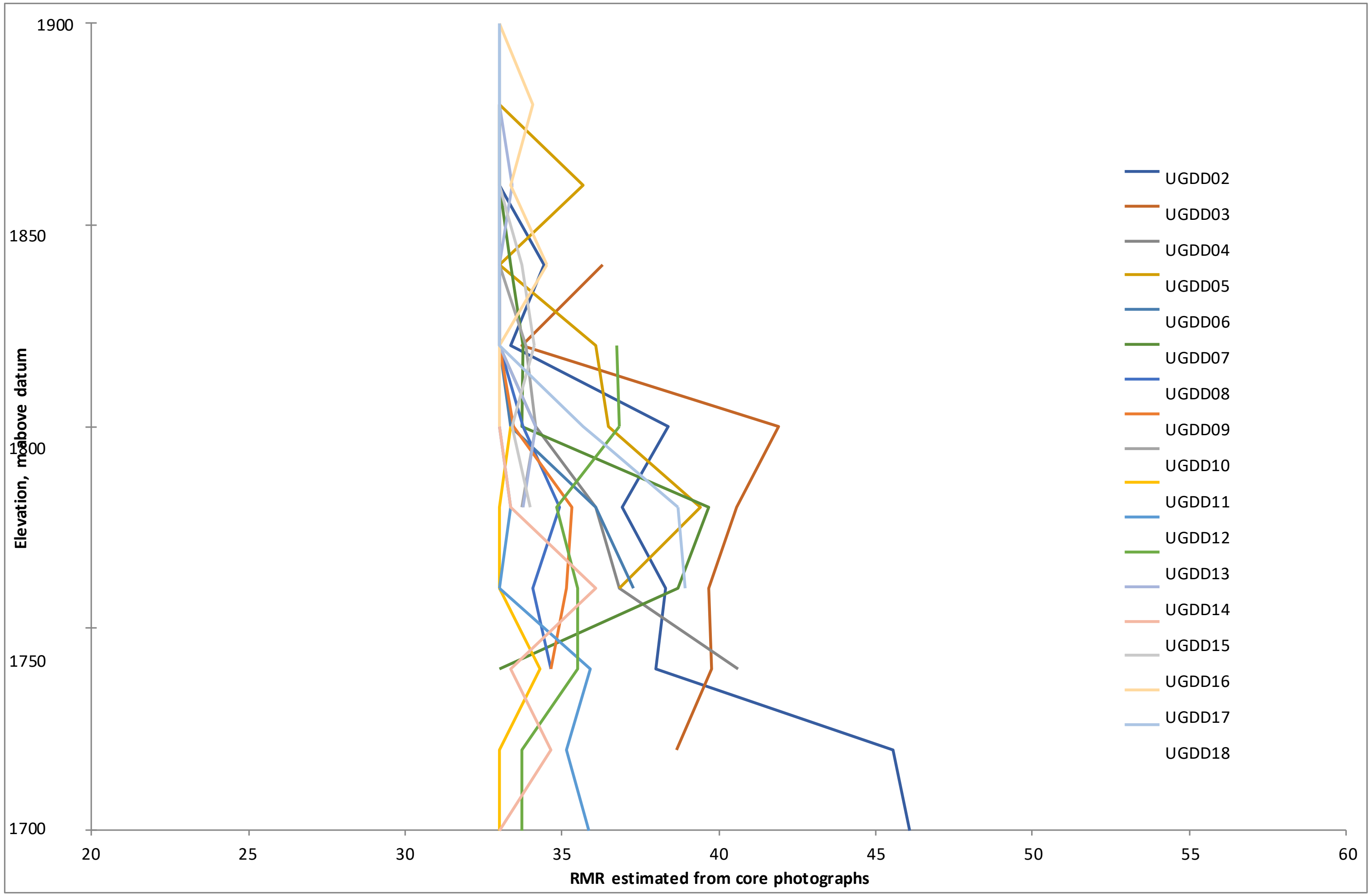


Figure 4 RMR values of oxidised ore

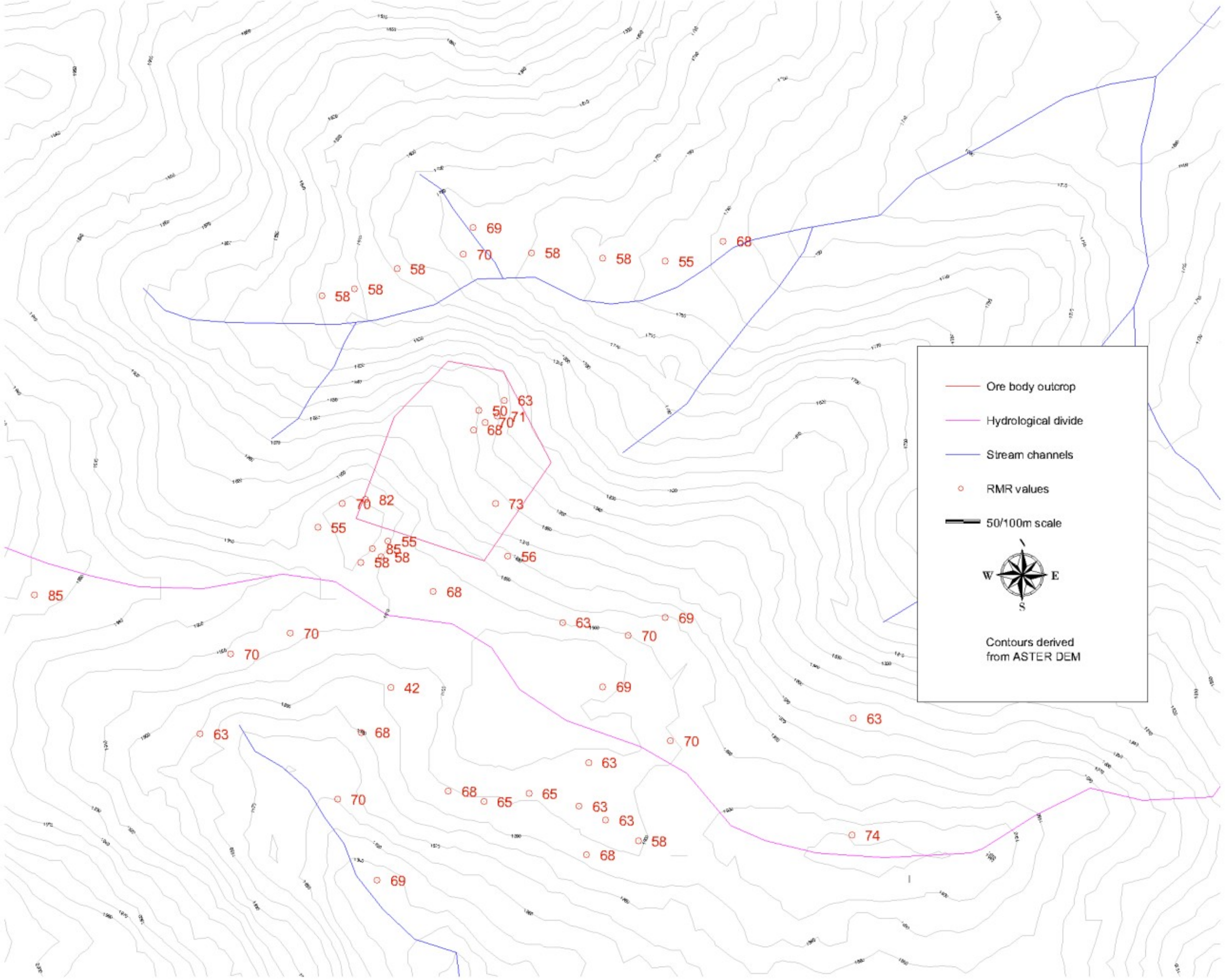


Figure 5 RMR values of host rock

Appendix A Borehole records used in this study

BH ID	X	Y	Z	Depth
UGDD01	4496961	565277.6	1862.998	285.5
UGDD02	4496923	565214.3	1887.850	401.3
UGDD03	4496996	565293.8	1857.236	138.5
UGDD04	4496901	565260.1	1875.067	123.5
UGDD05	4496828	565241.1	1895.180	139.0
UGDD06	4496877	565220.8	1890.401	133.35
UGDD07	4496920	565228.2	1882.951	130.0
UGDD08	4496955	565242.7	1873.978	124.0
UGDD09	4496931	565196.9	1891.884	126.2
UGDD10	4496889	565179.6	1901.681	122.2
UGDD11	4496925	565729.0	1820.729	151.5
UGDD12	4496853	565166.9	1908.013	125.0
UGDD13	4496922	565611.0	1827.364	151.0
UGDD14	4496937	565163.6	1905.233	132.0
UGDD15	4497040	565771.7	1803.832	250.0
UGDD16	4496903	565147.4	1912.362	134.0
UGDD17	4496869	565130.3	1919.683	110.0
UGDD18	4497005	565220.2	1883.000	125.4
UGDD19	4496998	565253.1	1873.342	117.0
UGDD20	4496876	565246.0	1884.148	125.0
UGDD21	4496970	565207.6	1885.889	104.5
UGDD22	4497031	565269.9	1867.203	136.0
UGDD23	4496844	565299.8	1880.514	117.0
UGDD24	4497044	565236.6	1869.208	134.0
UGDD25	4496888	565305.5	1870.758	120.0
UGRC01	4496820	565169.7	1908.800	33.0
UGRC02	4496868	565146.5	1913.200	34.0
UGRC03	4496889	565305.8	1871.100	34.0
UGRC04	4496959	565275.6	1863.300	27.0
UGRC05	4496928	565309.2	1858.000	13.0
UGRC06	4496923	565343.0	1850.300	32.0
UGRC07	4496970	565320.4	1847.300	34.0
UGRC08	4497022	565347.6	1833.500	31.0
UGRC09	4497000	565336.7	1837.600	22.0
UGRC10	4496930	565266.6	1867.200	34.0
UGRC11	4496998	565290.5	1857.700	34.0
UGRC12	4497018	565267.4	1869.100	34.0
UGRC13	4496976	565234.9	1877.700	34.0
UGRC14	4496922	565212.8	1888.000	34.0
UGRC15	4497010	565222.6	1882.500	34.0

BH ID	X	Y	Z	Depth
UGRC16	4496971	565184.4	1892.9	34.0
UGRC17	4496869	565204.8	1896.0	34.0
UGRC18	4496887	565244.7	1882.1	34.0
UGRC19	4496844	565090.1	1931.9	34.0
UGRC20	4496916	565163.8	1905.7	34.0
UGRC21	4497048	565240.9	1867.1	34.0
UGRC22	4497059	565284.2	1854.6	34.0
UGRC23	4496849	565295.5	1880.0	34.0
UGRC24	4496906	565106.9	1921.2	34.0
UGRC25	4496977	565140.8	1891.6	34.0
UGRC25A	4496978	565144.7	1891.6	34.0
UGRC26	4497025	565173.9	1875.2	34.0
UGRC27	4496840	565229.9	1895.3	34.0
UGRC28	4496610	565355.0	1921.1	34.0
UGRC29	4496612	565303.1	1915.4	34.0
UGRC30	4496657	565318.5	1915.5	34.0
UGRC31	4496749	565190.3	1906.1	34.0
UGRC32	4496795	565209.5	1904.0	34.0
UGRC33	4496777	565147.3	1914.3	34.0
UGRC34	4496745	565126.2	1909.8	34.0
UGRC35	4496754	565057.0	1915.3	34.0
UGRC36	4496794	565104.5	1923.8	34.0
UGRC37	4496794	565058.9	1923.9	34.0
UGRC38	4496748	565027.4	1918.4	34.0
UGRC39	4496779	564988.8	1921.7	34.0
UGRC40	4496828	565022.2	1922.5	34.0
UGRC41	4496871	565045.5	1922.0	34.0
UGRC42	4496913	565057.2	1913.6	34.0
UGRC43	4496852	564979.0	1912.4	34.0
UGRC44	4496809	564948.3	1919.6	34.0
UGRC45	4496842	564909.6	1912.6	34.0
UGRC46	4496798	564883.7	1925.9	34.0
UGRC47	4496775	564921.3	1926.5	34.0
UGRC48	4496759	564852.4	1929.8	34.0
UGRC49	4496783	564810.6	1932.9	34.0
UGRC50	4496824	564840.8	1921.1	34.0
UGRC51	4496811	564765.9	1933.8	34.0
UGRC52	4496772	564743.3	1942.9	34.0
UGRC53	4497046	565702.2	1785.4	34.0
UGRC54	4497051	565794.7	1803.5	34.0
UGRC55	4497020	565770.8	1807.9	34.0

Appendix B Photographs

1 Proposed Pit Location

2 Diamond drilling in the Ugur Deposit area

3 Surface outcrop of host rock, typical in the upper central area of proposed pit area

4 Example of host rock exposure in the stream valley to the east of the proposed pit area.

5 Toppling of host rock on the northern side of the stream, confirming the least favourable orientation for stability, as identified in the sensitivity analysis.

6 Spring identified on the ore deposit outcrop.

7 Exposure of weathered ore body in the southern edge of the proposed pit area.

8 Example of host rock outcrop in the centre of the proposed pit area.

9 Example of host rock outcrop on the western end and crest of the proposed pit area.

10 Example of host rock outcrop on the ridge to the west of the proposed pit area.

Appendix C Locations of host rock quality measurements

No.	COORDINATES			Joint No.	RANDOM	RQD	RMR	Q
	mE	mN	Z					
1	56514	449681	19	3	N	14	58	1.86
2	56513	449682	19	3	N	100	85	13.3
3	56515	449684	19	3	Y	15	55	1.5
4	56512	449690	19	3	Y	90	82	9
5	56508	449689	19	3	Y	50	70	5
6	56505	449686	19	3	Y	38	55	3.8
7	56511	449680	19	3	Y	32	58	3.2
8	56528	449703	18	3	Y	29	50	2.9
9	56532	449704	18	3	Y	42	63	4.2
10	56531	449702	18	3	Y	56	71	5.6
11	56529	449701	18	3	Y	54	70	5.4
12	56528	449700	18	3	Y	60	68	6
13	56531	449689	18	3	Y	70	73	7
14	56533	449681	18	3	Y	46	56	4.6
15	56522	449676	19	3	Y	60	68	6
16	56529	449645	18	3	Y	63	65	6.3
17	56524	449647	18	3	Y	53	68	5.3
18	56536	449647	18	4	Y	70	65	5.6
19	56463	449676	19	3	Y	100	85	10
20	56544	449651	19	3	Y	60	63	6
21	56543	449645	18	3	Y	55	63	5.5
22	56544	449638	18	3	Y	70	68	7
23	56547	449643	19	3	Y	67	63	6.7
24	56552	449640	19	3	Y	36	58	3.6
25	56564	449727	17	4	N	51	68	4.1
26	56556	449725	17	4	N	49	55	1.96
27	56546	449725	17	3	Y	58	58	2.9
28	56536	449726	17	4	N	56	58	2.24
29	56527	449729	17	4	N	95	69	3.8
30	56526	449726	17	3	Y	60	70	6
31	56583	449658	18	3	Y	43	63	4.3
32	56556	449654	19	4	Y	52	70	4.16
33	56546	449662	19	4	N	79	69	6.32
34	56556	449672	18	4	N	85	69	6.8
35	56550	449670	19	3	Y	71	70	7.1
36	56541	449672	19	3	Y	51	63	5.1
37	56583	449641	19	3	Y	79	74	7.9
38	56515	449662	18	4	N	45	42	1.8

No.	COORDINATES			Joint No.	RANDOM	RQD	RMR	Q
	mE	mN	Z					
39	565116	4496560	1870	3	Y	68	68	6.8
40	56508	449646	18	4	N	69	70	5.52
41	56513	449634	18	3	Y	78	69	7.8
42	56488	449655	18	4	N	54	63	4.32
43	56492	449667	19	3	Y	59	70	5.9
44	56501	449670	19	3	N	64	70	8.5
45	56516	449723	17	4	Y	61	58	2.44
46	56505	449719	17	3	Y	28	58	2.8
47	56510	449720	17	4	Y	39	58	1.56

Appendix D Summary of rockmass classifications

South African CSIR (RMR)

Classification	Range	No. in survey
Very good	80-100	3
Good	60-80	30
Fair	40-60	14
Poor	20-40	0
Very poor	0-20	0

Norwegian NGI (Q)

Classification	Range	No. in
Exceptionally good	400+	-
Extremely good	100-400	-
Very good	40-100	-
Good	10-40	2
Fair	4-10	31
Poor	1-4	14
Very poor	0.1-1	-
Extremely poor	0.01-0.1	-
Exceptionally poor	0.001-0.01	-

Appendix E Stereonet projections of selected stability scenarios

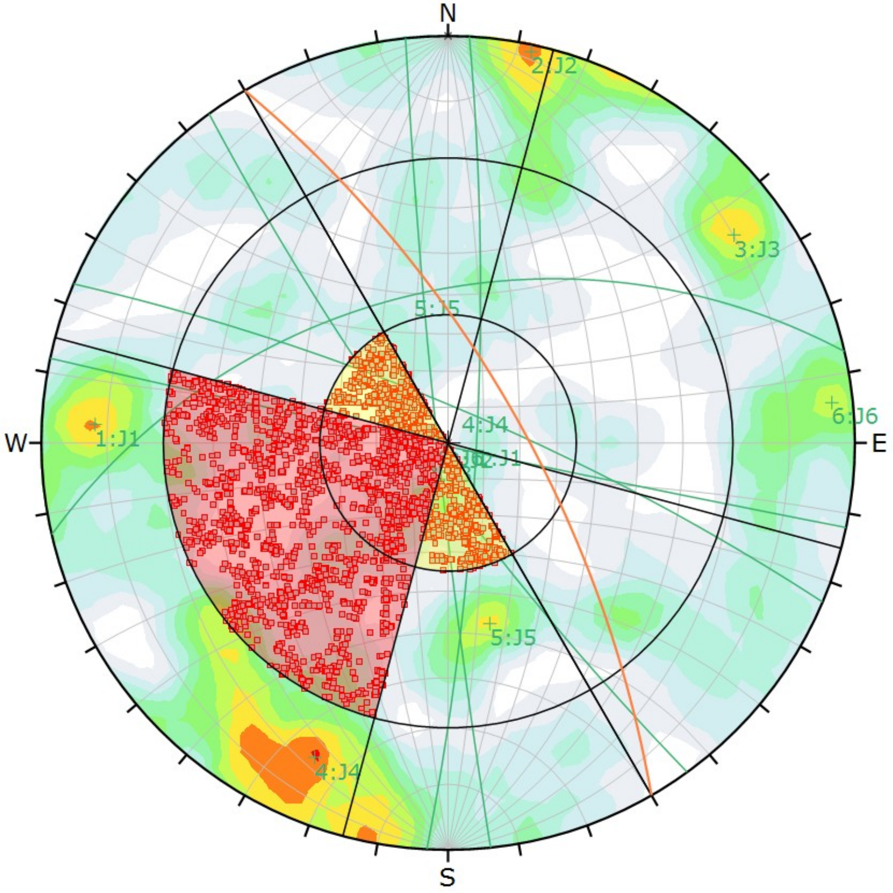
On the following pages:

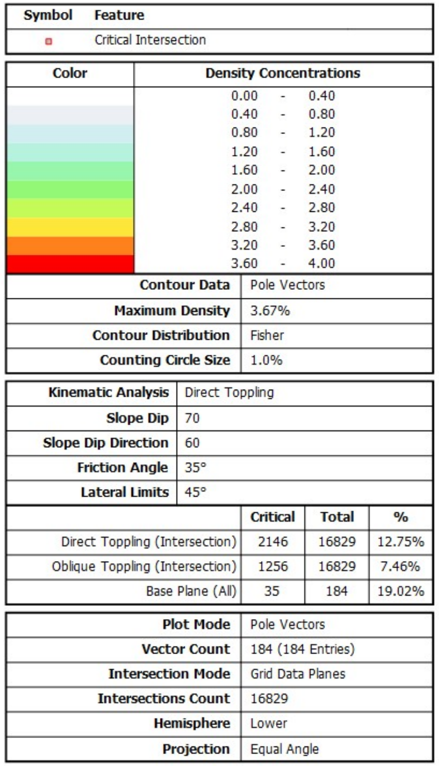
For pit face oriented at 060°

Toppling failure on bench
Wedge failure on bench
Toppling failure on overall slope
Wedge failure on overall slope

For pit face oriented at 165°

Toppling failure on bench
Wedge failure on bench
Toppling failure on overall slope
Wedge failure on overall slope





Symbol	Feature
▫	Critical Intersection

Color	Density Concentrations
⬜	0.00 - 0.40
⬜	0.40 - 0.80
🟦	0.80 - 1.20
🟦	1.20 - 1.60
🟩	1.60 - 2.00
🟩	2.00 - 2.40
🟩	2.40 - 2.80
🟨	2.80 - 3.20
🟧	3.20 - 3.60
🟥	3.60 - 4.00

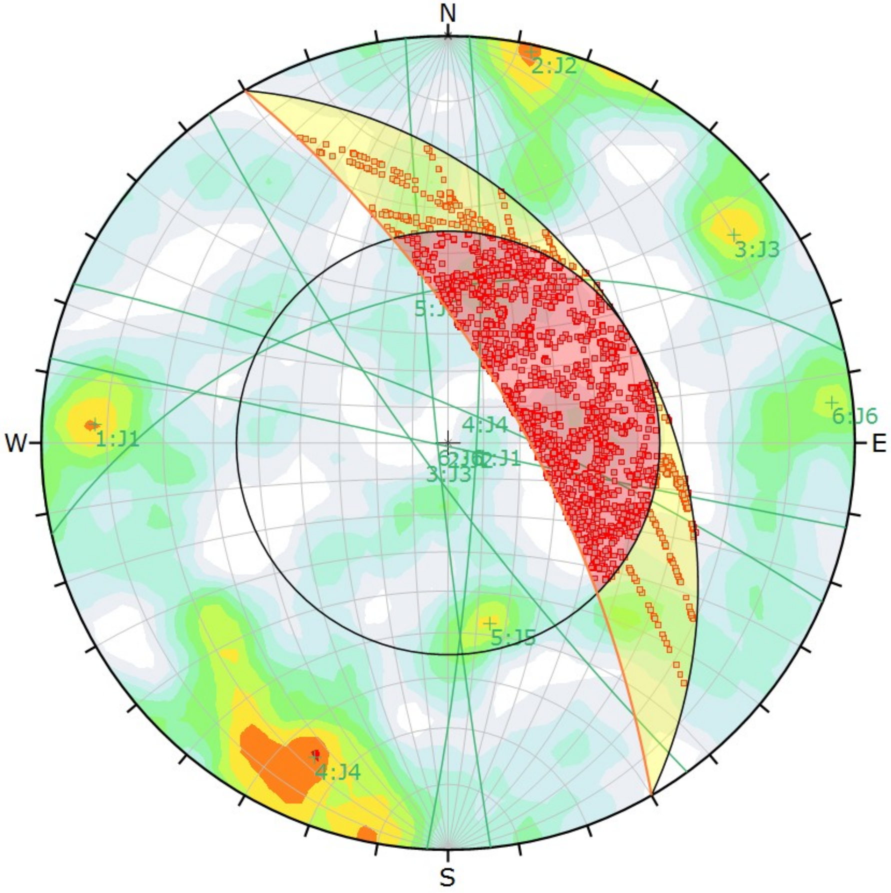
Contour Data	Pole Vectors
Maximum Density	3.67%
Contour Distribution	Fisher
Counting Circle Size	1.0%

Kinematic Analysis	Direct Toppling
Slope Dip	70
Slope Dip Direction	60
Friction Angle	35°
Lateral Limits	45°

	Critical	**Total**	**%**
Direct Toppling (Intersection)	2146	16829	12.75%
Oblique Toppling (Intersection)	1256	16829	7.46%
Base Plane (All)	35	184	19.02%

Plot Mode	Pole Vectors
Vector Count	184 (184 Entries)
Intersection Mode	Grid Data Planes
Intersections Count	16829
Hemisphere	Lower
Projection	Equal Angle

Pit face oriented at 060° - Toppling failure on bench



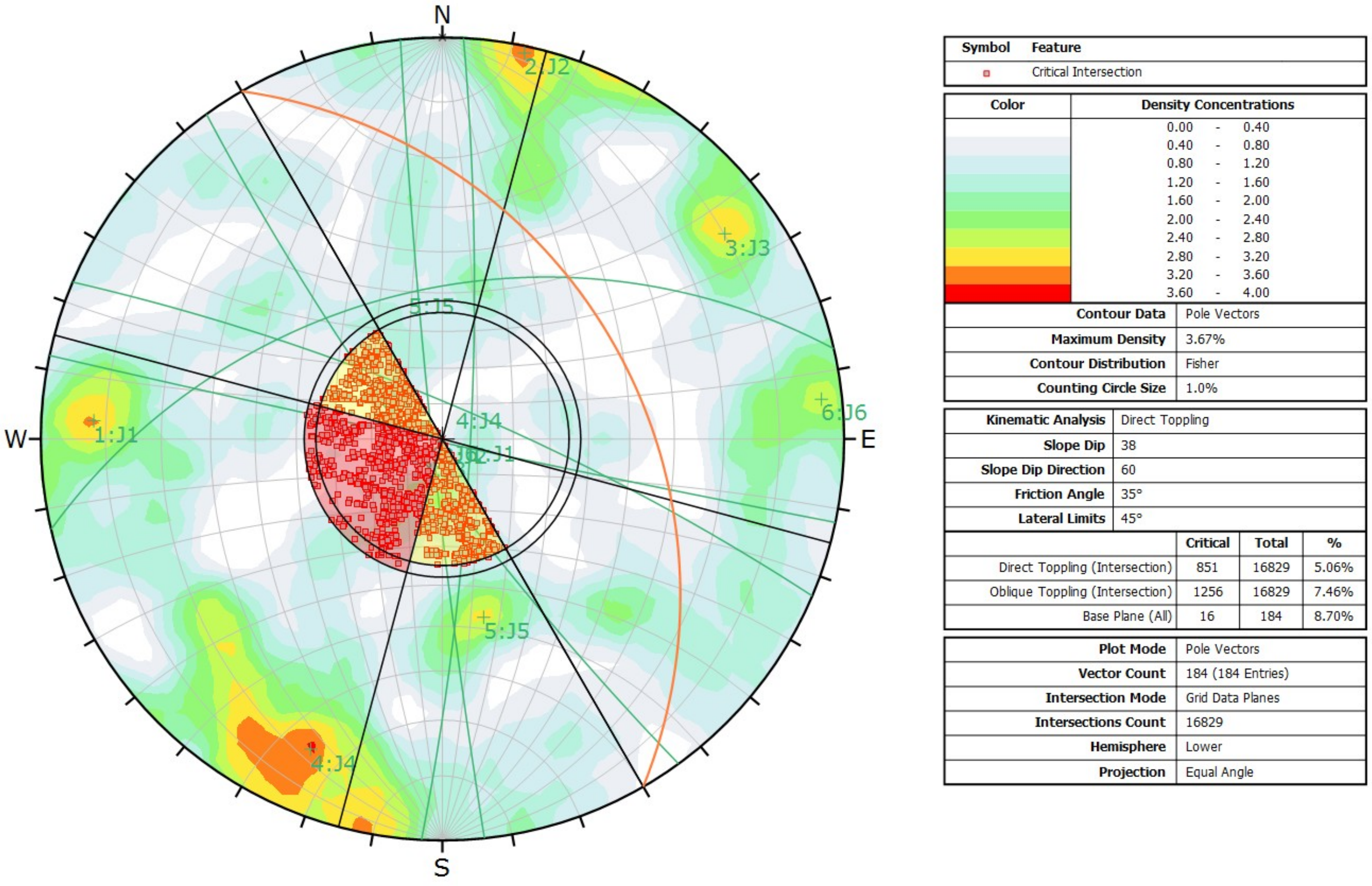
Symbol	Feature
▫	Critical Intersection

Color	Density Concentrations
⬜	0.00 - 0.40
⬜	0.40 - 0.80
⬜	0.80 - 1.20
🟩	1.20 - 1.60
🟩	1.60 - 2.00
🟩	2.00 - 2.40
🟩	2.40 - 2.80
🟨	2.80 - 3.20
🟧	3.20 - 3.60
🟥	3.60 - 4.00
Contour Data	Pole Vectors
Maximum Density	3.67%
Contour Distribution	Fisher
Counting Circle Size	1.0%

Kinematic Analysis	Wedge Sliding		
Slope Dip	70		
Slope Dip Direction	60		
Friction Angle	35°		
	Critical	**Total**	**%**
Wedge Sliding	2551	16829	15.16%

Plot Mode	Pole Vectors
Vector Count	184 (184 Entries)
Intersection Mode	Grid Data Planes
Intersections Count	16829
Hemisphere	Lower
Projection	Equal Angle

Pit face oriented at 060° - Wedge failure on bench



Symbol	Feature
▫	Critical Intersection

Color	Density Concentrations
⬜	0.00 - 0.40
⬜	0.40 - 0.80
🟦	0.80 - 1.20
🟦	1.20 - 1.60
🟩	1.60 - 2.00
🟩	2.00 - 2.40
🟩	2.40 - 2.80
🟨	2.80 - 3.20
🟧	3.20 - 3.60
🟥	3.60 - 4.00

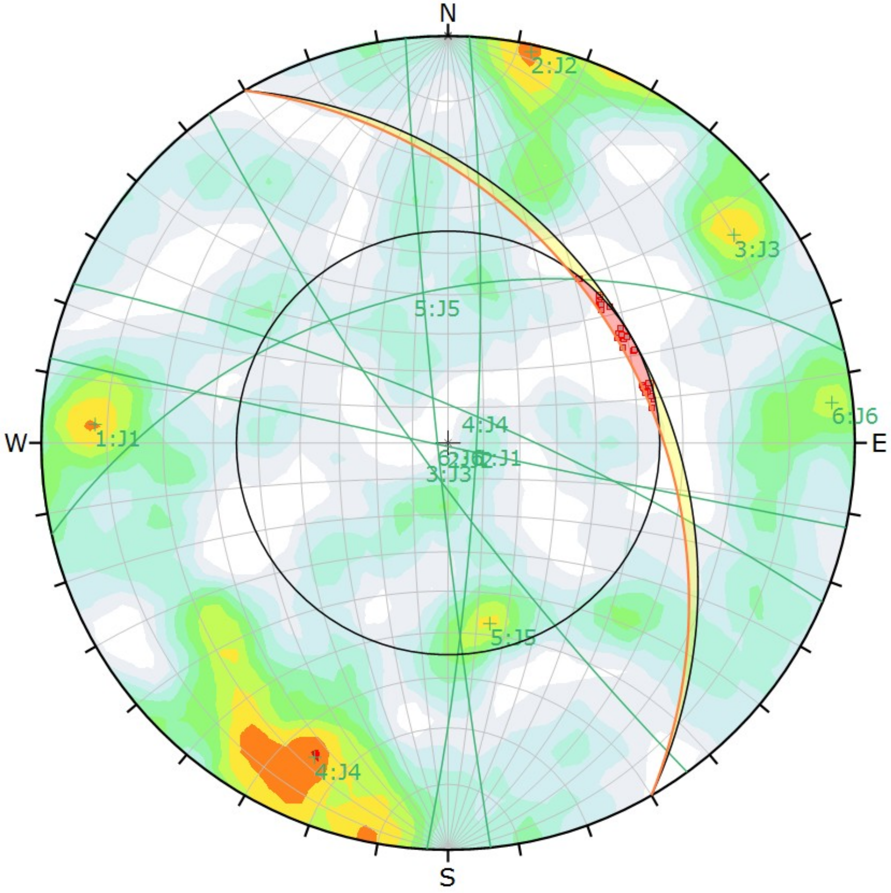
Contour Data	Pole Vectors
Maximum Density	3.67%
Contour Distribution	Fisher
Counting Circle Size	1.0%

Kinematic Analysis	Direct Toppling
Slope Dip	38
Slope Dip Direction	60
Friction Angle	35°
Lateral Limits	45°

	Critical	Total	%
Direct Toppling (Intersection)	851	16829	5.06%
Oblique Toppling (Intersection)	1256	16829	7.46%
Base Plane (All)	16	184	8.70%

Plot Mode	Pole Vectors
Vector Count	184 (184 Entries)
Intersection Mode	Grid Data Planes
Intersections Count	16829
Hemisphere	Lower
Projection	Equal Angle

Pit face oriented at 060° - Toppling failure on overall slope



Symbol	Feature
▫	Critical Intersection

Color	Density Concentrations
⬜	0.00 - 0.40
⬜	0.40 - 0.80
⬜	0.80 - 1.20
🟩	1.20 - 1.60
🟩	1.60 - 2.00
🟩	2.00 - 2.40
🟨	2.40 - 2.80
🟨	2.80 - 3.20
🟧	3.20 - 3.60
🟥	3.60 - 4.00

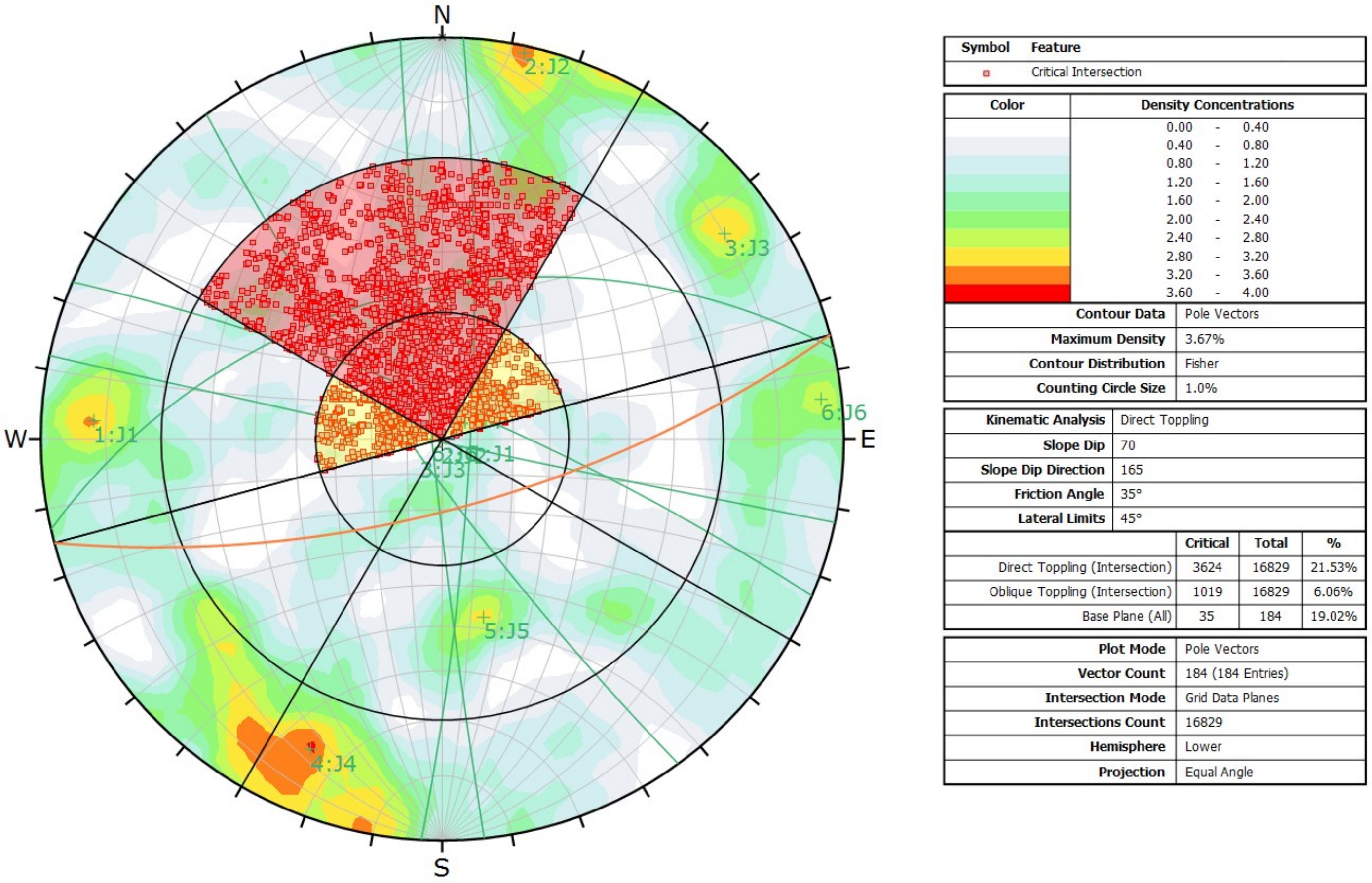
Contour Data	Pole Vectors
Maximum Density	3.67%
Contour Distribution	Fisher
Counting Circle Size	1.0%

Kinematic Analysis	Wedge Sliding
Slope Dip	38
Slope Dip Direction	60
Friction Angle	35°

	Critical	Total	%
Wedge Sliding	36	16829	0.21%

Plot Mode	Pole Vectors
Vector Count	184 (184 Entries)
Intersection Mode	Grid Data Planes
Intersections Count	16829
Hemisphere	Lower
Projection	Equal Angle

Pit face oriented at 060° - Wedge failure on overall slope



Symbol	Feature
▫	Critical Intersection

Color	Density Concentrations
⬜	0.00 - 0.40
⬜	0.40 - 0.80
🟦	0.80 - 1.20
🟦	1.20 - 1.60
🟩	1.60 - 2.00
🟩	2.00 - 2.40
🟩	2.40 - 2.80
🟨	2.80 - 3.20
🟧	3.20 - 3.60
🟥	3.60 - 4.00

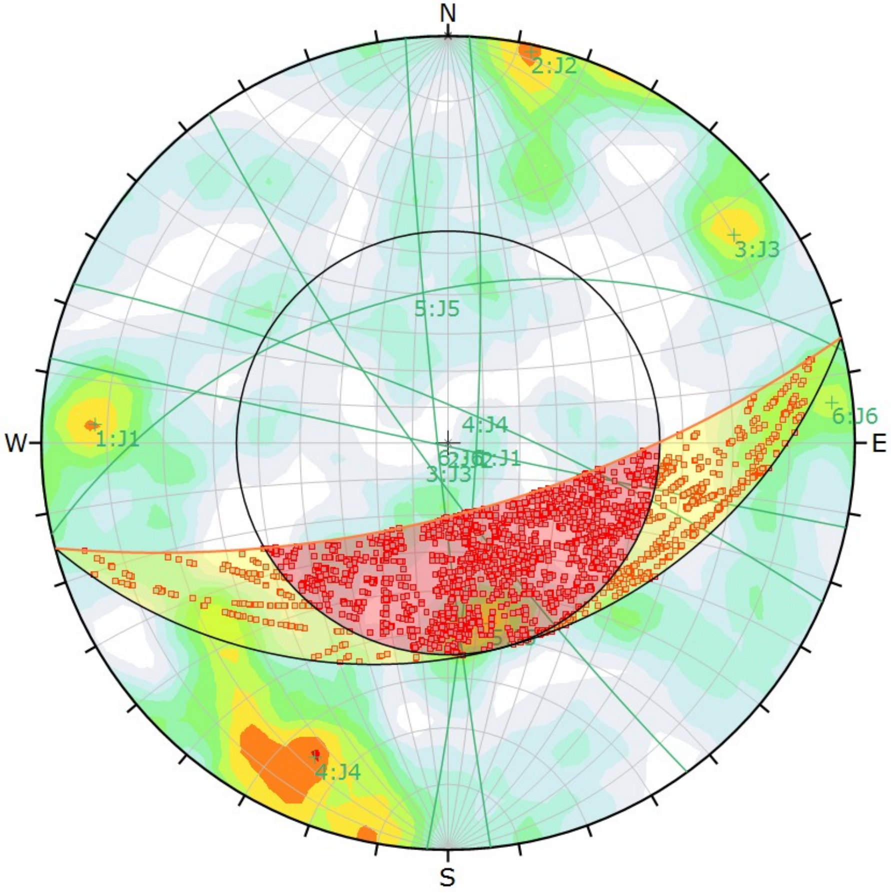
Contour Data	Pole Vectors
Maximum Density	3.67%
Contour Distribution	Fisher
Counting Circle Size	1.0%

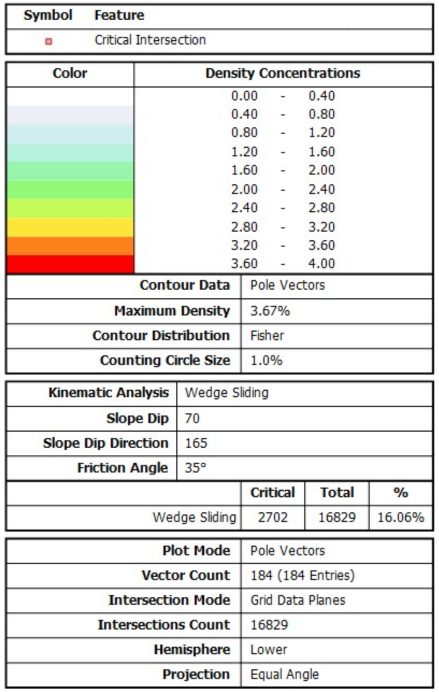
Kinematic Analysis	Direct Toppling
Slope Dip	70
Slope Dip Direction	165
Friction Angle	35°
Lateral Limits	45°

	Critical	Total	%
Direct Toppling (Intersection)	3624	16829	21.53%
Oblique Toppling (Intersection)	1019	16829	6.06%
Base Plane (All)	35	184	19.02%

Plot Mode	Pole Vectors
Vector Count	184 (184 Entries)
Intersection Mode	Grid Data Planes
Intersections Count	16829
Hemisphere	Lower
Projection	Equal Angle

Pit face oriented at 165° - Toppling failure on bench





Symbol	Feature
□	Critical Intersection

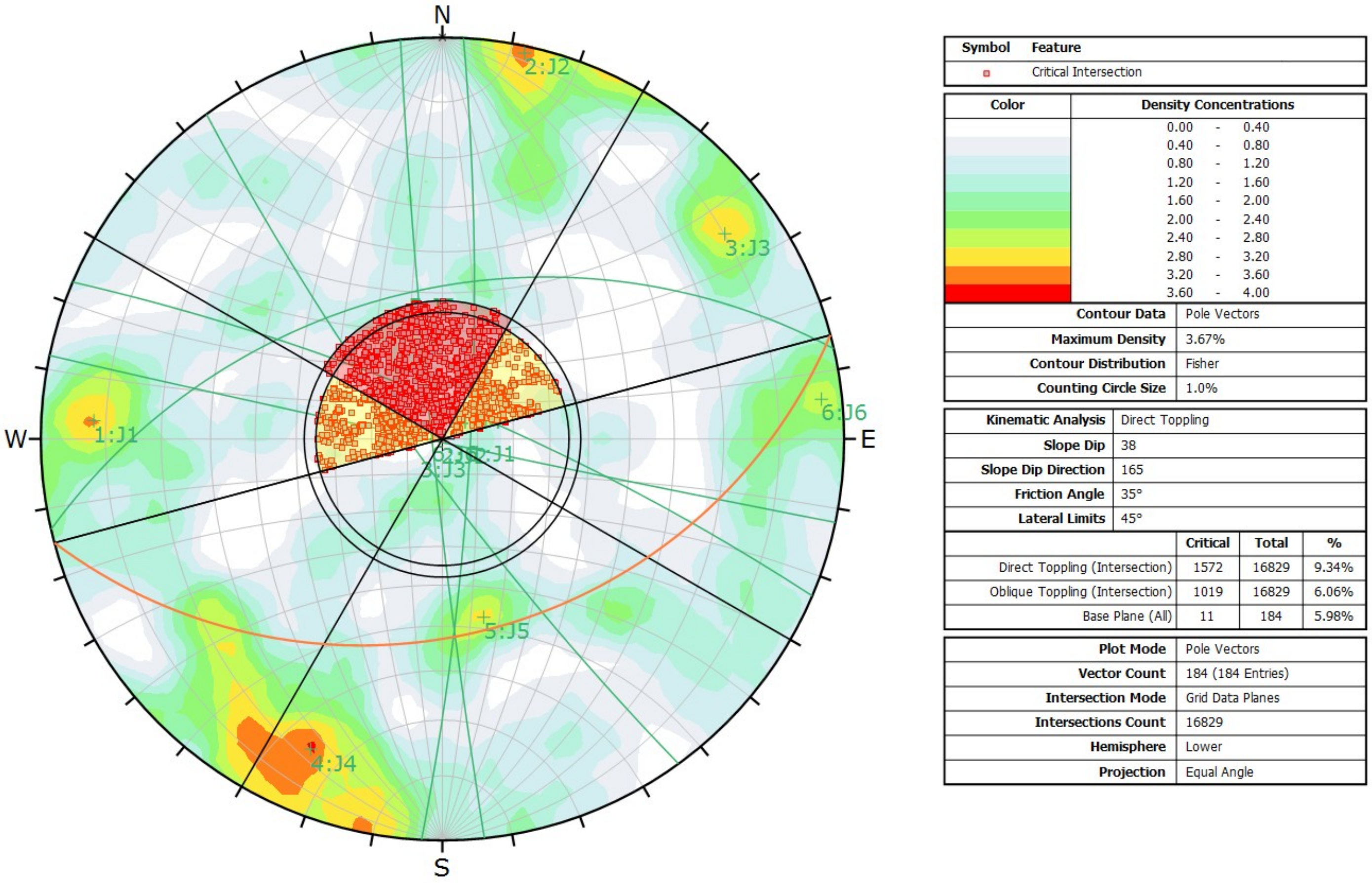
Color	Density Concentrations
⬜	0.00 - 0.40
⬜	0.40 - 0.80
🟦	0.80 - 1.20
🟩	1.20 - 1.60
🟩	1.60 - 2.00
🟩	2.00 - 2.40
🟨	2.40 - 2.80
🟨	2.80 - 3.20
🟧	3.20 - 3.60
🟥	3.60 - 4.00

Contour Data	Pole Vectors
Maximum Density	3.67%
Contour Distribution	Fisher
Counting Circle Size	1.0%

Kinematic Analysis	Wedge Sliding		
Slope Dip	70		
Slope Dip Direction	165		
Friction Angle	35°		
	Critical	**Total**	**%**
Wedge Sliding	2702	16829	16.06%

Plot Mode	Pole Vectors
Vector Count	184 (184 Entries)
Intersection Mode	Grid Data Planes
Intersections Count	16829
Hemisphere	Lower
Projection	Equal Angle

Pit face oriented at 165° - Wedge failure on bench



Symbol	Feature
▫	Critical Intersection

Color	Density Concentrations
⬜	0.00 - 0.40
⬜	0.40 - 0.80
🟦	0.80 - 1.20
🟦	1.20 - 1.60
🟩	1.60 - 2.00
🟩	2.00 - 2.40
🟩	2.40 - 2.80
🟨	2.80 - 3.20
🟧	3.20 - 3.60
🟥	3.60 - 4.00

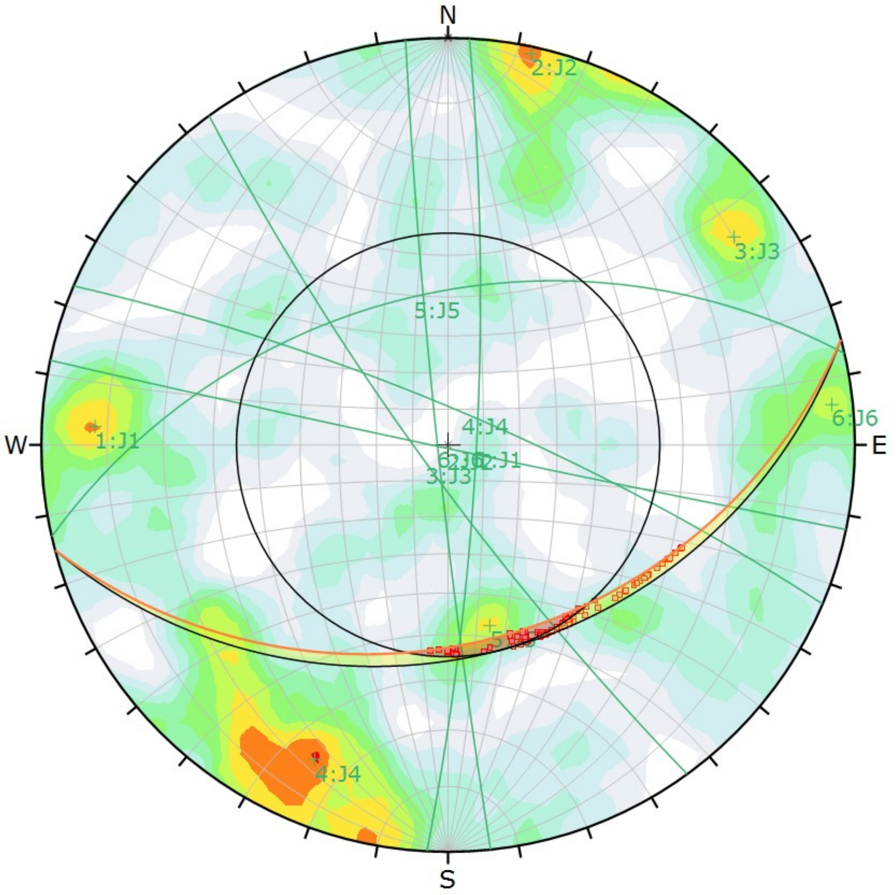
Contour Data	Pole Vectors
Maximum Density	3.67%
Contour Distribution	Fisher
Counting Circle Size	1.0%

Kinematic Analysis	Direct Toppling
Slope Dip	38
Slope Dip Direction	165
Friction Angle	35°
Lateral Limits	45°

	Critical	Total	%
Direct Toppling (Intersection)	1572	16829	9.34%
Oblique Toppling (Intersection)	1019	16829	6.06%
Base Plane (All)	11	184	5.98%

Plot Mode	Pole Vectors
Vector Count	184 (184 Entries)
Intersection Mode	Grid Data Planes
Intersections Count	16829
Hemisphere	Lower
Projection	Equal Angle

Pit face oriented at 165° - Toppling failure on overall slope



Symbol	Feature
▫	Critical Intersection

Color	Density Concentrations
⬜	0.00 - 0.40
⬜	0.40 - 0.80
🟦	0.80 - 1.20
🟦	1.20 - 1.60
🟩	1.60 - 2.00
🟩	2.00 - 2.40
🟩	2.40 - 2.80
🟨	2.80 - 3.20
🟧	3.20 - 3.60
🟥	3.60 - 4.00

Contour Data	Pole Vectors
Maximum Density	3.67%
Contour Distribution	Fisher
Counting Circle Size	1.0%

Kinematic Analysis	Wedge Sliding		
Slope Dip	38		
Slope Dip Direction	165		
Friction Angle	35°		
	Critical	**Total**	**%**
Wedge Sliding	137	16829	0.81%

Plot Mode	Pole Vectors
Vector Count	184 (184 Entries)
Intersection Mode	Grid Data Planes
Intersections Count	16829
Hemisphere	Lower
Projection	Equal Angle

Pit face oriented at 165° - Wedge failure on overall slope

Appendix F Point load test and UCS test certificates

On the following pages:

Company name: AT-GEOTECH
Company phone/fax: +994 3421948
Company email: office@at-geotech.com
Company web page: www.AT-GEOTECH.com

Project name: Ugur Mining
Test Date: 17-Mar-17
Test name: Point Load Strength Index of Rock
Test standard: ASTM D 5731

Note:

1. The client delivered four (4) rock samples to our laboratory for Point Load testing on March 17, 2017.

2. Insufficient length of each sample didn't allow us to perform 10 tests per sample for presenting average figure in accordance with the standard test procedure.

3. As much as possible tests were conducted from 4 samples.

Test No	BH	Sample Depth, m bgl	Length, L, mm	Dia./Depth, D, mm	Width, W, mm	Type: Diametral/ Axial, D or A	Perpendicular or Paralel, ⊥ or ‖	Failure load, P, kN	Equivalent core diameter square, D_e^2, mm^2	Correction factor, F	**Corrected Point Load Strength Index, $I_{s(50)}$, MPa**	Picture number before the test	Picture number after the test	Water content, %
1	2	8.70	200	86	-	D	‖	2124.12	7,396.00	1.28	**0.37**	2301	2302	2.3%
2	2	8.70	200	60	86	A	⊥	3138.13	6,569.92	1.24	**0.59**	2303	2304	2.3%
3	2	13.5	250	86	-	D	‖	2396.75	7,396.00	1.28	**0.41**	2305	2306	1.2%
4	2	13.5	250	68	86	A	⊥	2453.62	7,445.90	1.28	**0.42**	2307	2308	1.2%
5	2	55.6	230	86	-	D	‖	2843.93	7,396.00	1.28	**0.49**	2309	2310	0.6%
6	2	55.6	230	53	86	A	⊥	2983.18	5,803.43	1.21	**0.62**	2311	2312	0.6%
7	2	74.6	200	86	-	D	‖	1743.62	7,396.00	1.28	**0.30**	N/A	N/A	1.0%

Point Load Photos

UGDD-23

Point Load Photos

UGDD-23

Point Load Photos

UGDD-23

AT GEOLAB

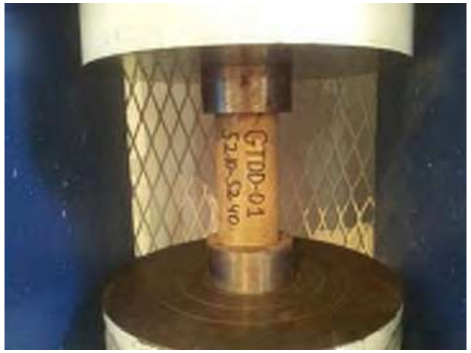
www.at-geolab.com
www.at-geotech.com
+994 55 909 08 58

Project name: 296-AIMC UGUR
Test Date: 11-May-17
Test name: Unconfined Compressive Strength of Intact Rock Core
Test standard: ASTM D 2938

Note:

1. The client delivered eight (8) rock samples to our laboratory for UCS testing on May 10, 2017.
2. Seven (7) samples were prepared and tested of eight (8). But one (1) sample was naturally cracked and can't be cut the required sample size for testing. (The relevant photos are included).

Test No	BH/ Location	Sample Depth, m bgl	Diameter, D, mm	Height, H, mm	Dimensional Conformance (H/D)	Rate of Loading, mm / min	Cross Sectional Area, A, mm^2	Uniaxial Compressive Strength, σuc, Mpa	Bulk density, kg/m^3	Photo number before the test	Photo number after the test	Water content, %	Sample Description
1	GTDD-01	52.10-52.40	63.0	129.5	2.1	0.75	3117.25	**20.02**	2,516	113451	114120	2.52%	Moderately strong, Secondary quartzites
2	GTDD-01	69.60-69.85	63.0	132.5	2.1	0.75	3117.25	**10.03**	2,717	114845	115438	1.08%	Moderately weak, Secondary quartzites
3	GTDD-01	72.65-73.00	63.0	131.8	2.1	0.75	3117.25	**25.71**	2,771	120731	121406	0.58%	Moderately strong, Secondary quartzites
4	GTDD-02	20.90-21.30	63.0	131.0	2.1	0.75	3117.25	**27.21**	2,891	122145	122759	1.97%	Moderately strong, Secondary quartzites
5	GTDD-02	37.65-37.80	63.0	130.8	2.1	0.75	3117.25	**38.01**	2,897	123313	123805	2.29%	Moderately strong, Secondary quartzites
6	GTDD-02	43.15-43.35	63.0	131.8	2.1	0.75	3117.25	**3.81**	2,307	124456	125106	8.93%	Weak, Secondary quartzites
7	GTDD-02	83.50-83.95	63.0	130.8	2.1	0.75	3117.25	**10.42**	2,516	130053	130631	4.12%	Moderately weak, Secondary quartzites

<table>
<tr><td colspan="2">Test № 1 GTDD-01_52.10-52.40m</td><td colspan="2">Test № 2 GTDD-01_69.60-69.85m</td></tr>
<tr><td></td><td></td><td></td><td></td></tr>
<tr><td>(Before test) _113451.JPG</td><td>(After test)_114120.JPG</td><td>(Before test) _114845.JPG</td><td>(After test)_115438.JPG</td></tr>
<tr><td colspan="2">Test № 3 GTDD-01_72.65-73.00m</td><td colspan="2">Test № 4 GTDD-02_20.90-21.30m</td></tr>
<tr><td></td><td></td><td></td><td></td></tr>
<tr><td>(Before test) _120731.JPG</td><td>(After test)_121406.JPG</td><td>(Before test) _122145.JPG</td><td>(After test)_122759.JPG</td></tr>
</table>

Test № 5 GTDD-02_37.65-37.80m		Test № 6 GTDD-02_43.15-43.35m	
			
(Before test) _123313.JPG	(After test)_123805.JPG	(Before test) _124456.JPG	(After test)_125106.JPG
Test № 7 GTDD-02_83.50-83.95m		Test № 8 GTDD-01_40.50-40.65m	
			N/A
(Before test) _130053JPG	(After test)_130631.JPG	(Cracked sample, can’t be tested) _125156JPG	(After test)_.JPG

DATAMINE

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