OreWin
Independent Mining Consultants

SSR MINING INC.

Çöpler Project

Çöpler District Master Plan 2020

November 2020

Job No. 19010

Bernard Peters

Technical Director – Mining

OreWin Pty Ltd

CONSENT of QUALIFIED PERSON

TO: SSR Mining Inc. (SSR Mining)

I, Bernard Peters, am an author of the technical report entitled: 'Çöpler District Master Plan 2020', with an effective date of 27 November 2020 (the "**Technical Report**").

I hereby consent to the public filing of the Technical Report, and to the use of extracts from, or a summary of, the Technical Report in SSR Mining's press release dated 30 November 2020 (the "**Disclosure Document**").

I confirm that I have read the Disclosure Document and that the Disclosure Document fairly and accurately represents the information contained in the sections of the Technical Report for which I am responsible.

Dated: 30 November 2020

/s/Bernard Peters

Bernard Peters
Technical Director – Mining
OreWin Pty Ltd

Sharron Sylvester

Technical Director – Geology

OreWin Pty Ltd

CONSENT of QUALIFIED PERSON

TO: SSR Mining Inc. (SSR Mining)

I, Sharron Sylvester, am an author of the technical report entitled: 'Çöpler District Master Plan 2020', with an effective date of 27 November 2020 (the "**Technical Report**").

I hereby consent to the public filing of the Technical Report, and to the use of extracts from, or a summary of, the Technical Report in SSR Mining's press release dated 30 November 2020 (the "**Disclosure Document**").

I confirm that I have read the Disclosure Document and that the Disclosure Document fairly and accurately represents the information contained in the sections of the Technical Report for which I am responsible.

Dated: 30 November 2020

/s/Sharron Sylvester

Sharron Sylvester

Technical Director – Geology

OreWin Pty Ltd

CERTIFICATE of AUTHOR

This Certificate of Author has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects Part 8.1.

a) Name, Address, Occupation:

Bernard Peters
OreWin Pty Ltd, 140 South Terrace, Adelaide South Australia 5000, Australia
Mining Engineer, employed as Technical Director – Mining.

b) Title and Date of Technical Report:

Çöpler District Master Plan 2020 (CDMP20) dated 30 November 2020 (the "Technical Report")

c) Qualifications:

I graduated from the University of Melbourne, Australia with a Bachelor of Engineering in Mining Engineering in 1986. I am a Fellow of the Australasian Institute of Mining and Metallurgy (no. 201743). I have practised my profession continuously since 1986 and have experience in mining operations and consulting at and for projects in various countries including Australia, Bolivia, Canada, Democratic Republic of the Congo, Indonesia, Kazakstan, Kyrgyzstan, Mongolia, Peru, Philippines, Russia, South Africa, Turkey, and USA. I have managed and been responsible for studies with multidisciplinary teams of professionals in the mining industry including geology, mining engineering, processing and infrastructure. As a result of my qualifications and experience, I am a Qualified Person as defined in National Instrument 43-101.

d) Site Inspection:

I visited the project 13–17 May 2019, 15–21 September 2019, 14–18 October 2019, 18–21 November 2019, and 27 February to 4 March 2020. The site visits included briefings from geology and exploration, mine, processing, environmental, permitting, and corporate personnel, site inspections of current and future areas for mining and plant and infrastructure, and discussions with other consultants. In addition, several visits to SSR Mining's head office in Denver Colorado were undertaken during the same timeframe for the purpose of project-related meetings.

e) Responsibilities:

I am responsible for the overall preparation of the CDMP20 and, the Mineral Reserve estimates, Sections 1 to 4; Sections 5 and 6; Section 13; Sections 15 to 27.

f) Independence:

I am independent of SSR Mining Inc. in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement:

I have been working on various aspects of the studies of the project since 2018.

h) Compliance with NI 43-101:

I have read National Instrument 43-101 and Form 43-101FI and the Technical Report has been prepared in compliance with same.

i) Disclosure:

As of 30 November 2020, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: 30 November 2020

/s/Bernard Peters

Bernard Peters
Technical Director – Mining
OreWin Pty Ltd

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CERTIFICATE of AUTHOR

This Certificate of Author has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects 30 June 2011, Part 8.1.

a) **Name, Address, Occupation:**
Sharron Sylvester
OreWin Pty Ltd, 140 South Terrace, Adelaide South Australia, 5000, Australia
Geologist, employed as Technical Director – Geology.

b) **Title and Date of Technical Report:**
Çöpler District Master Plan 2020 (CDMP20) dated 30 November 2020 (the "Technical Report")

c) **Qualifications:**
I graduated from the University of Adelaide, Australia with a Bachelor of Science Degree in Geology in 1989. I am a Member of the Australian Institute of Geoscientists (2512) and a Registered Professional Geologist (10125). I have practised my profession continuously since 1989 and have experience in the assessment, modelling, and resource estimation of mineral deposits, underground and open-pit mine geology, project management, and due diligence and valuation. I have extensive experience in a variety of geological terrains and commodities, including copper, gold, base metals, ferrous metals, and construction materials. As a result of my qualifications and experience, I am a Qualified Person as defined in National Instrument 43-101.

d) **Site Inspection:**
I visited the project 13–17 May 2019, 15–21 September 2019, 14–18 October 2019, 18–21 November 2019, and 27 February to 4 March 2020. The site visits included briefings from geology and exploration, mine, processing, environmental, permitting and corporate personnel, site inspections of current and future areas for mining and plant and infrastructure, and discussions with other consultants. In addition, several visits to SSR Mining's head office in Denver, Colorado were undertaken during the same timeframe for the purpose of project-related meetings. Visits to analytical laboratories were planned to be undertaken but not completed due to global travel restrictions related to Covid-19.

e) **Responsibilities:**
I am responsible for the preparation of the Mineral Resources, Sections 1 to 4; Section 7 to 12; Section 14; and Sections 25 to 27.

f) **Independence:**
I am independent of SSR Mining Inc. in accordance with the application of Section 1.5 of National Instrument 43-101.

g) **Prior Involvement:**
I have been working on the Mineral Resources of the project since 2019.

h) **Compliance with NI 43-101:**
I have read National Instrument 43-101 and Form 43-101FI and the Technical Report has been prepared in compliance with same.

i) **Disclosure:**
As of 30 November 2020, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: 30 November 2020

/s/Sharron Sylvester

Sharron Sylvester

Technical Director – Geology

OreWin Pty Ltd

IMPORTANT NOTICE

This notice is an integral component of the Çöpler District Master Plan 2020 Technical Report (CDMP20) and should be read in its entirety and must accompany every copy made of the Technical Report. The CDMP20 has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects.

The CDMP20 has been prepared for SSR Mining Inc. (SSR Mining) by OreWin Pty Ltd (OreWin). The CDMP20 is based on information and data supplied to OreWin by SSR Mining and other parties and where necessary OreWin has assumed that the supplied data and information are accurate and complete.

The CDMP20 includes a Preliminary Economic Assessment (PEA) and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the PEA represent forward-looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the PEA. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report under each relevant section.

The conclusions and estimates stated in the CDMP20 are to the accuracy stated in the CDMP20 only and rely on assumptions stated in the CDMP20. The results of further work may indicate that the conclusions, estimates and assumptions in the CDMP20 need to be revised or reviewed.

OreWin has used its experience and industry expertise to produce the estimates and approximations in the CDMP20. Where OreWin has made those estimates and approximations it does not warrant the accuracy of those amounts and it should also be noted that all estimates and approximations contained in the CDMP20 will be prone to fluctuations with time and changing industry circumstances.

The CDMP20 should be construed in light of the methods, procedures, and techniques used to prepare the CDMP20. Sections or parts of the CDMP20 should be read in context of the entire CDMP20 and should not be removed from their original context.

The CDMP20 is intended to be used by SSR Mining, subject to the terms and conditions of its contract with OreWin. Recognising that SSR Mining has legal and regulatory obligations, OreWin has consented to the filing of the CDMP20 with Canadian Securities Administrators and its System for Electronic Document Analysis and Retrieval (SEDAR). Except for the purposes legislated under provincial securities laws, any other use of this report by any third party is at that party's sole risk.

OreWin

Title Page

Project Name:	Çöpler Project
Title:	Çöpler District Master Plan 2020
Location:	Erzincan Province Turkey
Effective Date of Technical Report:	27 November 2020
Effective Date of Mineral Resources:	27 November 2020
Effective Date of Drilling Database:	27 November 2020
Effective Date of Mineral Reserves:	27 November 2020

Qualified Persons:

- Bernard Peters, BEng (Mining), FAusIMM (201743), employed by OreWin Pty Ltd as Technical Director – Mining, was responsible for the overall preparation of the CDMP20 and, the Mineral Reserve estimates, Sections 1 to 4; Sections 5 and 6; Section 13; Sections 15 to 27.
- Sharron Sylvester, BSc (Geol), RPGeo AIG (10125), employed by OreWin Pty Ltd as Technical Director – Geology, was responsible for the preparation of the Mineral Resources, Sections 1 to 4; Section 7 to 12; Section 14; Sections 25 to 27.

Signature Page

Project Name:	Çöpler Project
Title:	Çöpler District Master Plan 2020
Location:	Erzincan Province Turkey
Date of Signing:	30 November 2020
Effective Date of Technical Report:	27 November 2020

/s/ Bernard Peters

Bernard Peters, Technical Director – Mining, OreWin Pty Ltd, BEng (Mining), FAusIMM (201743)

/s/ Sharron Sylvester

Sharron Sylvester, Technical Director – Geology, OreWin Pty Ltd, BSc (Geol), RPGeo AIG (10125)

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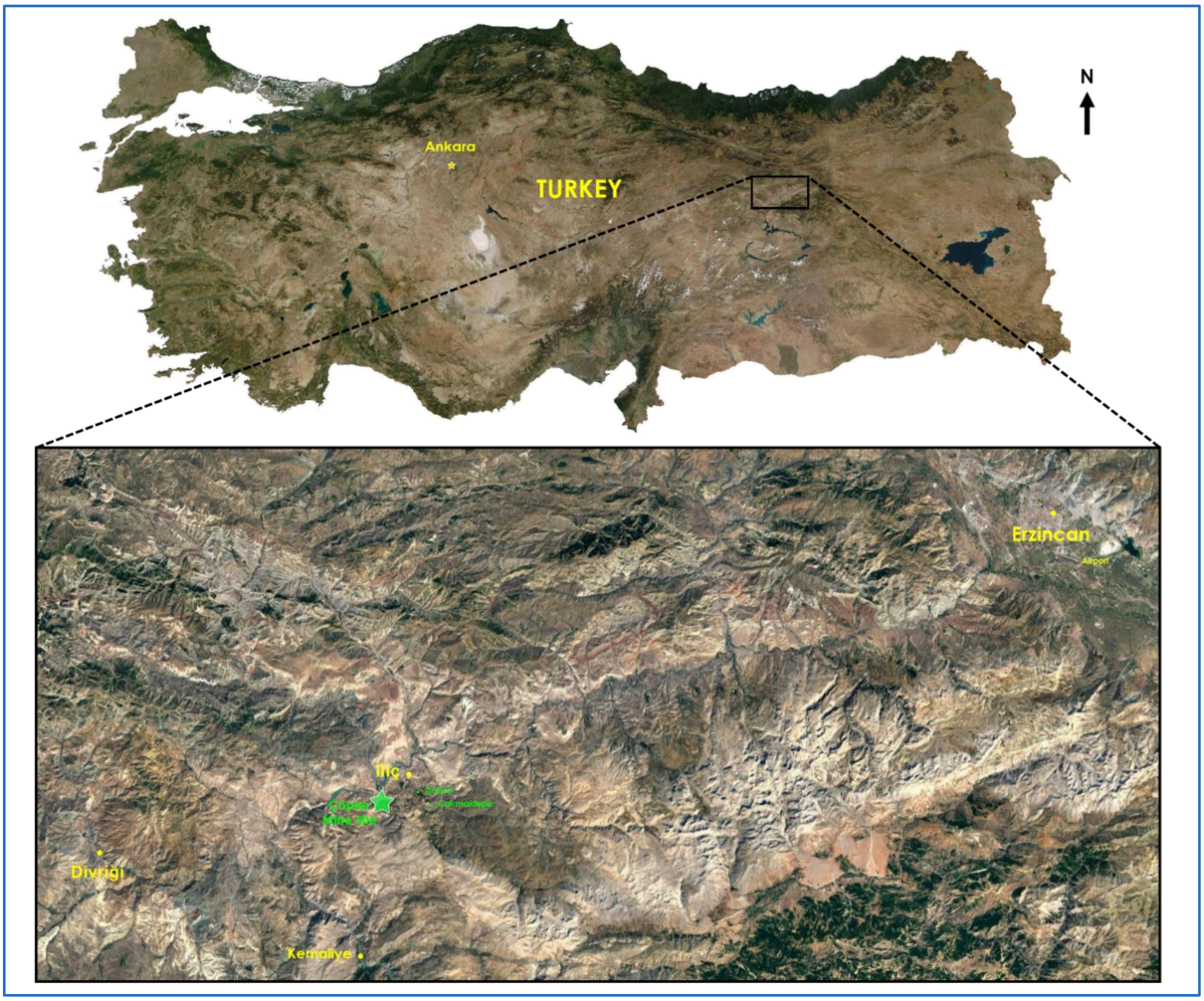
1 SUMMARY

1.1 Introduction

The Çöpler District Master Plan 2020 Technical Report (CDMP20) is an independent Technical Report prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for SSR Mining Inc. (SSR Mining), on the Çöpler project (the project), located in Turkey. The project comprises a number of mining licences covering Mineral Resources for the Çöpler, Çakmaktepe, Ardich, and Bayramdere deposits, Mineral Reserves on the Çöpler and Çakmaktepe open pit mines, oxide and sulfide processing facilities and supporting infrastructure. The Çöpler project is located in east-central Turkey, 120 km west of the city of Erzincan, in Erzincan Province, 40 km east of the iron-mining city of Divriği (one-hour drive), and 550 km east of Turkey's capital city, Ankara (Figure 1.1). The nearest urban centre, İliç, (approximate population 3,800), is located approximately 6 km north-east of the Çöpler mine.

Figure 1.1 Project Location Map



SSR Mining, 2020

SSR Mining is a gold mining company with four producing assets, located in the USA, Turkey, Canada, and Argentina, with development and exploration assets in the USA, Turkey, Mexico, Peru, and Canada. SSR Mining is listed on the NASDAQ Capital Markets (NASDAQ:SSRM), the Toronto Stock Exchange (TSX:SSRM), and on the Australian Stock Exchange (ASX:SSR).

The Çöpler project is owned and operated by Anagold Madencilik Sanayi ve Ticaret Anonim Şirketi (Anagold). SSR Mining controls 80% of the shares of Anagold, Lidya Madencilik Sanayi ve Ticaret A.Ş. (Lidya), controls 18.5%, and a bank wholly-owned by Çalık Holdings A.Ş., holds the remaining 1.5%. Exploration tenures surrounding the project area and mining at Çakmaktepe are subject to joint venture agreements between SSR Mining and Lidya that have varying interest proportions. SSR Mining controls 50% of the shares of Kartaltepe Madencilik Sanayi ve Ticaret Anonim Şirketi (Kartaltepe) and 50% of Tunçpinar Madencilik Sanayi ve Ticaret Anonim Şirketi (Tunçpinar). The other 50% of both is controlled by Lidya. More than 96% of the Mineral Resources are located on the Anagold owned 80% ground, with the remainder of the mineralisation within the 50%/50% ownership boundary.

SSR Mining has undertaken further study work that has been used to prepare the CDMP20 since the previous Technical Report on the project was issued in 2016 (2016 Technical Report). The 2016 Technical Report described the Çöpler Sulfide Expansion Project for construction and operation of a sulfide plant, which commenced commissioning in Q4'18.

The key features of the CDMP20 are:

- Updated Mineral Resources on the Çöpler, Çakmaktepe, and Ardich deposits.
- Updated Mineral Reserves on the Çöpler and Çakmaktepe deposits.
- The incorporation of a supplemental flotation circuit in the existing sulfide plant.
- Preliminary Economic Assessment (PEA) including the Ardich Mineral Resources.

The CDMP20 summarises the current SSR Mining development strategy for the Çöpler project. The CDMP20 includes analysis for two production scenarios: the Reserve Case and the PEA Case.

The Mineral Reserves are supported by feasibility study level work on the currently operated pits at the Çöpler and Çakmaktepe deposits and the oxide heap leach facility and sulfide plant in the Reserve Case. The processing analysis in the Reserve Case includes incorporation of a flotation circuit into the existing sulfide plant to upgrade sulfide sulfur (SS) to fully utilise grinding and pressure oxidation (POX) autoclave capacity. The flotation circuit is currently under design and preliminary construction works are underway pending final permitting, which is expected in late-2020.

The CDMP20 also includes a PEA on an expanded Çöpler project that includes the new predominantly-oxide Ardich deposit. The PEA Case analyses inclusion of production from Ardich and reflects the increased capital costs and infrastructure required to incorporate the new deposit. The PEA Case is a whole-of-project analysis that represents a significant change from the Reserve Case economics analysis results and production.

The Ardich deposit is a newly identified deposit that is separate to the other deposits on the property. Drilling is continuing at the Ardich deposit and it is expected that the drilling will further define the Mineral Resource. The progression of Ardich requires development of a new open pit that is approximately 6 km east of the current Çöpler pit and 1 km north of the Çakmaktepe pits. A plan showing facility location and the boundaries of the Reserve Case and the PEA Case is shown in Figure 1.2.

The Inferred Mineral Resource from Ardich is included in the PEA Case. The Ardich oxide Mineral Resources in the PEA Case represent a 226% increase in production tonnage at a grade that is 38% higher than the Çöpler oxide heap leach processing grades. The PEA Case includes assumptions for separate capital, infrastructure, and permitting that will be required to develop the Ardich Mineral Resources. The Ardich Mineral Resource were not included in the previous Technical Report and so the PEA represents a significant change in information.

The PEA Case is described in Section 24 of the CDMP20. The PEA Case is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

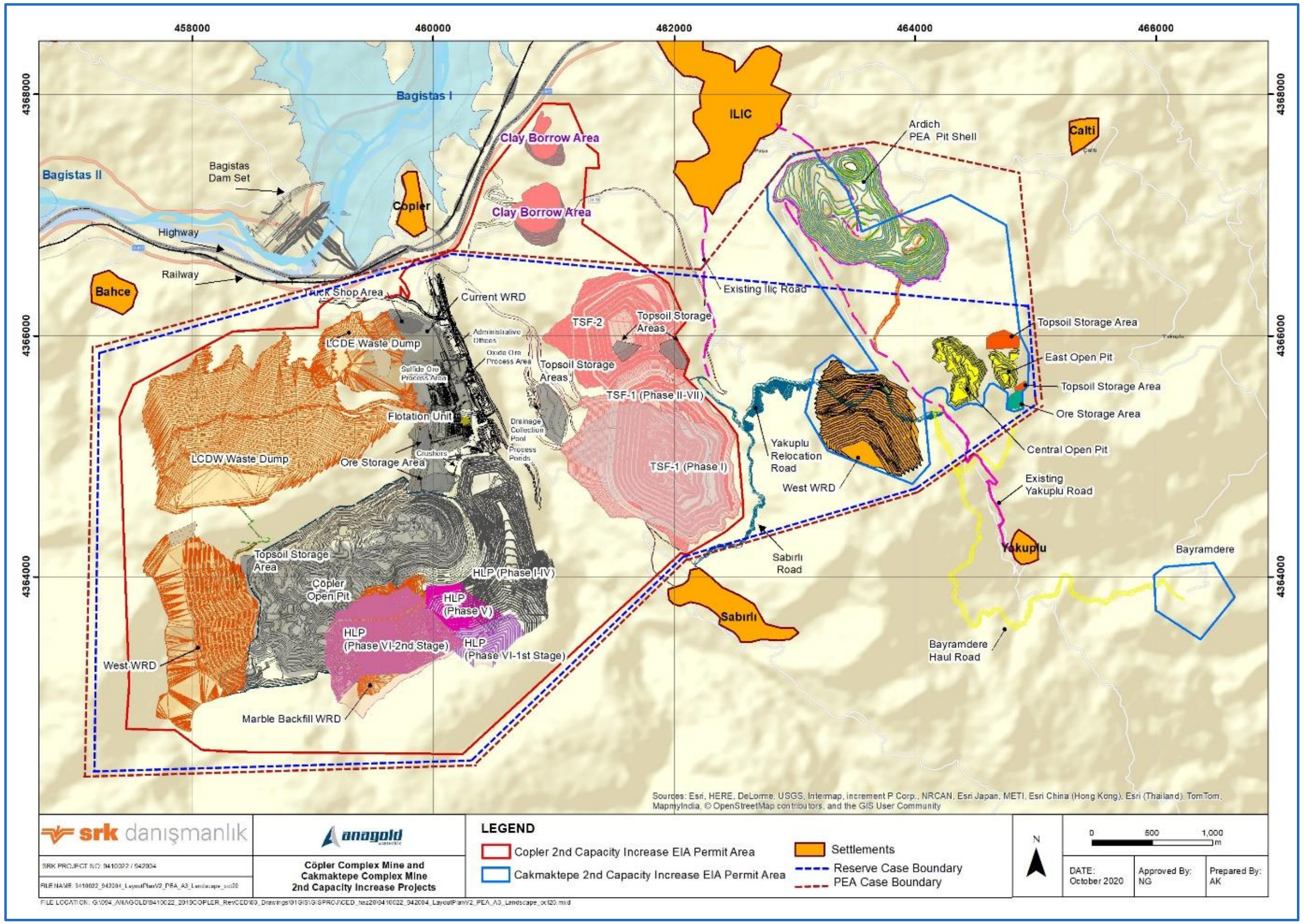
The key production and economic analysis from the CDMP20 are shown in Table 1.1.

The economic analysis uses long-term metal price assumptions of $1,585/oz gold, $20.25/oz silver, and $3.05/lb copper. These prices are based on a review of consensus price forecasts from financial institutions and similar studies recently published.

The Reserve Case production includes 7.7 Mt at 1.22 g/t Au oxide ore processed by heap leaching and 51.1 Mt at 2.24 g/t Au processed in the sulfide plant. Total gold production is 3.6 Moz. All mining is completed by 2032, oxide heap leach stacking is completed in 2031, while sulfide processing will continue from stockpiles until 2041. The Reserve Case shows an after-tax NPV at a 5% discount rate of $1.73 billion. The operation is cash positive in each year of the mine plan, therefore an IRR is not reported. The Reserve Case average all-in sustaining cost (AISC) is $945/oz gold.

The PEA Case production is 79.1 Mt at 2.13 g/t Au. The gold production in the PEA Case is 4.6 Moz. The increase in total production in the PEA Case is due to the addition of 20.3 Mt at 2.18 g/t Au from Ardich Mineral Resources. Like the Reserve Case, all mining is completed by 2032 in the PEA Case, oxide heap leach stacking is completed in 2031, while sulfide processing continues from stockpiles until 2042. The PEA Case shows an after-tax NPV at a 5% discount rate of $2.16 billion and the average AISC is $893/oz gold. The PEA Case is cash positive in each year of the mine plan.

Figure 1.2 CDMP20 Reserve Case and PEA Case Boundaries



Anagold, 2020

Table 1.1 CDMP20 Results Summary

Item	Unit	Reserve Case	PEA Case
Oxide Processed			
Heap Leach Quantity	kt	7,668	25,008
Au Feed Grade	g/t	1.22	1.69
Sulfide Processed			
Quantity Milled	kt	51,084	54,073
Au Feed Grade	g/t	2.24	2.33
Total Gold Produced			
Oxide – Gold	koz	256	956
Sulfide – Gold	koz	3,334	3,691
Total – Gold	koz	3,591	4,646
Oxide – Gold Recovery	%	73	68
Sulfide – Gold Recovery	%	91	91
5-Year Annual Average			
Average Gold Produced	kozpa	266	306
Free Cash Flow	$Mpa	224	249
Production Costs	$/oz gold	682	701
All-in Sustaining Costs	$/oz gold	865	886
Key Financial Results			
Production Costs	$/oz gold	748	726
All-in Sustaining Costs	$/oz gold	945	893
Site Operating Costs	$/t treated	47.09	42.87
After-Tax $NPV_{5\%}$	$M	1,733	2,164
Mine Life	years	21	22

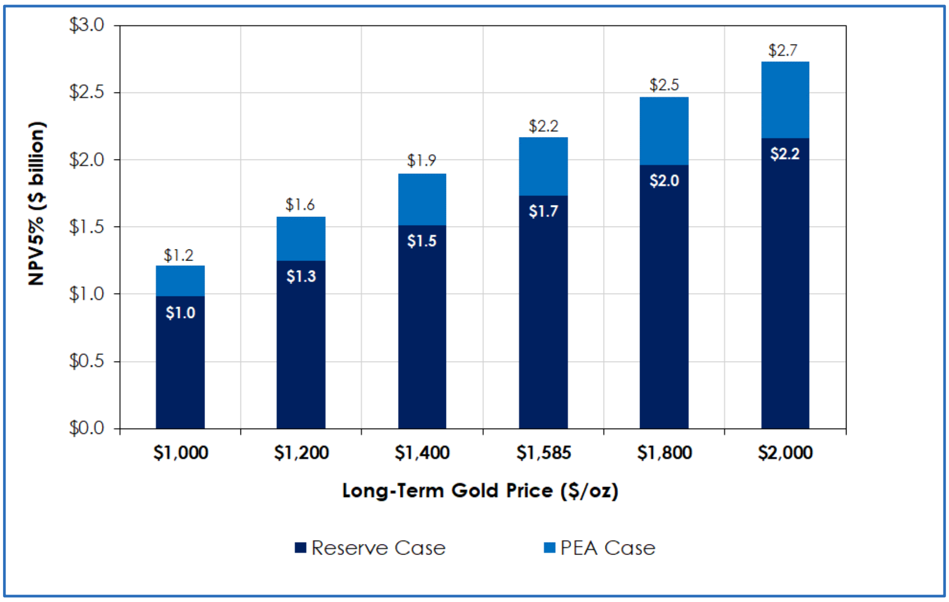
5-Year annual average is for the period 1 January 2021 through 31 December 2025

The after-tax net present value (NPV) sensitivity to metal price variation is shown in Table 1.2 and Figure 1.3 for gold prices from $1,000–$2,000/oz.

Table 1.2 Gold Price Sensitivity

After-Tax $NPV_{5\%}$ ($M)	**Long-Term Gold Price ($/oz)**					
Case	**1,000**	**1,200**	**1,400**	**1,585**	**1,800**	**2,000**
Reserve Case	981	1,251	1,510	1,733	1,962	2,162
PEA Case	1,211	1,574	1,896	2,164	2,464	2,730

Figure 1.3 Gold Price Sensitivity



OreWin, 2020

1.2 Mineral and Surface Rights

Anagold holds the exclusive right to engage in mining activities within the Çöpler project area. Anagold holds six granted licences covering a combined area of approximately 16,600 ha. Mineral title is held in the name of Anagold. Kartaltepe holds eight licences covering approximately 9,200 ha. The total near-mine tenement package is approximately 25,800 ha. Anagold currently holds sufficient surface rights to allow continued operation of the mining operation in the Reserve Case.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Çöpler project is serviced by road and rail networks. The mine is accessed from the main paved highway between Erzincan and Kemaliye. The project area is located in the Eastern Anatolia geographical district of Turkey. Mining operations are conducted year-round. The climate is typically continental with cold wet, winters and hot dry, summers.

1.4 History

The Çöpler region has been subject to gold and silver mining dating back at least to Roman times. The Turkish Geological Survey (MTA) carried out regional exploration work in the early-1960s that was predominately confined to geological mapping. In 1964, a local Turkish company started mining for manganese, continuing through until closing in 1973. Unimangan Manganez San A.Ş. (Unimangan) acquired the property in January 1979 and re-started manganese production, continuing until 1992.

In 1998, Anatolia Minerals Development Ltd (Anatolia) identified several porphyry-style gold–copper prospects in east-central Turkey and applied for exploration licences for these prospects. During this work, Anatolia identified a prospect in the Çöpler basin. This prospect and the supporting work were the basis for a joint venture agreement for exploration with Rio Tinto and Anatolia and in January 2004, Anatolia acquired the interests of Rio Tinto and Unimangan.

In August 2009, a joint venture agreement between Anatolia and Lidya was executed.

In February 2011, Anatolia merged with Avoca Resources Limited, an Australian company, to become Alacer Gold Corp. (Alacer). In September 2020, Alacer merged with SSR Mining.

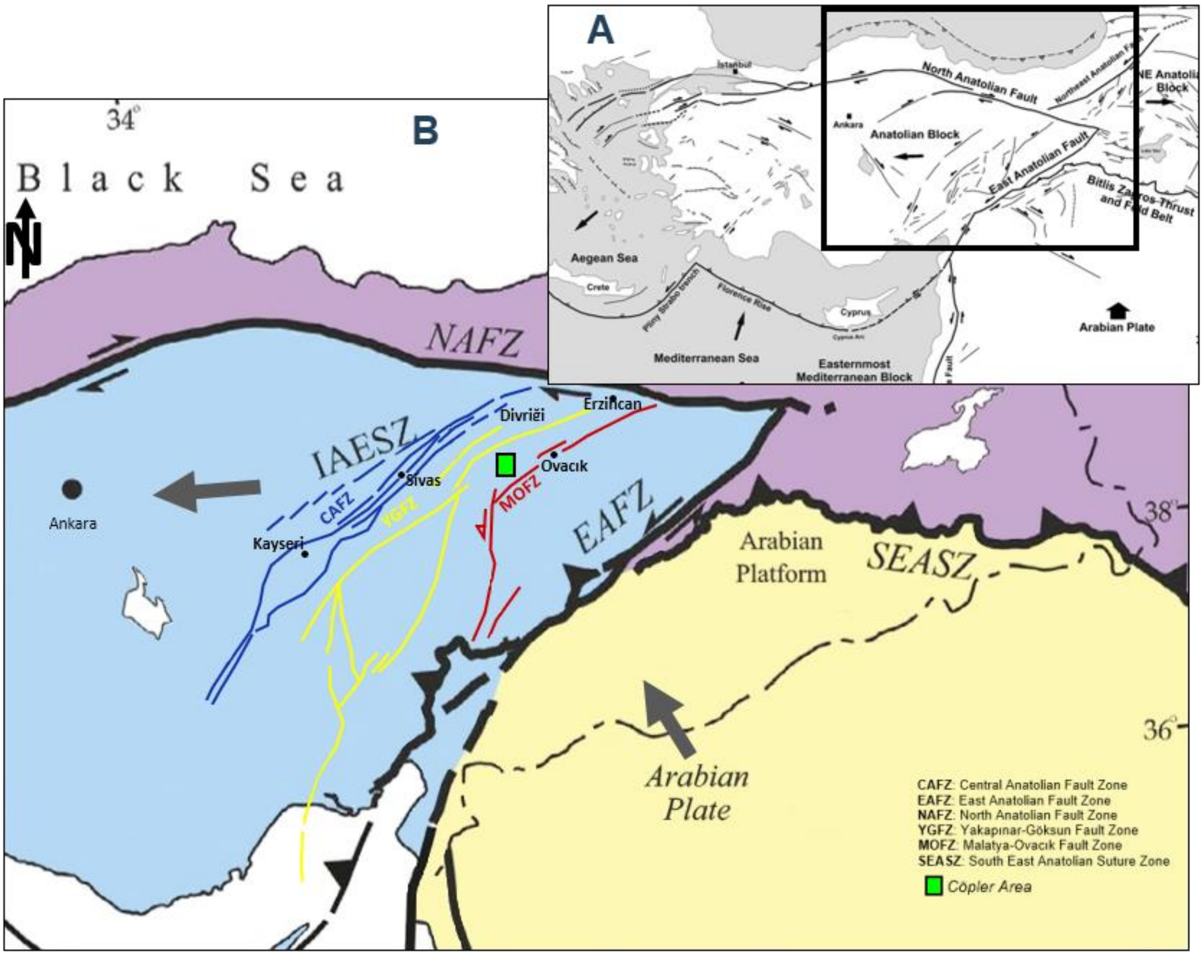
Technical Reports have been prepared on the project since 2003. The previous Technical Report on the project was issued in 2016. The 2016 Technical Report described the Çöpler Sulfide Expansion Project for construction and operation of a sulfide plant, which commenced commissioning in Q4'18.

1.5 Geological Setting and Mineralisation

The project is located near the northern margin of a complex collision zone that lies between the Pontide Belt / North Anatolian Fault, the Arabian Plate, and the East Anatolian Fault, which bounds several major plates. The region underwent crustal thickening related to the closure of a single ocean, or possibly several oceanic and micro-continental realms, in the Late Cretaceous to Early Tertiary period. Figure 1.4 illustrates the broad structural setting of the Anatolia region of Turkey. The Çöpler project area is located between Divriği and Ovacık.

![OreWin logo]

Figure 1.4 Structural Setting of Anatolia



SSR Mining, 2020

The gold, silver, and copper mineralisation of economic interest at Çöpler occurs in a porphyry-related epithermal environment, with most of the gold mineralisation concentrated in three zones: Main Zone, Manganese Zone, and Marble Zone. The mineralisation is present in five different forms:

- Stockwork and veins with disseminated marcasite, pyrite, and arsenopyrite.
- Clay-altered brecciated and carbonatised diorite with rhodochrosite veinlets, and disseminated marcasite, pyrite, realgar, orpiment, sphalerite, and galena.
- Massive marcasite and pyrite replacement bodies.
- Massive jarositic gossan.
- Massive manganese oxide.

Oxidation of this mineralisation has resulted in the formation of gossans, massive manganese oxide, and goethitic / jarositic assemblages hosting fine-grained free gold. The oxidised cap is underlain by primary and secondary sulfide mineralisation. Çöpler is a geologically complex system due to structural disturbance and multiple-stage diorite intrusions. The initial mineralisation concept model, based on geochemistry of an epithermal system overlying a copper–gold porphyry dome, continues to hold true with current modelling.

1.6 Exploration

Exploration completed since Anatolia's involvement in the Çöpler project commenced in 1998 has included:

- geological and reconnaissance mapping,
- rock chip, grab, soil, channel, and stream sediment geochemical sampling,
- ground geophysical surveys including ground magnetic, complex resistivity / induced polarisation (IP), time domain IP, and controlled source audio-frequency magneto-telluric (CSAMT) surveys,
- a regional helicopter-borne geophysical survey,
- reverse circulation (RC) and diamond core (DD) drilling programmes, and
- acquisition of satellite imagery.

Other related work has included: mining technical studies:

- geotechnical and hydrogeological studies,
- environmental and social baseline studies,
- studies in support of project permitting,
- metallurgical testwork and metallurgical studies, and
- condemnation evaluations.

The principal exploration technique at the Çöpler project has been RC and DD drilling, conducted in several campaigns starting in 2000. Initially, exploration was directed at evaluating the economic potential of the near-surface oxide mineralisation for the recovery of gold by either heap leaching or conventional milling techniques.

1.7 Drilling

Drilling at the Çöpler deposit commenced in 2000, and since that time a total of 2,554 holes have been drilled for 347,972.2 m.

Step-out drilling at the Çöpler deposit has defined most of the lateral boundaries of the mineralisation. There has been additional development drilling, as well as condemnation drilling of areas planned for infrastructure during the last few years. Infill drilling programmes have been conducted since 2007 to improve confidence in the short-term mine planning. Drilling in 2014 focused on mineralisation confirmation with a twin-hole programme. The Çöpler deposit continues to be tested using RC and DD drilling as production proceeds.

Development drilling continued in 2015 by improving sample coverage at depth in the Manganese Zone and along structural boundaries in the Main Zone. In addition to the drilling of in situ mineralisation, a stockpile drilling programme began in December 2015 to confirm sulfide stockpile ore grade, grade distribution, and mineralogy.

Drilling in 2016–2020 mainly focused on target generation to supplement the amount of oxide material in production. This was focused on the Main Zone, the West pit, and the Saddle areas. These drilling programmes aimed to test continuation of the main gold-bearing structures based on a re-interpretation of the Çöpler structural and mineralisation settings. In-pit drilling campaigns continue.

Drilling at Çakmaktepe commenced in 2012 and has resulted in the definition of three distinct mineralised zones: East, Central, and North. As production proceeded within the Çakmaktepe Central and East pits, additional targets were generated to provide push-back options around the pit design. A total of 130 DD holes have been completed since 2019 to test for continuation of the Çakmaktepe mineralisation to the north and the east.

After the initial discovery of mineralisation at Ardich, SSR Mining has undertaken several drilling programmes to better define the geological model and to improve resource inventories. Anagold has completed 304 DD holes at Ardich since late-2017, including holes for metallurgical testing and hydrogeological studies.

Drilling at Bayramdere commenced in 2007 as part of the near-mine exploration strategy. Since that time 118 holes have been drilled at Bayramdere for a total of 10,708.9 m.

1.8 Sampling Method, Approach and Analyses

From 2004 through late-2012, drillhole samples were prepared at ALS İzmir, Turkey (ALS İzmir) and analysed at ALS Vancouver, Canada (ALS Vancouver), (collectively ALS Global). From late-2012 through 2014, samples were prepared and analysed at ALS İzmir. In 2015, samples were prepared and analysed at the SGS laboratory in Ankara, Turkey (SGS). From 2015 to recent ALS İzmir is being used as the main laboratory and samples are being prepared and analysed there. Umpire analysis was completed by ACME Mineral Laboratories (ACME) in Ankara, Turkey.

ALS İzmir has ISO 9001:2008 certification, and ALS Vancouver is ISO/IEC 17025:2005 accredited for precious and base metal assay methods. SGS is certified to ISO 9001:2008 and OHSAS 18001. ACME is part of the Bureau Veritas (BV) group, globally certified to ISO9001:2008.

ALS Global and SGS are specialist analytical testing service companies; both are independent of SSR Mining.

Sampling and quality assurance and quality control (QA/QC) programmes have been in place for all RC and DD drilling conducted since the first drill programme. The QA/QC programme is currently still in use, although the insertion rates have been modified over time.

SSR Mining operates an on-site laboratory at Çöpler for assay of production samples. The on-site laboratory is certified to 17025:2017 but is not independent. It is primarily used in grade control.

1.9 Data Verification

Data verification procedures are well-established at the project. Routine ongoing checking of all data is undertaken prior to being uploaded to the database. This is followed by campaign-based independent data verification audits at milestone stages throughout data collection programmes.

For drillhole data, verification includes the checking of DGPS collar coordinates relative to topographic surveys, checking of down-hole surveys relative to adjacent readings and planned dip and azimuth of the hole, checking logged data entries to ensure they are consistent with log key sheets, cross-checking a subset of assay data with the original laboratory reports, and submission of and review of QA/QC data.

The QA/QC programme has historically consisted of a combination of QA/QC sample types that are designed to monitor different aspects of the sample preparation and assaying process: Blanks are routinely inserted in order to identify the presence of contamination through the sample preparation process; a variety of CRM standards are routinely inserted in order to monitor and measure the accuracy of the assay laboratory results over time; Field duplicates are routinely inserted as a means of monitoring and assessing sample homogeneity and inherent grade variability and to enable the determination of bias and precision between sample pairs; laboratory duplicates are inserted as a means of testing the precision of the laboratory measurements; and inter-laboratory pulp duplicates are submitted to alternative independent laboratory to assess for bias or drift. The rate of submission has been modified over time but is currently 3%–5% for blanks, CRMs, and duplicates, and 5%–10% for field duplicates.

None of the verification programmes have identified material issues with the supporting data.

1.10 Metallurgical Testwork

1.10.1 Oxide Testwork

The heap leaching facilities were commissioned at the Çöpler mine site in late-2010 and have operated continuously since that time. Operations were continuing at the CDMP20 effective date.

Metallurgical testwork on Çakmaktepe oxide material for heap leaching has been undertaken at the on-site metallurgical laboratory, initially under the supervision of Kappes, Cassiday & Associates. The initial testwork in 2015 undertook bottle roll and column leach tests. The results are comparable with the Çöpler oxide ore, with similar behaviour and leach kinetics. Subsequently, Çakmaktepe oxide ore has been heap leached along with Çöpler oxide ore. Oxide column testwork on oxide ore continues at the on-site laboratory.

Metallurgical testwork on Ardich oxide material for heap leaching has been undertaken at McClelland laboratories and supervised by Metallurgium. An initial testwork programme, including bottle roll and column leach, was carried out in 2019. This initial programme identified two distinct domains with respect to gold recovery based on sulfur content; <1% and 1%–2%. The column testwork results indicated that the listwanite, dolomite, and jasperoid lithologies have physical properties amenable to heap leaching. This initial test programme is being followed up with further testwork in 2020.

Analysis of the results of the metallurgical testwork and a review of the existing recovery models for use in economic analysis were undertaken in 2020. This was done for the oxide and sulfide processing, including the flotation circuit. The resulting recoveries have been used in the economic analysis for the CDMP20.

Oxide gold recoveries vary by lithology for Çöpler in the range 62.3%–78.4%, at Çakmaktepe the range is 61.0%–80.0% The average oxide recovery in the Reserve Case is 73%. At Ardich the testwork suggest recoveries will vary in the range 40.0%–73.0%.

1.10.2 Sulfide Testwork

The sulfide process plant commenced commissioning in Q4'18. The plant consists principally of a pressure oxidation (POX) leach followed by a cyanide leach to recover gold.

Significant testwork had been conducted on sulfide ores prior to commissioning of the sulfide plant, with pilot plant testwork campaigns and a significant number of batch variability tests on POX / cyanide leach completed.

Whilst a POX / cyanide leach circuit was implemented, significant work had also been undertaken on flotation of the gold-bearing sulfides as a process route, although ultimately this option was not selected for development. Flotation of a partial stream of the plant feed was considered to maximise the available capacity of the plant, including the POX autoclave and available oxygen supply. Further flotation testwork demonstrated that the addition of a small flotation plant into the existing sulfide process route would allow optimisation and maximisation of already installed capacities.

The testwork indicates that sulfur recovery through flotation is estimated to be 75% to concentrate with a corresponding 55% gold recovery. Flotation tails gold recovery is estimated at 43%.

The current determination of POX gold recovery is based on assessment of results for the pilot testwork programmes undertaken prior to commencement of operations and benchmarked with the existing operating data. An equation has been derived to calculate gold recovery by material type for all ore that is subject to POX; this includes direct POX feed and flotation concentrate. The Reserve Case average sulfide gold recovery is 91%.

1.11 Mineral Resource

1.11.1 Resource Modelling

1.11.1.1 Çöpler Deposit

The Çöpler deposit includes four mine areas: Main, Manganese, Marble, and West. The current Çöpler resource model, which was constructed by SSR Mining personnel, was completed in February 2016.

The cut-off date for the drillholes database was 15 July 2015. The data extract contained 1,957 drillholes with a total of 297,798.2 m of drilling. Of this, a total of 1,880 drillholes have collar coordinates within the extents used to construct the resource model. In general, the drillhole spacing ranged from 5–60 m, averaging approximately 20 m. Most drillholes are either vertical or inclined at 60°.

Wireframes were constructed for the four main geological units: diorite, metasediment, marble, and manganese-rich diorite. Drillhole data and surface mapping were developed into 3D solids that represent the major rock types using implicit modelling techniques. This process included generating contact surfaces used to define the division boundaries that represent the geological faults and lithological contacts.

The resource estimation method at Çöpler was developed to address the variable nature of the gold mineralisation while honouring the bi-modal distribution of the sulfur mineralisation that is critical for mine planning (material with a total sulfur grade <2% is sent to the heap leach while material with total sulfur grade ≥2% is sent to the sulfide stockpile for eventual processing at the POX plant). Since no obvious correlations were observed between gold and total sulfur, gold and sulfur were domained and estimated separately. Gold showed little correlation with lithology and was domained by mining areas to reflect the different trends of the mineralisation that commonly follow structures and/or lithological contacts. Due to the strong correlation between sulfur content and lithology, sulfur was domained by lithology. However, since each lithology may contain <2% S and ≥2% S material, each lithology was additionally separated into <2% S and ≥2% S sub-domains.

Probability assigned constrained kriging (PACK) was used to estimate the gold content of the mineralisation within an expanded mineralised wireframe. A probabilistic envelope was generated within the expanded gold shape to define the limits of the economic mineralisation. The wireframe and probabilistic envelope were used to prevent potentially economic assays from being 'smeared' into non-economic zones, and conversely to restrict waste assays from diluting the potentially economic mineralisation. Two PACK cell models were constructed for gold. The first (low-grade gold) model was applied to <2% S material that can be processed by heap leaching, and the second (high-grade gold) model was later applied to ≥2% S material to be processed by the POX plant.

Once constructed, the gold models were calibrated to historical production data, categorised by sulfur content (<2% S and ≥2% S), and mining area. Estimates were classified into Mineral Resource categories based on drillhole density and data quality.

Density values were assigned to the cell model based on lithological domain and depth below the surface.

1.11.1.2 Çakmaktepe Deposit

The Çakmaktepe deposit is located 6 km east of the current Çöpler pit and includes four areas: North, Central, East, and Southeast. The current Çakmaktepe resource model, which was constructed by SSR Mining personnel, was completed in February 2020.

The drillhole dataset used to develop the February 2020 resource model contained a total of 1,109 holes with a drilling date range of September 2007–October 2019. The total drilled metres input into the modelling was 119,001.1 m. Original sample lengths are predominately 1 m in length with some 2 m sampling across areas presumed to be waste. The mean sample length was 1.02 m. The shortest interval was 0.1 m with maximum length 3.1 m. Composited samples 5 m in length were used for statistical analysis, construction of interpretation boundaries, and grade estimation.

Mineralisation at Çakmaktepe follows structural controls and designated lithological contact orientations. Mineralised zones often incorporate multiple lithological units along the boundary rather than being hosted by a single rock type. For this reason, grade shells were constructed for gold and copper to allow estimation concordant with the mineralised zones instead of being controlled by samples residing within a single lithological unit. Mineralised trends were honoured in 3D with no specific grade cut-off used to bound the mineralised shapes. The resulting shapes for gold and copper are lenticular with thicknesses ranging from 5–40 m. On average, thicknesses are of the order of 6 m.

Sulfur grades correlate with lithological units: higher sulfur values are associated with diorite and metasediment, and lower sulfur values are in association with gossan, jasperoid, ophiolite, and marble.

A single geological cell model with 5 m x 5 m x 5 m parent cells was constructed to include the four deposit areas. Gold, silver, copper, sulfur, and carbon were estimated using inverse distance interpolation (ID) weighted to the power of three (ID3) and 5 m drillhole composites. Gold, copper, and silver were estimated using grade shells as hard boundaries. Sulfur and carbon estimates were constrained by modelled lithological units. All grade shell boundaries for metal estimates were treated as hard. Domains were treated as soft boundaries allowing the selection of samples from nearby domains.

Density values were assigned to the cell model based on lithological domain.

1.11.1.3 Ardich Deposit

The Ardich deposit is located 1.5 km north of Çakmaktepe and includes two areas: Main and East. The current Ardich resource model, which was constructed by SSR Mining personnel, was completed in June 2020.

The drillhole dataset used to develop the June 2020 resource model contained a total of 233 Ardich holes with a drilling date range of September 2017–December 2019. Total drilled metres for Ardich was 43,411.7 m. Original sample lengths are predominately 1 m. The shortest assayed interval was 0.2 m, the maximum length 3.0 m, and the mean sample length was 1.03 m. Composite samples 5 m in length were used for statistical analysis, construction of interpretation boundaries, and grade estimation.

The Ardich Mineral Resource estimate was based on a 3D geological solids model developed within constraining fault blocks. The main lithological units: ophiolite, listwanite, dolomite, and cataclasite, are offset by faults, creating rotated blocks that have moved up and down relative to each other. High-angle faults cross-cut the deposit with several low-angle structures carrying metal grades along the dolomite / listwanite contact. Mineralised trends follow the orientations of the structural controls and lithological contacts as they change within the fault blocks. Domains for Ardich are defined by these fault blocks.

Gold distribution is related to lithological contact zones and structural intersections. These zones tend to be narrow and localised. Control of the gold estimation is accomplished by using grade shells as hard boundaries. Mineralised gold grade shells were built using composites located along structural and lithological features. In some cases, the grade shell follows the lithology strata within a domain and extends across the interpreted fault to allow estimation of grades along the fault boundary.

A single geological cell model with 15 m x 15 m x 5 m parent cells was constructed to include the two deposit areas. Au was interpolated within grade shells using ID3 and S within lithological units using ID2, both using 5 m drillhole composites. All grade shells and lithological units were treated as hard boundaries. Domains were treated as soft boundaries allowing the selection of samples from nearby domains.

1.11.1.4 Bayramdere Deposit

The Bayramdere deposit is located approximately 6.3 km east of the Çöpler mine and 5 km south east of İliç. It is within the Kartaltepe Mining Licence 7083. This licence is an operational licence and is 50% SSR Mining-owned.

The Bayramdere mineralisation has an overall strike length of approximately 300 m. Mineralisation is localised within three stacked, shallow-dipping lodes that vary in depth between 30–40 m below topography. Mineralisation appears to be open to the east and south.

A resource model for Bayramdere was completed in 2016. Separate mineralisation domains were created for gold, silver, copper, and sulfur. In the creation of mineralised domains, a minimum mining width of 2.5 m was used based on anticipated open pit mining methods. Grade estimation was limited to the interpreted domains. Outside the mineralised domains a 'mineralised waste' estimate was completed. Lithological domains were used for estimates outside of the mineralisation domains. Ordinary kriging was used to estimate gold, silver, and copper mineralisation into parent cells of 10 m x 10 m x 5 m size with sub-celling permitted to 2 m x 2 m x 1 m to better honour the domain boundaries.

Density was assigned as a default for each of the mineralisation and lithological domains.

Although a small deposit, Bayramdere is relatively high grade and can support a high stripping ratio to access mineralisation.

1.11.2 Reasonable Prospects for Eventual Economic Extraction

All Mineral Resources in the CDMP20 were assessed for reasonable prospects for eventual economic extraction by reporting only material that fell within conceptual pit shells based on metal prices of $1,750/oz for gold, or as otherwise specified.

1.11.3 Mineral Resource Estimates

Mineral Resources are reported inclusive of Mineral Reserves and have been summarised by resource classification and oxidation state in Table 1.3.

Mineral Resources have been classified using the 2014 CIM Definition Standards (CIM, 2014) and were estimated by Sharron Sylvester BSc (Geology), RPGeo AIG (10125), employed by OreWin Pty Ltd as Technical Director – Geology. Mineral Resources are presented on a project basis and have an effective date of 27 November 2020.

Table 1.3 CDMP20 Mineral Resources Summary

CDMP20 Mineral Resources Summary (as at the Effective Date)							
Classification	Tonnage (kt)	Grades			Contained Metal		
		Au (g/t)	Ag (g/t)	Cu (%)	Gold (koz)	Silver (koz)	Copper (klb)
Çöpler Mine Oxide Mineral Resource							
Measured	287	1.29	7.75	0.09	12	72	540
Indicated	25,139	0.98	3.44	0.15	789	2,781	81,399
Measured + Indicated	25,427	0.98	3.49	0.15	801	2,853	81,939
Inferred	*33,083*	*0.96*	*7.16*	*0.13*	*1,017*	*7,614*	*94,935*
Çöpler Mine Sulfide Mineral Resource							
Measured	2,454	2.22	7.21	–	175	569	–
Indicated [5]	84,558	1.84	5.04	–	5,015	12,617	–
Measured + Indicated	87,012	1.86	4.71	–	5,190	13,186	–
Inferred	*34,073*	*1.54*	*12.72*	–	*1,692*	*13,937*	–
Çakmaktepe Oxide Mineral Resource							
Measured	–	–	–	–	–	–	–
Indicated [6]	3,626	1.53	8.50	–	179	990	–
Measured + Indicated	3,626	1.53	8.50	–	179	990	–
Inferred	*1,205*	*0.85*	*4.04*	–	*33*	*157*	–
Ardich Oxide Mineral Resource							
Measured	4,707	1.63	–	–	246	–	–
Indicated	12,817	1.62	–	–	666	–	–
Measured + Indicated	17,524	1.62	–	–	912	–	–
Inferred	*4,713*	*1.62*	–	–	*246*	–	–
Ardich Sulfide Mineral Resource							
Measured	695	2.56	–	–	57	–	–
Indicated	2,231	3.71	–	–	266	–	–
Measured + Indicated	2,926	3.43	–	–	323	–	–
Inferred	*782*	*4.24*	–	–	*107*	–	–
Bayramdere Oxide Mineral Resource							
Measured	–	–	–	–	–	–	–
Indicated	145	2.34	20.82	–	11	97	–
Measured + Indicated	145	2.34	20.82	–	11	97	–
Inferred	*8*	*2.17*	*19.95*	–	*1*	*5*	–
CPMD20 Mineral Resources Total							
Measured	8,143	1.87	2.45	0.00	490	641	540
Indicated	128,517	1.68	3.99	0.03	6,926	16,485	81,399
Measured + Indicated	136,660	1.69	3.90	0.03	7,416	17,126	81,939
Inferred	*73,865*	*1.30*	*9.14*	*0.06*	*3,094*	*21,713*	*94,935*

1. Mineral Resources have an effective date of 27 November 2020.
2. Mineral Resources are reported based on end of August 2020 topography surface.
3. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
4. Mineral Resources are shown on a 100% basis. Çöpler Mineral Resources are located on ground held 80% by SSR Mining, Çakmaktepe and Bayramdere Mineral Resources are located on ground held 50% by SSR Mining, and approximately 96% of Ardich Mineral Resources are located on ground held 80% by SSR Mining, with the remainder located on ground 50% held by SSR Mining.
5. Çöpler Sulfide Indicated total includes stockpiles: 6,674 kt @ 2.63 g/t Au (*).
6. Çakmaktepe Oxide Indicated total includes stockpiles: 11 kt @ 2.69 g/t Au (t).
7. At Çöpler: oxide is defined as material <2% total sulfur and sulfide material is ≥2% total sulfur.
8. At Ardich and Çakmaktepe, low-sulfur (LS) oxide is defined as material with <1% total sulfur, high-sulfur (HS) oxide is material with ≥1% and <2% total sulfur, and sulfide material is ≥2% total sulfur.
9. At Bayramdere: oxide is defined as material <2% total sulfur. There is no sulfide material at Bayramdere.
10. All Mineral Resources in the CDMP20 were assessed for reasonable prospects for eventual economic extraction by reporting only material that fell within conceptual pit shells based on metal prices of $1,750/oz for gold ($1,400 for gold and $19/oz for silver for Bayramdere). The following parameters were used:
    metallurgical recoveries in oxide: Çöpler 62.3%–78.4%, Çakmaktepe 38.0%–80.0%, Ardich 40.0%–73.0%, and Bayramdere 75.0%, and in sulfide: Çöpler 85.0%, and Ardich 82.9%;
    Au cut-off grades in oxide: Çöpler 0.32–0.41 g/t Au, Çakmaktepe 0.36–0.76 g/t Au, Ardich 0.30–0.55 g/t Au, and Bayramdere 0.35–0.50 g/t Au, and in sulfide: Çöpler 0.73 g/t Au and Ardich 0.77 g/t Au, (there are no credits for Ag or Cu in the cut-off grade calculations); allowances have been made for royalty payable.
11. Reported Mineral Resources contain no allowances for unplanned dilution or mining recovery.
12. Totals may vary due to rounding.

1.12 Mining Method

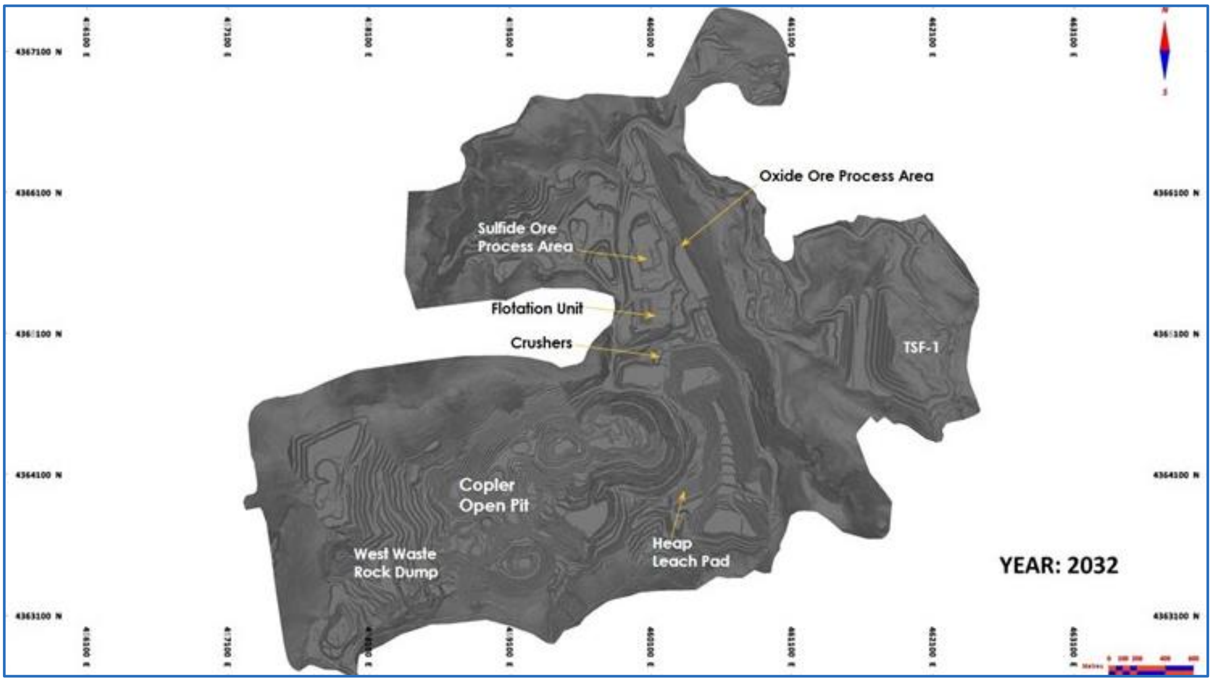
Open pit mining at the Çöpler project is carried out by a mining contractor and managed by Anagold. The mining method is a conventional open pit method with drill and blast and utilising excavators and trucks operating on bench heights of 5 m. The mining contractor provides operators, line supervisors, equipment, and ancillary facilities required for the mining operation. SSR Mining provides management, technical, mine planning, engineering, and grade control functions for the operation.

SSR Mining currently operates a sulfide process plant and an oxide heap leach facility. Costs are based on the actual operational costs and the project budget assumptions.

Production schedules and costs have been updated based on current site performance and contracts. There is currently only 7.9 Mt of oxide ore in the Mineral Reserves, of this 285 kt is remaining at the Çakmaktepe deposit, therefore most of the remaining mining will be at the Çöpler deposit.

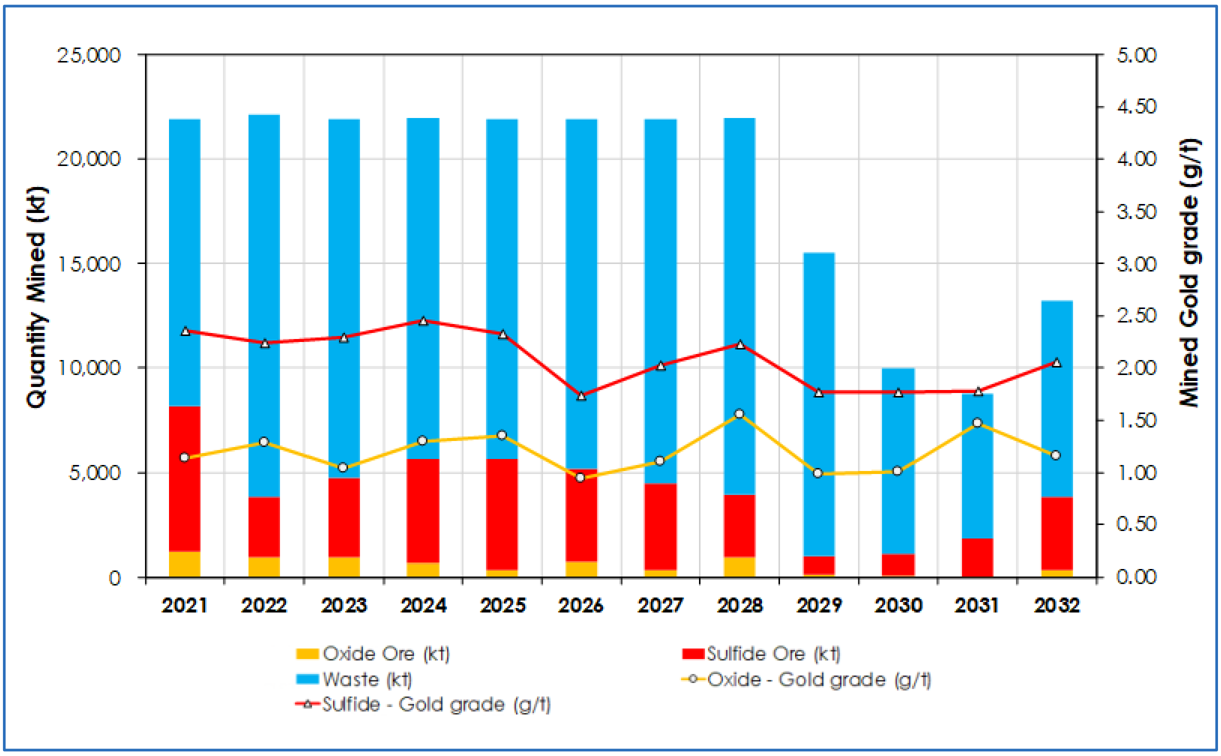
Pit designs from the 2016 Technical Report have been mined since 2016 and there is still significant ore remaining within those designs. In 2020 two additional phases in the Main Zone were designed and included in the Mineral Reserves. The Çöpler pit design for 2032, when in-pit mining is completed for the Reserve Case, is shown in Figure 1.5. Following completion of in-pit mining, the sulfide plant will be fed from stockpiles until 2041. The Reserve Case mining production is shown in Figure 1.6.

Figure 1.5 2032 Pit Plan – Çöpler



Anagold, 2020

Figure 1.6 CDMP20 Reserve Case Mining Production



OreWin, 2020

1.13 Recovery Methods

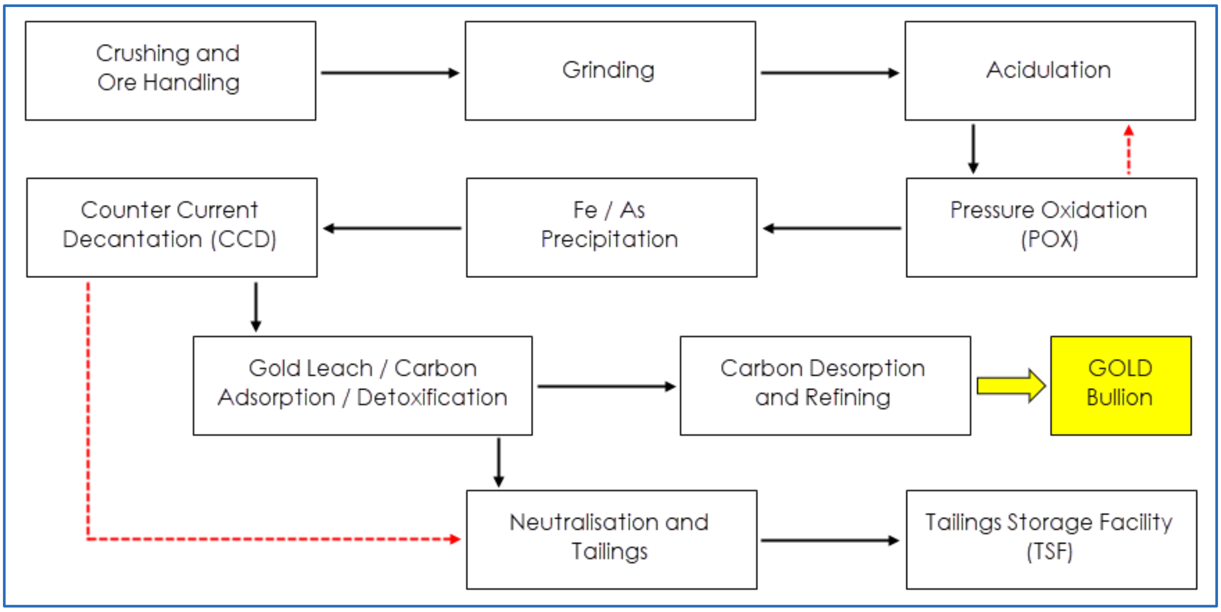
1.13.1 Sulfide Plant

The sulfide plant commenced commissioning in Q4'18. The basic flow sheet is shown in Figure 1.7 and comprises:

- Crushing and ore handling
- Grinding
- Acidulation
- Pressure oxidation
- Iron / arsenic precipitation
- Counter Current Decantation (CCD)
- Gold leach, carbon adsorption, and detoxification
- Carbon desorption and refining
- Neutralisation and tailings
- Tailing Storage Facility (TSF)

The sulfide plant performance from Q4'18 up to Q1'20, including commissioning and ramp-up, has achieved greater-than-design throughputs and approaches design gold recovery for the ore types processed.

Figure 1.7 Process Flow Sheet for Sulfide Plant

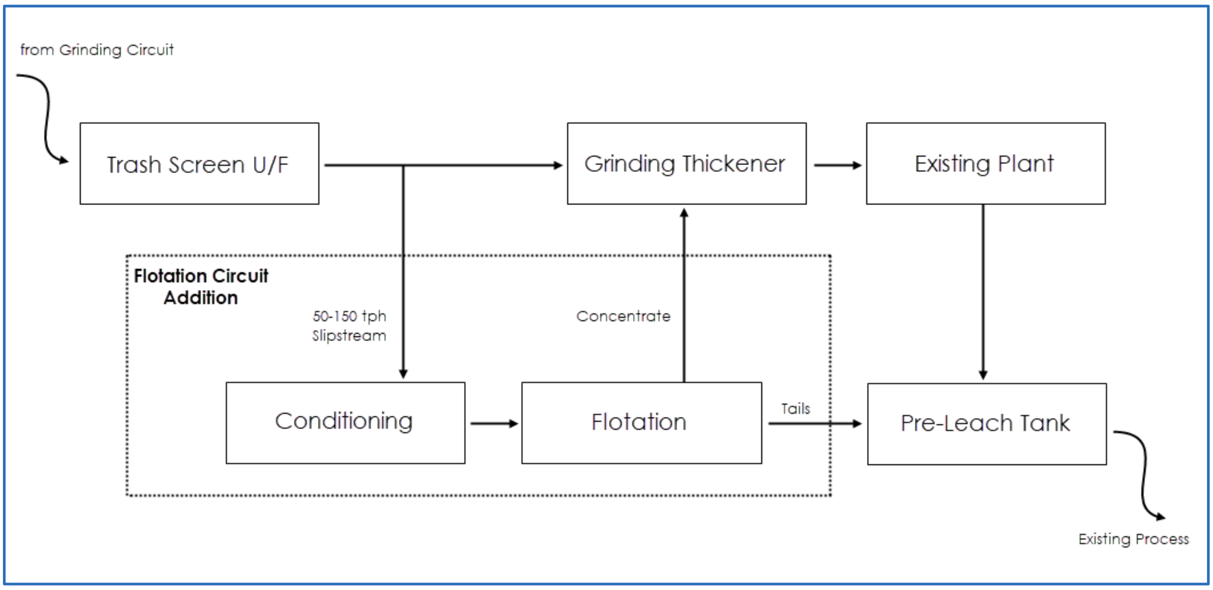


Anagold, 2020

The incorporation of a new flotation circuit in the existing sulfide plant to upgrade SS to fully utilise POX autoclave oxidation capacity is under design and construction. This addition to the sulfide plant is incorporated between grinding and acidulation, as shown in Figure 1.8, by taking a bleed / slip stream from the grinding thickener feed, floating gold-bearing sulfides, rejecting acid-consuming carbonates and returning the sulfide concentrate to the grinding thickener to be combined with direct POX feed. The gold not recovered to concentrate that remains in the flotation tails is directed to the gold leach circuit feed to recover this remaining gold, albeit at lower gold recoveries than ore that is treated through the POX autoclave circuit.

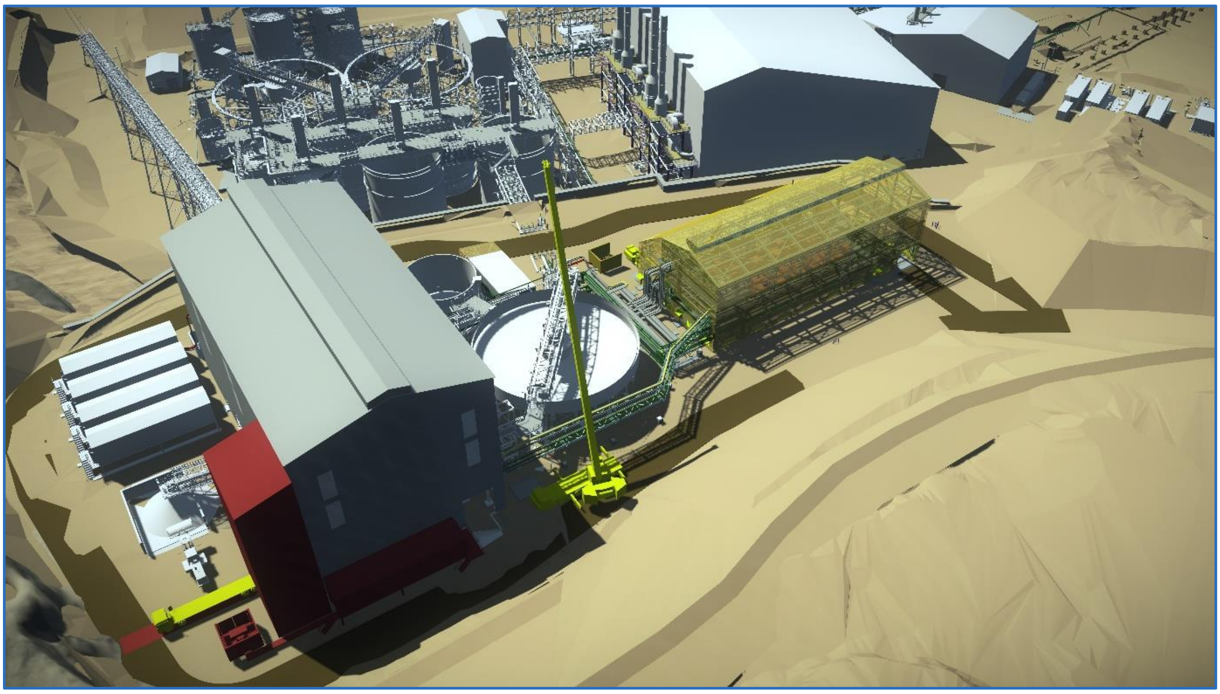
This will increase overall plant maximum throughput rate to 400 tph, allowing the grinding and POX circuit to operate at their maximum demonstrated capacities. The grinding circuit maximum volumetric flow throughput will increase from an original design limit of 306 tph to 400 tph, fully utilising latent capacity within the crushing and grinding circuit. The flotation plant is designed to operate in the throughput range of 50–150 tph to produce a concentrate that will supplement the feed ore SS to maximise autoclave SS up to 13.75 tph at a maximum autoclave feed rate of 280 tph. Figure 1.9 indicates the position of the flotation building. Operating performance of the autoclaves indicates that higher than design oxygen utilisations efficiencies are possible, which may allow greater than 13.75 tph SS to be treated. This oxygen utilisation efficiency along with increased oxygen availability is upside to the CDMP20 Reserve Case.

Figure 1.8 Flotation Block Flow Diagram



Anagold, 2020

Figure 1.9 Flotation Circuit Building Location



Anagold, 2020

The existing sulfide circuit, before the addition of flotation, has demonstrated additional latent capacity in throughput controlling sections of the circuit, crushing/grinding and autoclaves. The incorporation of flotation will allow the POX autoclaves to maximise throughput and SS oxidation capacity, utilising latent capacity in the process plant, in particular, the grinding and POX circuits. Fully utilising this latent capacity with the addition of a small flotation plant allows the increase in overall plant throughput at a minimal capital cost.

The POX autoclave circuit has demonstrated it can process up to a maximum of 280 tph feed and 13.75 tph SS, compared to design of 245 tph and 12.5 tph respectively. The limit of 13.75 tph SS is dictated by the capacity of the oxygen supply to effect oxidation of the sulfides. The flotation plant feed rate will be variable between 50–150 tph based on SS feed grade and the oxidation capacity of the POX autoclaves to oxidise sulfides. Operating performance of the autoclaves indicates that higher than design oxygen utilisations efficiencies are possible, which may allow greater than 13.75 tph SS to be treated. This oxygen utilisation efficiency along with increased oxygen availability is upside to the CDMP20 Reserve Case.

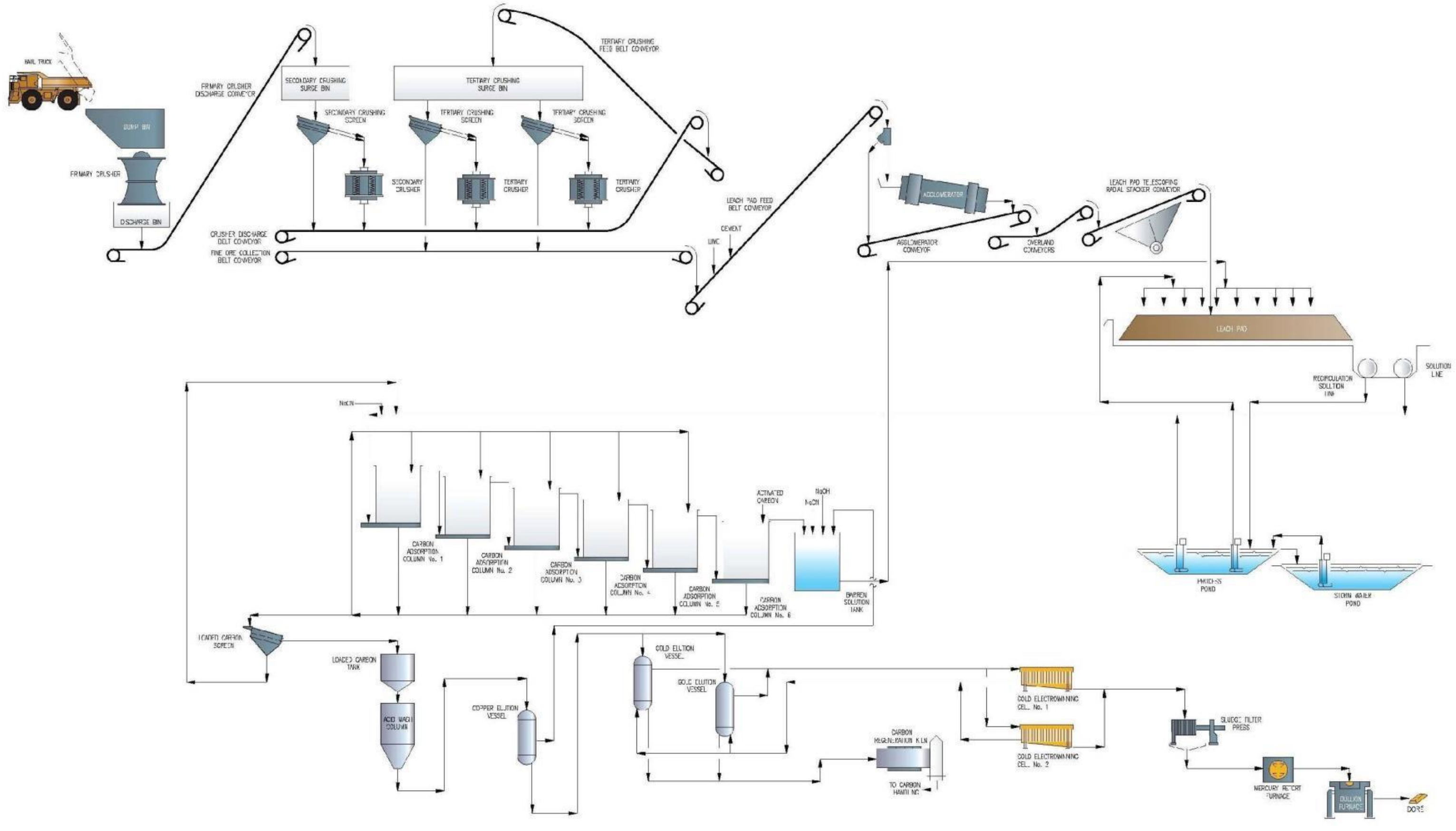
1.13.2 Oxide Ore Heap Leach Processing

In the Reserve Case production is predominantly from sulfide ore. The maximum oxide ore placed in any year is 1.2 Mt for a total production of 7.9 Mt.

The oxide heap leaching and associated facilities were commissioned in the second half of 2010 and initial gold production was achieved in Q4'10. The process was originally designed to treat approximately 6.0 Mtpa of ore by three-stage crushing (primary, secondary, and tertiary) to 80% passing 12.5 mm, agglomeration, and heap leaching on a lined heap leach pad with dilute alkaline sodium cyanide solution. Gold is recovered through a carbon-in-column (CIC) system, followed by stripping of metal values from carbon, electrowinning and smelting to yield a doré (containing gold and silver) suitable for sale. Control of copper in leach solutions is undertaken in a sulfidisation, acidification, recovery, and thickening (SART) plant, which also regenerates cyanide.

The oxide ore heap leach process flow sheet is shown in Figure 1.10.

Figure 1.10 Process Flow Sheet for Oxide Ore Heap Leach



SSR Mining, 2016

1.13.3 Project Infrastructure

1.13.3.1 Infrastructure

The facility infrastructure supports the mine and process areas of oxide heap leach and sulfide plant. The existing infrastructure, and the tailings storage and heap leach pad area once the planned expansion is complete, will be sufficient for the current Mineral Reserves. The infrastructure for the addition of flotation to the sulfide plant will be supported by the existing facility infrastructure with some components modified to meet the addition of the flotation circuit.

The current leach pad consists of four phases designed to accommodate approximately 58 Mt of oxide ore heap with a nominal maximum heap height of 100 m above the pad liner. An additional two phases (phase 5 and phase 6), with a capacity of 20 Mt are yet to be approved but are not required for the Mineral Reserve.

The Tailings Storage Facility (TSF) is developed and constructed in stages. The development of TSF 1 includes seven phases. TSF 1 phase 3 is under construction in Q4'20. Ongoing work in ensuring sufficient long-term capacity for storage of tailings has been undertaken. Studies by Anagold have determined that the effect of the addition of the flotation circuit to the sulfide plant would result in an increase in the solids content and improvement in the final settled density based on an increase in the rate of tailings consolidation.

TSF 1 has sufficient storage capacity (70.8 Mt) to accommodate the CDMP20 tailings. Scoping level investigations have identified additional TSF sites. An adjacent site, TSF 2, has been the subject of a PFS level study and can provide approximately 20 Mt of net additional tails storage capacity, if required in the future.

TSF 2 construction is not included in the mine plan but remains as an option for further expansions.

1.14 Market Studies

The markets for gold and silver doré are readily accessed and available to gold producers. Currently, 100% of the gold and silver is delivered to the Istanbul Gold Refinery. Copper precipitate is currently produced from the SART plant and sold into local markets in Turkey. The sulfide plant does not currently include a copper circuit. Provisions have been made in the plant design to include the copper circuit in the future if market conditions warrant.

1.15 Environmental and Permitting

The Çöpler mining and processing operations involve open pit mining from multiple pits, construction of multiple waste rock dumps (WRD) to accommodate mined materials, processing of oxide ores and placement on a heap leach pad, and processing of sulfide ores with placement of tailings in a TSF. These activities and facilities are carried out on treasury, pasture, and forestry lands, including some private lands.

In addition to the direct impacts on the involved lands, the operations impact the surrounding lands and the local communities. Physical impacts may include changes to local surface and groundwater (including potential pollution), air quality impacts particularly from dust, and increased noise and vibration from mining and processing activities.

Operation of the Çöpler mining and processing facilities, and subsequent mining at Çakmaktepe, have been investigated and authorised by means of a series of Environmental Impact Assessments (EIAs), with positive decisions obtained from the Turkish Ministry of Environment and Urban Planning (MEUP). These EIA's include specific actions designed to address all material impacts of the mining and processing operations. Anagold has remained in compliance with all aspects of the EIA and operating permits throughout the history of the project.

The original 2008 EIA obtained on 16 April 2008 included three main open pits (Manganese, Marble, and Main zones), five WRDs, a heap leach pad, a processing plant, and a TSF. The 2008 project description involved only the oxide resources.

The Çöpler mine started its open pit and heap leach operation in 2010 and first gold was poured in December 2010. Additional EIA investigations have been submitted and approved, as required, to support on-going mining and processing operations, including:

- EIA to allow operation of a mobile crushing plant, approved 10 April 2012.
- EIA to allow waste dump capacity expansion, oxide capacity expansion to 23,500 tpd and a SART plant, approved 17 May 2012.
- EIA to allow the sulfide plant and heap leach area expansion, approved 24 December 2014.
- EIA to allow the Çakmaktepe satellite pits expansion, approved 26 January 2017.
- EIA to allow a Çakmaktepe capacity increase, approved 9 August 2018.

In addition, pending EIA processes include:

- EIA to allow a second capacity expansion at Çöpler, including heap leach pads 5 and 6, TSF expansion, and operation of a flotation plant (the permitting process was started in December 2019 and a public hearing was held January 2020).
- EIA to allow second capacity increase on the Çakmaktepe EIA to include initial mining from Ardich in the EIA project description file, submitted in October 2020.

Subsequent to the EIA positive decisions, additional licences and permits were required to be issued by government agencies consistent with the Turkish governing laws and regulations. These include land access permits (treasury, pasture and forestry), environmental licences and permits, and workplace opening and operating permits, licences, and certificates.

1.16 Capital and Operating Costs

Capital and operating cost estimates have been developed based on the current project costs, the mine and process designs, and discussions with potential suppliers and contractors. The sulfide growth costs include the capital cost for the flotation circuit. The estimated capital costs are to a feasibility level of accuracy and include a contingency of 10%.

1.17 Capital Costs

Capital costs have been split into growth and sustaining costs. The sustaining costs also include the reclamation costs for closure. Capital costs do not include mining costs capitalised as deferred stripping. The CDMP20 Reserve Case capital costs to the end of 2021 and life-of-mine (LOM) are shown in Table 1.4.

Growth capital costs in the Reserve Case includes costs for:

- Flotation circuit
- Heap leach phase 4B
- Road relocation, studies, and project management
- Explosives magazine

Sustaining capital in the Reserve Case includes costs for:

- TSF expansion
- Project team
- Technical services
- Administration
- Assay laboratory
- Mining
- IT
- Sulfide and oxide processing
- Environment
- Mineral / lands rights
- Health and safety
- Security
- Supply chain
- Reclamation

Table 1.4 CDMP20 Reserve Case Capital Costs

Description	Capital Costs ($M)	
	Q4'20 and 2021	Total LOM
Oxide		
Growth	29	29
Sustaining	4	9
Sulfide		
Growth	29	29
Sustaining	59	421
Site		
Reclamation	2	103
Working and Other	8	14
Total	131	605

Capital costs do not include mining costs capitalised as deferred waste stripping

1.18 Operating Costs

Operating costs were estimated based on current site cost performance and contract costs, including actual operational costs for labour, consumables, contracts, and the Anagold budget assumptions. Operating costs have a base date of Q4'20 with no allowance for escalation. LOM average operating costs are shown in Table 1.5.

Table 1.5 Summary of LOM Average Operating Costs

Description	Total LOM ($M)	Operating Costs ($/t ore)		
		Years 1–5	Years 1–10	LOM
Mining	371	9.59	9.60	6.32
Ore Rehandle	62	0.67	0.79	1.05
Processing	1,872	28.28	29.45	31.86
Site Support	462	10.11	9.79	7.86
Total Operating Cost	2,767	48.64	49.63	47.09

Mining costs include waste stripping costs

1.19 CDMP20 Reserve Case

The Reserve Case production includes 7.7 Mt at 1.22 g/t Au oxide ore processed by heap leaching and 51.1 Mt at 2.24 g/t Au processed in the sulfide plant. Total gold production is 3.6 Moz. All mining is completed by 2032, oxide heap leach stacking is completed in 2031, while sulfide processing will continue from stockpiles until 2041. The Reserve Case shows an after-tax NPV at a 5% discount rate of $1.73 billion. The operation is cash positive in each year of the mine plan, therefore an IRR is not reported. The Reserve Case average all-in sustaining cost (AISC) is $945/oz gold. The key results of the Reserve Case economic analysis are shown in Table 1.6.

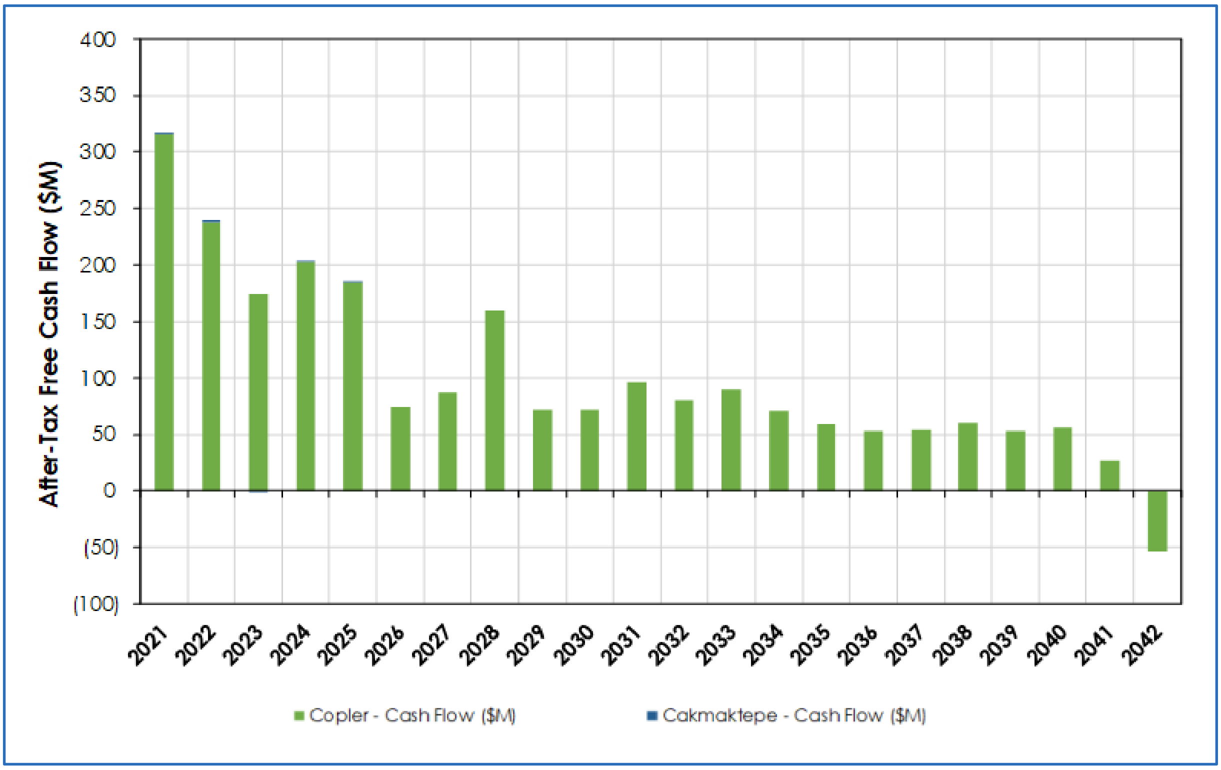
Table 1.6 CDMP20 Reserve Case Results Summary

Item	Unit	Reserve Case
Oxide Processed		
Heap Leach Quantity	kt	7,668
Au Feed Grade	g/t	1.22
Sulfide Processed		
Quantity Milled	kt	51,084
Au Feed Grade	g/t	2.24
Total Gold Produced		
Oxide – Gold	koz	256
Sulfide – Gold	koz	3,334
Total – Gold	koz	3,591
Oxide – Gold Recovery	%	73
Sulfide – Gold Recovery	%	91
5-Year Annual Average		
Average Gold Produced	kozpa	266
Free Cash Flow	$Mpa	224
Production Costs	$/oz gold	682
All-in Sustaining Costs	$/oz gold	865
Key Financial Results		
Production Costs	$/oz gold	748
All-in Sustaining Costs	$/oz gold	945
Site Operating Costs	$/t treated	47.09
After-Tax $NPV_{5\%}$	$M	1,733
Mine Life	years	21

5-Year Annual Average is for the period 1 January 2021 through 31 December 2025

The after-tax cash flow is shown in Figure 1.11. The NPV results for before and after-tax over a range of discount rates is shown in Table 1.7. The sulfide and oxide production profiles are shown in Figure 1.12 and Figure 1.13. Cash costs are shown in Table 1.8.

Figure 1.11 Reserve Case After-Tax Cash Flow

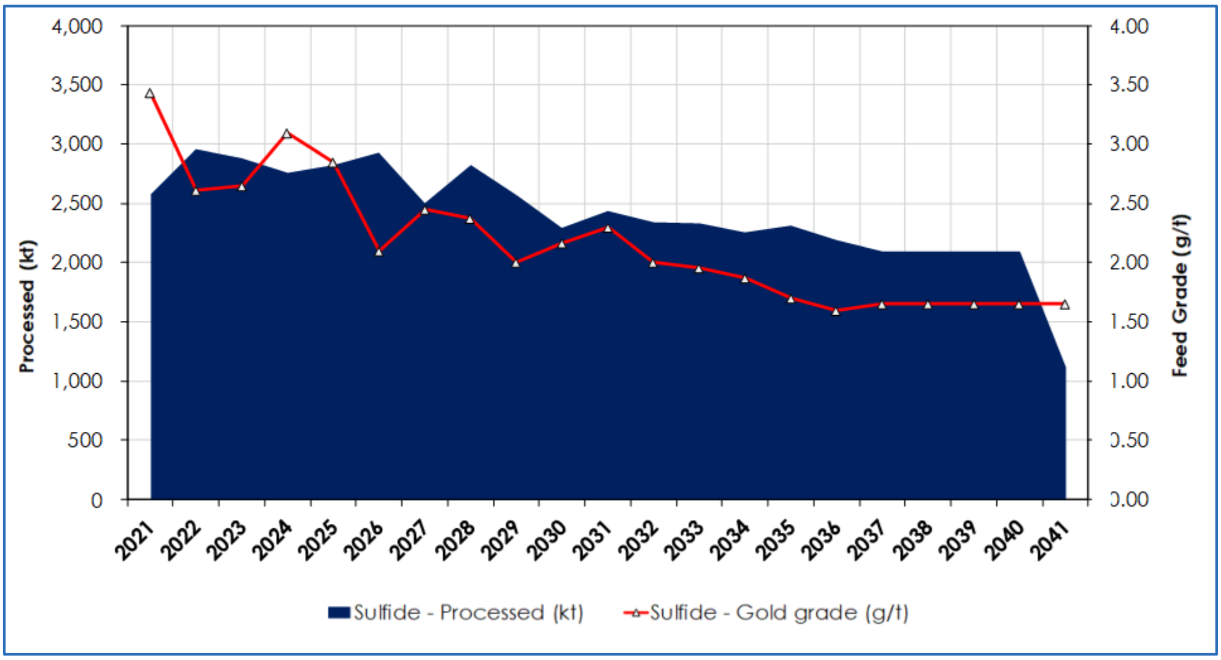


OreWin, 2020

Table 1.7 CDMP20 Reserve Case Before and After-Tax NPV

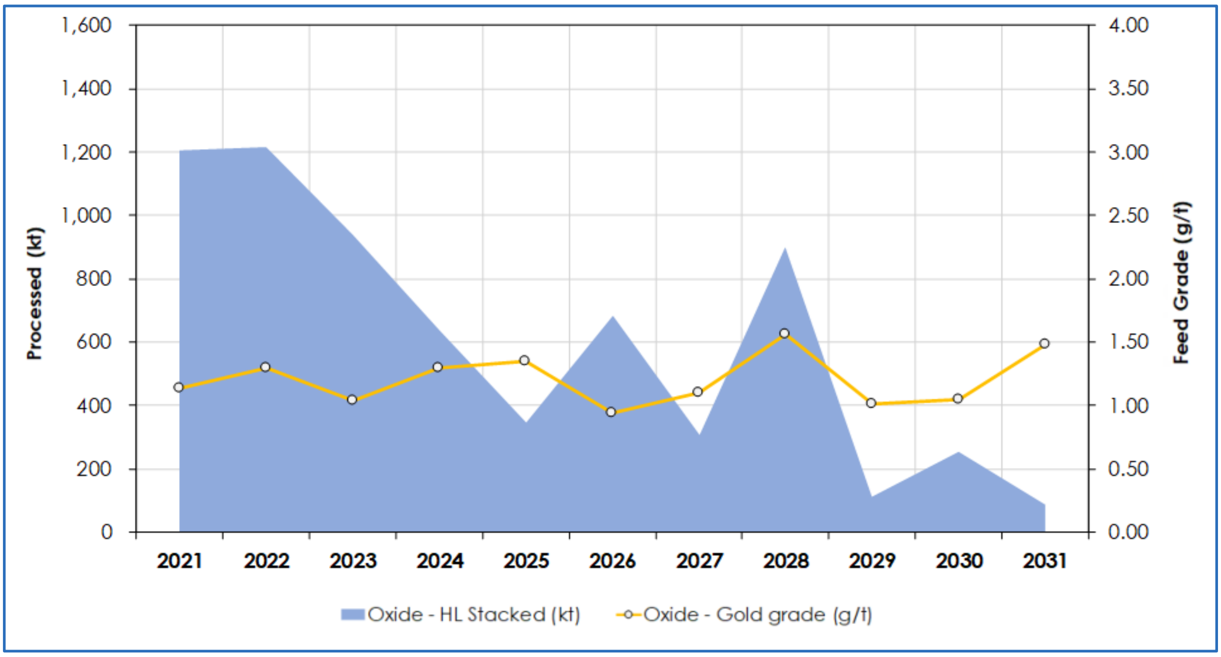
Discount Rate	Before-Tax NPV ($M)	After-Tax NPV ($M)
Undiscounted	2,397	2,306
5%	1,791	1,733
10%	1,434	1,393
12%	1,332	1,295

Figure 1.12 Reserve Case Sulfide Processing



OreWin, 2020

Figure 1.13 Reserve Case Oxide Heap Leach Processing



OreWin, 2020

Table 1.8 CDMP20 Reserve Case Cash Costs

Description	Units	Reserve Case
Mining and Rehandle	$M	420
Process, Freight, and Refining	$M	1,633
Site Support	$M	400
Royalties	$M	232
Total Production Costs	$M	2,686
Production Costs	$/oz gold	748
Sustaining Capital	$M	430
Fixed Lease Payments	$M	201
Exploration Cost - Sustaining	$M	14
Site G&A	$M	61
Total All-in Sustaining Costs	$M	3,392
All-in Sustaining Costs	$/oz gold	945

Process, Freight, and Refining includes by-product credits and excludes fixed lease costs
Royalties are calculated in the period incurred

A financial model was prepared using the Reserve Case production schedule and operating and capital assumptions on an annual basis. The assumptions for taxes and royalties were provided by SSR Mining. The corporate tax rate in Turkey is 22% for 2020 but reverts to 20% from 2021. The royalty rate for precious metals under Turkish Mining Law is variable and tied to metal prices. As Çöpler ores are processed on site, the applicable royalty rate is subject to a further 40% reduction for certain qualifying operating costs. The average royalty calculated as a proportion of gross revenue in the Reserve Case is approximately 4.2%.

Metal prices were estimated after analysis of consensus industry metal price forecasts and metal prices used in other comparable studies. The prices used for the economic analysis are shown in Table 1.9.

Table 1.9 CDMP20 Reserve Case Metal Price Assumptions

Metal	Unit	Metal Price by Year						
		Average	Q4'20	2021	2022	2023	2024	2025
Gold	$/oz	1,658	1,850	1,965	1,835	1,745	1,645	1,585
Silver	$/oz	21.55	20.05	24.15	22.70	21.80	20.75	20.25
Copper	$/lb	2.95	2.70	2.90	2.90	2.95	3.00	3.05

The estimates of cash flows have been prepared on a real basis with a base date of Q4'20 and a mid-year discounting is used to calculate NPV. All monetary figures have a base date of Q4'20 with no allowance for escalation and are expressed in US dollars (US$) unless otherwise stated. Production costs and AISCs are determined on a per ounce gold produced basis and do not consider the application of inventory movements or deferred waste stripping. Production costs do not equate to cash costs prepared under SSR Mining non-GAAP measures. AISCs do not equate to AISCs prepared under SSR Mining non-GAAP measures.

1.20 CDMP20 PEA Case

The CDMP20 also includes a Preliminary Economic Assessment (PEA) on an expanded Çöpler project that includes the new predominantly-oxide Ardich deposit. The PEA Case analyses inclusion of production from Ardich and reflects the increased capital costs and infrastructure required to incorporate the new deposit. The PEA Case is a whole-of-project analysis that represents a significant change from the Reserve Case economics analysis results and production.

The Ardich deposit is a newly discovered deposit that is separate to the other deposits on the property. Drilling is continuing at the Ardich deposit and it is expected that the drilling will further define the Mineral Resource. The development of Ardich requires development of a new open pit that is approximately 6 km east of the current Çöpler pit and 1 km north of the Çakmaktepe pits. A location plan showing site and the boundaries of the Reserve Case and the PEA Case are shown in Figure 1.2.

The Inferred Mineral Resource from Ardich is included in the PEA Case. The Ardich oxide Mineral Resources in the PEA Case represents a 226% increase in production tonnage at a grade that is 38% higher than the Çöpler oxide heap leach processing grades. The PEA Case includes assumptions for separate capital, infrastructure, and permitting that will be required to develop the Ardich Mineral Resources. The Ardich Mineral Resources were not included in the previous Technical Report and so the PEA represents a significant change in information.

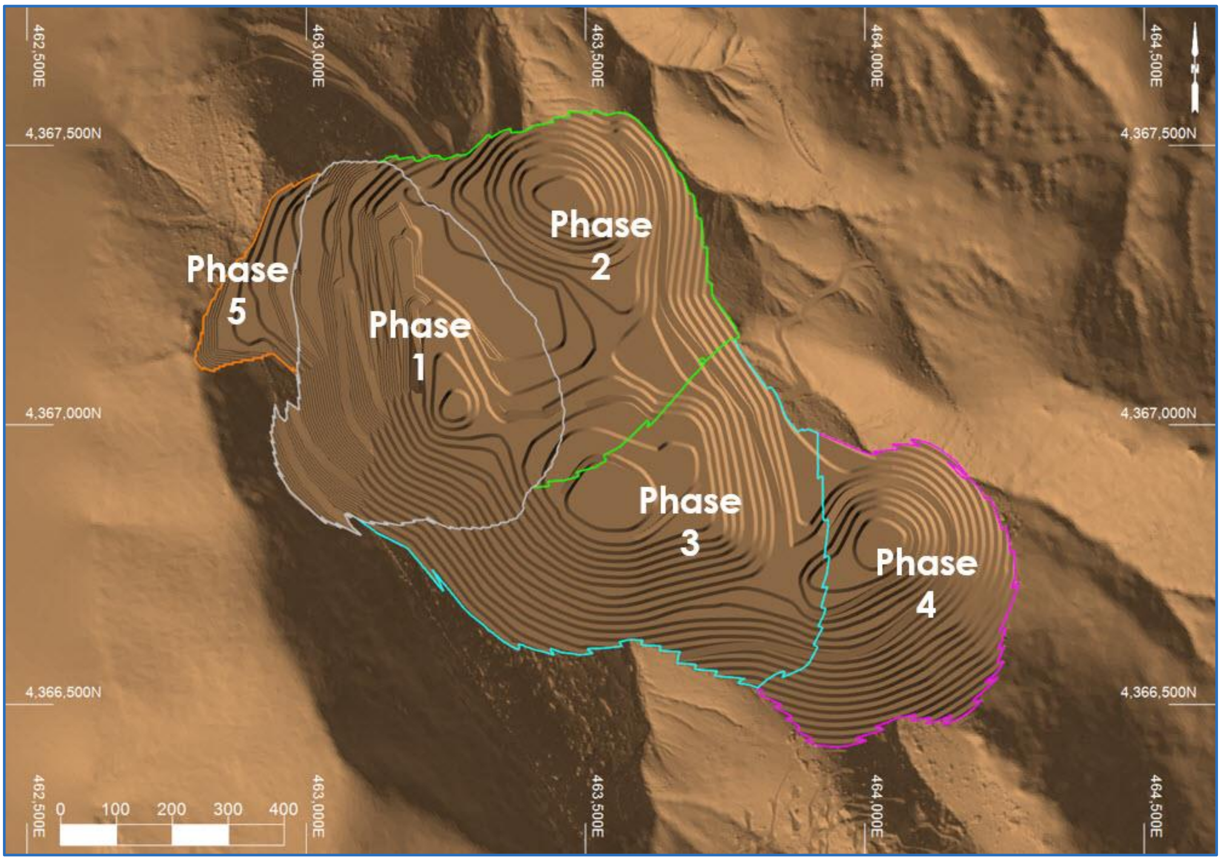
The PEA Case assumes Ardich oxide to be progressively stacked on the Çöpler heap. Ardich sulfide has been assumed to be placed in stockpile and treated once all the Çöpler sulfide has been treated.

The PEA Case production is 79.1 Mt at 2.13 g/t Au. The gold production in the PEA Case is 4.6 Moz. The increase in total production in the PEA Case is due to the addition of 20.3 Mt at 2.18 g/t Au from Ardich Mineral Resources. Like the Reserve Case, all mining is completed by 2032 in the PEA Case, oxide heap leach stacking is completed in 2031, while sulfide processing continues from stockpiles until 2042. The PEA Case shows an after-tax NPV at a 5% discount rate of $2.16 billion and the average AISC is $893/oz gold. The PEA Case is cash positive in each year of the mine plan.

The PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

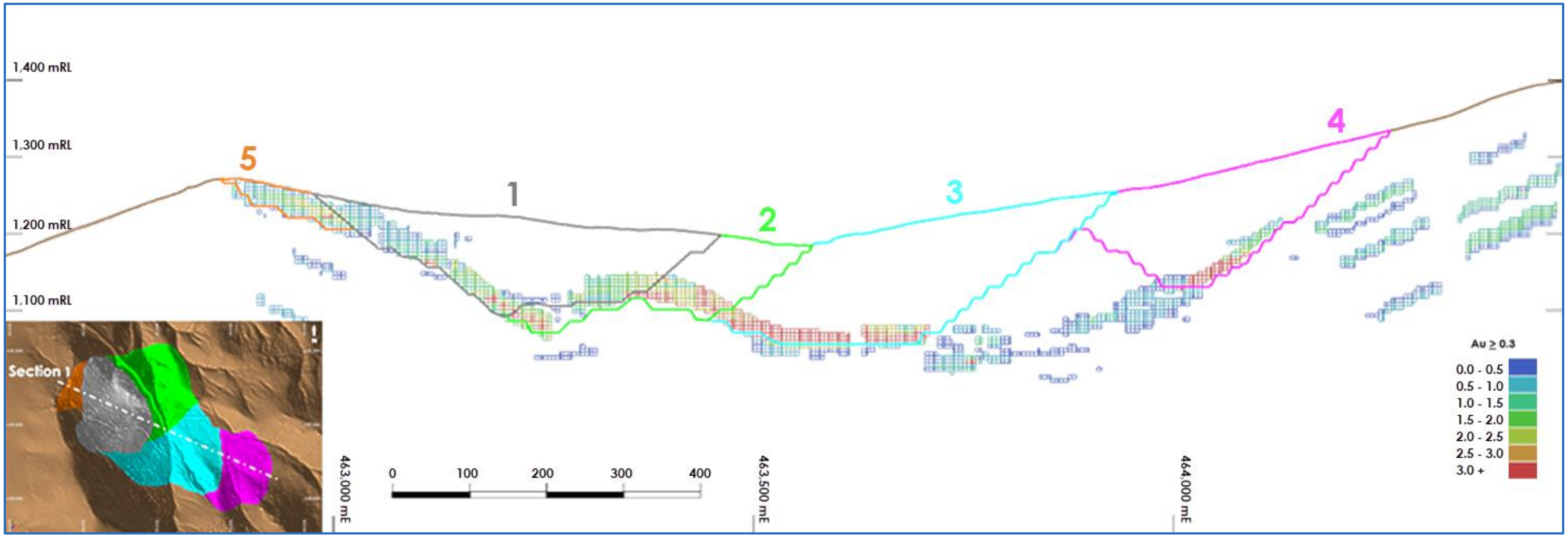
The PEA Case assumes that open pit mining is undertaken at Ardich using excavators and trucks and operated by a mining contractor, as is the case at the Çöpler mine. The Ardich production is primarily from oxide Mineral Resource. The pit has been split into five (5) phases for production scheduling. A plan and a long-section of the Ardich pit phases are shown in Figure 1.14 and Figure 1.15. Phase 5 is based on mostly Inferred Mineral Resource and although it is close to the surface and next to phase 1, phase 5 has been delayed to the end of the Ardich schedule so that the influence of the Inferred Mineral Resource is reduced. The phase 5 area has been targeted for resource definition drilling to improve the confidence in the estimates of grade in that area.

Figure 1.14 Ardich PEA Pit Design (plan view, looking down)



Anagold, 2020

Figure 1.15 Ardich Pit Phases (long-section 1, looking north-east)



OreWin, 2020

The key results of the PEA Case economic analysis are shown in Table 1.10. The after-tax cash flow is shown in Figure 1.16. The sulfide and oxide production profiles are shown in Figure 1.17 and Figure 1.18 respectively. The NPV results for before and after-tax over a range of discount rates is shown in Table 1.11. Gold unit costs are shown in Table 1.12.

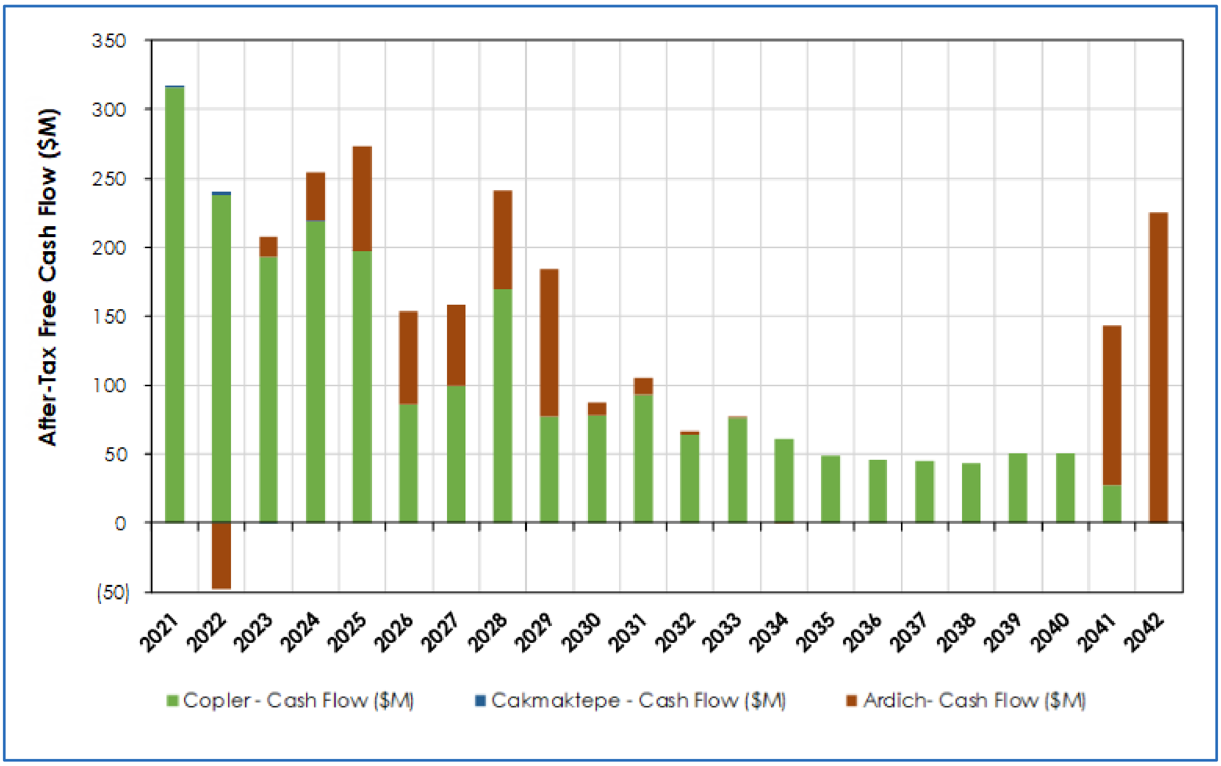
Table 1.10 CDMP20 PEA Case Results Summary

Item	Unit	PEA Case
Oxide Processed		
Heap Leach Quantity	kt	25,008
Au Feed Grade	g/t	1.69
Sulfide Processed		
Quantity Milled	kt	54,073
Au Feed Grade	g/t	2.33
Total Gold Produced		
Oxide – Gold	koz	956
Sulfide – Gold	koz	3,691
Total – Gold	koz	4,646
Oxide – Gold Recovery	%	68
Sulfide – Gold Recovery	%	91
5-Year Annual Average		
Average Gold Produced	kozpa	306
Free Cash Flow	$Mpa	249
Production Costs	$/oz gold	701
All-in Sustaining Costs	$/oz gold	886
Key Financial Results		
Production Costs	$/oz gold	726
All-in Sustaining Costs	$/oz gold	893
Site Operating Costs	$/t treated	42.87
After-Tax $NPV_{5\%}$	$M	2,164
Mine Life	years	22

5-Year annual average is for the period 1 January 2021 through 31 December 2025

OreWin

Figure 1.16 CDMP20 PEA Case After-Tax Cash Flow

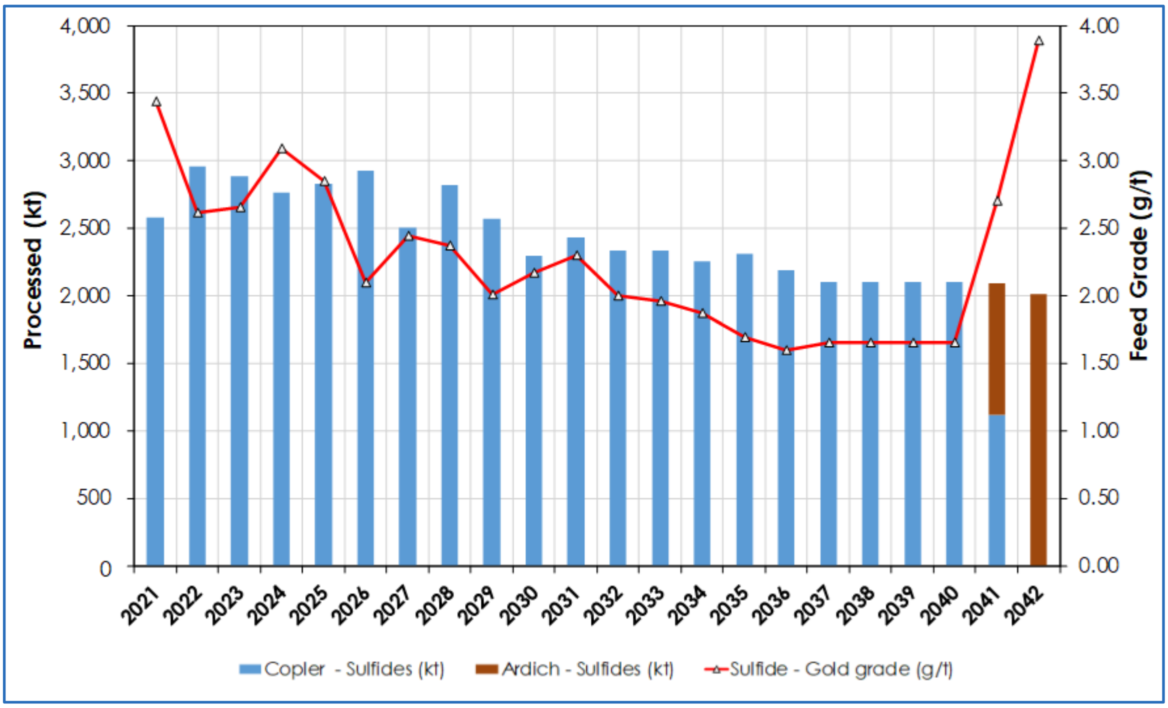


OreWin, 2020

Table 1.11 CDMP20 PEA Case Before and After-Tax NPV

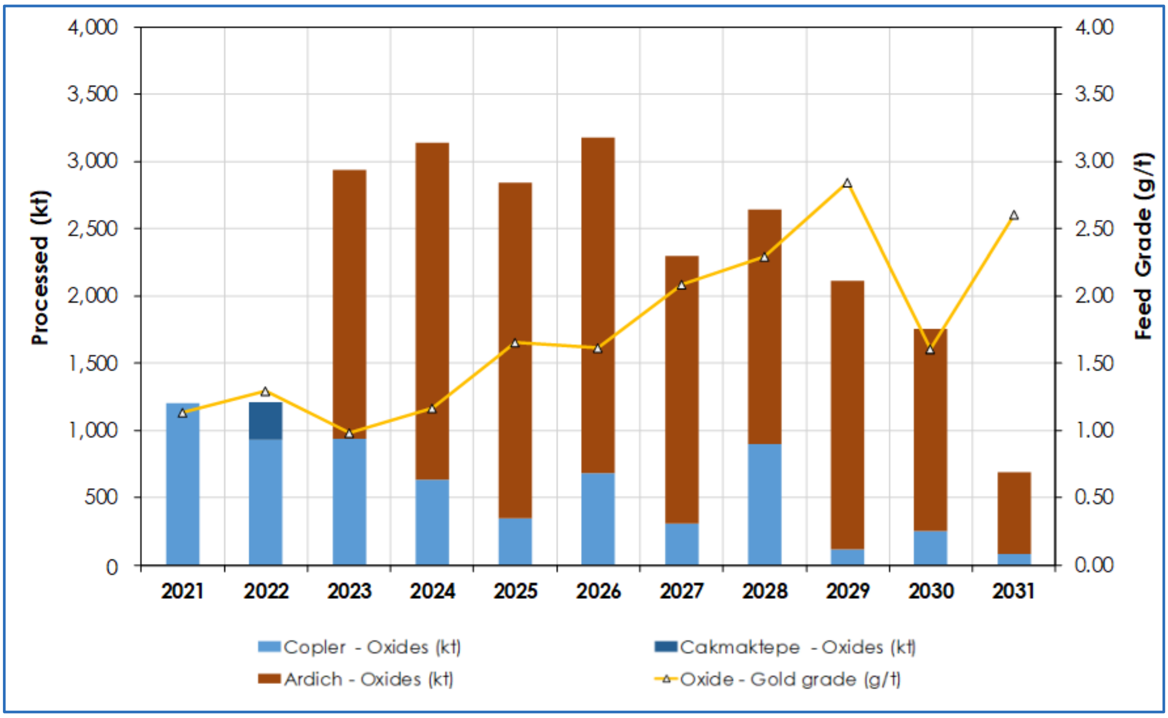
Discount Rate	Before-Tax NPV ($M)	After-Tax NPV ($M)
Undiscounted	3,312	3,033
5%	2,310	2,164
10%	1,767	1,680
12%	1,617	1,543

Figure 1.17 PEA Case Sulfide Production



OreWin, 2020

Figure 1.18 PEA Case Oxide Production



OreWin, 2020

Table 1.12 CDMP20 PEA Case Cash Costs

Description	Units	PEA Case
Mining and Rehandle	$M	727
Process, Freight, and Refining	$M	1,859
Site Support	$M	442
Royalties	$M	346
Total Production Costs	$M	3,374
Production Costs	$/oz gold	726
Sustaining Capital	$M	479
Fixed Lease Payments	$M	211
Exploration Cost – Sustaining	$M	18
Site G&A	$M	67
Total All-in Sustaining Costs	$M	4,150
All-in Sustaining Costs	$/oz gold	893

Process, Freight, and Refining includes by-product credits and excludes fixed lease costs
Royalties are calculated in the period incurred

The PEA Case is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Costs, taxation, and royalties used in the PEA Case were the same as in the Reserve Case. The prices used for the economic analysis are shown in Table 1.13.

Table 1.13 Metal Price Assumptions

Metal	Unit	Metal Price by Year						
		Average	Q4'20	2021	2022	2023	2024	2025
Gold	$/oz	1,644	1,850	1,965	1,835	1,745	1,645	1,585
Silver	$/oz	21.55	20.05	24.15	22.70	21.80	20.75	20.25
Copper	$/lb	2.95	2.70	2.90	2.90	2.95	3.00	3.05

The estimates of cash flows have been prepared on a real basis with a base date of Q4'20 and a mid-year discounting is used to calculate NPV. All monetary figures have a base date of Q4'20 with no allowance for escalation and are expressed in US dollars (US$) unless otherwise stated. Production costs and AISCs are determined on a per ounce gold produced basis and do not consider the application of inventory movements or deferred waste stripping. Production costs do not equate to cash costs prepared under SSR Mining non-GAAP measures. AISCs do not equate to AISCs prepared under SSR Mining non-GAAP measures.

1.21 CDMP20 Comparison with 2016 Technical Report

A comparison of gold production in the CDMP20 cases and the 2016 Technical Report was prepared. Figure 1.19 details the Reserve Case gold production, the 2016 Technical Report, and actual / near-term estimates. The 2020 production is based on actual for Q1'20 through Q3'20 and a forecast estimate for Q4'20. Figure 1.20 shows the incremental change in gold production in the PEA Case from the addition of Ardich.

Actual gold production from the Çöpler project matched with the 2016 Technical Report for 2016 and 2017, while a large increase was experienced in 2019. Projections for 2020–2021 are again forecast to outperform the 2016 Technical Report gold production. The Reserve Case metal production is very similar to the 2016 Technical Report profile, with only a small dip expected in 2022–2023, gains in 2024–2033, and an extension to the tail, overall adding 0.69 Moz of total gold production relative to the 2016 Technical Report (2021–LOM). The PEA Case removes much of the 2022–2023 dip in gold production, and strongly outperforms the 2016 Technical Report from 2024 through 2031. The PEA Case also adds a further extension to the tail gold production, adding 1.06 Moz total gold relative to the Reserve Case (1.75 Moz relative to the 2016 Technical Report).

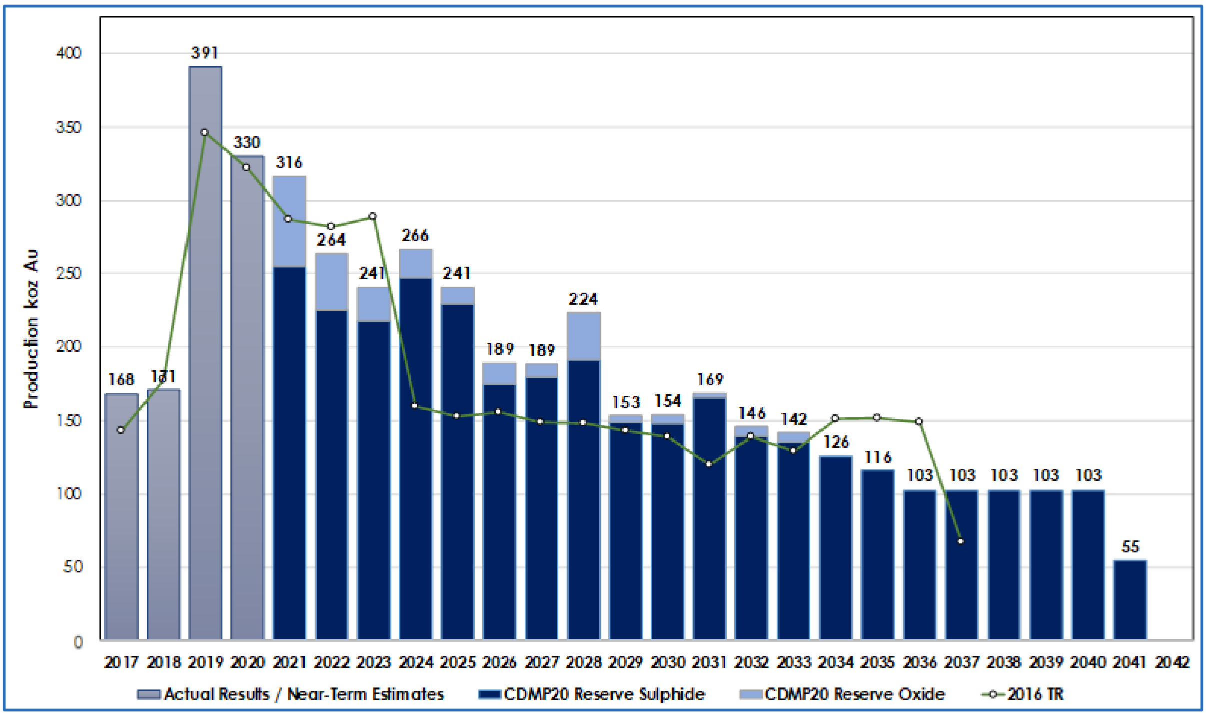
The PEA Case is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

1.22 Interpretation and Conclusions

The CDMP20 has identified additional Mineral Resources and additional Mineral Reserves when compared to prior studies.

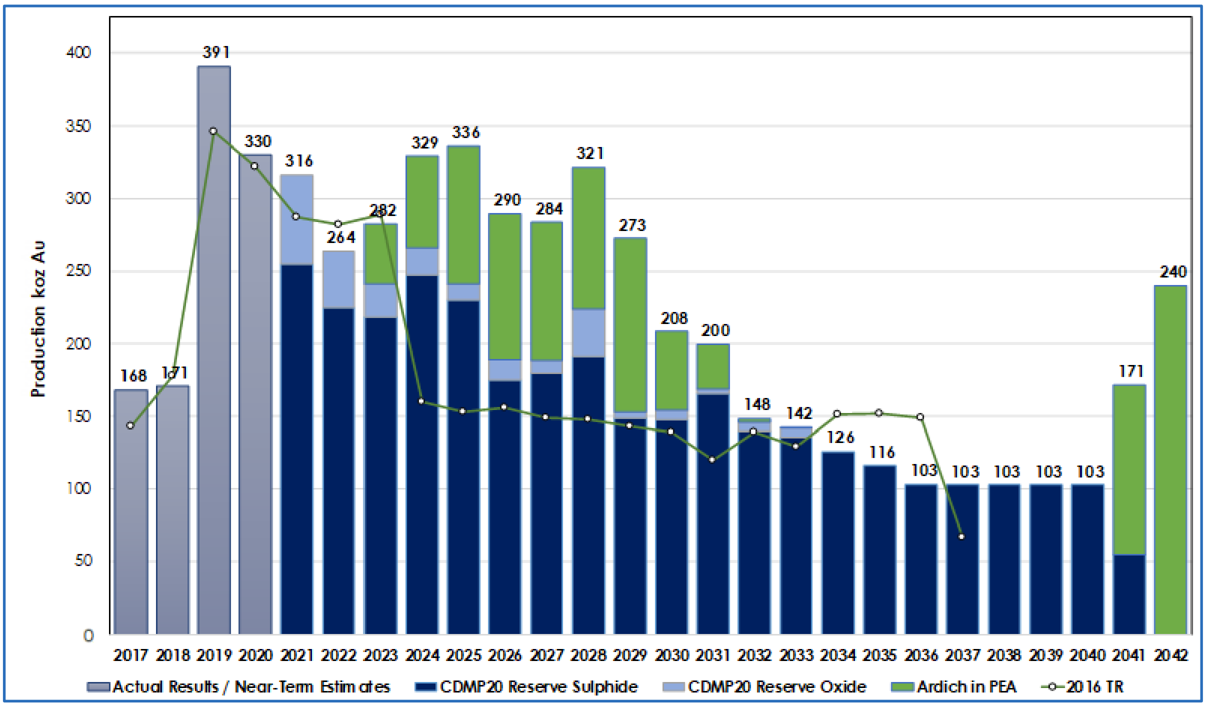
The PEA Case has indicated that there is significant potential value in the Ardich Mineral Resource that may provide an opportunity for SSR Mining to increase gold production. The PEA Case is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Figure 1.19 CDMP20 Reserve Case Gold Production



OreWin, 2020

Figure 1.20 CDMP20 PEA Case Gold Production



OreWin, 2020

1.23 Recommendations

Key recommendations from the CDMP20 are:

- Continue to update and evaluate the Çöpler District Master Plan as the existing Mineral Resources and Mineral Reserves are updated and as new prospects are advanced.
- Re-design of Çöpler pits at updated metal prices.
- Geotechnical review and study of the re-evaluation of the re-designs.
- Optimisation of the sulfide flotation circuit, POX and process operation.
- Metallurgical testwork on future oxide and sulfide ore sources
- Optimisation of the oxide heap leach circuit.
- Optimisation of the mining rates to increase gold production.
- Stockpile reconciliation and management studies.
- Review and adapt the ore control and stockpiling strategies to maximise gold production.
- Continue drilling at Ardich.
- Geotechnical studies of Ardich.
- Reconciliation studies of Çöpler.
- Update Çöpler and Ardich resource models and estimates.
- Further study of PEA Case and advance to next stage of study:
  - Geotechnical studies
  - EIA and permitting
  - Blasting studies
  - Metallurgical studies

2 INTRODUCTION

2.1 SSR Mining Inc.

The Çöpler project is owned and operated by Anagold Madencilik Sanayi ve Ticaret Anonim Şirketi (Anagold). SSR Mining controls 80% of the shares of Anagold, Lidya Madencilik Sanayi ve Ticaret A.Ş. (Lidya), controls 18.5%, and a bank wholly-owned by Çalık Holdings A.Ş., holds the remaining 1.5%.

Exploration tenures surrounding the project area and mining at Çakmaktepe are subject to joint venture agreements between SSR Mining and Lidya that have varying interest proportions. SSR Mining controls 50% of the shares of Kartaltepe Madencilik Sanayi ve Ticaret Anonim Şirketi (Kartaltepe) and 50% of Tunçpinar Madencilik Sanayi ve Ticaret Anonim Şirketi (Tunçpinar). The other 50% of both is controlled by Lidya. More than 96% of the Mineral Resource is located on the Anagold owned 80% ground, with the remainder of the mineralisation within the 50%/50% ownership boundary.

In most cases, the parent company will be referred to as SSR Mining throughout this Technical Report even though it may have been Alacer or Anatolia at the time referenced in the report. Anagold remains the operating company for the Çöpler project and is the entity that undertakes the day-to-day work for the project.

2.2 Terms of Reference and Purpose of the Report

The CDMP20 is an Independent Technical Report on the Çöpler project, prepared for SSR Mining as part of the strategy for expansion of the Çöpler project. The CDMP20 was prepared by OreWin Pty Ltd (OreWin), working with SSR Mining, Anagold, and their consultants.

This Report uses metric measurements except where otherwise noted. The currency used is US dollars (US$) unless otherwise stated.

2.3 Qualified Persons

The following people served as the Qualified Persons (QPs) as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101F1:

- Bernard Peters, BEng (Mining), FAusIMM (201743), employed by OreWin Pty Ltd as Technical Director – Mining, was responsible for the overall preparation of the CDMP20 and, the Mineral Reserve estimates, Sections 1 to 4; Sections 5 and 6; Section 13; Sections 15 to 27.
- Sharron Sylvester, BSc (Geol), RPGeo AIG (10125), employed by OreWin Pty Ltd as Technical Director – Geology, was responsible for the preparation of the Mineral Resources, Sections 1 to 4; Section 7 to 12; Section 14; Sections 25 to 27.

2.4 Site Visits and Scope of Personal Inspection

Site visits were performed as follows:

- Mr Bernard Peters visited the project 13–17 May 2019, 15–21 September 2019, 14–18 October 2019, 18–21 November 2019, and 27 February to 4 March 2020. The site visits included briefings from geology and exploration, mine, processing, environmental, permitting and corporate personnel, site inspections of current and future areas for mining and plant and infrastructure, and discussions with other consultants. In addition, several visits to SSR Mining's head office in Denver Colorado were undertaken during the same timeframe for the purpose of project-related meetings.
- Sharron Sylvester visited the project 13–17 May 2019, 15–21 September 2019, 14–18 October 2019, 18–21 November 2019, and 27 February to 4 March 2020. The site visits included briefings from geology and exploration, mine, processing, environmental, permitting and corporate personnel, site inspections of current and future areas for mining and plant and infrastructure, and discussions with other consultants. In addition, several visits to SSR Mining's head office in Denver, Colorado were undertaken during the same timeframe for the purpose of project-related meetings. Visits to analytical laboratories were planned to be undertaken but not completed due to global travel restrictions related to Covid-19.

2.5 Effective Dates

The report has a number of effective dates, as follows:

- Effective date of the Report: 27 November 2020.
- Date of drillhole database close-out for the Çöpler Mineral Resource estimate: 15 July 2015.
- Date of drillhole database close-out for the Çakmaktepe Mineral Resource estimate: 31 October 2019.
- Date of drillhole database close-out for the Ardich Mineral Resource estimate: 13 February 2020.
- Effective date of Mineral Resource update for mineralisation amenable to open pit mining methods: 27 November 2020.
- Effective date of Mineral Reserves: 27 November 2020.

2.6 Information Sources and References

The reports and documents listed in Section 6.1 (Previous Technical Reports), Section 3 (Reliance on Other Experts), and Section 27 (References) of this report were used to support the preparation of the report. Additional information was sought from SSR Mining personnel where required.

3 RELIANCE ON OTHER EXPERTS

3.1 Mineral Reserve

The Mineral Reserves were developed based on mine planning work completed in October 2020 and estimated based on the end-of-August 2020 topography surface.

Çöpler oxide ore cut-off grades vary from 0.47–0.59 g/t Au. Çöpler sulfide ore cut-off grade is 1.05 g/t Au. Çakmaktepe oxide cut-off grades vary from 0.52–0.69 g/t Au. There is no Çakmaktepe sulfide Mineral Reserve. Average oxide gold recoveries are 73% and average sulfide ore recoveries are 91%.

The cut-off grades for the Mineral Reserves estimates were determined using a gold price of $1,350/oz. There are no credits for silver or copper in the cut-off grade calculations. Economic analysis has been carried out using long-term metal prices of $1,585/oz gold, $20.25/oz silver, and $3.05/lb copper, and average metal prices of $1,658/oz gold, $21.55/oz silver, and $2.95/lb copper.

The Mineral Reserves statement is shown in Table 3.1. Mineral Reserves have been classified using the 2014 CIM Definition Standards (CIM, 2014) and were estimated by Bernard Peters BEng (Mining), FAusIMM (201743), employed by OreWin Pty Ltd as Technical Director – Mining. Mineral Reserves are presented on a project basis and have an effective date of 27 November 2020.

The CDMP20 Reserve Case is at a feasibility level of study. The Mineral Resource estimates that underpin the Mineral Reserves have been reported in the CDMP20 inclusive of dilution. Measured Mineral Resources were converted to Proven Mineral Reserves and Indicated Mineral Resources were converted to Probable Mineral Reserves. Inferred Mineral Resources were treated as waste and were not converted to Mineral Reserve. The CDMP20 Mineral Reserves have been demonstrated to be viable.

Table 3.1 CDMP20 Mineral Reserves Summary

CDMP20 Mineral Reserves Summary (as at the Effective Date)							
Classification	Tonnage (kt)	Grades			Contained Metal		
		Au (g/t)	Ag (g/t)	Cu (%)	Gold (koz)	Silver (koz)	Copper (klb)
Çöpler Mine – Oxide							
Proven Mineral Reserve	230	1.23	8.97	0.06	9	66	294
Probable Mineral Reserve	7,364	1.23	6.16	0.13	290	1,458	20,549
Probable – Stockpile	–	–	–	–	–	–	–
Total Mineral Reserve	7,595	1.23	6.24	0.12	299	1,525	20,843
Çöpler Mine – Sulfide							
Proven Mineral Reserve	2,140	2.42	7.63	–	166	525	–
Probable Mineral Reserve	42,461	2.18	5.73	–	2,970	7,819	–
Probable – Stockpile	6,674	2.63	–	–	564	–	–
Total Mineral Reserve	51,274	2.24	5.06	–	3,700	8,344	–
Çakmaktepe Mine – Oxide							
Proven Mineral Reserve	–	–	–	–	–	–	–
Probable Mineral Reserve	274	1.26	10.91	–	11	96	–
Probable – Stockpile	11	2.69	–	–	1	–	–
Total Mineral Reserve	285	1.32	10.49	–	12	96	–
CDMP20 – Oxide Reserve							
Proven Mineral Reserve	230	1.23	8.97	0.06	9	66	294
Probable Mineral Reserve	7,638	1.23	6.33	0.13	301	1,554	20,549
Probable – Stockpile	11	2.69	–	–	1	–	–
Total Mineral Reserve	7,879	1.23	6.40	0.12	311	1,621	20,843
CDMP20 – Sulfide Reserve							
Proven Mineral Reserve	2,140	2.42	7.63	–	166	525	–
Probable Mineral Reserve	42,461	2.18	5.73	–	2,970	7,819	–
Probable – Stockpile	6,674	2.63	–	–	564	–	–
Total Mineral Reserve	51,274	2.24	5.06	–	3,700	8,344	–
CDMP20 Mineral Reserves Total							
Proven Mineral Reserve	2,370	2.30	7.76	0.01	175	591	294
Probable Mineral Reserve	50,099	2.03	5.82	0.02	3,271	9,373	20,549
Probable – Stockpile	6,685	2.63	–	–	565	–	–
Total Mineral Reserve	59,154	2.11	5.24	0.02	4,011	9,964	20,843

1. Effective date of the CDMP20 Mineral Reserve is 27 November 2020.
2. The Mineral Reserves were developed based on mine planning work completed in October 2020 and estimated based on End of August 2020 topography surface.
3. Mineral Reserve cut-offs are based on a gold price of $1,350/oz; average oxide recoveries are 73% and average sulfide recoveries are 91%.
4. Çöpler oxide cut-off grades 0.47–0.59 g/t Au, Çöpler sulfide cut-off grade 1.05 g/t Au, Çakmaktepe oxide cut-off grades 0.52–0.71 g/t Au; all cut-off grades include allowance for royalty payable. There are no credits for silver or copper in the cut-off grade calculations. There is no Çakmaktepe Sulfide Mineral Reserve.
5. Economic analysis has been carried out using a long-term gold price of $1,585/oz. The economic analysis has used a Q4'20 start date.
6. Mineral Reserves tabulated include 403 kt at 2.47 g/t Au from the mine plan scheduled for September 2020.
7. Totals may vary due to rounding.

3.2 Mineral Tenure, Surface Rights, and Ownership

The QPs have not independently reviewed ownership of the project area and any underlying property agreements, mineral tenure, surface rights, or royalties. The QPs have fully relied upon, and disclaim responsibility for, information derived from SSR Mining and legal experts retained by SSR Mining for this information through the following document:

- Çöpler District Master Plan 2020 Feasibility Study Section 3 Property Title, Description, and Location, November 2020
- Biçer, İ., 2015: Mining Title Opinion of Turkish Legal Counsel: letter addressed to Anagold, from the legal firm Baycan Hukuk Bürosu, 26 April 2015, 8 p.
- Aritürk, R., 2020: Legal Opinion relating to permits and licenses: letter to Anagold, from the legal firm Elmadağ Attorneys & Counselors, 14 February 2020, 33 p.

This information is used in Section 4, Section 14, Section 15, and Section 22 of the CDMP20.

3.3 Environmental Studies, Permitting, Social and Community Impact

The QPs have fully relied upon, and disclaim responsibility for, information supplied by SSR Mining and experts retained by SSR Mining for information relating to the status of the current royalties and taxation regime for the project as follows:

- Çöpler District Master Plan 2020 Feasibility Study Section 12 Environmental Studies, Permitting, Social and Community, November 2020

This information is used in Section 4 and in Section 20 of the CDMP20.

3.4 Taxation and Royalties

The QPs have obtained information regarding the taxation and royalties for the project from information supplied by SSR Mining. The QPs have fully relied upon, and disclaim responsibility for, information derived from SSR Mining for this information through the following documents:

- Çöpler District Master Plan 2020 Feasibility Study Section 3 Property Title, Description, and Location, November 2020
- Çöpler District Master Plan 2020 Feasibility Study Section 14 Economic Analysis, November 2020

This information is used in Section 4 and in Section 22 of the CDMP20.

3.5 Marketing

The QPs have obtained information regarding the marketing status of the Project from information supplied by SSR Mining. The QPs have fully relied upon, and disclaim responsibility for, information derived from SSR Mining for this information through the following documents:

- Çöpler District Master Plan 2020 Feasibility Study Section 11 Market Studies and Contracts, November 2020

This information is used in Section 22 of the CDMP20.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Çöpler District Master Plan 2020 Technical Report (CDMP20) is an independent Technical Report prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for SSR Mining Inc. (SSR Mining), on the Çöpler project (the project), located in Turkey. The project consists of a number of mining licences covering Mineral Resources on the Çöpler, Çakmaktepe, Ardich, and Bayramdere deposits, Mineral Reserves on the Çöpler and Çakmaktepe open pit mines, oxide and sulfide processing facilities, and supporting infrastructure.

The Çöpler project is located in east-central Turkey, 120 km west of the city of Erzincan, in Erzincan Province, 40 km east of the iron-mining city of Divriği (one-hour drive), and 550 km east of Turkey's capital city, Ankara. The nearest urban centre, İliç, (approximate population 3,800), is located approximately 6 km east of the current Çöpler pit. Figure 4.1 illustrates the location of the project within the country of Turkey and indicates the deposit's proximity to surrounding communities.

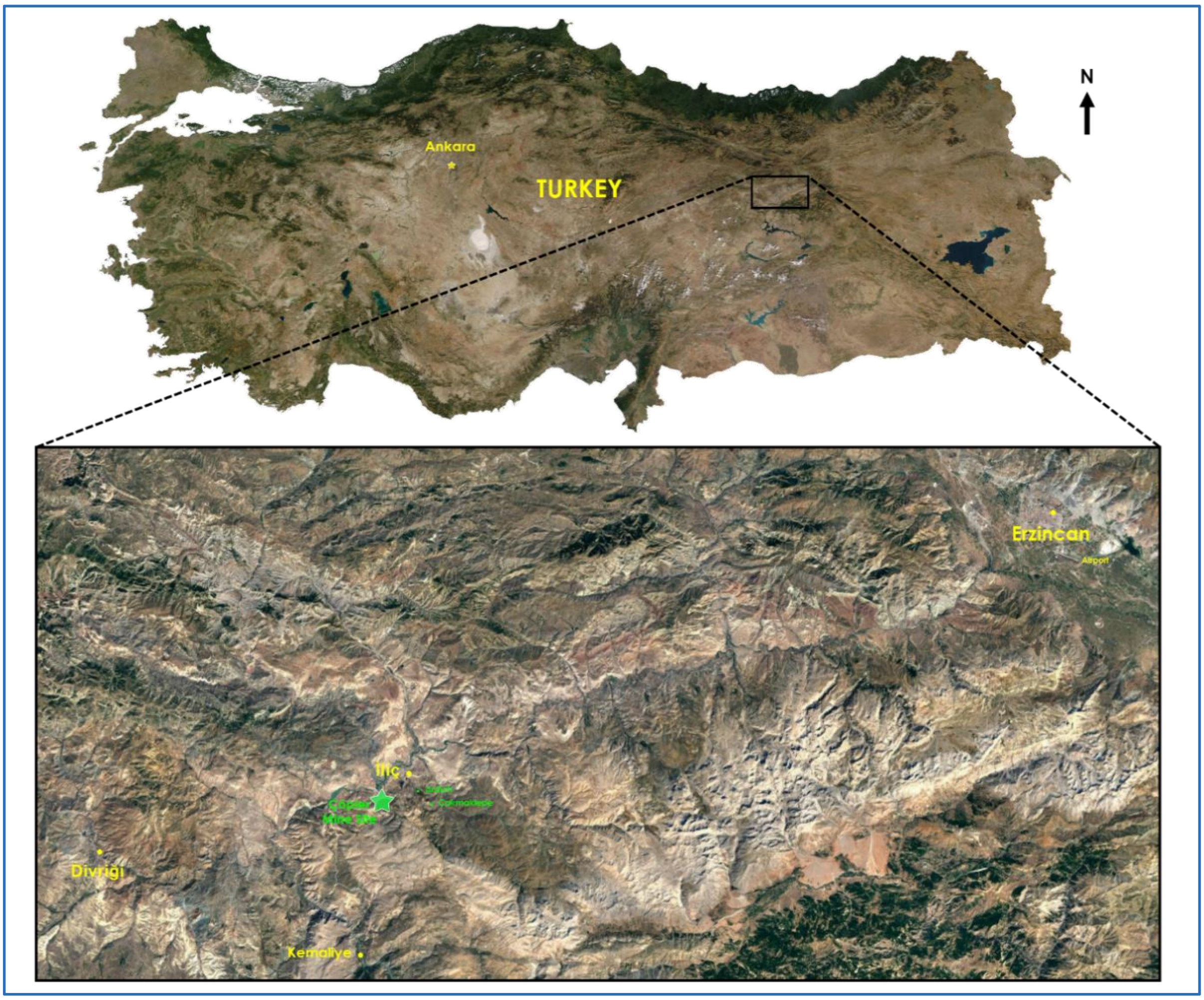
The Çöpler project uses the European 1950 (E1950) datum coordinate system, which is a Turkish Government requirement. The Çöpler deposit is located in UTM6 zone 37N of the E1950 coordinate system. The Çöpler project centroid is situated at approximately 459,975 mE and 4,364,420 mN and has an approximate elevation of 1,160 m above mean sea level (mamsl).

The Çöpler mining operations are located 900 m south-west of the İliç district centre, 650 m south of the Bahçe neighbourhood, 250 m south of the Çöpler village, and 180 m north of the Sabırlı village. The project site lies within the licence areas numbered 847, 49729, and 20067313 (Figure 4.2), which have been granted by the General Directorate of Mining and Petroleum Affairs (MAPEG).

The Çakmaktepe satellite mining operation is located 6 km east of the current Çöpler pit and 1.5 km south of İliç. The Çakmaktepe pits are located within Kartaltepe Licence 1054. Ore mined at Çakmaktepe is hauled and treated at the Çöpler facilities.

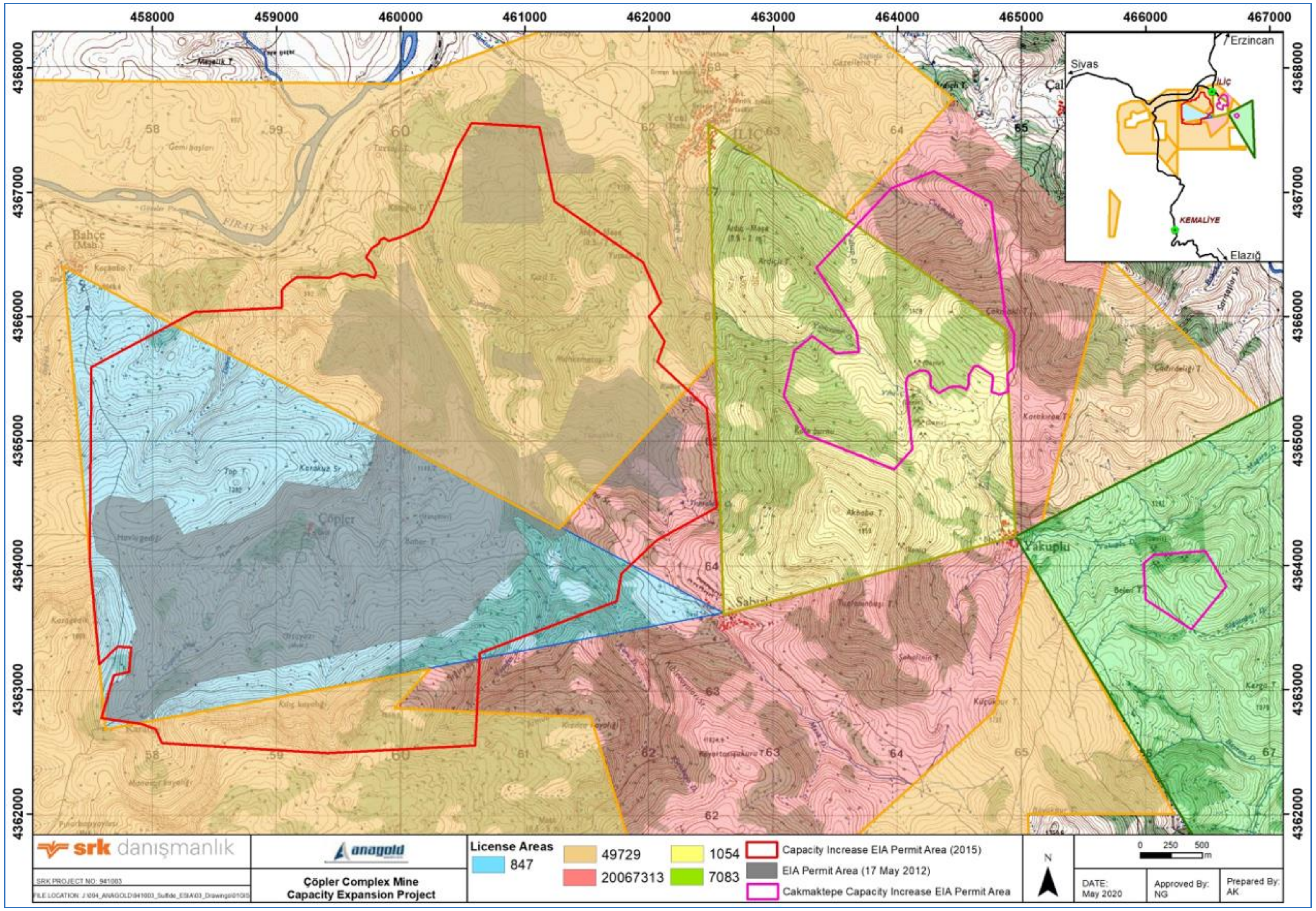
The currently permitted Environmental Impact Assessment (EIA) boundary incorporates 1,976 ha, whereas the footprint of the mine units covers a combined 1,097 ha.

Figure 4.1 Location of the Project



SSR Mining, 2020

Figure 4.2 Çöpler Project Licence and Surrounding Licences (UTM Grid)

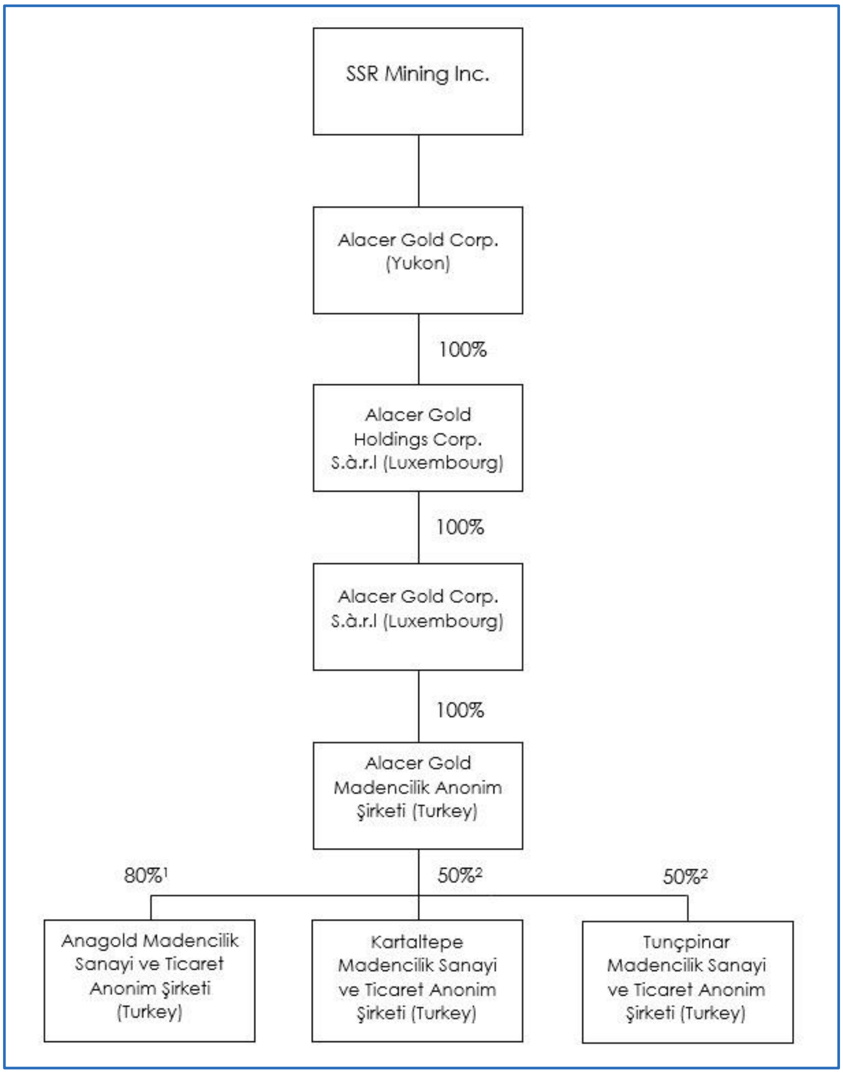


SSR Mining, 2020

4.2 Ownership

SSR Mining controls the Çöpler project through a series of companies that own the licence areas. The company structure that links SSR Mining to the Çöpler project is shown in Figure 4.3.

Figure 4.3 Ownership



[1] Lidya holds 18.5% of this entity and Bank Kombetare Tregtare SHA, a bank wholly-owned by Çalık Holdings A.Ş., holds the remaining 1.5%.
[2] Lidya holds the remaining 50% of the entity.

The Çöpler project is owned and operated by Anagold Madencilik Sanayi ve Ticaret Anonim Şirketi (Anagold). SSR Mining controls 80% of the shares of Anagold, Lidya Madencilik Sanayi ve Ticaret A.Ş. (Lidya), controls 18.5%, and a bank wholly-owned by Çalık Holdings A.Ş., holds the remaining 1.5%.

Exploration tenures surrounding the project area and mining at Çakmaktepe are subject to joint venture agreements between SSR Mining and Lidya that have varying interest proportions. SSR Mining controls 50% of the shares of Kartaltepe Madencilik Sanayi ve Ticaret Anonim Şirketi (Kartaltepe) and 50% of Tunçpinar Madencilik Sanayi ve Ticaret Anonim Şirketi (Tunçpinar). The other 50% of both is controlled by Lidya.

More than 96% of the Mineral Resources are located on the Anagold-held 80% ground, with the remainder of the mineralisation within the 50%/50% ownership boundary.

Kartaltepe sells Anagold ore from Çakmaktepe at an agreed rate. SSR Mining holds a 2% NSR on the Kartaltepe licences, receivable after repayment of a historical royalty pre-payment, which had an outstanding value of $1.8M as of 31 December 2019.

The Çöpler deposit, including the Mineral Resources and Mineral Reserves, is wholly-owned by Anagold. Çakmaktepe is wholly-owned by Kartaltepe. Ardich, Mavialtin, Bayramdere, Aslantepe, and Findiklidere have areas owned by both Anagold and Kartaltepe.

The PEA Case has only analysed Mineral Resources located on the Anagold licence.

4.3 Mineral Tenure

Anagold holds the exclusive right to engage in mining activities within the Çöpler project area. Anagold holds six granted licences (Table 4.1) covering a combined area of approximately 16,600 ha. Mineral title is held in the name of Anagold. Kartaltepe holds eight licences covering approximately 9,200 ha. The total near-mine tenement package is approximately 25,800 ha. Anagold currently holds sufficient surface rights to allow continued operation of the mining operation in the Reserve Case. The major licence boundaries are shown in Figure 4.4.

The granted licences include two clay borrow pit licences, being 76817 and 76818.

The Çöpler mine and associated infrastructure are hosted within the triangular-shaped concession 847. Anagold has applied for and received approval from the Mining Affairs Committee to be granted extensions to the three Anagold licences that have expired (76817, 76818, and 50237). Formal licence extension is awaiting approval from the Energy and Natural Resources Ministry. Anagold retains ownership of these licences pending formal approval.

Anagold has confirmed that charges and administrative expenses due to the Turkish Ministry of Energy and Natural Resources, Directorate General of Mining and Petroleum Affairs (MAPEG) have been paid, and all Anagold licences were in good standing as of November 2020.

The mined Çakmaktepe pits are all on Kartaltepe Licence 1054. Bayramdere prospect is on Kartaltepe Licence 7083. These licences are operational licences.

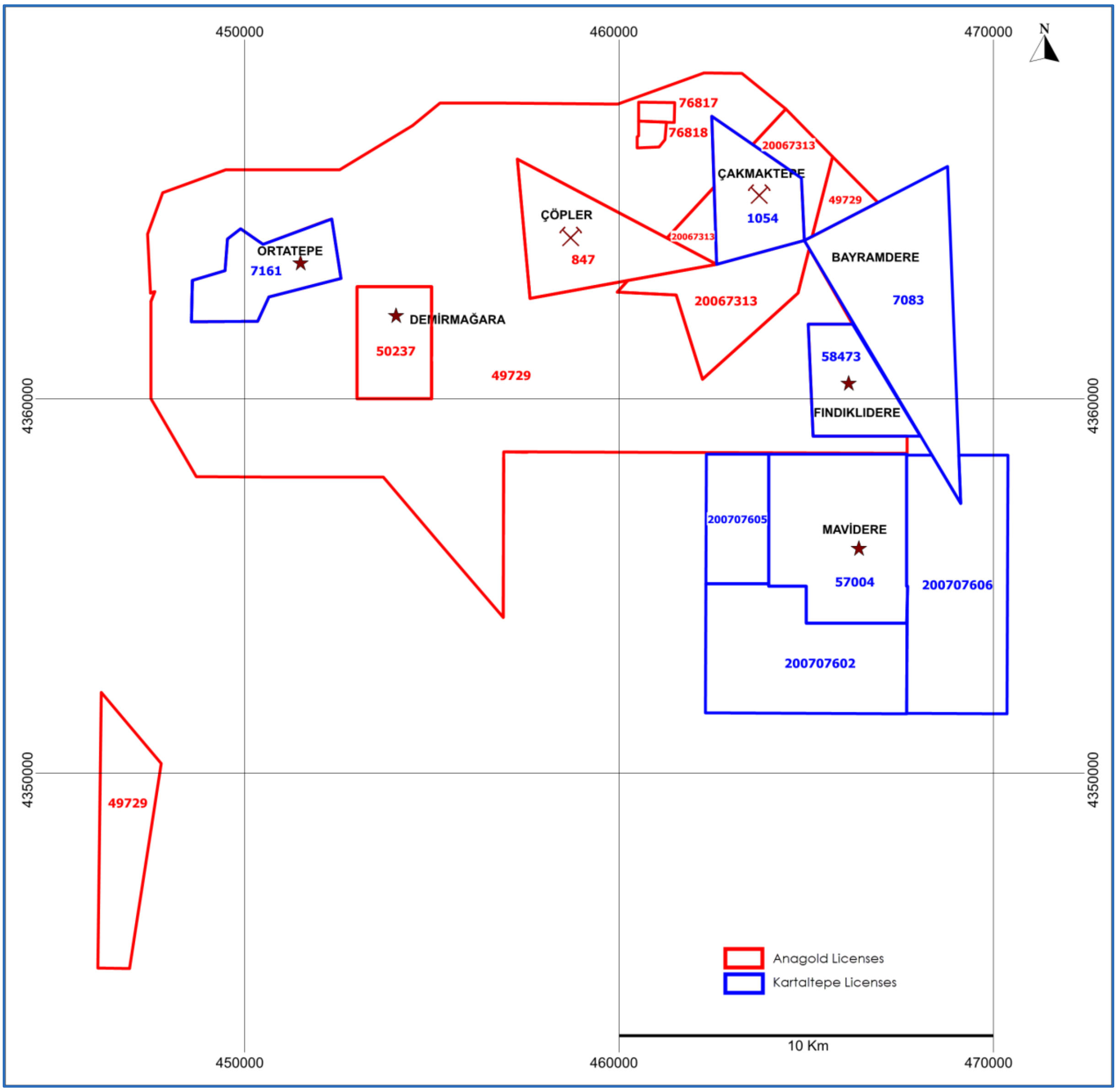
The three expired Kartaltepe licences (200707602, 200707605 and 200707606) were combined, and an operation project was prepared and submitted to receive an operation licence. The process continues. Kartaltepe maintains ownership of these licences during this process.

OreWin

Table 4.1 Granted Licences and Operating Permits

Province	Town	Village	Registration No.	Licence No.	Licence Area (ha)	Licence Type	Licence Group	Operation Permit	Operation Permit Area (ha)	Licence Issue Date	Licence Expiry Date	Licensee	Project
Erzincan	İliç	Çöpler	1027313	847	941.92	Operation	IV (Metallic)	Au+Ag+Cu+Hg Mn	Au+Ag+Cu+Hg: 941.92 Mn: 941.92	6/11/1986	6/11/2026	Anagold	Çöpler-Çöpler Saddle
Erzincan	İliç	Çöpler	2384036	49729	13,747.51	Operation	IV (Metallic)	Au+Ag+Cu+Mo	909.50	4/08/2016	4/08/2026	Anagold	Ardich-Çöpler Saddle-Kiziltepe-Meseburnu
Erzincan	İliç	Ortatepe	2386272	50237	600.00	Operation	IV (Metallic)	Au	18.07	21/03/2008	21/03/2018	Anagold	Elmadere-Demirmagara
Erzincan	İliç	Sabırlı	3095732	20067313	1,184.91	Operation	IV (Metallic)	Au+Ag+Cu	216.41	16/02/2012	16/02/2022	Anagold	Çakmaktepe Se-Ardich
Erzincan	İliç	Çöpler	3201587	76817	49.32	Operation	I-B (Brick Tile Clay)	Clay	6.68	15/07/2009	15/07/2019	Anagold	Clay Licence
Erzincan	İliç	Çöpler	3201588	76818	49.09	Operation	I-B (Brick Tile Clay)	Clay	49.09	15/07/2009	15/07/2019	Anagold	Clay Licence
				Total (ha)	16,572.75								
Erzincan	Kemaliye	Kabataş	2450158	57004	1,564.69	Operation	IV (Metallic)	Au+Cu	931.87	2/09/2018	2/09/2023	Kartaltepe	Mavidere
Erzincan	Kemaliye		3129489	200707602	1,572.23	Pending Operation	IV (Metallic)	–	–	2/08/2007	2/08/2012	Kartaltepe	Mavidere
Erzincan	Kemaliye		3129490	200707605	577.92	Pending Operation	IV (Metallic)	–	–	2/08/2007	2/08/2012	Kartaltepe	Mavidere
Erzincan	Kemaliye		3129496	200707606	1,818.11	Pending Operation	IV (Metallic)	–	–	2/08/2007	2/08/2012	Kartaltepe	Mavidere
Erzincan	İliç		1032544	58473	606.60	Operation	IV (Metallic)	Fe+Cu	7.54	16/11/2017	16/11/2027	Kartaltepe	Findiklidere
Erzincan	İliç	Yakuplu	1032719	7083	1,756.55	Operation	IV (Metallic)	Au+Ag+Cu+Fe Cr	Au+Ag+Cu+Fe: 175.00 Cr: 607.47	2/04/2011	2/04/2021	Kartaltepe	Bayramdere-Aslantepe-Saridere
Erzincan	İliç	Yakuplu	1027026	1054	660.87	Operation	IV (Metallic)	Au+Ag+Cu+Fe	359.33	30/07/2017	30/07/2027	Kartaltepe	Çakmaktepe
Erzincan	İliç	Ortatepe	2003094	7161	642.68	Operation	IV (Metallic)	Fe	214.65	7/05/2013	7/05/2023	Kartaltepe	Ortatepe
				Total (ha)	9,199.65								

Figure 4.4 Tenure Layout Plan



Anagold, 2020

4.4 Surface Rights

SSR Mining currently holds sufficient surface rights to support the Reserve Case oxide heap leach mining operations and sulfide processing and tailings disposal.

4.5 Taxation

The Turkish government implemented a temporary taxation rate increase from 20% to 22% for the periods of 2018–2020. From 2021 onwards, the effective tax rate is expected to return to 20%.

The CDMP20 economic analysis applies a corporate tax rate of 22% for Q4'20 and then the reduced 20% for 2021 onwards.

For tax purposes, a 20% accelerated depreciation rate is applicable for both the oxide and sulfide capital. The depreciation period is 10 years for general mining equipment, if not specifically defined by the tax office.

Investment incentive certificates are available for investments that promote economic development. Investment incentive certificates can be classified as strategic in specific circumstances and such certificates provide additional incentives. Anagold received a strategic incentive certificate for the sulfide process plant. An investment incentive certificate generates credits that offset corporate income taxes generated by the investment. The amount of investment credits generated from the investment incentive certificate is based on eligible capital expenditures. The investment credits generated by the strategic investment incentive certificate reduce the corporate tax rate to a minimum of 2% in a given tax period until the last quarter of 2023, thereafter it is assumed subsequent non-strategic investment incentive certificates will be available and the minimum rate will be 4%. Incentive tax credits can be carried forward to future tax periods indefinitely until exhaustion. Incentive tax credits and other tax pools are determined in the local currency, Turkish Lira, and subject to devaluation and revaluation as fluctuations against the US dollar occur. The cash flow model is prepared on a constant Turkish Lira basis.

Value-added tax (VAT) in Turkey is levied at 18% and the project is eligible for the Turkish exemptions for mining projects and mining equipment purchases. In the CDMP20 assumes the cash flows are not subject to VAT.

Import duties are not included in the capital cost estimate for mining related imported equipment because they are exempted in the incentive certificates.

4.6 Royalties

Under Turkish Mining Law, the royalty rate for precious metals is variable and tied to metal prices. The Çöpler project is subject to a mineral production royalty that is based on a sliding scale to gold price and is payable to the Turkish government. In September 2020 a presidential decree was issued, increasing the prescribed royalty rates by 25%.

Table 4.2 details the relevant prescribed royalty rates along with the revised rates following the September 2020 presidential decree. The royalties are calculated on total revenue with deductions allowed for processing and haulage costs of ore. Revenue from by-products (silver and copper) is included in the total revenue used for royalty calculations.

The royalty rates outlined in Table 4.2 apply to sellers of raw ore. Royalty rates are reduced by 40% for ore processed in country, as an incentive to process ore locally. As the Çöpler project produces its gold doré on site, the Çöpler project is eligible for a 40% reduction to the royalty rate.

Table 4.2 Gold Royalty Rates

Gold Price ($/oz)		Prescribed Royalty Rate (%)	Revised Royalty Rate (%)
From	To		
0	800	1.00	1.25
800	900	2.00	2.50
900	1,000	3.00	3.75
1,000	1,100	4.00	5.00
1,100	1,200	5.00	6.25
1,200	1,300	6.00	7.50
1,300	1,400	7.00	8.75
1,400	1,500	8.00	10.00
1,500	1,600	9.00	11.25
1,600	1,700	10.00	12.50
1,700	1,800	11.00	13.75
1,800	1,900	12.00	15.00
1,900	2,000	13.00	16.25
2,000	2,100	14.00	17.50
2,100	+	15.00	18.75

The Çöpler project effective life-of-mine (LOM) royalty rate based on the financial model metal price assumptions and applicable deductions is approximately 4.2%.

Other than the royalty payments, there are no other known back-in rights, payments, or other agreements and encumbrances to which the property is subject.

4.7 Environmental Liabilities

There are no known existing environmental liabilities for the Çöpler project, except for SSR Mining's obligation for ultimate reclamation and closure.

4.8 Permits

The EIA permitting for the Çöpler mine oxide ore was completed in April 2008 with the issuance of an EIA positive certificate. All of the necessary operation permits have already been obtained for the oxide inventory. These include:

- explosive storage permit,
- permit for water abstraction from groundwater sources,
- EIA positive certificate for power transmission line construction,
- land acquisition permits for forest areas and pasturelands,
- hazardous workplace permit, and
- operating permits.

The EIA permitting process for the Sulfide Expansion Project was commenced on 7 April 2014 and completed with the receipt of an 'EIA Positive Statement' on 24 December 2014. In addition to an EIA approval, other permits required for the Sulfide Expansion Project involved an expanded workplace opening permit, additional operating permits, and land acquisition permits for forest areas and pasture lands.

Additional EIA studies conducted, and environmental permits received for the Çöpler mine since the start of the gold mining operations are as follows:

- EIA permit, dated 10 April 2012, for the operation of mobile crushing plant.
- EIA permit, dated 17 May 2012, for the capacity expansion involving:
  - Increasing operation rate to 23,500 tpd.
  - Increasing Çöpler waste rock dump (WRD) footprint area.
  - Adding a sulfidisation, acidification, recovery, and thickening (SART) plant to the process in order to decrease the cyanide consumption due to the high copper content of the ore.
- EIA permit, dated 24 December 2014, for the capacity expansion involving:
  - Sulfide plant expansion
  - Heap leach area expansion
- EIA permit, dated 26 January 2017, for the Çakmaktepe satellite pits expansion.
- EIA permit, dated 9 August 2018, for the Çakmaktepe expansion for the new defined Central pit.

To the extent known, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the property that have not been discussed in this CDMP20.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Çöpler project is accessed from the main paved highway between Erzincan and Kemaliye, crossing the Karasu River and passing by the village of İliç. From İliç there is an additional 4.5 km of road to reach the Çöpler mine site.

The Ankara to Erzincan railway line, operated by the Turkish State Railway Company, (TCDD), runs parallel to the south bank of the Karasu River and passes within 2 km north of the site at a point between the train stations at İliç and Bağıştaş. The railway line connects the site with Ankara and the west as well as with seaports to the north on the Black Sea, and to the south on the Mediterranean Sea. Overnight passenger sleeper cars are available between Erzincan and Ankara.

The reservoirs of the Bağıştaş I & II hydro-electric power plants (HEPP) are 350 m and 1,800 m away from the Çöpler mine site, respectively. The embankment of Bağıştaş I Dam originally covered a portion of the existing highway, railroad, and railroad station until these were relocated before dam construction was completed. Construction routes for the railroad and highway were located between the new Çöpler village and the Çöpler mine site. The bridge on the north-east side of İliç was relocated to further east of the embankment.

There are regular commercial airline flights from Istanbul and Ankara to the regional cities of Erzincan, Erzurum, Malatya, Elazığ, and Sivas. Driving from the regional cities to the project site takes between two to four hours on paved highways. Driving from Ankara to the site takes approximately eight hours.

5.2 Local Resources and Infrastructure

The district of İliç has a population of approximately 6,990 inhabitants and is located approximately 6 km east of the current Çöpler pit. The district has a hospital, schools, municipal offices, a fire station, a police station, and a Gendarmerie post. The primary economic activity in the region is sheep herding for wool, meat, and dairy products. Other agricultural activities include bee keeping for honey production and, some wheat farming along the Karasu River. Additionally, there is some light manufacturing and grain milling performed in İliç.

The workforce for the SSR Mining exploration programmes has primarily included residents drawn from the local communities of Çöpler, İliç, and Sabırlı.

Turkish telecommunications are up to European standards. High-speed, fibre-optic internet access is in operation at the mine site.

Initially, electrical power at 380 V and 50 Hz, was available in İliç and at the mine site. This was upgraded to support the project by the construction of a 40 km long 154 kV power line from the substation at Divriği to the mine site. The power supply was further upgraded when the hydroelectric dam near the mine site was commissioned. Çöpler is now connected to the national grid by a 6 km 154 kV powerline from the Bağıştaş sub-station.

Sufficient local fresh water supply exists to support the mining and processing operations. Ground water resources include seven production wells with a 25,728 m³/day extraction permit. Further information on project infrastructure is included in Section 18. Section 20 contains additional data on the project social setting.

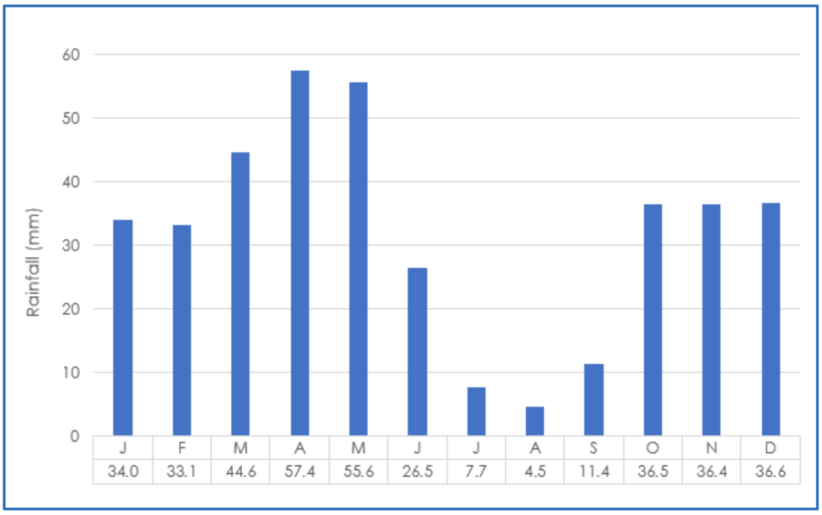
5.3 Climate

Site climate data were developed during previous studies. No additional climate data were generated for the CDMP20.

Mining operations are conducted year-round. The climate is typically continental with cold wet winters and hot dry summers. In winter, the night-time temperature can drop to –25°C although the average is usually a few degrees below freezing. The July temperature frequently exceeds +40°C but the climate is usually pleasantly warm outside of these extremes. The average monthly temperature ranges from +3.7°C for the coldest month of January to +23.9°C for August, the warmest month.

Most precipitation occurs in the winter and spring. Monthly average rainfall values are shown in Figure 5.1. The average annual rainfall for the site is 384.3 mm. Snowfall is common during the period mid-November through February, but with little, if any, accumulation. Snow depth assessments are based on the Divriği meteorological weather station, located 41 km west of the project area, which shows maximum snow-pack depths at approximately 200 mm for 1985.

Figure 5.1 Average Monthly Rainfall for Çöpler Project Area



Anagold, 2016

The frost depth is less than 0.3 m, based on local information, with 0.5 m selected as the design frost depth limit.

The maximum wind speed recorded at the Divriği weather station in 2004 ranges from 15–25 m/s, with variable directions mainly from the north, south, and east.

5.4 Hydrogeology

SRK compiled and updated the project conceptual hydrogeological model with new geological data, established a new numerical model and used it to evaluate the hydrogeology of the project area.

5.4.1 Existing Data Evaluation, Field Investigation, Hydrogeology Conceptual Model

Within the regional hydrology area, lithological units are defined in three main classes according to their underground water transport and transmission properties. These units are:

- Impervious units.
- Low permeate units: such units contain some thin layers that are more permeable than other layers with small extensions and provide water through sources with a flow rate of less than 1 L/s.
- Conductive units and very permeable units: Munzur limestone and Quaternary alluvium units.

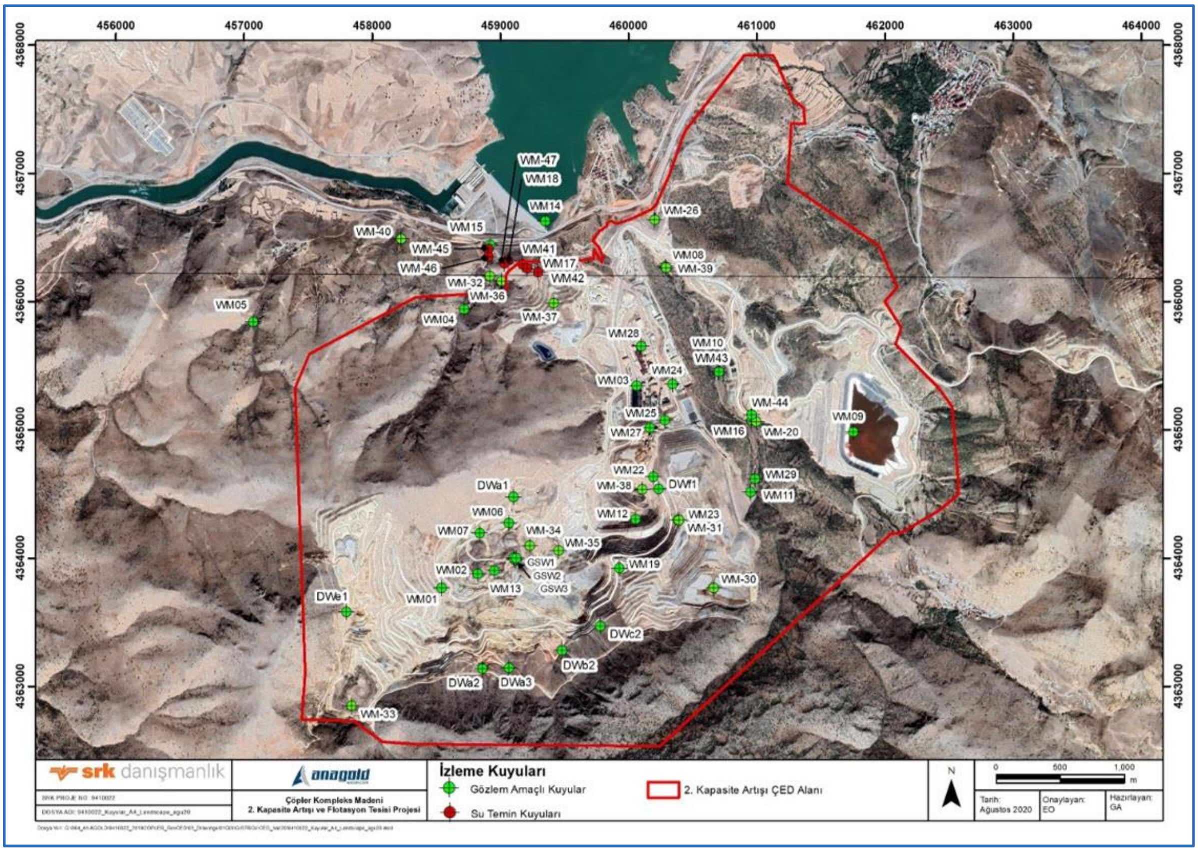
The regional geology is a complex structural assemblage of fault-bounded blocks including the following stratigraphy:

- Munzur limestone: grey to blue-grey, fine-grained to recrystallised marbles. Much of the unit displays various degrees of karst development. Bedding within the unit is indistinct to massive. This limestone group is also named the Çöpler limestone in the vicinity of the area where Mineral Resources have been estimated.
- Metasediment: fine-grained argillite sequences consisting of interbedded siltstones, shale units, marls, and sandy siltstones. The thermal and hydrothermal impact to this unit from the intrusions resulted in the creation of the skarns and hornfels.
- Ophiolitic mélange: ophiolitic mélange consists of diabase and serpentinite units. Serpentinisation is non-uniform and appears to be best developed near major fault zones.
- Diabase: the diabase is located within the upper zone of the ophiolitic mélange. The rock mass consists of green to greenish black. In general, joint surfaces are covered with calcite and iron oxide sealing. In places, the rock mass shows blocky textures embedded in a fine matrix.
- Diorite to granodiorite intrusions: beige and light brown, medium to coarse-grained plutons. This formation has intruded into the pre-existing argillite's and Munzur limestone. This includes fine to medium-grained quartz, feldspar, biotite, and amphibole minerals.
- Skarn: the skarn zone is developed along the granodiorite contact with the limestone and ophiolitic mélange. This zone was developed under elevated pressure and temperature conditions during intrusion of the granodiorite mass. The skarn units are black to dark brown, silicified, moderately weathered and includes frequent solution cavities.

5.4.2 Well Installation

A total of 56 wells for groundwater observation, testing, and water supply purposes have been drilled. Forty-one of the wells were drilled prior to 2018, 10 were for groundwater control and slope stability studies in 2018, two were for waste storage area observation purposes, and three were developed in 2018 as part of the sulfide expansion project for additional water supply. Hydrogeology wells drilled are shown in Figure 5.2.

Figure 5.2 Groundwater Wells



Anagold, 2020

Groundwater is expected to be recharged through the infiltration of precipitation through secondary porosity in the bedrock terrain. Groundwater elevation data indicates that the flow direction is generally northward to the Karasu River through the Munzur limestone. During the resource drilling and subsequent monitoring well installation programmes, perched groundwater conditions were reported above the clay-altered intrusions. It is anticipated that the perched groundwater is present in restricted areas. The volume of water held in storage as perched groundwater is unknown.

Groundwater elevations at the Çöpler project range from 1,328.5 m at Well GMW-10 (southern end of the site) to 864.7 m at Well GMW-09 (northern end of the site). Observations of cavernous features (karst) during drilling and high values of hydraulic conductivity from aquifer tests suggest an area of karst development in the limestone near the Karasu River, at boreholes GMW-09 and GMW-24. This was incorporated into the groundwater flow model as an area of high hydraulic conductivity near these wells and along the Sabırlı Fault.

5.5 Physiography

The Çöpler project is located in a roughly east–west oriented valley at altitudes of 1,100–1,300 m. The valley is surrounded by limestone mountains that rise to more than 2,500 m on the north and south sides of the project area. These mountains are at the western end of the Munzur range, which rises to more than 3,300 m between Ovacık and Kemah.

The region is sparsely vegetated, predominantly with semi-arid brush and scrub trees including dwarf oaks and junipers.

The following are the site data developed during previous studies for the design of the project:

- Latitude: 39° 25' North
- Longitude: 38° 32' East
- Elevation: 1,150 mamsl
- Frost depth: 500 mm
- Snow load: 145 kg/m^2
- Wind load: 40 m/sec, Exposure 'C'
- Earthquake zone: second order, Ao = 0.20
- Atmospheric pressure (average): 880.5 millibars
- Maximum design temperature: +40°C
- Minimum design temperature: –25°C
- Annual rainfall: 384 mm
- Maximum snowfall depth: 200 mm (estimated)
- Design maximum rainfall: 24 hours, 76 mm

6 HISTORY

The region around the Çöpler project has been subject to gold and silver mining dating back at least to Roman times, with historical bullion production estimated at approximately 50 koz of gold. A copper-rich slag pile of approximately 2.5 kt is located at the western edge of the district and is believed to be waste from ancient production. Although the district contains copper mineralisation, there appears to have been little production targeting copper. There are several additional minor slag piles scattered around the property thought to be from ancient, small-scale gold and by-product copper production.

The Turkish Geological Survey (MTA) carried out regional exploration work in the early 1960s that was predominately confined to geological mapping. In 1964, a local Turkish company started mining for manganese, continuing through until closing in 1973 and producing approximately 7.3 kt of manganese ore during its active life. Unimangan Manganez San A.Ş. (Unimangan) acquired the property in January 1979 and re-started manganese production, producing 1–5 ktpa of ore until ceasing operations in 1992.

In 1998, Anatolia Minerals Development Ltd (Anatolia) identified several porphyry-style gold–copper prospects in east-central Turkey and applied for exploration licences for these prospects. This work was based upon the earlier work by MTA in the 1960s. During this effort, Anatolia delineated a prospect in the Çöpler basin formed by an altered and mineralised granodiorite, intruded metasediment, and limestone. This prospect and the supporting work was the basis for a joint venture agreement for exploration with Rio Tinto.

During the period of the joint venture, exploration drilling of the Çöpler deposit was completed and a Mineral Resource estimate was developed with three mineralised zones: the Main, Manganese, and Marble zones. In January 2004, Anatolia acquired sole control over the project and maintained exclusivity until 2009, at which time a joint venture with Lidya was executed.

In February 2011, Anatolia merged with Avoca Resources Limited to form Alacer Gold Corp. (Alacer). In September 2020 Alacer merged with SSR Mining.

Today the Çöpler project is owned and operated by Anagold Madencilik Sanayi ve Ticaret Anonim Şirketi (Anagold). SSR Mining controls 80% of the shares of Anagold, Lidya Madencilik Sanayi ve Ticaret A.Ş. (Lidya), controls 18.5%, and a bank wholly-owned by Çalık Holdings A.Ş., holds the remaining 1.5%.

In most cases the company will be referred to as SSR Mining throughout this Technical Report even though it may have been Alacer or Anatolia at the time referenced in the report.

The previous Technical Report was the 2016 Technical Report on the Çöpler project and Çöpler Sulfide Expansion Project, which updated the Mineral Resource and Mineral Reserve estimates and the Sulfide Expansion Project status from the 2015 Technical Report and provided updated information on the current detailed engineering phase.

The previous reporting of Mineral Resources and Mineral Reserves was in the Annual Information Form for the year ended 31 December 2019 and filed 4 February 2020 (Alacer). Those statements on Mineral Resources and Mineral Reserves have been used for comparison.

6.1 Previous Technical Reports

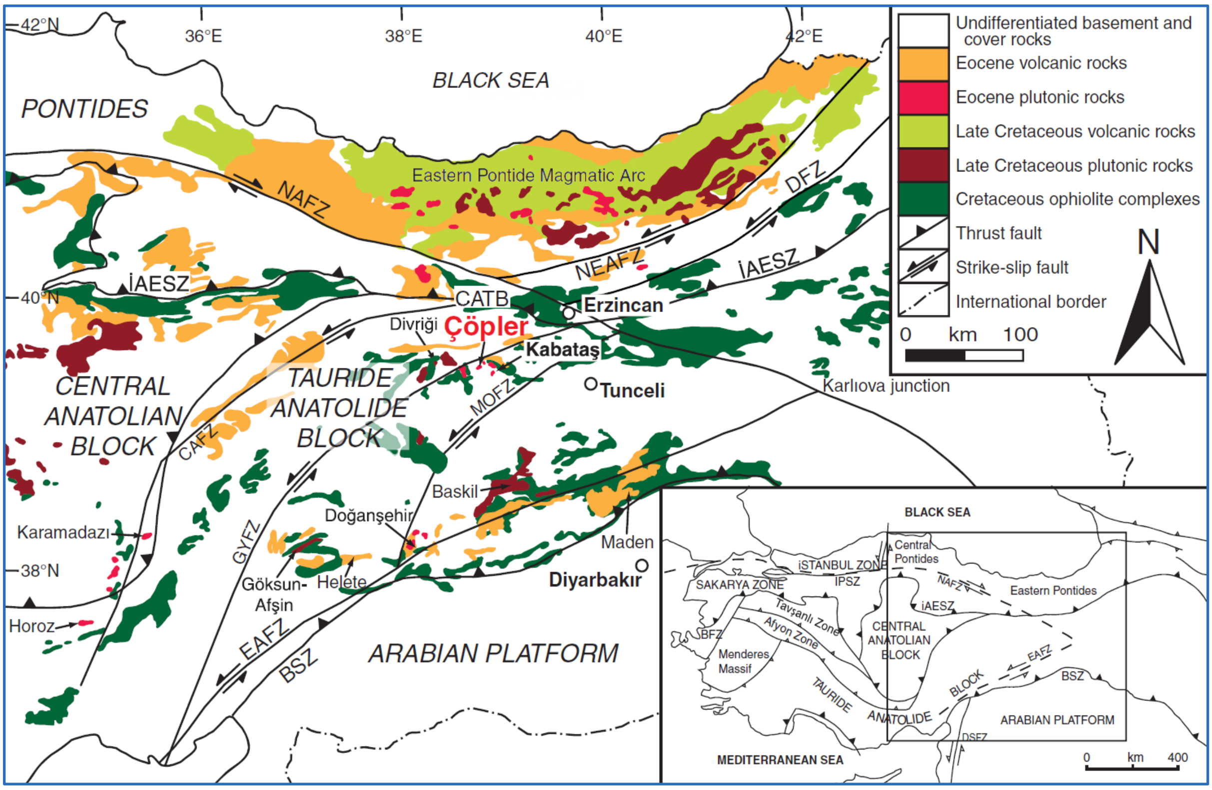
The following Technical Reports have been filed on the Çöpler project (in chronological order):

- Watts, Griffis and McQuat Limited, 2003. Update of the Geology and Mineral Resources of the Çöpler Prospect, 1 May 2003.
- Independent Mining Consultants, Inc., 2005. Çöpler Project Resource Estimate, 19 October 2005.
- Marek, J.M., Pennstrom, W.J., Reynolds, T., 2006. Çöpler Gold Project Feasibility Study, 30 May 2006.
- Marek, J.M., Moores, R.C., Pennstrom, W.J., Reynolds, T., 2007. Çöpler Gold Project, 2 March 2007, as amended 30 April 2007.
- Easton, C.L., Malhotra, D., Marek, J.M., Moores, R.C., and Pennstrom, W.J., 2008. Çöpler Gold Project East Central Turkey Preliminary Assessment Sulfide Ore Processing, 4 February 2008.
- Marek, J.M., Benbow, R.D., and Pennstrom, W.J., 2008. Çöpler Gold Project East Central Turkey, 5 December 2008 (amended and restated; supersedes 11 July 2008 version).
- Altman, K., Liskowich, M., Mukhopadhyay, D.K., and Shoemaker, S.J., 2011. Çöpler Sulfide Expansion Project Prefeasibility Study, 27 March 2011.
- Altman, K., Bascombe, L., Benbow, R.D., Mach, L., and Shoemaker, S.J., 2012. Çöpler Resource Update, Erzincan Province, Turkey, 30 March 2012.
- Altman, K., Bair, D., Bascombe, L., Benbow, R., Mach, L., and Swanson, B., 2013. Çöpler Mineral Resource Update, Erzincan Province, Turkey, 28 March 2013.
- Armstrong, D., Bascombe, L., Bohling, R., Kiel, R., Liskowich, M., Parker, H.M., Parshley, J., Seibel, G., and Swanson, B., 2014. Çöpler Sulfide Expansion Project Feasibility Study, Erzincan Province, Turkey, 29 July 2014.
- Bascombe, L., Benbow, R.D., Birch, R.G., Bohling, R., Francis, J., Khoury, C., Kiel, R., Liskowich, M., Marsden, J., Parker, H.M., Parshley, J., Seibel, G., and Statham, S., 2015. Çöpler Sulfide Expansion Project Feasibility Update, Erzincan Province Turkey, 27 March 2015.
- David, D., Kiel, R., Liskowich, M., Parshley, J., Marsden, J., Seibel, G., Parker, H., Bascombe, L., Benbow, R., Statham, S., Francis, J., and Smolonogov, S., 2016. Çöpler Mine, Erzincan Province, Turkey, 9 June 2016

7 GEOLOGICAL SETTING AND MINERALISATION

The Çöpler district is located near the north margin of a complex collision zone and to the south of the prominent North Anatolian Fault Zone (Figure 7.1). The collision zone, and subsequent crustal thickening, is related to the closure of the northern branch of the Neotethys ocean, resulting from the northward subduction and coming together of the Pontides and Tauride Anatolide Block in the Late Cretaceous to Early Tertiary. In this intensely-deformed tectonic region, east–west trending imbricated structures were cut by north–north-east trending strike-slip faults during the Late Cretaceous to Paleogene period, further complicating the geology.

Figure 7.1 Geological Setting of the Çöpler District



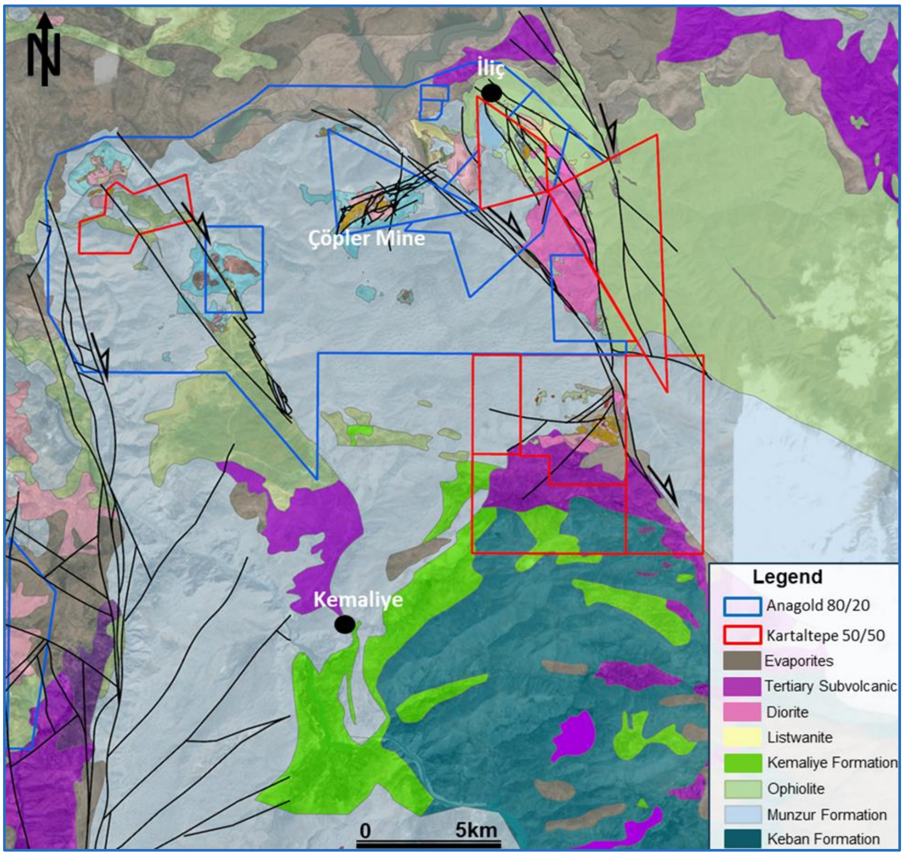
İmer, 2012

The Çöpler district deposits (Çöpler, Çakmaktepe, Ardich, and Bayramdere) are hosted within the Tethyan mineralised belt, a major global mineralised terrain for gold, copper, and base metals, stretching from Indo-China into Europe through Eurasia.

Three main rock assemblages are exposed in the Çöpler district (Figure 7.2).

- The first assemblage includes the Keban, Munzur, and Kemaliye formations. These units are tectonically overlain by ophiolitic nappes (Ovacık Formation of Özgül and Turşucu 1984).
- The second assemblage includes Middle Eocene magmatic and sedimentary rocks.
- The third assemblage includes the Oligocene to Recent sedimentary Sivas basin.

Figure 7.2 Geological and Structural Map of the Çöpler District



SSR Mining, 2020

7.1 Geological Setting – Çöpler Deposit

7.1.1 Geology – Çöpler Deposit

The Çöpler deposit is centred on composite diorite to monzonite porphyry stocks that are part of the Eocene Çöpler Kabataş magmatic complex dated (by İmer et al., 2013) at:

- 43.8 ± 0.3 Ma and 44.2 ± 0.2 Ma (from 40 Ar/39 Ar analysis of igneous biotite), and
- 44.1 ± 0.4 Ma (from igneous hornblende).

The magmatic rocks have intruded into both the Keban and Munzur formations.

Metamorphic rocks of the Permian to Upper Cretaceous Keban formation shelf sequences vary in composition between siliciclastic and calcareous, with fine to medium-grained sandstone interbedded with mudstone, and locally thick sections of fine laminated mudstone. The sedimentary units are folded with a resolved fold axis plunging at approximately 25→200 (plunge→plunge direction) from bedding measurements in the Çöpler pits. The limestone of the Upper Triassic to Late Cretaceous (Upper Campanian) Munzur formation overlies the folded Keban formation with a structural contact represented by cataclasite at the base of the Munzur formation. Intense shearing in the underlying sedimentary rocks is also observed, with top-to-south kinematics.

Stratigraphically, the Munzur formation overlies the Keban. However, stratigraphic mapping of the Munzur formation to the north of Çöpler shows homoclinal structure with consistent bedding in the limestones at approximately 40 / 060 (dip / dip-direction) indicating juxtaposition of structural blocks. The Munzur allochthon was thrusted onto Permo-Triassic metamorphic basement in the Late Cretaceous (Özgül and Turşucu 1984). This structural contact pre-dates Eocene Çöpler Kabataş intrusions, which appear to have intruded across the sheared contact between Keban metamorphic rocks (Main Zone) and Munzur limestone (Manganese Zone).

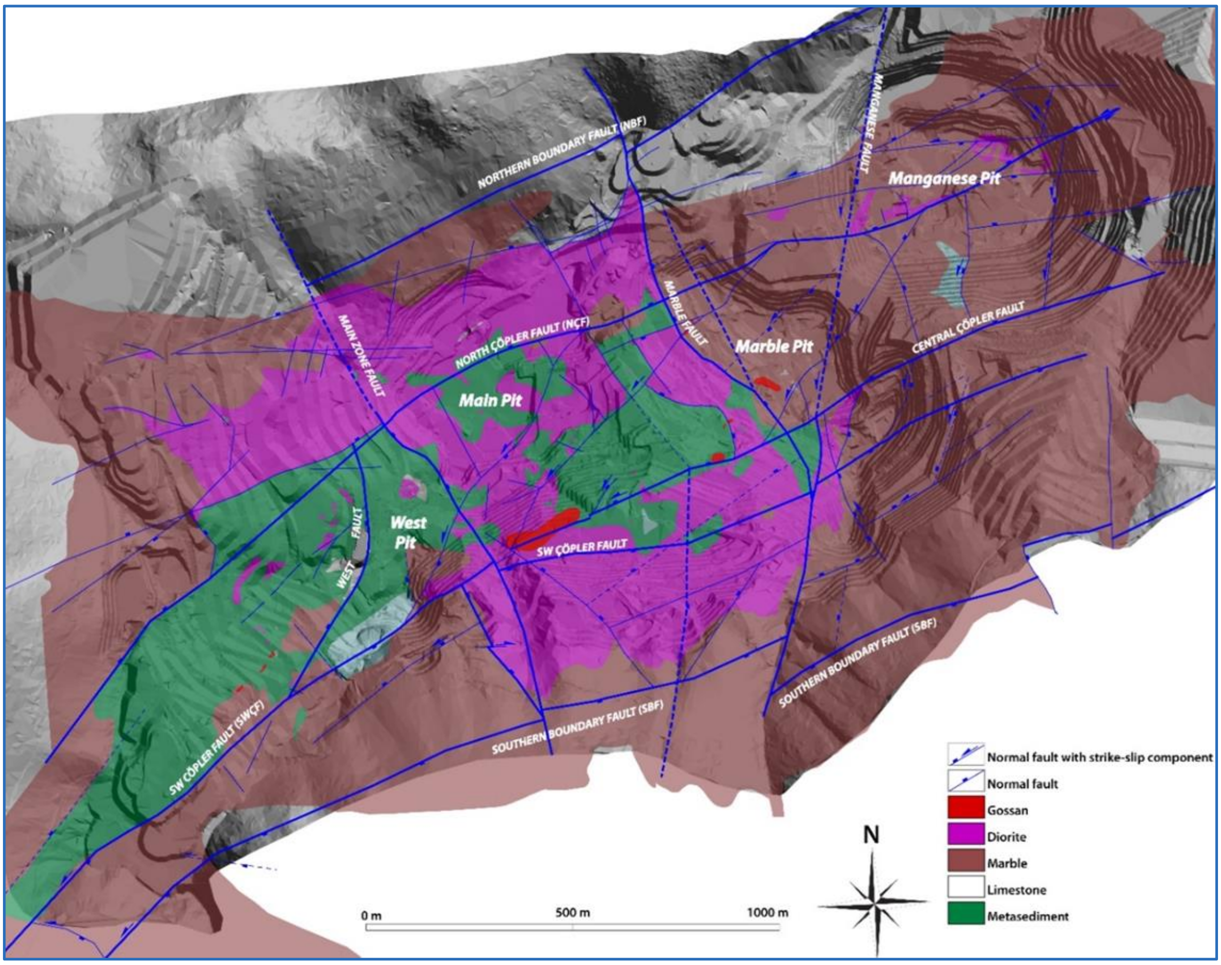
The Çöpler intrusion is a hornblende–quartz diorite-porphyry that shows strong argillic alteration. Some fresh outcrop occurs in the central part of the Main Zone and also as remnants within the Manganese Zone. In its least-altered state, the diorite-porphyry is relatively pristine with well-preserved hornblende, biotite, and K-feldspar phenocrysts in a granular matrix of plagioclase and quartz with prominent magnetite. Flow alignment of the hornblende phenocrysts can be seen in places. Gradational transitions to argillic-altered rocks are evident in outcrop and drill core on a centimetre scale.

The primary control on the location of the Çöpler intrusion appears to have been the hornfels-carbonate contact. The contact of the Çöpler intrusion has a roughly rectilinear shape, suggesting control by pre-existing east–north-east trending faults, and by a set of north–north-west trending fractures. The north–north-west striking bedding may also have exerted a local control in the central part of the intrusion where many intrusive contacts are parallel to bedding and have a sill-like morphology. However, it is considered more likely that this reflects the north–north-west trending fracture control referred to above.

A pronounced ground magnetic anomaly is centred on the core of the porphyry, which has been modelled to reflect the potassically altered core of the stock-like barren porphyry system dipping steeply towards the south. In addition, there are several dykes and intrusive apophyses; most notably, a brecciated and strongly clay-altered intrusion centred on the Manganese Zone.

In the area of the Çöpler deposit, two dominant sets of faults are present. These faults are approximately parallel to the long axis of the deposit and are oriented east–north-east. These are referred to as longitudinal faults. The other set of faults are transverse to the longitudinal faults and referred to as cross-faults (Figure 7.3). The major cross-faults include from east–west; Manganese fault, Marble fault, Main Zone fault, and West fault.

Figure 7.3 Çöpler Deposit Geological Map



SSR Mining, 2020

The longitudinal faults include the Northern Boundary fault (NBF), North Çöpler fault (NÇF), Central Çöpler fault, South-West Çöpler fault, and Southern Boundary fault (SBF). Among these, previously the Central and South-West Çöpler faults were thought to be the same fault and dipping towards the south.

Weathering has resulted in oxidation of the mineralisation close to surface. The oxidised cap is underlain by primary and secondary sulfide mineralisation. In addition to the gold–silver–copper mineralisation of economic interest, arsenic, lead, magnesium, manganese, mercury, and zinc are also present.

7.1.2 Mineralisation – Çöpler Deposit

The mineralisation at the Çöpler deposit area is exposed in four adjacent open pits from east to west: Manganese pit, Marble pit, Main pit and West pit. The pits expose economic parts of the same orebody and the three eastern pits will likely join up as the mining progresses. The predominant rock types in the mine include limestone/marble, metamorphic rocks (mainly hornfels) and diorite-tonalite porphyry, locally with equigranular biotite-granodiorite intrusions. Supergene enrichment enhanced along syn-mineralisation and post-mineralisation structures plays an important role in localising high-grade gold mineralisation at lithological contacts, late-stage faults and shear zones, and fault / contact intersections.

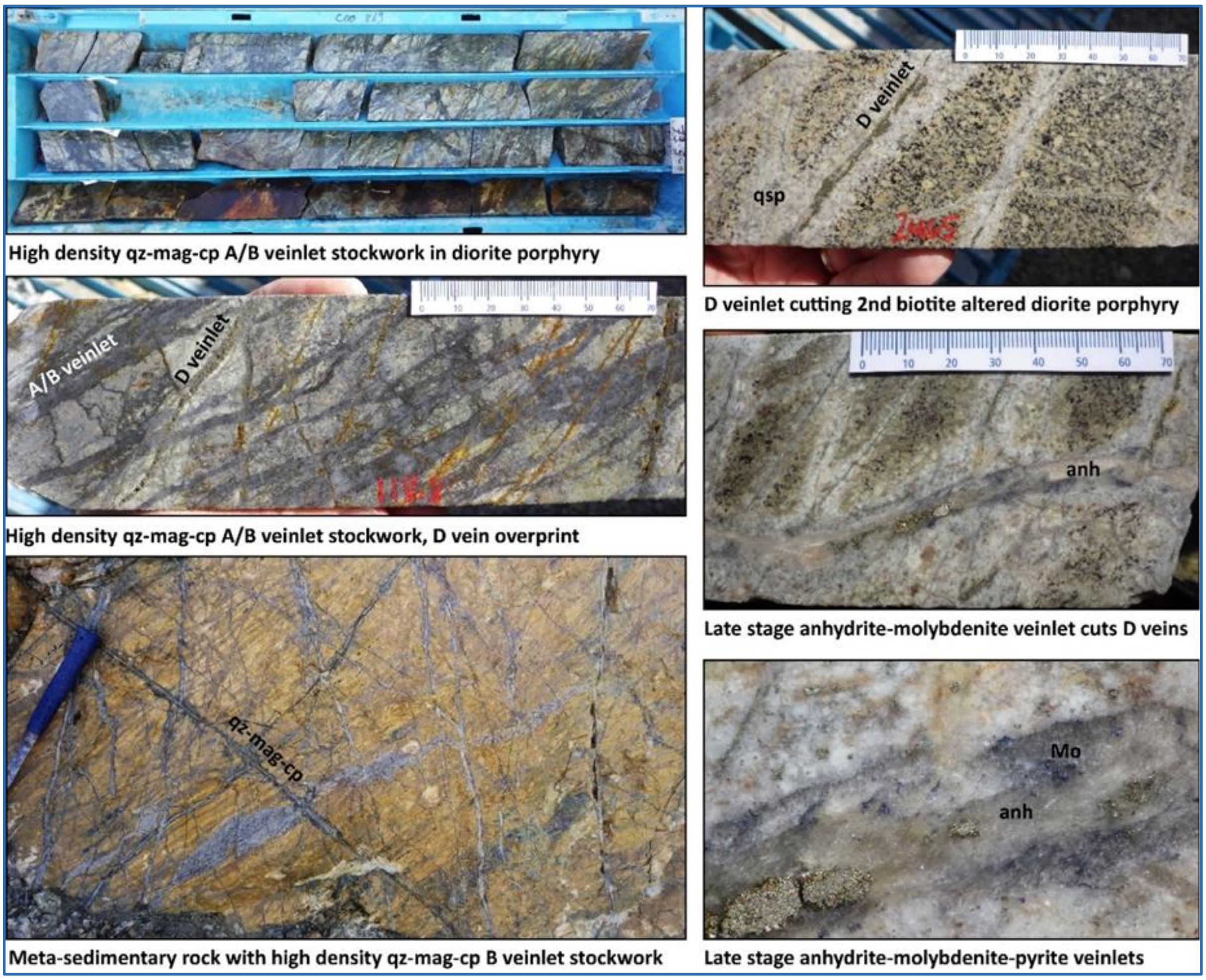
In general, three closely related mineralisation styles can be identified across the six primary areas at the Çöpler deposit.

7.1.2.1 Three Mineralisation Styles at the Çöpler Deposit

Low-Grade Porphyry Vein Mineralisation

Low-grade sub-economic porphyry copper–gold–molybdenum mineralisation is characterised by well-developed alteration zones that are complex and superimposed on each other. Late-stage porphyry mineralisation is hosted in diorite-tonalite porphyry as dominant sheeted veinlet arrays and as stockworks in metamorphic wall rocks and intruded into a low-grade to barren diorite porphyry system (Figure 7.4). Porphyry veinlets are best exposed in the Main pit since the volume of outcropping intrusions is much greater than in other areas of the mine. Early, irregular high-temperature quartz–chalcopyrite–magnetite veinlets are overprinted by 'D' veinlets with pyrite±quartz and symmetric feldspar-destructive phyllic halos (Figure 7.4). Dense 'A'/'B' veinlets occur as sheeted arrays and lesser stockworks in the intrusions but form well developed dense stockworks in the surrounding metamorphic wall rocks (Figure 7.4). Late-stage anhydrite veinlets with pyrite and molybdenite appear to overprint the 'D' veins, (Tripp, 2017; internal company report).

Figure 7.4 Çöpler Deposit Porphyry Vein Mineralisation



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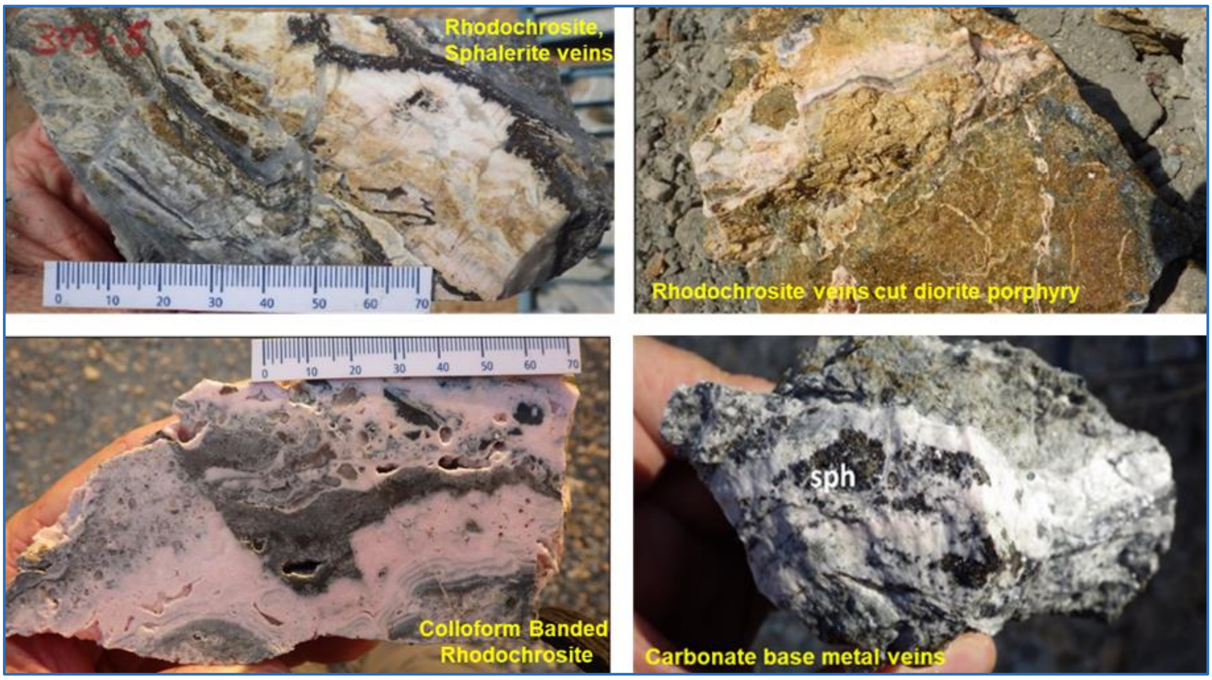
Intermediate Sulfidation Epithermal Mineralisation

Intermediate sulfidation epithermal mineralisation is primarily observed in the Manganese pit as clusters of bright pink, banded, colloform, rhodochrosite base metal sulfide veins and breccia lodes, with a spatial association with elevated gold grades, (Figure 7.5). Carbonate base metal veins contain base metal sulfides sphalerite±galena±chalcopyrite in a gangue of calcite, ferroan dolomite, and/or manganese carbonates (rhodochrosite) or realgar. In the Main pit, the base metal carbonate veins are coarsely crystalline, compared with the Manganese pit where the veins display brecciation, colloform banding, and locally bladed calcite replaced by silica in the interpreted higher level position.

Figure 7.5 Çöpler Deposit Intermediate Sulfidation Epithermal Mineralisation

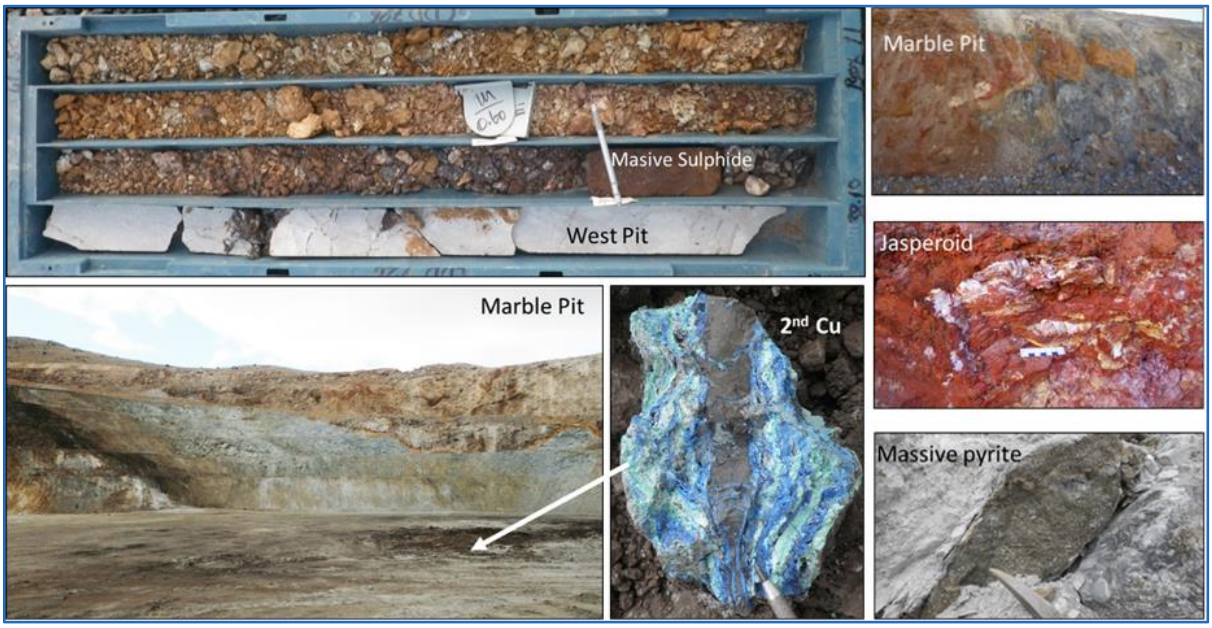


SSR Mining, 2020

Iron Skarn and Carbonate Replacement Mineralisation

Iron skarn and related carbonate replacement oxide gold mineralisation developed along faults, shear zones, and within karstic spaces. It is observed as iron oxide-rich zones as well as gossan-like and jarosite formations developed by oxidation of previous pyrite-rich mineralisation, (Figure 7.6). This replacement type mineralisation appears to be derived from previously formed distal skarn mineralisation. The age of gossan / jasperoid development possibly relates to post-mineralisation weathering of primary Eocene sulfide mineralisation in semi-arid conditions, either at the contact where gossan is found, or remobilised from a nearby source, likely continuing up to the present day.

Figure 7.6 Çöpler Deposit Porphyry Vein Mineralisation



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7.1.2.2 Six Mineralisation Areas at the Çöpler Deposit

Main Zone Mineralisation

The Main Zone lies in the west portion of the Çöpler deposit and occupies a footprint of approximately 750 m north–south by 1 km east–west. Typical depths of mineralisation range to 200 m below surface. Disseminated quartz–pyrite–arsenopyrite epithermal veinlets are primarily hosted in diorite and metasediment with some marble-hosted mineralisation on the eastern margin of the zone. Oxidation has occurred, and oxide mineralisation occurs from near-surface to depths of approximately 40 m below surface, with the thickest development over ridges and thinning in the intervening valleys.

Minor volumes of massive sulfide pyrite mineralisation occur within the Main Zone.

Main Zone West Mineralisation

Main Zone West is located in the north-west corner of the Çöpler deposit at the contact between diorite, marble, and the basement metasediment. This mineralisation is hosted within narrow gossans located at the contact, and in sub-parallel veinlets containing disseminated sulfides within the marble and metasediment. Main Zone West has a strike length of approximately 750 m and is approximately 75 m wide.

Main Zone East Mineralisation

The Main Zone East represents a portion of the mineralisation lying between the Manganese Zone and Main Zone. The geology in this area is typified by narrow, weakly to moderately-mineralised gossans located at the contact between the basement metasediment and the overlying marble. It is postulated that the gossan is sourced from the diorite located in the Manganese Zone and has been emplaced along the metasediment / marble contact as the diorite has crystallised.

Manganese Zone Mineralisation

The Manganese Zone occupies the eastern end of the Çöpler deposit. This zone is approximately 650 m wide north–south by 650 m east–west. The pre-mining surface expression of this area consists predominately of marble. A moderately-sized intrusion of diorite occurs sub-surface. A large proportion of the Manganese Zone mineralisation is associated with the contact between this diorite and the surrounding marble. Mineralisation ranges from surface to approximately 400 m depth.

Free gold mineralisation occurs in the marble with minimal associated sulfides. Disseminated quartz–sulfide mineralisation occurs in clay-altered and brecciated diorites as well as locally carbonate-altered diorite. Moderate volumes of massive sulfide pyrite mineralisation occur within the Manganese Zone. It appears that 'leachable' mineralisation is a combination of free gold in marble and supergene oxidised mineralisation in both marble and diorite. Leachable oxide mineralisation occurs to +200 m below surface.

Marble Zone Mineralisation

The Marble Zone occurs in the south-eastern portion of the Çöpler deposit and is associated with a north-east striking fault contact between marble to the east and metasediment and intrusions to the west. The geology in this area is typified by large 'plugs' of gossan and diorite that have formed at the junctions between large-scale faults, where mineralising fluid flow has been considerable. The width of the Marble Zone is approximately 350 m, and the strike length is 300 m east–north-east. The depth of mineralisation ranges from surface to approximately 160 m below surface.

Mineralisation occurs as both disseminated sulfides in veinlets and massive sulfide along the marble contact. Oxidation has occurred along the north-east structure resulting in greater depths of oxidised mineralisation than that seen in the Main Zone.

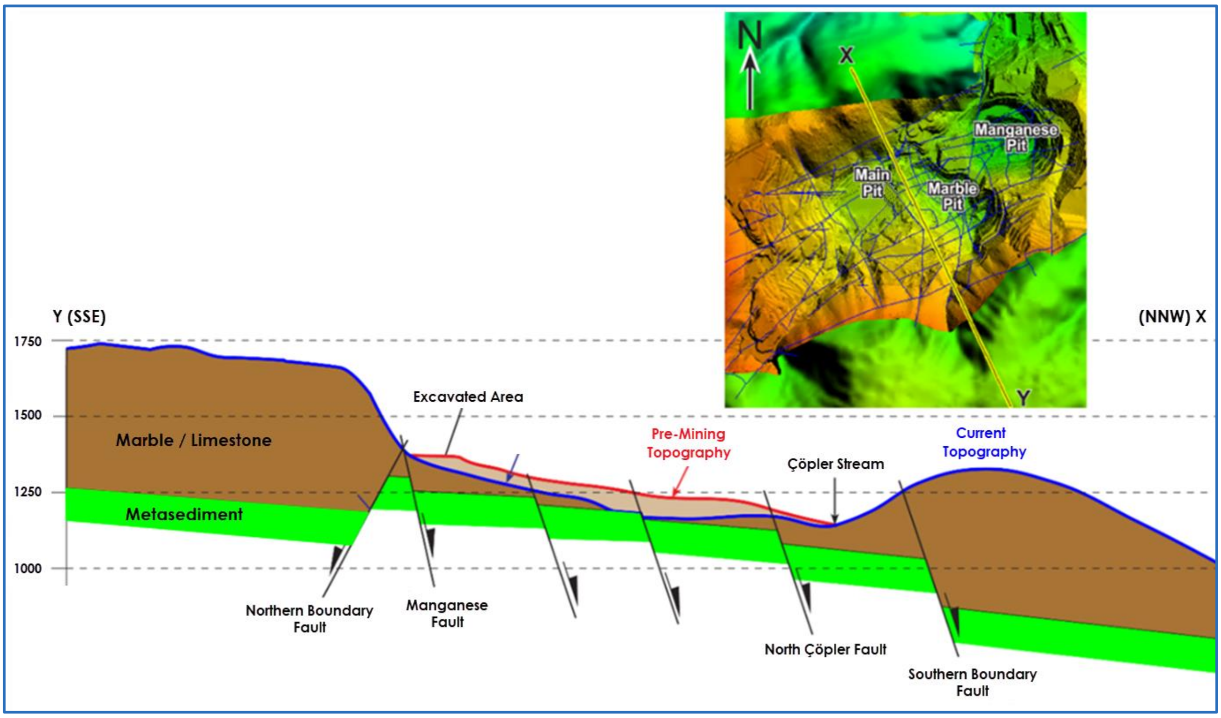
West Zone Mineralisation

The West Zone occupies the westernmost portion of the Çöpler deposit and is located at the contact between the basement metasediment and the overlying limestone/marble, where a large-scale north-east trending fault is located. Mineralisation is present within veinlets containing disseminated sulfides, massive sulfides, and oxidised gossan. The West Zone has a strike length of approximately 700 m north-east and is approximately 150 m wide. Multiple narrow mineralised zones are present sub-parallel to the faulted contact and occur to a depth of approximately 150 m below surface.

7.1.3 Structure – Çöpler Deposit

The Çöpler deposit area demonstrates trans-tensional deformation. The extensional deformation in the area dominates over strike-slip motion as indicated by the lack of compressional structures and the presence of normal movement on all faults. Structurally, the Çöpler deposit occurs in a horst-like feature developed within a sinistral trans-tensional strike-slip setting (Figure 7.7). The two boundary faults delimit the northern and southern extent of the gossan-like, oxidised, supergene, gold-bearing deposits. The northern and southern boundary faults are located almost at the present boundaries of the mine and they dip away from the mine, thereby defining the horst geometry. In addition, the deposit is traversed by a number of cross-cutting normal faults (with or without strike-slip components) in various orientations that complicate but localise the geometry and position of oxidised ore (Kaymakçı, 2017, internal company reporting).

Figure 7.7 Simplified Schematic of the Çöpler Deposit Structures (cross-section)



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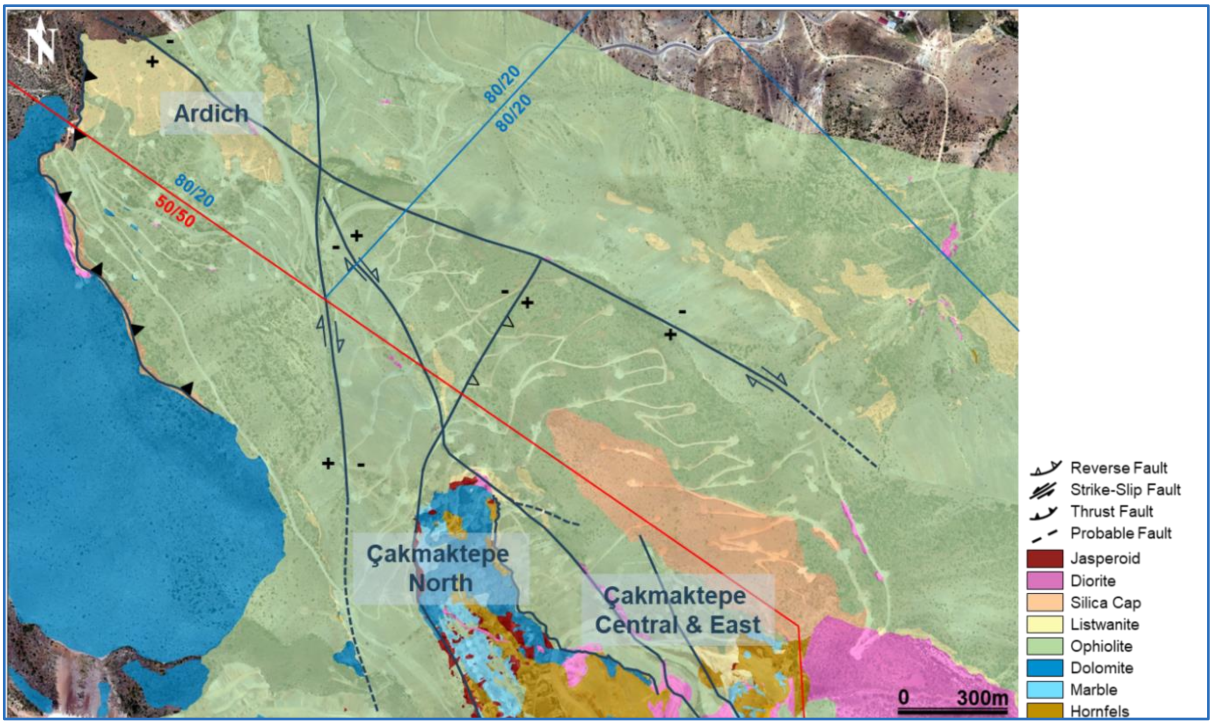
7.2 Geological Setting – Çakmaktepe Deposit

7.2.1 Geology – Çakmaktepe Deposit

The Çakmaktepe deposit is made up of a number of mineralised zones (Figure 7.8). The deposit area mainly comprises various Palaeozoic metamorphic rocks and marbles belonging to the Keban formation that constitute the basement and Mesozoic platform carbonates (e.g. Munzur limestone). All these units are tectonically overlain by ophiolitic mélange rocks. These ophiolitic rocks originated from the northern branch of the Neotethys ocean, the former position of which is delineated by the Ankara–Erzincan suture zone. The emplacement of the ophiolitic units took place at the end of Upper Cretaceous with north to south motion.

Figure 7.8 Geological Map of the Çakmaktepe and Ardich Deposits



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The youngest units include Eocene and younger magmatic rocks, volcaniclastics, and various sedimentary units that unconformably overlie and seal the Munzur limestone, its basement and the ophiolitic units and their tectonics contacts. All of these units are intruded by widespread intermediate igneous rocks that are exposed mainly at the northern and western parts of the Munzur mountains and southern margin of Sivas Basin.

Listwanite has formed in structurally deformed areas by the percolation of CO_2-rich fluids along the contacts of ultramafic rocks that are part of the ophiolite complex. Sulfidic jasperoid is present, caused by silica-sulfide metasomatism of the Munzur dolomites. Both listwanite and jasperoid are important host rocks for gold and silver mineralisation.

7.2.2 Mineralisation – Çakmaktepe Deposit

The Çakmaktepe deposit is a structurally controlled gold–silver–copper deposit, displaying both epithermal and replacement mineralisation styles. Mineralisation is primarily associated with jasperoid and listwanite. At depth, mineralisation transitions below the base of oxidation to disseminated pyrite, vein sulfides, and massive sulfide horizons, generally occurring within shear zones, along shallow thrusts, in diorite sills, and on intrusion margins.

As with the Çöpler deposit, Çakmaktepe is considered to be the result of a mineralised intrusion that generated suitable conditions for mineralisation to be localised into a favourable geological setting of ophiolite, limestone, and hornfels lithologies (Figure 7.9). A complex system of faults and thrusts have allowed mineralised fluids and diorite dykes and sills associated with the epithermal system to permeate into the stratigraphy. Steep-dipping, shear-hosted mineralisation is characterised at Çakmaktepe North, whereas flatter, early-stage, thrust-related mineralisation is characterised at Çakmaktepe East, Çakmaktepe South-East and Çakmaktepe Central. Key to each structurally associated style of mineralisation is the juxtaposition of ophiolites against limestone and hornfels to create the right geochemical conditions for the deposition of gold and other metals.

Figure 7.9 Geological Mapping within Çakmaktepe Central Pit

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The Çakmaktepe North area is strongly sheared with epithermal characteristics and grade associations with intrusive diorite dykes. The bulk of the mineralisation is structurally confined to a major sub-vertical shear zone (Main Shear). The Main Shear varies in width from 5–40 m, has been defined to a depth of 200–250 m below surface, and dips at approximately 70° to the east. Surface mapping and sampling have defined the mineralised extent of the shear as being over 1 km in length.

Mineralisation at Çakmaktepe North is not solely contained within the shear zone, but also occurs along flat thrust structures and lithological contacts cut by the shear zone. Contacts between ophiolite and limestone, limestone and hornfels, and all lithologies in contact with intrusive diorite sills and dykes are generally mineralised. The listwanite horizon is the most favourable host rock for gold mineralisation. Diorite intrusions show evidence of hydrothermal activity that either takes the form of massive iron-dominated replacement (magnetite, specular hematite, or pyrite) or sheeted crystalline quartz veins bearing jasperoids closer to diorite contacts.

Other mineralised zones within the Çakmaktepe deposit are referred to as 'contact' styles of mineralisation where iron, sulfur, gold, copper, and silver have been emplaced along thrust surfaces where ophiolite is next to limestone and metasediment. Epithermal veining and replacement alteration and textures are prevalent. Skarn and metasomatic mineralisation occurs in contact with intrusive diorite dykes, sills, and stocks.

Oxide mineralisation at Çakmaktepe is predominantly characterised by silica–iron–carbonate rich jasperoid, less-siliceous iron-rich gossan, and epithermal veined and brecciated limestone.

7.3 Geological Setting – Ardich Deposit

7.3.1 Geology – Ardich Deposit

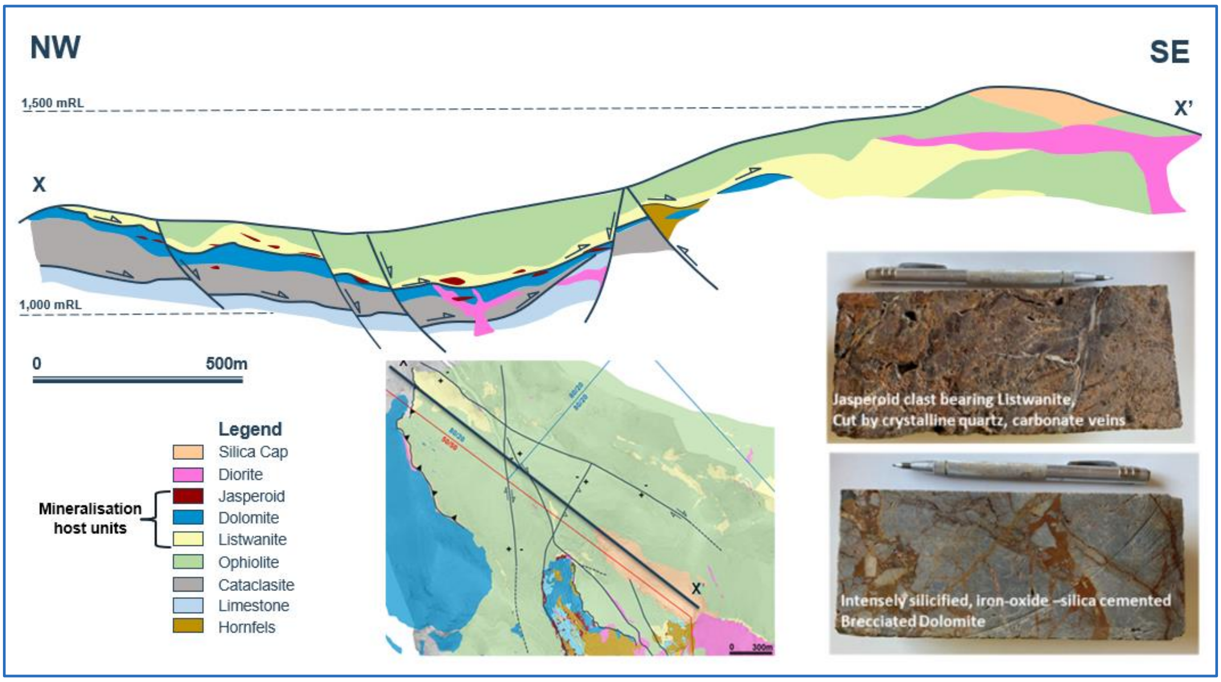
The Ardich deposit is located immediately to the north-west of the Çakmaktepe deposit (Figure 7.8). The north-western portion of Ardich and the Çakmaktepe North mineralised zone are in close proximity to each other, as are the Ardich Southeast and Çakmaktepe East mineralised zones. While there are some characteristic differences between Ardich and Çakmaktepe, the local geology is generally very similar.

The mineralisation at Ardich occurs at a higher stratigraphic level that that seen at Çakmaktepe. The emphasis at Ardich is on the ophiolitic mélange rocks that have been thrust into place on top of the basement metasediment and carbonate lithologies.

The local geology at Ardich is dominated by ophiolites, listwanite, and dolomites and limestones, with mineralisation occurring along low-angle thrust zones between ophiolites, listwanite, and dolomites and limestones (Figure 7.10). This occurs within a complex north-west trending structural zone that is cut by multiple high-angle faults that together result in creating multiple rotated fault blocks and mineralised zones.

The mineralisation at Ardich is considered to be related to fluids associated with diorite intrusions at depth, much like those observed at the Çöpler and Çakmaktepe deposits. Diorite dykes are present but not common at Ardich, unlike the adjacent Çakmaktepe deposit and nearby Çöpler deposit where diorite is a dominant lithology.

Figure 7.10 Schematic of Ardich Geological Setting with Mineralisation Examples



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7.3.2 Mineralisation – Ardich Deposit

The mineralisation at Ardich is related to crystalline and chalcedonic quartz veins within the brecciated and silicified listwanite and dolomite zones. The mineralisation is predominantly in the form of oxide, with sulfide mineralisation confined to limited pyrite-rich jasperoid zones. Clay / gossan in jasperoids or limestone karstic boundaries also contain high-grade gold across Ardich.

Gold grades increase at dolomite / listwanite contacts and within silica-rich listwanites that act as horizontal traps for higher grade gold-bearing mineralisation. Increases in gold grade can be seen along the lithological contacts. Elevated grades can exist within either contact lithology. Several drillholes show a very rapid down-hole change in gold grade from mineralised to unmineralised material, indicating that mineralisation is tightly constrained instead of disseminated across the deposit. Due to these relationships, the three-dimensional model indicates that the main mineralised zone is tabular and almost flat-lying.

7.4 Geological Setting – Bayramdere Deposit

7.4.1 Geology – Bayramdere Deposit

The Bayramdere deposit is an oxide gold and copper deposit with similar geological and mineralisation characteristics to the Çakmaktepe and Ardich deposits. The geology is dominated by ophiolites that have been thrust over the limestone and dolomite, which are in turn intruded by granodioritic stocks. Gossans are generally observed as lenses and confined by normal faults.

The Bayramdere deposit is structurally controlled, displaying a replacement gold (minor copper, minor silver) mineralisation style. The deposit is dominantly represented by near-surface oxide mineralisation, primarily associated with iron-rich gossan.

The Bayramdere geological regime is considered to be the result of a mineralised intrusion generating suitable conditions for mineralisation to be localised into a favourable geological setting. A complex system of faults and thrusts have allowed mineralised fluids and diorite dykes and sills associated with the epithermal system to permeate into the stratigraphy. Key to each structurally associated style of mineralisation is the juxtaposition of ophiolites against limestone (±hornfels) to create the right geochemical conditions for the deposition of gold and other metals.

7.4.2 Mineralisation – Bayramdere Deposit

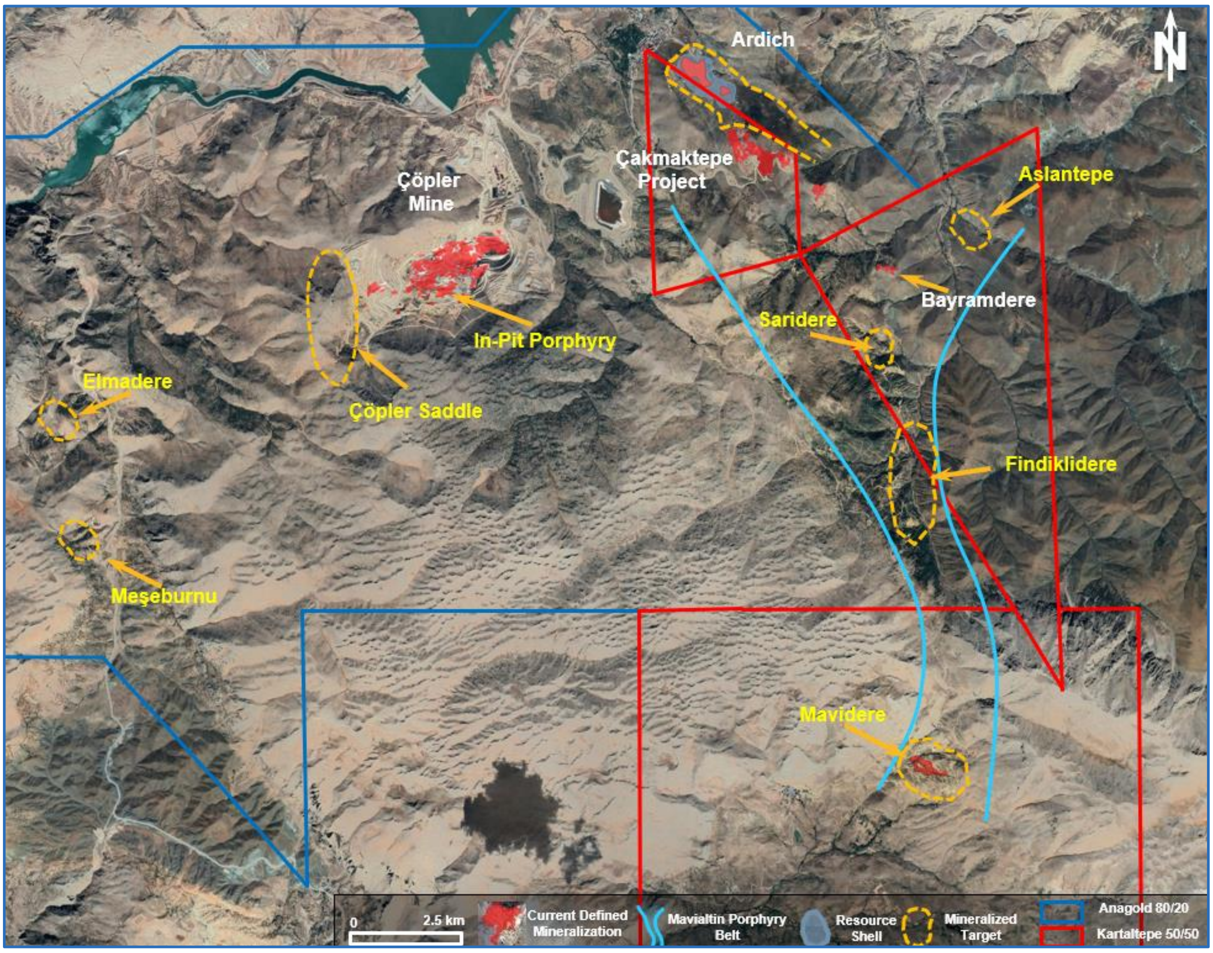
The Bayramdere mineralisation is localised within three stacked, shallow-dipping lodes that have formed at the contacts of limestone and ophiolite lithologies, with mineralisation replacing limestone along the contacts. The limestone/ophiolite contacts are low-angle thrusts, with limestone typically being trapped as wedges of material within a dominantly ophiolite stratigraphy. Mineralisation occurs within shallow iron-rich gossan horizons.

7.5 Geological Setting – Regional Prospects and Targets

Since 2000, SSR Mining exploration programmes within the Çöpler district have identified several new gold-dominant and copper–gold prospects. The gold-dominant regional prospects include the Çöpler Saddle and Elmadere. Copper–gold prospects are Aslantepe, Sarıdere, Findiklidere and Mavidere porphyries located within the Mavialtin Porphyry Belt (Figure 7.11) and the early exploration stage Meşeburnu porphyry located west of the Çöpler deposit.

Each of these prospects is discussed below.

Figure 7.11 Çöpler District Exploration Projects



Anagold, 2020

7.5.1 Geology – Çöpler Saddle

The Çöpler Saddle prospect borders the western flank of the Çöpler mine. The Çöpler Saddle is associated with a shear zone defined as an arc-like structure that trends north–south for approximately 2 km, Figure 7.11. Along the shear zone, the geology is dominated by limestone, marble, and hornfels units that are in turn intruded by small-scale microdioritic to granodioritic stocks. These lithologies were subjected to silica-clay alteration with iron oxide developments along the local structures as well as clay-pyrite alteration. At the south of the zone, silica is mainly observed as jasperoid lenses, of approximately 2 m long and 1 m wide, which occur along the hornfels and marble contacts. At the centre of the zone, less silica is observed and larger gossan-like mineralised iron oxide bodies have formed.

7.5.2 Geology – Meşeburnu and Elmadere

The Meşeburnu and Elmadere prospects (former Demirmağara project licence group) are located approximately 7 km south-west of the Çöpler deposit (Figure 7.11). The area is covered by ophiolites, limestone, and metamorphic rocks that are intruded by dioritic to granodioritic stocks. Three types of mineralisation have been identified in the area:

- Gold-bearing skarn and jasperoid occurrences along limestone and granodiorite contacts.
- Epithermal gold mineralisation developed along ophiolite, listwanite, and limestone structural contacts (referred to as Elmadere mineralisation).
- Meşeburnu copper–gold porphyry mineralisation.

Gold-bearing skarn and jasperoid occurrences were tested with drilling between 2001–2017, however only short gold-mineralised intervals were intersected. Mapping and sampling in Elmadere and Meşeburnu prospects are ongoing to define drilling targets.

7.5.3 Geology – Mavialtin Porphyry Belt Prospects

The Mavialtin Porphyry Belt is a structural corridor approximately 6–7 km wide and extending over approximately 20 km from the Çakmaktepe deposit in the north to the Mavidere porphyry deposit in the south (Figure 7.11). The Mavialtin Porphyry Belt contains the Mavidere, Findiklidere, Saridere, and Aslantepe porphyry copper–gold prospects.

7.5.3.1 Geology – Mavidere

The Mavidere porphyry copper–gold mineralisation is hosted by hornblende–biotite monzonite to monzogranite to granodioritic phases of a shallow porphyritic intrusive hosted by metamorphic and crystallised limestone. At the centre of the porphyry system, the intrusive phases were subjected to mainly potassic alteration with clay and minor sericite overprinting covering an area of 800 m x 400 m. The porphyry system appears to continue underneath the moraine cover to the east and south.

Previous exploration activities included:

- surface mapping,
- geochemistry (soil, rock, stream sediment sampling),
- geophysical studies (Induced Polarisation (IP) and surface magnetics), and
- RC and DD drilling.

The prospect was first drilled in 2001, with 1,780 m at eight locations. In 2008, 22 additional holes were drilled totalling 7,761 m, with the preliminary results announced in 2009. From 2011 through 2013, 77 DD holes totalling 20,653.3 m and 68 RC holes totalling 7,512 m were completed. Field studies and mapping in 2018 identified additional mineralised zones, some of which were drill tested in 2018 and 2019. Drillhole MD06, drilled in 2019, returned a highly prospective intercept of 269.1 m at 0.34% Cu and 0.55 g/t Au from surface.

7.5.3.2 Geology – Aslantepe

The geology of the Aslantepe porphyry copper–gold prospect is dominated by ophiolites thrusted over Jurassic to Cretaceous limestone, both of which are intruded by dioritic to granodioritic stocks and dykes. The Aslantepe intrusives outcrop in a narrow corridor subjected to propylitic, potassic, and clay alteration. The potassic zone is characterised by well-developed intense quartz–sulfide stockwork veinlets with secondary biotite, K-feldspar, and magnetite. In 2018, two additional DD holes were drilled at Aslantepe, with drillhole AT07 intersecting 63.9 m at 0.22% Cu and 0.45 g/t Au from 46.7 m down-hole. The mineralisation appears to be dipping underneath the ophiolites.

7.5.3.3 Geology – Sarıdere

The Sarıdere porphyry copper–gold prospect is covered by metamorphic limestone and ophiolite, which are in turn intruded by tonalitic to granodioritic stocks. The prospect was initially identified by stream sediment and soil anomalies. In 2018 and 2019, exploration activities identified potassic-altered porphyry intrusive outcrops covering an area of approximately 800 m x 500 m, with a phyllic alteration halo around the potassic zone of 4.3 km x 0.6 km. Seven DD holes totalling 1,461.5 m were drilled from 2007 through 2013 at the margin of the porphyry system, testing the elevated soil geochemistry. These holes intersected short intervals of copper–gold mineralisation.

7.5.3.4 Geology – Fındıklıdere

The Findiklidere porphyry copper–gold prospect is covered by massive Jurassic to Cretaceous limestone, which has been over-thrusted by ophiolites on the eastern flank. These units were intruded by fine to medium-grained tonalitic to granodioritic intrusive stocks. The porphyry copper mineralisation is characterised by well-developed stockwork quartz–magnetite–pyrite veins with copper. Peripheral iron–copper–gold skarns are observed within the limestone. In 2018, the geology, structure, and alteration was re-mapped to better understand the porphyry potential of the prospect. Results of this field work indicated that the porphyry mineralisation was potentially continuing underneath the ophiolitic body to the south-west of the known porphyry mineralisation. In 2019, DD hole FD02 was drilled to test porphyry potential beneath the ophiolitic cover. The hole was mineralised over 234.4 m (down-hole) with some higher grade intervals such as 32.1 m at 0.84% Cu and 0.37 g/t Au from 13.4 m and 16.5 m at 1.27% Cu and 0.07 g/t Au from 139.5 m.

The abovementioned drilling results were announced within the exploration press release dated 14 February 2020.

8 DEPOSIT TYPES

Porphyry copper–gold systems host some of the most widely distributed mineralisation types at convergent plate boundaries, including porphyry deposits centred on intrusions; skarn, carbonate-replacement, and sediment hosted gold deposits in increasingly peripheral locations; and high to intermediate-sulfidation epithermal deposits.

The alteration and mineralisation in porphyry copper–gold systems are zoned outward from the stocks or dyke swarms, which typically comprise several generations of intermediate to felsic porphyry intrusions. Porphyry copper (± gold, ± molybdenum) deposits are centred on the intrusions, whereas carbonate wall rocks commonly host proximal copper–gold skarns, less common distal zinc–lead and/or gold skarns, and, beyond the skarn front, carbonate-replacement copper and/or zinc–lead–silver (± gold) deposits, and/or sediment-hosted (distal-disseminated) gold deposits. Peripheral mineralisation is less conspicuous in non-carbonate wall rocks but may include base metal-bearing or gold-bearing veins and mantos (Sillitoe, 2010). Skarn deposits are typically hosted in mineralogically simple fine-grained clastic and carbonate sedimentary rocks. Skarn mineralogy and metal content is largely dependent on the crystallisation history and genesis of associated plutons (Meinert et al., 2005).

The Çöpler district is located at the edge of a convergent plate boundary. It is characterised by a complex structural history and is associated with intermediate intrusive and carbonate-rich host lithologies. As such, porphyry copper–gold systems and related styles of mineralisation are appropriate models to be applied across the Çöpler district.

The Çöpler deposit consists of three major mineralisation types that are closely associated with each other: low-grade subeconomic porphyry copper–gold–molybdenum mineralisation characterised by well-developed alteration zones and stockwork quartz veins (Main Zone); intermediate sulfidation epithermal mineralisation observed in the Manganese Zone as clusters of bright pink, banded, colloform rhodochrosite base metal sulfide veins and breccia lodes; and iron–gold (± copper) skarn with related carbonate replacement gold mineralisation.

The setting, alteration mineralogy, and mineralisation characteristics of the Manganese Zone are somewhat consistent with an intermediate sulfidation epithermal system, as defined in Hedenquist et al., (2000).

Exploration programmes modelled on epithermal-style deposits have shown success in the Çöpler district. A multi-phase porphyry model with a barren trapping system and a possible mineralised porphyry underneath it is also applicable.

9 EXPLORATION

9.1 Exploration – Çöpler Deposit

Exploration of the Çöpler deposit has been conducted by SSR Mining and its predecessors since September 1998. Work completed has included:

- geological and reconnaissance mapping,
- rock chip, grab, soil, channel and stream sediment geochemical sampling,
- ground geophysical surveys including ground magnetic, complex resistivity / IP, time domain IP and controlled source audio-frequency magneto-telluric (CSAMT) surveys,
- a regional helicopter-borne geophysical survey,
- RC and DD drilling programmes,
- acquisition of satellite imagery,
- mining technical studies,
- geotechnical and hydrogeological studies,
- environmental baseline studies,
- studies in support of project permitting,
- metallurgical testwork and studies, and
- condemnation evaluations.

The principal exploration technique used at Çöpler has been RC and DD drilling, conducted in multiple campaigns since 2000. Initially, exploration was directed at evaluating the economic potential of the near-surface oxide mineralisation for the recovery of gold by either heap leaching or conventional milling techniques.

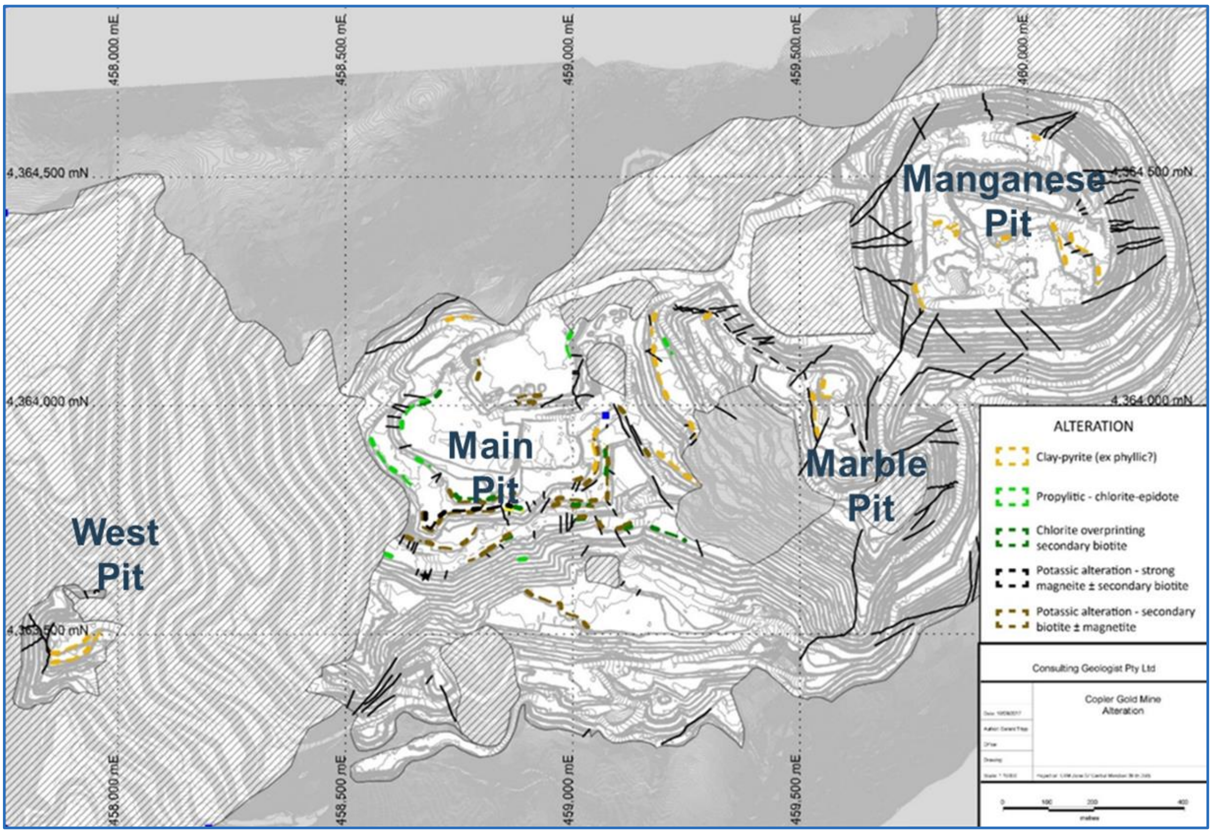
In 2013, drilling occurred primarily in the western portion of the Main Zone and on the northern edge of the Çöpler deposit. Drilling during 2014 focused on verification of existing drilling results through a twin hole programme. Drilling in 2015 provided data coverage at depth in the Manganese Zone, infill drilling in the Main Zone, and testing of low-sulfur mineralisation below the oxidation boundary. Drilling continues to better define both the oxide and sulfide portions of the deposit.

9.1.1 Geological Mapping – Çöpler Deposit

Surface mapping and sampling has been undertaken over the life of the project, culminating in a detailed geological map of the Çöpler valley, shown in Figure 7.2.

Geological mapping is used in support of exploration vectoring, exploration activities, infrastructure locations, mine planning, and environmental monitoring. One of the aims of the mapping studies was to provide sufficient information to define mineralisation types and structural settings for the Çöpler deposits. Alteration zones, such as the high-temperature porphyry alteration preserved in the southern wall of the Main Zone (shown in Figure 9.1), were identified through detailed bench wall mapping during the target generation programmes.

Figure 9.1 Çöpler Deposit Map of Alteration Minerals



Anagold, 2020

9.1.2 Geochemical Sampling – Çöpler Deposit

Extensive sampling programmes have been, and continue to be, conducted within the Çöpler area, leading to the identification of significant gold anomalies including the near-mine discovery of the Çöpler Saddle on the western flank of the Çöpler mine.

9.1.3 Geophysics – Çöpler Deposit

Various ground and airborne geophysical surveys have been conducted at the Çöpler deposit as well as across the wider Çöpler district since mid-2000. Surveys carried out include ground magnetic, complex resistivity / IP, time domain IP, and CSAMT surveys, as well as a regional helicopter-borne aeromagnetic survey that included the broader Çöpler district.

Physical property measurements are collected regularly on outcrops and DD core samples, including magnetic susceptibility, resistivity, and chargeability. Additionally, four samples from DD hole CDD067 were sent to Systems Exploration in Australia for a detailed physical property analysis.

9.2 Exploration – Çakmaktepe Deposit

The Çakmaktepe deposit and surrounding mineralised zones were identified by stream sediment samples with elevated gold geochemistry.

Drilling at Çakmaktepe started in 2012. The recent drilling (2019 onwards) has been designed to improve the known Mineral Resources identified at Çakmaktepe North. Data collected to date includes magnetic geophysical surveys, outcrop and bench wall mapping, rock and soil sampling, and both RC and DD drilling.

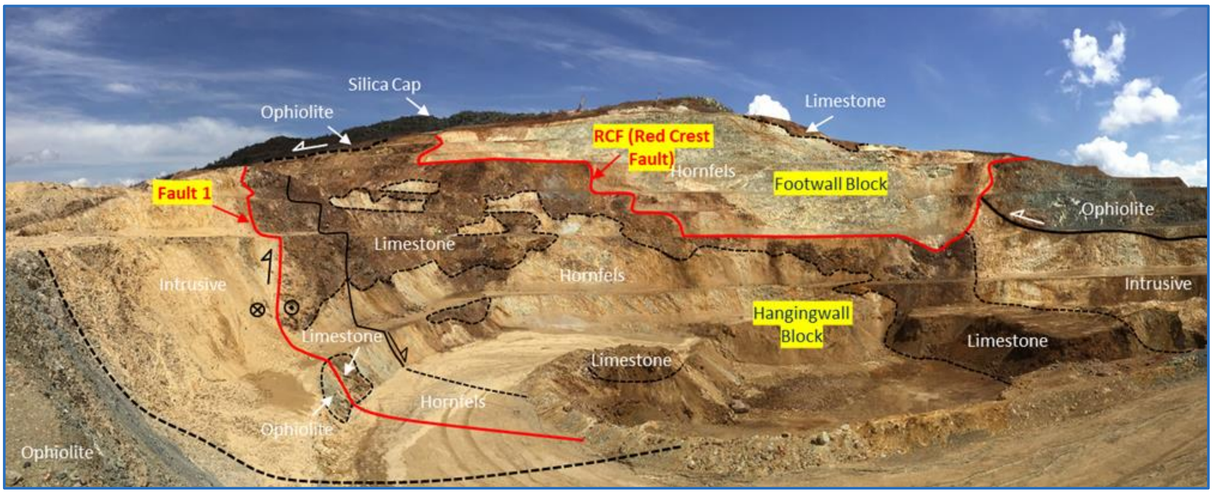
9.2.1 Geological Mapping – Çakmaktepe Deposit

The first geological mapping study in the area was conducted in 2000.

Mapping in 2014–2016 focused on deposit-wide surface geology definition at a scale of 1:1,000, reducing to 1:500 scale for the Çakmaktepe Mineral Resource area. The establishment of a network of drill tracks and pads on the sides of hills and ridges resulted in new rock exposures that have been subjected to detail geological mapping. Mapping included the collection of lithological, alteration, geochemical, and structural data.

An additional mapping study within the Çakmaktepe deposit was initiated as the Çakmaktepe operation advanced in late-2018. Details from the bench walls were collected and integrated into the drillhole dataset (mapping example shown in Figure 9.2). This has resulted in a more-accurate geological model for further pit extension exploration drilling.

Figure 9.2 Çakmaktepe Deposit – Example East Pit Geological and Structural Map



Anagold, 2020

9.2.2 Geochemical Sampling – Çakmaktepe Deposit

Geochemical sampling programmes at Çakmaktepe were initiated in 2014 and included rock chip and soil sampling (Table 9.1). Geochemical sampling was also used to define areas of alteration and mineralisation that inform additional detailed sampling surveys.

Table 9.1 Number of Geochemical Samples within the Çakmaktepe Deposit

Year	Rock Chip	Soil
2014	661	341
2015	3,527	–
2016	356	270
2017	63	1,638
2019	540	–
2020	13	–
Total	**5,160**	**2,249**

A total of 5,160 rock chip samples have been collected from the Çakmaktepe deposit since 2014. During 2019, rock chip sampling extended into bench wall and haulage roadsides to help define the extents of the deposit more accurately.

Soil sampling programmes were initiated during the 2010 exploration programme. The deposit has been fully covered with a 50 m x 50 m sampling grid totalling 2,249 samples.

Stream sediment sampling was carried out on a regional scale as part of target generation programmes since 2002. A total of 851 sediment samples have been collected.

9.3 Exploration – Ardich Deposit

Exploration activities across the Ardich deposit began in 2017 and included geological mapping, geochemical sampling, and DD drilling programmes.

9.3.1 Geological Mapping – Ardich Deposit

The Ardich deposit was discovered in 2017 during detailed geological mapping and rock sampling programmes. Results of the mapping study highlighted the potential of the Ardich deposit and its extension to the south. The mineralisation identified to date continues approximately 4 km on a north-westerly trend.

9.3.2 Geochemical Sampling – Ardich Deposit

Geochemical sampling programmes at Ardich have included rock chip / channel and soil sampling, (Table 9.2). Most of the geochemical sampling campaigns across the Ardich deposit were designed based on findings from the geological mapping programmes.

Table 9.2 Number of Geochemical Samples within the Ardich Deposit

Year	Rock Chip / Channel	Soil
2017	175	125
2018	912	–
2019	880	1,718
2020	140	–
Total	**2,107**	**1,843**

A total of 2,107 rock chip / channel samples have been collected since 2017 from outcrops across the Ardich deposit. Rock chip / channel sampling has been the most representative surface sampling, collected directly from altered rock exposures. As the drilling programmes continue, newly opened drill tracks and pads give good access to new rock exposures that are subjected to rock sampling and geological mapping.

Soil sampling was completed in early-2000 as part of a regional geochemical reconnaissance programme, with early targets being potentially mineralised listwanite-capped faults. SSR Mining started regional systematic soil sampling on 200 m x 200 m grids to cover all tenements in 2011. At the Ardich deposit, a total of 1,843 soil samples were collected on a sampling grid of 50 m x 50 m, which was reduced to 25 m x 25 m in gold-anomalous areas in 2017–2019.

10 DRILLING

All drillhole counts in this section include holes drilled for resource definition, geotechnical, and, metallurgical purposes.

10.1 Drilling – Çöpler Deposit

The Çöpler deposit continues to be tested by RC and DD drilling. The details of drillholes utilised in this Mineral Resource update for the Çöpler deposit are presented in Table 10.1. Typically, the drillhole spacing at surface is a nominal 50 m, however, in some areas the drill spacing has been reduced to 25 m (Figure 10.1).

Table 10.1 Drilling History – Çöpler Deposit

Year	Hole Type	Number of Holes	Metres Drilled	Total Metres / Year
2000	DD	4	971.5	971.5
2001	DD	10	2,254.4	
	RC	32	4,065.9	6,320.3
2002	DD	31	6,575.6	
	RC	1	120.0	
	Other	2	140.0	6,835.6
2003	DD	33	2,975.7	2,975.7
2004	DD	37	4,413.5	
	RC	228	11,036.0	
	Other	16	1,185.3	16,634.8
2005	DD	24	4,776.4	
	RC	177	29,009.7	
	Other	16	1,276.0	35,062.1
2006	DD	17	2,102.6	
	RC	94	12,878.0	
	Other	24	877.0	15,857.6
2007	DD	74	16,513.2	
	RC	125	16,998.5	
	Other	40	924.2	34,435.9
2008	DD	35	5,059.4	
	RC	41	4,904.0	9,963.4
2009	DD	23	5,789.5	
	RC	34	4,346.0	10,135.5
2010	DD	14	1,916.1	
	RC	1	144.5	2,060.6

Year	Hole Type	Number of Holes	Metres Drilled	Total Metres / Year
2011	DD	115	29,359.0	
	RC	150	17,983.0	47,342.0
2012	DD	145	50,156.5	
	RC	120	13,884.5	64,041.0
2013	DD	126	33,040.9	
	RC	53	4,545.0	37,585.9
2014	DD	12	1,296.5	1,296.5
2015	DD	59	6,214.1	
	RC	69	6,564.0	12,778.1
2016	DD	148	3,826.5	
	RC	94	2,194.0	6,020.5
2017	DD	41	3,370.5	3,370.5
2018	DD	109	10,745.0	10,745.0
2019	DD	62	7,607.7	7,607.7
2020	DD	118	15,932.0	15,932.0
Total	RC	1,237	214,896.6	
	DD	1,219	128,673.1	
	Other	98	4,402.5	
	All Types	**2,554**	**347,972.2**	

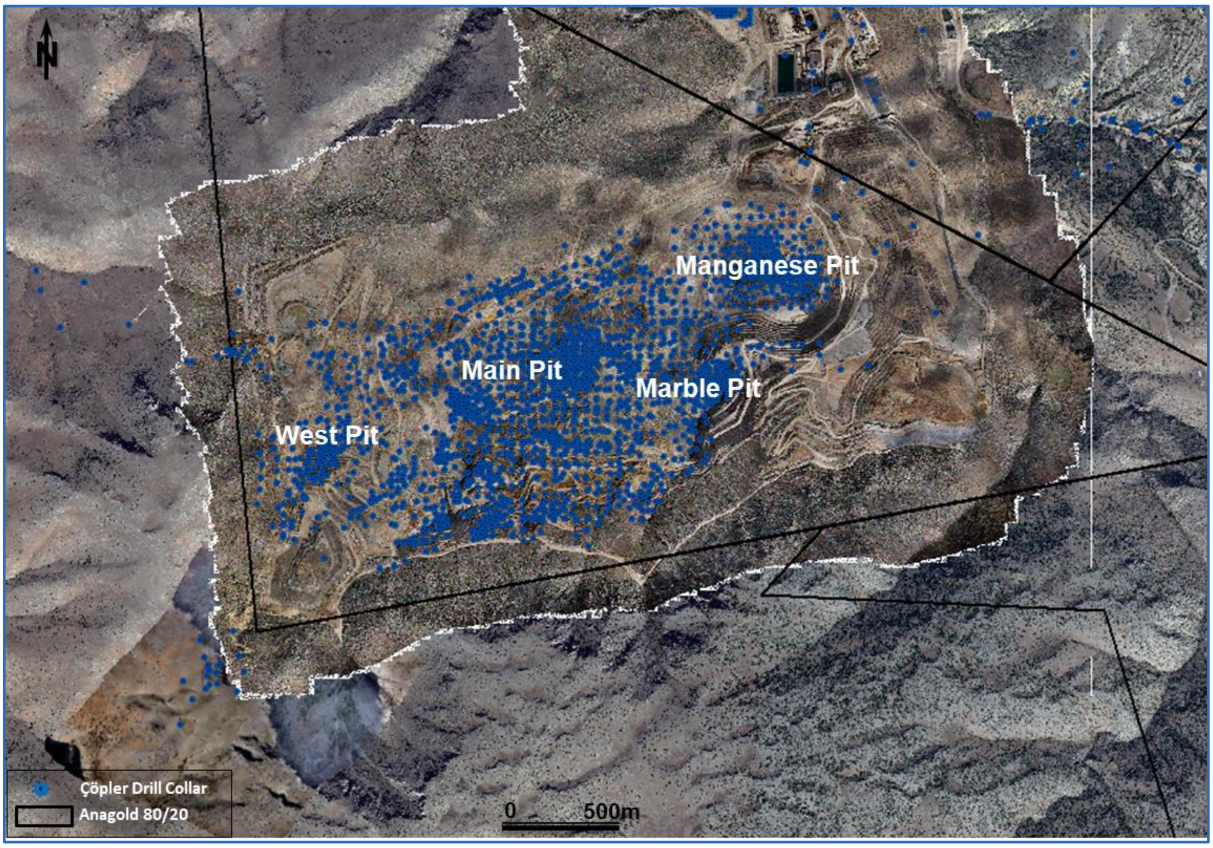
Step-out drilling at the Çöpler deposit has defined most of the lateral boundaries of the mineralisation. There has been additional development drilling, as well as condemnation drilling of areas planned for infrastructure during the last few years. In order to improve confidence in the short-range mine planning, infill drilling programmes have been conducted since 2007.

Drilling in 2014 focused on confirmation of the mineralisation with a twin hole programme.

Development drilling continued in 2015 by improving sample coverage at depth in the Manganese Zone and along structural boundaries in the Main Zone. In addition to the drilling of in situ mineralisation, a stockpile drilling programme began in December 2015 to confirm sulfide stockpile ore grade, grade distribution, and mineralogy.

Drilling in 2016–2020 mainly concentrated on target generation to increase the amount of oxide material for the production portfolio. This was focused on the Main Zone, West Zone, and the Çöpler Saddle areas. More specifically, the programme aimed to test continuation of the main gold-bearing structures based on a re-interpretation of the Çöpler structural and mineralisation settings. In-pit drilling campaigns continue with extensive exploration programmes to define additional oxide gold potential.

Figure 10.1 Drillhole Collar Location Plan – Çöpler Deposit

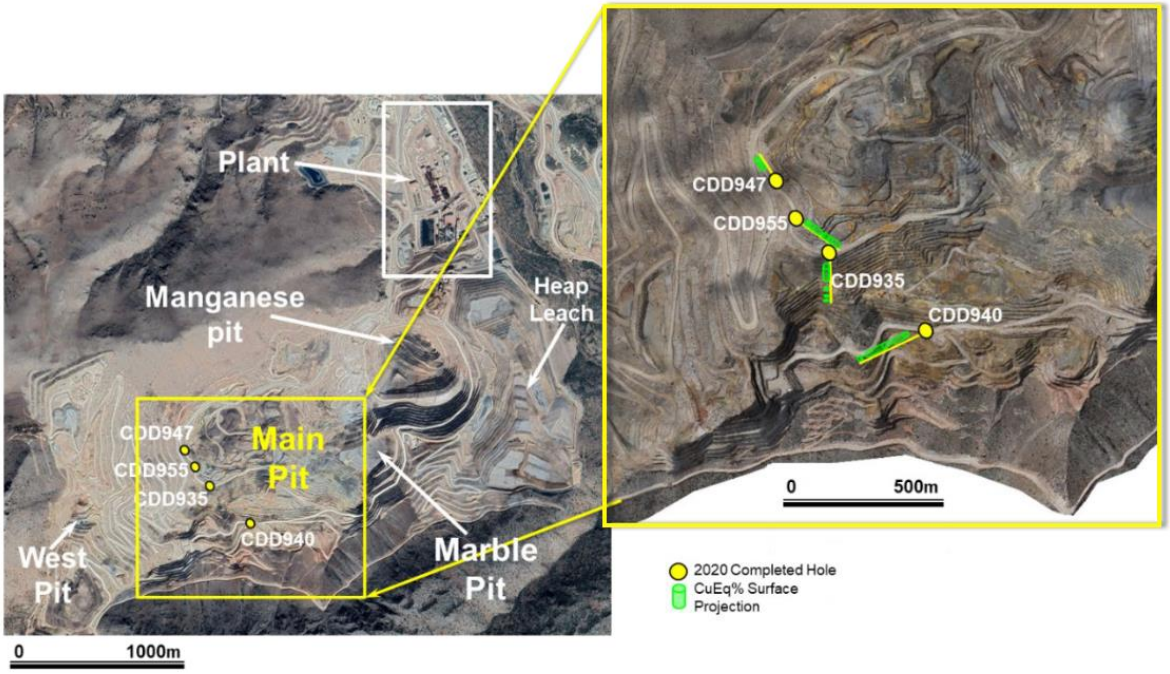


Anagold, 2020

Very recent drilling from within Main pit has identified what is known as the 'C2' target, as reported by SSR Mining in its announcement dated 25 November 2020 (SSR Mining, 2020). Four HQ DD holes totalling 1,882.5 m intersected gold-rich copper porphyry mineralisation starting at, or close to, the bottom of the ultimate Çöpler Main pit (see Figure 10.2). Significant intercepts to date include:

- CDD935: returned 0.86% CuEq over 108.6 m from 103.1 m, including 1.19% CuEq over 8.6 m from 146.3 m, 1.40% CuEq over 23.6 m from 161 m, and 1.36% CuEq over 5.3 m from 199 m.
- CDD940: returned 0.71% CuEq over 81.5 m from 271.2 m, including 1.29% CuEq over 10.8 m from 274.2 m, 1.28% CuEq over 5.7 m from 308.5 m, 1.30% CuEq over 9 m from 327.6 m, and 0.34% CuEq over 74.7 m from 359.7 m
- CDD947: returned 1.14% CuEq over 49.6 m from 156.9 m, including 1.06% CuEq over 10.0 m from 162.3 m, 1.29% CuEq over 6.7 m from 181.0 m, 2.82% CuEq over 9.8 m from 194.7 m, 1.20% CuEq over 18.4 m from 237.8 m, and 0.30% CuEq over 127.7 m from 303.3 m
- CDD955: returned 0.74% CuEq over 241.5 m from 37 m, including 1.77% CuEq over 32 m from 96.2 m, 1.92% CuEq over 17.4 m from 136.2 m, and 0.42% CuEq over 166.2 m from 287.5 m.

Figure 10.2 Drillhole Location Plan – Çöpler C2 Drilling



Anagold, 2020

Some of the newly discovered porphyry intrusive is exposed in parts of the lower benches of the Çöpler pit. The porphyry has well-developed stockwork and sheeted sulfide-rich quartz veins. Where exposed in the pit benches, these veins are locally overprinted by thicker quartz–sericite–sulfide veins. The mineralised intrusive was subjected to potassic alteration that is characterised by the development of K-Feldspar, secondary biotite, quartz, and magnetite, as veins and as replacement of earlier rock-forming minerals. Potassic alteration is overprinted by a supergene clay alteration and locally overprinted by chlorite and sericite. The copper mineralisation is predominantly chalcopyrite formed as disseminations in the matrix and as thin veins associated with quartz accompanied with rare molybdenite mineralisation. There is elevated arsenic in some places, but this does not seem to be directly correlated to the copper mineralisation. The gold mineralisation is not visible.

Drilling at Çöpler is currently ongoing, focusing on collecting additional information to assist with the evolving understanding of the extent and calibre of the C2 target.

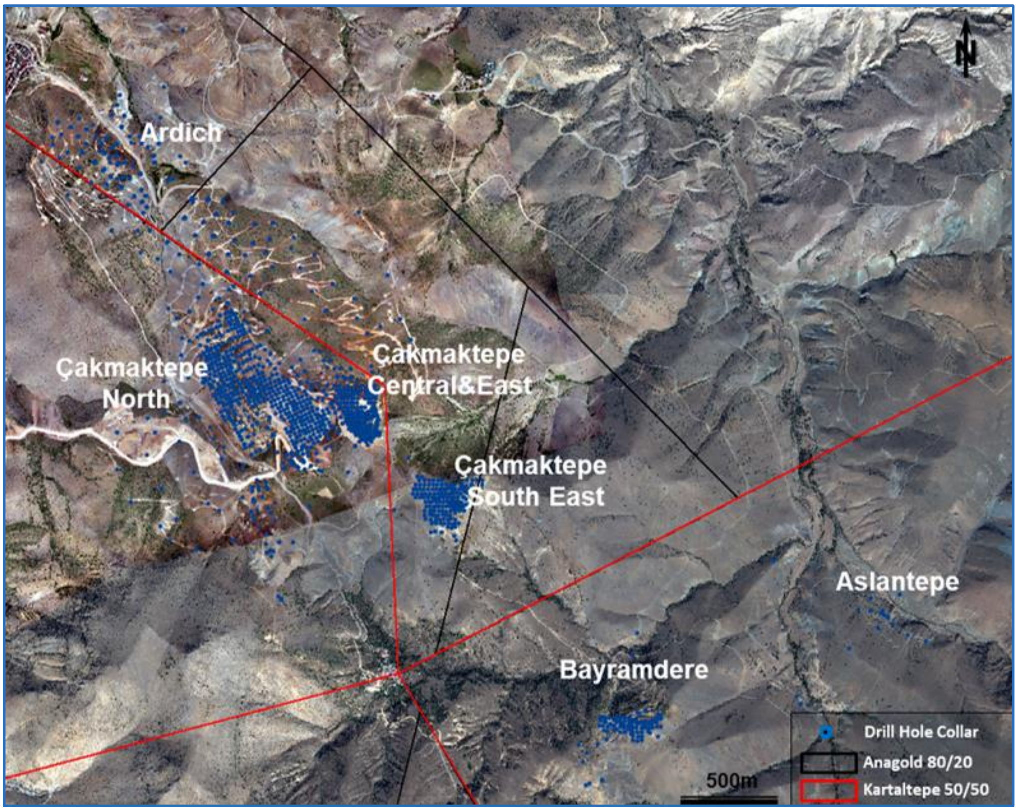
10.2 Drilling – Çakmaktepe Deposit

A total of 1,177 drillholes have been drilled at the Çakmaktepe deposit since 2012. This included 528 RC holes, 564 DD holes, and the remainder a mixture of RC and DD. Drilling to obtain samples for metallurgical testing and hydrogeological studies has also been undertaken at Ardich. As production proceeded within the Çakmaktepe Central and Çakmaktepe East pits, additional targets were generated to provide push-back options within the pit design. A total of 130 DD holes have been completed since 2019 to test for continuation of the Çakmaktepe deposit, Figure 10.3 and Table 10.2.

Table 10.2 Drilling History – Çakmaktepe Deposit

Year	Number of Drillholes	Drilled Metres
2012	21	2,287.5
2013	7	962.0
2014	162	15,976.7
2015	256	21,463.2
2016	485	64,108.6
2017	116	9,366.2
2019	75	5,919.4
2020	55	7,430.8
Total	**1,177**	**127,514.4**

Figure 10.3 Drillhole Collar Location Plan – Ardich, Çakmaktepe, and Bayramdere Deposits



Anagold, 2020

10.3 Drilling – Ardich Deposit

A total of 304 DD holes have been drilled at the Ardich deposit since late-2017, Figure 10.3 and Table 10.3. After the initial discovery of the Ardich deposit, DD drilling programmes have continued to better-define the mineralisation and to improve the Mineral Resource estimates. Drilling to obtain samples for metallurgical testing and hydrogeological studies has also been undertaken at Ardich.

Table 10.3 Drilling History – Ardich Deposit

Year	Number of Drillholes	Drilled Metres
2017	9	1,374.1
2018	91	14,216.4
2019	133	27,821.2
2020	71	18,932.7
Total	**304**	**62,344.4**

A total of 175 drillholes were included in the previously-announced Ardich Mineral Resource (announcement dated 22 November 2019, drillholes AR1–AR175). Since the data cut-off date for the October 2019 Mineral Resource, data has been obtained for an additional 129 drillholes (AR176–AR304).

A drillhole collar plot is shown in Figure 10.4, indicating the various generations of drilling.

The target of the post-2019 drilling has been two-fold:

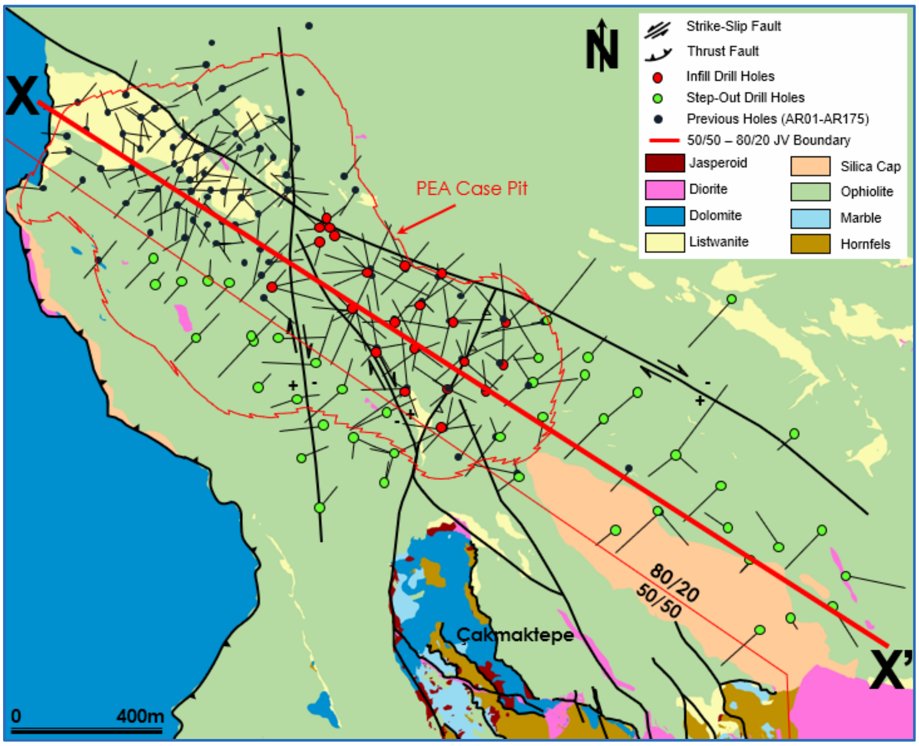
- Infill drilling within the bounds of the 2019 resource model area.
- Step-out drilling to the west, south, and south-west of the 2019 resource area.

Significant mineralisation has been intersected in the recent drilling, both within the bounds of the 2019 resource, and peripheral to the previously-modelled mineralisation.

The results from the drilling completed since 13 February 2020 provide encouragement for extension of the mineralised zones beyond the extents of the Mineral Resource reported in Section 14.3. Significant step-out intercepts to date include:

- AR274: returned 40.7 m @ 7.48 g/t Au from 154.8 m, including: 5 m @ 30.0 g/t Au from 186.5 m.
- AR273: returned 29.5 m @ 3.01 g/t Au from 191.0 m, including 2 m @ 10.16 g/t Au from 202.5 m and 3 m @ 6.78 g/t Au from 214.5 m.
- AR280: returned 24.5 m @ 4.18 g/t Au from 246.0 m, including 2.3 m @ 18.1 g/t Au from 264.0 m.

Figure 10.4 Drillhole Collar Location Plan – Ardich



Anagold, 2020
See Figure 14.32 for X–X' long-section

Drillholes AR1 through AR233 have contributed to updated (2020) resource modelling for Ardich, which is discussed at length in Section 14.3. The 2020 update resulted not only in a larger inventory than that previously-announced (22 November 2019), but is also a higher confidence inventory. The data cut-off date for updated Ardich resource model was 13 February 2020.

Once all data for the recent drilling has been obtained, SSR Mining intends to update the resource model for Ardich.

Drilling at Ardich is currently ongoing.

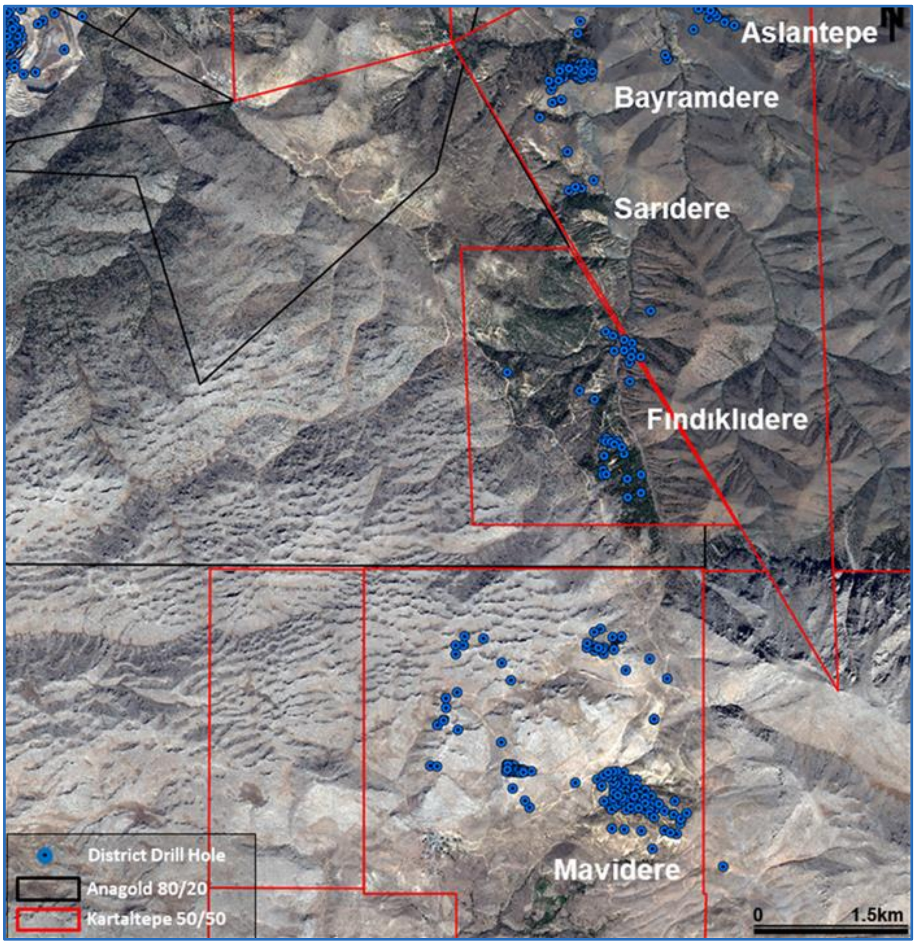
10.4 Drilling – Mavialtin Porphyry Belt Prospects

Drilling within the Mavialtin Porphyry Belt first started in early-2000. Re-interpretation of historical drillholes and detailed mapping programmes resulted in the definition of new drill targets in subsequent years. A total of 158 DD holes and 81 RC holes have been completed between 2001–2020 at various targets within the Mavialtin Porphyry Belt, Figure 10.5 and Table 10.4.

Table 10.4 Drilling History – Mavialtin Porphyry Belt Prospects

Project	Year	Number of Drillholes	Drilled Metres
Aslantepe	2014	15	2,278.7
	2018	2	440.3
	2020	1	400.8
	Aslantepe Total	**18**	**3,119.8**
Bayramdere	2007	4	763.5
	2013	28	4,024.0
	2014	68	4,698.3
	2015	17	669.9
	2016	1	98.0
	Bayramdere Total	**118**	**10,708.9**
Fındıklıdere	2008	4	1,085.3
	2012	15	5,132.0
	2013	4	1,091.2
	2014	3	825.5
	2019	5	2,501.5
	2020	1	434.0
	Fındıklıdere Total	**32**	**11,069.5**
Mavidere	2001	8	1,780.3
	2008	22	7,761.1
	2011	22	3,806.2
	2012	37	10,479.5
	2013	78	11,171.6
	2018	5	2,119.8
	2019	4	1,567.1
	Mavidere Total	**176**	**38,685.6**

Figure 10.5 Drillhole Collar Location Plan – Mavialtin Porphyry Belt Prospects



Anagold, 2020

10.5 Grid Coordinate Systems

The Çöpler project uses the European 1950 (E1950) datum coordinate system – this is a Turkish Government requirement.

The Çöpler project is in UTM6 zone 37N of the E1950 coordinate system. Until 2014, drillhole collars were surveyed by the mine surveyors in the E1950 UTM3 coordinate system and then converted to E1950 UTM6 before making them available to other personnel. The conversion from UTM3 to UTM6 was achieved by subtracting 1,746 m (–1,746 m) from the UTM3 northing coordinate and adding 17 m (+17 m) to the UTM3 easting coordinate. There is no rotation, scaling, or change in elevation between the E1950 UTM3 and E1950 UTM6 systems. Since March 2014, collar coordinates have been and are being collected in the ED1950 UTM6 coordinate system.

10.6 Collar and Down-hole Surveys

Up until 2014, drillhole collars were surveyed by Anagold surveyors using a Topcon differential global positioning system (DGPS) instrument. Approximately 4% of the drillholes up to 2014 have planned collar locations, rather than surveyed collar data. After 2014, the exploration department managed the collection of collar survey coordinates with the use of a differential GPS (DGPS). All collar survey data is checked prior to being stored within the corporate drillhole database.

Down-hole surveys are collected for all drillholes. Prior to 2009, down-hole surveys were undertaken using a Reflex Instruments Limited (Reflex) single shot down-hole camera. In 2009, a Reflex multi-shot down-hole camera was introduced on the project. Drilling contractors upgraded to a Reflex EZ Trac tool for down-hole survey data collection through to the end of 2017, thereafter, to date the majority of the drillholes have been down-hole surveyed using Reflex S Process V2.5.0650 and Devico PeeWee. Survey measurements were taken every 10 m down-hole, and data provided with raw files to record quality assurance and quality control (QA/QC) for each survey.

The depth of the surveys varies between drillholes and is dependent on the depth and angle of the drillhole.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

From 2004 to late-2012, samples were prepared at ALS İzmir, Turkey (ALS İzmir) and analysed at ALS Vancouver, Canada (ALS Vancouver), (collectively ALS Global). From late-2012 through 2014, samples were prepared and analysed at ALS İzmir. Samples in 2015 were prepared and analysed at the SGS laboratory in Ankara, Turkey (SGS). From 2015 to current, ALS İzmir is being used as the main laboratory and samples are being prepared and analysed there. Umpire analysis was completed by ACME Mineral Laboratories (ACME) in Ankara, Turkey.

SGS is certified to ISO 9001:2008 and OHSAS 18001, ALS İzmir has ISO 9001:2008 certification, and ALS Vancouver is ISO/IEC 17025:2005 accredited for precious and base metal assay methods. ACME is part of the Bureau Veritas (BV) group, globally certified to ISO9001:2008.

ALS Global and SGS are specialist analytical testing service companies, both independent of SSR Mining.

Samples from the 2000–2003 drilling programme were submitted to OMAC Laboratories Limited (OMAC) in Loughrea, Ireland. ALS Global assumed ownership of OMAC in 2011.

Detailed sampling and QA/QC procedures for RC and DD drilling were instigated and have been in use since the first drill programme. The QA/QC procedures have been retained by SSR Mining, although the insertion rates have been modified for some of the later programmes.

SSR Mining operates an on-site laboratory for assay of production samples. The on-site laboratory is certified to 17025:2017 but is not independent. It is primarily used for the analysis of grade control samples.

11.1 Sample Collection

11.1.1 Reverse Circulation Drilling Sample Collection

Historically, RC drilling was completed with a 4.5–4.75 inch (11.4–12.0 cm) diameter down-the-hole hammer. RC cuttings were passed through a cyclone with a 10 inch (25.4 cm) port for sample collection. RC drill intervals were 1 m in length and cuttings for the entire 1 m sample interval were collected from the cyclone underflow in large reinforced plastic bags. Prior to 2015, RC samples were split using a Jones splitter.

Since 2015, RC drilling has been completed with a nominal 5.25 inch face sampling hammer with centre sample return to a rig-side mounted sampling system. The sampling system included of a cyclone, sending 1 m samples to a rotary cone splitter. The rotary cone sample splitter was adjusted to maintain a representative sample volume. RC chip samples, to a weight of 3–5 kg, were collected in calico bags for analysis. All sample bags are clearly numbered and labelled with the drillhole name and sample number. Residual samples were collected in PVC bags and stored in a bag farm for six months in case re-logging, duplicate sampling, metallurgical sampling, or follow-up QA/QC was required.

The rig sampler sieves a small portion of the residual sample from the large plastic bag and places the sieved portion in a plastic chip tray to provide a sample for logging and as an enduring geological record. The plastic chip trays are photographed.

RC drilling is generally only used above the water table. The water table is closer to the surface in the northern region of the Main Zone, and for that reason, diamond drilling is the preferred method in this zone.

The following QA/QC samples are collected during the RC sampling process:

- Certified Reference Materials (CRMs) are inserted into each sample batch at a rate of two CRMs in every 40 samples (1-in-20 insertion rate).
- Prior to 2015, blank samples were inserted into each batch at a rate of one blank in every 60 samples (1-in-60 insertion rate). Since 2015, this has been changed to a 1-in-30 insertion rate.
- Field duplicate samples are collected by splitting an RC sample twice to collect two independently numbered samples of the same interval or selecting a quarter of the remnant core. Historically, field duplicates were collected and inserted into the sample job at a rate of 1-in-40 samples. In 2015, field duplicate insertion rates were increased to 1-in-20.

11.1.2 Diamond Drilling Sample Collection

Up until 2017, the diamond drilling undertaken on the project has generally been HQ or NQ diameter. HQ core has a nominal diameter of 63.5 mm while NQ has a nominal size of 47.6 mm. Approximately 90% of the DD core drilled at Çöpler and Çakmaktepe is HQ. Some drillholes are started with HQ and then reduced in size to NQ further down the hole.

Of the more recent drilling at Ardich, approximately 60% was completed with HQ core, and the remainder was mostly PQ sized core (very few holes were NQ core). PQ core has a nominal diameter of 85 mm.

Drill core is boxed at the rig by the driller and transported to the sample preparation facility on site for logging by SSR Mining exploration staff.

Logging includes the collection of lithological, alteration, and structural information. Since 2017, drill core has also undergone a detailed geotechnical logging process including a detailed 'mining rock mass rating' to 'rock mass rating' system. In addition, core samples are collected every 10 m to undertake point load IS50 testing for uniaxial compressive strength (UCS).

Diamond core that is competent is sawn in half longitudinally with a diamond saw at the core yard. Core that is broken or rubbly is sampled using a spatula to take approximately half the sample. Half the core is placed in a sample bag and the remaining half is returned to the correct position in the core tray. Sample numbers are assigned, and sample tags are placed in the sample bags and recorded in the master sample list. Sample intervals are typically 1 m down-hole.

Prior to 2015, QA/QC samples were collected routinely during the sampling process. CRMs were inserted into each sample job at a rate of 1-in-20. Blank samples were inserted into each sample job at a rate of 1-in-60. Field duplicate samples were collected by cutting the remaining half core portion into two and selecting one quarter of the remaining sample to be submitted as the field duplicate. Field duplicates are collected and inserted into the sample job at a rate of 1-in-40 samples. From 2015 onwards, the field duplicate insertion rate was increased to 1-in-20.

11.1.3 Drillhole Logging and Data Collection

RC chip samples are collected by field staff for review by the logging geologist. Similarly, core samples are metre marked by field staff in preparation for the logging geologist.

Drill core is subjected to detail logging using SSR Mining geological codes and logging formats. Information captured includes lithology, structure, alteration, mineralisation, and geotechnical data on veining, joint frequency, and joint sets.

Until September 2019, all geological data was recorded onto hard-copy logs and then transcribed into text files, using data-loading templates, ready for loading into the corporate relational SQL database. Since September 2019, hard copy logs have been replaced with data loading templates on touchpads with direct links to the company server. Files located on the server are uploaded into the corporate database regularly following appropriate checking of the data entry.

Until 2017, the SQL drilling database was managed by the SSR Mining exploration team located at the Çöpler mine site. Thereafter, the exploration database is controlled and managed by the SSR Mining exploration team located at the head office in Ankara.

11.2 Sample Preparation

11.2.1 Reverse Circulation Sample Preparation

The majority of historical RC sample preparation was completed at ALS İzmir. From late-2012 through to the end of 2013, pulp samples weighing approximately 150 g were sent to ALS Vancouver. All samples in 2014 were generated and analysed by ALS İzmir. In 2015, samples were sent to SGS for preparation and assay. Since 2015, ALS Global is being used as the main laboratory.

11.2.2 Diamond Drilling Sample Preparation

The majority of historical DD sample preparation was completed at ALS İzmir. From late-2012 through to the end of 2013, pulp samples weighing approximately 150 g were sent to ALS Vancouver. All samples in 2014 were generated and analysed by ALS İzmir. In 2015, samples were sent to SGS for preparation and assay. Since 2015, ALS Global is being used as the main laboratory.

11.3 Sample Analysis

In the period 2004–2014, samples analysed for Au at ALS Vancouver used method Au-AA25, which is a fire assay of a 30 g sample followed by atomic absorption spectroscopy (AAS). The lower and upper detection limits are 0.01 g/t Au and 100 g/t Au respectively. Samples that returned Au grades above the upper detection limit were re-analysed using the gravimetric method Au-GRA21.

Analysis of an additional 33 elements was performed using the ALS Global method ME-ICP61, which involves a four-acid (perchloric, nitric, hydrofluoric and hydrochloric acid) digestion (four-acid digest), followed by inductively coupled plasma-atomic emission spectroscopy (ICP-AES). Ag, Cu, Pb, Zn, and Mn are among the 33 elements analysed by this method.

In 2015, samples sent to SGS were analysed using the Au fire assay method FAA303, which also uses a 30 g sample and ICP-AES. Detection limits are 0.01 g/t Au. When content was detected above 3 g/t Au, method FAG303 using a gravimetric finish was added.

A 36 element analysis was performed at SGS with ICP40B method, which involves a four-acid digest (4A) followed by analysis via inductively coupled plasma-optical emission spectroscopy (ICP-OES).

From 2016 to recent, samples have been sent to ALS İzmir. Until 2019, Au-AA23 method was used, involving a fire assay of a 30 g sample followed by AAS with the lower and upper detection limits being 0.01 g/t Au and 10 g/t Au respectively. Samples that returned grades above the upper Au detection limit were re-analysed using the gravimetric method Au-GRA21. Since 2019, Au-AA24 method with a 50 g sample and lower detection limit of 0.005 g/t Au has been used. For Au grades above the upper detection limit, gravimetric method Au-GRA22 with a 50 g sample is used.

11.4 Sample Security

Drill core and RC chips are transported to the core storage facility by either the drilling company personnel or Anagold geological staff. Once at the facility, the samples are kept in a secure location while logging and sampling is conducted. The DD core storage facility is enclosed by a fence and gate that is locked at night and when the geology staff are absent. When samples are transported off site a commercial carrier is used.

11.5 QA/QC Procedures

The QA/QC programme has historically consisted of a combination of QA/QC sample types that are designed to monitor different aspects of the sample preparation and assaying process.

Blanks consist of non-mineralised samples that are submitted in order to identify the presence of contamination through the sample preparation process. Prior to 2015, blank samples comprised of commercially available pulp samples. As pulp blanks require neither crushing nor pulverising, they are of limited value in terms of identifying contamination through those aspects of the sample preparation process. Therefore, commencing in 2015, the pulp samples were switched to a coarse quartz material that would allow for better monitoring of sample contamination. Blank samples have been inserted routinely into all sample batches. If a blank sample returns an assay grade above an acceptable limit, contamination from a previous mineralised sample is assumed to have occurred at either the crushing or pulverisation stage. The first sample in a drillhole is typically a blank, after which blanks are inserted into the sample batch at a nominal rate of 1-in-60 samples. The insertion rate was updated and for the period 2015–2020 to approximately 1-in-30 for diamond drillholes.

CRM samples are inserted into sample submissions in order to monitor and measure the accuracy of the assay laboratory results over time. CRMs have been inserted into sample submissions at a nominal rate of 1-in-30. The frequency was increased from 3% to 5% in 2015. A number of different CRMs have been selected for use at varying Au and Cu grades over the life of the project. Pulp blanks have been used to determine the accuracy of assay results at very low grades, and as such are inserted using the same logic as CRMs. The combined insertion rate of pulp blanks and CRMs is a nominal 1-in-20 samples. For the period 2015–2020, the combined rate is approximately 1-in-25.

Field duplicates are used as a means of monitoring and assessing sample homogeneity and inherent grade variability and enable the determination of bias and precision between sample pairs. Field duplicates have been routinely inserted into both RC and DD sample submissions since drilling began. DD field duplicates are generated by cutting the residual half core sample into halves again and submitting one of the resultant quarters of core as the field duplicate. RC field duplicates are generated by splitting the RC sample twice to create two samples from the same interval. Field duplicates have historically been submitted at a nominal rate of 1-in-40 samples. In 2015, the field duplicate insertion rate was increased to 1-in-20. Since 2017 for DD samples, duplicate samples are being collected as laboratory duplicates instead of quarter core field duplicate samples.

12 DATA VERIFICATION

12.1 Çöpler Deposit Data Verification

12.1.1 Data Verification in Support of Technical Reports – Çöpler

Independent data verification was conducted in various stages during the compilation of technical reports on the project between 2003–2012. No material issues were identified with the supporting data.

In 2014, an independent database audit and review of available QA/QC data was undertaken to ensure the data are of sufficient quality to support resource estimation (the 2014 audit). The database audit covered data collected from 2000 to December 2013.

A further independent audit of the Çöpler deposit database as of 15 July 2015 was completed that year to verify the data are of sufficient quality to support Mineral Resource estimation of gold, copper, and silver for the Çöpler deposit (the 2015 audit). The 2015 audit focused on the 121 drillholes (12,959.8 m) completed since the 2014 audit. Available QA/QC data were evaluated to ensure the assay data are suitable to support resource estimation.

The abovementioned verification work is discussed in detail in the 2016 Technical Report.

A recent data audit covering new data obtained from 2015 through 2020 was completed in June 2020 (the 2020 audit):

- Yetkin, E., 2020 (2020a). Çöpler Project Drill Data Validation, Verification & QA/QC Review. 30 June 2020.

The 2020 audit discusses some minor inconsistencies and outliers but overall confirms the previous findings that the Çöpler drillhole data sampling and assaying is of a good standard and suitable for the purpose of Mineral Resource estimation and the reporting of exploration results.

12.1.2 Collar Location – Çöpler

The 2014 audit indicated that Anagold had not retained the original collar survey documentation provided by the mine site survey department. Additionally, collar locations were not able to be wholly confirmed because approximately one-third of the drillhole collars have been lost (either mined away or buried).

The 2015 audit recommended that SSR Mining re-survey the remaining drillhole collars, update the current database, and archive the survey coordinates because SSR Mining lacks original surveyor's records. Procedures were modified after 2016 and all collars are now surveyed using DGPS, with original survey data retained. Due to active operations, collar surveys could not be taken from the mined-out areas.

The 2020 audit identified nine drillholes that had erroneous collar coordinates and recommended that these be rectified prior to use of that data in future resource updates.

12.1.3 Down-hole Surveys – Çöpler

The 2014 audit indicated approximately 32% of the holes had a recorded down-hole survey, while the remaining holes used the planned drill azimuth and inclination. The 2014 audit was unable to verify the down-hole surveys because SSR Mining did not have the original films, records, or documents available.

The 2015 audit recommended that SSR Mining initiate a procedure to retain the down-hole survey data as they are collected. The data is to be reviewed by the responsible geologist, then signed, dated, and added to the drillhole folder. The 2015 audit also recommended that SSR Mining apply the proper magnetic declination correction of 5.6° East rather than the 3.0° East correction currently being applied. The declination correction varied from 4.5° East in 2000 to 5.6° East in 2014. The correction applied should be based on the year the data were collected. After 2016, correction of magnetic declination is completed on an annual basis.

The actual end-of-hole location for 245 drillholes was compared to the planned end-of-hole location in the 2014 audit. The average absolute variation was 3.9 m east–west, 5.8 m north–south, and 3.0 m in the vertical directions. This variation is within the resource model cell dimension of 10 m x 10 m x 5 m (Çöpler model); however, the audit recommended that all DD holes with lengths of greater than 300 m should be surveyed down the hole.

In 2015, a check on the corrected database indicated that no drillholes contained excessive deviation.

The 2020 audit reported that all holes are currently down-hole surveyed using a north-seeking Gyro instrument (Reflex), with the only exceptions being those holes with technical issues such as rods stuck or hole collapses. A comparison of successive down-hole survey readings for a given drillhole was undertaken using a maximum 5° variation over 30 m (0.17°/m) in either inclination or azimuth to flag records with excessive deviations. A total of 61 spurious readings were deemed to be out of acceptable limits, and a recommendation was made to remove those data from the overall database.

12.1.4 Geology, Density, and Geotechnical Logs – Çöpler

In the 2014 audit, SSR Mining was not able to provide all the requested geology, geotechnical, and density logs to support the audit due to missing drill logs. The 2016 Technical Report recommended that SSR Mining attempt to locate original logs for the missing holes. The 2016 Technical Report also recommended the SSR Mining senior exploration geologists review, sign, and date the final logs. Since that time, all scanned original logs and original data entry worksheets are kept in the drillhole folder, and final review and sign-off by a senior / principal geologist is required.

In 2015, no material errors were identified.

Yetkin, (2020a) compared original lithology log sheets with respect to the lithology table in the database. A total of seven lithology codes in 165 entries did not have a match in the current lookup table. A total of eight drillholes had missing intervals in the lithology logs. Some mismatched entries were noted in from/to, geology codes, and oxidation state.

Since 2017, SSR Mining has extended the geotechnical logging programme to include all holes drilled to obtain more data for the geotechnical characterisation of site. A comprehensive mining rock mass rating programme was established and included in the logging activities. Until detailed domain descriptions were achieved, mining rock mass rating logging for each exploration hole continued. In total, 295 holes were logged and stored within the database. In addition, an Optic Televiewer tool provided rapid and high-resolution oriented images of the drillhole. This technique was integrated into the exploration programme and used on selected holes within each zone at Çöpler. Reliable and accurate data were collected and assisted in the interpretation of kinematic analysis for mineralised / non-mineralised zones.

Regardless of material type, samples are collected every 3 m for density measurements. During the core cutting process, samples are cut in half with one-half going in the analytical sample bag and the other half being placed in an aluminium ‘boat' with a labelled aluminium tag for the density measurements. Samples are placed in an oven at 90°C for 24 hours to ensure sufficient dryness and weighed in air (A), then coated in paraffin wax to ensure all open pore spaces are sealed with wax. The wax-coated sample is then weighed in air (B) and weighed submerged in the water below the balance (C). The sample is then returned to the core box.

$$\text{Density} = A / (B - C - [\,(B - A) / 0.86^{*}])$$ *where * 0.86 = density of wax*

Density data were reviewed in the 2020 audit. There are a total of 23 readings in the 2015–2020 dataset that are in the outlier range of less than 1.5 t/m^3 or greater than 4.0 t/m^3. These data are to be considered when undertaking analysis into potential bottom and top cuts for data to be used in future resource updates.

12.1.5 Assays – Çöpler

12.1.5.1 2000–2003

Assay laboratory certificates from the OMAC laboratory for drilling prior to the 2004 were not available. ALS Global assumed ownership of OMAC in 2011. Anagold obtained the electronic records from OMAC, however, the laboratory certificates were not able to be located.

The 2016 Technical Report used statistical methods (histograms and quantile-quantile (QQ) plots) to validate the OMAC data against the ALS Global data and found the data to compare well. A divergence at approximately 4.0 g/t Au is seen in the QQ plot is explained by the inclusion of a few higher grade composites. These composite grades were confirmed by subsequent SSR Mining drilling in the vicinity. OMAC drilling data represents 6% of the total metreage within the resource modelling database extract.

12.1.5.2 2004–2015

ALS Global assay results for the period 2004–2013 for Au, Ag, As, Cu, Fe, Mn, S, and Zn were compared to the Anagold database. The results of the comparison are presented in Table 12.1.

Table 12.1 ALS Global Assay Audit Summary – Çöpler

Element	Number of Assays	Number of Differences	% Difference
Au	193,255	1,979	1.00
Ag	191,215	562	0.30
As	191,215	865	0.50
Cu	191,215	1,457	0.80
Fe	191,215	822	0.40
Mn	182,619	3,362	1.80
S	192,215	692	0.40
Zn	192,215	1,030	0.50

The higher error rate observed for Mn is due to conversion of MnO assays to Mn values. A conversion factor of 0.7745 was used to convert MnO assays to Mn. It did not appear, however, that a constant conversion factor was applied to the values in the Anagold database.

The 2015 audit compared Au (and cyanide-soluble Au assays where available), Ag, As, Cu, Fe, Mn, S, and Zn from 995 samples analysed by ALS Global and noted only one error: the database contained a Cu assay of 1.0% for sample number 333474 rather than the correct value of 1.074% (ALS Global certificate IZ140478). A further comparison was undertaken of Au (both fire assay (FA) and cyanide-soluble), Ag, As, Cu, Fe, Mn, S, and Zn from 11,228 samples analysed by SGS and noted 53 errors for Au and nine errors for Cu. A list of sample numbers, assay values, and associated SGS certificates was sent to SSR Mining staff for review and to be used to update the database.

12.1.5.3 2015–2020

The 2020 audit reviewed the assay database as at 3 June 2020 including drillholes CDD593 through CDD904 (n=284) comprising 31,757 assay records. The database includes only DD holes.

There are a total of 1,309 duplicate intervals identified including drillholes CDD687 through CDD691, CDD693 through CDD708 and CDD710 through CDD715. The assays of these drillholes with duplicate intervals belong to ALS Global and the on-site laboratory. As the on-site laboratory's assays do not have multi-element assays it is recommended that the data for these duplicate assay intervals be removed from the database.

There are no missing assay values identified for Au (FA). Missing assays for Ag (4A) and Cu (4A) solely belong to drillholes CDD681–CDD686 where the analyses were carried out at the on-site laboratory. Missing assays for S (4A) were both for samples assayed in SGS (CDD593 through CDD625) and at the on-site laboratory. It is understood that the cyanide leach Au assay (AuCL), sulfur infrared combustion analysis (S (Leco)), sulfide sulfur infrared combustion analysis (SS (Leco)), and carbon infrared combustion analysis ((C (Leco)) are run on a selective basis. For the period where SGS was being used as the main laboratory, S (Leco) analysis was used instead of S (4A).

There were four different laboratories used for assays and geochemical analyses during March 2015–April 2020, these were:

- SGS
- ALS Global
- BV (ACME)
- On-site laboratory

The variety of laboratories resulted in a variety of method codes for fire assay, four-acid digestion, multi-element, and Leco analyses.

For DDs the sample is prepared as half core. The highest 1% of assays (AuFA) were checked for transcription errors. There are no major errors identified.

Consistency checks were performed on the assay table. All consistency checks passed other than the duplicate intervals noted, and the missing assay intervals highlighted in the previous table.

12.1.6 Witness Samples – Çöpler

Ten witness samples obtained from blast hole cuttings were submitted to both the on-site laboratory and to ALS Global in 2014. The average of the ALS Global Au assay results is 8% higher than the mean of the results provided by the on-site laboratory. If the result from one high-grade sample (above 4 g/t Au) is removed from the comparison, the average ALS Global Au grade is 3% higher than for the on-site laboratory. This is considered acceptable agreement between the two laboratories.

12.1.7 Quality Assurance / Quality Control (QA/QC) Results – Çöpler

12.1.7.1 Screen Analyses – Çöpler

As part of the 2014 audit, 1,724 crusher screen test results were reviewed. The screen test results were obtained from 387 ALS Global certificates that reported the percent passing a 2 mm screen. All but eight samples exceeded the specification of 70% passing 2 mm. A review of 3,945 pulveriser screen test results was made from 750 ALS Global certificates for material passing a 75 µm screen. There were 443 samples (11%) that did not meet the specification of 85% passing 75 µm. There is a marked improvement in pulverisation starting approximately July 2013.

There were very few ALS Global screen test results from 2014, but a review of 2015 crusher and pulveriser screen test results from SGS showed that all of the 681 crusher screen test results met the specification of 70% passing 2 mm and only one of the 680 pulveriser screen test results failed to meet the specification of 85% passing 75 µm.

12.1.7.2 Certified Reference Material (CRM) Samples – Çöpler

A total of 26 different CRMs have been used at Çöpler, all obtained from Ore Research and Exploration P/L, located in Australia (OREAS)).

A review of the CRM results from the samples submitted to OMAC (2000–2003) indicated that acceptable accuracy was achieved by OMAC: in a dataset of 651 Au CRMs and blanks, analyses for 97% fell within the ±2 standard deviation (SD) accepted range.

SSR Mining used 11 different CRMs in the period 2013–2015. These were inserted at a frequency of 5%. However, three CRMs were primarily used to monitor assay accuracy, these were: OREAS152B, OREAS502B, and OREAS504B. An overall relative bias for these CRMs is within 5% and concludes the assay accuracy is sufficient for resource estimation.

The 2020 audit summarised the CRM performance by element. A total of 25 different CRMs were used during this period. ALS Global, SGS, BV (ACME), and the on-site laboratory all demonstrated an acceptable overall performance for the listed Au CRMs used during the programme. The performance of CRMs for Ag, Cu, S, and C was also reviewed. Other than the noted outliers, in general ALS Global, SGS, and BV (ACME) had an overall acceptable performance for these elements.

The 2020 audit recommends that timely monitoring of the CRM performance will ensure that the replicate assays stay within range, that systematic analytical drift is promptly corrected, and that mis-labelled samples are promptly identified. The extreme-outlier cases need to be investigated and if these are found to be mis-labelling then the organisational procedures should be reviewed and updated. If it transpires that these are not mis-labelled samples and the errors are found to be laboratory-related, then re-assay procedures needed to confirm the assays for the relevant batches.

12.1.7.3 Blank Samples – Çöpler

The 2014 audit reviewed the results from 2,437 blank samples from 10 blank material sources, which were blindly inserted into drill sample submissions. Although the results indicated that there was likely some carry-over contamination of gold, the amount of contamination was not considered to be sufficiently high to materially affect project assay results.

The 2015 audit reviewed 264 blank samples from two blank material sources, which were blindly inserted into drill sample submissions. Based on these sample results, there did not appear to be any indication of sample contamination. However, based on the number of blank samples, it appeared only 1-in-60 samples was submitted as a blank. This is below industry-leading practices of a submission rate of 1-in-20 samples.

The 2020 audit outlined the blank sample rate as approximately 1-in-30. The coarse blank material currently used is commercially purchased from ALS İzmir. SGS blanks (a total of 106) were all below threshold values for Au, Ag, and S (Leco). SGS blanks returned a total of three results that were slightly higher than the threshold of 5 ppm Cu and one result that was slightly higher than the threshold of 0.05% for C (Leco). On-site laboratory blanks (a total of 13) were all below threshold values for Au (fire assay and cyanide leach) and S (Leco). For C (Leco) there was only one result above the threshold value of 0.2% C (Leco). BV (ACME) blanks (a total of eight samples) were all below threshold values for Au, Ag, Cu, S (4A), S (Leco), and C (Leco) analyses. The number of blank samples submitted for the umpire laboratory at BV (ACME) should be between approximately 3% to 5% of all samples sent.

12.1.7.4 Duplicate Samples – Çöpler

During 2000–2003, coarse reject and pulp reject duplicate samples were submitted. An issue was noted, possibly as a result of coarse gold in the coarse rejects. The pulp reject duplicates showed excellent agreement.

The 2020 audit stated that the overall duplicate sampling ratio as approximately 15:1, which is in line with industry standards. During this period the main laboratory changed from SGS (Ankara) to ALS İzmir. Also, the duplicate sampling procedure changed from field duplicate to laboratory duplicate. These changes are very well observed in the overall reproducibility and sampling homogeneity displayed by the absolute relative difference parameter. Maximum absolute relative difference in AuFA duplicate pairs is 226% from SGS and 54% from ALS İzmir duplicate sample populations. This reflects the change in the duplicate sampling procedure where the field duplicates show the sampling variance plus geological variance between the sample pairs, but the laboratory duplicate of coarse rejects have increased homogeneity therefore lower variance in assays of the duplicate pairs.

12.1.7.5 Check Assays – Çöpler

For the 2000–2003 drilling, 403 check samples of prepared coarse reject material and 203 samples of fine reject material were submitted for check Au (±Cu, ±Ag) assays at OMAC, ALS Global, and Bondar Clegg. This work was carried out as a quality control review of both the sample preparation at ALS Izmir and also the accuracy of analyses at OMAC. Excellent agreement was found between intra-laboratory duplicate AuFA analyses carried out at OMAC, and inter-laboratory analyses between OMAC, ALS Global, and Bondar Clegg.

It does not appear that check samples were submitted from 2005–2009, or from 2011–2014. Historical pulp and sample reject material pre-2013 are no longer available, therefore check assays cannot be submitted for this period. There were 308 samples (3.5%) selected from the 2009 and 2010 drill programmes. These samples were submitted to ACME for analysis. Both pulp rejects and field duplicates were submitted as check samples.

Based on 111 results, the Au assays from ALS Global in 2009–2010 were biased 6% high compared to ACME for the RC holes. Based on 51 results, the Au assays from ALS Global are biased 8% higher than ACME for the DD holes.

In 2015, SSR Mining submitted 318 samples to BV (ACME) as check samples. This submission included 301 check samples (pulps), 11 CRMs, and six blank samples. Review of the Au, Ag, Cu, and S results indicates that SGS is biased 2.2% low for Au, 6.9% low for Ag, and 11% high for S, when compared to BV (ACME). There was no bias noted for Cu results.

Au and Ag are within the ±10% limit commonly used by industry to determine whether check results are acceptable or are of concern. S is very close to the ±10% limit.

The 2020 audit reviewed a total of 1,267 pulp duplicate samples were submitted for umpire assaying, which represents 5% of the total sampling. Although the main laboratory changed from SGS to ALS Global during that period, the umpire laboratory (BV (ACME)) did not change. A total of 216 drillholes were included in the umpire dataset out of 284 for this period. The umpire duplicate assays confirm the reproducibility of the primary laboratory Au analyses. There are isolated higher difference assays among the umpire dataset, especially in the lower grades, but the averages show a high correlation.

12.1.8 Discussion – Çöpler

The independent QA/QC review confirms that the Çöpler drillhole data sampling and assaying is of a good standard and suitable for the purpose of mineral resource estimation and the reporting of exploration results. This is especially true for gold, which is the primary metal of economic interest. The confidence in the silver, copper, sulfur, and carbon analyses is at a level that at minimum supports modelling for geometallurgical and by-product metal characterisation.

12.2 Çakmaktepe Deposit Data Verification

12.2.1 Data Verification in Support of Technical Reports – Çakmaktepe

Independent data verification was conducted on the Çakmaktepe drilling databases and available QA/QC sample data for drilling completed from the first Çakmaktepe hole drilled on 27 September 2012 to the established data cut-off date for the Mineral Resource modelling of 9 December 2019.

This verification was completed in three campaigns as drill programmes progressed, and is reported in three reports:

- Cube Consulting (2016a). Çöpler Near-Mine Projects Data Verification for Mineral Resource Estimation. 29 April 2016.
- Mineral Consultancy, 2017. Çakmaktepe Drill Data Validation, Verification & QA/QC Review. 17 November 2017.
- Yetkin, E., 2020 (2020b). Çakmaktepe Project Drill Data Validation, Verification & QA/QC Review. 29 February 2020.

It was concluded that the Çakmaktepe drillhole data sampling and assaying is of a high standard and suitable for the purpose of Mineral Resource estimation and the reporting of exploration results.

12.2.2 Collar Location – Çakmaktepe

Collar positions were verified against the pre-mine topographic surface DTM to check for inconsistencies in elevation. The threshold difference between the DTM and the drillhole collar elevation used for validation was variable over the different reviews:

- ±2 m in 2016
- ±4 m in 2017
- ±3 m in 2020

Five 2015 holes were found to be outside the then current tolerance limit, but all of these are within ±4 m, which is the uppermost limit used in the overall data verification. Seventeen recent holes were found to have a difference outside the 2020 tolerance limits, some substantially so (up to 38.08 m).

As mining is currently halted at Çakmaktepe, these discrepancies can be resolved by re-surveying the collar locations.

12.2.3 Down-hole Surveys – Çakmaktepe

The majority of the Çakmaktepe drillholes AR176 thru AR233 were downhole surveyed using a multi-shot (Devico or Reflex) with readings spaced at 10–12 m on average (range of 2–93 m).

Thirty-one holes (6%) were found to have no down-hole survey data.

A comparison of successive down-hole survey readings for a given drillhole was undertaken using a maximum 5° variation over 30 m (0.17°/m) in either inclination or azimuth to flag records with excessive deviations. A total of 50 spurious readings were deemed to be out of acceptable limits, and a recommendation was made to remove those data from the overall database.

The recommended magnetic declination correction discussed in Section 12.1.3 has been implemented for Çakmaktepe data.

12.2.4 Geology, Density, and Geotechnical Logs – Çakmaktepe

The drillhole database lithology table was checked for alphanumeric categorical code validity and interval reporting consistency with the log key sheets. Several mis-matches were identified, mostly arising from the incorrect combination of upper and lower-case and one new code introduced. All other entries were found to be identical to the codes provided in log key sheets. Seven intervals were shown to have missing lithology records.

Some minor discrepancies were identified in other coding in the database, such as upper-case ‘redox’ records and mixed-case ‘geogrp’ codes used, causing different unique categories to be created, and a new code created in the ‘redox’ table that does not appear in the log key sheet.

Density data were reviewed during all three of the Çakmaktepe verification campaigns. Density measurements are collected using the same process described in Section 12.1.4 for the Çöpler deposit. A systematic truncation from four decimal places to three decimal places was observed, and several transcription errors in FROM and TO records were identified. Manually calculated spot check values were within ~2% of the density reading supplied in the resource database. The density samples are representative in a spatial and geological context. On a total project basis, there are no obvious density outliers.

12.2.5 Assays – Çakmaktepe

There were two different independent laboratories used for assays and geochemical analyses for the entire Çakmaktepe database to date, these were:

- ALS Global
- BV (ACME)
- SGS

The variety of laboratories resulted in a variety of method codes for fire assay, four-acid digestion, multi-element, and Leco analyses.

In consistency checks on the 'tblVWDHAssays_ALL' assay table, four samples were found to have missing assay entries. The highest 1% of assays were checked for transcription errors, with no major errors identified, although some gravimetric results were not given priority over fire assay results.

12.2.6 Witness Samples – Çakmaktepe

No witness samples are known of for Çakmaktepe.

12.2.7 Quality Assurance / Quality Control (QA/QC) Results – Çakmaktepe

Çakmaktepe QA/QC data was independently reviewed on a campaign basis at milestone times in the evolution of the exploration programme. There are currently three individual reports describing the results. The collective results are reported in this section.

The Çakmaktepe QA/QC programme follows suggested guidelines for QC sample insertion rates:

- 3%–5% CRMs and blanks
- 5%–10% field duplicates
- 3%–5% pulp duplicates
- 5% of coarse rejects/pulps to a third-party external laboratory

12.2.7.1 Screen Analyses – Çakmaktepe

No screen analysis has been undertaken to date on Çakmaktepe material.

12.2.7.2 Certified Reference Material (CRM) Samples – Çakmaktepe

A total of 39 different CRMs have been used over time at Çakmaktepe. The earlies review identified 28 different CRMs, but this has been reduced to a more manageable ten types in the most recent campaign. Of the ten, four are Au-only, six include Cu, S, and ±Ag, four of which include S (Leco) and two of those include C (Leco).

The principal assay laboratory for drill samples has changed over time:

- ALS Chemex prior to 2015
- SGS Ankara 2015–2016
- ALS İzmir 2017–onwards

Umpire samples were principally submitted to the BV (ACME) laboratory.

Au CRMs were submitted across the entire Çakmaktepe database. The average insertion rate was of the order of 3.4%, which meets the guideline.

The performance of the CRM sample data was assessed by plotting the laboratory assay values for Au (FA and CL), Ag (4A), Cu (4A), S (4A and Leco), and C (Leco) of the CRMs against time on control charts.

A review of the CRM results from the samples submitted indicated that:

- Prior to 2015, ALS Chemex showed a consistent negative bias across most CRMs, with three CRMs having a negative bias of greater than 5%. The negative bias decreases with increasing grade of the CRM and is most apparent in the grade range of less than 0.5 g/t Au.
- In 2015–2016, SGS consistently had good to excellent performance over the range of grades for the six Au CRMs in use.
- After 2017, ALS had an acceptable overall performance, although consistently had issues with isolated ±2SDs as well as seven failed cases of ±3SDs, all of the fails likely being mis labelled samples.

BV (ACME) showed consistently good results.

Timely monitoring of the CRM performance will ensure that the replicate assays stay within range, that systematic analytical drift is promptly corrected, and that mis-labelled samples are promptly identified. Any extreme-outlier cases need to be investigated and if these are found to be mis-labelling then the organisational procedures should be reviewed and updated. If it transpires that these are not mis-labelled samples and the errors are found to be laboratory-related, then re-assay procedures needed to confirm the assays for the relevant batches.

12.2.7.3 Blank Samples – Çakmaktepe

Blanks were inserted into the sample stream as a check for cross-contamination during sample preparation. The insertion rate was of the order of 2.7%, which does not quite meet the guideline.

Au assays for blanks were assessed by charting the laboratory assay values and assessing performance versus the maximum accepted threshold value of 0.05 g/t Au, which is 10 times the lower detection limit (DL). The threshold value was 0.1 g/t Au prior to 2015.

All Au blank assays but two were within acceptable limits, and one of the fails was considered to be a mis-labelling issue.

The most-recent campaign reviewed blanks performance for Ag, Cu, S (4A and Leco) and C (Leco). Ag, all blank assays were below the maximum threshold value of 0.5 g/t Ag. For Cu, the threshold level is 10 ppm Cu and there was only one sample that assayed 40% above the threshold value. For sulfur (4A and Leco) the threshold value is 0.1% S and all blank sample assays were below this value. For carbon (Leco) the threshold value is 0.1% C and there are five assays above the threshold. These results show that the blank material used to monitor Au and other elements may not be suitable for C analysis, or that the samples are contaminated during the sample preparation. Other than these no obvious contamination issues are apparent within the assay database.

There were no blank samples submitted within the umpire sample set to BV (ACME). This does not comply with the guideline.

12.2.7.4 Duplicate Samples – Çakmaktepe

Duplicate sample data was analysed to determine the reproducibility of assays according to the combination of geological, sampling, and analytic variances. The insertion rate was of the order of 5%, which meets the guideline.

The measure of acceptable duplicate results has changed from campaign to campaign:

- In the 2016 review, acceptance was based on the relative mean paired difference (RMPD) scatterplot and the and the measurement of the relative precision error between pairs based on the average coefficient of variation (ACV). The sample precision for the RC and DD results are acceptable for this style of mineralisation with ACVs of 30% and 37% respectively.

  The DD duplicates have a higher variability than the RC samples partly as a result of the differing sample volumes between the original ½ core sample and the duplicate ¼ core sample. The smaller duplicate sample size and inherent nugget effect is contributing to the poor precision. In addition, the ¼ core duplicate samples are showing an overall negative bias of 6% when compared to the original sample, a result of the smaller sample volume failing to adequately capture the sparse particulate gold distribution and biasing results to the low side. The improved precision of the RC samples is a reflection of the increased sample volume when compared to the DD duplicate samples, which allows more representative sampling of the gold grade population. The RMPD plots for the two drill types show that lower precision or poorer repeatability for gold is particularly evident below approximately 1.5 g/t Au. This may be partly the result of an analytical precision error identified at lower grades.

- In the 2017 review, the DD duplicates have an average absolute relative difference of 0.355 for Au, which falls above the rule-of-thumb of 0.20–0.30 absolute relative difference range for acceptable laboratory duplicate samples. The Au assay variance for DD duplicates is –8.9%, which falls within the rule-of-thumb of ±10% precision window. The RC duplicates have an average absolute relative difference of 0.240 for Au, which falls within the rule-of-thumb of 0.20–0.30 absolute relative difference range and the Au assay variance for RC duplicates is –0.9%, which falls within the rule-of-thumb of ±10% precision window.

The DD duplicates have a higher variability than the RC samples partly as a result of the differing sample volumes between the original ½ core sample and duplicate ¼ core samples. The smaller duplicate sample size and inherent nugget effect is contributing to lessened assay reproducibility. The improved precision of the RC duplicates is a reflection of the increased sample volume and inherent sample homogeneity of RC chips when compared to DD duplicate samples, which allows more representative sampling of the gold grade population.

- In the 2020 review, the DD duplicates have an average absolute relative difference of 0.027 for Au and 0.018 for S (4A), both of which fall within the rule-of-thumb of 0.10–0.20 absolute relative difference range for acceptable laboratory duplicate samples (note the tighter range for acceptable results relative to 2017). The Au assay variance for DD duplicates is –2.3% and the S(4A) is 1.8%, both of which fall within the rule-of-thumb of ±10% precision window.

The high precision of the duplicates reflects the inherent sample homogeneity of laboratory-prepared duplicate samples from coarse rejects, which allows more representative sampling of the grade population.

12.2.7.5 Check Assays – Çakmaktepe

All three QA/QC campaigns report the results of umpire assays with pulp duplicates submitted to BV (ACME) for independent analysis.

The rate of check assay was lower for the first campaign (2%), but the overall average is 4.6%, which is approaching the guideline.

In the first campaign (2017), the overall precision for the pulp duplicates had an average coefficient of variation (ACV) of 22%, which is above the upper level of the acceptable limit (10%–20%) for this type of duplicate sample. The SGS laboratory shows a higher variability, with an ACV of 36%, which is double that of the ALS laboratory with an acceptable ACV of 18%. The analytical precision issue at SGS is not apparent within the CRM data, suggesting the problem may lie with sample preparation and the production of pulps with poor homogeneity. The poor repeatability is most pronounced in the grade range below 0.6 g/t Au at SGS. A similar but smaller effect is also present at ALS for assays below 1.5 g/t Au. Part of the grade variability may be the result of the nugget effect, particularly at lower grades.

In the two subsequent campaigns, the results generally show low-level artefacts due to differing DLs between the two laboratories, and the occasional outlier result, but overall, the scatter plots demonstrate strong linear correlation.

The check assays in the two most-recent QA/QC reviews have an average absolute relative difference of 0.069 and 0.158 for Au, which fall below or within the rule-of-thumb of 0.10–0.20 absolute relative difference range for acceptable laboratory duplicate samples in every campaign. The Au assay variance for duplicates is –0.05% and 2.0%, both of which fall within the rule-of-thumb of ±10% precision window.

12.2.8 Discussion – Çakmaktepe

The independent QA/QC reviews confirm that the Çakmaktepe drillhole data sampling and assaying is of a high standard and suitable for the purpose of mineral resource estimation and the reporting of exploration results. This is especially true for gold, which is the primary metal of economic interest. The confidence in the silver, copper, sulfur, and carbon analyses is at a level that at minimum supports modelling for geometallurgical and by-product metal characterisation.

12.3 Ardich Deposit Data Verification

12.3.1 Data Verification in Support of Technical Reports – Ardich

Independent data verification was conducted on the Ardich drilling databases and available QA/QC sample data for drilling completed from the first Ardich hole drilled on 1 August 2017 to the established data cut-off date for the Mineral Resource modelling of 9 December 2019.

This verification was completed in stages as drill programmes progressed, and is reported in six reports:

- Mineral Consultancy, 2018. Ardich Project Drill Data QA/QC Review. 28 February 2018.
- Yetkin, E., 2018 (2018a). Ardich Project Drill Data QA/QC Review. 29 July 2018.
- Yetkin, E., 2018 (2018b). Ardich Project Drill Data QA/QC Review. 29 October 2018.
- Yetkin, E., 2019 (2019a). Ardich Project Drill Data Validation, Verification & QA/QC Review. 8 March 2019.
- Yetkin, E., 2019 (2019b). Ardich Project Drill Data Validation, Verification & QA/QC Review. 31 October 2019.
- Yetkin, E., 2020 (2020c). Ardich Project Drill Data Validation, Verification & QA/QC Review. 30 March 2020.

It was concluded that the Ardich drillhole data sampling and assaying is of a high standard and suitable for the purpose of Mineral Resource estimation and the reporting of exploration results.

12.3.2 Collar Location – Ardich

Collar positions were verified against the pre-mine topographic surface DTM to check for inconsistencies in elevation. The threshold difference between the DTM and the drillhole collar elevation used for validation was a ±4 m difference in data up to 2020, at which time the tolerance was decreased to ±3 m.

One hole was found to have a difference outside the tolerance limits – AR214, with 7.12 m difference. All other differences were < 3 m.

As Ardich has not been mined to date, this discrepancy can be resolved by re-surveying the collar location.

12.3.3 Down-hole Surveys – Ardich

All of the Ardich drillholes AR176 thru AR233 were downhole surveyed using a multi-shot (Devico or Reflex) with readings spaced at 10 m on average (range of 4–110 m).

Six holes were found to have no down-hole survey data.

A comparison of successive down-hole survey readings for a given drillhole was undertaken using a maximum 5° variation over 30 m (0.17°/m) in either inclination or azimuth to flag records with excessive deviations. A total of 34 spurious readings were deemed to be out of acceptable limits, and a recommendation was made to remove those data from the overall database.

The recommended magnetic declination correction discussed in Section 12.1.3 has been implemented for Ardich data.

12.3.4 Geology, Density, and Geotechnical Logs – Ardich

The drillhole database lithology table was checked for alphanumeric categorical code validity and interval reporting consistency with the log key sheets. No mis-matches were identified, and all entries were found to be identical to the codes provided in log key sheets. One lithology interval was shown to have an overlapping FROM–TO and there were three intervals that were missing lithology records.

Some minor discrepancies were identified in other coding in the database, such as lower-case 'fault' codes used instead of upper-case, causing two different unique categories to be created, and some new codes created in the 'redox' and 'alteration' tables that do not appear in the log key sheet.

Density data were reviewed during three of the six Ardich verification campaigns. Density measurements are collected using the same process described in Section 12.1.4 for the Çöpler deposit. A systematic truncation from four decimal places to three decimal places was observed, and several transcription errors in FROM–TO records were identified. Manually calculated spot check values were within ~2% of the density reading supplied in the resource database. The density samples are representative in a spatial and geological context. On a total project basis, there are no obvious density outliers.

12.3.5 Assays – Ardich

There were two different independent laboratories used for assays and geochemical analyses for the entire Ardich database to date, these were:

- ALS Global
- BV (ACME)

The variety of laboratories resulted in a variety of method codes for fire assay, four-acid digestion, multi-element, and Leco analyses.

In consistency checks on the "tblVWDHAssays_ALL" assay table, four samples were found to have missing assay entries. The highest 1% of assays were checked for transcription errors, with no major errors identified.

12.3.6 Witness Samples – Ardich

No witness samples are known of for Ardich.

12.3.7 Quality Assurance / Quality Control (QA/QC) Results – Ardich

Ardich QA/QC data was independently reviewed on a campaign basis at milestone times in the evolution of the exploration programme. There are currently six individual reports describing the results. The collective results are reported in this section.

The Ardich QA/QC programme follows suggested guidelines for QC sample insertion rates:

- 3%–5% CRMs and blanks
- 5%–10% field duplicates
- 3%–5% pulp duplicates
- 5% of coarse rejects/pulps to a third-party external laboratory

12.3.7.1 Screen Analyses – Ardich

No screen analysis has been undertaken to date on Ardich material.

12.3.7.2 Certified Reference Material (CRM) Samples – Ardich

The principal assay laboratory for drill samples was ALS İzmir, with umpire samples principally submitted to the BV (ACME) laboratory.

Au CRMs were submitted across the entire Ardich database, plus S (Leco) and C (Leco) CRMs in the later programmes. The average insertion rate was of the order of 3.5%, which meets the guideline.

The performance of the CRM sample data was assessed by plotting the laboratory assay values for Au (FA and CL), Ag (4A), Cu (4A), S (4A and Leco), and C (Leco) of the CRMs against time on control charts.

A review of the CRM results from the samples submitted indicated that both ALS İzmir and BV (ACME) had acceptable overall performances for the listed Au CRMs used during the programme, although ALS consistently had issues with isolated ±2SDs as well as failed cases of ±3SDs. In general, the ALS shows high bias in almost all Au CRMs at varying levels, being more evident in low Au and cut-off Au grades, which are also responsible for the most of the +2SD and +3SD occurrences. Few of these failed cases appeared to be as a result of mis-labelling. No unexplained extreme outliers were identified. Several CRMs had insufficient data to identify any change in performance over time.

The performance of Ag, Cu, S, and C CRMs was also reviewed, showing ALS had an acceptable overall performance with isolated cases to be followed up for Ag, Cu and S.

ALS and BV (ACME) performance both for S (4A) and S (Leco) are generally acceptable other than calibration-related bias noted for low grade CRMs. C (Leco) performance of OREAS20A (ALS and BV (ACME)) and OREAS25A (ALS and BV (ACME)) returned acceptable results both for low-grade and cut-off grade.

Timely monitoring of the CRM performance will ensure that the replicate assays stay within range, that systematic analytical drift is promptly corrected, and that mis-labelled samples are promptly identified. The extreme-outlier cases need to be investigated and if these are found to be mis-labelling then the organisational procedures should be reviewed and updated. If it transpires that these are not mis-labelled samples and the errors are found to be laboratory-related, then re-assay procedures needed to confirm the assays for the relevant batches.

12.3.7.3 Blank Samples – Ardich

Blanks were inserted into the sample stream as a check for cross-contamination during sample preparation. The insertion rate was of the order of 3%, which meets the guideline.

For ALS İzmir, Au assays for blanks were assessed by charting the laboratory assay values and assessing performance versus the maximum accepted threshold value of 0.05 g/t Au, which is 10 times the lower detection limit (DL). All blank assays were below 3DL except for one sample, however it was noted that there were several occurrences where consecutive blanks assayed close to the threshold.

For Ag, all blank assays were below the maximum threshold value of 0.5 g/t Ag. For Cu, the threshold level is 10 ppm Cu and there were several samples that assayed slightly above, at, or close to the threshold value. For sulfur (both for 4A and Leco) the threshold value is 0.1% S and all blank sample assays were below this value. For carbon (Leco) the threshold value is 0.1% C and there are 92 assays above the threshold. These results show that the blank material used to monitor Au and other elements may not be suitable for C analysis, or that the samples are contaminated during the sample preparation. Other than these no obvious contamination issues are apparent within the assay database.

For BV (ACME) blanks returned all below threshold values for Au, Ag, Cu, S (4A), S (Leco), and C (Leco) analyses. Only nine blanks were submitted to BV (ACME), which does not meet the 3%–5% insertion rate guideline.

12.3.7.4 Duplicate Samples – Ardich

Duplicate sample data was analysed to determine the reproducibility of assays according to the combination of geological, sampling, and analytic variances. The insertion rate was of the order of 5%, which meets the guideline.

The duplicates in each of the six QA/QC reviews have an average absolute relative difference of between 0.029–0.140 for Au, with a sample-weighted average of approximately 0.053, which falls within or below the rule-of-thumb of 0.10–0.20 absolute relative difference range for acceptable laboratory duplicate samples for each campaign.

The Au assay variance in each campaign, given by the average percent difference, is within the range of –2.6%–3.8%, with an average of approximately –0.25%, which falls within the rule-of-thumb of ±10% precision window.

The absolute relative difference and average percent difference results were equally encouraging for S (4A) where data was obtained (from drillhole AR56 onwards).

The high precision of the duplicates reflects the inherent sample homogeneity of laboratory-prepared duplicate samples from coarse rejects, which allows more representative sampling of the grade population.

12.3.7.5 Check Assays – Ardich

All six QA/QC campaigns report the results of umpire assays with pulp duplicates submitted to BV (ACME) for independent analysis.

The rate of check assay was lower for the earlier campaigns, as low as 2% in the first campaign, but the overall average is 4.7%, which is approaching the guideline.

Generally, the results show low-level artefacts due to differing DLs between the two laboratories, and the occasional outlier result, but overall, the scatter plots demonstrate strong linear correlation.

The check assays in each of the six QA/QC reviews have an average absolute relative difference of between 0.042–0.078 for Au, with a sample-weighted average of approximately 0.064, which falls below the rule-of-thumb of 0.10–0.20 absolute relative difference range for acceptable laboratory duplicate samples in every campaign.

Two of the earlier campaigns showed questionable performance for Ag and S; a result that is considered to be moderated by the small number of samples submitted in these early campaigns.

12.3.8 Discussion – Ardich

The independent QA/QC reviews confirm that the Ardich drillhole data sampling and assaying is of a high standard and suitable for the purpose of mineral resource estimation and the reporting of exploration results. This is especially true for gold, which is the primary metal of economic interest. The confidence in the silver, copper, sulfur, and carbon analyses is at a level that at minimum supports modelling for geometallurgical and by-product metal characterisation.

12.4 Bayramdere Deposit Data Verification

The Bayramdere sampling project was part of the near-mine programme that also included the Yakuplu East and Yakuplu Southeast areas.

12.4.1 Data Verification in Support of Technical Reports – Bayramdere

Independent data verification was conducted during and immediately following the 2015 drilling programme on the project, and a data audit for Bayramdere drilling was completed in January 2016 (Cube Consulting, 2016b).

12.4.2 Drilling – Bayramdere

A total of 118 drillholes have been drilled at Bayramdere for a total length of 10,708.9 m, inclusive of metallurgical and geotechnical holes. The assay database includes 8,283 sample intervals for a total assayed length of 10,483.4 m.

The independent data verification concluded that the sample data is considered to be of an acceptable standard and appropriate for the purpose of Mineral Resource estimation and the reporting of exploration results.

12.4.3 Collar Location – Bayramdere

Raw survey pick-up verification data for 13 holes were supplied for Yakuplu East and Yakuplu South-east areas only – none for Bayramdere. The review of the 13 holes showed that the collar coordinates in the database are considered accurate for use in Mineral Resource estimation.

Bayramdere collar positions have been verified spatially against the supplied topographic surface DTMs to check for elevation errors. Only one hole was found to have varied by more than 2 m in RL when compared with the topography DTM. Hole BDD006 was identified with an erroneous elevation, however the collar details in the database appeared to have been the planned coordinates and this was considered the likely reason for the discrepancy.

12.4.4 Down-hole Surveys – Bayramdere

The Reflex multi-shot hardcopy PDF (by Devico), and associated Excel spreadsheets were cross-checked against the database. A total of 161 downhole surveys were checked. It was concluded that all conversions from magnetic to true declination were completed correctly and the down-hole database is robust.

There were two down-hole surveys for which excessive deviation existed between successive downhole azimuth readings, these are BDD031 at 45 m and BDRD003 at 80 m.

12.4.5 Geology, Density, and Geotechnical Logs – Bayramdere

Generally, the data from the original handwritten geology logs matches well to the data entry spreadsheet and the corresponding entry in the supplied databases. A number of data discrepancies were identified and included FROM and/or TO intervals not matching the original logging sheets and one case where a single entry in a database table was not recorded on the original log sheets. The geology logging discrepancies are not considered significant, with minor data transcription errors and adjustments for re-logging/ re-interpretation are to be expected as the project develops.

Where available, the hardcopy density measurement sheets (PDF) were cross-checked against the digital data entry spreadsheets and the 'DH Specific Gravity' table in the supplied databases. No data discrepancies or spurious values were identified for the Yakuplu East and Yakuplu South-east density data. A 'specific gravity' table was not provided in the Bayramdere database.

DD core and RC sample recovery and geotechnical data entry spreadsheets and some limited hardcopy logging sheets were supplied, however these could not be cross-referenced, as the corresponding data tables were not provided in the supplied databases.

12.4.6 Assays – Bayramdere

The finalised laboratory digital assay files (CSV) were imported into the supplied databases, and then cross-checked against the original assay database table using MS Access queries. In addition, the certified laboratory analysis certificates (PDF) were used to verify the finalised laboratory files (CSV). Only the key elements Au, Cu, and Ag were checked.

A total of 11,028 assay records (for Au, Cu, and Ag) were cross-checked. Assays above the upper detection limit for the analyte have been re-assayed using an ore-grade analysis and were correctly given precedence over the original assay as summarised below;

- 54 Cu AA46 assays were substituted for Cu ME-ICP61 values > 10,000 ppm,
- 71 Ag AA46 assays were substituted for Ag ME-ICP61 values > 100 g/t.

A total of 22 Au-AA25 assays were identified where the Au check assay was given precedence over the original Au-AA25 assay. The original Au assay should always be given precedence over any subsequent Au repeat or check assays to maintain consistency in the Au assay population.

In addition, the following observations were also made;

- 179 Cu assays (method CL-ICPMS) have been assigned precedence over Cu-4A_ICPES method,
- 127 Ag assays are from results of an unknown method and take precedence over the original Ag (ME-ICP61) method, and are not found in the supplied database or laboratory files,
- One Ag-AA46 check value reported as >1,500 g/t Ag, which is the upper limit for this method, and was reset in the database to a value of 1,500 g/t Ag.

Although some negligible errors have been identified, the data verification process has demonstrated the drilling data to be of a high standard and suitable for the purpose of mineral resource estimation and the reporting of exploration results.

12.4.7 Witness Samples – Bayramdere

No witness samples are known of for Bayramdere.

12.4.8 QA/QC Results – Bayramdere

Quality control sampling at Bayramdere consisted of CRMs, blank reference material samples, field duplicates, laboratory duplicates, and umpire duplicates for a combined total insertion rate of 11%.

The Bayramdere QA/QC programme follows suggested guidelines for QC sample insertion rates:

- 3%–5% CRMs and blanks
- 5%–10% field duplicates
- 3%–5% pulp duplicates
- 5% of coarse rejects/pulps to a third-party external laboratory

12.4.8.1 CRM Samples – Bayramdere

A total of 27 unique Au-specific CRMs were used during the drilling programme, these range in grade from 0.13 g/t Au to 5.61 g/t Au.

A total of 263 samples from 17 different CRMs were inserted, comprising approximately 3.2% of the total Bayramdere samples.

A bias was observed in data from ALS Global, with two of the 17 CRMs used at Bayramdere (G303-8 and G312-7) having a negative bias of greater than 5%. The scale of the negative bias decreases with increasing grade of the CRM and is most apparent in the grade range of less than 0.8 g/t Au. For G312-7, the performance at the umpire laboratory (ACME) shows no bias indicating possible instrument calibration issues at ALS Global during this period for low-grade Au analyses.

Although the replicate CRM assays do show some precision and bias errors on a CRM and laboratory basis, there is no consistent trend for a particular CRM, indicating the accuracy issue lies at the ALS Global laboratory. The internal CRMs used by the laboratory for QC purposes do not show the same bias trend.

12.4.8.2 Blank Samples – Bayramdere

The low-level Au CRMs have certified values below the detection limit for Au analysis and were treated as blanks.

A total of 855 blank samples were inserted during the entire near-mine programme into the sample stream. This comprises approximately 2.4% of the total samples submitted.

All blank assays were below the maximum accepted value of 0.1 g/t Au. No obvious contamination issues are apparent within the primary assay data.

12.4.8.3 Duplicate Samples – Bayramdere

A total of 502 duplicate samples were inserted for the Bayramdere database. Of this total 230 were field duplicates and 272 were umpire pulp duplicates. This total number of duplicates comprises approximately 6% of the total samples submitted.

Analysis of the field duplicates quantifies the total sampling error from sample collection and preparation to analysis at the laboratory.

For Bayramdere, the small amount of field duplicate data (both DD and RC) above 0.1 g/t Au was insufficient for meaningful analysis. The following conclusions are based on data from the total near-mine programme.

Results for the filtered (0.1 g/t Au) Yakuplu Southeast RC field duplicates indicate sample precision is acceptable for this style of mineralisation with an average coefficient of variation (ACV) of 28%. The acceptable levels for assessing analytical quality are an ACV of 20% for best practice, with 40% being the upper tolerance limit for acceptable precision. No significant bias in the original samples was evident. Results for the filtered (0.1 g/t Au) Yakuplu East DD field duplicates indicated sample precision was acceptable for this style of mineralisation with an ACV of 25%. A very slight negative bias in the original samples above 1.0 g/t Au was evident.

Overall, the results for the filtered Ag and Cu field duplicates (>2 g/t Ag and >1,000 ppm Cu) on the combined projects, show acceptable precision for this style of mineralisation.

For Bayramdere, the majority of the pulp duplicate data available was from DD holes; there were insufficient RC data above 0.1 g/t Au to make any reliable conclusions.

Results for the filtered (0.1 g/t Au) Bayramdere umpire duplicates indicate that the sample precision was acceptable for this style of mineralisation with an ACV of 16%. There was no bias observed between the original and duplicate samples.

12.4.8.4 Check Assays – Bayramdere

The insertion rate for check assays at Bayramdere is 4%, which falls below the guideline.

The overall precision for the pulp duplicates had an average coefficient of variation (ACV) of 22%, which is above the upper level of the acceptable limit (10%–20%) for this type of duplicate sample. The SGS laboratory shows a higher variability, with an ACV of 36%, which is double that of the ALS laboratory with an acceptable ACV of 18%. The analytical precision issue at SGS is not apparent within the CRM data, suggesting the problem may lie with sample preparation and the production of pulps with poor homogeneity. The poor repeatability is most pronounced in the grade range below 0.6 g/t Au at SGS. A similar but smaller effect is also present at ALS for assays below 1.5 g/t Au. Part of the grade variability may be the result of the nugget effect, particularly at lower grades.

12.5 CDMP20 Data Verification Discussion

The SSR Mining QA/QC programme includes CRMs, blanks, sample preparation duplicates, field duplicates, and check assays at umpire laboratories, and is considered to be acceptable according to industry standards. The following improvements have been adopted since the 2016 Technical Report:

- Drillhole collar coordinates and elevations identified as being in disagreement with field survey data (raw) or detailed topographic data, were reconciled and corrected.
- The errors in magnetic declination correction was followed up and applied on a regular basis. Review of the difference between planned and surveyed down-hole orientation and declination was completed, and calibration of survey equipment was provided regularly.
- Out-of-tolerance down-hole survey records with excessive deviations were removed. The differences identified between the raw survey files and the stored dataset was investigated and the data was updated accordingly. Raw survey files are stored for every measurement to control if needed.
- Lithological codes from all exploration projects have been reviewed and adjusted where necessary to provide consistency of logging codes throughout the database. Internal spot checks of lithological codes within the database were regularly completed to create a sustainable environment in the database.
- The outliers in density readings were followed up and either removed or corrected, as appropriate.
- The identified assay check discrepancies were followed up and corrections were applied within the database.

* The few extreme outliers that were interpreted to be mis-classified (mis-labelled) were followed up, and control mechanisms were inserted into the sampling procedure to minimise mis-classification issues.
* Internal QA/QC reporting was followed up in a regular manner, with additional spot checks completed upon assay uploading to the database. Timely corrections were made in the database like for the mis-labelled CRMs. The changing nature of the laboratory performance creates the need for close monitoring and follow-up of the reported assays for all methods of interest, including multi-element data. ALS Global was notified of identified bias issues.

QA/QC monitoring for Leco and cyanide leach analyses are partly integrated into the programme. The use of specific CRMs, blanks, and duplicates will be reconsidered if these analyses become material during the programme.

The independent reviewer recommends that:

> In addition to internal QA/QC reporting, spot checks should be done in a regular fashion and upon uploading to assays and other data to the database. This will have immediate benefits by ensuring corrections are made in the database in a timely manner, such as for mis-labelled CRMs.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Oxide Ore for Heap Leaching

13.1.1 Testwork – Çöpler Oxide

Metallurgical testwork for Çöpler oxide ore for heap leaching commenced in September 2004. The majority of this testing was carried out by Resource Development Inc. (RDi) of Wheat Ridge Colorado, with oversight from Ausenco Limited of Brisbane, Australia, and Pennstrom Consulting of Highlands Ranch, Colorado. Additional follow-up metallurgical testwork was conducted by AMMTEC Limited (AMMTEC) of Perth, Australia in 2009 and McClelland Laboratories, and supervised by Metallurgium.

The heap leaching facilities were commissioned at the Çöpler project in late-2010 and have operated continuously since that time. Operations are currently continuing.

13.1.2 Testwork – Çakmaktepe Oxide

Metallurgical testwork on Çakmaktepe oxide ore for heap leaching was undertaken at the on-site metallurgical laboratory, initially under the supervision of Kappes, Cassiday & Associates. The initial testwork in 2015 undertook bottle roll and column leach tests. The results compare to the Çöpler oxide ore, with similar behaviour and leach kinetics. Subsequently, Çakmaktepe oxide ore was heap leached together with Çöpler oxide ore.

13.1.3 Testwork – Ardich Oxide

Metallurgical testwork on Ardich oxide for heap leaching has been undertaken at McClelland laboratories and supervised by Metallurgium. An initial testwork programme including bottle roll and column leach was carried out in 2019. This initial programme identified two distinct domains with respect to gold recovery based on sulfide sulfur (SS) content of <1% and 1%–2%. The column test results indicated that the listwanite, dolomite, and jasperoid lithologies have physical properties amenable to heap leaching. The column tests were undertaken at a crush size of P_{80} of 12.5 mm. This initial test programme is being followed up in 2020 with further testwork.

13.1.3.1 Ardich Crushing Testwork

Crushing testwork on six Ardich composite samples was performed as part of the 2019 McClelland testwork programme, Crushing Work index (CWi), and Abrasion index (Ai). The CWi values ranged from 4.0–6.9 kWh/t, indicating that the material was very soft. The jasperoid was the hardest material, with a CWi of 6.9 kWh/t. The Ai values ranged from 0.12–0.90. The jasperoid was the most abrasive (0.90, Very Abrasive), whereas all other lithology types ranged from 0.12–0.26 (Abrasive to Moderately Abrasive).

13.1.4 Testwork – Bayramdere Oxide

Metallurgical testwork has been completed to characterise the Bayramdere oxide mineralisation and determine its suitability for potential heap leaching. In total, five PQ DD holes were completed in 2014 for this purpose and 91 m of half-core have been provided for intermittent bottle roll leach (IBRL) test and column leach testing.

In the IBRL tests, the gold extraction ranges from 54% to 97% at the end of 11 tests with the consumption of 0.85 kg/t NaCN. Test information is shown in Table 13.1.

Table 13.1 Summary of Bayramdere Intermittent Bottle Roll Leach Test Results

Number of IBRL Tests	11
Average Au (%)	80%
Median Au (%)	84
Minimum Au (%)	54
Maximum	97
NaCN (kg/t)	0.85

In the column test, gold extraction is 84% in the two duplicate columns. The test results are summarised in Table 13.2.

Table 13.2 Summary of Bayramdere Column Test Results

Test ID	Grind Size P_{80} (mm)	Au (g/t)			Gold Extraction (%)	Reagents (kg/t)	
		Calculated Head Grade	Recovered Grade	Weighted Residue		NaCN	Lime
1	8.0	4.08	3.41	0.67	84	0.65	0.20
2	9.1	3.91	3.27	0.65	84	0.61	0.10

Final gold extraction in column testing is approximately 84% with reasonable leach kinetics. The extracted gold quantity will be economic for heap leach processing if haul costs are not excessive.

13.1.5 Heap Leach Gold Recovery

The heap leaching process gold recovery assumptions have been updated to reflect actual performance of the operation between Q4'10 and December 2019. The gold recovery assumptions are summarised for Çöpler oxide in Table 13.3, Çakmaktepe oxide in Table 13.4 (including Bayramdere), and Ardich oxide in Table 13.5.

Table 13.3 Çöpler Gold Recovery Assumptions for Heap Leaching of Oxide

	Çöpler Zone					
Oxide Ore Type	**Manganese**	**Marble**	**Main**	**Main East**	**Main West**	**West**
Diorite	71.2	62.3	71.2	71.2	62.3	62.3
Metasediment	66.8	66.8	66.8	66.8	66.8	66.8
Limestone/Marble	78.4	75.7	68.6	78.4	75.7	75.7
Gossan	71.2	65.1	71.2	71.2	65.1	65.1
Manganese Diorite	71.2	62.3	71.2	71.2	62.3	62.3

Table 13.4 Çakmaktepe Gold Recovery Assumptions for Heap Leaching of Oxide (incl. Bayramdere)

	Çakmaktepe Zone				
Oxide Ore Type	**Central**	**North**	**East**	**Southeast**	**Bayramdere**
Limestone/Marble	70.0	59.0	67.0	–	75.0
Metasediment	80.0	14.0	–	–	–
Gossan	–	59.0	67.0	75.0	75.0
Jasperoid	73.0	59.0	–	–	–
Diorite	61.0	38.0	–	–	–
Ophiolite	70.0	63.0	67.0	75.0	75.0

Table 13.5 Ardich Gold Recovery Assumptions for Heap Leaching of Oxide

	Ardich Zone	
	Main	**East**
Sulfur <1%		
Jasperoid	50.0	50.0
Listwanite	73.0	55.0
Dolomite	73.0	55.0
Sulfur 1%–2%		
Jasperoid	40.0	40.0
Listwanite	58.0	45.0
Dolomite	58.0	45.0

The original gold recovery assumptions for Çöpler ores were developed in 2008, based on the results of column leach and bottle roll testing performed by RDi between 2005–2008. These recovery assumptions are reviewed and updated annually based on the following information:

- An analysis of the results of additional column leach and bottle roll tests performed on monthly composite samples of heap leach feed material conducted at the Çöpler project from July 2011 through December 2019.
- Use of a MS Excel-based heap leach production model that is calibrated against actual gold production data at the Çöpler mine from start-up of the operation in late-2010 through December 2019.

The recovery values listed in Table 13.3, Table 13.4, and Table 13.5 consider heap leaching of ore crushed to 80% passing 12.5 mm, agglomerated, and placed on a lined heap leach pad for treatment.

13.2 Sulfide Ores

Sulfide material (i.e. material with >2% sulfur content) is not suitable for treatment by the heap leaching process.

13.2.1 Historical Testwork – Çöpler Sulfide

Historical testing was conducted on samples from the sulfide material in several phases. RDi performed several sulfide processing scoping-level investigations from 2006–2009. A two-phase programme on sulfide samples was conducted at SGS in 2009 and 2010 to support a pre-feasibility study (PFS) completed in 2011, (Samuel, 2011). A QEMSCAN (quantitative evaluation of minerals by scanning electron microscopy) mineralogy study on three sulfide (and six oxide) samples was performed by AMMTEC in December 2008.

The historical work completed at both RDi and SGS concentrated on evaluating sulfide processing options, including direct cyanidation, flotation, cyanidation of flotation concentrates, pressure oxidation (POX) coupled with cyanidation, and roasting coupled with cyanidation. The evaluation of the historical data in the PFS resulted in the selection of POX coupled with cyanidation as the process to further evaluate with testing and a feasibility study (FS).

Initial metallurgical testwork carried out by RDi indicated that 11%–30% of the gold content in the Çöpler sulfide material may be amenable to whole-ore cyanidation, as demonstrated by diagnostic leaching. Between 60%–80% of the gold content was found to be associated with sulfide minerals and would require some type of oxidation step to liberate the gold for cyanidation.

The RDi scoping studies indicated that pre-treatment using POX was the most effective treatment and displayed the potential to achieve greater than 90% gold extractions. Flotation tests indicated that gold could be recovered by flotation, but the concentrates were low-grade with relatively high mass pulls, and relatively low gold recovery. Testwork indicated that flotation concentrate, and tailings did not leach well using cyanide, even after being finely ground.

13.2.2 Sulfide Mineralogy

In December 2008, SSR Mining commissioned AMMTEC to complete a QEMSCAN precious metals search (PMS), trace mineral search (TMS), and energy dispersive spectra signal (EDS) mineralogy analyses performed on three sulfide mineralisation samples. Analyses were performed on samples of diorite, metasediment, and massive pyrite rock types.

The findings from the 2008 QEMSCAN analyses indicated that the gangue mineralisation in the sulfide mineralisation is composed mainly of quartz, micas / clays, and feldspars, (displaying relative abundances of approximately 31%, 27%, and 21%, respectively). The sulfide mineralisation consists of pyrite, arsenopyrite, chalcopyrite, and sphalerite.

A gold deportment study was performed by AMTEL Ltd. (AMTEL) on samples of MC4 composite after flotation separation. Although flotation was not part of the flow sheet, it is a useful method of concentrating the sulfides (the main gold carriers) to improve analysis statistics. The combined concentrate represented 18.5% of the feed mass and assayed 9.8 g/t Au and 23% SS. Recoveries of gold and sulfur to concentrate were 72.7% and 90% respectively. Flotation tailings assayed 0.68 g/t Au and 0.48% SS.

The detailed mineralogical analysis is summarised in Table 13.6 and confirms that the gold is primarily carried by sulfide minerals. In the calculated head, 83% of all gold is in sulfides (free or locked) and only 2.4% was held in rock. The remainder of the gold (14%) was present as free gold, and this correlates well with a direct cyanidation recovery of only 17% when the ore was ground to a P_{80} of 90 µm.

Of the gold that is in sulfides, the majority (78%) is in sub-microscopic form. This confirms the refractory nature of the ore and explains why oxidation of the sulfides is necessary to make the gold available for leaching. Arsenopyrite was the sulfide mineral found to have the highest contained gold, averaging 123 g/t Au by one measure and 182 g/t Au by a second. Gold in pyrite was more than an order of magnitude lower than arsenopyrite and averaged 7.0 g/t Au. Marcasite, a mineral chemically similar to pyrite, carried an average of 17.8 g/t Au. Of the gold contained in sulfides, 50% was found to be in arsenopyrite, 25% in pyrite, and 20% in marcasite.

In summary, the AMTEL gold department study is consistent with previous mineralogy studies and confirms that a large portion of the gold is present as sub-microscopic particles, primarily in sulfides, largely arsenopyrite. The study also concluded that whole-ore oxidation would be required as a pre-treatment to cyanidation to liberate the majority of the gold contained in the sulfide materials.

13.2.3 Direct Cyanidation

Hazen performed direct cyanidation carbon-in-leach (CIL) tests at various grind sizes with no pre-treatment on the individual sulfide rock type composites to establish baseline gold extractions. The goal of these tests was to examine gold extraction variability with grind size. These samples were subsequently used to prepare feed composites used in the Hazen pilot plant programme.

The testwork demonstrated that the bulk of the Çöpler sulfide samples are refractory to direct cyanidation, and that extractions do not improve significantly with finer grinding.

Table 13.6 Gold Deportment in Flotation Separated Streams

Form and Carrier of Gold	Concentrate (g/t)	Tails (g/t)
Assayed Grade	10.187 ± 0.167	0.837 ± 0.028
Free / Liberated Gold Grains		
>40 µm	0.106	0.004 *
5–4 µm	0.346	0.003
<5 µm	0.871	0.146
Exposed Associated Gold Grains		
Free Sulfides +5 µm	0.350	0.018
–5 µm	–	–
Rock-Sulfide Composites	0.125	0.052
Rock Particles	0.021	0.035
Enclosed Associated Gold Grains		
Free Sulfides +5 µm	0.977	0.007
–5 µm	0.292	0.029
Rock-Sulfide Composites	0.338	0.023
Rock Particles	0.014	0.031
Sub-microscopic Gold		
Free Sulfides +5 µm	4.156	0.020
–5 µm	1.244	0.157
Associated Sulfides	1.605	0.304
Total (mineralogically counted)	10.444 (102.5%)	0.829 (99.0%)

* From a very small number of grains (1 free grain, from ~2 kg of material)

13.2.4 Flotation Testwork

13.2.4.1 Pre-2006 Testwork

A range of testwork was conducted by RDi (Wheatridge, CO), starting in 2001. This flotation testwork indicated poor flotation responses over a range of reagents tested. Approximately 50% of the arsenopyrite was found to be extremely fine grained resulting in the poor flotation response. Direct cyanide leaching testwork completed at the same time indicated that the gold mineralisation was highly refractory, with gold extractions ranging from 3% to 16%, with an average of 10%. The use of nitric acid to completely decompose the sulfide minerals resulted in excellent gold extractions. Mineralogical investigation indicated that most of the gold was associated with arsenopyrite, with lesser amounts associated with pyrite and marcasite, and the gold was sub-microscopic in nature.

The gold particle size distribution was determined to be very fine (in addition to being intimately associated with iron–arsenic sulfide minerals). This ruled out gravity concentration as an option to be considered.

The need for a refractory ore treatment process was recognised early on in the investigation of Çöpler sulfide ores.

Ultrafine grinding of two samples of rougher flotation concentrate to 99% <3 µm and 80% <4 µm, respectively, realised gold extractions of 25% and 59%, respectively. Cyanide and lime consumptions were extremely high. When combined with the poor flotation recoveries achieved, the process was viewed as unattractive.

Biological oxidation was tested at Little Bear Laboratories (Golden, CO). The results of tests on rougher concentrate samples indicated that gold extractions were directly proportional to the extent of arsenic oxidation; 85% gold extraction was achieved at 76% arsenic oxidation and 93% gold extraction at 88% arsenic oxidation. Cyanide and lime consumptions were high (8–16 kg/t and 8–19 kg/t respectively). When combined with the poor flotation recoveries achieved, the process was viewed as unattractive.

13.2.4.2 2006 Testwork

Additional flotation testwork confirmed the generally poor response to flotation, with gold recoveries ranging from 72% to 76% into 18%–30% by weight of the feed mass. The mass pull had to be increased dramatically to further increase gold recovery, i.e. 86% gold recovery into 58% of feed mass. A number of reagent schemes and a range of primary grind sizes were tested. There was little benefit observed to grinding below 80% minus 75 µm.

Direct cyanide leaching of the flotation concentrate, after grinding to 80% minus 4.6 µm, yielded 72% gold extraction. Direct cyanide leaching of the flotation tailings yielded gold extractions between 29%–34%.

13.2.4.3 2007 Testwork

This testwork was conducted on two composite samples of material that was considered to represent the majority of the orebody; metasediment and diorite. Diagnostic leaching tests indicated that both of these materials responded similarly. Direct cyanide leach gold extractions were 11%–12%, gold associated with arsenopyrite accounted for an additional 30%–34% (after roasting at 425°C), gold associated with other iron sulfide minerals accounted for a further 42%–48% (after roasting at 625°C), leaving 7%–17% of the gold remaining in the residue.

13.2.4.4 2009 Testwork

RDi conducted additional metallurgical testwork on a composite sulfide sample comprised of 38% metasediment, 37% diorite, 19% gossan, and 6% massive pyrite ore types. The results of this work confirmed the preferred approach of whole ore POX:

- Gravity concentration tests using the Knelson concentrator and Gemini table confirmed that the ore was not amenable to gravity concentration.
- Flotation tests confirmed the results of earlier work with gold recoveries in the range of 66%–71% into a rougher concentrate containing 23%–27% by weight of the feed.
- Cleaner flotation tests indicated poor upgrading of the rougher concentrate, with a final concentrate grading 12–14 g/t Au and heavy losses to the cleaner tails.
- Diagnostic leaching tests indicated that 29% of the gold was free milling, 37% associated with arsenopyrite, 19% associated with other iron sulfides, and 15% locked in the non-sulfide gangue minerals. These results were reasonably consistent with the earlier work.
- Direct cyanide leaching of the rougher flotation concentrate yielded 28% gold extraction.
- Direct cyanide leaching of the rougher flotation concentrate after fine grinding yielded 27% gold extraction.
- Two-stage roasting, and cyanide leaching of the rougher concentrate yielded 65% gold extraction.
- Single-stage oxidising roasting and cyanide leaching of the rougher concentrate yielded 58% gold extraction.
- Pressure oxidation and cyanide leaching of the rougher concentrate yielded 93% gold extraction.

SGS conducted a range of metallurgical testwork on samples of metasediment and diorite material to further evaluate flotation and POX response.

- Flotation testwork was conducted using a range of conditions and reagent schemes. Overall recoveries to the rougher concentrate varied from 60% to 80% into 16%–47% of the feed weight (i.e. 60% into 16% of the mass; 80% into 47% of the mass).
- Sequential flotation indicated that recoveries of 80%–83% could potentially be achieved into 28%–33% of the feed weight, using finer grind size (38–45 µm) and longer flotation times.
- Direct cyanide leaching yielded gold extractions of 29%–32%, consistent with earlier work.
- Direct cyanide leaching of the rougher tails gave gold extractions of 30%.
- Direct cyanide leaching of the bulk scavenger tails gave gold extractions of 10%–15%.
- Direct cyanide leaching of the slimes (rejected prior to flotation) gave gold extractions of 8%–11%.

The results of this work indicated that it would still be difficult to develop an effective flotation approach for the treatment of Çöpler sulfide ores.

13.2.4.5 2011–2012 Testwork

In late-2012, additional flotation testwork was commissioned at FLSmidth (formerly Dawson Laboratories, Inc.) to re-visit flotation and to test the most current and promising reagent schemes and conditions on representative samples of Çöpler sulfide ore. The purpose of this work was to ensure that no opportunity to upgrade the Çöpler sulfide ore using flotation was being missed. The results of this work confirmed the results of earlier work, indicating that the best performance achievable was 80% gold recovery into a rougher concentrate containing 28% by weight of the feed. Subsequent cleaner flotation indicated that 63% of the gold in feed could be recovered into a concentrate containing 16% of the feed mass with a grade of approximately 11 g/t Au. Additional tests were run to determine gold extraction by direct cyanide leaching of the concentrate, direct cyanide leaching of the tails, and ultra-fine grinding and cyanide leaching of the concentrate. None of these options resulted in an attractive approach to treatment of Çöpler sulfide ore.

13.2.4.6 2019 Testwork

Testwork was conducted in 2019 on fresh material from the existing sulfide circuit. A total of 20 tests were conducted as part of this programme.

The key variables considered in determining throughput for flotation are SS flotation recovery and flotation mass pull. Gold recovery to concentrate and gold recovery of the flotation tails are also determined. A total of eight flotation testwork tests are considered: T7–12, T15, and T16. These tests are considered due to their relative commonality of flotation conditions, and the SS feed grade is within the range that the flotation plant is expected to operate.

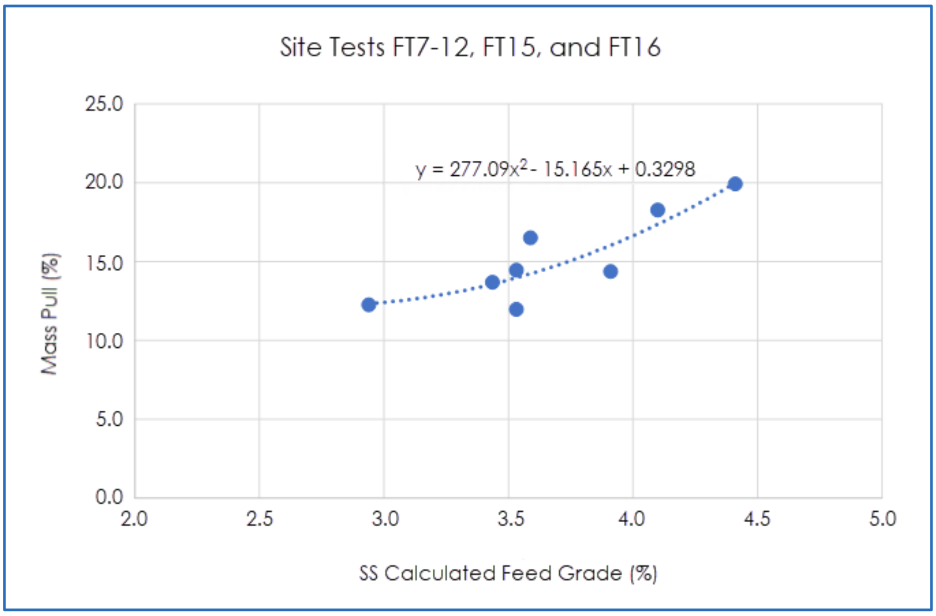
Table 13.7 summarises the key testwork results.

Table 13.7 2019 Testwork Summary

Test No.	Calculated Head Grade			Total Time (minutes)	Mass Pull (%)	Recovery Flotation			Bottle Roll – Flotation Tail			
	Au (g/t)	C (%)	SS (%)			Au (%)	C (%)	SS (%)	Calculated Head Grade (Au g/t)	Tail (Au g/t)	Gold Recovery (g/t)	Gold Recovery (%)
T7	4.60	0.74	3.91	25	14.4	50.3	2.0	79.2	2.34	1.17	1.17	50
T8	4.44	0.85	3.59	25	16.5	51.8	6.5	81.2	2.26	1.04	1.22	54
T9	4.95	0.79	2.94	25	12.3	43.0	4.9	65.1	3.16	1.68	1.48	47
T10	4.77	0.74	3.53	25	14.5	43.3	4.8	65.8	2.62	1.56	1.06	41
T11	5.21	0.75	3.53	25	12.0	45.3	4.8	69.1	2.91	1.65	1.26	43
T12	5.26	0.83	3.44	25	13.7	44.9	5.9	65.8	2.88	1.65	1.23	43
T15	4.40	1.87	4.41	30	19.9	55.8	12.1	75.5	2.78	1.32	1.45	52
T16	4.47	1.65	4.10	25	18.3	43.2	12.9	72.1	3.06	1.45	1.61	53
Average	4.76	1.03	3.68	–	15.2	47.2	6.7	71.7	–	–	–	48

The mass pull for sulfide flotation is typically related to SS grade. Figure 13.1, shows the relationship of mass pull to SS feed grade.

Figure 13.1 Feed SS% – Mass Pull Relationship



Anagold, 2020

Float Concentrate Mass Pull = 277.09 x Feed SS%^{2} – 15.165 x Feed SS% + 0.3298

13.2.5 Testwork – Comminution

The comminution properties for the three major ore domains (metasediment, diorite, and manganese diorite) have been measured during all testwork stages. Rock competence drives semi-autogenous grind (SAG) mill selection, Bond Work Index (BWi) drives ball mill selection, and Ai is used to estimate media and mill liner consumption rates.

The results are summarised in Table 13.8.

Table 13.8 Summary of Comminution Test Results

Domain	Testwork Phase	BWi (kWh/t)	Ai (g)	DWi (kwh/m³)	Axb	SPi (minutes)
Diorite	PFS	13.1	0.0630	1.70	145.0	–
	DFS1-3	10.3	0.0498	1.58	164.4	–
	DFS4	14.0	0.3142	3.71	70.6	54.6
	DFS4	12.6	0.1121	2.08	125.1	29.2
	DFS4	10.7	–	–	–	15.4
	DFS4	10.2	–	3.32	78.9	12.3
	DFS4	9.5	0.0694	2.10	122.5	19.1
	DFS4	12.3	–	3.40	80.0	20.9
	DFS4	13.4	0.2230	2.87	91.3	28.8
	DFS4	9.1	–	–	–	14.2
	DFS4	12.1	0.4353	2.69	98.6	39.1
	DFS5	14.4	0.1758	4.05	63.6	6.0
Metasediment	PFS	15.5	0.2380	6.45	41.6	–
	DFS1-3	13.1	0.1801	3.68	72.4	–
	DFS4	13.0	0.2584	3.04	84.6	76.7
	DFS4	12.6	0.2178	3.40	78.4	39.1
	DFS4	14.2	0.1891	5.12	51.2	62.5
	DFS4	15.7	0.2941	6.71	39.9	106.1
	DFS4	19.9	0.5702	6.07	45.0	161.3
	DFS4	13.0	–	6.19	43.0	80.6
	DFS4	12.0	–	–	–	59.0
	DFS4	18.1	0.4963	8.16	33.1	109.3
	DFS4	17.3	0.3313	4.46	59.6	86.6
	DFS4	12.6	0.2984	5.96	45.4	84.7
	DFS5	16.0	0.1591	4.75	54.9	26.2
Manganese Diorite	PFS	13.4	0.0330	2.69	104.0	–
	DFS4	9.1	–	–	–	3.8
	DFS4	15.4	0.2380	4.80	54.4	56.6
	DFS4	10.4	–	–	–	5.8
	DFS4	11.3	–	1.22	205.3	10.3
	DFS5	14.6	0.1490	4.01	64.8	8.5

As part of the flotation circuit sizing, Wood determined the throughput capacity of the installed crushing and grinding circuit reviewing testwork and plant actual performance.

The review of the grinding circuit determined that the throughput has exceeded design expectations since commissioning due to the processing of ore that is softer than the design comminution testwork identified. The design maximum feed rate of 306 t/h was achievable with close to full milling power being consumed. However, an average throughput rate of 370 t/h was achieved in the period late-2019 through early-2020 with the SAG and ball mills drawing approximately half of their design power.

Wood prepared a simulation model of the comminution circuit (in JKSimMet) that was calibrated to this actual plant performance. This calibrated simulation was then used to estimate plant performance with future harder ores, having properties approximating design expectations. Wood's simulation showed that the plant is expected to be able to process the target rate of 400 t/h of design-hardness ore with the mills at full design operating power.

13.2.6 Testwork – POX

Three continuous pilot plant programmes have been conducted for the POX sulfide plant; the first two programmes at Hazen Research, Inc. (Hazen) comprising a total of four test campaigns, and the third programme at SGS Lakefield Oretest, Perth (SGS Perth). Three campaigns were completed during the first pilot plant programme, with the first campaign commencing in February 2012. The second pilot programme incorporating one campaign, was conducted in December 2012. The third pilot programme, conducted in August 2015, included a single campaign that tested multiple lithologies at high and low-acidulation extents.

The pilot plant facility for the first pilot programme included the following continuous circuits: acidulation, POX autoclave, hot cure (HC), primary neutralisation (PN), six-stage counter current decantation (CCD), and mixed sulfide precipitation (MSP). Ore preparation (grinding), cyanidation, activated carbon gold recovery, cyanide destruction, tailings neutralisation, and final tailings production were all completed on a batch basis.

Campaign 1, in the first pilot plant programme, explored ranges of process operations and established preferred operating conditions. Campaigns 2 and 3 evaluated different feed combinations, and the last 30-hour run of campaign 3 (run 27) tested the preferred conditions using the ore feed blend judged by SSR Mining to be most representative of early commercial plant operation.

Campaign 4 was used to obtain key process information to populate the process design criteria document.

Several changes were made to the original pilot plant configuration for campaign 4. A hot cure circuit was added, and the sulfide precipitation circuit was converted from the stirred-tank reactor system used in campaigns 1–3 to an inline reactor with a very short retention time. A tailings neutralisation (TN) circuit was also added in order to generate samples for rheological and solid liquid separation studies. The number of CCD thickeners was reduced from six to three to reduce inventory and simplify operation.

To produce the ore feed blend for campaign 4, Hazen composited 89 samples from 280 quarter-core samples. From these 89 samples, splits were taken to provide Variability Study 2 (VS2) samples for comminution studies, samples for FLSmidth for clay mineralogy testing to include x-ray diffraction (XRD) Rietveld analysis and cation exchange capacity (CEC) swelling clay analysis, and samples for initial and detailed chemical analyses.

After selecting sample material for the VS2 testing, the remaining mass from the rejected splits was composited with samples of ore types from the January 2012 Turkish core samples in storage at Hazen to prepare the campaign 4 feed blend, called master composite 4 (MC4). The following ore types were represented in MC4:

- Metasediment (28.5%)
- Diorite (49.8%)
- Massive pyrite (3.5%)
- Manganese diorite (20%, note run 28 incorrectly used 16.7%)
- Gossan (1.5%)

In 2015, Anagold performed confirmatory pilot testing on a range of ore-types and composite blends treated at 'high' and 'low' acidulation conditions. This programme comprised a single pilot plant campaign, campaign 5, which was conducted at SGS Perth during August and September. Apart from testing the impact of acidulation chemistry, one of the key purposes of the campaign was to produce samples for repeat thickener vendor testing. This was prompted by the inconsistent vendor data generated during campaigns 1–4.

13.2.7 Overall Circuit Performance

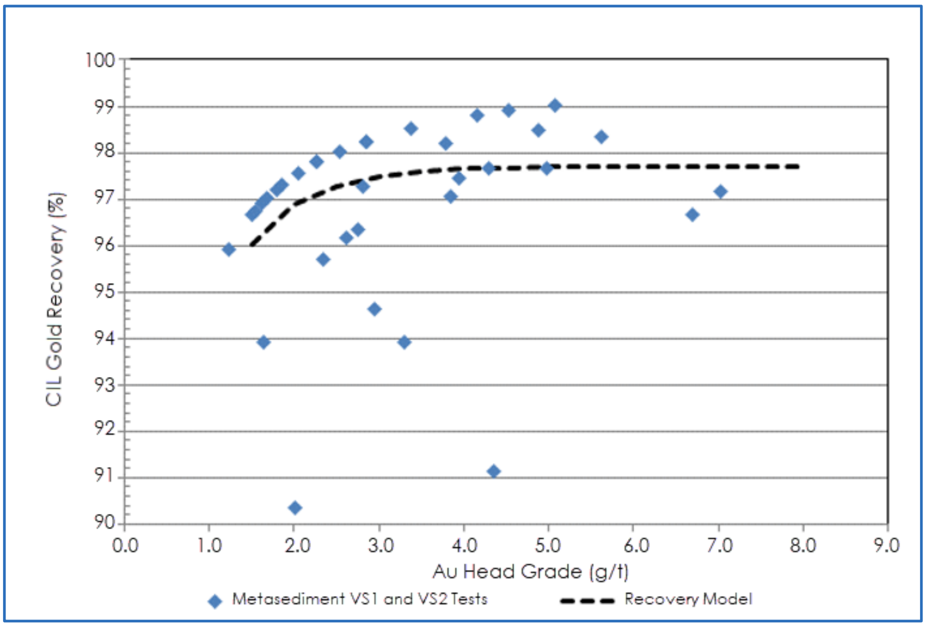
The recovery of gold across a laboratory carbon-in-pulp (CIP) circuit was measured for a number of variability samples representing each of the three major ore types. The set of results useful for predictive recovery work was arrived at by excluding results where the test conditions were not representative of the design operating conditions. Results were excluded where: the head grade was less than 1.5 g/t Au, the target oxidation level was not attained, or where the free acid at the end of the test was less than 20 g/L. Out of 158 tests, only 77 tests were conducted under conditions representing design.

In addition to the testwork, the commercial sulfide POX plant commenced commissioning in December 2018, with actual results reviewed to February 2020 to validate the recovery.

13.2.7.1 POX Gold Recovery

The gold recovery results of the acceptable tests are plotted in Figure 13.2, Figure 13.3, and Figure 13.4, together with an appropriate recovery model curve in each instance.

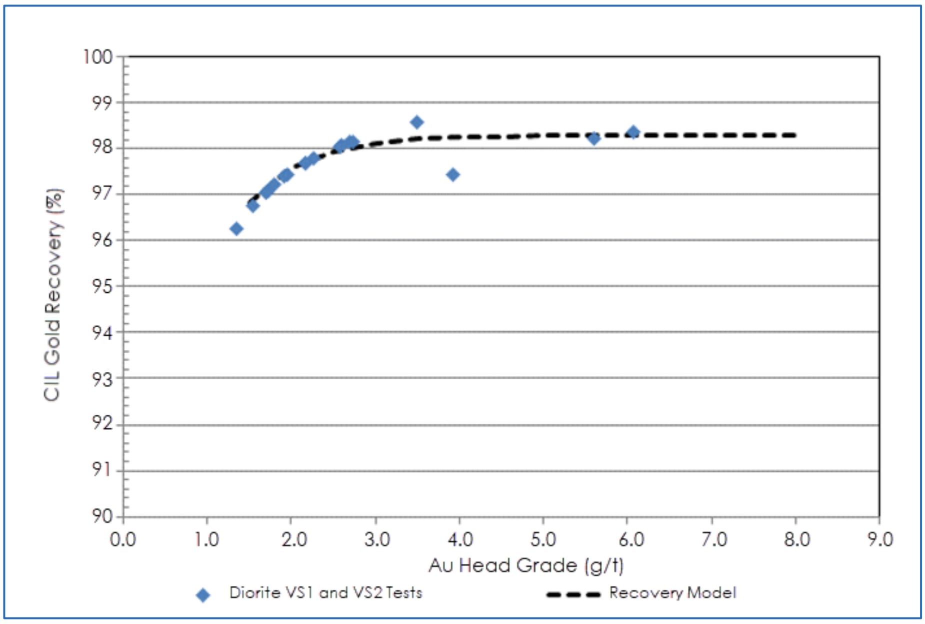
Figure 13.2 Metasediment Gold Recovery Results and Model



Anagold, 2016

The results are plotted in terms of feed grade so that predictions of recovery during operations can be made by knowing the feed grade.

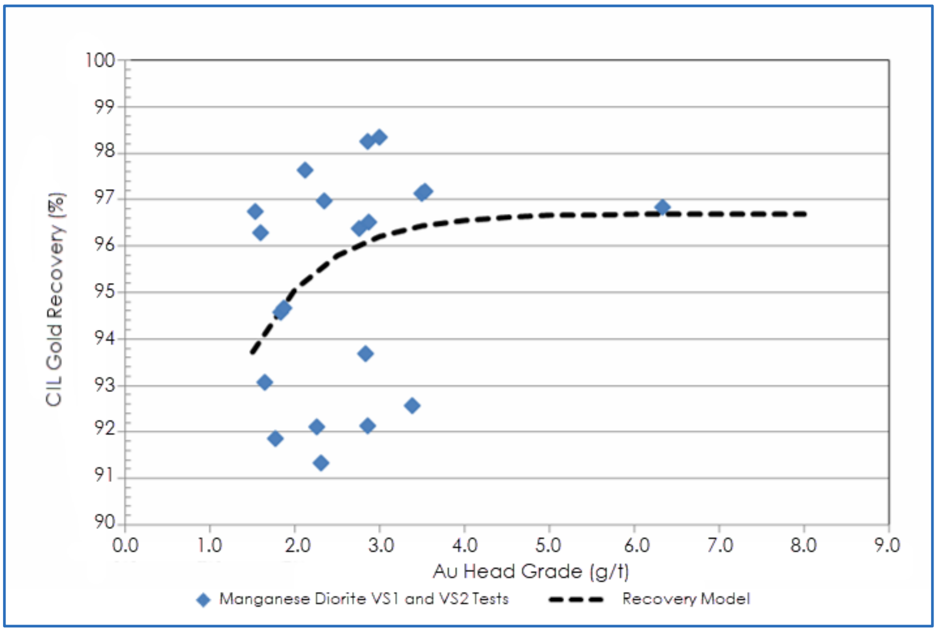
Figure 13.3 Diorite Gold Recovery and Model



Anagold, 2016

Note that Figure 13.2 and Figure 13.3 show a number of results that tend to form a regular curve at the top of the datasets. In each instance, where the results are on this curve the solid tails Au grade was below the limit of detection and an assigned tails grade, equal to half the limit of detection, was set for calculation purposes.

Figure 13.4 Manganese Diorite Gold Recovery and Model



Anagold, 2016

The recovery model is represented by the equation:

Gold Recovery (%) = a x (1 – exp (–b x (Au head grade in g/t – c))) + d

Parameter ‘a’ is the only one of the four that has a direct process meaning, representing the maximum recovery the equation can generate. The parameter ‘d’ represents circuit losses in a commercial operation.

The parameters used to generate the curves in Figure 13.2, Figure 13.3, and Figure 13.4 are shown in Table 13.9, and include an allowance for operational losses of 1%.

Table 13.9 Gold POX Recovery Model Parameters

Material Type	a	b	c	d
Metasediment	97.7	1.4	–1.4	–1.0
Diorite	98.3	1.4	–1.5	–1.0
Manganese Diorite	96.7	1.2	–1.4	–1.0

The POX commissioning and ramp-up allowances in Table 13.10 have been made on top of the base recoveries.

Table 13.10 Commissioning and Ramp-up Allowances

Recovery Corrections	Gold Recovery Deduction (%)
Commissioning to June 2019	–3.30
Ramp-up July 2019–June 2020	–2.30
Flotation Commissioning	–0.75

13.2.7.2 POX Silver Recovery

The silver recovery pattern is much less clear than gold because silver is not released by the oxidation process. Silver recovery is determined from actual plant recovery over the period January 2019–February 2020.

The silver recovery calculates to 3.0%.

13.2.7.3 Flotation Gold Recovery

From the testwork, it is estimated that the flotation concentrate reporting to the POX circuit will achieve the same overall recovery as the ore directly reporting to POX. Gold recovery to the flotation concentrate is estimated to be 55%.

The flotation tails reporting directly to the leach circuit is estimated to have a gold recovery of 43%, based on testwork.

An allowance of 0.75% reduced gold recovery during commissioning and ramp-up of the flotation circuit (Year-1 of flotation operation) has been included.

13.2.8 Variability

The POX metallurgical variability test programme (batch testing) was conducted on samples representing each of the main Çöpler ore types (metasediment, diorite, and manganese diorite) and representing the full grade spectrum, in terms of Au, Ag, Cu, SS, and carbonate for each type. The flow sheet development testing, at both batch and pilot scale, were conducted on composites representing early plant operation. The sulfur levels, which are critical for POX process operation, were similar to the actual SS levels. Predicted performance from the testwork during development compares well with the overall actual performance.

13.3 Mineral Processing and Metallurgical Discussion

A large amount of POX testwork has been performed on Çöpler sulfide ore across a number of pilot plant campaigns. The processes used have been shown to be robust, as demonstrated through operational performance during commissioning and ramp-up to February 2020.

The addition of a flotation circuit to the sulfide plant is estimated to provide stability and flexibility to the POX circuit operation to maximise throughput and oxygen utilisation by maintaining optimum sulfur grade to the autoclaves.

Given the limited flotation testwork undertaken, ongoing flotation testwork is recommended to further optimise flotation performance and gold recovery to concentrate.

Ongoing testwork and analysis is also recommended on POX oxidation and leach recovery to improve and optimise circuit performance.

Further metallurgical testing of Ardich material types, both oxide and sulfide, is recommended to optimise the feeds to the heap leach and POX and flotation circuits, respectively.

14 MINERAL RESOURCE ESTIMATES

Mineral Resources for the project have been estimated using industry best practices (CIM, 2019), and conform to the requirements of CIM Definition Standards (CIM, 2014).

14.1 Çöpler Deposit

At Çöpler, a resource model was constructed to define the geometry of the gold mineralisation. Grades were estimated using exploration drilling data and then calibrated against the production grade control data. Steps for the gold modelling process included:

- Creation of wireframes that constrain gold mineralisation.

  This step incorporated structural trends to guide the shape of the wireframes along known geological features within the deposit. Mineralised trends commonly followed lithological contacts, such as the diorite / marble contact, and structural features identified by surface mapping. A total of 15 trends across the deposit were used to produce a 3D solid of the gold mineralisation (the gold mineralisation shell). Trends were developed using the geological model, pit mapping and blast hole data.

- Gold mineralisation was then estimated using a method termed probability assigned constrained kriging (PACK) and then trimmed using the gold mineralisation shell.

  PACK first uses a probabilistic model or envelope (indicator envelope) to define the limits of the potentially economic mineralisation. The model cells and drillhole composites within these indicator envelopes were then used for grade estimations. The PACK process was designed to prevent economic grades inside the indicator envelope from being smeared into the waste and restricts low-grade material outside the indicator envelopes from diluting the mineralised material inside the envelope.

- The parameters used to construct the indicator envelopes were calibrated such that the estimated tonnes and grades approximated the historical production data. These calibrations were performed by area, material type, and time period so the calibrations could be studied and evaluated in detail.

Au, Ag, and Cu were interpolated into the parent cells using ordinary kriging (OK), while As, Mn, Fe, and Zn were interpolated using inverse distance method, weighted to the power of two (ID2).

14.1.1 Çöpler Mineral Resource Estimate – Key Assumptions

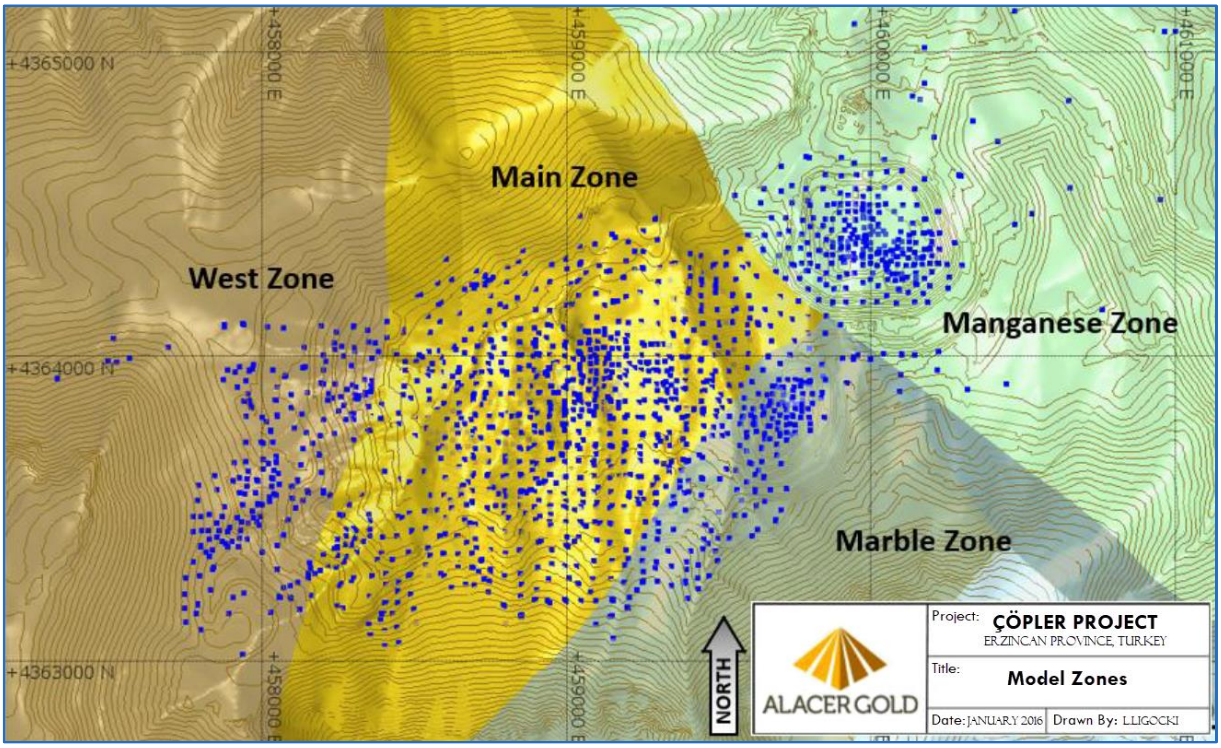
The estimation methods at Çöpler were designed to address the variable nature of the epithermal, structural, and disseminated styles of gold mineralisation, while honouring the bi-modal distribution of the sulfur mineralisation and the oxide / sulfide boundary used to define the material types for mine planning. The modelling methods were designed so the:

- Mineral Resources could be updated with additional drilling.
- Changes in cut-off grades could be re-calibrated using up-to-date production data.

Although Ag and Cu were estimated and used in the mining studies, the model design focused on the gold mineralisation as it is the dominant economic contributor at Çöpler.

No obvious correlations were observed between Au and total sulfur; they were therefore domained and estimated separately. Since Au also showed little correlation with lithology, it was domained simply by model zone (Manganese, Main, Marble, and West), which reflects the different trends of the mineralisation that commonly follow structures and lithological contacts in the zones (see Figure 14.1).

Figure 14.1 Çöpler Model Zones



SSR Mining, 2016

The percentage of total sulfur is the main criterion used to delineate between 'oxide' and 'sulfide' material types:

- Oxide material (S <2%) is processed using a heap leach method and has a cut-off grade of approximately 0.3 g/t Au.
- Sulfide material (S ≥ 2%) is processed in the sulfide plant and has a cut-off grade of approximately 1.5 g/t Au.

Total sulfur assay data exhibits a bi-modal distribution with a distinct inflection point at 2% S, and also shows a good correlation with logged lithology. The 2% S inflection point also agrees well with a 1% pyrite break point in the drillhole logs.

As a result, sulfur was modelled using oxide and sulfide sub-domains within each lithology, and gold PACK models were constructed separately for oxide and sulfide within each lithology using the respective Au cut-offs.

The gold models were then reconciled to historical production data and the resource modelling parameters were adjusted to best match the historical data. Mineral Resource categories were applied to each model cell based on a combination of parameters including drillhole density and data quality.

14.1.2 Çöpler Base Indicator Model

In order to constrain the cell model into a manageable file size, model cells were only generated within the gold mineralisation shell. The upper surface of the gold mineralisation shell honours the original (pre-mining) topography projected upwards by approximately 30 m, and extends beyond drilling by approximately 300 m. This allows cells at the corners of the orthogonal model to be excluded, thereby reducing the model size by approximately 40% without impacting the area of interest.

A parent cell size of 10 m x 10 m x 5 m was selected, with the 10 m easting and northing dimensions representing approximately one half the average drillhole spacing, and the 5 m height of the cells representing the mining bench height. Cell model prototype parameters are provided in Table 14.1. The Mineral Resource model has an implicit selective mining unit (SMU) size of 5 m x 10 m x 5 m. The cell model is not rotated.

Table 14.1 Çöpler Block Model Parameters

Direction	Minimum (m)	Maximum (m)	Range (m)	Cell Size (m)	No. of Cells
East	457,100	461,100	4,000	10	400
North	4,362,500	4,365,100	2,600	10	260
RL	400	1,750	1,350	5	270

Drillhole intervals were composited to 10 m down-hole lengths and then assigned Au indicator values based on their composited Au grade. The sulfur indicator values were assigned to 5 m composites. Composites below the threshold were assigned '0' and composites at or above the threshold were assigned '1'.

Gold and sulfur indicator values were then interpolated into the parent cell model using ID2 and the parameters shown in Table 14.2. The interpolated indicators represent a distance-weighted average of the composite indicators that occur within the search neighbourhood and therefore have values anywhere in the range 0–1. The interpolated indicator was used to create an envelope encapsulating the mineralisation above 0.3 g/t Au (the indicator envelope). An example cross-section of the cells within the indicator envelope juxtaposed with drilling with Au grades above and below the 0.3 g/t Au cut-off is shown in Figure 14.2 (note: the PACK threshold has not been applied to the model cells in Figure 14.2.)

Exploratory data analyses (EDA) and capping studies were performed on samples within the indicator envelope.

Table 14.2 Çöpler Gold and Sulfur Indicator Estimation Parameters

	Indicators	Search Pass	Samples			Search Distance (m)		
			Min.	Max.	Max./Hole	East	North	RL
Au	1 & 2	1	3	20	2	40	40	30
		2	3	20	2	60	60	40
		3	2	20	2	150	150	75
S	1	1	3	12	2	40	40	40
		2	3	12	2	60	60	60
		3	2	12	2	160	160	160

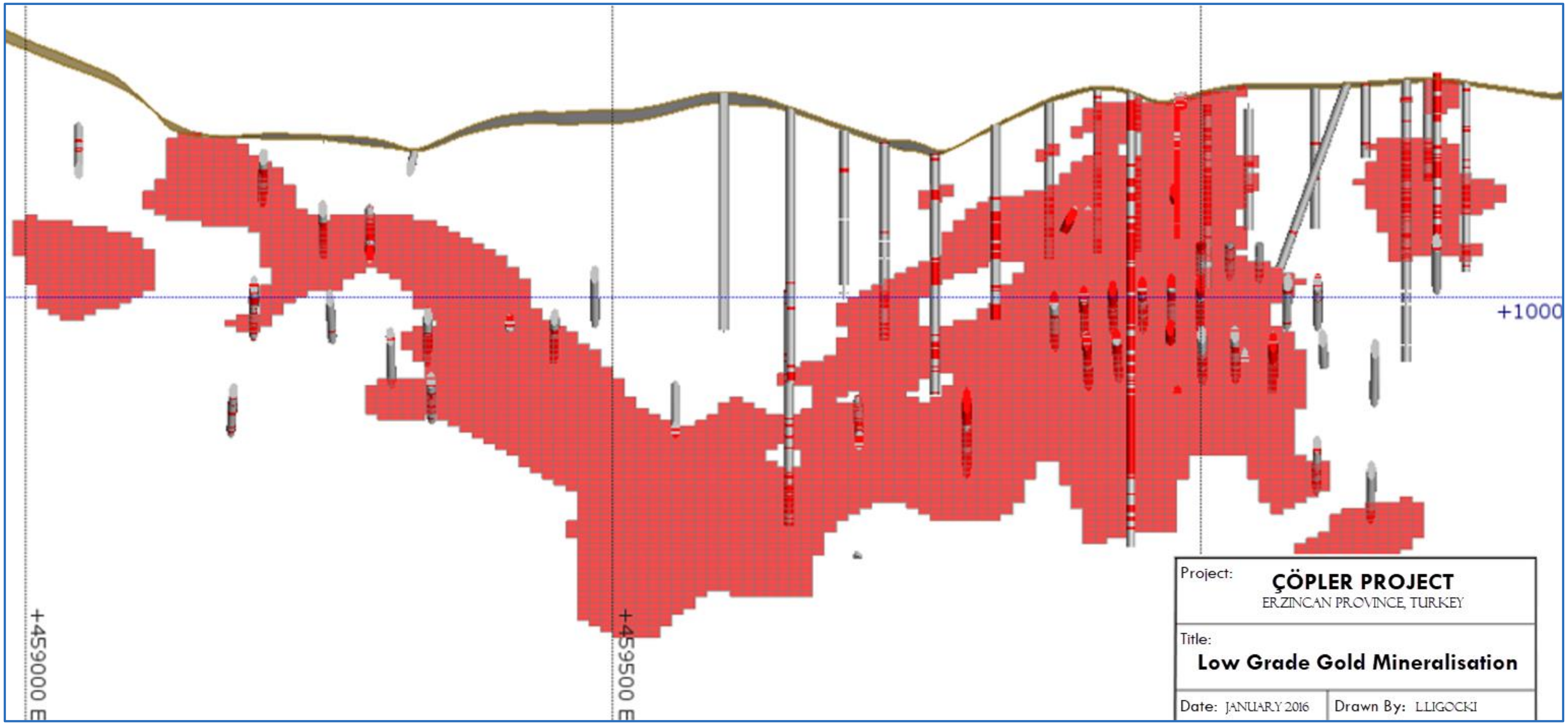
14.1.1 Çöpler Domains

The model cells within the indicator envelope were assigned into four zones that represent the four geologically distinct zones (Manganese, Main, Marble, and West) using wireframe solids.

The position of the boundary between the Manganese Zone and the Main Zone was selected between discrete diorite intrusive events. The boundaries for the Marble Zone were selected along one limb of a diorite intrusion associated with a region of higher grade gold mineralisation. The boundary direction then follows the north-easterly trend of the mineralisation. The extension of this boundary includes a larger discrete diorite intrusion that carries minor gold mineralisation along its contact with the metasediment.

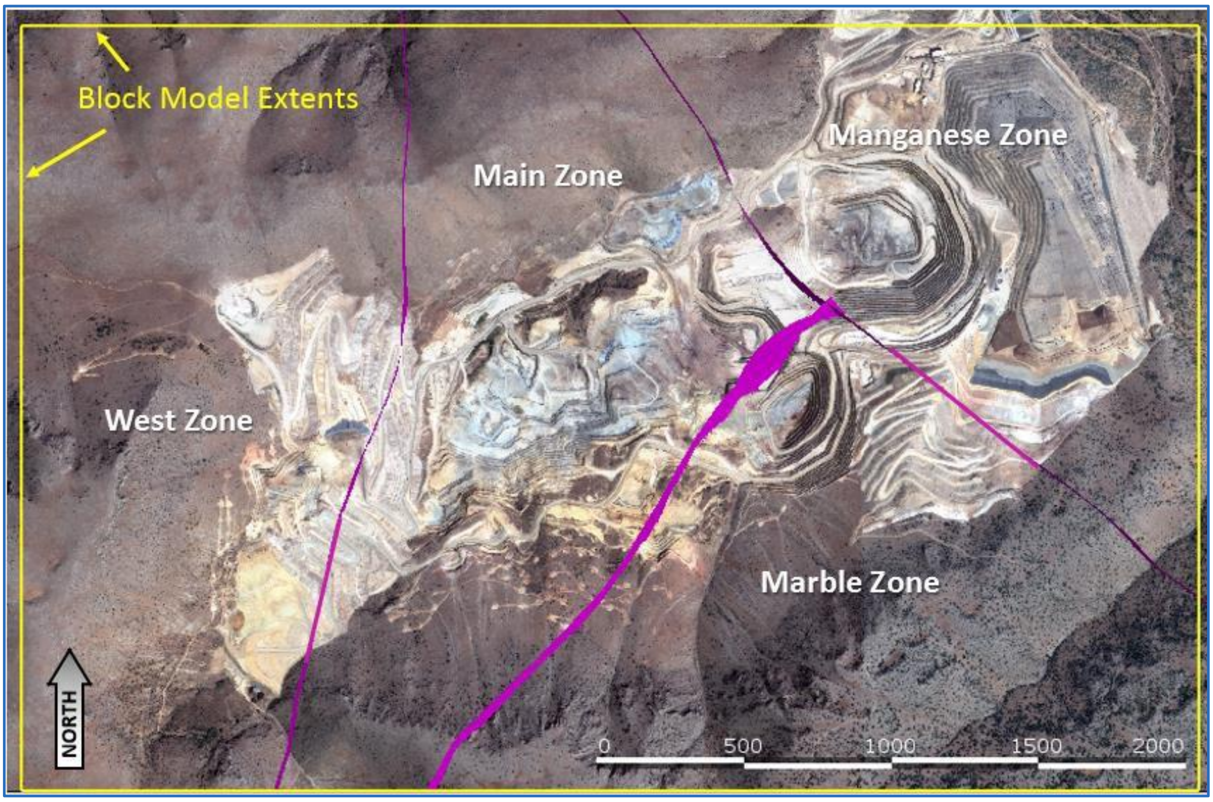
The tops of the model zone boundaries wireframe solids were trimmed to the original (pre-mining) topography. The extents of the model zone boundaries exclude exploration drillholes to the far north and east of the pit area. Figure 14.3 shows the spatial relationships between the four model zones used in the resource modellings relative to the pits as at 2015.

Figure 14.2 Çöpler Low-Grade Gold Indicator Envelope – Cross-Section 4,364,400 mN, (looking north)



SSR Mining, 2016
Red cells show the extents of the Indicator Envelope
Drillholes are coloured by Au grade: red intervals show Au ≥0.3 g/t

Figure 14.3 Çöpler Resource Model Zones and Model Prototype Extents



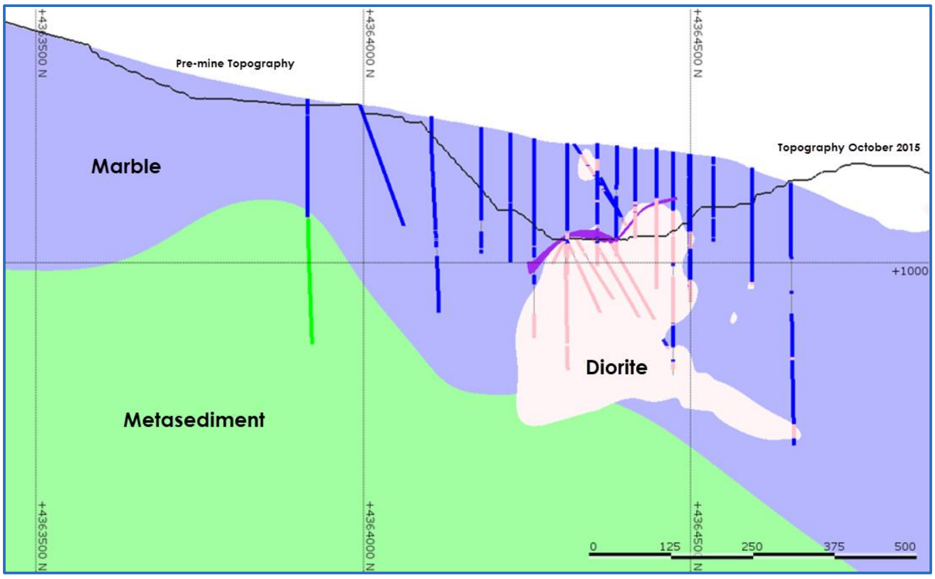
SSR Mining, 2016

14.1.2 Çöpler Geological Model

Exploration drillhole data and surface mapping were used to create 3D solid interpretation wireframes for the four main geological units: marble, diorite, metasediment, and manganese diorite. Surface mapping was used to provide indicative contact locations in areas of sparse drilling. In areas where the two datasets did not match, priority was given to the drillhole data. Blast hole data were not used to generate the lithology interpretations but were referenced to provide guidance in zones of wide-spaced drilling and in areas with missing drillhole data. The interpretation was adjusted in the Manganese Zone after referencing the blast hole data.

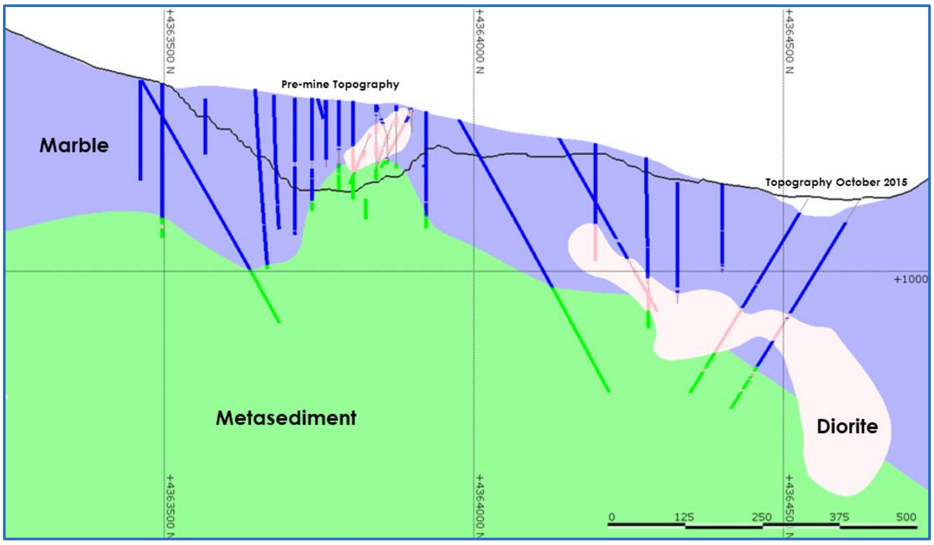
Construction of the lithology wireframes was made within a defined boundary, sufficient in size to cover areas of interest for resource modelling. Some typical cross-sections illustrating the lithology interpretations are shown Figure 14.4 and Figure 14.5.

Figure 14.4 Çöpler Lithology Model – Manganese Domain, Cross-Section 459,900 mE, (looking West)



SSR Mining, 2016
Manganese rich zone, shown in purple

Figure 14.5 Çöpler Lithology – Marble Zone, Cross-Section 459,700 mE, (looking west)



SSR Mining, 2016

14.1.3 Çöpler Data Summary

The cut-off date for the export of the drillholes from the database to be used in the resource modelling was 15 July 2015. The extract contained 1,957 drillholes with a total of 297,798.2 m of drilling. Of this, a total of 1,880 drillholes have collar coordinates within the extents used to construct the resource model. In general, the drillhole spacing ranged from 5–60 m, averaging approximately 20 m. Most drillholes are either vertical or inclined at 60°. Approximately 2% of the drillholes had missing assays; these were set to a null value and not used in the statistics or mineral resource estimation.

14.1.4 Çöpler Exploratory Data Analysis

14.1.4.1 Çöpler Summary Statistics

A mixture of sample lengths was submitted to the laboratory for assay analysis for both RC and DD holes. In areas perceived to be waste, some 1 m RC sample intervals were combined into a 2 m sample length. For initial statistical studies, the drillhole dataset was composited to 1 m intervals to provide equal support. For grade estimation, the samples were composited into 5 m down-hole composite intervals. Table 14.3 and Table 14.4 summarise the key statistics for samples located within the interpreted mineralisation envelope. One metre composites for Au had an initial EDA top cut threshold of 40 g/t Au applied globally to limit skewing the overall mean. The 40 g/t Au cap applied to only 89 of >243,000 composites.

Table 14.3 Çöpler Drillhole Au Statistics by Lithology (based on top cut 1 m composites)

	Au Statistics						
Lithology	Count	Min.	Max. *	Mean	Std. Dev.	Variance	CV
All Data							
Diorite	73,458	0.005	40.0	0.70	2.11	4.46	3.00
Metasediment	97,085	0.005	40.0	0.59	1.56	2.43	2.62
Marble	71,995	0.005	40.0	0.41	1.75	3.06	4.22
Manganese Diorite	1,219	0.010	40.0	4.04	5.73	32.84	1.42
Oxide Data (S <2%)							
Diorite	30,312	0.005	40.0	0.53	2.29	5.26	4.32
Metasediment	31,838	0.005	40.0	0.36	1.45	2.11	4.04
Marble	70,208	0.005	40.0	0.39	1.66	2.75	4.28
Manganese Diorite	853	0.010	40.0	4.52	5.65	31.93	1.25
Sulfide Data (S ≥2%)							
Diorite	43,146	0.005	40.0	0.83	1.97	3.86	2.38
Metasediment	65,247	0.005	40.0	0.71	1.59	2.54	2.25
Marble	1,787	0.005	40.0	1.50	3.76	14.11	2.50
Manganese Diorite	366	0.020	40.0	2.92	5.77	33.28	1.97

* 40 g/t Au is the EDA top cut threshold

Table 14.4 Çöpler Key Statistics (based on top cut 1 m Composites)

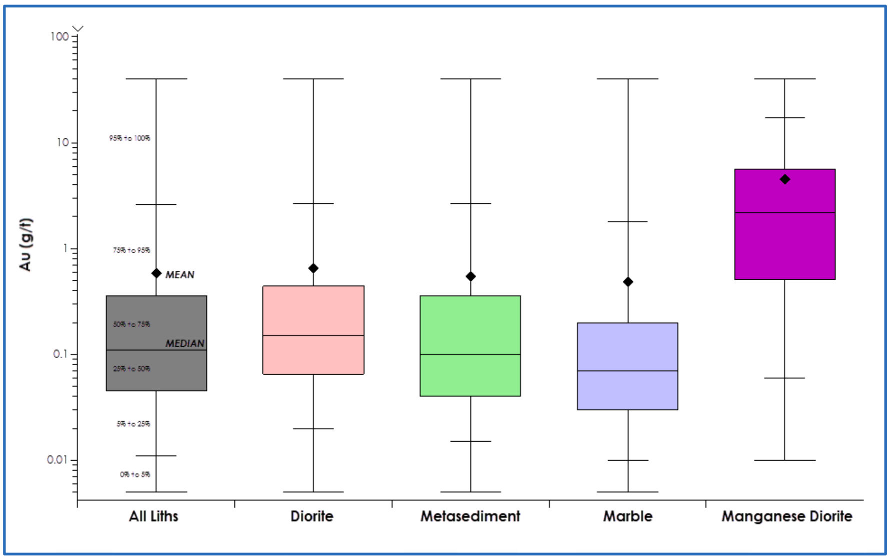
Grade	Unit	Count	Min.	Max.	Mean	Std. Dev.	Variance	CV
Au	g/t	245,124	0.005	40.0 *	0.59	1.86	3.46	3.14
Ag	g/t	245,122	0.25	1,500	2.1	13.6	185	6.47
Cu	%	239,969	0.001	22.80	0.08	0.23	0.05	2.81
S	%	245,122	0.005	50.00	2.24	2.65	7.05	1.18
As	ppm	244,888	2.5	81,644	586	1,494	2.2×10^6	2.55
Mn	ppm	245,055	2.5	630,000	2,023	7,787	60.6×10^6	3.85

* 40 g/t Au is the EDA top cut threshold

14.1.4.2 Çöpler Box Plots

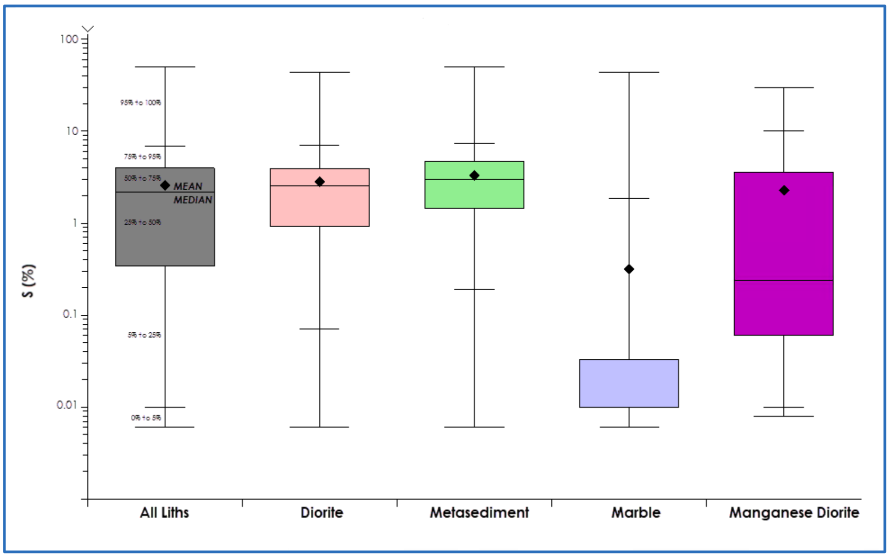
Box plots were categorised by lithology. Examples of box plots for AuFA and S are shown in Figure 14.6 and Figure 14.7.

Figure 14.6 Çöpler Log Box Plot of 1 m AuFA Composites by Lithology



Modified from SSR Mining, 2016

Figure 14.7 Çöpler Log Box Plot of 1 m S% Composites by Lithology



Modified from SSR Mining, 2016

14.1.4.3 Çöpler Correlation Coefficients

Correlation coefficients and scatterplots of the elements with the higher correlations were constructed. Correlation coefficients are summarised in Table 14.5.

Table 14.5 Çöpler Correlation Coefficients using 5 m Composites

	AuFA (g/t)	Ag (g/t)	Cu (%)	As (ppm)	Fe (%)	Mn (ppm)	Zn (ppm)	S (%)
AuFA (g/t)	**1.00**							
Ag (g/t)	0.27	**1.00**						
Cu (%)	0.09	0.05	**1.00**					
As (ppm)	0.41	0.17	0.03	**1.00**				
Fe (%)	0.14	0.02	0.42	0.21	**1.00**			
Mn (ppm)	0.16	0.31	0.03	0.17	–0.02	**1.00**		
Zn (ppm)	0.07	0.16	0.17	0.03	0.26	0.06	**1.00**	
S (%)	0.05	0.07	0.18	0.23	0.26	0.02	0.14	**1.00**

14.1.4.4 Çöpler Statistical Review

Detailed statistical analyses were undertaken to assist with the understanding of the mineralisation distribution in the various domains. The statistical review included typical univariate statistics (tabulations, histograms, box plots) and bivariate statistics (scatter plots, correlations).

A summary of key findings follows:

- A histogram of sulfur grade in the 1 m composites shows a bi-modal distribution, with the lowest mode at or near trace S (27% of the dataset), and the second mode at approximately 3.25% S (7% of the dataset).
- Mean Au grade statistics are similar for diorite, metasediment, and marble but higher in the manganese diorite. When reviewing the data spatially, however, the higher grade Au mineralisation commonly occurs along the lithological contacts.
- Mean Ag grades are similar for diorite and metasediment, but lower in marble and higher in manganese diorite.
- Mean Cu grades varied between lithologies, but in general are higher in the diorite and metasediment.
- Mean Au grades in diorite, metasediment, and the marble are higher within the sulfide material. Manganese diorite carries a higher mean Au grade within the oxide material relative to the sulfide material.
- Distinctively different sulfur populations were observed for each lithology (although each lithology hosts both low and high-sulfur mineralisation) suggesting that sulfur should be domained by lithology for estimation. This approach was taken on the current model.
- The diorite, metasediment, and manganese diorite showed similar As grades, but the marble As was lower.

- There is moderate correlation between:
  - Au and As
  - Cu and Fe
- Minor correlations occur between:
  - Au and Ag
  - Ag and As
  - Ag and Mn
- While correlation probably exists between gold and sulfur on a mineralogical level, as suggested by the correlation between gold and arsenic, and the observed presence of arsenopyrite (FeAsS), this correlation is probably masked by the much larger episode of non-auriferous sulfide mineralisation. This suggests that it is reasonable to model silver, copper, zinc, arsenic, and manganese using the gold statistical model.

14.1.5 Çöpler Core Recovery

Basic statistics (categorised by oxide (S <2%) and sulfide (S ≥2%)), histograms, quantile-quantile (QQ) plots, and box plots binned by core recovery were performed with the following results:

- No correlation was identified between any of the elemental grades and core recovery.
- There is no obvious increase or decrease in Au grade with lower core recovery.

In 2014, two nearest-neighbour (NN) models were constructed to quantify the influence of the drillhole assays with low DD core recoveries. The first NN model was constructed using only composites with core recoveries >60% (96% of composites), and the second NN model was constructed using all composites that were used in the resource model. All estimation parameters were kept the same for both estimations. The average grades of the NN models were then compared, and the difference was found to be less than 0.1% relative.

14.1.6 Çöpler Twin Holes

Twenty-three twin hole comparisons were made between various combinations of DD holes and RC holes for Au, S and Cu. An additional 10 twin hole comparisons were made for gold between PQ core holes and either DD holes or RC holes. To aid the interpretation, the water table was plotted and the correlations between the twin hole grades were ranked and summarised with the following results:

- The average RC Au grade is slightly higher than the average DD hole grade.
- No significant changes in grades were noted for the RC holes above or below the water table.
- For sulfur, little difference in grade was noted between DD holes and RC holes.
- For Cu, little difference was noted between the DD holes and RC holes, but the grades were very low.

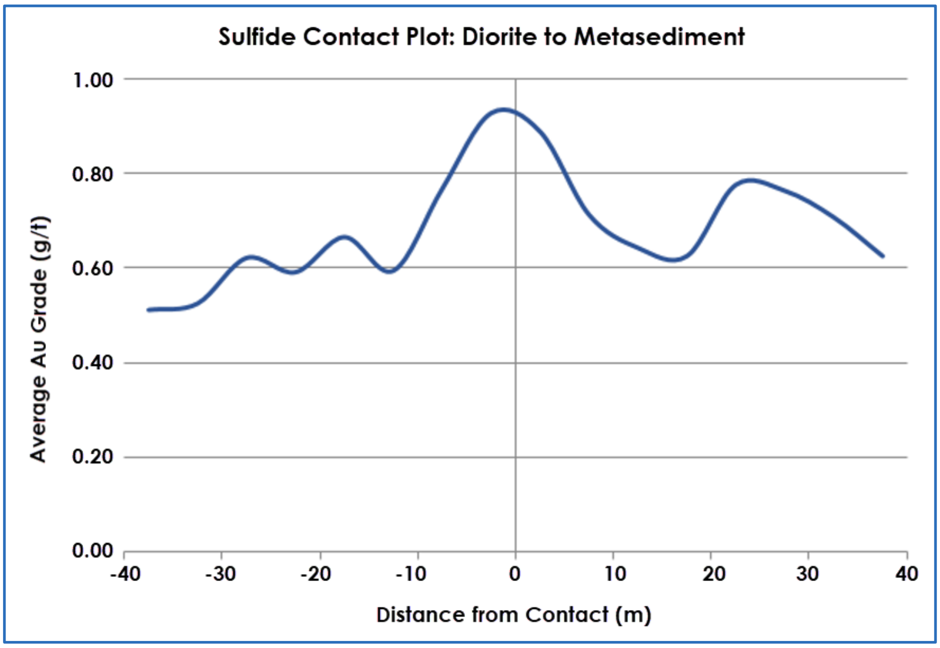
- The PQ holes showed approximately 6% higher grade, but the dataset is limited.

In conclusion, the twin hole comparisons was considered to provide good agreement between the drilling types.

14.1.7 Çöpler Contact Plots

Contact plots were constructed for the different combinations of lithological contacts and categorised by material located within the oxide or sulfide portion of the deposit. The oxide and sulfide boundary used for the plots was defined by an interpreted oxidation surface based on visual logging. An example of a contact plot of Au across the diorite / metasediment contact for sulfide material is shown in Figure 14.8.

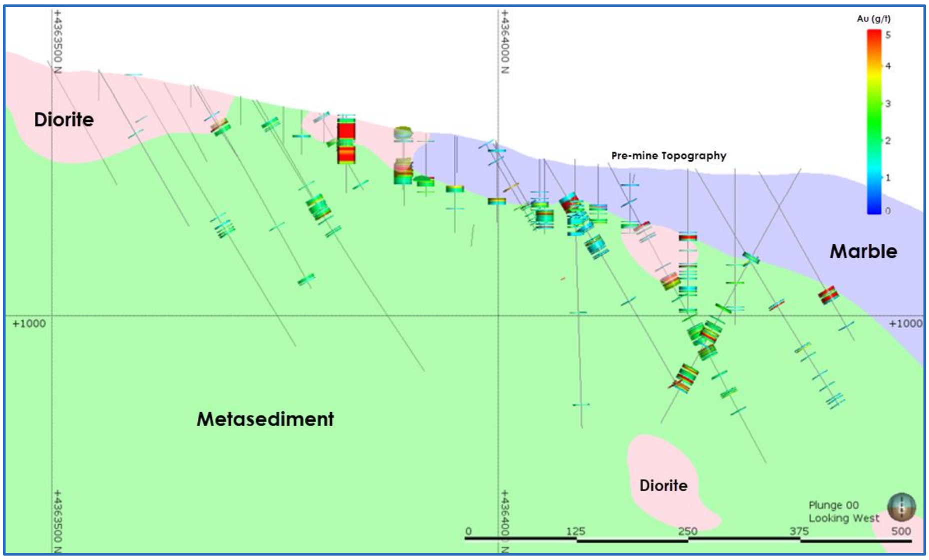
Figure 14.8 Çöpler Contact Plot of Au across the Diorite / Metasediment Contact for Sulfide Material



SSR Mining, 2016

In general, no hard contacts were observed for Au. As shown in Figure 14.9, the higher grade Au mineralisation commonly occurs along the lithological contacts, indicating that the gold mineralisation should not be modelled separately for each of the lithological domains.

Figure 14.9 Çöpler Drillhole Au Assays above 1.0 g/t at Lithology Contacts, Cross-Section 459,400 mE, (looking west)



SSR Mining, 2016

14.1.8 Çöpler Top Cutting

In mineral deposits with skewed distributions, it is not uncommon for a small number of the highest assays to account for a significant and disproportionate quantity of the total metal content in the model estimates. Although these assays are real and reproducible, they commonly show little continuity, and can add a significant amount of uncertainty to a mineral resource estimate.

One method of constraining the influence of these samples is to apply a top cut to the assays before compositing and grade estimation.

To determine appropriate top cuts, statistical studies were performed for each of the domains categorised by S <2% and S ≥2%. The top cut studies performed were:

- Checking for discontinuities (kinks) in cumulative log probability plots.
- Decile analysis, (Parrish, 1997).
- Quantifying the number of high-grade samples lying in close proximity to each other (DIST).

Results for each of these methods were compared and a top cut threshold was selected. Top cutting was performed on the 1 m composites prior to compositing into the 5 m composites used for the grade estimations. Au was studied and capped by domain and low / high-sulfur category. Top cut thresholds for Ag, Cu, S, As, Fe, Mn, and Zn were applied globally.

The top cut thresholds applied before compositing are summarised in Table 14.6 and Table 14.7.

Table 14.6 Çöpler Top Cuts for Au

Domain	Top Cut Au (g/t)
Oxide (S <2%)	
Manganese Zone	18.0
Main Zone	16.0
Marble Zone	30.0
West Zone	16.0
Sulfide (S ≥2%)	
Manganese Zone	18.0
Main Zone	14.0
Marble Zone	25.0
West Zone	14.0

Table 14.7 Çöpler Top Cuts for Non-Au Elements, Applied Globally

Element	Unit	Top Cut
Ag	g/t	300
Cu	%	5.0
S	%	20.0
C	%	13.0
As	ppm	30,000
Fe	%	50
Mn	ppm	100,000
Zn	ppm	60,000

14.1.9 Çöpler Drillhole Compositing

Samples used for grade estimation were prepared by first compositing the raw sample lengths to 1 m down-hole intervals. Au composites were capped globally at 40 g/t Au for the EDA. The 1 m composites were subsequently top cut at the relevant threshold according to the statistics of each model zone and oxide / sulfide domain. These 1 m composites were then composited into 5 m down-hole for additional statistical analysis and grade estimation.

The 5 m composite interval for grade estimation was selected as it was considered to notionally match the mining bench height. The 5 m composites were not truncated at lithological contacts, nor domain boundaries. Statistics of the 5 m gold composites used for grade estimation are summarised in Table 14.8.

Table 14.8 Çöpler Drillhole Au Statistics (based on 5 m composites)

Domain	Count	Min.	Max.	Mean	Std. Dev.	CV
Model Zone						
Manganese Zone	9,597	0.005	18.00	0.74	1.60	2.16
Main Zone	31,200	0.005	15.58	0.54	1.09	2.01
Marble Zone	4,598	0.005	30.00	0.76	2.60	3.40
West Zone	4,212	0.005	12.69	0.24	0.69	2.88
Lithological Domain – Oxide (S <2%)						
Diorite	6,115	0.005	17.04	0.58	1.48	2.54
Metasediment	13,941	0.005	15.58	0.31	0.80	2.61
Marble	3,696	0.005	30.00	0.76	2.70	3.56
Manganese Diorite	2,534	0.005	12.69	0.26	0.83	3.16
Lithological Domain – Sulfide (S ≥2%)						
Diorite	3,482	0.007	18.00	1.02	1.75	1.72
Metasediment	17,259	0.005	15.32	0.73	1.25	1.70
Marble	902	0.005	22.03	0.79	2.16	2.74
Manganese Diorite	1,678	0.005	5.08	0.20	0.36	1.80

14.1.10 Çöpler Variography

The EDA showed that the trends of the Au mineralisation followed lithological contacts and structures that vary by domain. As a result, variograms (correlograms) were calculated for Au, Ag, and Cu composites for each domain categorised by oxide (S <2%) and sulfide (S ≥2%).

The directions of the anisotropy axes were determined by creating multi-directional variograms, variogram models, and visual observation of the tabular shaped trends of the mineralisation. After the anisotropy had been determined, three directional variograms were calculated and modelled in each of the three primary directions of anisotropy. Given the low and high-sulfur domain variograms showed similar structures, albeit with the low-sulfur domain variogram structures better defined, the low-sulfur domain variograms were used for the grade estimation. Variogram parameters are summarised in Table 14.9.

Table 14.9 Çöpler Variogram Parameters used in Grade Estimation

Model Zone	Element	Azimuth	Incline	Axis	Nugget	Variogram Structure			Range		
						c1	c2	c3	a1	a2	a3
Manganese Zone	Ag	302	0	X	0.17	0.27	0.54	0.02	51	82	250
		212	52	Y		0.27	0.54	0.02	19	82	235
		32	38	Z		0.27	0.54	0.02	36	73	112
	Au	302	0	X	0.12	0.55	0.32	0.01	43	88	222
		212	52	Y		0.55	0.32	0.01	20	63	235
		32	38	Z		0.55	0.32	0.01	30	40	11
	Cu	302	0	X	0.21	0.26	0.36	0.17	22	58	234
		212	52	Y		0.26	0.36	0.17	12	74	100
		32	38	Z		0.26	0.36	0.17	48	52	95
Main Zone	Ag	147	0	X	0.37	0.24	0.26	0.13	22	68	196
		57	10	Y		0.24	0.26	0.13	22	63	192
		237	80	Z		0.24	0.26	0.13	9	3	124
	Au	147	0	X	0.17	0.47	0.26	0.10	0	53	92
		57	10	Y		0.47	0.26	0.10	12	74	192
		237	80	Z		0.47	0.26	0.10	12	48	196
	Cu	147	0	X	0.17	0.41	0.25	0.17	18	93	300
		57	10	Y		0.41	0.25	0.17	26	65	182
		237	80	Z		0.41	0.25	0.17	10	33	300
Marble Zone	Ag	210	0	X	0.19	0.30	0.51	0.00	48	83	242
		120	50	Y		0.30	0.51	0.00	72	90	192
		300	40	Z		0.30	0.51	0.00	33	80	200
	Au	210	0	X	0.06	0.82	0.08	0.04	52	77	106
		120	50	Y		0.82	0.08	0.04	21	53	121
		300	40	Z		0.82	0.08	0.04	21	55	200
	Cu	210	0	X	0.26	0.47	0.27	0.00	27	114	186
		120	50	Y		0.47	0.27	0.00	49	99	121
		300	40	Z		0.47	0.27	0.00	31	83	250
West Zone	Ag	50	0	X	0.24	0.03	0.25	0.48	17	50	106
		320	65	Y		0.03	0.25	0.48	20	108	140
		140	25	Z		0.03	0.25	0.48	9	53	105
	Au	50	0	X	0.20	0.11	0.68	0.01	17	42	91
		320	65	Y		0.11	0.68	0.01	24	40	115
		140	25	Z		0.11	0.68	0.01	32	48	105
	Cu	50	0	X	0.08	0.54	0.35	0.03	46	300	400
		320	65	Y		0.54	0.35	0.03	42	194	300
		140	25	Z		0.54	0.35	0.03	200	500	500

14.1.11 Çöpler Sulfur Model

The total sulfur model was designed to emulate the hard 2% S threshold used during ore control to delineate material to be processed on the heap leach pad or sent to the POX plant.

EDA showed that sulfur should be modelled separately in each of the four main lithological units (diorite, metasediment, marble, and manganese diorite). The sulfur estimate proved to be very sensitive. Minor changes in the estimation parameters causes the reclassification of material from high to low-sulfur and vice versa. The change in the sulfur categorisation has an impact on what cut-off grade is used and what mining and processing cost is applied.

To match the proportion of material greater than and less than 2% sulfur in each lithological domain, a sulfur indicator was generated using a discriminator of 2% sulfur. To accomplish this, a sulfur indicator field was created in the drillhole data, and populated as follows:

- S Indicator = 0 where S < 2%
- S Indicator = 1 where S ≥ 2%

The S indicator was then interpolated into the cell model using NN and ID2 methods. The ID2 interpolated indicators represent a distance-weighted average of the composite indicators and therefore have values anywhere in the range 0–1. In contrast, the NN interpolated indicators represent only the closest composite indicator and therefore can only have the value ‘0’ or ‘1’.

The number of cells above and below 2% sulfur was initially defined using the NN result (Indicator 0 = S <2% and Indicator 1 = S ≥2%). The ID2 indicator estimate was calibrated against the NN model to make the proportion of low and high-sulfur material honour the NN proportions. Sulfur indicator ID2 estimate thresholds that honoured the results of the NN estimation for low-sulfur / high-sulfur proportions were:

- Diorite = 0.50
- Metasediments = 0.51
- Marble = 0.26
- Manganese diorite = 0.36

Table 14.10 Çöpler Sulfur Estimation Parameters

Model Variable	Method	Pass	Samples		Max. Samples Per Hole	Search Distance (m)		
			Minimum	Maximum		X	Y	Z
S	ID2	1	3	12	2	40	40	40
S	ID2	2	3	12	2	60	60	60
S	ID2	3	2	12	2	160	160	160

Sample indicator estimate limits by lithology and material type are summarised in Table 14.11, and a typical cross-section is shown in Figure 14.10.

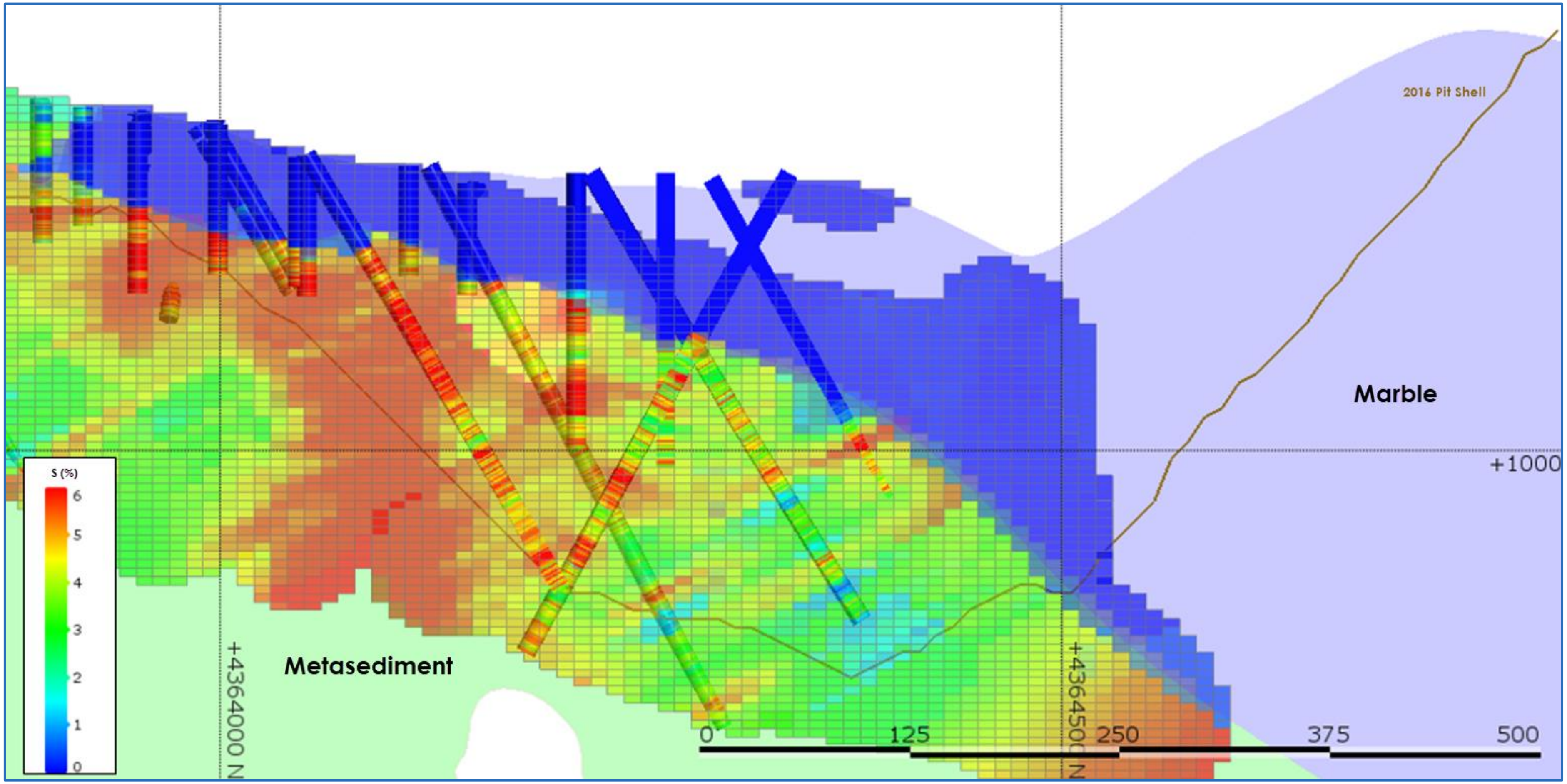
Table 14.11 Çöpler Sulfur Indicator Estimate Limits

	Lithology	Sulfur Indicator Samples		Sulfur Indicator Model Cells	
		Minimum	Maximum	Minimum	Maximum
Oxide (S <2%)	Diorite	0.00	0.58	0.00	0.50
	Metasediment	0.00	0.59	0.00	0.51
	Marble	0.00	0.30	0.00	0.26
	Manganese Diorite	0.00	0.40	0.00	0.36
Sulfide (S ≥2%)	Diorite	0.40	1.00	0.50	1.00
	Metasediment	0.40	1.00	0.51	1.00
	Marble	0.21	1.00	0.26	1.00
	Manganese Diorite	0.29	1.00	0.36	1.00

A soft boundary approach was achieved at lithological contacts by slightly raising the maximum indicator estimate for the oxide estimate and lowering the minimum indicator estimate for the sulfide estimate.

The sulfur model was not constrained by the mineralisation envelope. This means sulfur was also estimated into the waste rock cells; this was for the purpose of waste rock characterisation.

Figure 14.10 Çöpler Cross-Section of Sulfur Model, 459,400 mE, (looking west)



SSR Mining, 2016
The distinct breaks in sulfur grades approximate lithological contacts (note gradational grades at boundary due to soft boundary approach)

14.1.12 Çöpler Gold and Other Metal Models

A total of nine elements, Au, Ag, Cu, S, C, Zn, Fe, As, and Mn were estimated. Au, Cu, and Ag were estimated using ordinary kriging (OK) and the remaining elements were interpolated using the ID2 method. Zn, Fe, As, and Mn, which are only used for material-type classification, were restricted to within the mineralisation envelope.

Estimation parameters used in the PACK model are shown in Table 14.12. All cells were estimated using a discretisation matrix of 3 x 3 x 1.

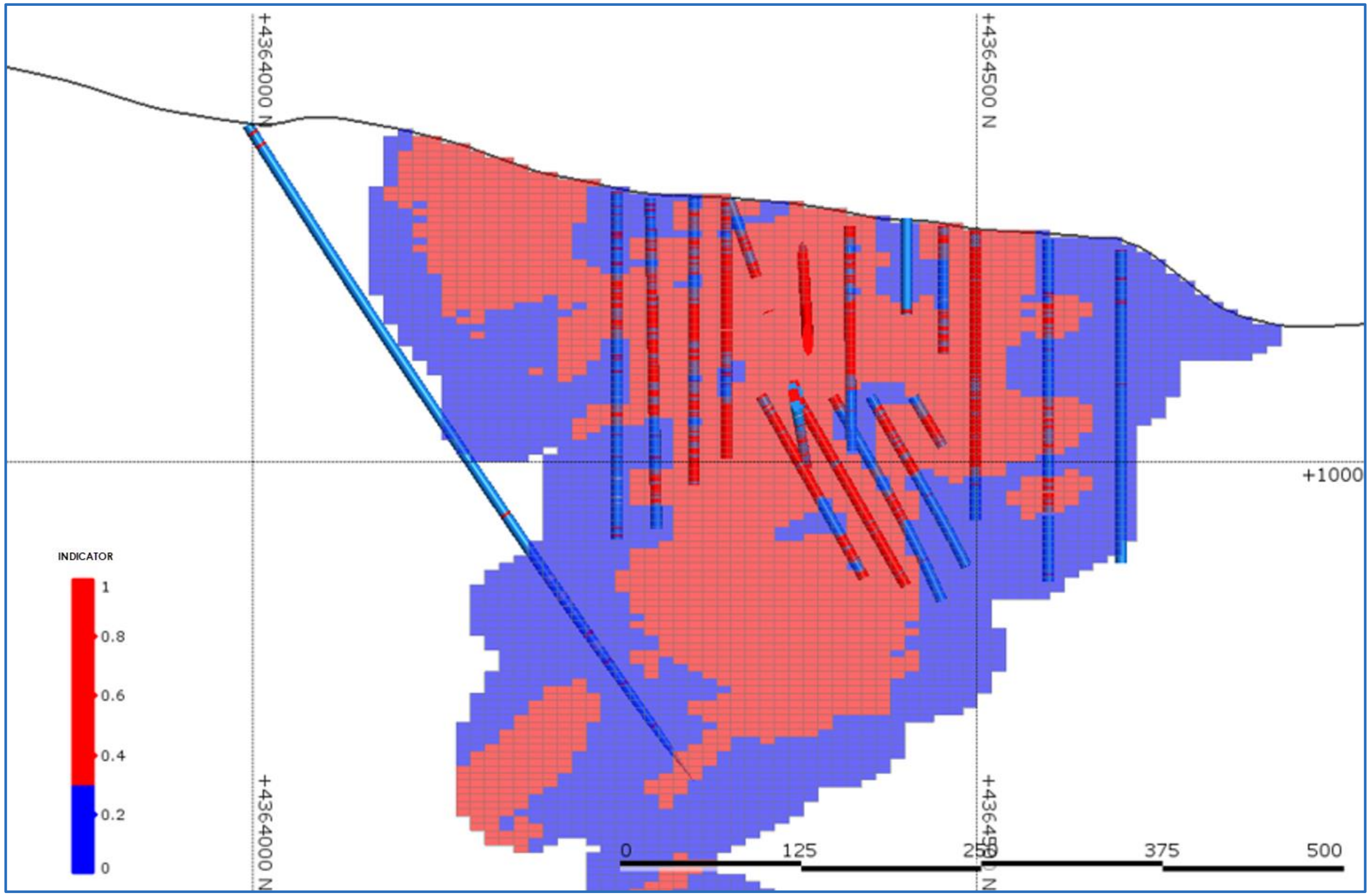
The volume of the mineralisation envelope was calibrated to past production by:

1. Creating a production cell model:
    - Constructing a 3 m x 3 m x 5 m cell model in the areas that had already been mined.
    - Populating the 3 m x 3 m x 5 m cells with the ore control tonnes and grades estimated from blast hole assays.
    - Tabulating ore control tonnes and grade from January 2014–October 2015.
2. Building an indicator model and estimation of gold grade:
    - The low-grade estimates were achieved using an indicator approach defined by an 0.3 g/t Au discriminator. First a low-grade Au indicator field was established in the drillhole 5 m composite file: if the composite grade was <0.3 g/t Au, the low-grade indicator field was set to zero (IND1=0); if the composite grade was ≥0.3 g/t Au, the low-grade indicator was set to one (IND1=1). The low-grade indicator was then interpolated into all cells using ID2, and those cells with an estimated low-grade indicator of greater than 0.3 (i.e. IND1 > 0.3) were selected to define the indicator envelope. Only composites within the indicator envelope were used to estimate the Au grade, Figure 14.11.
    - Similarly, a high-grade gold estimate was developed using a high-sulfur indicator model with a discriminator of 1.5 g/t Au to reflect the higher cut-off required for processing the material through the POX plant. The high-grade gold estimate uses the same indicator estimate threshold of 0.3 (i.e. IND2 > 0.3) to define the boundary limits.
    - The low-grade gold estimates were applied to those cells with estimated sulfur grades <2%, and the high-grade gold estimates were applied to those cells with estimated sulfur grades ≥2% S. Figure 14.12 shows the cells with S ≥2% within the indicator envelope. Figure 14.13 shows the estimated Au grades after combining the low-sulfur and high-sulfur indicator gold estimates.
3. Calibrating the PACK model:
    - The PACK model parameters were then adjusted so that the gold ounces in the PACK model approximates the gold ounces reported from the ore control model; this is explained further in Section 14.1.16. The calibrations were categorised by domain and by oxide / sulfur material types, Table 14.13.
    - After the gold ounces were calibrated by zone and material type, cells with estimated Au grades below the selected indicator threshold were set to waste.

Table 14.12 Çöpler Summary of the Estimation Parameters Used in the PACK Estimations

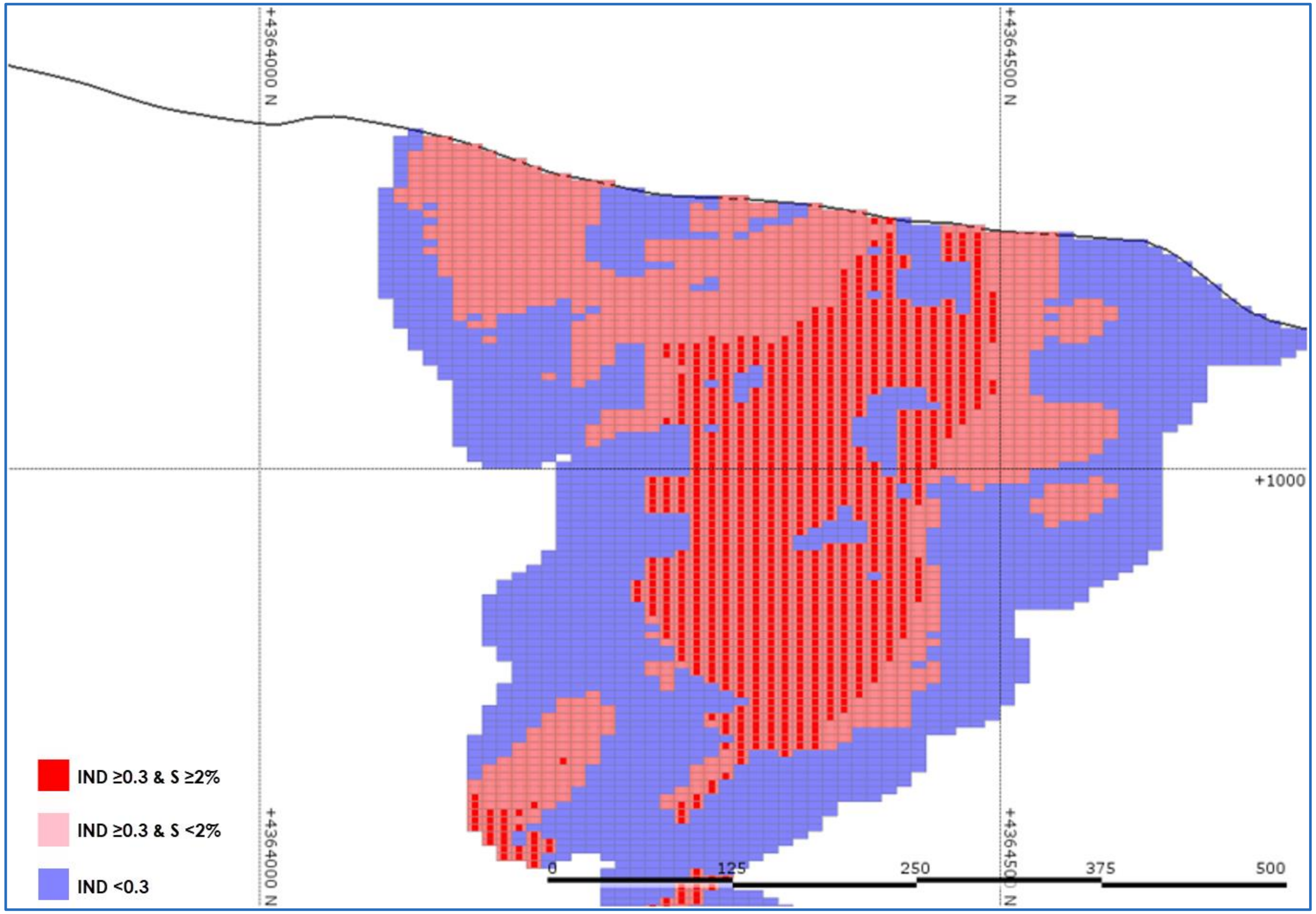
Zone	Element	Azimuth	Incline	Search Pass 1			Search Pass 2			Search Pass 3		
				Distance	Minimum	Maximum	Distance	Minimum	Maximum	Distance	Minimum	Maximum
All Zones	Sulfur	90	0	40	3	12	60	3	12	160	2	12
		0	0	40	3	12	60	3	12	160	2	12
		0	–90	40	3	12	60	3	12	160	2	12
Manganese Zone	Low-grade Au, Ag, Cu	302	0	40	3	12	60	3	12	150	2	12
		212	–52	40	3	12	60	3	12	150	2	12
		32	–38	30	3	12	40	3	12	80	2	12
	High-grade Au	302	0	40	3	12	60	3	12	150	2	12
		212	–52	40	3	12	60	3	12	150	2	12
		32	–38	30	3	12	40	3	12	80	2	12
Main Zone	Low-grade Au, Ag, Cu	147	0	40	3	12	60	3	12	150	2	12
		57	–10	40	3	12	60	3	12	150	2	12
		237	–80	20	3	12	30	3	12	75	2	12
	High-grade Au	147	0	40	3	12	60	3	12	150	2	12
		57	–10	40	3	12	60	3	12	150	2	12
		237	–80	20	3	12	30	3	12	75	2	12
Marble Zone	Low-grade Au, Ag, Cu	210	0	40	3	12	60	3	12	150	2	12
		120	–50	40	3	12	60	3	12	150	2	12
		300	–40	20	3	12	30	3	12	75	2	12
	High-grade Au	210	0	40	3	12	60	3	12	150	2	12
		120	–50	40	3	12	60	3	12	150	2	12
		300	–40	20	3	12	30	3	12	75	2	12
West Zone	Low-grade Au, Ag, Cu	50	0	40	3	12	60	3	12	150	2	12
		320	–65	40	3	12	60	3	12	150	2	12
		140	–25	20	3	12	30	3	12	75	2	12
	High-grade Au	50	0	40	3	12	60	3	12	150	2	12
		320	–65	40	3	12	60	3	12	150	2	12
		140	–25	20	3	12	30	3	12	75	2	12

Figure 14.11 Çöpler Geometry of the Gold Indicator PACK Model within Gold 3D Solid; Cross-Section 460,000 mE, (looking west)



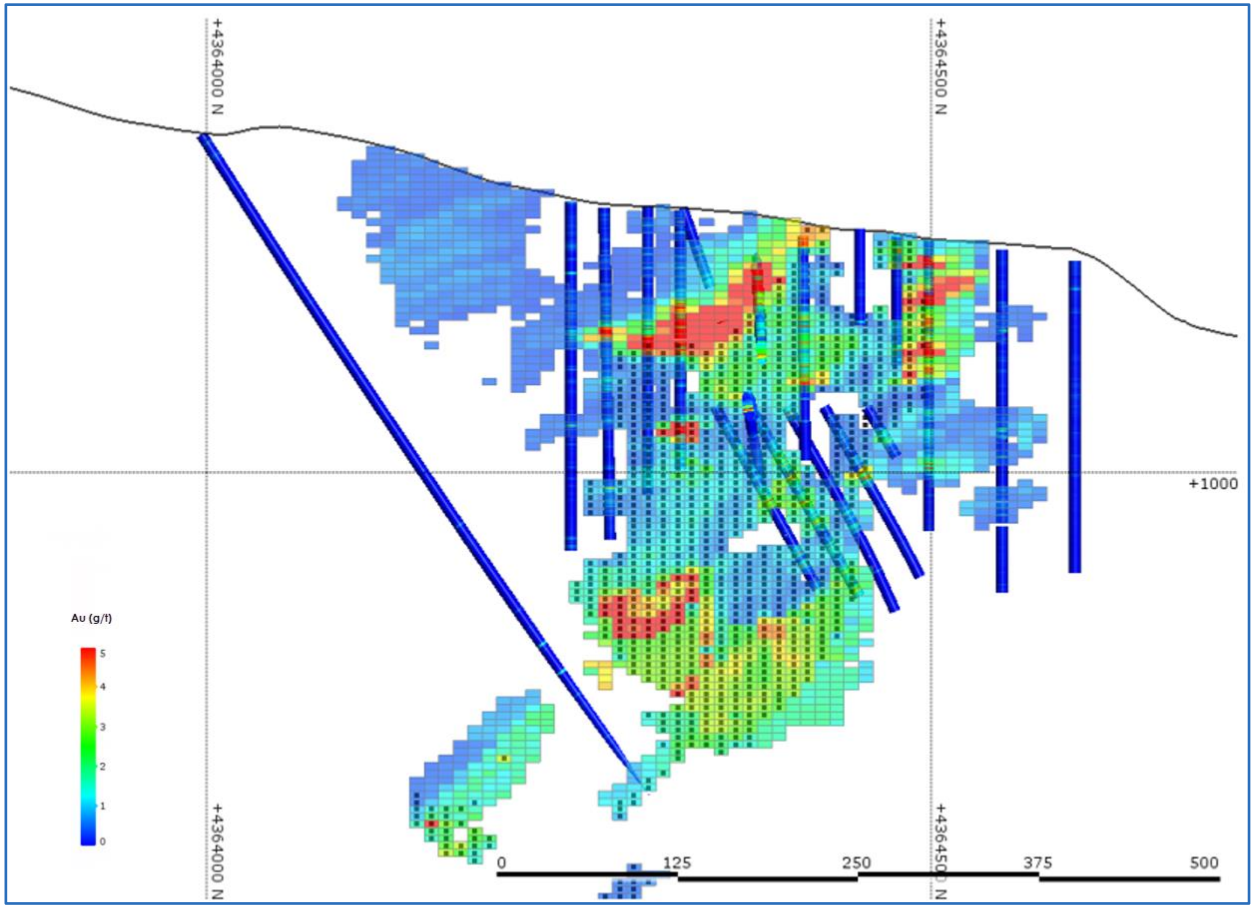
SSR Mining, 2016

Figure 14.12 Çöpler Geometry of the Sulfur Categorisation in the Gold Indicator PACK Model within the Gold Mineralisation Shell; Cross-Section 460,000 mE, (looking west)



SSR Mining, 2016

Figure 14.13 Çöpler Combined Indicator and Sulfur PACK Gold Estimates; Cross-Section 460,000 mE, (looking west)



SSR Mining, 2016
Drillholes and model use the same Au legend
Black dots indicate blocks with S ≥2.0%

Table 14.13 Çöpler Selected Gold Indicator Values for Oxide Material by Domain

Model Zone	Estimated Indicator Threshold (IND)
Manganese Zone	0.30
Main Zone	0.56
Marble Zone	0.35
West Zone	0.47

Sulfide production data showed higher grades and lower tonnes across all domains when compared to the PACK estimate, therefore an indicator threshold was not applied to reduce sulfide tonnes in the resource model.

Various periods of production data were reviewed with data from the 2014–2015 era used for model calibration.

Estimated Au grades in cells with an indicator estimate less than the indicator estimate threshold were set to a waste grade of 0.001 g/t Au.

14.1.13 Çöpler Resource Classification

Grade estimates were classified using the following SSR Mining guidelines:

- Indicated Mineral Resource should be quantified within relative ±15% with 90% confidence on an annual basis, and
- Measured Mineral Resources should be known within ±15% with 90% confidence on a quarterly basis.

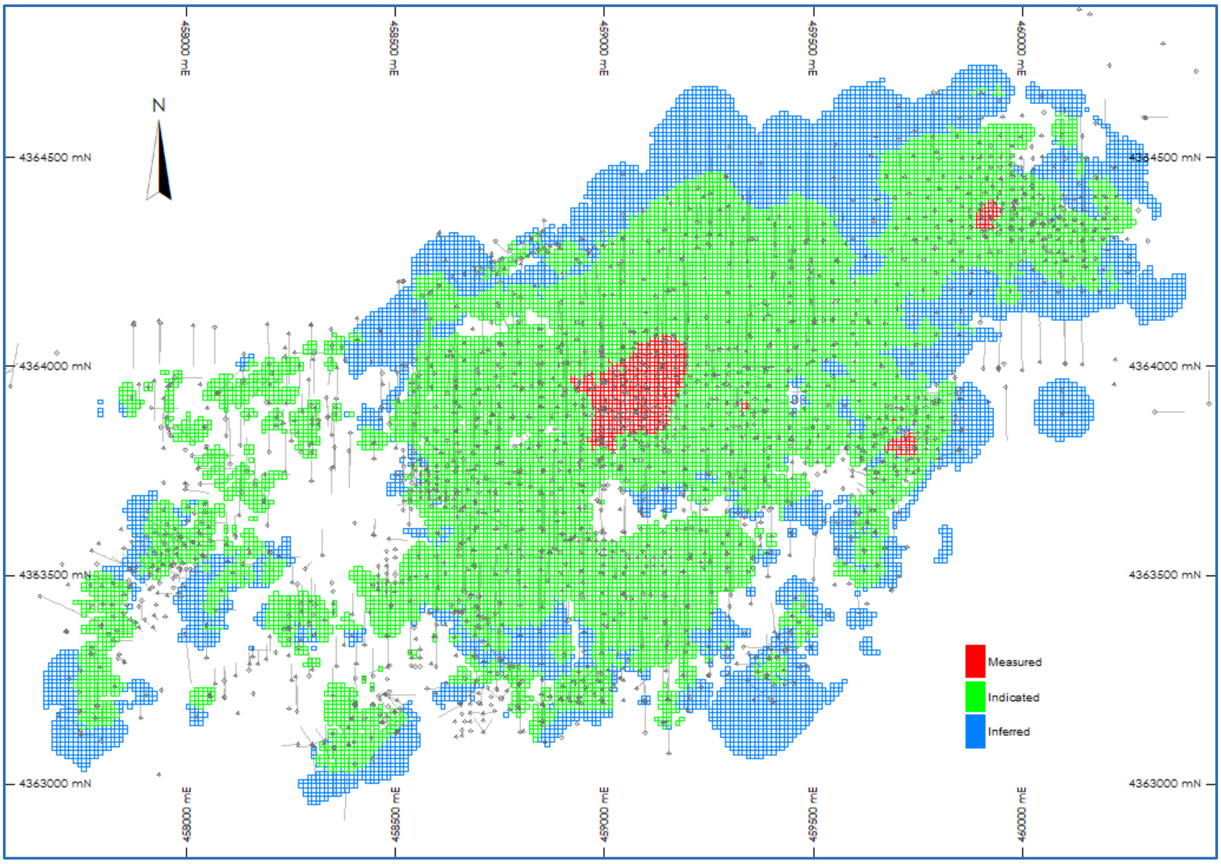
Based on these guidelines, the drilling is generally sufficiently close-spaced enough to permit confirmation of or assumption of continuity (Measured vs. Indicated, respectively) between data points. For the Çöpler model, a drillhole spacing study was performed to determine the nominal drillhole spacing required to classify material as Indicated.

Confidence limits were calculated on a single block that represents one month of POX production (based on 1.9 Mtpa). The confidence limits, a review of continuity on sections and plans, and an assessment of data quality were used to determine minimum drillhole spacing by domain. A spacing of 40 m x 40 m in the Marble Zone, 50 m x 50 m in the Manganese Zone and West Zones, and 60 m x 60 m in the Main Zone was required to meet the requirements for Indicated. An 80 m x 80 m spacing was required for Inferred in all domains. Model Cells with a drillhole spacing that was greater than 80 m were not classified as Mineral Resource.

The resultant classification was then 'smoothed' to remove the isolated cells that are not of the same classification tenor as the proximal surrounding cells.

The resulting classification shows the majority of the deposit can be classified as Indicated with Inferred cells forming a halo around the Indicated mineralisation, Figure 14.14. A small quantity of cells classified as Measured.

Figure 14.14 Projected Plan View of Çöpler Resource Classification



OreWin, 2020
Only model cells with Au >0.3 g/t shown

14.1.14 Çöpler Density Model Construction

Density measurements were performed on representative DD core by the site exploration geologists. Measurements were taken using the wax-coated water displacement method (Archimedes method). Results were then sent to the Anagold database manager where they were loaded into the corporate database.

Density values were assigned to the cell model based on rock type and depth below the surface. The density samples were first flagged by lithological code. Since lithological codes were not available for many of the density samples, Lithology was assigned using the lithological wireframes for all density values.

In total, 5,678 density measurements were used to estimate density. Since the majority of the measurements were taken in the diorite, marble and metasediment, the densities for these units are considered to be more reliable than the resulting manganese diorite density value used.

The density samples were then flagged by depth using wireframe solids for the three depth categories. The fourth category (>60 m) was considered as the default, and no solids were constructed for this category.

Density data were then reviewed spatially and statistically. Density values that fell outside expected upper and lower density limits (shown in Table 14.14) were considered to be outliers and removed, Table 14.14.

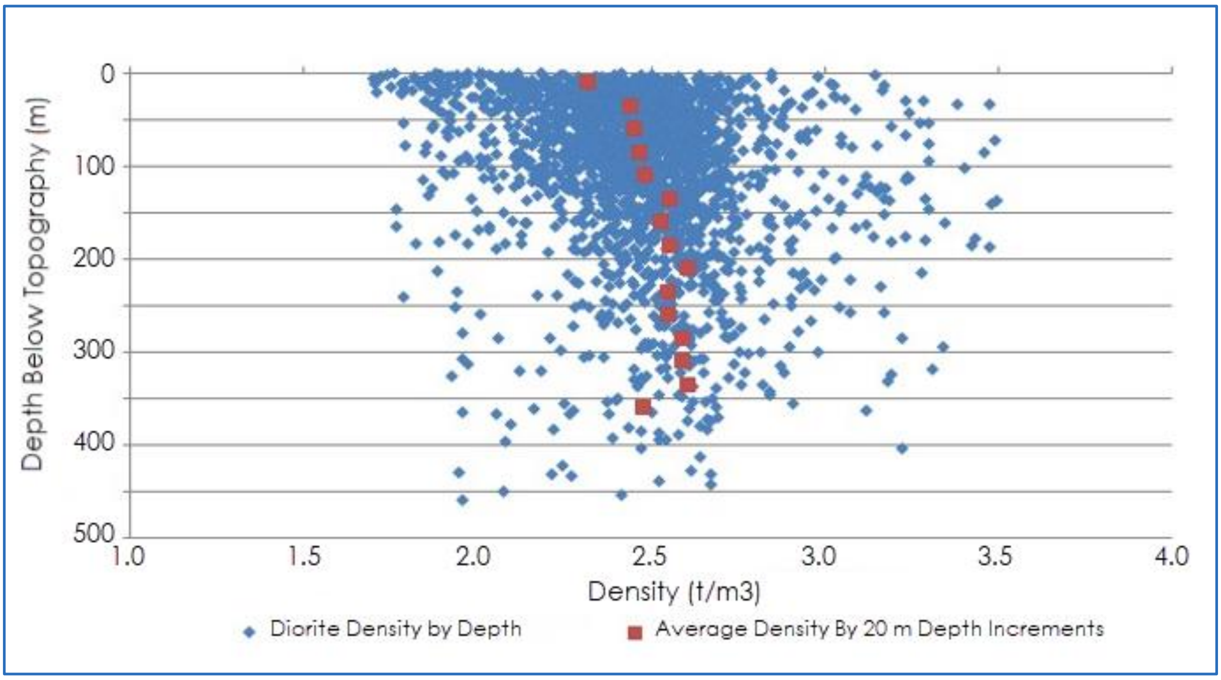
Table 14.14 Çöpler Upper and Lower Density Limits by Lithology

Lithology	Density Lower Limit (t/m^3)	Density Upper Limit (t/m^3)
Diorite	1.7	3.5
Metasediment	1.7	3.5
Marble	1.7	3.5
Manganese Diorite	–	–

The data were plotted by depth below the original topographic surface and categorised by lithology. The mean density was then calculated in 20 m depth bins below the original topographic surface. Based on the statistical analysis, density values were assigned by lithology and depth to the resource model. Density values are plotted by depth for the diorite in Figure 14.15 and for the metasediment in Figure 14.16. Since very little change in density with depth was noted for the marble, a single density value was applied to all cells in the resource model coded as marble.

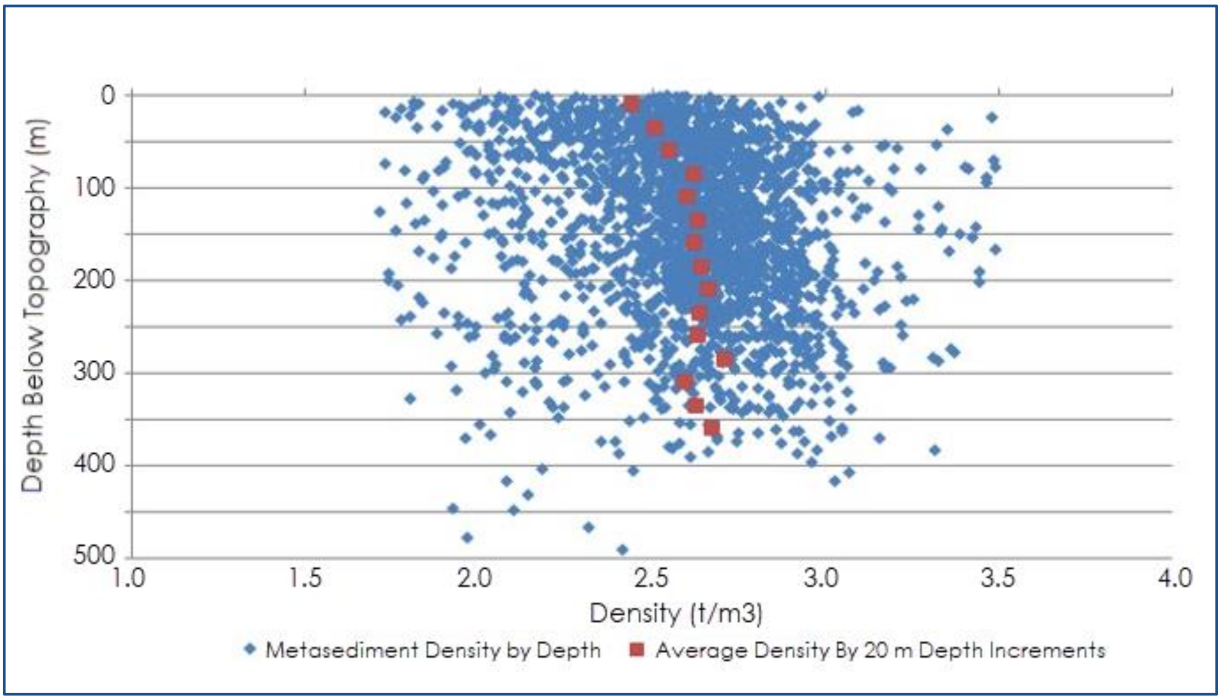
These data reflect the observed geology showing that the diorites and metasediment are more weathered closer to the surface, and the degree of weathering decreases with depth below the surface, resulting in an increase in density with depth as shown in Table 14.15. Densities used in the resource model are summarised in Figure 14.16.

Figure 14.15 Çöpler Diorite Density Values by Depth Below Surface



SSR Mining, 2016

Figure 14.16 Çöpler Metasediment Density Values by Depth Below Surface



SSR Mining, 2016

Table 14.15 Density Values Assigned to the Çöpler Block Model by Rock Type and Vertical Depth Below Surface

Rock Type	Depth (m)	No. Density Data	Assigned Density (t/m³)
Diorite	0–20	111	2.22
	20–40	173	2.42
	40–60	155	2.44
	60+	1,653	2.50
Metasediment	0–20	86	2.38
	20–40	209	2.51
	40–60	219	2.54
	60+	1,769	2.63
Marble	all	1,099	2.57
Manganese Diorite	all	23	2.63

14.1.15 Çöpler Model Validation

Model validation was approached in several ways, as described in the following sections.

14.1.15.1 Çöpler Visual Inspection

The estimated Au grades in the model were compared to the composite grades by visual inspection in plan views, north–south cross-sections, and east–west cross-sections.

In general, the model and composite grades compared well.

14.1.15.2 Çöpler Global Bias

The cell model was checked for global bias by comparing the mean Au, Ag, Cu, and S grades (with no cut-off) from the model (OK/ID2 grades) with means from NN estimates for cells of Indicated classification. The NN estimator produces a theoretically unbiased (de-clustered) estimate of the mean value when no cut-off grade is imposed and provides a reasonable basis for checking the performance of different estimation methods.

In general, an estimate is considered acceptable if the bias is at or below 5% (relative difference). Table 14.16 shows the bias results on a global basis.

Table 14.16 Çöpler Global Bias by Zone and Oxide / Sulfide Category

Zone	Element	Oxide / Sulfide	OK Estimates	NN Estimates	Rel. Diff. (%)
All Zones Combined	Au	Oxide	0.745	0.751	–0.7
	Au	Sulfide	1.045	1.005	3.9
	Ag	All	3.399	3.341	1.8
	Cu		0.029	0.029	0.9
	S		2.995	3.018	–0.8
Manganese Zone	Au	Oxide	1.151	1.147	0.3
	Au	Sulfide	1.571	1.482	6.1
	Ag	All	5.631	5.530	1.8
	Cu		0.019	0.018	3.3
	S		2.286	2.292	–0.2
Main Zone	Au	Oxide	0.514	0.513	0.1
	Au	Sulfide	0.961	0.930	3.4
	Ag	All	2.887	2.819	2.4
	Cu		0.034	0.034	1.0
	S		3.287	3.314	–0.8
Marble Zone	Au	Oxide	1.448	1.503	–3.6
	Au	Sulfide	1.802	1.751	2.9
	Ag	All	2.174	2.190	–0.7
	Cu		0.028	0.027	1.0
	S		1.423	1.417	0.4
West Zone	Au	Oxide	0.612	0.662	–7.4
	Au	Sulfide	0.428	0.421	1.6
	Ag	All	3.573	3.809	–6.2
	Cu		0.018	0.019	–4.0
	S		2.296	2.350	–2.3

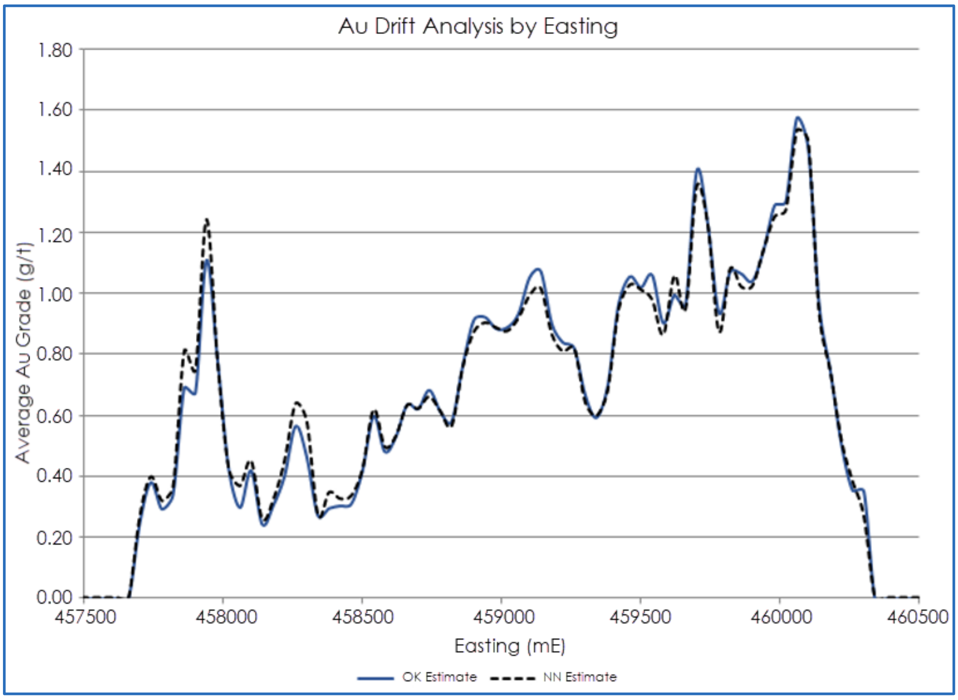
The West Zone shows the highest variance for oxide Au, however given the limited amount of drilling, mineralised material, and scheduled mining in this area, the variances were not considered likely to significantly impact operations.

14.1.15.3 Çöpler Local Bias

Local trends in the grade estimates (also known as drift analysis) were assessed by plotting the mean values from the NN estimate versus the kriged results for Indicated model cells in east–west, north–south and vertical directions (swath plots). Swath plots are shown in Figure 14.17, Figure 14.18, and Figure 14.19.

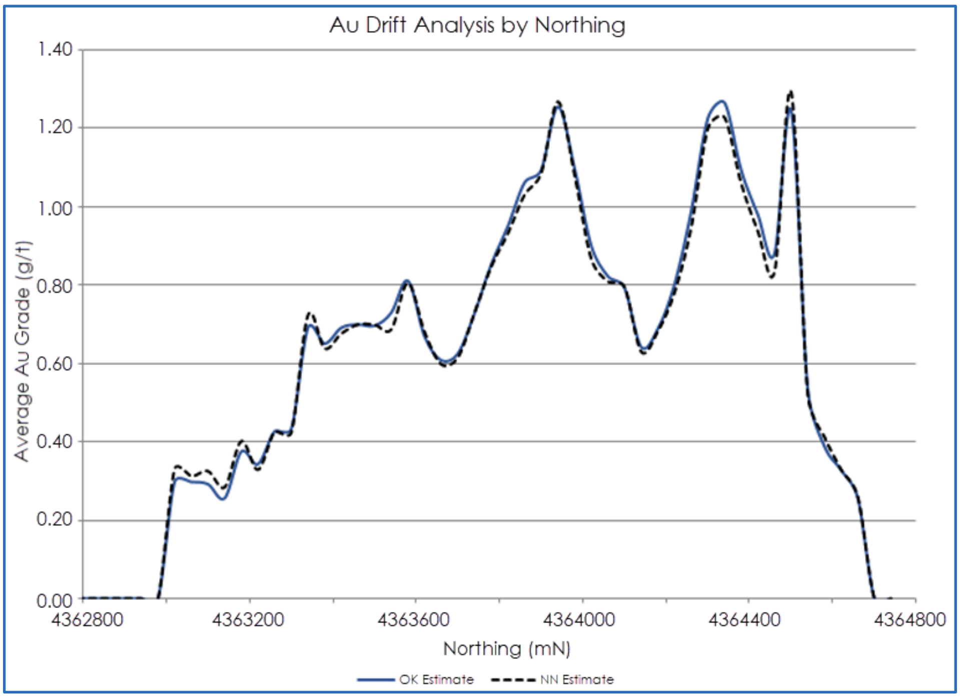
The swath grade profile plots help assess the local mean grades and are used here to validate grade trends in the model. The global comparisons agree well, however the swath plots do illustrate the existence of slight local differences between the NN and kriged model grades.

Figure 14.17 Çöpler Au Grade Swath Plot by Easting



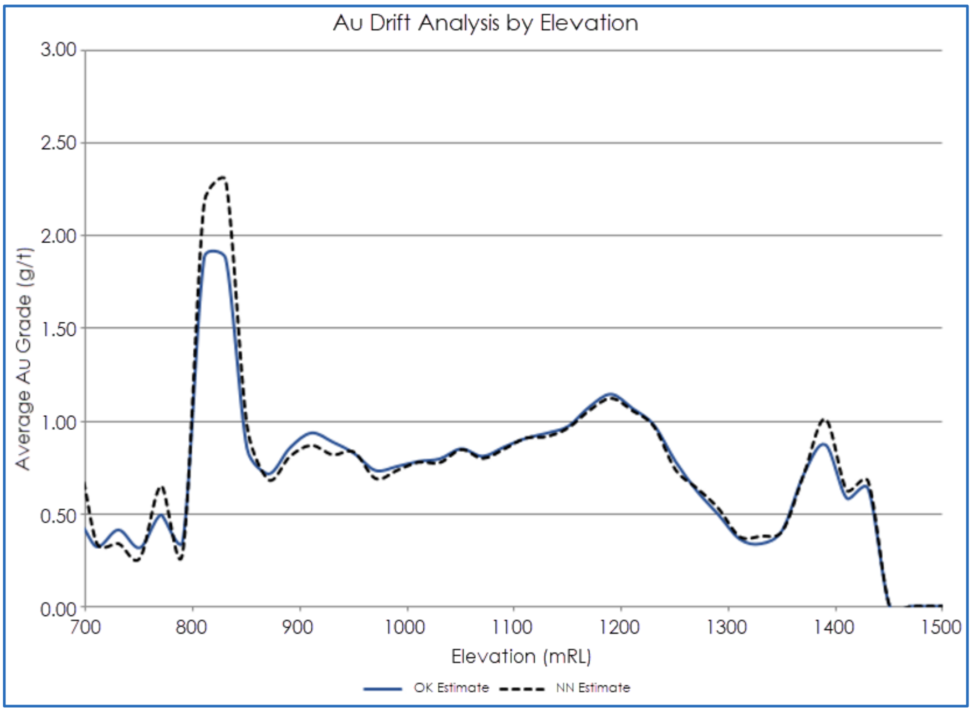
SSR Mining, 2016

Figure 14.18 Çöpler Au Swath Plot by Northing



SSR Mining, 2016

Figure 14.19 Çöpler Au Swath Plot by Elevation



SSR Mining, 2016

14.1.16 Çöpler Change of Support

Tonnes, grades, and resulting mass content from the grade estimates were calibrated against mine production data. Comparisons between the resource model and the ore control model (dig line model) were performed by material type, mine area, and time period. Indicator thresholds were modified for oxide material to minimise the variance between predicted resource model ounces and estimates using blast hole production data. The calibration step assumes historical mining practices will closely follow future mine operations.

The resource model calibration process involved:

- Reporting resource model ore tonnes and grades (Au and S) within each mining area. Mine production ore tonnes, Au grade, and material type (oxide or sulfide) was tracked by mining area through grade control.
- Tabulating material type above the relevant Au cut-off within each mining area. The estimated indicator threshold by cell, ranging from 0–1, was included for all material.
- Increasing the indicator threshold by individual 'mine domain' to obtain similar contained ounces in the resource model when compared to the grade control / production data.

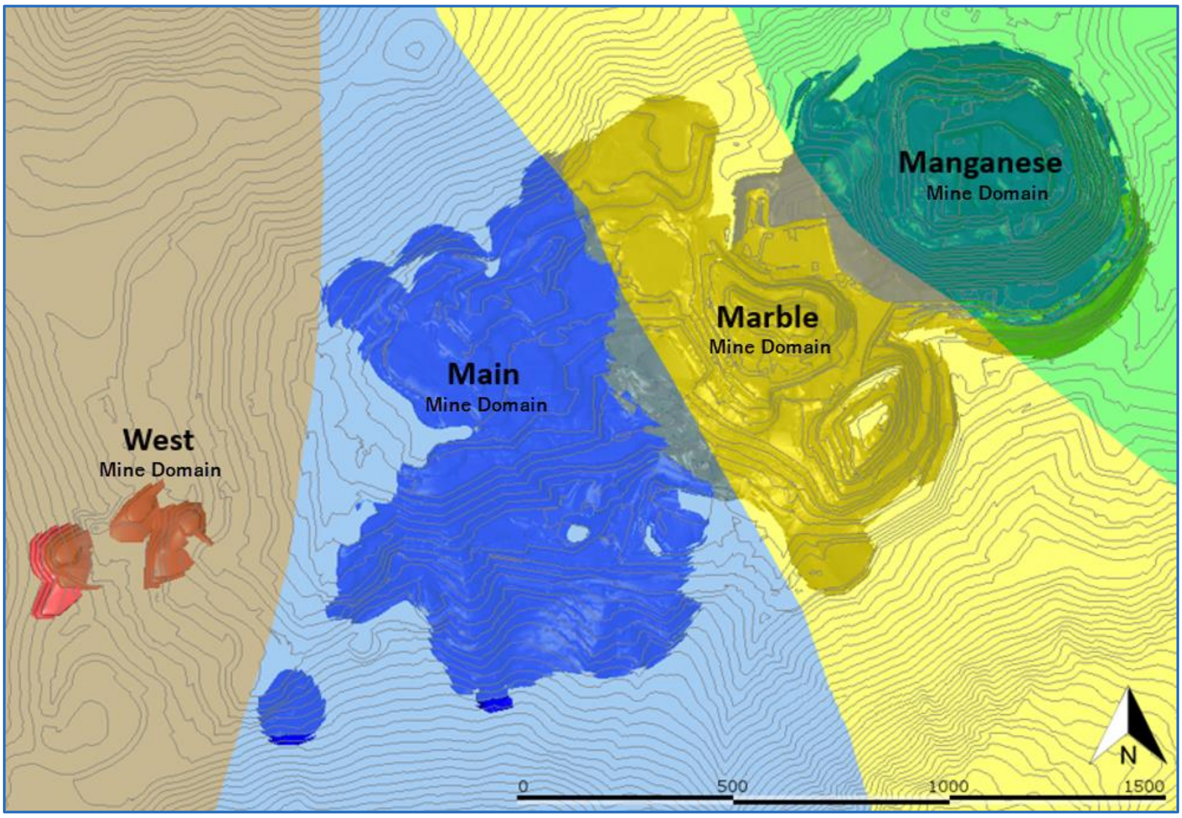
Mine domains were used in the resource model to allow the calibration according to the mined pit areas. The individual mine domains were used to calibrate estimated gold ounces with production information. Mine domains are shown in Figure 14.20.

Mine domain boundaries were generated and positioned based on mine design. Mine domains are not the same as model zones, despite having identical nomenclature.

Table 14.17 shows the relative difference of the ore control estimates when compared to the resource model estimates. Adjustment of the indicator threshold allowed calibration to achieve an overall variance on contained gold ounces of 1% for oxide and 9.4% for sulfide.

In the oxide, ore control has higher tonnage at slightly lower grade indicating the resource estimates have been slightly under-smoothed. The opposite appears to be the case in the sulfide. The resource model may not be capturing the higher grade, short-range structures that are seen in ore control with elevated Au grades.

Figure 14.20 Çöpler Mine Domains for Calibration of Gold Ounces (projected plan view)



SSR Mining, 2016

Table 14.17 Relative Difference: Ore Control vs. Resource Model Estimates at Çöpler

Ore Type	Mine Domain	Relative Difference (%)		
		Tonnes	Au Grade	Contained Gold
Oxide	Manganese	6.5	–4.7	1.5
	Main	6.7	–5.3	1.0
	Marble	–0.1	1.4	1.3
	West	12.0	–11.5	–0.9
	Total	5.1	–3.9	1.0
Sulfide	Manganese	–4.4	40.6	34.4
	Main	–2.4	1.0	–1.4
	Marble	–21.1	14.3	–9.8
	West	–2.5	–5.4	–7.7
	Total	–5.5	15.7	9.4
Total		2.7	1.4	4.1

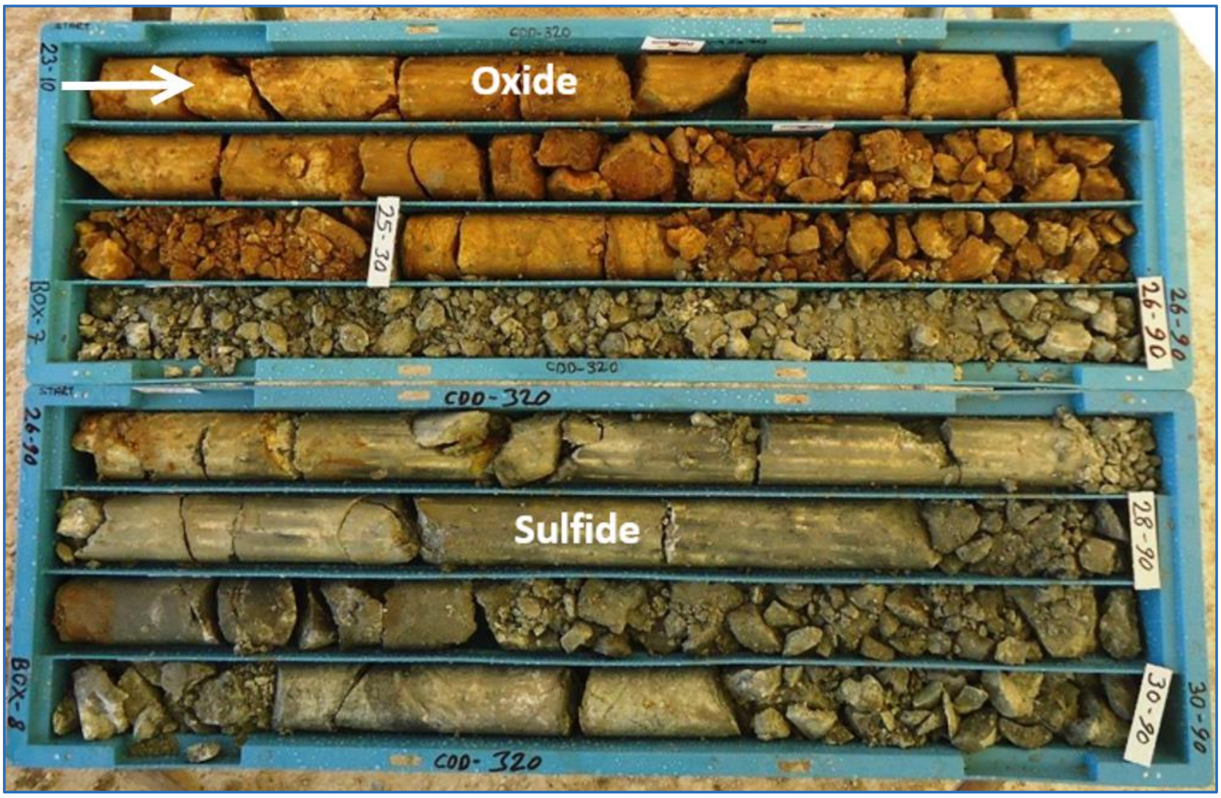
Positive percentages indicate ore control estimate is higher than the resource model estimate

14.1.17 Çöpler Oxidation Model

The oxidation model reflects oxidation due to surficial weathering and/or oxidation resulting from the manganese alteration. Oxide (low-sulfur material (S <2%)) can be processed by heap leaching while sulfide (high-sulfur material (S ≥2%)) is processed through the POX plant.

The low/high-sulfur criteria were further finessed using the logged colour codes and pyrite percentages recorded in the drillhole logs. Review of the logs showed a generally relatively sharp colour change from orange–brown tones to grey–black tones (Figure 14.21).
A wireframe was constructed to represent this logged colour change. The wireframe was further refined using the logged visual estimates of pyrite. Near-surface material is highly oxidised and usually does not include visually identifiable sulfides, while visual sulfide percentage increases with depth to a point (pyrite ≥1%) where the percent pyrite can be estimated and recorded in the drill logs. In general, the 1% visual pyrite boundary matched the red–grey colour boundary within approximately 5 m, but locally deviated up to 10 m. The 5 m variance is considered to be within the accuracy of the data, as it reflects the composite sample length and the mining bench height.

Figure 14.21 Çöpler Drill Core Showing Colour Change from Oxide to Sulfide



SSR Mining, 2016

The resulting oxide-sulfide wireframe boundary was compared to the sulfur-estimates model. This comparison showed that the S <2% and S ≥2% domains matched the oxide-sulfide boundary reasonably well, although there are local areas of material with S <2% below the oxide-sulfide surface which are due in part to deeper weathering along structures. As a result, the oxide boundary surface is considered to be somewhat conservative locally in estimating the amount of oxide material.

Blast hole data from Main Zone that contains both Au fire assays (AuFA) and cyanide leach assays (AuCN) show that the gold recovery significantly decreases below the oxide / sulfide boundary. This implies there is low-sulfur material below the oxide / sulfide boundary that has not oxidised, and hence lower recoveries are obtained by cyanide leaching. As a result, the oxide / sulfide boundary is used in the Main Zone to delineate material types. In the Manganese and Marble zones, however, the estimated sulfur content is used to delineate material.

In the eastern portion of the Çöpler deposit, the oxidation profile is better-developed and follows the diorite intrusion. This contrasts with the much shallower oxidation profile in the western portion of the mining operation.

14.1.18 Çöpler Assessment of Reasonable Prospects of Eventual Economic Extraction

Mineral Resource estimates were shown to meet reasonable prospects for eventual economic extraction criteria by reporting only material that was contained within a conceptual pit shell using metal prices of $1,750/oz for gold and the parameters summarised in Table 14.18. These parameters, with the exception of the gold price, are the same parameters as those used to define the Mineral Reserve pit.

Table 14.18 Summary of Key Parameters Used in 2020 Conceptual Pit Shell at Çöpler

Description	Unit	Minimum	Maximum
Heap Leach Gold Recovery	%	62.3	78.4
POX Gold Recovery	%	85.0	85.0
Mining Cost per tonne mined	$/t	1.89	1.89
Process Costs Heap Leach	$/t	12.31	12.31
Process Costs POX	$/t	31.00	31.00
Site Support per tonne processed	$/t	3.17	6.60
Internal Au Cut-off – Heap Leach	g/t	0.32	0.41
Internal Au Cut-off – POX	g/t	0.73	0.73
Royalty	%	2.0	2 .0

14.1.19 Çöpler Deposit Mineral Resource Tabulation

Mineral Resources are reported inclusive of Mineral Reserves in Table 14.19 according to resource classification and material type. Mineral Resources that are not Mineral Reserves have not demonstrated economic viability. The overall tonnage and grade estimate have increased for oxide and sulfide material from the previously-reported estimate in 2019. This change is predominantly due to the change in gold price from $1,500/oz in 2019 to $1,750/oz in 2020, and the associated resultant drop in cut-off grades. The pit shell used to constrain the resource has been updated to reflect the increase in gold price. Depletion from mining has been included.

Table 14.19 Çöpler Mineral Resource Table by Classification and Oxide State

Mineral Resource Statement for the Çöpler Deposit (as at the Effective Date)							
Cut-off Grade (Au g/t)	**Material Type**	**Resource Category / Material**	**Tonnage (kt)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Contained Gold (koz)**
Variable	Oxide	Measured	287	1.29	7.75	0.09	12
		Indicated	25,139	0.98	3.44	0.15	789
		Indicated – Stockpile	–	–	–	–	–
		Measured + Indicated	25,427	0.98	3.49	0.15	801
		Inferred	33,083	0.96	7.16	0.13	1,017
0.73	Sulfide	Measured	2,454	2.22	7.21	–	175
		Indicated	77,884	1.78	5.04	–	4,451
		Indicated – Stockpile	6,674	2.63	–	–	564
		Measured + Indicated	87,012	1.86	4.71	–	5,190
		Inferred	34,073	1.54	12.72	–	1,692
Variable	Total	Measured	2,741	2.12	7.27	0.01	187
		Indicated	109,697	1.65	4.37	0.03	5,804
		Measured + Indicated	112,438	1.66	4.44	0.03	5,991
		Inferred	67,156	1.25	9.98	0.06	2,709

1. Mineral Resources have an effective date of 27 November 2020.
2. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. Mineral Resources are shown on a 100% basis, of which SSR Mining owns 80%.
4. In the Main pit, oxide is defined as material above the interpreted oxide surface. All material beneath the oxide surface in this area is classified as sulfide. A transitional zone was not used. The Manganese and Marble pit are divided into oxide material (<2% total sulfur) and sulfide material (≥2% total sulfur).
5. All Mineral Resources in the CDMP20 were assessed for reasonable prospects for eventual economic extraction by reporting only material that fell within conceptual pit shells based on metal prices of $1,750/oz for gold. The following parameters were used: metallurgical recoveries in oxide 62.3%–78.4%, and in sulfide 85.0%; Au cut-off grades in oxide 0.32–0.41 g/t Au, and in sulfide 0.73 g/t Au, (there are no credits for Ag or Cu in the cut-off grade calculations); allowances have been made for royalty payable.
6. Reported Mineral Resources contain no allowances for unplanned dilution or mining recovery.
7. Tonnage and grade measurements are in metric units. Contained gold is reported in troy ounces.
8. Tonnages in Table 14.19 are rounded to the nearest thousand tonnes; grades are rounded to two decimal places. As a result, totals may not match.

14.2 Çakmaktepe

The Çöpler district hosts various styles of mineralisation, mainly epithermal, skarn and contact style gold and gold–copper mineralisation. The Çakmaktepe North zone of the Çakmaktepe deposit is a strongly sheared zone with strong epithermal characteristics and grade associations with intrusive diorite dykes. As with the other prospects the mineral association is dominantly Au–Cu–Ag. Other mineralised zones belonging to the Çakmaktepe deposit are generally contact styles of mineralisation where Au–Cu–Ag have been emplaced along thrust surfaces next to ophiolite, limestone and metasediment. Epithermal veining and replacement alteration textures are prevalent.

Oxide mining began in the Çakmaktepe Central and East pits in November 2018. Mining continued through September 2019 within the same two pits. Oxide ore material was transported to the Çöpler oxide processing facility for inclusion on the heap leach pad.

A geological model was constructed along with a cell model estimating grades for Au, Cu, Ag, S, and C. Estimated grades were constrained by mineralised envelopes.

14.2.1 Çakmaktepe Domains

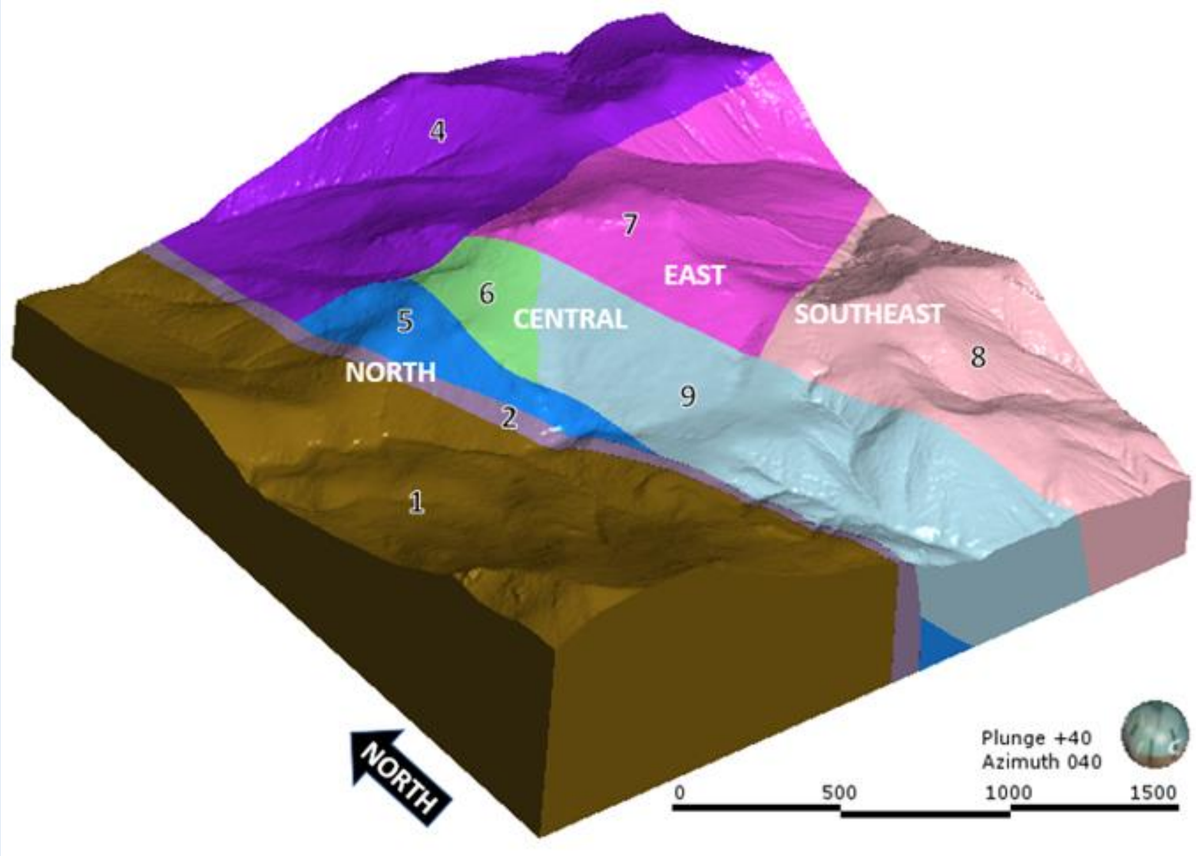
At Çakmaktepe, mineralisation follows structural controls and designated lithological contact orientations. Grades trends and element associations were investigated, and a number of separate domains were identified and are shown in Figure 14.22.

Mineralisation at Çakmaktepe often overlaps multiple lithological units along its boundary, rather than being hosted within a single rock type. For this reason, grade shells were constructed for gold and copper to constrain estimates within mineralised zones. These were constructed by manually selecting 5 m composites along identified mineralisation trends. Mineralised trends were honoured in 3D with no grade cut used to bound mineralised shapes. The resulting mineralised shapes for gold and copper are lenticular with thicknesses ranging from 5–40 m, the average thicknesses being approximately 6 m.

Grade shells were also developed for silver. However, because silver mineralisation tends to be more dispersed and more difficult to follow across the deposit than gold, different methods were used for silver grade shells depending on which area was being modelled. For Çakmaktepe North, the silver grade shell was constructed using the same trends identified for gold since this area is strongly controlled by structural features. For the remaining areas of Çakmaktepe, general trends resembling lithological orientations were used to construct 4 g/t Ag grade shells. These shells tend to be less continuous than the gold and copper vein lenses but can be followed across the deposit areas as an overall trend.

Sulfur grades follow lithological units. Higher S values are seen in diorite and metasediment, with decreased S in gossan, jasperoid, ophiolite, and marble.

Figure 14.22 Çakmaktepe Model Domains (oblique view)



SSR Mining, 2020

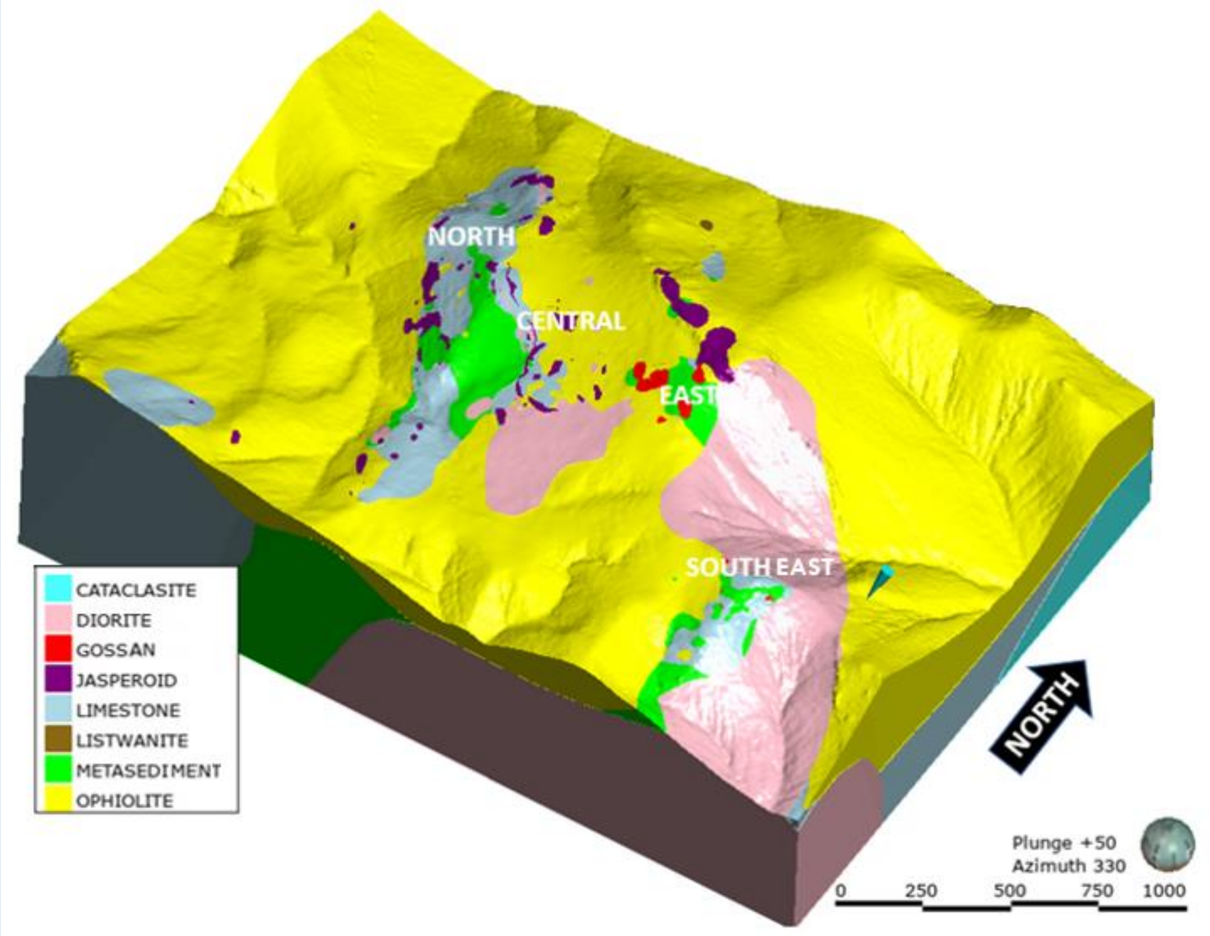
The key points in relation to Çakmaktepe mineralisation domains are:

- Çakmaktepe North is located on a vertical shear structure with elevated metal grades within jasperoid unit. Several low-angle structures dipping to the north-east carry grades along the marble to metasediment contact. Intrusive diorite/s, orientated vertically, cross-cut all other lithological units. Mineralisation within/around the diorite is limited in Çakmaktepe North.
- Çakmaktepe Central mineralisation follows the marble contact, which dips gradually to the north-east. The marble unit is approximately 15 m thick and located between the ophiolite and metasediment units.
- Mineralisation in the Çakmaktepe East area is near-surface and within the gossan unit, which is relatively flat lying and localised.
- The Southeast area seems to be controlled by a massive diorite body with gossan at the surface. Mineralisation is weak and near surface.

14.2.2 Çakmaktepe Geological Model

The Çakmaktepe deposit includes four distinct areas, North, Central, East, and South-east. A single geological model was constructed to include the four areas (see Figure 14.23).

Figure 14.23 Çakmaktepe Geological Model and Deposit Areas (oblique view)



SSR Mining, 2020

Contacts for lithological shapes used the raw logged interval depth in 3D space. Surfaces were generated through implicit modelling of contact locations in the drillholes.

Construction of the lithological shapes assumed the following:

- Diorites are intrusive units that can exist as large bodies or thin sills cross cutting other units.
- Jasperoid is an alteration product but treated here as a lithological unit. Jasperoids occur along shear zones and are high in pyrite. Jasperoid can exist in pods and can be discordant to surrounding stratigraphy.
- Gossan is primarily the result of surficial oxidation, with the shape influenced by the local topographic elevation.

- In most areas, marble overlays metasediment, with ophiolite above marble.
- Offsets in lithological units help to define fault locations and structural boundaries.

A series of fault surface wireframes were developed in an effort to represent the structural knowledge at Çakmaktepe. These structures extended beyond the Çakmaktepe model area to take into consideration the spatial relationships between Çakmaktepe and Ardich. The incorporation of modelled 3D faults into the geological model highlighted a discrepancy between the Ardich lithological concept and the Çakmaktepe geological units. Given the correlation of the two deposits was not clearly defined at the time of this model, interpreted faults were excluded from the Çakmaktepe geological model.

14.2.3 Çakmaktepe Data Summary

The cut-off date for the export of the drillholes from the database to be used in the resource modelling was 31 October 2019. The extract contained 1,109 drillholes with a drilling date range of September 2007–October 2019, totalling of 119,001 m of drilling.

14.2.3.1 Çakmaktepe Drillhole Compositing

The original sample lengths in the Çakmaktepe dataset are predominately 1 m, with some 2 m sampling through zones presumed at the time of drilling to be waste. The average sample length is 1.02 m. The shortest interval was 0.1 m and the maximum length was 3.1 m.

Assayed intervals below the laboratory detection limit are stored as half the laboratory detection limit.

Samples were composited to 5 m lengths for use in statistical analysis and construction of mineralisation boundaries. Often, composites along lithological boundaries were selected to match geological control with mineralisation.

During compositing, missing data is denoted as –99 and excluded from the composite calculation. Composites do not truncate at geological boundaries. End (or tail) composites of <5 m are length-weighted during the grade estimate.

Composites are flagged within the mineralisation shapes. Lithology is also coded into the composite file based on the interpreted shapes.

14.2.4 Çakmaktepe Exploratory Data Analysis

14.2.4.1 Çakmaktepe Summary Statistics

Table 14.20 summarises the univariate statistics calculated for each gold shell constructed in the North and Central areas. Mean grades within the grade zones vary from low grade (0.53 g/t) to moderately high (2.02 g/t).

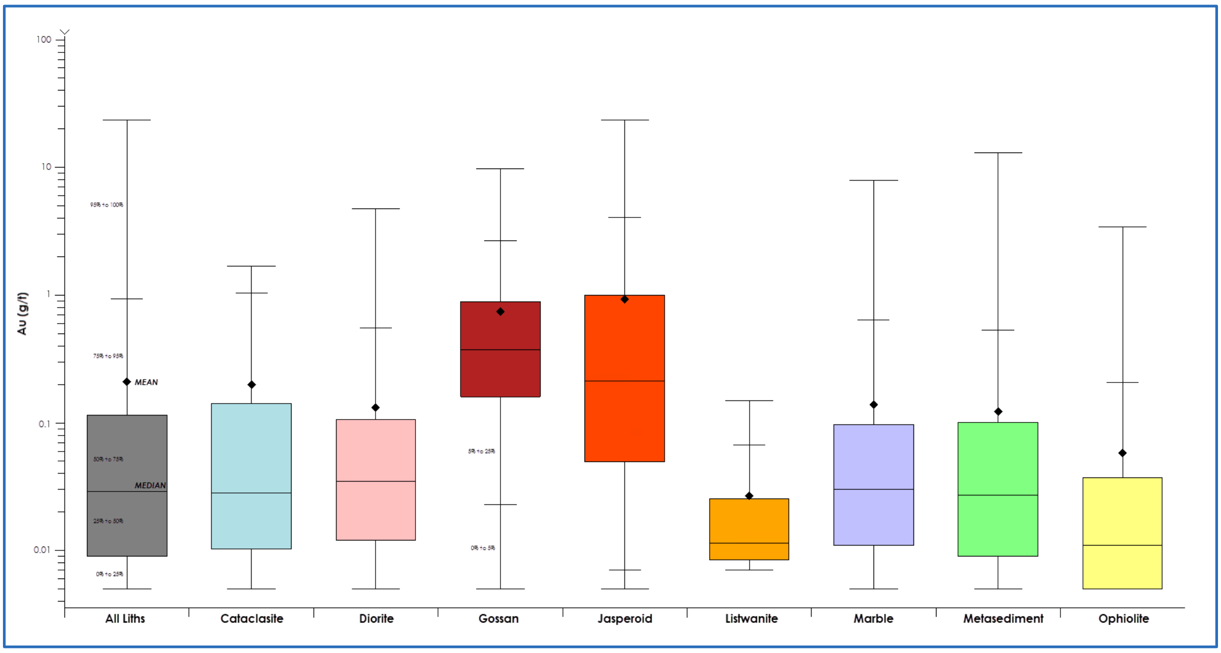
Table 14.20 Çakmaktepe Gold Statistics (based on 5 m Composites)

Gold Grade Shell	Count	Min.	Max.	Mean	Std. Dev.	Variance	CV
1	801	0.005	23.62	1.51	2.07	4.29	1.37
2	185	0.005	6.61	0.53	0.99	0.98	1.89
3	78	0.005	4.02	1.18	0.95	0.90	0.81
4	100	0.005	20.25	1.86	2.87	8.22	1.54
5	33	0.005	11.38	2.02	3.17	10.05	1.57
6	68	0.005	4.57	0.70	0.85	0.73	1.21
7	82	0.005	4.71	0.92	0.93	0.87	1.01
8	352	0.005	13.10	1.89	2.22	4.92	1.17
9	38	0.005	5.13	0.89	0.92	0.84	1.03
10	15	0.046	5.28	0.95	1.51	2.27	1.59
11-CE_0pt3	474	0.007	6.34	0.73	0.86	0.74	1.17
12-CSE_0pt3	223	0.007	9.80	0.85	1.16	1.36	1.38

14.2.4.2 Çakmaktepe Box Plots

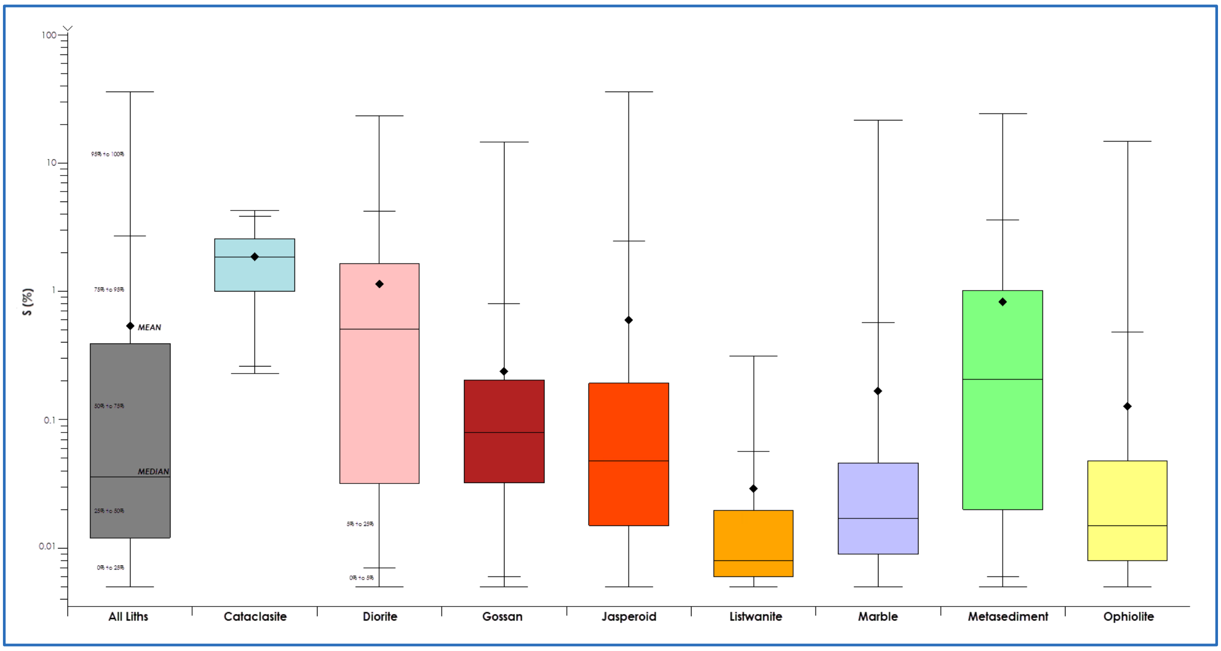
Box plots were created to facilitate comparisons of metal grades between lithologies and domains. Example box plots for gold and sulfur are shown in Figure 14.24 and Figure 14.25.

Figure 14.24 Çakmaktepe Box Plot of 5 m Au Composites by Lithology



Modified from SSR Mining, 2020

Figure 14.25 Çakmaktepe Box Plot of 5 m Sulfur Composites by Lithology



Modified from SSR Mining, 2020

14.2.4.3 Çakmaktepe Correlation Coefficients

Mineralisation tends to spatially follow lithological contacts. There is variable correlation between Au, Ag, and Cu. Sulfur shows a direct association to lithological unit. Correlation coefficients are summarised in Table 14.21.

Table 14.21 Çakmaktepe Correlation Coefficients using 5 m Composites

	AuFA (g/t)	Ag (g/t)	Cu (%)	S (%)	C (%)	As (ppm)
AuFA (g/t)	**1.00**					
Ag (g/t)	0.31	**1.00**				
Cu (%)	0.23	0.38	**1.00**			
S (%)	0.08	0.06	0.07	**1.00**		
C (%)	–0.02	–0.01	–0.03	–0.16	**1.00**	
As (ppm)	0.56	0.31	0.27	0.14	0.02	**1.00**

14.2.4.4 Çöpler Statistical Review

Detailed statistical analyses were undertaken to assist with the understanding of the mineralisation distribution in the various domains. The statistical review included typical univariate statistics (tabulations, histograms, box plots) and bivariate statistics (scatter plots, correlations).

A summary of key findings follows:

- The box plot confirms observations made from histograms and probability plots that gossan and jasperoid contain significantly higher Au grades and the remaining units (diorite, metasediment, ophiolite, and marble) have lower Au grades.
- Box plots of sulfur show higher sulfur content in diorite and metasediment with moderate sulfur grades in gossan and jasperoid. Low sulfur is consistently seen in ophiolite and marble. For this reason, the sulfur estimate uses lithologic contacts as domain boundaries.
- Mineralisation tends to spatially follow lithological contacts.
- For Çakmaktepe Central, the probability plot is relatively straight, indicating only one population is present in the distribution.

14.2.5 Çakmaktepe Core Recovery

Core recovery is calculated on a per metre basis of recovered core and entered into the database as a percentage. In general, core recoveries are between 80%–90%, reflecting strongly sheared, brecciated, altered and in areas of limestone, karstic ground (cavities) being drilled at Çakmaktepe.

14.2.6 Çakmaktepe Twin Holes

A set of four twin holes were drilled at Çakmaktepe. Each pair contained one DD and one RC hole. Scissor holes were used for validation of grade distribution, grade tenor, orebody boundary definition and metallurgical test sample collection. Many drill sections contain drill fans, testing grade recurrence within 5m to 10m of original holes and confirming mineralised orientation.

14.2.7 Çakmaktepe Contact Plots

Contact plots were created to show grades change across geological boundaries. Jasperoid and gossan are favourable mineralisation hosts and show abrupt grade changes when compared to the other lithologies (marble, metasediment, diorite, ophiolite).

Grade shell boundaries were constructed to follow lithological contacts and were used as hard domains in the grade estimation process.

14.2.8 Çakmaktepe Top Cutting

To determine appropriate top cuts, statistical studies were performed for Au, Ag, Cu, and S. The study looked at:

- Probable top cut thresholds based on indicator correlation
- Change in CV for the highest grade sample population
- Log probability plots by domain
- Top sample value curves by domain

Top cuts were selected based on the log probability plot, supported by the projection of the data trend to the expected upper grade (y-axis value) using the top sample value curve (Table 14.22).

Top cutting occurred after compositing to 5 m. A spatial review of top values by domain shows randomly spaced samples rather than a localised body of higher grades.

High yield limits were included outside of the grade shells to restrict the extrapolation of higher grades within the applied search distance. For Au, sample values above 4 g/t Au were restricted to a distance of 10 m x 10 m x 5 m in the East and Southeast areas. For Central, a high yield limit of 8 g/t Au was used. The high yield limit was increased to 12 g/t Au in the Çakmaktepe North area. For copper, samples above 2% Cu were restricted to 10 m x 10 m x 5 m in Central and 3% Cu in North and East.

Table 14.22 Çakmaktepe Top Cuts for Au, Cu, and Ag

Element	Çakmaktepe Area	Top Cut Grade	No. Samples Cut
Au (g/t)	North	15.0	2
	Central	9.0	7
	East	5.5	1
	Southeast	5.0	4
Cu (%)	North	4.0	2
	Central	3.0	2
	East	4.0	2
	Southeast	1.0	2
Ag (g/t)	North	180	2
	Central	130	3
	East	150	5
	Southeast	60	5

14.2.9 Çakmaktepe Resource Model Estimation

14.2.9.1 Çakmaktepe Cell Model

A cell model was constructed by first coding the interpreted lithology shapes into the cells. These were then flagged by each of the grade shells and model domains. A project-wide solid was used to trim out distant cells at model edges.

The cell model limits are shown in Table 14.23.

The model was not rotated, and no sub-celling was used.

Table 14.23 Çakmaktepe Cell Model Parameters

Direction	Minimum (m)	Maximum (m)	Range (m)	Cell Size (m)	No. of Cells
East	463,400	465,700	2,300	5	460
North	4,364,800	4,366,700	1,900	5	380
RL	1,050	1,850	800	5	160

14.2.9.2 Çakmaktepe Estimation Method

Au, Ag, Cu, S, and C were interpolated using ID3 and NN methods. Au, Cu, and Ag were estimated according to grade shell constraints. S and C were estimated by modelled lithological units. All grade shell boundaries were treated as hard. Mineralisation domains were treated as soft boundaries allowing the selection of samples from nearby domains.

A single search distance was used within the gold, copper, and silver grade shells. A two-pass method was used to estimate cells outside of the grade shells. Search ranges and sample requirements varied by estimation pass.

A summary of the estimation parameters is shown in Table 14.24.

Table 14.24 Çakmaktepe Estimation Parameters Inside Gold Grade Shells

Gold Grade Shell	Search Orientation			Axis Distance			
	Azimuth	Plunge	Dip	Major	Semi-Major	Minor	Min./Max.
1	68	–77	0	100	100	30	3/12
2	42	–18	0	100	100	30	3/12
3	350	–14	0	100	100	30	3/12
4	350	–14	0	100	100	30	3/12
5	296	–40	0	100	100	30	3/12
6	60	–77	0	100	100	30	3/12
7	316	–4	0	100	100	30	3/12
8a	67	–18	0	100	100	30	3/12
8b	316	–4	0	100	100	30	3/12
9	42	–18	0	100	100	30	3/12
10	266	–2	0	100	100	30	3/12
11	90	–2	0	100	100	30	3/12
12	207	–18	0	100	100	30	3/12

Search orientations were selected to match the mineralised dip and dip-direction. The gold grade shell 8 is large in size, with the mineralisation changing orientations at its two ends. For this reason, grade shell 8 was split into 8A and 8B sections to estimate grade at two different orientations.

Au was interpolated within each gold grade shell using only composite samples inside the shell. Au grade was then interpolated into cells outside the grade shell using domain-specific parameters (Table 14.25).

Table 14.25 Çakmaktepe Estimation Parameters for Au, Cu and Ag by Domain

Gold Domain	Search Orientation			Axis Distance – Pass 1				Axis Distance – Pass 2			
	Azimuth	Plunge	Dip	Major	Semi-Major	Minor	Min./Max.	Major	Semi-Major	Minor	Min./Max.
1	350	–14	0	40	40	30	3/12	80	80	60	2/12
2	68	–77	0	40	40	30	3/12	80	80	60	2/12
3	72	–64	0	40	40	30	3/12	80	80	60	2/12
4	245	–52	0	40	40	30	3/12	80	80	60	2/12
5	42	–18	0	40	40	30	3/12	80	80	60	2/12
6	67	–18	0	40	40	30	3/12	80	80	60	2/12
7	90	–2	0	40	40	30	3/12	80	80	60	2/12
8	207	–18	0	40	40	30	3/12	80	80	60	2/12
9	316	–4	0	40	40	30	3/12	80	80	60	2/12

Cu was estimated using the same method as Au, by first interpolating grade within the copper grade shells and then interpolating outside the grade shells in two passes.

A summary of the estimation parameters for Cu are shown in Table 14.26 and Table 14.27.

Table 14.26 Çakmaktepe Estimation Parameters Inside Copper Grade Shells

Copper Grade Shell	Search Orientation			Axis Distance			
	Azimuth	Plunge	Dip	Major	Semi-Major	Minor	Min./Max.
1a	67	–18	0	100	100	30	3/12
1b	316	–4	0	100	100	30	3/12
2	68	–77	0	100	100	30	3/12
3	42	–18	0	100	100	30	3/12
4	350	–14	0	100	100	30	3/12
5	42	–18	0	100	100	30	3/12
6	345	–27	0	100	100	30	3/12
7	42	–18	0	100	100	30	3/12
8	52	–78	0	100	100	30	3/12
9	350	–14	0	100	100	30	3/12
10	90	–2	0	100	100	30	3/12
11	207	–8	0	100	100	30	3/12

Ag estimation followed the same technique as Au and Cu by interpolating within the silver grade shell and then interpolating outside the grade shell by domain.

A summary of the estimation parameters for Ag are shown in Table 14.26 and Table 14.27.

Table 14.27 Çakmaktepe Estimation Parameters Inside Silver Grade Shells

Ag Domain	Search Orientation			Axis Distance			
	Azimuth	Plunge	Dip	Major	Semi-Major	Minor	Min./Max.
1	350	–14	0	100	100	50	3/12
2	256	–86	0	100	100	50	3/12
3	72	–64	0	100	100	50	3/12
4	245	–52	0	100	100	50	3/12
5	42	–18	0	100	100	50	3/12
6	67	–18	0	100	100	50	3/12
7	90	–2	0	100	100	50	3/12
8	207	–18	0	100	100	50	3/12
9	316	–4	0	100	100	50	3/12

Sulfur and carbon content is linked to lithology. Lithological shapes were used as hard boundaries to interpolate S and C grades. No preferred orientation of S or C grades was observed; therefore, a spherical search was used. A two-pass estimate was run on S and C.

A summary of the estimation parameters for sulfur and carbon are shown in Table 14.28.

Table 14.28 Çakmaktepe Estimation Parameters for Sulfur and Carbon

Lithology	Search Orientation			Axis Distance – Pass 1				Axis Distance – Pass 2			
	Azimuth	Plunge	Dip	Major	Semi-Major	Minor	Min./Max.	Major	Semi-Major	Minor	Min./Max.
Gossan	0	0	0	30	30	30	3/12	90	90	90	2/12
All Other	0	0	0	40	30	30	3/12	90	90	90	2/12

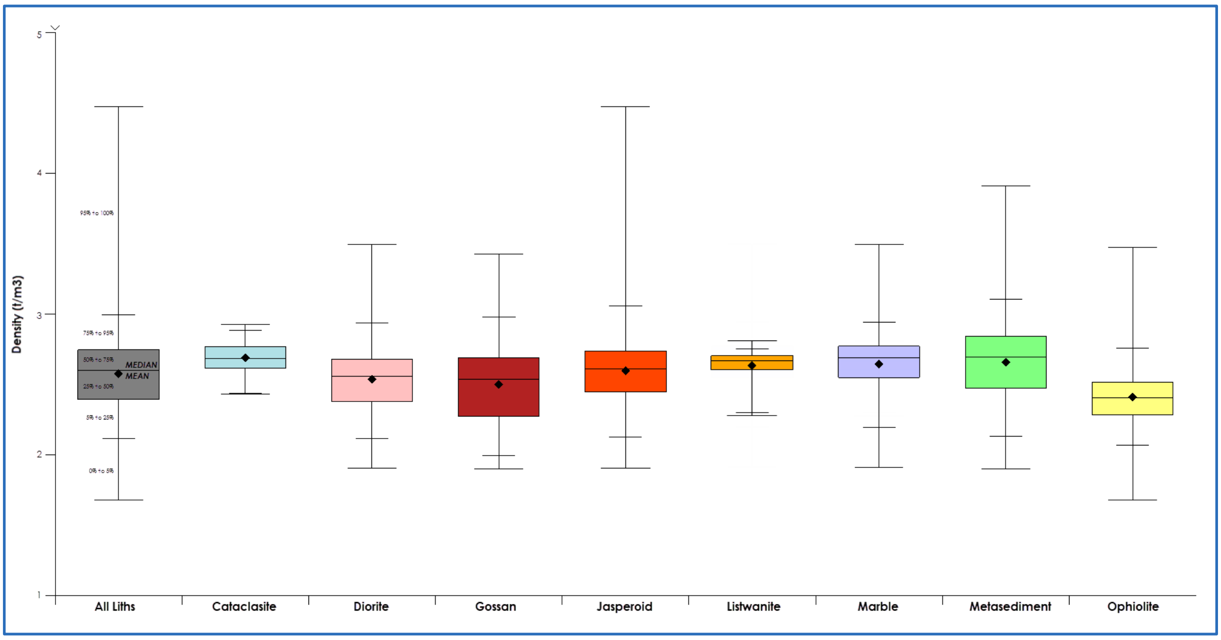
A NN estimate was completed for all variables using the same composites, same domains, same search ranges and same top cut values as the ID3 estimates.

The resulting NN model was used for estimation validation to detect potential estimation bias by domain.

14.2.10 Çakmaktepe Density Model

Density measurements were collected on DD core samples spaced nominally 3 m apart down-hole. Samples were wax-coated when necessary to reduce the influence of porosity and void space. Density values were statistically analysed by lithology with outliers and non-representative values excluded from the analysis (Figure 14.26).

Figure 14.26 Çakmaktepe Density by Lithology



Modified from SSR Mining, 2020.

A review of histograms of density within each rock type aided in the selection of bottom and top cut values.

Selected lower and upper cut density values by lithology are shown in Table 14.29.

Table 14.29 Çakmaktepe Bottom and Top Density Cut Values

Lithology	No. of Density Data	Bottom Cut	Top Cut
Cataclasite	33	2.60	2.62
Diorite	1,496	2.00	3.00
Gossan	407	2.00	2.90
Jasperoid	1,972	2.00	3.20
Listwanite	29	2.28	2.80
Marble	4,041	1.91	3.50
Metasediment	4,114	2.00	3.30
Ophiolite	3,400	2.00	3.00

Çakmaktepe North is drilled predominately with DD core holes and shows good spatial coverage of density samples. The Central area is drilled almost exclusively with RC holes containing very few density samples and limited sample coverage. The East area has close-spaced density sampling throughout. Southeast has limited density sampling. The limited density sampling in Central and Southeast prevented the estimation of density values across Çakmaktepe.

14.2.11 Çakmaktepe Resource Classification

Grade estimates were classified using the following SSR Mining guidelines:

- Indicated Mineral Resource should be quantified within relative ±15% with 90% confidence on an annual basis, and
- Measured Mineral Resources should be known within ±15% with 90% confidence on a quarterly basis.

Several methods were explored to classify model estimates within the Çakmaktepe model. This included classification by sample spacing, grade shell modelling, and comparisons to older methods based on a drill spacing study.

In 2017, a drill spacing study was completed to classify each deposit using confidence limits. Confidence limits were calculated on a single mass that represents the average of one month's heap leach production (185,000 t/month or 2.2 Mt/a) from Çakmaktepe. A relative ±15% with 90% confidence was used to identify Indicated volumes. This resulted in using model cells satisfying sample spacing from 12–22 m as Indicated. Indicated model cells were only assigned within modelled grades shells for Au and Cu.

Based on mine reconciliation from the Central and East areas, more gold ounces were produced than the 2017 resource model predicted. Reasons for the variances included the low bias of exploration drilling and the restrictive classification method. Conservative model classification placed a high percentage of gold ounces in the Inferred category, despite the deposit being drilled on a regular 20 m x 25 m grid.

A test of classification by sample spacing was run on the Çakmaktepe drillhole dataset. The sample spacing method involved assignment of Inferred to cells having an average distance to the first two samples (from different holes) to the cell centroid of 35 m. Model cells were assigned to the Indicated category if the average sample spacing was 20 m or less. After assignment of classification by sample spacing and by the 2017 method, mineralised material above 0.5 g/t Au was compared by deposit area. The comparison showed, the sample spacing method:

- Increased classified ounces for all deposits by 5% (Indicated and Inferred).
- Re-classified Inferred ounces to Indicated. This change was drastic in some areas with more than 70% of the Inferred blocks in the North area being re-classified as Indicated.
- Did not include a parameter for grade continuity causing a high percentage of material above 0.5 g/t Au to be assigned to the Indicated category.

The sample spacing method was generous towards Indicated classification and did not adhere to the notion of Indicated mineralisation having verified grade continuity.

To include grade continuity based on geological evidence into the classification parameters, only cells within modelled grades shells (Au, Cu, and Ag) were assigned to the Indicated category. This segregated mineralisation based on model support.

The classification method used in this model combined sample spacing with modelled shapes. First, cells with an average sample spacing of 35 m were assigned Inferred. Then cells within the modelled grade shells having a sample spacing of 20 m or less were given an Indicated classification.

No Çakmaktepe estimates were classified as Measured.

Inspection of the geological model and gold mineralisation continuity within each geological unit in Southeast shows poor geological continuity and limited Au grade continuity. Therefore, all cells in the Southeast deposit were set to Inferred, despite the 25 m x 25 m drillhole spacing.

In summary, assignment of model classification followed these steps:

- Sample spacing was calculated based on samples from drillholes containing assay values. The calculation of sample spacing did not use limiting boundaries such as domains or lithological shapes.
- Inferred and indicated classification was assigned on the basis of drill sample distances (20 m and 35 m).
- Indicated classification was then restricted to those cells within the modelled mineral grade shells for gold, copper, and silver.
- Southeast estimates were set to Inferred.

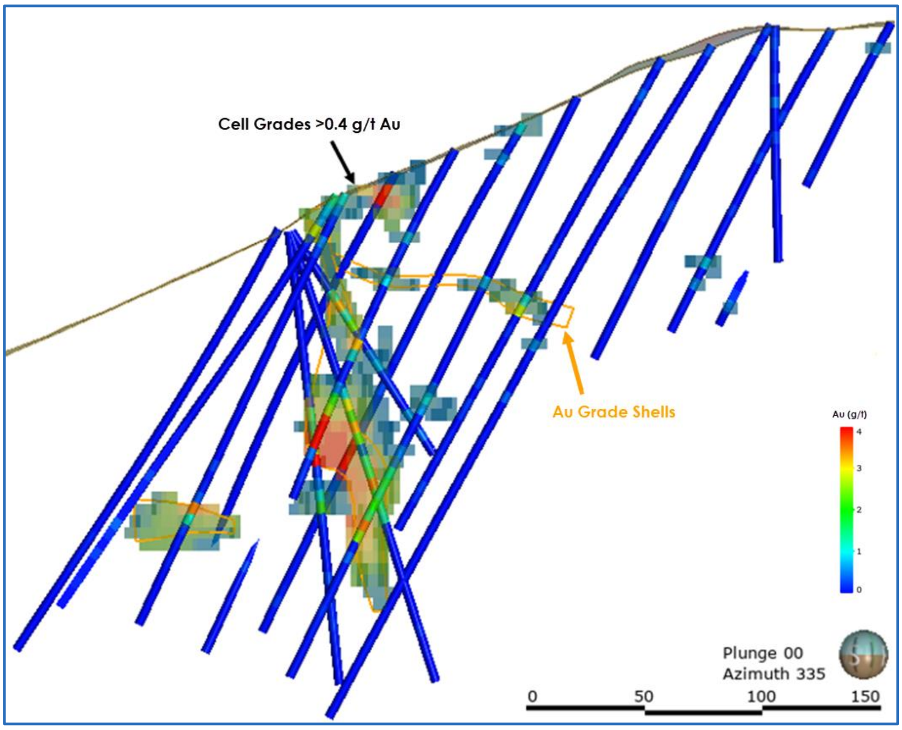
14.2.12 Çakmaktepe Model Validation

Validation of the 5 m x 5 m x 5 m model estimates included visual inspection of grade estimates, comparisons of cell grades to drillhole data, checks for global bias, check of local bias (swath plot), metal reduction calculation, and comparison of estimates within the Central and East areas using grade tonnage curves.

14.2.12.1 Çakmaktepe Visual Inspection

Visual inspection of plans and sections and 3D visualisation confirmed that the cell model estimates honour the drillhole data and grade shell boundaries. An example cross-section and bench-section are presented in Figure 14.27 and Figure 14.28, respectively. These figures illustrate the spatial distribution of Au grades and their relationship to grade shells.

Figure 14.27 Çakmaktepe North Schematic Section with Drilling and Au Model Estimates

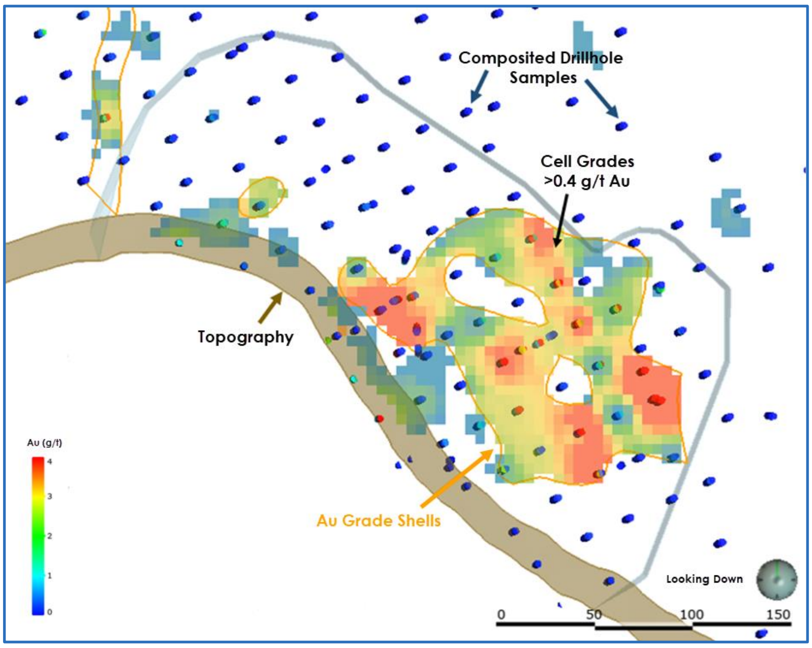


SSR Mining, 2020

Grade estimates within the grade shells were visually confirmed by comparing the grade of the cell with the grade shell boundary. Higher grades exist inside the grade shell with a drop in grade tenor evident when crossing the grade shell boundary. Grade shells follow geological features such as lithological contacts and the Çakmaktepe North shear structure. Estimates outside of the grade shells were set to generalised orientations honouring the trends of the low-grade mineralisation and orientations of the major lithological units.

Hard grade boundaries were used for gold, silver, and copper. The sharp changes in grade are expected, rather than being an artefact of the estimate, due to the close relationship between mineralisation and structural features. This relationship is supported by close-spaced drilling throughout Çakmaktepe and crossing holes in areas such as the shear zone in Çakmaktepe North.

Figure 14.28 Çakmaktepe Central Bench Plan with Drilling and Au Model Estimates



SSR Mining, 2020

14.2.12.2 Çakmaktepe Global Bias

The cell model was checked for global bias by comparing the mean Au, Ag, and Cu grades at a zero cut-off from the ID3 model with means from NN estimates for Indicated and Inferred estimates.

Estimates were calculated within the grade shells to eliminate the influence of high-volume, marginal material on the estimation performance. The NN estimator produces a theoretically unbiased (de-clustered) estimate of the average value when no cut-off grade is imposed and provides a basis for checking the performance of different estimation methods. In general, an estimate is considered acceptable if the bias is less than 5%.

The global bias analysis is shown in Table 14.30. Domain 4 is the northern-most domain containing only a few samples from the North area. There are few modelled tonnes in this domain. There are no modelled gold veins in domain 4 with only a small portion of a copper grade shell in this domain. The NN estimate indicates the ID3 estimate in this domain may be bias high.

Table 14.30 Çakmaktepe Global Bias by Domain

Location	Domain	Element	ID3 Estimates	NN Estimates	Rel. Diff. (%)
Çakmaktepe North	1	Au	1.604	1.688	–5.0
		Ag	12.130	13.457	–9.9
		Cu	0.187	0.174	7.8
	2	Au	1.298	1.325	–2.0
		Ag	11.856	11.710	1.3
		Cu	0.196	0.201	–2.4
	3	Au	0.001	0.001	0.0
		Ag	10.932	11.381	–3.9
		Cu	0.231	0.239	–3.3
	4	Au	0.001	0.001	0.0
		Ag	15.193	12.831	18.4
		Cu	0.407	0.371	9.7
	5	Au	0.748	0.764	–2.2
		Ag	15.112	18.032	–16.2
		Cu	0.260	0.248	5.1
Çakmaktepe Central	6	Au	0.860	0.833	3.3
		Ag	15.112	18.032	–16.2
		Cu	0.150	0.148	1.5
	9	Au	1.509	1.450	4.1
		Ag	10.632	11.165	–4.8
		Cu	0.327	0.306	6.9
Çakmaktepe East	7	Au	0.820	0.793	3.3
		Ag	15.150	14.226	6.5
		Cu	0.344	0.316	8.8
Çakmaktepe Southeast	8	Au	0.917	0.961	–4.6
		Ag	10.854	10.827	0.3
		Cu	0.279	0.269	4.0
All	**Total**	Au	1.178	1.184	–0.5
		Ag	12.552	12.910	–2.8
		Cu	0.261	0.251	4.4

14.2.12.3 Çakmaktepe Local Bias

Local trends in the grade estimates (also known as drift analysis) were assessed by plotting the mean values from the ID3 estimate versus the NN results for Indicated model cells in east–west, north–south and vertical directions (swath plots).

Swath plots were constructed by project area (North, Central, East) to ensure that local variability would not be lost due to different mineralised zones having the same easting or northing. Southeast contains only Inferred material and no swath plots were generated.

Although the global comparisons agree well, the swath plots illustrate the existence of slight local differences between the NN and ID3 model grades. Most of these variances occur in the peaks and valleys of high-grade and low-grade when encountering gaps in mineralisation.

There is good correspondence between the ID3 and the NN estimate in North and Central. The close comparison to each estimate is seen even at model edges where data density is limited. The close comparison in estimates is primarily due to grade shell domaining restricting the selection of samples for the estimate.

Swath plots for the Au estimates in Çakmaktepe Central are shown in Figure 14.29, Figure 14.30 and Figure 14.31.

Figure 14.29 Çakmaktepe Central Au Swath Plot by Easting

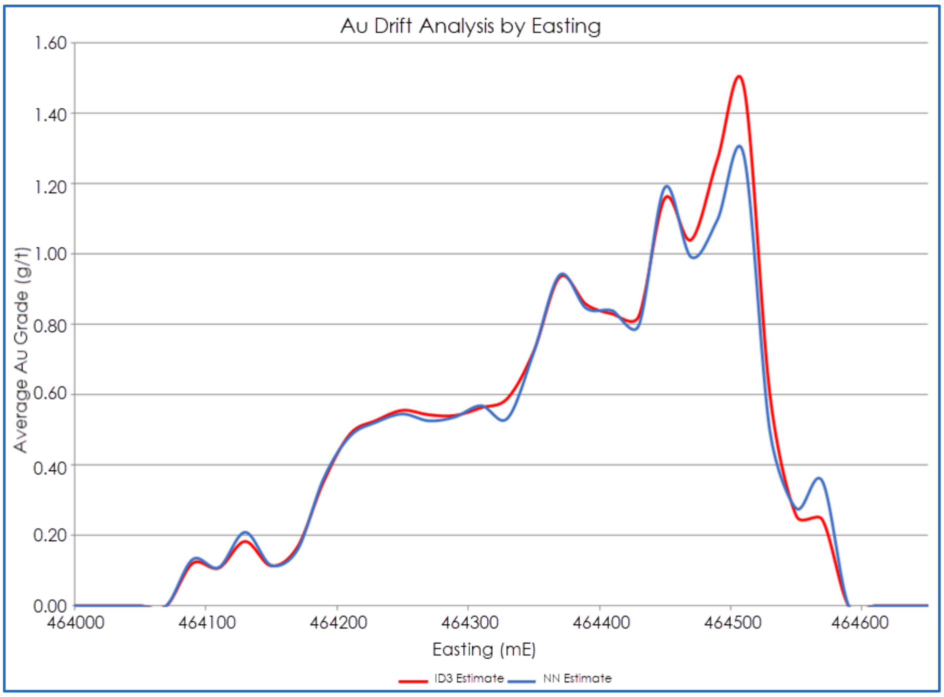


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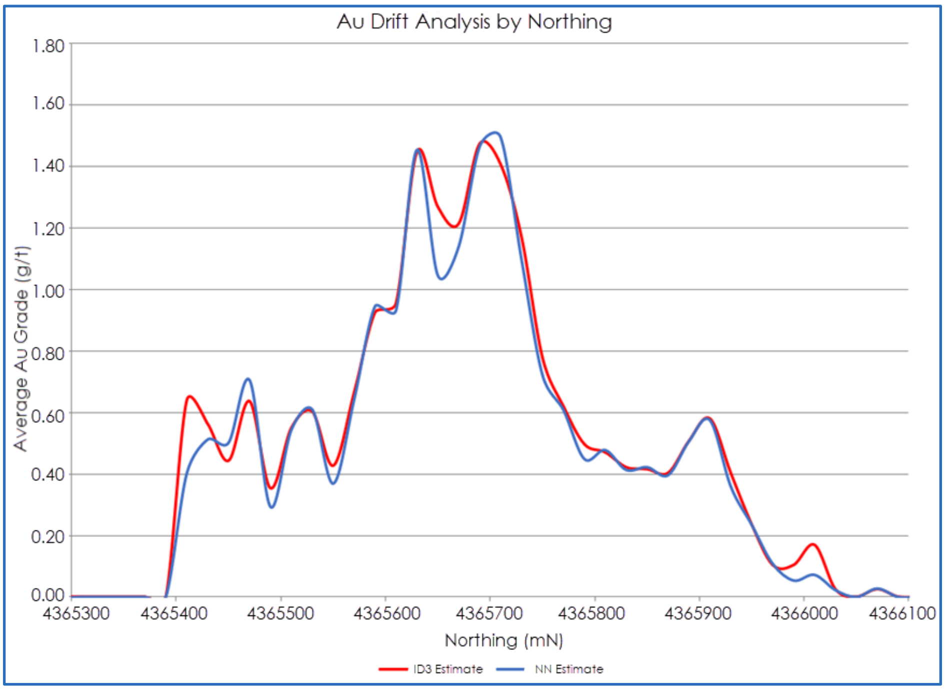
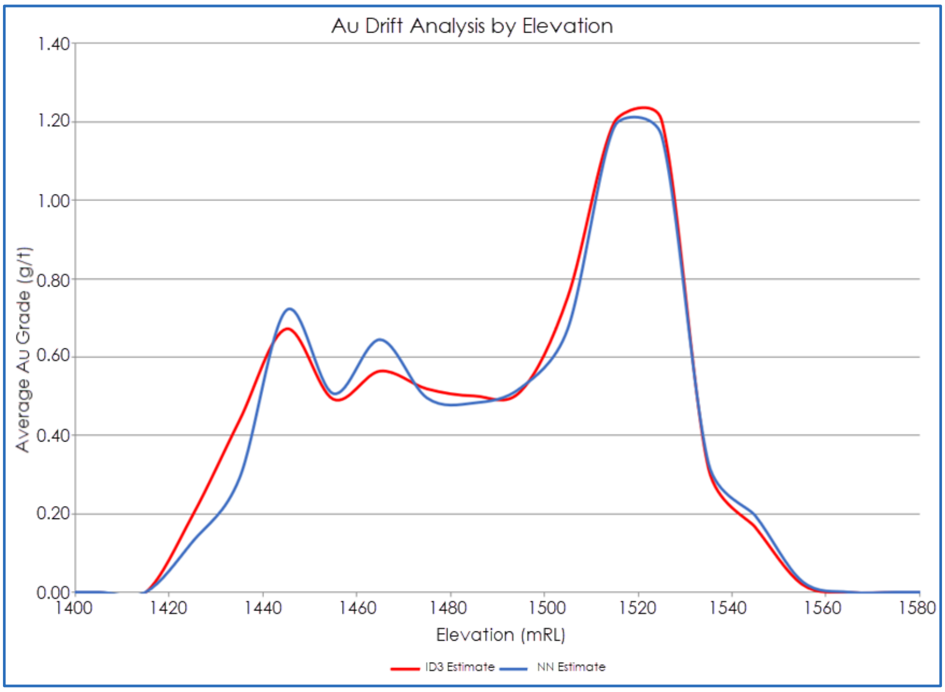
Figure 14.30 Çakmaktepe Central Au Swath Plot by Northing



SSR Mining, 2020

Figure 14.31 Çakmaktepe Central Au Swath Plot by Elevation



SSR Mining, 2020

14.2.13 Çakmaktepe Comparison to Production Data

Mining occurred in the Çakmaktepe Central and East pits, primarily during 2019. Blast hole data from these two pits were used to construct an Au grade estimate for comparison of the production model to the Mineral Resource model.

The production model, using blast hole assay data, was set up to follow the same parent cell size used in the resource model of 5 m x 5 m x 5 m. This generates cell centroids with the same centroid coordinates as the resource model for relational comparisons by cell.

Plotting the grade / tonnage curve for Au shows the number of tonnes to be similar in both models, with a crossover of the resource to production model tonnes occurring between the 0.8–2.8 g/t Au cut-offs. A large variance is seen when comparing Au grades between the two models. The increased grade in the production model results in more gold ounces. The largest positive and negative variances between the two models were investigated. The following observations were made:

- Estimate variances exist throughout the two cell models. An overall bias towards higher grade blast holes results in higher cell grades in the production model.
- Comparison of cut-off grades shows a larger variance in gold ounces between the two models as the cut-off grade is increased. Variances were plotted on a grade / tonnage curve by pit for a comparison of gold ounces by area.
- Variances were not limited to specific locations. Positive and negative variances were mixed throughout the Central and East pit. This suggests the selected modelling method for the resource grade estimation is not bias high or low, but likely producing a gold model more generalised than the variability seen within the deposit.
- When using the ID3 interpolation method, cell grades closely match drillhole composite values. Investigation of areas where exploration drilling crosses cells shows lower estimated grades in the resource model and higher estimated grades in the production model. This illustrates the variance in the two drillhole datasets – exploration to blast hole data.

These observations indicate that the variances between the two datasets are likely greater than the software tools available to match the deposit grade distribution and short-range variability to the resource model. To compensate for the model variances, increasing the exploration drill density to the deposit variability is preferred. However, increasing the exploration drill density is probably not feasible due to the high inherent variance seen in the deposit. This presents a risk that mining may not match the predictive abilities of the resource model using the available exploration data.

14.2.14 Çakmaktepe Assessment of Reasonable Prospects of Eventual Economic Extraction

Mineral Resource estimates were shown to meet reasonable prospects for eventual economic extraction criteria by reporting only material that was contained within a conceptual pit shell using metal prices of $1,750/oz for gold with the parameters summarised in Table 14.31. These parameters, with the exception of the gold price, are the same parameters as those used to define the Mineral Reserve pit.

Table 14.31 Summary of Key Parameters Used in Conceptual Pit Shell at Çakmaktepe

Description	Unit	Minimum	Maximum
Heap Leach Gold Recovery	%	38.0	80.0
Mining Cost per tonne mined	$/t	1.59	1.59
Process Costs Heap Leach	$/t	14.16	14.16
Site Support per tonne processed	$/t	3.17	3.17
Internal Au Cut-off – Heap Leach	g/t	0.36	0.76
Royalty	%	4.0	4.0

14.2.15 Çakmaktepe Mineral Resource Tabulation

Çakmaktepe Mineral Resources are reported inclusive of Mineral Reserves and have been tabulated by resource classification and oxidation state in Table 14.32. Mineral Resources are presented on a 100% basis.

Table 14.32 Çakmaktepe Mineral Resource Table by Classification and Oxide State

Mineral Resource Estimate for the Çakmaktepe Deposit (as at the Effective Date)				
Material Type	**Resource Category Material**	**Tonnes (kt)**	**Au (g/t)**	**Contained Gold (koz)**
Oxide	Indicated	3,615	1.53	178
	Indicated – Stockpile	11	2.69	1
	Total Indicated	**3,626**	**1.53**	**179**
	Inferred	1,205	0.85	33

1. Mineral Resources have an effective date of 27 November 2020.
2. Mineral Resources are reported inclusive of Mineral Reserves; Mineral Resources that are not Mineral Reserves have not demonstrated economic viability.
3. Mineral Resources are shown on a 100% basis. of which SSR Mining owns 50%.
4. Oxide is defined as material with <2% total sulfur. Internal categorisation of oxide to low-sulfur (LS) and high-sulfur (HS) does occur for mine planning purposes based on a 1% total sulfur threshold. There is no sulfide Mineral Resource at Çakmaktepe.
5. All Mineral Resources in the CDMP20 were assessed for reasonable prospects for eventual economic extraction by reporting only material that fell within conceptual pit shells based on metal prices of $1,750/oz for gold. The following parameters were used: metallurgical recoveries in oxide: 38.0%–80.0%; Au cut-off grades in oxide: 0.36–0.76 g/t Au, (there are no credits for Ag or Cu in the cut-off grade calculations); allowances have been made for royalty payable.
6. Reported Mineral Resources contain no allowances for unplanned dilution, or mining recovery. Tonnage and grade measurements are in metric units. Contained gold is reported in troy ounces.
   Tonnages are rounded to the nearest thousand tonnes; grades are rounded to two decimal places. As a result, totals may not match.

14.2.16 Çakmaktepe Metal Reduction

The effective amount of metal removed by outlier restriction can be evaluated by comparing the ID3 capped model to an uncapped ID3 model. An assessment of this effective amount by domain is presented in Table 14.33. The amount of metal removed was evaluated at a 0.5 g/t Au cut-off for Measured, Indicated and Inferred model cells to allow a comparison of each domain. Au grade capping reduced the overall mean grade of the estimate by 0.8%. The largest variance is domain 8 which corresponds to the Southeast deposit with high grade variability.

Table 14.33 Çakmaktepe Quantity of Gold Metal Removed

Domain	Tonnage (kt)	Uncapped ID3 Mean	Capped ID3 Mean	Variance
1	1,655	1.168	1.155	–1.1%
2	4,811	1.244	1.240	–0.3%
3	628	0.760	0.760	0.0%
4	4,878	0.830	0.830	0.0%
5	1,077	1.089	1.089	0.0%
6	513	0.948	0.948	0.0%
7	1,201	0.953	0.951	–0.2%
8	990	1.079	1.023	–5.2%
9	3,053	1.355	1.332	–1.7%
Total	**18,807**		**Average**	**–0.8%**

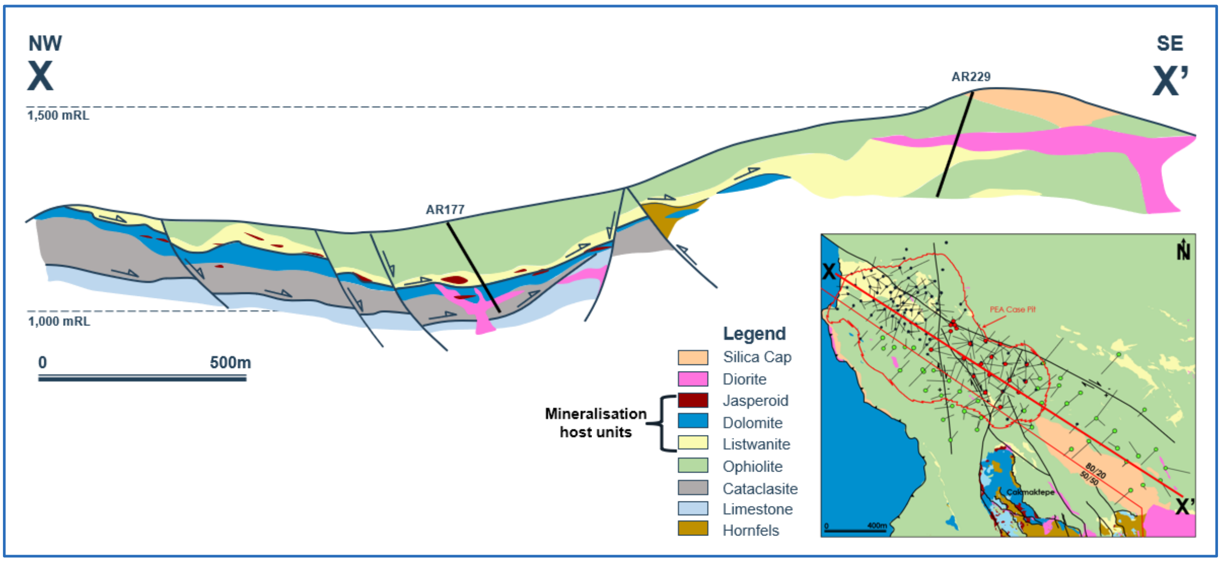
14.3 Ardich

The latest mineral asset to be intensively studied in the Çöpler district suite of mineralised zones is Ardich, which is located approximately 6 km east of the current Çöpler pit and 1 km north of the Çakmaktepe pits. The Ardich deposit is accessed by the İliç-Yakuplu village road, which is open throughout the year.

Ardich mineralisation was discovered in August 2017. Ardich does not appear to have hosted historical mining or trenching in the way that Çöpler and Çakmaktepe have.

The Ardich deposit is a listwanite-dolomite hosted gold replacement deposit with mineralisation occurring along fault zones between listwanite, ophiolites, hornfels, dolomites, and limestones (Figure 14.32). Mineralisation and alteration extend along a north-west trend, parallel to major structures controlling both mineralisation and block rotations. Au grades increase at dolomite-listwanite contacts and within silica-rich listwanites. The mineralisation is predominantly oxide, with sulfide mineralisation confined to pyrite-rich jasperoid zones. Based on available drillhole data, the main mineralised zone appears to be tabular and almost flat lying.

Figure 14.32 Ardich Geology Schematic (long-section)



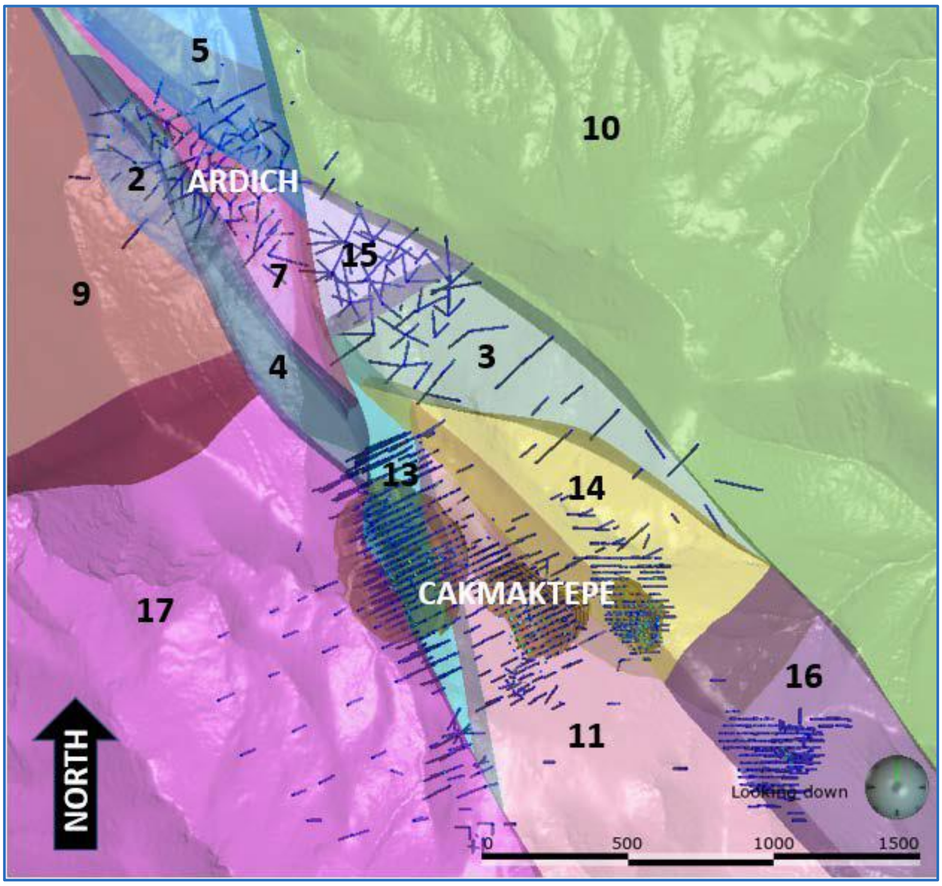
SSR Mining, 2020
See Figure 10.4 for larger image of collar plot

14.3.1 Ardich Domains

At Ardich, The Mineral Resource estimate was based on a 3D geological solids model developed within constraining fault blocks (Figure 14.33). Lithologies are offset by faults that create rotated blocks that have moved vertically relative to each other as well as pivoted / rotated within their own boundaries. High-angle faults cross-cut the deposit with several low-angle structures carrying mineralisation along the dolomite-listwanite contacts. Mineralised trends follow the orientations of the structural features and lithological contacts as they change within the fault blocks. Domains for Ardich are defined by these fault blocks.

Mineralised zones often exist along the boundary between two lithologies, rather than being hosted entirely within a single lithology. Gold grade shells were constructed to allow estimation to honour mineralised zones instead of being bound by lithological domains. This allows the grade estimation process to use samples on both sides of the lithological contact to estimate cell grades.

Figure 14.33 Ardich Model Domains in Plan View



14.3.2 Ardich Geological Model

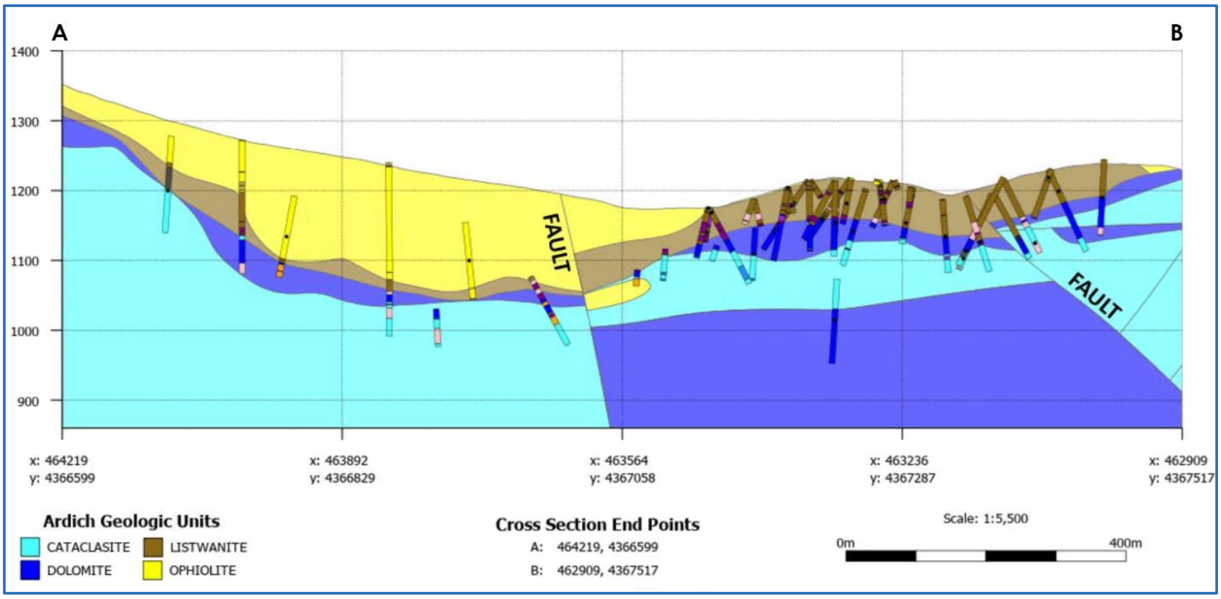
A 3D solids model was generated using drillhole logging and surface mapping (Figure 14.34). Logging was given priority, with surface mapping used primarily to help define where lithological boundaries daylight at the surface.

Fan drilling from surface provided information on geological features from several crossing angles. Sample spacing varied depending upon the depth of the feature encountered in the drillhole.

Interpreted faults are used as bounding limits to create fault blocks. The fault blocks are then used as domains for statistical reporting and the application of estimation parameters during the grade estimate. Some fault blocks were large and needed to be further subdivided to generate a better local grade estimate. Faults were interpreted based on the relative offset of lithological units in the drilling. Fault locations are loosely tied to changes seen in lithological mapping at surface. Each fault block contains a lithological package independent of adjacent fault blocks. The stacked lithological contacts within each fault block show a range of dip angles from near-horizontal to dips of up to 45°.

Jasperoid intercepts were grouped into sub-units for vein modelling. Jasperoid is an alteration product, but at Ardich is treated as an individual rock type for geological modelling, sulfur and carbon estimates, and metallurgical assessment.

Figure 14.34 Ardich Lithology Model, (looking south-west)



SSR Mining, 2020

14.3.3 Ardich Data Summary

Exploration drilling at Ardich utilised surface PQ and HQ triple-tube diamond core drilling. No RC drilling has occurred to date at Ardich.

A drillhole dataset for Ardich was obtained in MS Access format on 13 February 2020. Drillhole data was reviewed for transcription errors, sample interval overlaps and gaps, collar location with respect to topography, and geological logging consistency. The dataset contained a total of 233 Ardich holes ('AR' series of holes) that were geologically logged and had assay results. The total drilled metres for the Ardich dataset equalled 43,411.7 m. The dataset also included Çakmaktepe holes due to the close proximity of the two deposits.

Assayed intervals below the laboratory detection limit are stored as half the laboratory detection limit.

Data errors were recorded in an MS Excel spreadsheet and sent to the project geologist for checks and correction of the master Datashed database.

14.3.4 Ardich Exploratory Data Analysis

14.3.4.1 Ardich Drillhole Compositing

Original sample lengths are predominately 1 m in length. Of the total 39,956 samples from the AR drillhole series, 1,048 samples are less than 1 m and 1,583 samples are greater than 1.5 m. Sample length is based on manual selection by the logging geologist according to variabilities seen in the core, such as lithological contacts and alteration. Ardich does not contain consistent visual indicators of grade to decide sample breaks by metal content. Sample length is not correlated to Au grade.

Composite samples of 5 m in length were used for statistical analysis, construction of grade shells and grade estimation. During compositing, missing data is denoted as –99 and excluded from the composite calculation. Composites do not truncate at geological boundaries. End (or tail) composites of <5 m are retained and length-weighted during the grade estimate.

The mean grade for un-composited intervals was 0.33 g/t Au while the composited mean grade was 0.32 g/t Au. The maximum grade sample of 30.3 g/t Au (1 m interval) was composited into a 5 m interval of 13.49 g/t Au.

Contacts for lithological shapes used the raw logged interval depth in 3D space. Surfaces were generated through implicit modelling of contact locations in the drillholes. Adjustments are made in some cases to manage detailed logging and minimise the number of small shapes generated. This was done where:

- Segments denoted as ignored (no samples, core loss, cavity, overburden) were converted to the surrounding lithology when the segment was shorter than 3.0 m.
- Interior and exterior segments shorter than 3.0 m are simplified by filtering out intervals during the lithological solid generation. These short logged intervals remain within the drillhole database and are used in the modelling.

14.3.4.2 Ardich Summary Statistics

Table 14.34 summarises the univariate statistics calculated for each element considered for modelling within Ardich. The CV, (standard deviation divided by mean), is a measure of relative dispersion of the grade distribution.

Mean grades tend to be low since they are not reported here by specific domains. Ag and Cu grades are low throughout the deposit and were not estimated in the resource model.

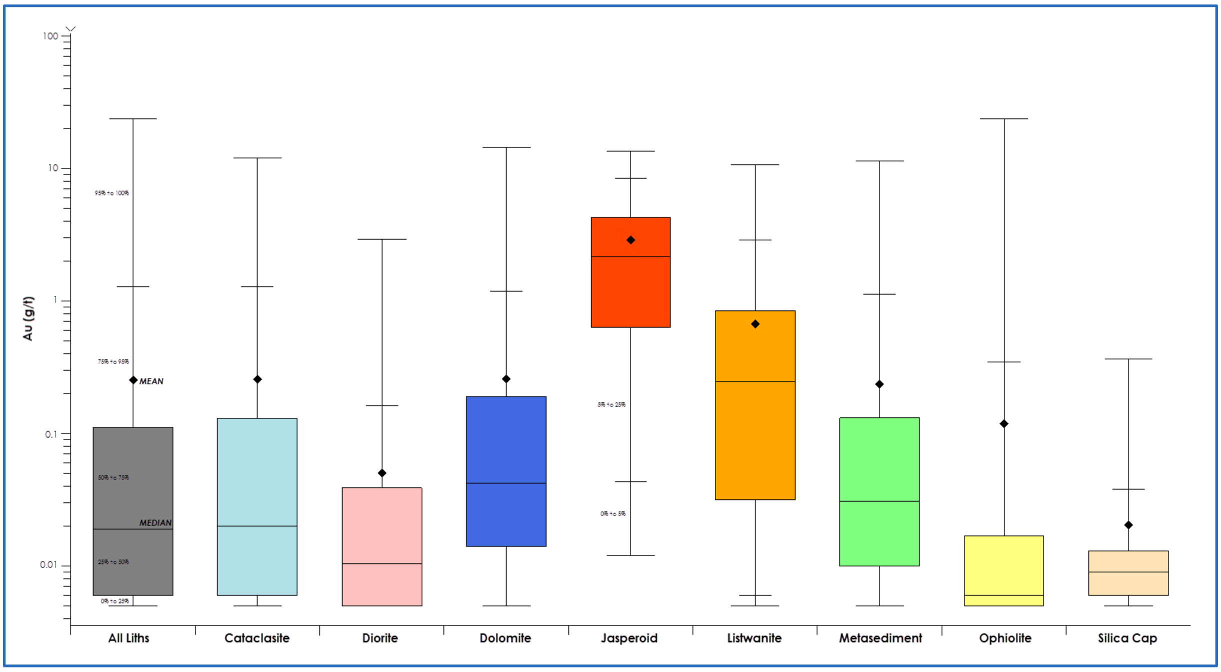
Table 14.34 Ardich Key Element Univariate Statistics (based on 5 m composites)

Metal	Count	Min.	Max.	Mean	Std. Dev.	Variance	CV
Au (ppm)	8,512	0.005	13.49	0.32	0.97	0.94	3.07
Ag (ppm)	8,512	0.250	30.70	0.67	1.37	1.86	2.05
Cu (%)	8,512	0.0001	7.483	0.01	0.08	0.01	16.09
S (%)	8,512	0.005	12.70	0.59	1.08	1.17	1.83
C (%)	8,067	0.009	13.11	3.45	3.65	13.34	1.06

14.3.4.3 Ardich Box Plots

Box plots were created to facilitate comparisons of metal grades between lithologies and domains. Box plots for Au in the Ardich 5 m composites is shown in Figure 14.35.

Figure 14.35 Ardich Box Plot of 5 m Au Composites by Lithology

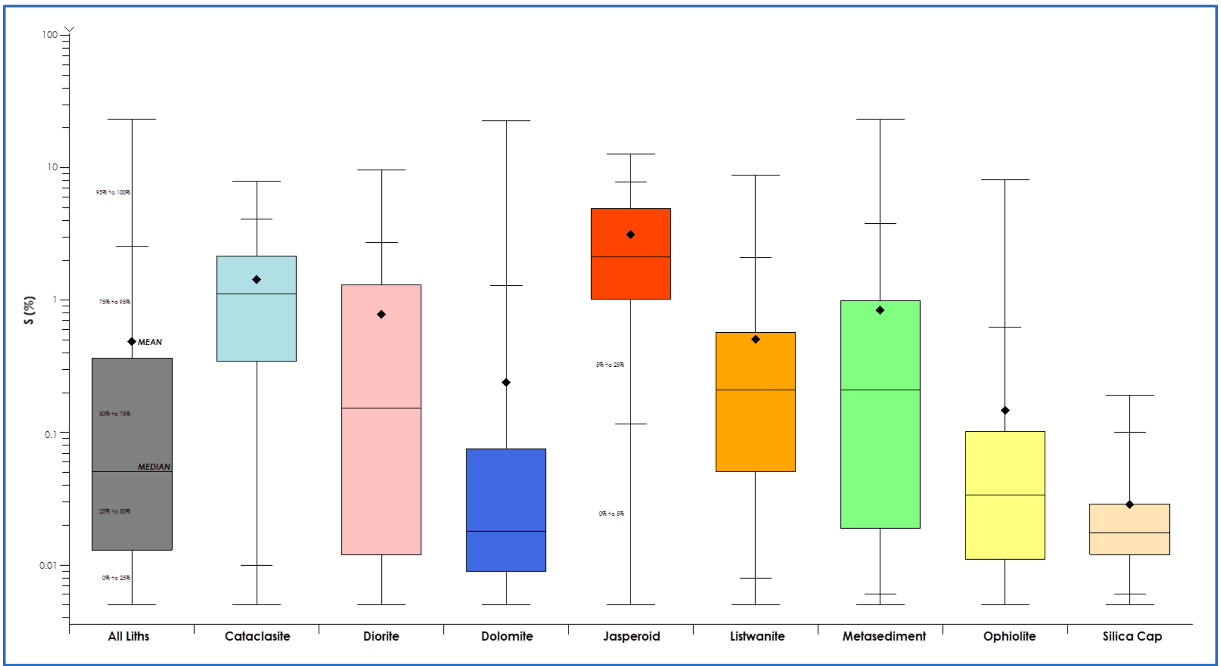


Modified from SSR Mining, 2020

The box plot confirms the lithological differences, with jasperoid and listwanite containing higher Au grades and the remaining units (cataclasite, diorite, dolomite, ophiolite) having lower Au grades.

Box plots of sulfur (Figure 14.36) show higher sulfur content in jasperoid and cataclasite with lower sulfur grades in listwanite, dolomite and ophiolite. Low sulfur is consistently seen in ophiolite. For this reason, the sulfur estimate uses lithological contacts as domain boundaries.

Figure 14.36 Ardich Box Plot of 5 m Sulfur Composites by Lithology



Modified from SSR Mining, 2020

14.3.4.4 Ardich Correlation Coefficients

Mineralisation trends at Ardich spatially follow structural and lithological contacts. The strongest association is Au and Ag, with reasonable correlation between Au and As (Table 14.35). Cu tends to be too low in grade to correlate with the other metals. Elevated Ag grades are seen in the same orientations as the lithological contacts but don't track to Au or Cu grade intensity (high to low grades). Higher grade Ag occurs along the gold grade shells and at structural intersections. Sulfur is closely associated with lithological units with very limited correlation to metal content.

Table 14.35 Ardich Correlation Coefficients using 5 m Composites

	AuFA (g/t)	Ag (g/t)	Cu (%)	As (ppm)	S (%)	C (%)
AuFA (g/t)	**1.00**					
Ag (g/t)	0.67	**1.00**				
Cu (%)	0.01	0.18	**1.00**			
As (ppm)	0.53	0.40	0.01	**1.00**		
S (%)	0.28	0.28	0.09	0.35	**1.00**	
C (%)	0.06	0.03	0.02	0.04	–0.09	**1.00**

14.3.5 Ardich Core Recovery

Exploration drilling at Ardich utilised surface PQ and HQ triple-tube diamond core drilling. Overall, Ardich drill core recovery is very good with a mean recovery over 92%. Review of the core photographs supports the high recovery percentage. Some variation in the number of core fractures was observed in different locations. It was also noted from core photos that more highly fractured ground was associated with the gold mineralised zone. More competent material is encountered external to the main orebody.

Raw interval lengths were compared to core recovery collected during geotechnical logging. No correlation is seen between Au grade and core recovery.

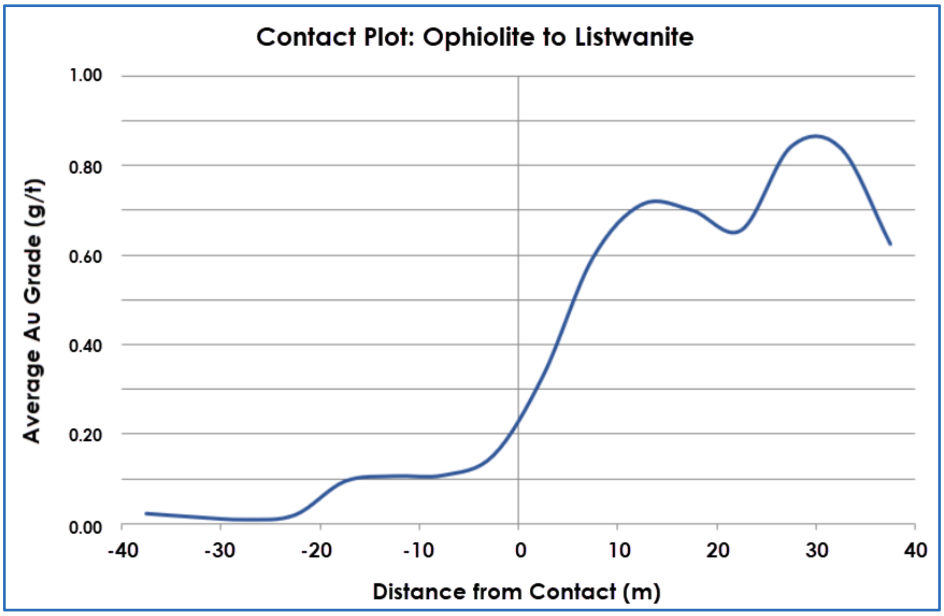
14.3.6 Ardich Twin Holes

There have been no twin holes drilled at Ardich.

14.3.7 Ardich Contact Plots

Contact plots were created between each of the domains to show how grades change across lithological boundaries. Jasperoid and listwanite are favourable hosts to mineralisation and show changes in grade when adjacent to other lithologies (ophiolite, dolomite, cataclasite). An example of a contact between ophiolite and listwanite is shown in Figure 14.37.

Figure 14.37 Ardich Contact Plot of Au Across the Ophiolite / Listwanite Contact



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14.3.8 Ardich Top Cutting

An analysis of the need for top cutting grades at Ardich was undertaken. Top cuts were selected for Au, S, and C based on the log probability plot, supported by the projection of the expected upper grade using the grade curves.

Top cuts selected are shown in Table 14.36.

Table 14.36 Ardich Top Cuts for Au, S, and C

Element	Ardich Area / Domain	Top Cut Grade	No. Samples Cut
Au (g/t)	External	5.0	2
	Mineralised (1–9)	12.0	4
S (%)	Dolomite	5.0	6
	Ophiolite	3.5	5
	Cataclasite	7.0	2
	Listwanite	4.5	6
	Jasperoid	9.0	5
	Diorite	15.0	–
	Metasediment	16.0	–
	Silica Cap	5.0	0
C (%)	Dolomite	13.0	1
	Ophiolite	6.0	77
	Cataclasite	10.5	44
	Listwanite	10.0	4
	Jasperoid	5.0	15
	Diorite	8.0	4
	Metasediment	14.0	–
	Silica Cap	6.0	2

Basic statistics were calculated for individual gold mineralised shapes. Mean grades by gold grade shell ranged from 0.80–4.63 g/t Au. The number of samples within each grade shell were limited, with seven of the nine shells containing less than 100 samples. Shells 1 and 7 were the exception, with 353 and 150 samples respectively.

Due to the limited number of samples, all samples within the modelled grade shells were grouped together for top cut evaluation. A single top cut grade was used for all mineralised gold grade shells. Composites external to the grade shells were also evaluated collectively. These two groups form the 'External' and 'Mineralised' top cuts shown in Table 14.36.

Top cuts were implemented after compositing to 5 m. A spatial review of values above 8 g/t Au shows good coverage of higher Au grades across Ardich. The higher Au grades are located along the mineralised contact zone.

High yield limits were used outside of the grade shells to restrict the extrapolation of high grades. For gold, sample values external to the grade shells above 3 g/t Au were restricted to a distance of 15 m x 15 m x 5 m, the size of one parent cell.

14.3.9 Ardich Resource Model Estimation

Gold mineralisation at Ardich is related to lithological contact zones and structural intersections. The mineralised zones tend to be narrow and localised rather than diffuse or disseminated. Mineralised gold shells were developed using composites located along structural and lithological features. In some cases, the grade shell follows the lithological strata within a domain and extends across the interpreted fault to allow estimation of grade along the fault boundary.

Inverse distance method, weighted to the power of three (ID3) was selected as the interpolation method. Weighting the distance to the power of three was selected in preference to the power of two (i.e. ID2) to help limit smoothing and be more responsive to the rapid changes in Au grade across the deposit. A parent cell size of 15 m x 15 m x 5 m was selected. Sub-cells retain the domain-relevant parent cell grades.

14.3.9.1 Ardich Cell Model

The cell model was constructed by first coding according to interpreted lithology shapes, then flagging by each of the grade shells and model domains.

A perimeter solid was used to trim cells outside the shape. Applying such a perimeter reduces the cell model file size and speeds the estimation processing. Care was taken not to exclude any cells that are necessary for subsequent analysis.

The cell model parameters are shown in Table 14.37.

Table 14.37 Ardich Cell Model Prototype Parameters

Direction	Minimum (m)	Maximum (m)	Range (m)	Cell Size (m)	No. of Cells
East	462,700	465,300	2,600	15	520
North	4,365,800	4,367,700	1,900	15	380
RL	850	1,670	820	5	164

To honour lithological shape volumes and boundary edges, a 5 m x 5 m x 5 m sub-cell was used throughout the model.

The model was not rotated.

14.3.9.2 Ardich Estimation Method

Au was interpolated using ID3 and NN using grade shell boundaries for sample selection. Sulfur was interpolated using ID2 and NN within the modelled lithological units. All grade shells and lithological units were treated as hard boundaries, meaning only samples within the shape were used to estimate cells within the same shape. Domains were treated as soft boundaries allowing the selection of samples from nearby domains if located within the search range.

A single search distance of 100 m was used within the gold grade shells to estimate grades. The search range within the mineralised shells is more reliant upon the extents of the gold grade shell instead of the 100 m search distance. The number of samples within the grade shell is limited so a large search distance with a minimum of two samples was used to obtain an estimate. Decreasing the search distance or increasing the minimum samples causes cells within the grade shell not to receive an estimate.

A summary of the estimation parameters are shown in Table 14.38.

Table 14.38 Ardich Estimation Parameters Inside Gold Grade Shells

Gold Grade Shell	Search Orientation			Axis Distance			
	Azimuth	Plunge	Dip	Major	Semi-Major	Minor	Min./Max.
1	145	–6	0	100	100	30	2/10
2	324	–14	0	100	100	30	2/10
3	86	–38	0	100	100	30	2/10
4	75	–32	0	100	100	30	2/10
5	302	–11	0	100	100	30	2/10
6	70	–88	0	50	50	20	2/10
7	32	–45	0	100	100	30	2/10
8	337	–36	0	100	100	30	2/10
9	325	–6	0	100	100	30	2/10

A two-pass method was used to estimate cells outside of the grade shells. Search ranges and sample requirements varied by estimation pass. The first pass only uses samples within 30 m to estimate grades with the second pass extending out to a maximum of 80 m.

The search orientations for cell estimates external to the grade shells were set to the general orientation of the lithological units within each fault block domain. Domains are based on the lithological fault blocks that extend across Ardich.

A summary of the estimation parameters for gold outside of grade shells are shown in Table 14.39.

Table 14.39 Ardich Estimation Parameters for Gold Outside Gold Grade Shells

Gold Grade Shell	Search Orientation			Axis Distance – Pass 1				Axis Distance – Pass 2			
	Azimuth	Plunge	Dip	Major	Semi-Major	Minor	Min./Max.	Major	Semi-Major	Minor	Min./Max.
1	42	–18	0	30	30	10	2/10	70	60	20	2/12
2	84	–27	0	30	30	10	2/10	70	60	20	2/12
3	338	–35	0	30	30	10	2/10	70	60	20	2/12
4	254	–14	0	30	30	10	2/10	70	70	20	2/12
5	26	–42	0	30	30	15	2/10	80	80	30	2/12
6	190	–16	0	30	30	20	2/10	80	80	20	2/12
7	78	–14	0	30	30	15	2/10	80	80	30	2/12
8	30	–52	0	30	30	10	2/10	70	70	20	2/12
9	118	–30	0	30	30	10	2/10	70	70	20	2/12
10	325	–15	0	30	30	10	2/10	70	70	20	2/12
11	56	–20	0	30	30	15	2/10	80	80	30	2/12
12	335	–2	0	30	30	10	2/10	60	60	15	2/12
13	72	–64	0	30	30	10	2/10	70	70	20	2/12
14	90	–2	0	30	30	15	2/10	80	80	30	2/12
15	20	–18	0	30	30	10	2/10	80	80	20	2/12
16	207	–18	0	30	30	10	2/10	60	60	15	2/12
17	350	–14	0	30	30	10	2/10	70	70	20	2/12
Shear	68	–77	0	70	70	30	2/12	–	–	–	–

Sulfur and carbon are directly related to lithological units. A spherical search was used within each lithological shape and interpolated using the ID2 method. A summary of the estimation parameters for sulfur are shown in Table 14.40.

Table 14.40 Ardich Estimation Parameters for Sulfur and Carbon Outside Grade Shells

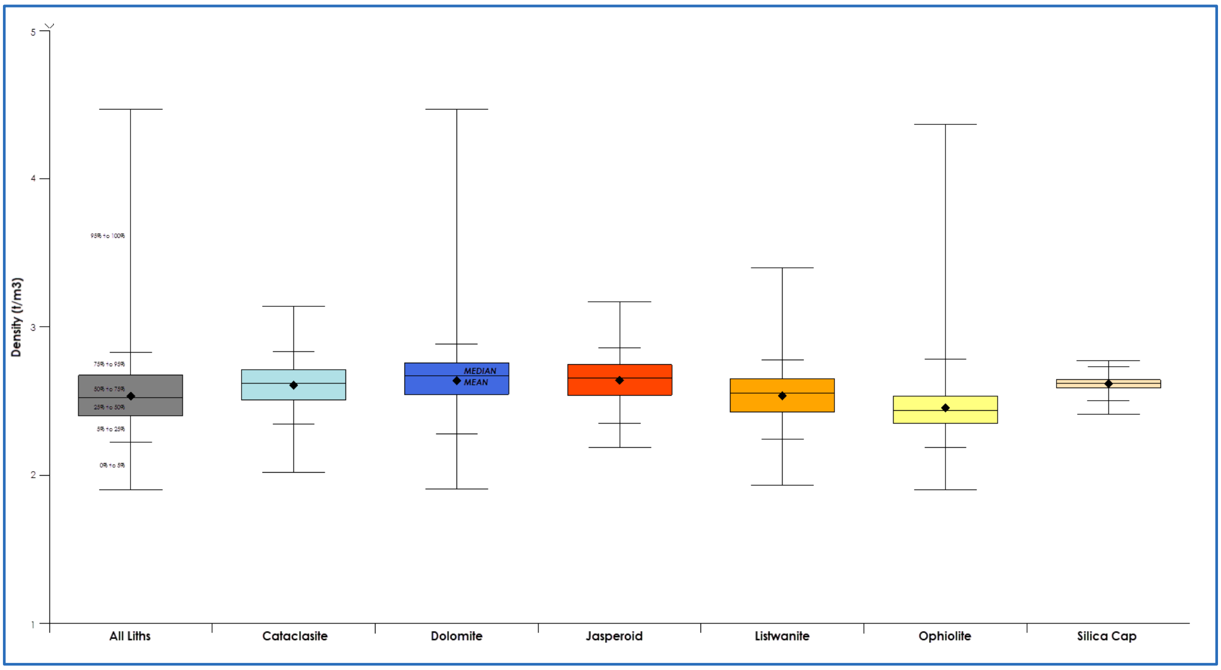
Lithology	Search Orientation			Axis Distance – Pass 1				Axis Distance – Pass 2			
	Azimuth	Plunge	Dip	Major	Semi-Major	Minor	Min./Max.	Major	Semi-Major	Minor	Min./Max.
All	0	0	0	30	30	30	3/12	90	90	90	2/12

A NN estimate was completed for Au and S using the same composites as the ID estimates. The NN estimate also used the same domains, search ranges, and top cut values as the ID3 estimates. The resulting NN model was used for model validation.

14.3.10 Ardich Density Model

Density measurements were collected on DD core samples spaced approximately 3 m apart down-hole. Samples were wax-coated when necessary to reduce the influence of porosity and void space. Density values were statistically analysed by lithology with outliers and non-representative values excluded from the analysis, (Figure 14.38).

Figure 14.38 Ardich Density by Lithology



Modified from SSR Mining, 2020

A review of histograms of density within each rock type aided in the selection of bottom and top cut values. Selected bottom and top cut density values by lithology are shown in Table 14.41

Table 14.41 Ardich Lower and Upper Density Cap Values

Lithology	Bottom Cut	Top Cut	No. of Samples Bottom Cut	No. of Samples Top Cut
Cataclasite	2.2	3.0	2	3
Dolomite	2.0	3.2	16	16
Jasperoid	2.2	3.0	1	2
Listwanite	2.0	3.0	6	4
Ophiolite	2.0	3.2	60	33
Silica Cap	2.5	2.7	3	8

Ardich is drilled entirely with DD core holes showing good spatial coverage of density samples along each drillhole. Continued collection of density samples at close intervals may allow the estimation of density values when drill coverage extends across the deposit.

14.3.11 Ardich Resource Classification

Grade estimates were classified using the following SSR Mining guidelines:

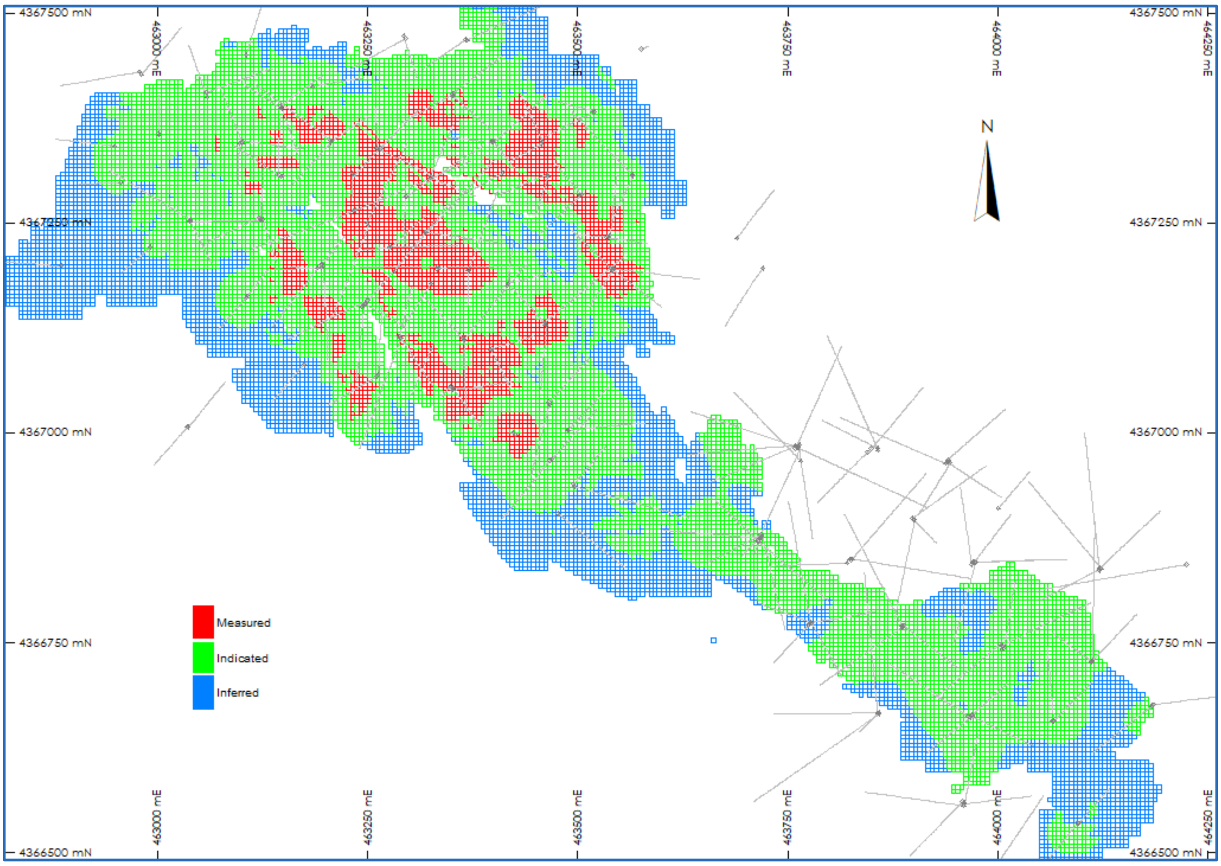
- Indicated Mineral Resource should be quantified within relative ±15% with 90% confidence on an annual basis, and
- Measured Mineral Resources should be known within ±15% with 90% confidence on a quarterly basis.

Mineral Resources were classified based on a drill spacing study and observed continuity of geology and mineralisation.

Drillhole spacing for support of classification of Inferred Mineral Resources could be obtained when sample spacing was within 70 m x 60 m. In domains with adequate drill spacing, 80 m x 80 m was used. For Indicated Mineral Resource classification, the drillhole spacing reduced to a 35 m x 35 m spacing. Appropriate drillhole pattern spacing selection was based on the belief that the mineralisation is structurally controlled, mineral continuity varies within each domain and adequate data quality has been achieved. Gold mineralisation occurs in lenses rather than as a massive homogenous body, thus reducing the confidence in connection of the multiple pod-like mineralised bodies.

The resulting classification shows the substantial portion of the deposit can be classified as Indicated with Inferred cells forming a halo around the Indicated mineralisation and Measured Mineral Resource encased within the Indicated zone, Figure 14.14.

Figure 14.39 Projected Plan View of Ardich Resource Classification



OreWin, 2020
Only model cells with Au >0.3 g/t shown

14.3.12 Ardich Model Validation

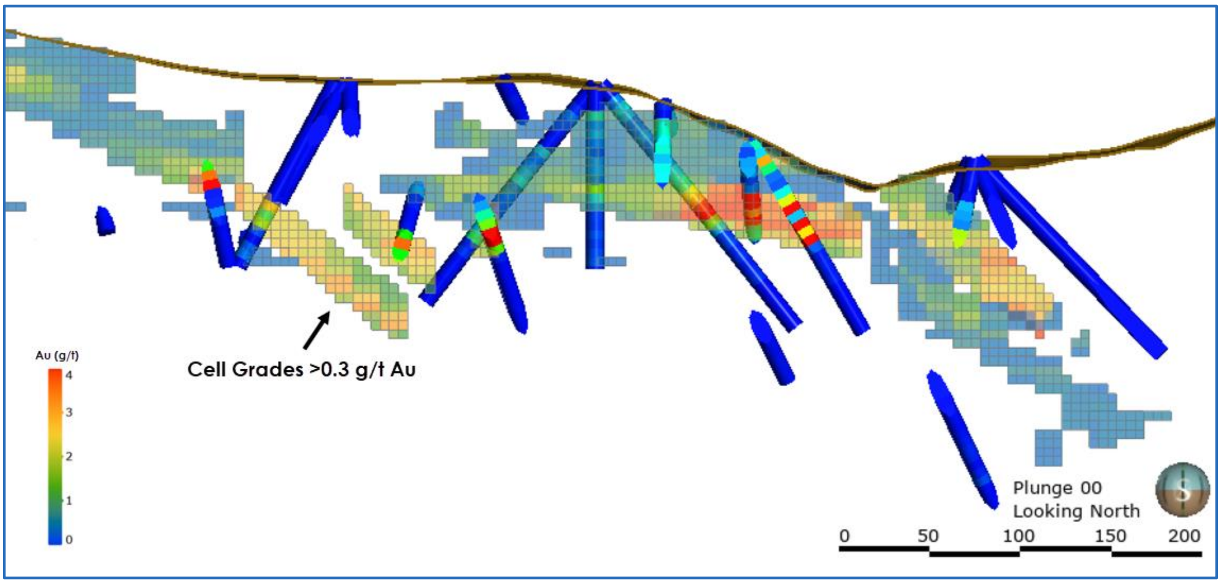
Validation of the model estimates included visual inspection of cell grades relative to drillhole composites, checks for global bias, check of local bias (swath plot), and comparisons to other estimation methods.

14.3.12.1 Ardich Visual Inspection

Visual inspection of plans and sections and 3D visualisation confirmed that the cell model estimates honour the drillhole data and grade shell boundaries. An example cross-section and bench-section are presented in Figure 14.40 and Figure 14.41.

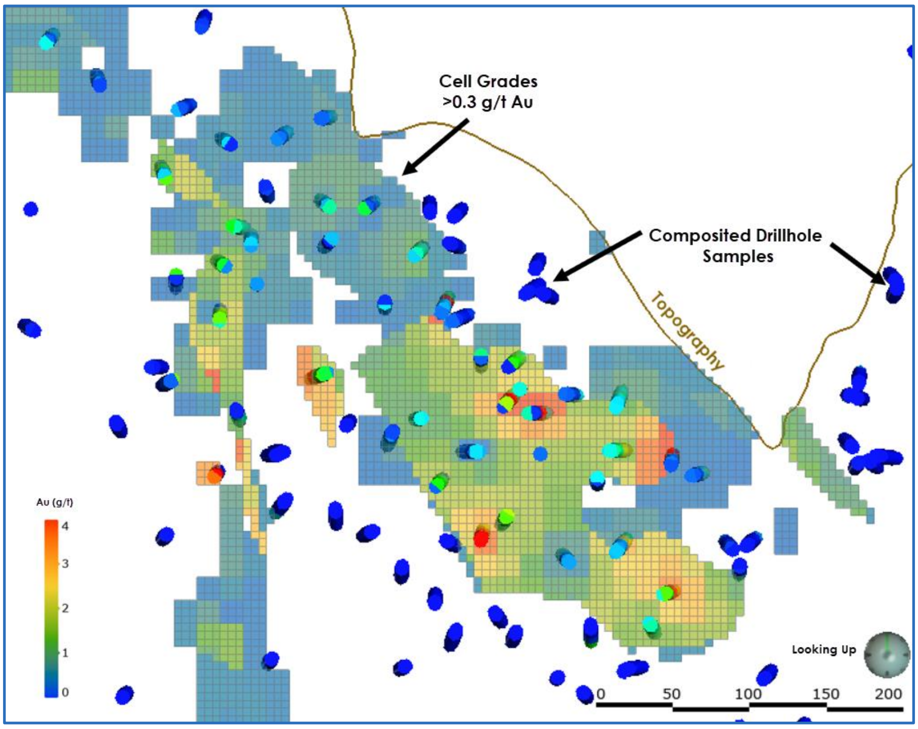
Cell grade estimates within the grade shells were visually confirmed. The grade shells follow lithological contacts and structural intersections. The use of hard grade boundaries can be seen for gold and sulfur. These sharp changes in grade are expected, rather than being an artefact of the estimate due to the close adherence of mineralisation to geological features.

Figure 14.40 Ardich Cross-Section 4,367,190 mN with Drilling and Au Model Estimates



SSR Mining, 2020

Figure 14.41 Ardich Bench Plan with Drilling and Au Model Estimates



SSR Mining, 2020

The relationship of gold mineralisation to ophiolite was checked visually and statistically and the ophiolite was deemed to be waste rock. In some areas the estimation of Au pushes grade into the ophiolite lithology. To correct for this extension of Au grade, a calculation was used to cap ophiolite cells to 0.5 g/t Au. In some areas, the logged jasperoid shape is near the ophiolite contact creating high-grade cells. Further refinement of the grade shells and jasperoid shapes are needed to reduce the occurrence of elevated Au grade in the ophiolite.

14.3.12.2 Ardich Global Bias

The cell model was checked for global bias by comparing the mean Au grades at a zero cut-off from the ID3 model with means from NN estimates.

The NN estimator produces a theoretically unbiased (de-clustered) estimate of the mean value when no cut-off grade is imposed and provides a basis for checking the performance of different estimation methods. In general, an estimate is considered acceptable here if the bias is at or below 5%.

The global bias is shown in Table 14.42.

Table 14.42 Ardich Global Au Bias by Domain

Domain	Element	ID3 Estimates	NN Estimates	Rel. Diff. (%)
1	Au	0.048	0.044	−7.0%
2		0.226	0.231	2.3%
3		0.157	0.156	−0.5%
4		0.322	0.310	−3.8%
5		0.166	0.155	−6.3%
6		0.403	0.394	−2.5%
7		0.461	0.457	−0.9%
8		0.242	0.238	−1.5%
9		0.103	0.102	−0.5%
10		0.042	0.041	−2.9%
15		0.147	0.143	−3.1%
	Total	0.157	0.154	−2.1%

The overall relative difference (including all domains) was 2.1%. Higher variances were seen in domains 1 and 5. Domain 1 is located at the base of the model with limited drilling and tonnes. Domain 5 is the northern most portion of Ardich with a steep dip of lithological strata and mineralisation. Due to the domain location, influence of these areas on the deposit is minimal.

As an additional check, bias within the gold grade shells was reviewed. The overall variance in the gold grade shells is 3.2%, which is below the 5% tolerance.

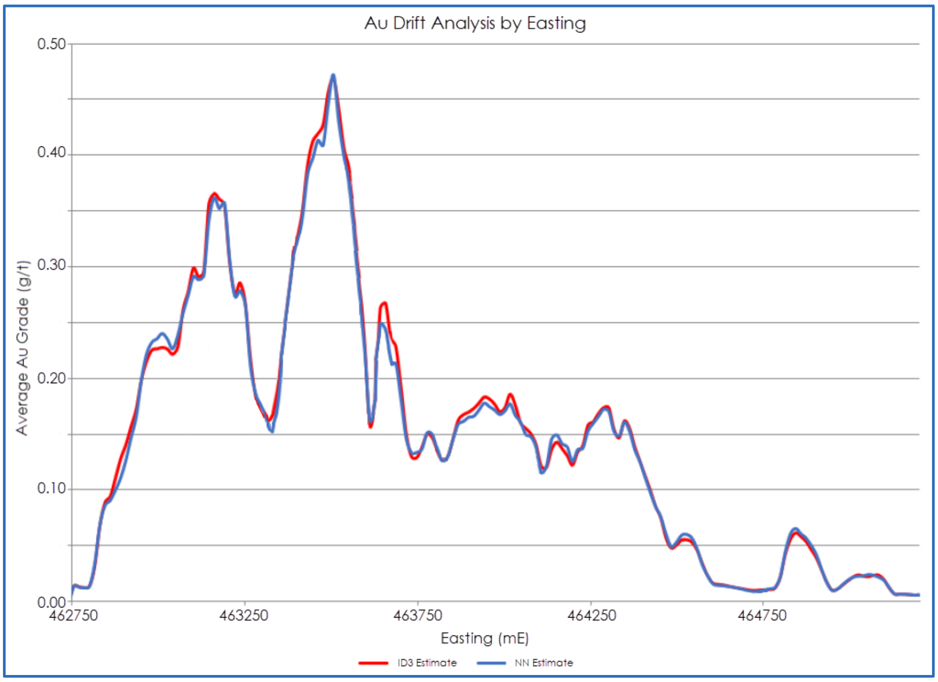
14.3.12.3 Ardich Local Bias

Local trends in the grade estimates (also known as drift analysis) were assessed by plotting the mean values from the ID3 estimate versus the NN results for Indicated model cells in east–west, north–south and vertical directions (swath plots).

There is good correspondence between the NN and ID3 estimate in all axis directions. The comparison of each estimate diverges at model edges where data density is limited. The close comparison in estimates is primarily due to grade shell domaining restricting the selection of samples for the estimate.

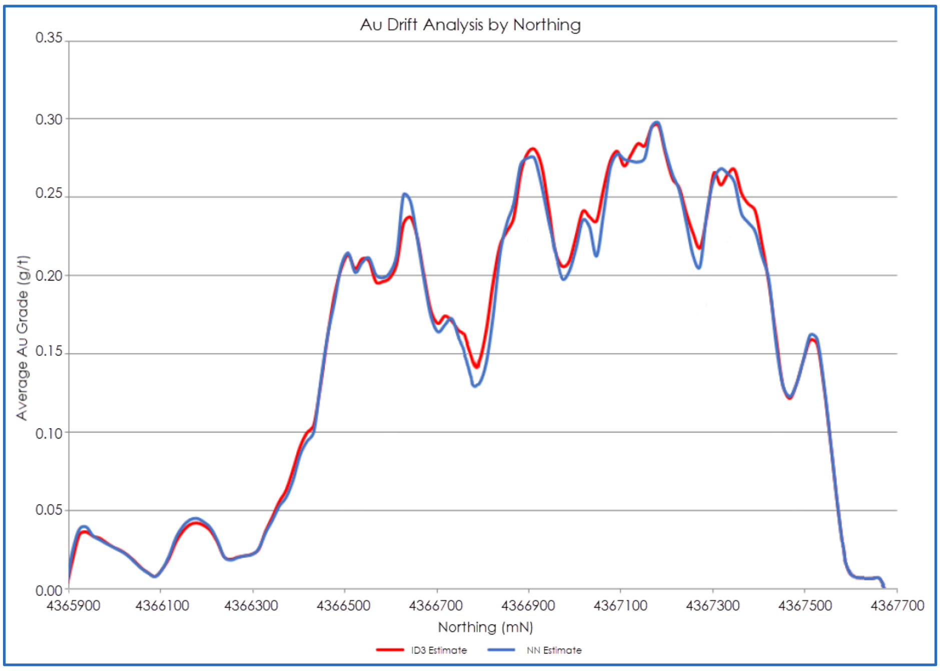
Swath plots for the Au estimate in Ardich are shown in Figure 14.42, Figure 14.43, and Figure 14.44.

Figure 14.42 Ardich Au Grade Trend Plot by Easting



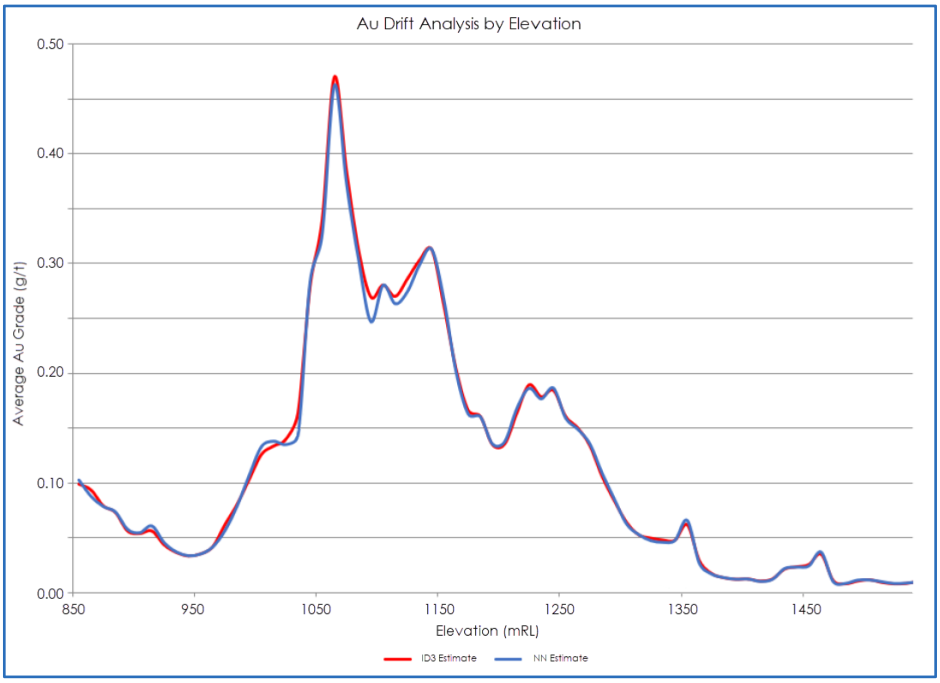
SSR Mining, 2020

Figure 14.43 Ardich Au Grade Trend Plot by Northing



SSR Mining, 2020

Figure 14.44 Ardich Au Grade Trend Plot by Elevation



SSR Mining, 2020

14.3.13 Ardich Change of Support

Mining has not occurred at the Ardich project area and therefore no production data is available. A 5 m mining bench is anticipated, with 5 m blast holes likely to be used. Grade estimation at Ardich is based on 5 m assay composites, interpreted mineralised shapes, and fault domains to estimate resource model tonnes and grade.

14.3.14 Ardich Assessment of Reasonable Prospects of Eventual Economic Extraction

Mineral Resource estimates were shown to meet reasonable prospects for eventual economic extraction criteria by reporting only material that was contained within a conceptual pit shell using metal prices of $1,750/oz for gold with the parameters summarised in Table 14.43.

Table 14.43 Summary of Key Parameters Used in Conceptual Pit Shell at Ardich

Description	Unit	Minimum	Maximum
Heap Leach Gold Recovery	%	40.0	73.0
POX Gold Recovery	%	82.9	82.9
Mining Cost per tonne mined	$/t	1.61	1.61
Process Costs Heap Leach	$/t	8.89	8.89
Process Costs POX	$/t	32.53	32.53
Site Support per tonne processed	$/t	3.17	6.60
Internal Au Cut-off – Heap Leach	g/t	0.30	0.55
Internal Au Cut-off – POX	g/t	0.77	0.77
Royalty	%	1.5	1.5

14.3.15 Ardich Mineral Resource Tabulation

Ardich Mineral Resources have been tabulated by resource classification and oxidation state in Table 14.44. Mineral Resources are presented on a 100% basis.

Mineral Resources that are not Mineral Reserves have not demonstrated economic viability.

The overall tonnage and grade estimate have increased for oxide and sulfide material from the previously-reported estimate in 2019. This change is predominantly due to the change in gold price from $1,500/oz in 2019 to $1,750/oz in 2020, and the associated resultant drop in cut-off grades. The pit shell used to constrain the resource has been updated to reflect the increase in gold price. There has been no depletion from mining.

Resource pit shells were generated by OreWin using a metal price assumption of $1,750/oz gold. Gold mineralisation modelled at Ardich is primarily oxidised with a smaller portion of sulfur mineralisation having estimated total sulfur grades >2%. Low-sulfur (LS) oxide is defined as material with <1% total sulfur. High-sulfur (HS) oxide is material with total sulfur >1% and <2%. Sulfide material has ≥2% total sulfur. The Mineral Resources are shown in Table 14.44.

Internal cut-off grades for oxide material range from 0.30–0.55 g/t Au. Sulfide is material with >2% total sulfur above a 1.1 g/t Au cut-off.

Table 14.44 Ardich Mineral Resource Table by Classification and Oxide State

Mineral Resource Estimate for the Ardich Deposit (as at the Effective Date)				
Material Type	**Resource Category Material**	**Tonnes (kt)**	**Au (g/t)**	**Contained Gold (koz)**
Oxide (LS+HS)	Measured	4,707	1.63	246
	Indicated	12,817	1.62	666
	Measured + Indicated	17,524	1.62	912
	Inferred	4,713	1.62	246
Sulfide	Measured	695	2.56	57
	Indicated	2,231	3.71	266
	Measured + Indicated	2,926	3.43	323
	Inferred	782	4.24	107
Total	Measured	5,402	1.75	303
	Indicated	15,048	1.93	932
	Measured + Indicated	20,451	1.88	1,235
	Inferred	5,495	1.99	352

1. Mineral Resources have an effective date of 27 November 2020.
2. Mineral Resources that are not Mineral Reserves have not demonstrated economic viability.
3. Mineral Resources are shown on a 100% basis. More than 96% of the Mineral Resources are located on the SSR Mining owned 80% ground, with the remainder of the mineralisation within the 50%/50% ownership boundary.
4. Low-sulfur (LS) oxide is defined as material with <1% total sulfur, high-sulfur (HS) oxide is material with total sulfur >1% and <2%, and sulfide material has ≥2% total sulfur.
5. All Mineral Resources in the CDMP20 were assessed for reasonable prospects for eventual economic extraction by reporting only material that fell within conceptual pit shells based on metal prices of $1,750/oz for gold. The following parameters were used: metallurgical recoveries in oxide 40.0%–73.0%, and in sulfide 82.9%; Au cut-off grades in oxide 0.30–0.55 g/t Au, and in sulfide 0.77 g/t Au, (there are no credits for Ag or Cu in the cut-off grade calculations); allowances have been made for royalty payable.
6. Reported Mineral Resources contain no allowances for unplanned dilution, or mining recovery. Tonnage and grade measurements are in metric units. Contained gold is reported in troy ounces.
7. Tonnages are rounded to the nearest thousand tonnes; grades are rounded to two decimal places. As a result, totals may not match.

14.4 Bayramdere Deposit

The Bayramdere deposit is located approximately 6.3 km east of the Çöpler mine and 5 km south-east of İliç. Bayramdere is within the Kartaltepe Mining Licence 7083. This licence is an operational licence and is 50% Anagold-held.

Soil samples have been collected across the prospect on a 100 m x 100 m grid. Soil copper and gold anomalies are identified as coincident with each other, but the copper anomaly covers a larger area.

The Bayramdere mineralisation has an overall strike length of approximately 300 m. Mineralisation is localised within three stacked, shallow-dipping lodes that are very close to the surface, varying in depth 30–40 m below topography. Mineralisation appears to be open to the east and south.

The mineralisation has formed at the contacts of limestone and ophiolite lithologies with mineralisation replacing limestone along the contacts. The limestone to ophiolite contacts are low-angle thrusts, with limestone typically being trapped as wedges of material within a dominantly ophiolite stratigraphy. Mineralisation occurs within iron-rich gossan horizons.

Although a small deposit, Bayramdere is relatively high grade and can support a high stripping ratio to access mineralisation.

Small-scale open pit iron ore mining has occurred historically at Bayramdere. Iron mineralisation can be associated with gold mineralisation.

14.4.1 Bayramdere Domains

The geological interpretation was represented in the geological model through the creation of mineralised domains based on the continuity of the geology and mineralisation identified specific to each deposit and mineralised zone within the deposit. Separate domains were created for gold, silver, copper, and sulfur. In the creation of mineralised domains, a minimum mining width of 2.5 m was used based on anticipated open pit mining methods.

14.4.2 Bayramdere Geological Model

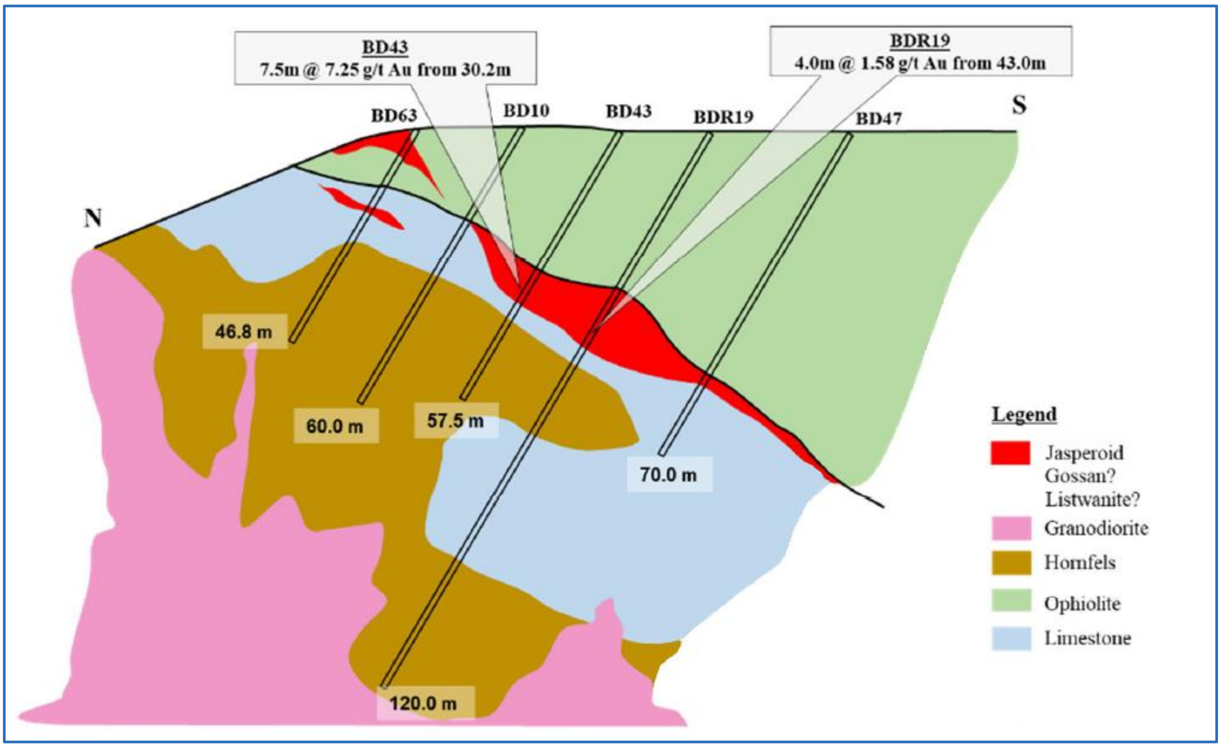
The Bayramdere deposit is a structurally controlled gold±minor copper±minor silver deposit displaying both epithermal and replacement mineralisation styles. At this stage of exploration, the deposit is dominantly represented by near-surface oxide mineralisation to a depth of up to 180 m below surface. Mineralisation is primarily associated with jasperoid and iron-rich gossan. Secondary pyrite is a commonly visible component within the jasperoids.

At depth, mineralisation transitions below the base of complete oxidation to disseminated pyrite, vein sulfides, and massive sulfide horizons generally occurring within shear zones, along shallow thrusts and diorite sill and dyke margins. The extent of sulfide mineralisation has not been tested.

As with the other Çöpler district deposits, Bayramdere is considered to be the result of a mineralised intrusion generating suitable conditions for mineralisation to be localised into a favourable geological setting of ophiolite, limestone, and hornfels lithologies (see Figure 14.45). A complex system of faults and thrusts have allowed mineralised fluids and diorite dykes and sills associated with the epithermal system to permeate into the stratigraphy.

Like the Çakmaktepe deposit, Bayramdere is associated with flat thrust structures. Key to each structurally associated style of mineralisation is the juxtaposition of ophiolites against limestone + hornfels to create suitable geochemical conditions for gold and other metals deposition. Ophiolite is not associated with mineralisation at Çöpler, this association at present is considered to be unique to Bayramdere and Çakmaktepe.

Figure 14.45 Bayramdere Geology Schematic Section



SSR Mining, 2017

14.4.3 Bayramdere Data Summary

The Bayramdere deposit has been drilled on 25 m lines with 20–25 m spaced holes on each line.

A total of 115 resource definition drillholes have been drilled at Bayramdere for a total length of 10,708.9 m, inclusive of metallurgical holes. The assay database includes 8,283 sample intervals for a total assayed length of 10,483.4 m.

When categorised according to type of drilling (excluding geotechnical and metallurgical drillholes), 30% are RC samples, 65% DD core samples, and 6% are a combination of RC and DD core.

Drilling has been completed on drill grids aligned at right angles to mineralisation trends or lithology dip and strike. Several areas contain scissor holes that test mineralisation at 180° from each other.

14.4.4 Bayramdere Drillhole Compositing

Sample compositing has not been applied. The predominant sample length is 1.0 m (52%), followed by 2 m as the next most prevalent length (17%).

14.4.5 Bayramdere Top Cutting

High-grade top cuts were applied after selecting appropriate limits based on cumulative frequency plots and value grade curves of the upper portion of the sample population.

14.4.6 Bayramdere Cell Model

The Bayramdere cell model parameters are shown in Table 14.45.

Sub-celling was permitted to 2 m x 2 m x 1 m to better honour the domain boundaries.

Table 14.45 Bayramdere Cell Model Prototype Parameters

Direction	Minimum (m)	Maximum (m)	Range (m)	Cell Size (m)	No. of Cells
East	466,000	466,600	600	10	60
North	4,363,800	4,364,100	300	10	30
RL	1,250	1,420	170	5	34

14.4.7 Bayramdere Estimation Method

Estimation was limited to the interpreted domains, with each domain informed only by samples contained within that domain. Outside the mineralised domains a 'mineralised waste' estimate was completed.

Mineralisation domains were also developed for silver, copper, and sulfur.

Lithological domains were used for estimates outside of the mineralisation domains.

Ordinary kriging was used to estimate Au, Ag, and Cu into parent cells. Variography was completed to inform estimation.

14.4.8 Bayramdere Density Model

Density has been assigned as a default for each of the mineralisation and lithological domains (see Table 14.46 and Table 14.47 respectively). The assigned densities reflect the arithmetic average of the domain-relevant data taken from DD core samples.

Table 14.46 Bayramdere Density Values for Mineralisation Domains

Domain	Density (t/m³)
mz100	2.69
mz101	2.49
mz200	2.55
mz201	2.79
mz300	2.29
mz301	2.29
mz400	2.49
mz500	2.42
mz600	2.49
mz700	2.49

Table 14.47 Bayramdere Density Values for Lithology Domains

Domain	Weathering State	Density (t/m³)
Gossan	Weathered	2.50
Diorite		2.44
Limestone		2.54
Ophiolite		2.36
Gossan	Fresh	2.50
Diorite		2.44
Limestone		2.54
Ophiolite		2.36
Overburden	All	1.40

14.4.9 Bayramdere Resource Classification

Grade estimates were classified using the following SSR Mining guidelines:

- Indicated Mineral Resource should be quantified within relative ±15% with 90% confidence on an annual basis, and
- Measured Mineral Resources should be known within ±15% with 90% confidence on a quarterly basis.

Drillhole spacing for support of classification of Inferred Mineral Resources was required to be 50 m x 25 m spacing. For Indicated Mineral Resource classification, the drillhole spacing requirement was reduced to 25 m x 25 m spacing. Appropriate drillhole pattern spacing selection was based on the understanding of the nature of the mineralisation being structurally controlled, mineral continuity, and assessment of data quality.

The drillhole spacing at Bayramdere is considered sufficient to support grade continuity, geological continuity, depth and lateral extents of mineralisation.

No Bayramdere estimates were classified in the Measured category.

Mineral Resources were tabulated using multiple cut-off grades due to variable recoveries and based on gold price only. Cut-off grades vary from 0.35–0.50 g/t Au and are calculated based on the equation:

$$Xc = Po / (r * (V-R))$$

where Xc = Cut-off Grade (g/t), Po = processing cost of ore (USD/tonne of ore), r = recovery, V = gold sell price ($/g), R = refining costs ($/g).

Mineral Resources are reported inclusive of Mineral Reserves.

14.4.10 Bayramdere Validation

Bayramdere grade estimates were validated against alternate interpolation methods. Estimated grades were compared to an ID2 model to check for global bias. Swath plots were used to check for a local bias. The estimated Au grades in the model were compared to the composite grades by visual inspection in plan views and cross-sections. Composite samples were queried by domain to confirm appropriate sample flagging.

14.4.11 Bayramdere Assessment of Reasonable Prospects of Eventual Economic Extraction

Mineral Resource estimates were shown to meet reasonable prospects for eventual economic extraction criteria by reporting only material that was contained within a conceptual pit shell using metal prices of $1,400/oz for gold and $19/oz for silver, with the parameters summarised in Table 14.48. These parameters have not been updated since 2017, primarily because no further work has been completed at Bayramdere since that time.

Table 14.48 Summary of Key Parameters Used in Conceptual Pit Shell at Bayramdere

Description	Unit	Minimum	Maximum
Heap Leach Gold Recovery	%	75.0	75.0
Mining Cost per tonne mined	$/t	1.75	1.75
Process Costs Heap Leach	$/t	9.99	9.99
Site Support per tonne processed	$/t	3.19	3.19
Internal Au Cut-off – Heap Leach	g/t	0.35	0.50
Royalty	%	2.0	2.0

14.4.12 Bayramdere Mineral Resource Tabulation

Bayramdere Mineral Resources have been tabulated by resource classification in Table 14.49. Mineral Resources are presented on a 100% basis. The entire Mineral Resource is oxide material.

Mineral Resources that are not Mineral Reserves have not demonstrated economic viability.

Table 14.49 Bayramdere Mineral Resource Table by Classification

Mineral Resource Estimate for the Bayramdere Deposit (as at the Effective Date)					
Resource Category	**Tonnes (kt)**	**Au (g/t)**	**Ag (g/t)**	**Contained Gold (koz)**	**Contained Silver (koz)**
Measured	–	–		–	
Indicated	145	2.34	20.82	11	97
Measured + Indicated	**145**	**2.34**	**20.82**	**11**	**97**
Inferred	*8*	*2.17*	*19.95*	*1*	*5*

1. Mineral Resources have an effective date of 27 November 2020.
2. Mineral Resources that are not Mineral Reserves have not demonstrated economic viability.
3. Mineral Resources are shown on a 100% basis, of which SSR Mining owns 50%.
4. Oxide is defined as material with <2% total sulfur. All Mineral Resource at Bayramdere is oxide.
5. All Mineral Resources in the CDMP20 were assessed for reasonable prospects for eventual economic extraction by reporting only material that fell within conceptual pit shells based on metal prices of $1,400/oz for gold. The following parameters were used: metallurgical recoveries in oxide: 75.0%; Au cut-off grades in oxide: 0.35–0.50 g/t Au; allowances have been made for royalty payable.
6. Reported Mineral Resources contain no allowances for unplanned dilution, or mining recovery. Tonnage and grade measurements are in metric units. Contained gold is reported in troy ounces.
7. Tonnages in Table 14.32 are rounded to the nearest thousand tonnes; grades are rounded to two decimal places. As a result, totals may not match.

14.5 Comparison of 2020 Mineral Resource Inventory to 2019 Mineral Resource Inventory

A summary of the entire CDMP20 Mineral Resource inventory is shown in Table 14.50.

The differences between the CDMP20 Mineral Resources and the previous Mineral Resources reported as at 31 December 2019 are shown in Table 14.51 for each deposit, material type, and classification.

The differences are a function of the following changes:

- Reduction in cut-off grades due to lower unit costs, higher throughputs in the sulfide plant, and increased gold price
- Larger conceptual pit shell selecting additional model cells above the cut-off
- Review of metallurgical recoveries
- Update to Çakmaktepe and Ardich resource models to incorporate recent drillhole data
- Review of Mineral Resource classification method
- Depletion through mining since 31 December 2019

Overall, there has been a 51% increase in tonnage above the cut-off across all combined Mineral Resource categories, with a corresponding 32% increase in contained gold.

Table 14.50 CDMP20 Mineral Resources Summary – All Deposits

CDMP20 Mineral Resources Summary (as at the Effective Date)							
Classification	**Tonnage (kt)**	**Grades**			**Contained Metal**		
		Au (g/t)	**Ag (g/t)**	**Cu (%)**	**Gold (koz)**	**Silver (koz)**	**Copper (klb)**
Çöpler Mine Oxide Mineral Resource							
Measured	287	1.29	7.75	0.09	12	72	540
Indicated	25,139	0.98	3.44	0.15	789	2,781	81,399
Measured + Indicated	25,427	0.98	3.49	0.15	801	2,853	81,939
Inferred	*33,083*	*0.96*	*7.16*	*0.13*	*1,017*	*7,614*	*94,935*
Çöpler Mine Sulfide Mineral Resource							
Measured	2,454	2.22	7.21	–	175	569	–
Indicated [5]	84,558	1.84	5.04	–	5,015	12,617	–
Measured + Indicated	87,012	1.86	4.71	–	5,190	13,186	–
Inferred	*34,073*	*1.54*	*12.72*	–	*1,692*	*13,937*	–
Çakmaktepe Oxide Mineral Resource							
Measured	–	–	–	–	–	–	–
Indicated [6]	3,626	1.53	8.50	–	179	990	–
Measured + Indicated	3,626	1.53	8.50	–	179	990	–
Inferred	*1,205*	*0.85*	*4.04*	–	*33*	*157*	–
Ardich Oxide Mineral Resource							
Measured	4,707	1.63	–	–	246	–	–
Indicated	12,817	1.62	–	–	666	–	–
Measured + Indicated	17,524	1.62	–	–	912	–	–
Inferred	*4,713*	*1.62*	–	–	*246*	–	–
Ardich Sulfide Mineral Resource							
Measured	695	2.56	–	–	57	–	–
Indicated	2,231	3.71	–	–	266	–	–
Measured + Indicated	2,926	3.43	–	–	323	–	–
Inferred	*782*	*4.24*	–	–	*107*	–	–
Bayramdere Oxide Mineral Resource							
Measured	–	–	–	–	–	–	–
Indicated	145	2.34	20.82	–	11	97	–
Measured + Indicated	145	2.34	20.82	–	11	97	–
Inferred	*8*	*2.17*	*19.95*	–	*1*	*5*	–
CPMD20 Mineral Resources Total							
Measured	8,143	1.87	2.45	0.00	490	641	540
Indicated	128,517	1.68	3.99	0.03	6,926	16,485	81,399
Measured + Indicated	136,660	1.69	3.90	0.03	7,416	17,126	81,939
Inferred	*73,865*	*1.30*	*9.14*	*0.06*	*3,094*	*21,713*	*94,935*

1. Mineral Resources have an effective date of 27 November 2020.
2. Mineral Resources are reported based on end of August 2020 topography surface.
3. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
4. Mineral Resources are shown on a 100% basis. Çöpler Mineral Resources are located on ground held 80% by SSR Mining, Çakmaktepe and Bayramdere Mineral Resources are located on ground held 50% by SSR Mining, and approximately 96% of Ardich Mineral Resources are located on ground held 80% by SSR Mining, with the remainder located on ground 50% held by SSR Mining.
5. Çöpler Sulfide Indicated total includes stockpiles: 6,674 kt @ 2.63 g/t Au.
6. Çakmaktepe Oxide Indicated total includes stockpiles: 11 kt @ 2.69 g/t Au.
7. At Çöpler: oxide is defined as material <2% total sulfur and sulfide material is ≥2% total sulfur.
8. At Ardich and Çakmaktepe, low-sulfur (LS) oxide is defined as material with <1% total sulfur, high-sulfur (HS) oxide is material with ≥1% and <2% total sulfur, and sulfide material is ≥2% total sulfur.
9. At Bayramdere: oxide is defined as material <2% total sulfur. There is no sulfide material at Bayramdere.
10. All Mineral Resources in the CDMP20 were assessed for reasonable prospects for eventual economic extraction by reporting only material that fell within conceptual pit shells based on metal prices of $1,750/oz for gold ($1,400 for gold and $19/oz for silver for Bayramdere). The following parameters were used:
    metallurgical recoveries in oxide: Çöpler 62.3%–78.4%, Çakmaktepe 38.0%–80.0%, Ardich 40.0%–73.0%, and Bayramdere 75.0%, and in sulfide: Çöpler 85.0%, and Ardich 82.9%;
    Au cut-off grades in oxide: Çöpler 0.32–0.41 g/t Au, Çakmaktepe 0.36–0.76 g/t Au, Ardich 0.30–0.55 g/t Au, and Bayramdere 0.35–0.50 g/t Au, and in sulfide: Çöpler 0.73 g/t Au and Ardich 0.77 g/t Au, (there are no credits for Ag or Cu in the cut-off grade calculations); allowances have been made for royalty payable.
11. Reported Mineral Resources contain no allowances for unplanned dilution or mining recovery.
12. Totals may vary due to rounding.

Table 14.51 CDMP20 Mineral Resources Compared to 2019 Mineral Resources

Relative Difference between CDMP20 Mineral Resources (as at the Effective Date) and 2019 Mineral Resources (as at 31 December 2019)							
Classification	**Tonnage (kt)**	**Grades**			**Contained Metal**		
		Au (g/t)	**Ag (g/t)**	**Cu (%)**	**Gold (koz)**	**Silver (koz)**	**Copper (klb)**
Çöpler Mine Oxide Mineral Resource							
Measured	New to 2020						
Indicated	+99%	+1%	–19%	+22%	+101%	+60%	+143%
Measured + Indicated	+101%	+1%	–18%	+22%	+104%	+65%	+145%
Inferred	+77%	+12%	+2%	–6%	+98%	+80%	+66%
Çöpler Mine Sulfide Mineral Resource							
Measured	New to 2020						
Indicated [*]	+26%	–14%	–7%	New to 2020	+8%	+17%	New to 2020
Measured + Indicated	+30%	–14%	–6%	New to 2020	+11%	+22%	New to 2020
Inferred	+171%	–22%	+6%	New to 2020	+110%	+188%	New to 2020
Çakmaktepe Oxide Mineral Resource							
Measured	*n/a*	*n/a*	*n/a*	*n/a*	*n/a*	*n/a*	*n/a*
Indicated [†]	+78%	–21%	–13%	*n/a*	+41%	+54%	*n/a*
Measured + Indicated	+78%	–21%	–13%	*n/a*	+41%	+54%	*n/a*
Inferred	–28%	–4%	–37%	*n/a*	–31%	–55%	*n/a*
Ardich Oxide Mineral Resource							
Measured	New to 2020						
Indicated	–10%	+9%	*n/a*	*n/a*	–2%	*n/a*	*n/a*
Measured + Indicated	+23%	+9%	*n/a*	*n/a*	+35%	*n/a*	*n/a*
Inferred	–36%	–5%	*n/a*	*n/a*	–39%	*n/a*	*n/a*
Ardich Sulfide Mineral Resource							
Measured	New to 2020						
Indicated	+34%	+41%	*n/a*	*n/a*	+90%	*n/a*	*n/a*
Measured + Indicated	+76%	+31%	*n/a*	*n/a*	+131%	*n/a*	*n/a*
Inferred	–47%	+4%	*n/a*	*n/a*	–44%	*n/a*	*n/a*
Bayramdere Oxide Mineral Resource							
Measured	*n/a*	*n/a*	*n/a*	*n/a*	*n/a*	*n/a*	*n/a*
Indicated	0%	0%	0%	*n/a*	*n/a*	0%	*n/a*
Measured + Indicated	0%	0%	0%	*n/a*	*n/a*	0%	*n/a*
Inferred	0%	0%	0%	*n/a*	*n/a*	0%	*n/a*
CPMD20 Mineral Resources Total							
Measured	New to 2020						
Indicated	+31%	–12%	–5%	New to 2020	+15%	+24%	+664%
Measured + Indicated	+40%	–12%	–8%	New to 2020	+24%	+29%	+675%
Inferred	+77%	–11%	+31%	New to 2020	+58%	+131%	+287%

'n/a' indicates that the value was reported in neither 2019 nor 2020
'New to 2020' indicates that the value is reported in 2020 but there was no equivalent value reported in 2019
Bayramdere Mineral Resource is unchanged since 2019

15 MINERAL RESERVE ESTIMATES

15.1 Summary

Open pit mining at the Çöpler project is carried out by a mining contractor and managed by SSR Mining. The mining method is a conventional open pit method with drill and blast to facilitate extraction utilising excavators and trucks. SSR Mining currently operates a sulfide process plant and an oxide heap leach facility. Costs are based on actual operational costs and the Anagold budget assumptions.

The Mineral Reserves were developed based on mine planning work completed in October 2020 and estimated based on an end-of-August 2020 topography surface. Çöpler oxide ore cut-off grades vary from 0.47–0.59 g/t Au. The Çöpler sulfide ore cut-off grade is 1.05 g/t Au. Çakmaktepe Oxide cut-off grades vary from 0.52 g/t Au to 0.69 g/t Au. There is no Çakmaktepe Sulfide Mineral Reserve. Average oxide gold recoveries are 73% and average sulfide gold recoveries are 91%.

The cut-off grades for the Mineral Reserves estimates are based on a gold price of $1,350/oz. There are no credits for silver or copper in the cut-off grade calculations. Economic analysis has been carried out using long-term metal prices of $1,585/oz gold, $20.25/oz silver, and $3.05/lb copper, and average metal prices of $1,658/oz gold, $21.55/oz silver, and $2.95/lb copper.

15.2 Mineral Reserve Statement

The Mineral Reserves statement is shown in Table 15.1. Mineral Reserves have been classified using the 2014 CIM Definition Standards and were estimated by Bernard Peters BEng (Mining), FAusIMM (201743), employed by OreWin Pty Ltd as Technical Director – Mining. Mineral Reserves are presented on a project basis and have an effective date of 27 November 2020.

The CDMP20 Reserve Case is at a feasibility level of study. The Mineral Resource estimates have been reported in the CDMP20. The Mineral Resource models include dilution. Measured Mineral Resources were converted to Proven Mineral Reserves, and Indicated Mineral Resources were converted to Probable Mineral Reserves. Inferred Mineral Resources were treated as waste and were not converted to Mineral Reserve. The Çöpler Mineral Reserve has been demonstrated to be viable by the CDMP20.

Table 15.1 CDMP20 Mineral Reserves

CDMP20 Mineral Reserves Summary (as at the Effective Date)							
Classification	Tonnage (kt)	Grades			Contained Metal		
		Au (g/t)	Ag (g/t)	Cu (%)	Gold (koz)	Silver (koz)	Copper (klb)
Çöpler Mine – Oxide							
Proven Mineral Reserve	230	1.23	8.97	0.06	9	66	294
Probable Mineral Reserve	7,364	1.23	6.16	0.13	290	1,458	20,549
Probable – Stockpile	–	–	–	–	–	–	–
Total Mineral Reserve	7,595	1.23	6.24	0.12	299	1,525	20,843
Çöpler Mine – Sulfide							
Proven Mineral Reserve	2,140	2.42	7.63	–	166	525	–
Probable Mineral Reserve	42,461	2.18	5.73	–	2,970	7,819	–
Probable – Stockpile	6,674	2.63	–	–	564	–	–
Total Mineral Reserve	51,274	2.24	5.06	–	3,700	8,344	–
Çakmaktepe Mine – Oxide							
Proven Mineral Reserve	–	–	–	–	–	–	–
Probable Mineral Reserve	274	1.26	10.91	–	11	96	–
Probable – Stockpile	11	2.69	–	–	1	–	–
Total Mineral Reserve	285	1.32	10.49	–	12	96	–
CDMP20 – Oxide Reserve							
Proven Mineral Reserve	230	1.23	8.97	0.06	9	66	294
Probable Mineral Reserve	7,638	1.23	6.33	0.13	301	1,554	20,549
Probable – Stockpile	11	2.69	–	–	1	–	–
Total Mineral Reserve	7,879	1.23	6.40	0.12	311	1,621	20,843
CDMP20 – Sulfide Reserve							
Proven Mineral Reserve	2,140	2.42	7.63	–	166	525	–
Probable Mineral Reserve	42,461	2.18	5.73	–	2,970	7,819	–
Probable – Stockpile	6,674	2.63	–	–	564	–	–
Total Mineral Reserve	51,274	2.24	5.06	–	3,700	8,344	–
CDMP20 Mineral Reserves Total							
Proven Mineral Reserve	2,370	2.30	7.76	0.01	175	591	294
Probable Mineral Reserve	50,099	2.03	5.82	0.02	3,271	9,373	20,549
Probable – Stockpile	6,685	2.63	–	–	565	–	–
Total Mineral Reserve	59,154	2.11	5.24	0.02	4,011	9,964	20,843

1. Effective date of the CDMP20 Mineral Reserve is 27 November 2020.
2. The Mineral Reserves were developed based on mine planning work completed in October 2020 and estimated based on End of August 2020 topography surface.
3. Mineral Reserve cut-offs are based on $1,350/oz gold price; average oxide recoveries are 73% and average sulfide recoveries are 91%.
4. Çöpler oxide cut-off grades 0.47–0.59 g/t Au, Çöpler sulfide cut-off grade 1.05 g/t Au, Çakmaktepe oxide cut-off grades 0.52–0.71 g/t Au; all cut-off grades include allowance for royalty payable. There are no credits for silver or copper in the cut-off grade calculations. There is no Çakmaktepe Sulfide Mineral Reserve.
5. Economic analysis has been carried out using a long-term gold price of $1,585/oz. The economic analysis has used a Q4'20 start date.
6. Mineral Reserves tabulated include 403 kt at 2.47 g/t Au from the mine plan scheduled for September 2020.
7. Totals may vary due to rounding.

Significant factors that could materially affect the Mineral Reserve are:

- Environmental, Permitting, Social, and Community – the Çöpler project is subject to the laws and regulations of Turkey, the mine has a number of local communities that are nearby. In order to operate the mine, Anagold must maintain appropriate relations with all the authorities and stakeholders. Social, community and government relations are managed by Anagold and include programmes and engagement with the local communities and both local and national governments. Anagold has remained in compliance with all aspects of the EIA and operating permits throughout the history of the project.
- Seismic impacts – the Çöpler project is located in an area with a history of significant seismic activity that could negatively impact mining operations.
- Metal price impacts – gold is the primary revenue element and silver and copper are produced as by-products. The ore is mined at an elevated cut-off grade and low grade ore is stockpiled for processing after mining is completed. The use of the elevated cut-off grade serves to mitigate the risks from periods of lower gold prices.
- Mining impacts – the mining equipment is suitable for a selective mining unit (SMU) of approximately 3 m x 3 m x 5 m. This allows for selectivity in mining and enhances the opportunities for blending the feed to the sulfide plant. The total mining rates in the CDMP20 mine plan are at 22.5 Mtpa, In the past, total mining rates of 36.5 Mtpa have been achieved, increasing the total mining rate may allow gold to be brought forward in the production schedule but will require additional stockpile storage areas.
- Geotechnical impacts – slope recommendations have significant impacts on the Mineral Reserve and the continued study will allow the Mineral reserves to be maximised.
- Processing impacts – the processing analysis in the Reserve Case includes incorporation of a flotation circuit into the existing sulfide plant to upgrade sulfide sulfur (SS) to fully utilise grinding and pressure oxidation (POX) autoclave capacity. Continued debottlenecking of the sulfide plant and optimisation of the flotation circuit when it commences operations may improve costs and recoveries, changing cut-off grades and impacting the Mineral Reserve.
- The addition of the flotation circuit to the sulfide plant requires new grade control protocols and a new associated stockpile strategy will be implemented to manage the required sulfide plant feed blend. It is likely that there will need to be a modification of the stockpiling cut-offs and procedures for both short-term and longer term blending, such as increasing the number of active mining areas, increasing the mining rate, and increasing the size or number of ROM stockpiles.

15.3 Comparison of 2020 Mineral Reserve to 2019 Mineral Reserve

A summary of the entire CDMP20 Mineral Reserves is shown in Table 15.1

The differences between the CDMP20 Mineral Resources and the previous Mineral Resources reported as at 31 December 2019 are shown in Table 15.2 for each deposit, material type, and classification.

The differences are a function of the following changes:

- New designs for two new phases beneath the Çöpler pit
- Reduction in cut-off grades from the increased throughput provided by the flotation circuit, reduced unit costs, and increased gold price
- Review of metallurgical recoveries
- Review of Çakmaktepe North
- Depletion through mining since 31 December 2019

Overall, there has been a 39% increase in tonnage above the cut-off across both Mineral Reserve categories, with a corresponding 22% increase in contained gold.

A review of the Çakmaktepe North Mineral Reserve included in the End of Year 2019 Mineral Reserve Statement, suggested it should be removed from Mineral Reserve due to the high stripping ratio and complex pit design requirements. Çakmaktepe North should be studied in the future and revaluated using different parameters and assumptions to determine if it is suitable to be included in the Mineral Reserve. Çakmaktepe North remains in the Mineral Resource estimate.

Table 15.2 CDMP20 Mineral Reserves Compared to 2019 Mineral Reserves

Relative Difference between CDMP20 Mineral Reserves (as at the Effective Date) and 2019 Mineral Reserves (as at 31 December 2019)							
Classification	**Tonnage (kt)**	**Grades**			**Contained Metal**		
		Au (g/t)	**Ag (g/t)**	**Cu (%)**	**Gold (koz)**	**Silver (koz)**	**Copper (klb)**
Çöpler Mine – Oxide							
Proven Mineral Reserve	New to 2020						
Probable Mineral Reserve	23%	12%	-2%	29%	37%	21%	59%
Probable – Stockpile	–100%	–100%	n/a	n/a	–100%	n/a	n/a
Çöpler Oxide Proven + Probable	27%	12%	0%	27%	41%	26%	61%
Çöpler Mine – Sulfide							
Proven Mineral Reserve	New to 2020						
Probable Mineral Reserve	+50%	–16%	–14%	New to 2020	+25%	+28%	New to 2020
Probable – Stockpile	–3%	–5%	n/a	n/a	–7%	n/a	n/a
Çöpler Sulfide Proven + Probable	+46%	–15%	–6%	New to 2020	+24%	+37%	New to 2020
Çakmaktepe – Oxide							
Proven Mineral Reserve	n/a	n/a	n/a	n/a	n/a	n/a	n/a
Probable Mineral Reserve	–79%	–41%	+8%	n/a	–88%	–78%	n/a
Probable – Stockpile	0%	0%	n/a	n/a	0%	n/a	n/a
Çakmaktepe Oxide Proven + Probable	–79%	–38%	+5%	n/a	–87%	–78%	n/a
CDMP20 Oxide Reserve							
Proven Mineral Reserve	New to 2020						
Probable Mineral Reserve	+5%	–4%	–9%	+29%	0%	–5%	+30%
Probable – Stockpile	–100%	–100%	n/a	n/a	–100%	n/a	n/a
CDMP20 Oxide Proven + Probable	+7%	–4%	–8%	+28%	+3%	–1%	+32%
CDMP20 Sulfide Reserve							
Proven Mineral Reserve	New to 2020						
Probable Mineral Reserve	+50%	–16%	–14%	New to 2020	+25%	+28%	New to 2020
Probable – Stockpile	–3%	–5%	n/a	n/a	–7%	n/a	n/a
CDMP20 Sulfide Proven + Probable	+46%	–15%	–6%	New to 2020	+24%	+37%	New to 2020
Overall CDMP20 Reserve							
Proven Mineral Reserve	New to 2020						
Probable Mineral Reserve	+40%	–13%	–14%	+380%	+22%	+21%	+30%
Probable – Stockpile	–3%	–4%	n/a	n/a	–7%	n/a	n/a
Overall CDMP20 Proven + Probable	+39%	–12%	–7%	+398%	+22%	+29%	+32%

'n/a' indicates that the value was reported in neither 2019 nor 2020
'New to 2020' indicates that material of this category is reported in 2020 but there was no equivalent material reported in 2019

15.4 Other Mineral Reserve Reporting

15.4.1 US SEC Industry Guide 7

The Mineral Reserves reported for NI 43-101 are also applicable for reporting the Ore Reserve under the US SEC Industry Guide 7. OreWin estimated the Çöpler project Mineral Reserves for the NI 43-101 CDMP20 Technical Report, which are based on work at a feasibility study level. The definitions of the Mineral Reserve classifications under NI 43-101 are the Canadian Institute of Mining (CIM) Definition Standards – For Mineral Resources and Mineral Reserves, prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on 11 December 2005. The definitions below are quoted from the CIM Definition Standards – For Mineral Resources and Mineral Reserves, page 5.

After consideration of guidelines and other information regarding the declaration of Reserves for the United States Securities and Exchange Commission (US SEC) reporting, OreWin considers that the CDMP20 Feasibility Study is suitable for declaring a Reserve as defined in US Industry Guide 7.

Documentation underlying Mineral Reserves determined in accordance with Industry Guide 7 generally includes the following:

- A 'final' feasibility study.
- Utilisation of the historical three-year average price for the commodity that is expected to be mined in determining economic viability.
- Primary environmental analysis has been submitted to government authorities.

15.4.1.1 Bankable Study

CDMP20 is a bankable feasibility study that supports the project finance SSR Mining has for the project. The project is in operation and the detailed work of the Reserve Case demonstrates the Mineral Reserve is viable. The finance facility has been provided by a syndicate of international financial institutions and export credit agencies representing the governments of Canada, the United States and Australia, along with 15 commercial banks. Drawdown of the loan has been completed. OreWin therefore considers it reasonable to conclude that the bankable study test in US SEC Industry Guide 7 has been met.

15.4.1.2 Test Price for Commodities

The Base Case economic analysis has been prepared using current long-term metal price estimates of:

- Gold $1,370/oz
- Silver $16.33/oz
- Copper $2.84/lb

The 2005 SME Guide Section 53 describes how the Test Price for commodities should be applied.

"If a Mineral Reserve is reported using a price lower than the test price, the forward-looking discounted cash flow must be positive, and the Reserve Sensitivity Test (based on an undiscounted cash flow) need not be performed. When applicable, a statement should be made that a Reserves Sensitivity Test was completed, or that such a test was not applicable."

The metal prices for the previous three years, the three-year trailing averages and the metal prices used for the Base Case Financial Analysis are shown in Table 15.3. The sensitivity analysis using the 3-year trailing averages shows the after-tax $NPV_{5\%}$ is $1,052M and demonstrates the forward-looking discounted cash flow is still positive for those prices.

Table 15.3 Metal Price Summary

Year Ended	Gold ($/oz)	Silver ($/oz)	Copper ($/lb)
Annual Average Metal Prices			
2018	1,285	16.83	3.14
2019	1,264	15.25	2.82
2020	1,562	16.90	2.57
3-Year Trailing Average	1,370	16.33	2.84
Base Case Financial Analysis			
Q4'20	1,850	20.05	2.70
2021	1,965	24.15	2.90
2022	1,835	22.70	2.90
2023	1,745	21.80	2.95
2024	1,645	20.75	3.00
2025 onwards	1,585	20.25	3.05

15.4.1.3 Primary Environmental Analysis Submission

The 2007 SME Guide Section 56 describes how the permitting and legal requirements of US SEC Industry Guide 7 should be applied. It indicates that:

"To demonstrate reasonable expectation that all permits, ancillary rights and authorizations can be obtained, the reporting entity must show understanding of the procedures to be followed to obtain such permits, ancillary rights and authorizations. Demonstrating earlier success in getting the necessary permits can be used to document the likelihood of success."

Operation of the Çöpler mining and processing facilities, and subsequent mining at Çakmaktepe, have been investigated and authorised by means of a series of EIAs, with positive decisions obtained from the Turkish Ministry of Environment and Urban Planning (MEUP). These EIA's include specific actions designed to address all material impacts of the mining and processing operations. Anagold has remained in compliance with all aspects of the EIA and operating permits throughout the history of the project. SSR Mining has completed a comprehensive Environmental and Social Impact Assessment (ESIA) for Çöpler. The culmination of many years of independent work and research carried out by both international and Turkish experts, the ESIA identifies and assesses the potential environmental and social impacts of the project, including cumulative impacts, focusing on key areas such as biodiversity, water resources, cultural heritage, and resettlement. The ESIA also sets out measures through all project phases to avoid, minimise, mitigate, and manage potential adverse impacts to acceptable levels established by Turkish regulatory requirements and good international industry practice.

It is considered reasonable to assume that the environmental permitting will continue to remain in place and without resulting in a change to the CDMP20 Mineral Reserve.

16 MINING METHODS

The objective of the CDMP20 is to provide a consistent and structured growth plan for the business. Mine plans were updated to improve metal production, through a revised development sequence, available information was then consolidated into a growth strategy, for communication to all levels of the business, using recognised progress reporting systems.

16.1 Geotechnical

16.1.1 Pit Slope Stability – Çöpler

This section contains a summary of the feasibility study level mining geotechnical investigation and design conducted for the Çöpler mine. Much of this work has been prepared prior to 2020, the work and the recommendations are still applied to the mine designs and workings.

The Çöpler mine maintains an on-site geotechnical monitoring programme that consists of 58 prisms, 33 extensometers, a long-range synthetic aperture radar, and daily data and field monitoring. Additional work is currently in progress to implement pit slope depressurisation. It is expected that pit slope depressurisation will be used extensively throughout the Main pit as the sulfide pit phases are progressed.

In April 2015, Golder Associates (Golder) completed a pit slope optimisation study intended to further optimise the pit slope angles as defined in their earlier study completed in April 2014. This programme included the drilling of five oriented geotechnical core holes to identify any prevalent jointing throughout the Çöpler deposit.

Golder completed the 2015 pit slope optimisation study using recommendations from the 2014 Golder pit slope review with the intention of identifying opportunities to increase definition of potential problem areas within the Çöpler pit to allow for mine planning and design to take advantage of steeper slope angles in some areas. No material changes in pit slope recommendations were made with the updated report. Anagold chose to continue using the more conservative slope angle recommendations made by Golder in 2014.

The results of the study have provided SSR Mining with a much better definition of potential highwall conditions. Not all slope angle recommendations made by Golder were able to be fully followed due to a lack of data and modelling of alteration zones within the Çöpler deposit. Where slope angles were not able to be further refined, Golder recommended that SSR Mining follow the recommendations set forth in the 2014 geotechnical review.

16.1.2 RQD Model

RQD is used as a simple and inexpensive indication of rock mass quality. RQD does not account for joint orientation, continuity, or gouge material. Joints sets parallel to the core axis will not intersect the core and therefore is it recommended to use RQD in combination with other geotechnical inputs. RQD is a measure of percent core recovery with artificial fractures ignored.

At the Çöpler project, it has been determined that RQD is a generally reliable indicator of alteration. Therefore, areas with RQD modelled as being less than 15% are considered altered.

Standard testing of RQD was collected on 661 core holes, 30 of which were drilled within the pit for metallurgical purposes. The 661 holes represent approximately 34% of all drilling in the Çöpler deposit. The Main pit contains RQD measurements for holes evenly spaced with data gaps occurring in the Manganese, Marble, and West pits.

RQD was interpolated in the resource model using the inverse distance method, weighted to the power of two (ID2) with 2 m drillhole composites. A total of six domains were used to estimate RQD values and included a distinction between oxide and sulfide material. To account for the variance in sample spacing, a two-pass approach was used to capture available samples. Model cell estimates were limited to the search distances used with no attempt to assign RQDs to un-estimated cells.

16.1.3 Pit Slope Design Parameters

The pit slope design parameters remain unchanged and those applied for each deposit are shown in Table 16.1. Note that for Çakmaktepe design parameters are in relation to the Central pit and based on the 2018 Golder study which are defined based on azimuth (i.e. direction the slope faces).

Table 16.1 Çöpler Mine Pit Slope Parameters

Çöpler Rock Type	Interramp Slope Angle Çöpler Pits	
	Altered RQD<15%	Un-altered (Fresh) RQD>15%
Diorite	23	38
Metasediment	32	43
Marble	50.5	50.5
Gossan Massive Sulfides	40	40

Çakmaktepe Slope Direction	Interramp Slope Angle Çakmaktepe Central Pits
0° to 180° (south-west wall)	34
180° to 360° (all other walls)	40

Golder site review, Çöpler and Golder 2018 for Çakmaktepe

16.1.4 Mine Operations Monitoring and Management

Pit slopes in the Çöpler pit are monitored daily to ensure safety and stability. Daily inspections of the active mining areas are conducted by shift engineers to identify hazards such as unstable rock on benches above, excessive water in and around the highwalls, and any visible cracking and movement of the highwalls. In addition, Anagold employs a geotechnical management team consisting of surveyors, geologists, and geotechnical engineers. This team conducts regular highwall inspections, measurement of movement through extensometers and prism surveys, and data collection and interpretation of the long-range synthetic aperture radar measurements.

Mining at Çöpler utilises perimeter pre-split blasting techniques in areas where competent rock is encountered (typically, limestone/marble, unaltered metasediment, and unaltered diorite). The pre-split holes are drilled according to the bench face angle recommendations as shown in Table 16.1. Blasting is conducted in a manner to minimise back-break through delays and adequate relief. A typical pre-split highwall at Çöpler is shown in Figure 16.1.

Where pre-splitting is not practical, highwalls are sloped by excavator to the recommended bench face angle. A typical bench face without pre-splitting is shown in Figure 16.2.

Figure 16.1 Typical 15 m Pre-Split at Çöpler Mine

Anagold, 2020

Figure 16.2 Typical Bench Face without Pre-Split

Anagold, 2016

16.1.5 Geotechnical Domains

Based on the 2014 Golder geotechnical site review, the following geotechnical domain categories are considered appropriate for design recommendations to be founded upon:

- Marble / Limestone – characterised by competent rocks and marbleised near the Çöpler intrusion.
- Fresh diorite – characterised as a fresh to slightly weathered or altered moderately strong rock.
- Hydrothermally altered diorite – alteration sufficient to significantly reduce strength relative to fresh diorite, but without the shearing and intense clay alteration of contact and fault zones.
- Weathered diorite and metasediment – highly weathered, extremely weak rock and soil that occurs in the oxidised zone (depth typically to 30 m).
- Fresh metasediment – fresh to slightly weathered, weak to moderately strong rock consisting of a turbidite sequence that may also be structurally complex near faults.
- Hydrothermally altered metasediment – alteration sufficient to significantly reduce strength relative to fresh metasediment, but without the shearing and intense clay alteration of Contact and Fault zones.
- Fault gouge including intrusive contact and intense sulfide alteration – slicken sided plastic clay with rock fragments that occurs in fault zones including the intrusive contacts.

The character and extent of the hydrothermal alteration beyond the fault zones is poorly defined. Where data are lacking within the alteration zones the most conservative pit slope angle is assumed, representing up-side potential should the alteration zone be further defined in the geological model.

The above listed geotechnical domains are mostly well known and modelled in a geologic model. The alteration zones, however, vary significantly and have not been modelled to an extent to where variations by alteration type are well defined. It has been recommended by Golder that the best way to identify alteration zones is by modelling RQD in the geologic model. For this purpose, RQD values of 15% and less are considered altered and RQD values greater than 15% are considered un-altered, or fresh.

16.1.6 Pit Dewatering

Earlier studies have predicted the formation of pit lakes at various stages of mining. Golder's hydrogeological study was used to predict pit lake formation. The groundwater flow model predicted that a pit lake would form over time after mining. These results, in conjunction with the acid rock drainage (ARD) work being conducted by SRK Turkey, are being used to predict pit lake water quality.

Sources of groundwater recharge include direct infiltration of precipitation and/or infiltration during storm water run-off events throughout the entire site. Fractured or karstic openings in the bedrock and alluvial sediments along drainages are considered to be the predominant pathways for infiltration. The main hydrogeological units and features considered in the groundwater model were:

- Munzur limestone (modelled hydraulic conductivity = 0.6 m/day)
- Diorite (modelled hydraulic conductivity = 0.0002 m/day)
- Metasediments (modelled hydraulic conductivity = 0.0002 m/day)
- Alluvium (modelled hydraulic conductivity = 10 m/day)
- Various fault systems (Sabirli, Çöpler, and Other) (modelled hydraulic conductivity = 6.1 m/day)

The calibrated groundwater model was used to predict pit inflows and pit lake development based on a pit design with a maximum depth to 875 m. This analysis estimated pit inflow at less than approximately 1,100 m^3/day. Estimations of pit lake formation suggest that over a 100-year scenario, based on a pit design with a maximum depth to 875 m, pit lake water elevations are projected to reach the 906 m elevation (±20 m). Modelling results indicate that water from beneath the Lower Çöpler West waste rock dump (WRD) will take more than 1,000-years to flow to the Karasu River. Groundwater located beneath the Lower Çöpler East WRD is estimated to discharge to the Karasu River within approximately 300-years.

Revisions to the pit design since the groundwater model was constructed and calibrated (in 2012) show that the minimum pit elevation (895 mRL) will be higher than the minimum pit elevation simulated in the model (875 mRL). Additionally, the area on the north side of the pit and the southern and south-eastern portions of the pit will be mined to a lower elevation than simulated in the model. Limestone in these areas may increase discharge to the pit during dewatering and may impact the formation of a pit lake following closure. Updating and possibly recalibrating the model based on the revised ultimate pit configuration and available data since 2012 would be required to better quantify the magnitude of the increase or impact.

16.2 Mine Plan

SSR Mining carried out the mine planning and scheduling work for the Çöpler open pits. The Reserve Case is an update of the work previously called the Çöpler Sulfide Expansion Project. Pit designs from the 2016 Technical Report have been mined since 2016 and there is still significant ore remaining within the designs. In 2020 two additional phases on the main zone area were designed and included in the Mineral Reserve. Sulfide flotation has been added to the sulfide processing assumptions, along with associated infrastructure. Production schedules and costs have been updated based on current site performance and contracts.

The mine plan has a start date of October 2020 and schedules 51.9 Mt of ore, along with 179.2 Mt of waste, in three pit phases from the Çöpler deposit and the final phase of the Çakmaktepe deposits. There is currently only 285 kt remaining at the Çakmaktepe deposit, therefore most of the remaining mining will be at the Çöpler deposit. The open pit Mineral Reserves are mined over approximately 21 years.

The optimisation for the additional phases, Reserve Case pit designs, and production scheduling were completed using Measured and Indicated Mineral Resources only, with Inferred Mineral Resources treated as waste.

The parameters, costs and throughput assumptions used to prepare cut-off grades and the production schedule are listed in the following sections.

16.2.1 Ore Definition

A revised set of processing parameters was used to calculate the internal Au cut-off grades for ore definition. The cut-off grades for the CDMP20 were calculated using the parameters described in the following sections.

16.2.1.1 Oxide Heap Leach Parameters

Table 16.2 details the gold recovery parameters by material type and location.

Table 16.2 Heap Leach – Gold Recoveries

Location	Unit	Material Types						
		LMS	META	GOSS	JAS	DIO	MNDIO	OPH
Çöpler Manganese	%	78.4	66.8	71.2	–	71.2	71.2	–
Çöpler Main	%	68.6	66.8	71.2	–	71.2	71.2	–
Çöpler Marble	%	75.7	66.8	65.1	–	62.3	62.3	–
Çakmaktepe Central	%	70.0	80.0	–	73.0	61.0	–	70.0

Table 16.3 details the silver recovery parameters by material type and location.

Table 16.3 Heap Leach – Silver Recoveries

Location	Unit	Material Types						
		LMS	META	GOSS	JAS	DIO	MNDIO	OPH
Çöpler Manganese	%	27.3	32.5	27.5	–	37.8	37.8	–
Çöpler Main	%	24.6	32.5	27.5	–	37.8	37.8	–
Çöpler Marble	%	34.0	32.5	27.5	–	32.0	32.0	–
Çakmaktepe Central	%	17.0	28.0	–	17.0	24.0	–	19.0

Table 16.4 details the copper recovery parameters by material type and location.

Table 16.4 Heap Leach – Copper Recoveries

Location	Unit	Material Types						
		LMS	META	GOSS	JAS	DIO	MNDIO	OPH
Çöpler Manganese	%	3.5	13.8	3.3	–	15.8	15.8	–
Çöpler Main	%	3.5	13.8	3.3	–	15.8	15.8	–
Çöpler Marble	%	3.5	13.8	3.3	–	15.8	15.8	–
Çakmaktepe	%	–	–	–	–	–	–	–

Table 16.5 details the operating costs by location.

Table 16.5 Oxide Operating Costs

Parameter	Unit	Çöpler	Çakmaktepe
Rehandle Cost	$/t	0.32	0.64
Processing – Fixed	$/t	3.05	3.05
Processing – Variable	$/t	8.94	8.94
G&A (Process and Site)	$/t	3.17	3.17
Ore Haulage	$/t	–	1.53
Mining Cost	$/t mined	1.89	1.59

16.2.1.2 Sulfide Plant Parameters

The following sections outline the processing parameters for the sulfide plant. Average LOM sulfide gold recoveries are 91%.

Throughput

Total Plant Throughput = Direct POX Feed + Float Plant Feed

POX Plant Throughput = Direct POX Feed + Float Plant Concentrate

Table 16.6 details the maximum plant throughputs for each part of the plant. The front-end limit of 400 tph means when the flotation plant is running at full capacity (i.e. 150 tph), the direct feed to the POX circuit will be limited to 250 tph.

Table 16.6 Plant Throughput Limits

Parameter	Unit	Maximum Throughputs
Float Plant	t/hr	150
POX Plant	t/hr	280
Total	t/hr	400

Float Plant Throughput = 216345 x Feed SS%^{2} – 30592 x Feed SS% + 980.24

Float Concentrate Mass Pull = 277.09 x Feed SS%^{2} – 15.17 x Feed SS% + 0.33

The POX circuit throughput is also limited by the sulfide sulfur (SS) in the feed to the autoclave, which must be less than 13.75 tph. If the SS content is too high, then the POX circuit throughput will need to be reduced until the rate is less than 13.75 tph SS.

Recovery – POX Gold

POX Gold Recovery = a x (1 - EXP(- b x (Au(g/t) - c))) + d.

Table 16.7 details the POX gold recovery factors by material type.

Table 16.7 POX – Gold Recovery Parameters

Material Type	a	b	c	d
Limestone/Marble	98.3	1.4	–1.5	–1.00
Metasediment	97.7	1.4	–1.4	–1.00
Gossan	98.3	1.4	–1.5	–1.00
Jasperoid	98.3	1.4	–1.5	–1.00
Diorite	98.3	1.4	–1.5	–1.00
Mn Diorite	96.7	1.2	–1.4	–1.00
Ophiolite	98.3	1.4	–1.5	–1.00

Recovery – Float Plant

Float Concentrate Gold Recovery = 55%

Float Tails Gold Recovery = 43%

Float Concentrate SS Recovery = 75%

Table 16.8 details the operating costs by location

Table 16.8 Sulfide Operating Costs

Parameter	Unit	Amount
Rehandle Cost	$/t	0.90
Processing – Fixed	$/t	8.32
Processing – Variable	$/t	19.10
Processing – Variable (SS)	$/t SS	2.68
G&A (Process and Site)	$/t	6.60

16.2.1.3 Metal Prices and Realisation Assumptions

Cut-off grades were determined using a gold price of $1,350/oz. There are no credits for silver or copper in the cut-off grade calculations. Table 16.9 details revenue and realisation assumptions for the Au cut-off grades.

Table 16.9 Au Cut-off Grade Revenue and Realisation Assumptions

Parameter	Unit	Au Cut-off Assumption
Payment and Deductions		
Gold	$/oz	1,350
Payable	%	100
Treatment and Refining		
Selling	$/oz	8.54
Royalties		
Çöpler	%	2
Çakmaktepe	%	4

16.2.2 Ore Cut-off Grades

Internal cut-off grades have been calculated for each of the material types based on the economic inputs and assumptions outlined in Section 16.2.1 and are shown in Table 16.10. Internal cut-off grades have been used to calculate process quantities within the Reserve pit stages.

The addition of the flotation circuit to the sulfide plant requires new grade control protocols and a new associated stockpile strategy will be implemented to manage the required sulfide plant feed blend. It is likely that there will need to be a modification of the stockpiling cut-offs and procedures for both short-term and longer term blending, such as increasing the number of active mining areas, increasing the mining rate, and increasing the size or number of ROM stockpiles.

Table 16.10 Internal Au Cut-off Grades

Mining Area	Ore Type	Rock Type	Zone	COG (Au g/t)
Çöpler	Oxide	Limestone/Marble	Manganese	0.47
			Main	0.53
			Marble	0.48
		Metasediment	Manganese	0.55
			Main	
			Marble	
		Gossan	Manganese	0.51
			Main	0.51
			Marble	0.56
		Diorite	Manganese	0.51
			Main	0.51
			Marble	0.59
		Mn Diorite	Manganese	0.51
			Main	0.51
			Marble	0.59
	Sulfide	All	All	1.05
Çakmaktepe	Oxide	Limestone/Breccia	Central	0.60
		Jasperoid		0.57
		Diorite		0.69
		Metasediment		0.52
		Ophiolite		0.60

16.2.3 Pit Design

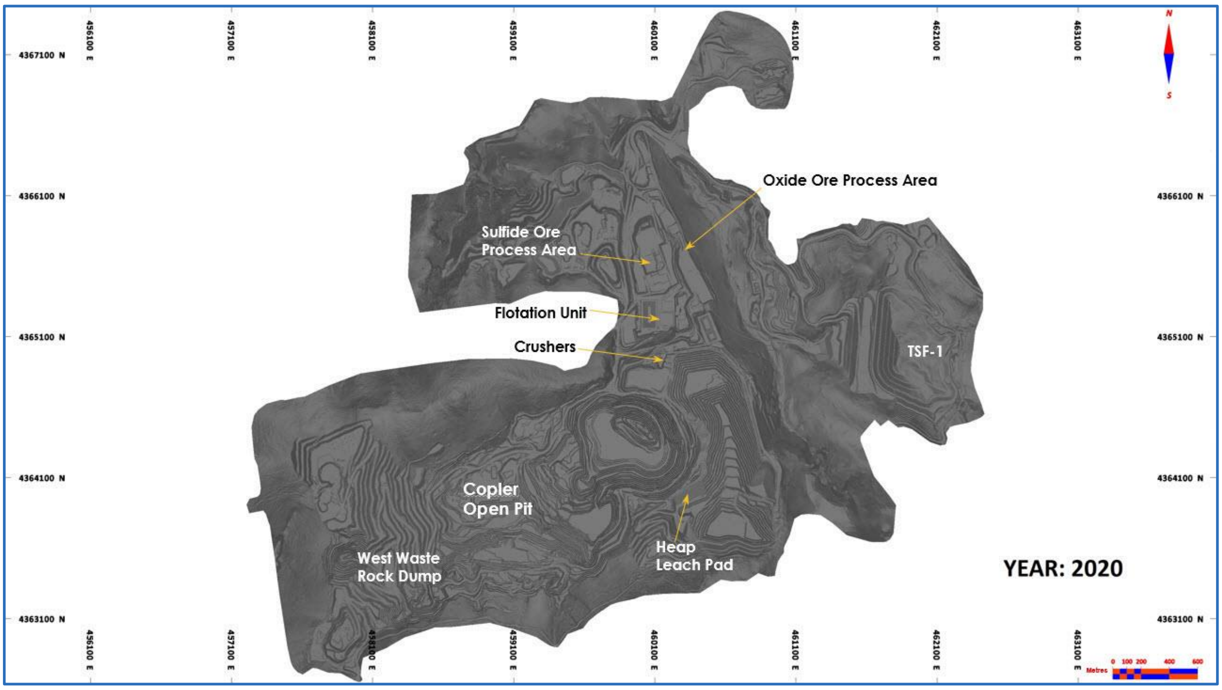
Pit designs from the 2016 Technical Report have been mined since 2016 and there is still significant ore remaining within the designs. In 2020 two additional phases on the Main Zone area were designed and included in the Mineral Reserve. The 2020 pit optimisation work on Çöpler indicated there was more Mineral Resource below the Çöpler pits that may be included in designs. It is recommended that a comprehensive review, optimisation and design of Çöpler be undertaken to evaluate the potential for increased Mineral Reserves.

The key aims of the pit designs are:

- Minimise mining costs and maximise economic return by exposing the highest value ore with minimum waste mining.
- Address operational requirements for loading, hauling, slope stability, and rockfall, as follows:
  - Loading – the phases were designed with a minimum operational width of 15–30 m between phases (depending on bench configuration) to allow efficient mining for the equipment scale.
  - Hauling – generally, two exit haul roads per phase were included: the west-bound exit to the crusher, low-grade stockpile, and west dump; and the east-bound exit to the potentially acid forming (PAF) and non-acid forming (NAF) dumps. Haul roads were generally 15 m wide at a 10% gradient. Single-lane haulage traffic is allowed in the lower benches of the mine and is set at 10 m wide.

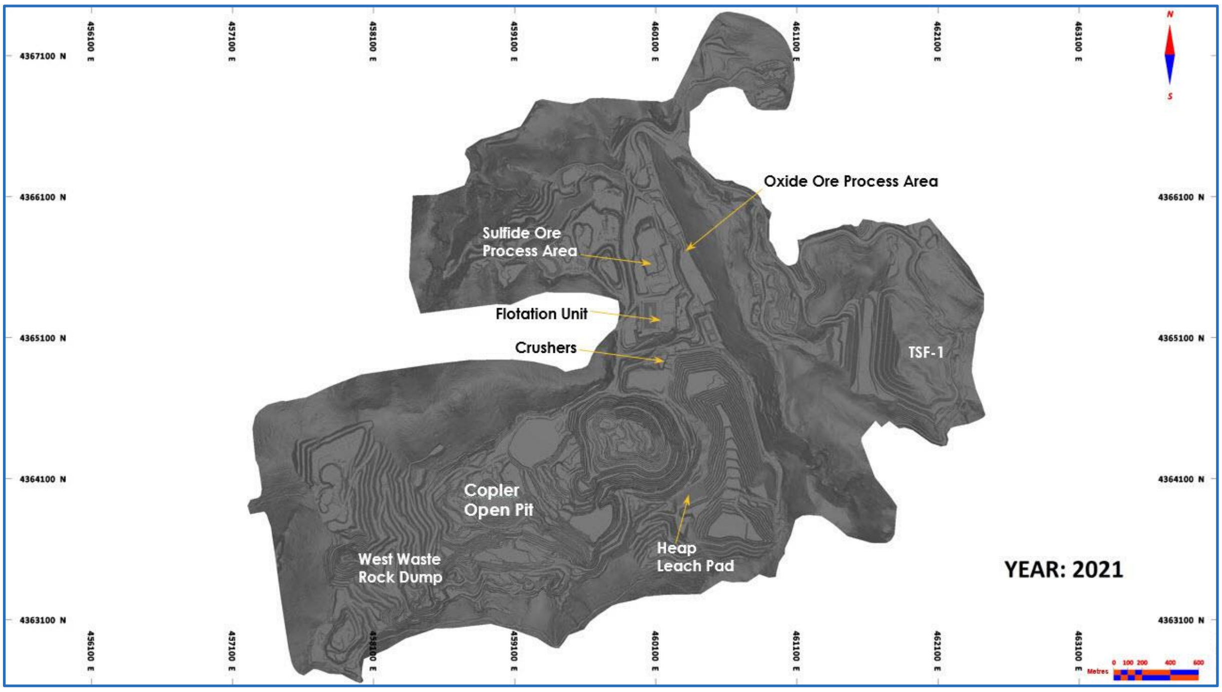
Figure 16.3 illustrates the current surface topography for Çöpler, including the processing plants. Plans showing the annual face positions for Çöpler are in Figure 16.3 through Figure 16.15.

Figure 16.3 2020 Pit Plan – Çöpler



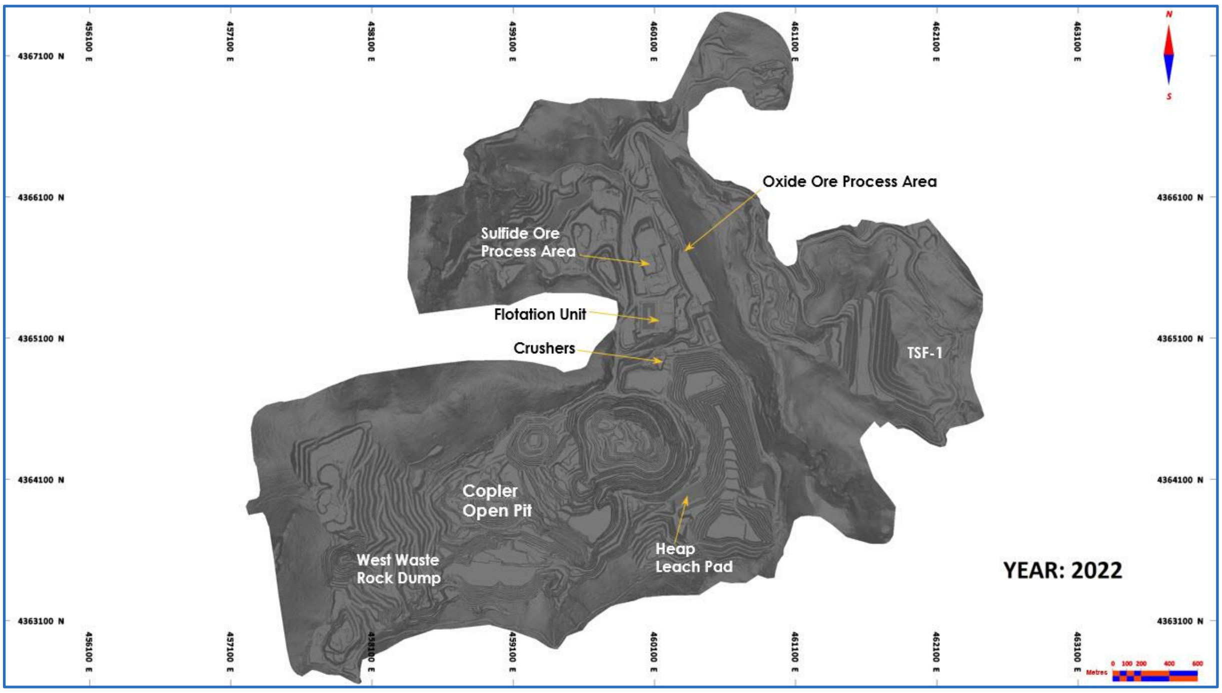
Anagold, 2020

Figure 16.4 2021 Pit Plan – Çöpler



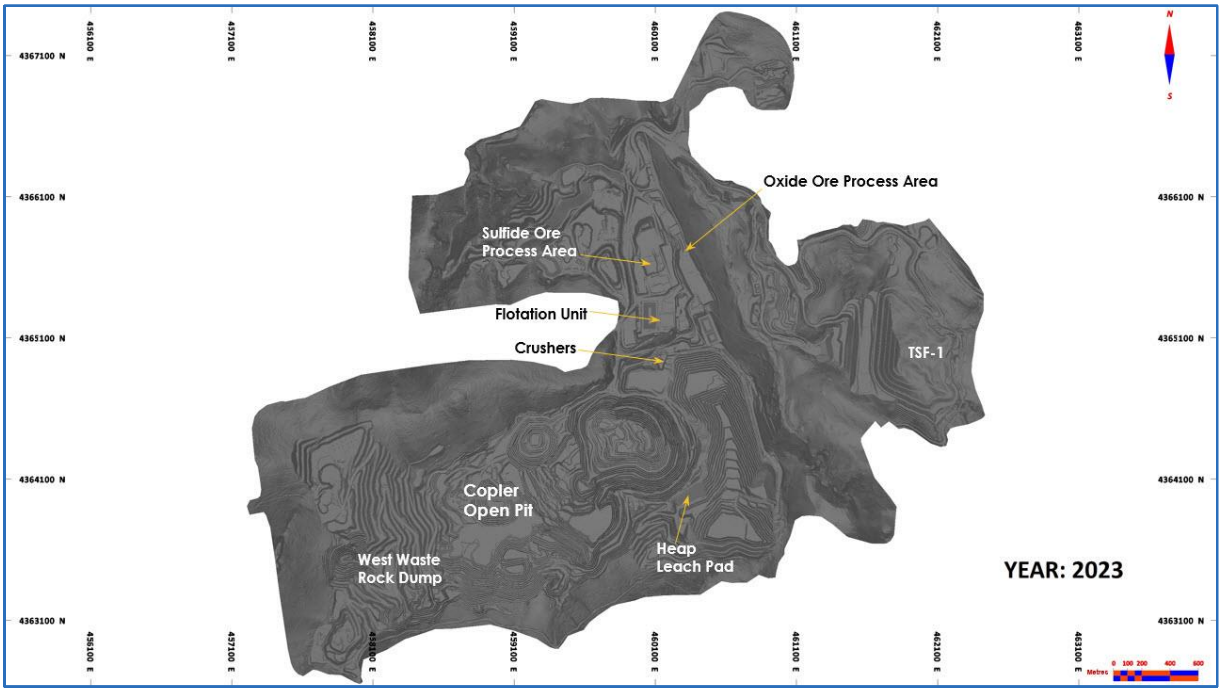
Anagold, 2020

Figure 16.5 2022 Pit Plan – Çöpler



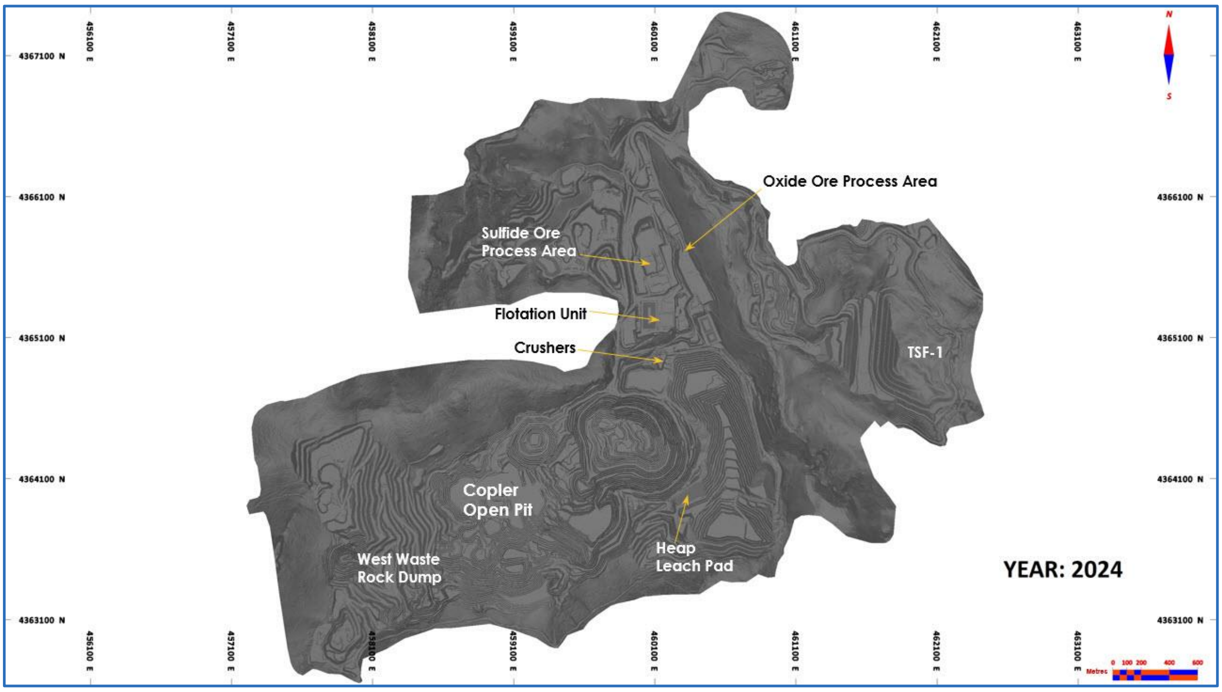
Anagold, 2020

Figure 16.6 2023 Pit Plan – Çöpler



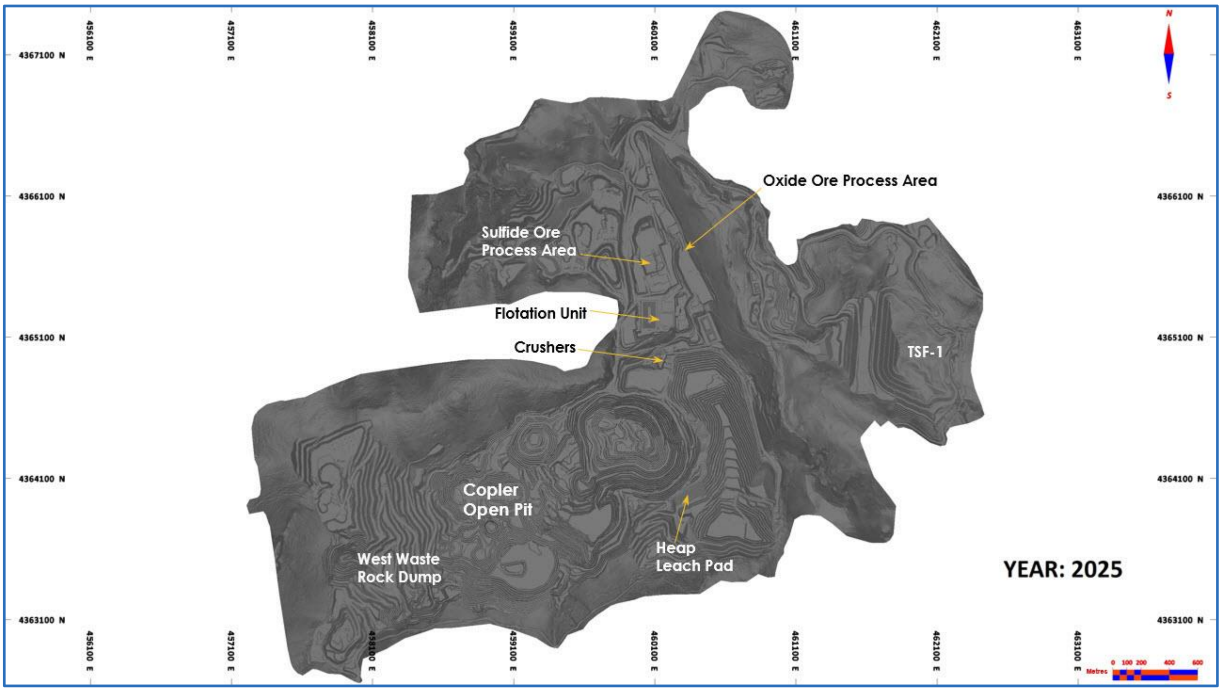
Anagold, 2020

Figure 16.7 2024 Pit Plan – Çöpler



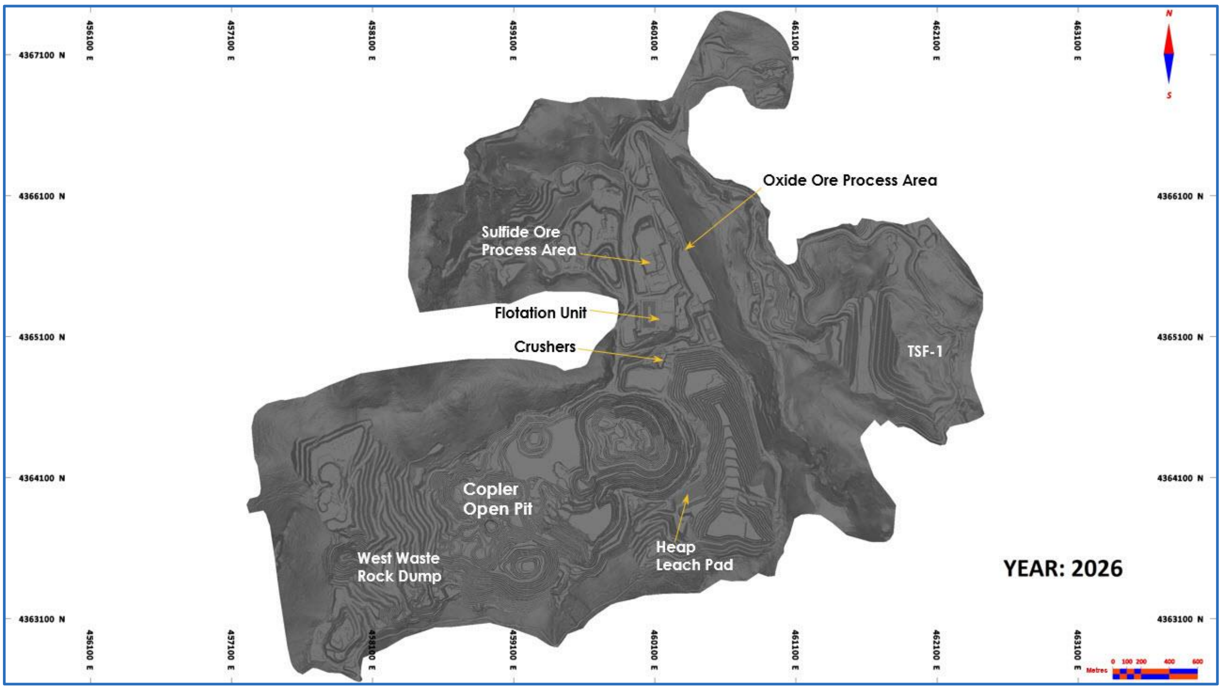
Anagold, 2020

Figure 16.8 2025 Pit Plan – Çöpler



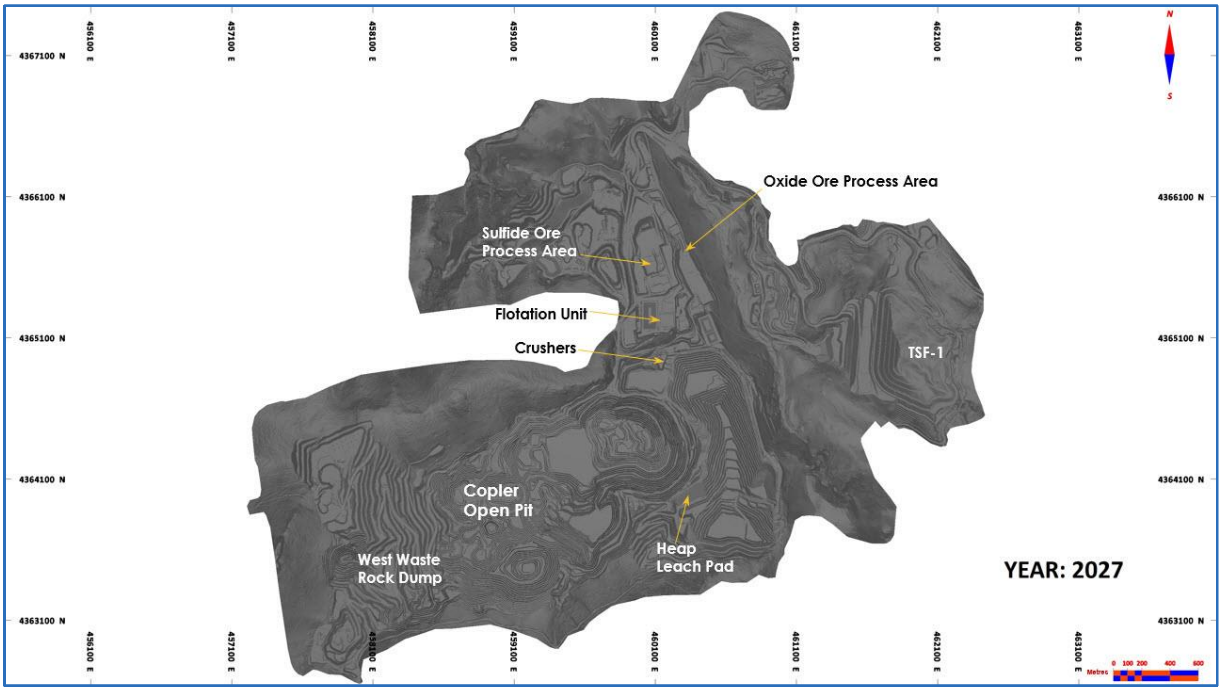
Anagold, 2020

Figure 16.9 2026 Pit Plan – Çöpler



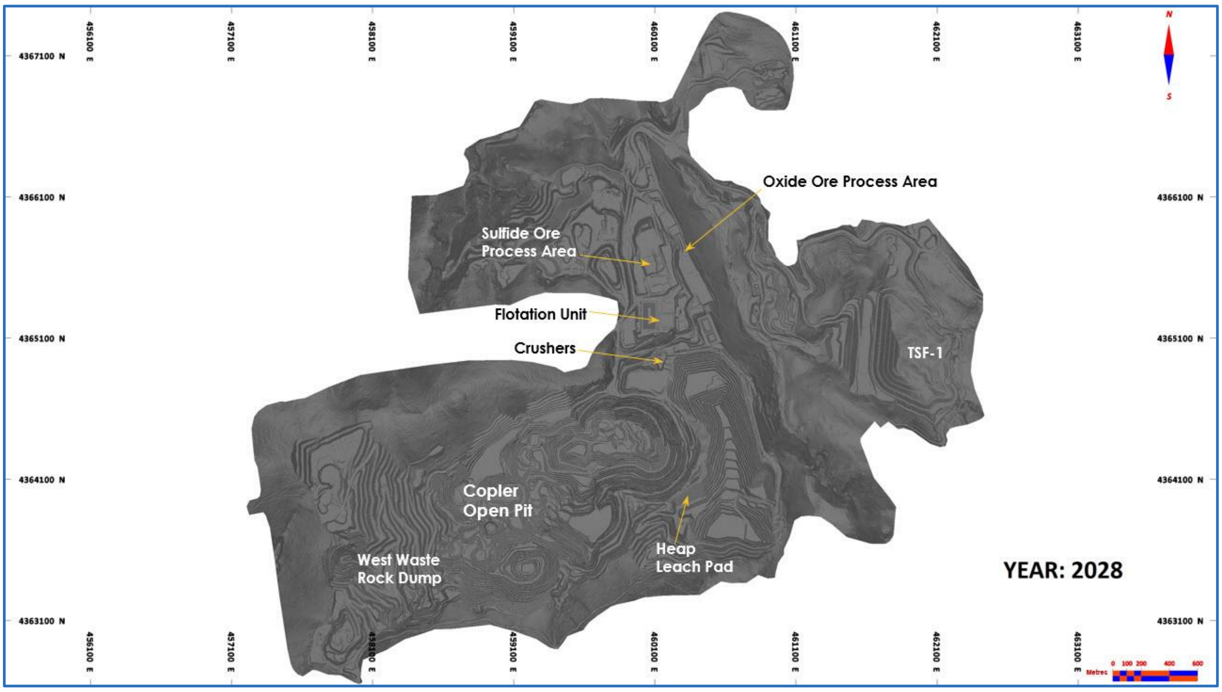
Anagold, 2020

Figure 16.10 2027 Pit Plan – Çöpler



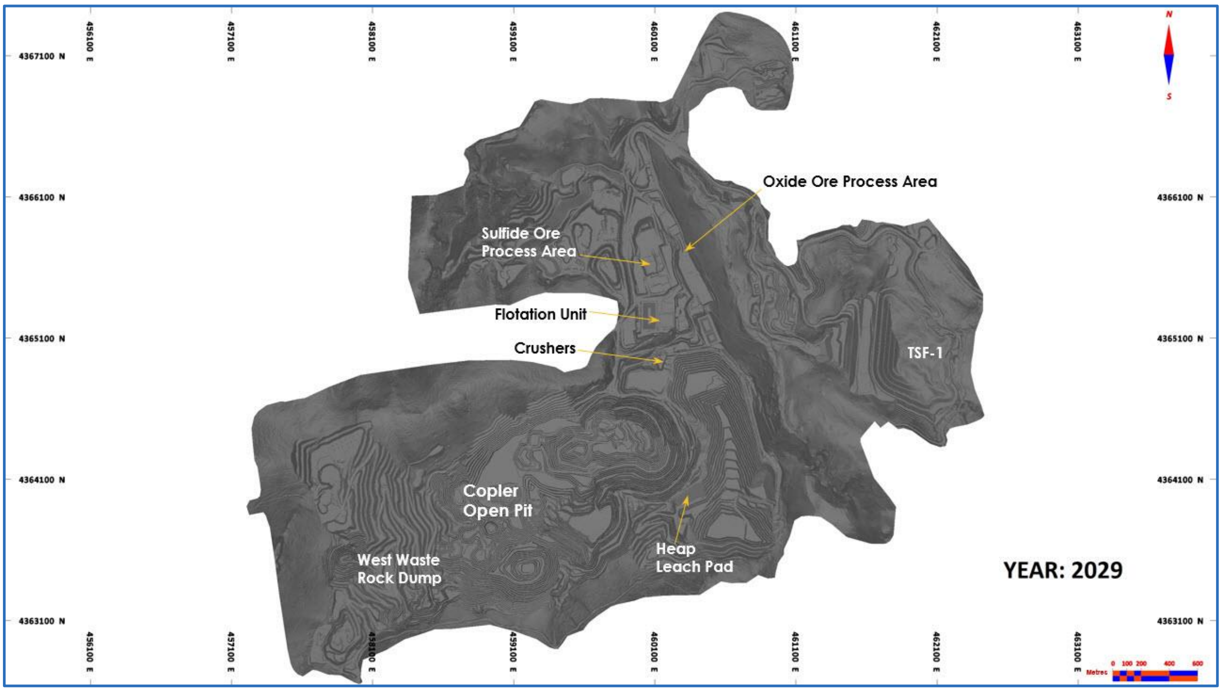
Anagold, 2020

Figure 16.11 2028 Pit Plan – Çöpler



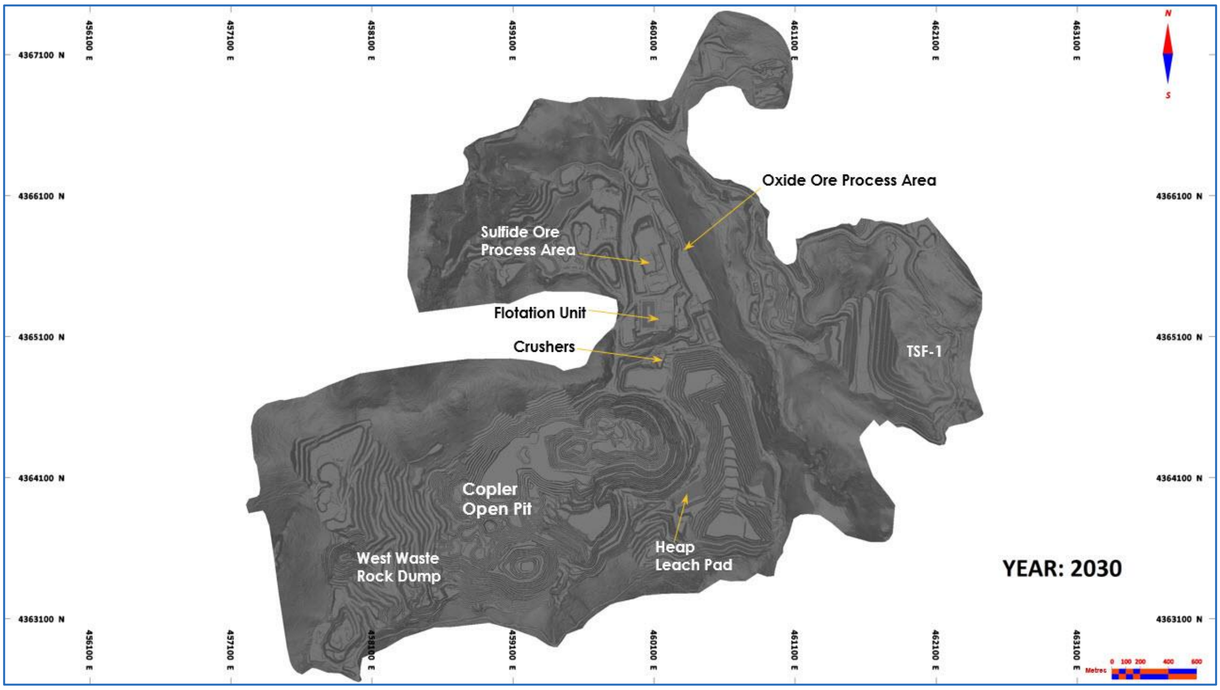
Anagold, 2020

Figure 16.12 2029 Pit Plan – Çöpler



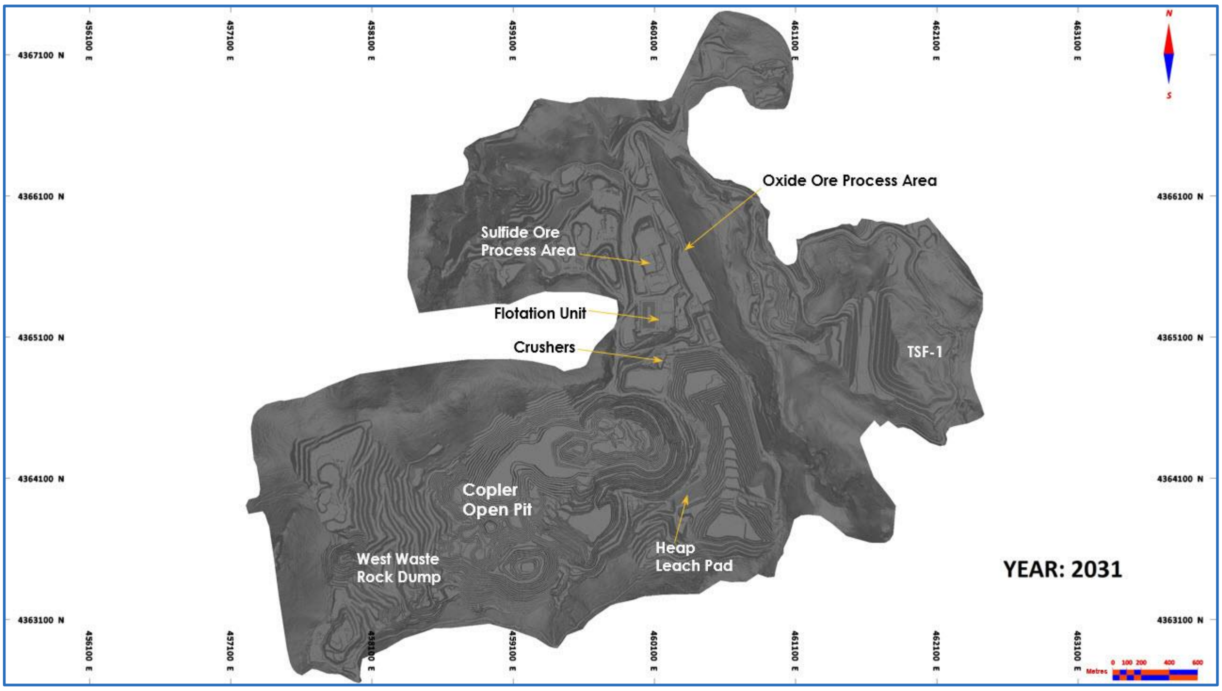
Anagold, 2020

Figure 16.13 2030 Pit Plan – Çöpler



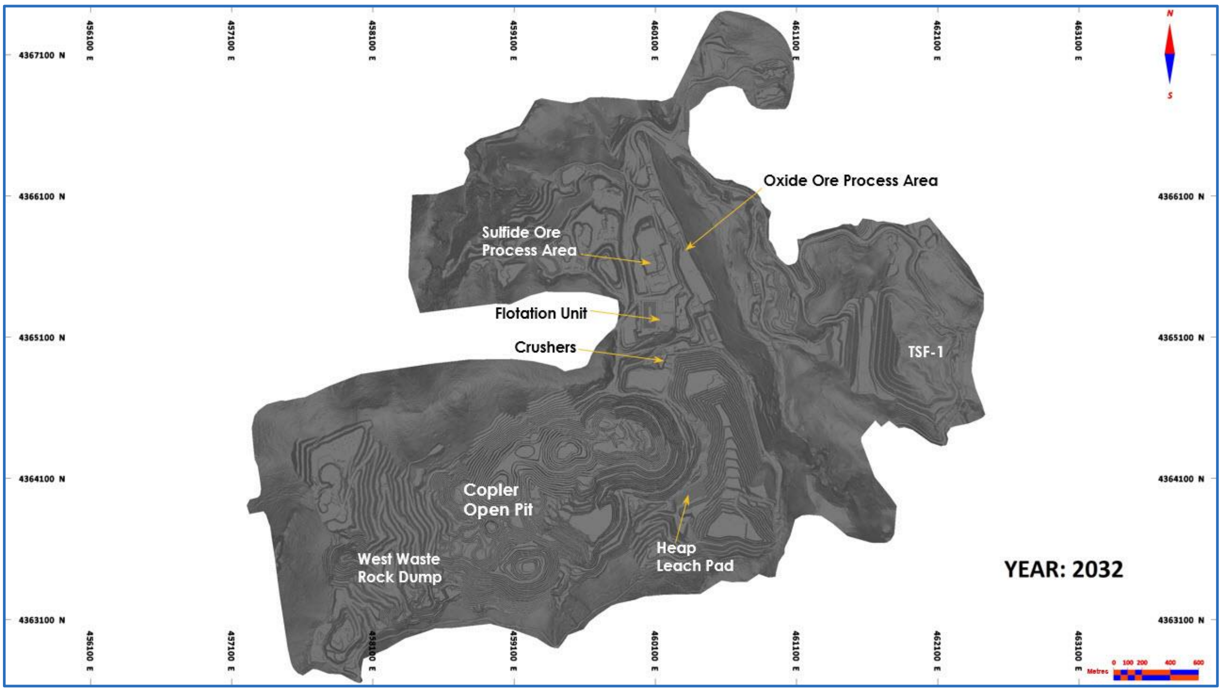
Anagold, 2020

Figure 16.14 2031 Pit Plan – Çöpler



Anagold, 2020

Figure 16.15 2032 Pit Plan – Çöpler

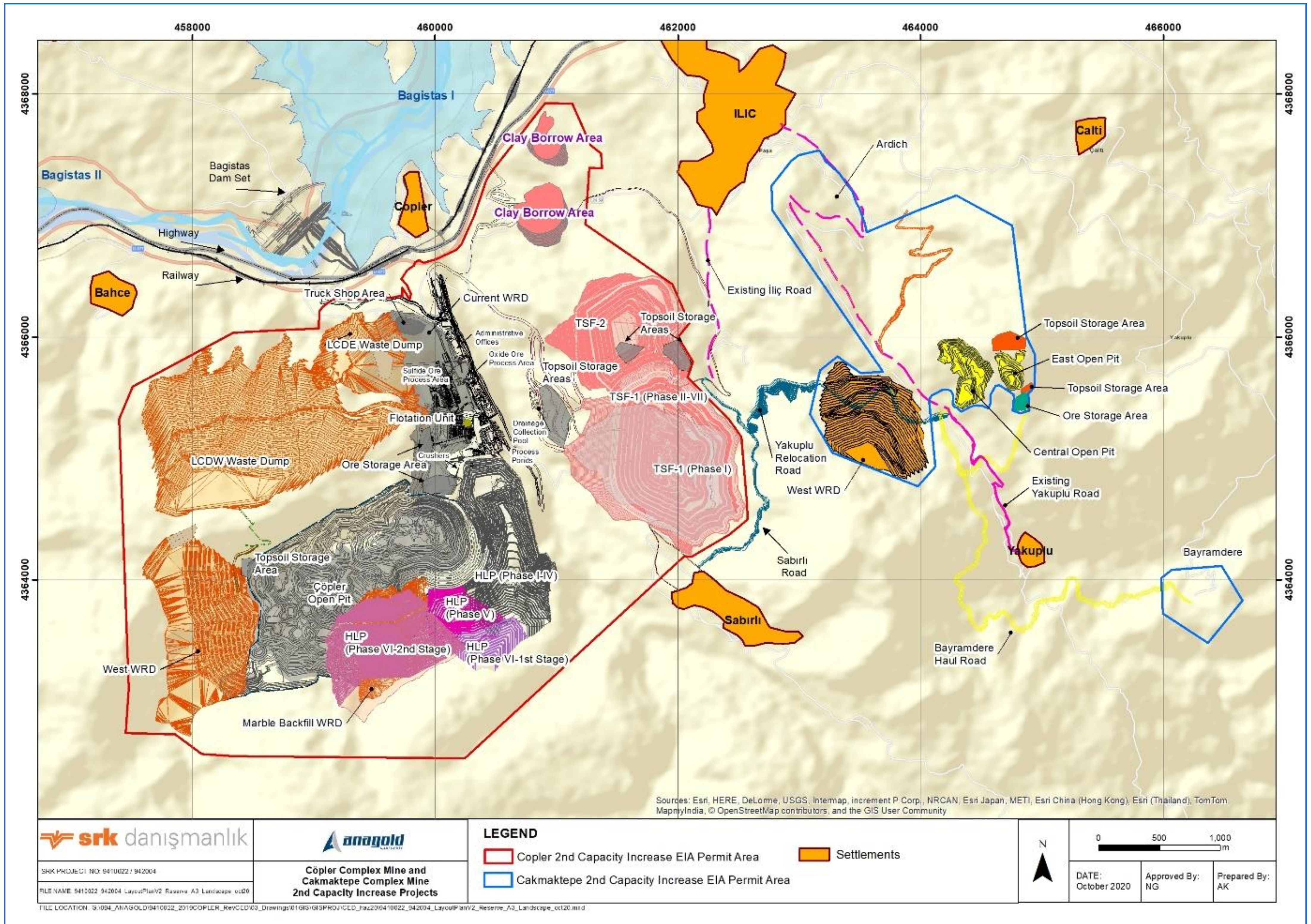


Anagold, 2020

16.2.4 Waste Dump and Stockpile Design

The mine plan allows for the use of five WRDs to store mined waste rock and sulfide ore that is extracted during mining operations. These five WRDs are Lower Çöpler East, Lower Çöpler West, Upper Çöpler, West, and Marble Backfill WRDs. Current operations do not use the Lower Çöpler West and Marble Backfill WRDs. The Lower Çöpler East and Upper Çöpler WRDs will primarily be utilised as sulfide ore stockpile areas, with the Upper Çöpler WRD being mined out to allow for future pushback extension of the Marble pit towards the north and allow for leach pad extensions to the west. Figure 16.16 shows the site layout.

Figure 16.16 Reserve Case Site Plan



Anagold, 2016

The Lower Çöpler East WRD has a capacity of 14.9 Mm^3 (26.8 Mt) of mine waste and 5.5 Mm^3 (9.9 Mt) of sulfide ore. The total surface area impacted by the Lower Çöpler East WRD is 51.5 ha. The Lower Çöpler West WRD has a capacity of 94.6 Mm^3 (170.3 Mt) of mine waste and 12.4 Mm^3 (22.3 Mt) of sulfide ore. The total surface area impacted by the Lower Çöpler West WRD is 206.5 ha. The Upper Çöpler WRD has a capacity of 7.6 Mm^3 (13.6 Mt) of sulfide ore. The total surface area impacted by the Upper Çöpler WRD is 26.1 ha. The West WRD complex has a capacity of 34.4 Mm^3 (61.9 Mt) of mine waste. The total surface area impacted by the West WRD is 108.9 ha.

An estimated 69.8 Mt of waste rock will be consumed in the construction of the tailings storage facility, haul road, and tailings pipeline corridor. Total constructed waste rock storage capacity is 155.0 Mm^3 (279.1 Mt). The total surface area impacted by all WRDs and stockpiles are 366.9 ha. When possible and economically preferable, waste rock will be backfilled within mined out areas of the pits as they become available.

16.2.4.1 Waste Rock Dump (WRD) Geotechnical Design

The WRDs will generally consist of 15 m tall lifts deposited at the waste material's angle-of-repose of approximately 1.33H:1V. The typical bench width will be 17 m and 15 m wide haul roads will be used to construct the WRDs. The WRDs will have overall slopes ranging from approximately 2.5H:1V to 2.6H:1V.

In February 2014, Golder completed an evaluation of the geotechnical stability of the four WRD designs (Golder, 2014a). This evaluation was updated in May 2015 (Golder, 2015b) to account for the updated material properties developed by Golder during the pit slope optimisation study and the updated waste dump designs and layouts developed by SSR Mining. Six of the most critical cross-sections were evaluated to determine the minimum Factor of Safety (FOS) for the proposed waste dumps. The sections were aligned to pass through the highest part of the waste piles, the steepest waste pile slopes, and the steepest foundation grades.

In addition to static stability analyses, pseudo-static stability analyses were performed to account for seismic loading conditions for the WRDs. The pseudo-static analyses were conducted based on the procedure proposed by Hynes-Griffin and Franklin (1984) in which a horizontal acceleration equal to 50% of the peak ground acceleration at bedrock is applied to the model. The design criteria peak ground acceleration is 0.30 g for the magnitude 7.0 operating basis earthquake (OBE). Therefore, a horizontal pseudo-static acceleration of 0.15 g was applied to the WRD sections in the seismic stability analyses.

The results of the stability analysis are summarised in Table 16.11.

Table 16.11 Waste Rock Dump (WRD) Design Factor of Safety (FOS)

Waste Dump	Section	Loading Condition	Failure Surface Location	Minimum Computed FOS
Lower Çöpler East Dump	A	Static	Shallow	1.4
		Pseudo-Static		1.1
		Static	Deep	1.9
		Pseudo-Static		1.3
	B	Static	Shallow	1.7
		Pseudo-Static		1.3
		Static	Deep	1.9
		Pseudo-Static		1.3
Lower Çöpler West Dump	C	Static	Shallow	1.7
		Pseudo-Static		1.3
		Static	Deep	1.9
		Pseudo-Static		1.3
	D	Static	Shallow	1.6
		Pseudo-Static		1.2
		Static	Deep	1.8
		Pseudo-Static		1.3
West Çöpler Dump	E	Static	Shallow	1.6
		Pseudo-Static		1.1
		Static	Deep	1.9
		Pseudo-Static		1.3
	F	Static	Shallow	1.6
		Pseudo-Static		1.2
		Static	Deep	2.0
		Pseudo-Static		1.4

The Lower Çöpler East WRD facility will be constructed over a portion of the existing Northeast WRD. Foundation conditions underlying the existing Northeast WRD and the proposed Lower Çöpler East facility consist of Munzur Limestone. Minimum computed factors of safety for the Lower Çöpler East facility are 1.4 and 1.1 for static and seismic loading conditions, respectively.

The Lower Çöpler West WRD facility will be founded on Munzur limestone. Limit equilibrium stability analyses indicate minimum computed FOS of 1.6 and 1.2 for static and seismic loading conditions, respectively (Golder, 2015b).

The West WRD is to be constructed adjacent to the Çöpler open pit and will be founded on Munzur limestone and metasediment with sporadic diorite intrusions. Minimum computed FOS are 1.9 and 1.3 for static and seismic loading conditions, respectively.

16.2.4.2 Waste Rock Geochemical Review

Anagold mines and monitors the waste rock types to determine PAF and NAF material according to the Çöpler waste rock management plan to ensure proper disposal of PAF material as it is encountered during the ore control process. SRK established the criteria for identifying PAF and NAF material as shown in Table 16.12.

Table 16.12 Waste Rock Geochemical Classification

Lithology	Sulfide Sulfur (SS%) Cut-off Grade	Waste Rock Groups	Descriptions
Diorite	0.8	PAF/High-sulfide diorite	Diorite with SS ≥0.8%
		NAF/Low-sulfide diorite	Diorite SS <0.8%
Metasediment	0.8	PAF/High-sulfide MTS	Metasediment with SS ≥0.8%
		NAF/Low-sulfide MTS	Metasediment with SS <0.8%
Limestone/ Marble	2	High-sulfide LMS	Limestone with SS ≥2%.
		Low-sulfide LMS	Limestone with SS <2%.
Gossan	–	Gossan – NAF	All Gossan unit
MnOx	–	MnOx – NAF	All MnOx unit
Massive Pyrite	–	Massive Pyrite – PAF	All Massive Pyrite unit

In September 2015 SRK completed a Geochemical Impact Assessment for the Çöpler WRD facilities. The key findings from the SRK report suggests that all WRD facilities at Çöpler, except one, have a neutralising potential (NP) to acid potential (AP) ratio of greater than 20:1; indicating that the Çöpler material has excellent neutralisation capacity for ARD. The one exception to this was the West WRD which was estimated to have a NP:AP ratio 1:3. It was recommended that SSR Mining optimise the WRD construction sequencing in order to take advantage of the neutralisation potential of the other WRD facilities by blending higher quantities of NAF material into the West WRD. SSR Mining anticipates that this will be a readily achievable solution that will not add any additional costs to the Project.

A series of waste rock samples representing the LOM distribution were tested by SRK in order to measure the immediate reactivity, future acid potential, and long-term acid potential of the waste rock.

In regard to immediate reactivity, a paste pH test was conducted that resulted in all samples generating near-neutral and slightly alkaline paste pH.

In regard to future acid potential, a large majority of all samples taken reside above the NP:AP 1:1 boundary. The remainder of the samples that fall below the 1:1 boundary are extremely close to the 1:1 boundary and should only pose a minimal risk to ARD generation. In terms of long-term acid potential, only two samples registered below the 1:1 NP:AP ratio.

16.2.5 Ore Stockpiles, Rehandle and Blending

Oxide and sulfide ore are processed through separate crushing circuits.

Oxide ore that is unable to be directly dumped into the crushing circuit is placed on the appropriate stockpile for processing at a later time. Oxide ore is typically segregated according to clay content and grade. The processing engineer determines the desired blend on a daily basis in order to maintain a consistent feed grade and rock type blend going to the heap leach pad.

All sulfide ore is currently placed in one of three primary stockpiles: High-grade, medium-grade, and low-grade. Sulfide ore is directed to the primary stockpiles or to the crusher pad. There is no allowance for material to be directly dumped into the sulfide crushing circuit. All material is rehandled by a loader from the crushing pad into the crushing circuit.

The following Au grade bin assumptions were used for the Mineral Reserves:

- High-grade Au >4.0 g/t Au
- Medium-grade Au 2.0–4.0 g/t Au
- Low-grade Au 1.05–2.0 g/t Au

Currently site typically experience a lack of sulfide sulfur (SS) feed to the POX, requiring additional cost to run the POX plant. The flotation plant (under construction) will upgrade (increase) the SS feed into the POX circuit. For the POX autoclave to operate autogenously, SS feed must be above 10.20 tph and less than 13.75 tph to achieve target oxidation with current oxygen availability. If the SS feed rate is too high, then the feed to the plant will need to be reduced until the POX SS feed rate is less than 13.75 tph limit. Operating performance of the autoclaves indicates that higher than design oxygen utilisations efficiencies are possible, which may allow greater than 13.75 tph sulfide sulfur to be treated. This oxygen utilisation efficiency along with increased oxygen availability is upside to the CDMP20 Reserve Case.

Plant feed will therefore need to be blended to achieve the target SS feed range of 10.20–13.75 tph into POX.

To blend on SS feed, new grade control protocols will need to be developed and implemented on site. Site grade control is currently being done on Au and C grades. New 'grade bins' will need to be developed incorporating SS grade, to aid in achieving the ideal range for SS feed into the plant. The new grade bins will need to be used to develop a new stockpile strategy.

The following SS grade bin assumptions were used for the Mineral Reserves inside each Au grade bin:

- High-grade SS >6.0% SS
- Medium-grade SS 3.0% to 6.0% SS
- Low-grade SS <3.0% SS

The effectiveness of these new grade bins in controlling the SS blend will need to be measured as the plant matures and adjustments to the grade bin parameters (and size of stockpiles) may be required.

The smallest parcel size for plant feed considered for the Mineral Reserves was one month.

The operation will need to be in control of the plant feed blend at a more granular level than was modelled for the Mineral Reserves. If maintaining a plant feed blend a more granular level be found to be problematic, there are several measures that site could implement to manage both short-term and longer term concerns:

- Mine working areas

  Given the relatively small size of the mining fleet, the number of active mining working areas could be increased, increasing mining selectivity, and therefore improving the blending capacity from the mine.

- Stockpile size

  The size of stockpiles could be adjusted to reduce feed impacts from short-term fluctuations coming from the mine.

- Mining rate

  Given the current site contract mining arrangement, site could ramp up the mining rates to reach sufficient material (of the required type) to maintain the required blend.

- Variation of grade bins

  Grade bin designations could be adjusted to have better control of the grade bands that are causing problems in the plant feed blend.

16.2.6 Grade Control

All ore control operations are managed by Anagold technical staff. Anagold maintains an on-site laboratory with the capacity to assay an average of 600 blast hole samples per day.

Prior to sampling, blast holes are identified as ‘potential ore’ (oxide or sulfide) or ‘potential waste’ (oxide or sulfide) based on grade control data from the bench above and the mining model prediction. A 10 m outside buffer is then applied to the potential ore areas to ensure appropriate sampling density. All potential ore blast holes are sampled for AuFA (fire assay for Au). Approximately 50% of potential ore blast holes are sampled for AuCN (cyanide soluble Au), total carbon, and total sulfur. Additionally, all potential sulfide ore blast holes are sampled for SS. Approximately 25% of potential waste blast holes are sampled for AuFA, AuCN, total carbon, and total sulfur.

Sampling of the blast hole drill cuttings is performed according to the formal procedure by using a sample scoop to extract a complete cross-section of the cutting pile. The sampled cuttings are deposited into a canvas bag, which is labelled with the drillhole identifier (ID) and with a laboratory information management system (LIMS) bar code tag inserted into the bag with the cuttings. Sample bags are sealed and sent to the on-site laboratory for analysis. The sample scoop is cleaned prior to collecting each sample to avoid contamination between samples.

Assay results are uploaded to the ore control database with reference to each specific drillhole ID. The assay results are then estimated into a cell model with parent cell sizes of 3 m x 3 m x 5 m using OK to estimate ore grade and type. The ore control geologist will then digitise mining shapes with a minimum width of 3 m (to match the SMU) and minimum tonnage of 500 t. These mining shapes are then sent to the survey group for layout in the mine using colour coded flagging under the supervision of the ore control geologist.

To effectively blend the sulfide feed on SS content, new grade control protocols will need to be developed and implemented on site. Site grade control is currently being done on Au and C grades. Therefore, new grade control protocols will be required to support the new stockpile grade bins.

16.3 Mine Production Schedule

The Reserve Case has examined production from two open pit mining locations at the Çöpler mine, the Çöpler deposit and the Çakmaktepe deposit. The Çakmaktepe pit, which contains only oxide ore, is almost exhausted. Therefore, the bulk of the oxide ore and 100% of the sulfide ore is sourced from the Çöpler pit. SSR Mining has prepared the open pit production schedules. The case adopted for the Reserve Case assumes the addition of the flotation circuit to the sulfide plant, is based on Mineral Reserves only, and does not include Inferred Mineral Resources.

16.3.1 Scheduling Assumptions

The following scheduling methodology was used to balance mine, mill, and stockpile quantities:

- Heap leach:
  - Oxide ore is not limited by processing capacity.
  - Oxide ore that is unable to be directly dumped into the oxide crushing circuit is placed in the appropriate stockpile for future processing.
  - Oxide ore is segregated dependent on clay content and average grade.
- Sulfide plant:
  - All sulfide ore is segregated into one of three primary gold stockpiles: high-grade, medium-grade, and low-grade, which are each further split by SS grade.
  - Existing stockpiles are mined at the average grade of each stockpile.
  - All material is rehandled by a loader from the crushing pad into the crushing circuit (no direct tipping).
  - The flotation circuit is planned to be commissioned in mid-2021.
  - Plant throughput capacity is calculated from the available mill hours and varies by material type.
- The production schedules are based on Proven and Probable Mineral Reserves only. No Inferred Mineral Resources were used.
- The open pit schedules were based on mining inventories by bench reported within the pit stages.
- Low-grade stockpiling was used to balance the mining rate where necessary.

16.3.2 Production Schedule

The input assumptions for Reserve Case were adjusted based on current mine and production performances including throughput rates and recoveries.

All throughput rates are reported inclusive of all availability and utilisation factors on a calendar year. Total mine production is limited to an annual average of 22.5 Mtpa. The throughput assumptions are supported by current mining rates including productivity allowances for winter and summer conditions. Mining rates are limited based on vertical advance and bench configuration in order to ensure that the schedule is achievable. Production is not limited by the mining rate and increases in rate would be possible to bring forward oxide ore or increase stockpiling to bring higher grade feed to the sulfide plant.

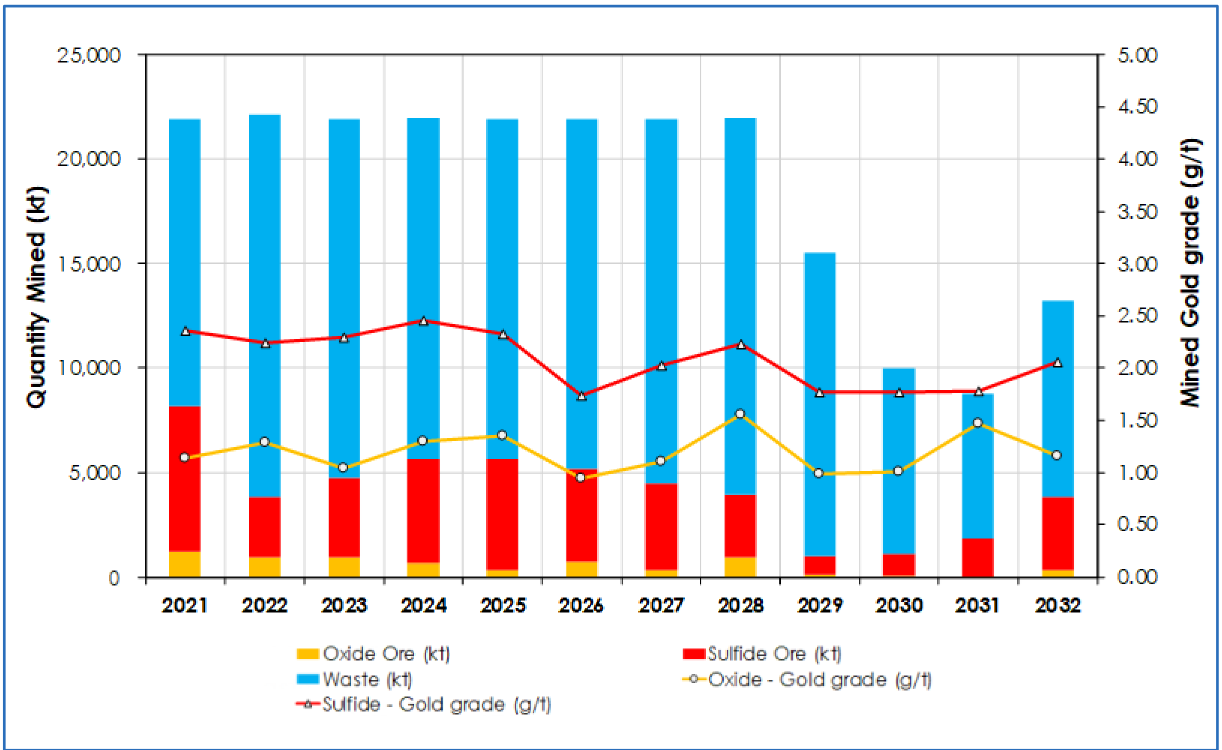
Mining in the Reserve Case is completed in 2032 the sulfide plant is then fed from stockpiles.

The objective of the production schedule is to maximise the early cash flow by delaying costs and bringing revenue forward with ore feed to meet concentrator throughput capacity. Considerations for the LOM scheduling include:

- Ensuring continuous ore supply to the concentrator by delivering the highest value ore first and meeting physical mining and milling hours capacity constraints.
- Achieving excavator productivities and sinking rates to deliver ore at maximum utilisation of milling hours available at the concentrator.
- Maximising annual utilisation hours for the mine loading equipment.
- Maintaining a balance of ore throughput rates (material types) and mill cut-off grades that allows milling hours to be maximised.

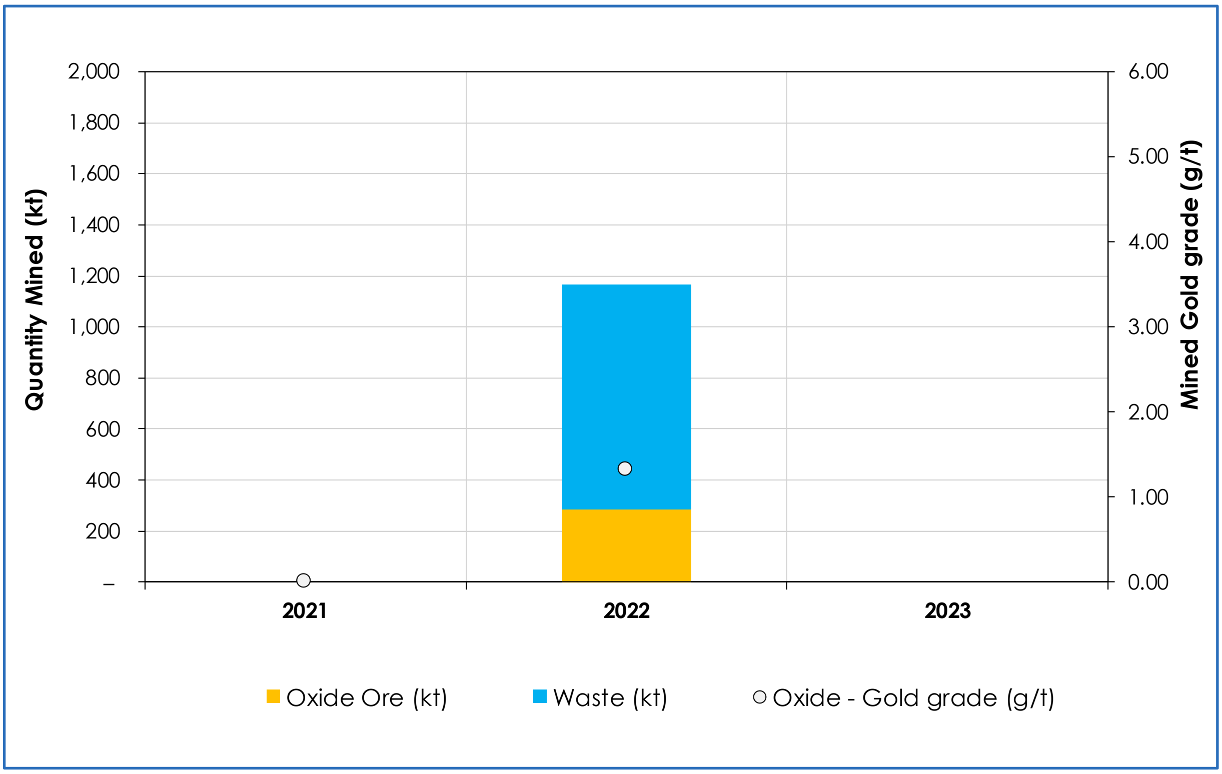
The mine schedule incorporates strategic stockpiling considerations by optimising the number of excavators on the benches of early phases, increasing the opportunity to raise mill cut-off grades. This leads to stockpiling medium-grade and low-grade material and sending higher grade ore to the mill sooner. The open pit total movement is shown in Figure 16.17 to Figure 16.19.The mining schedule by material type is in Table 16.13.

Figure 16.17 CDMP20 Reserve Case Mining Production



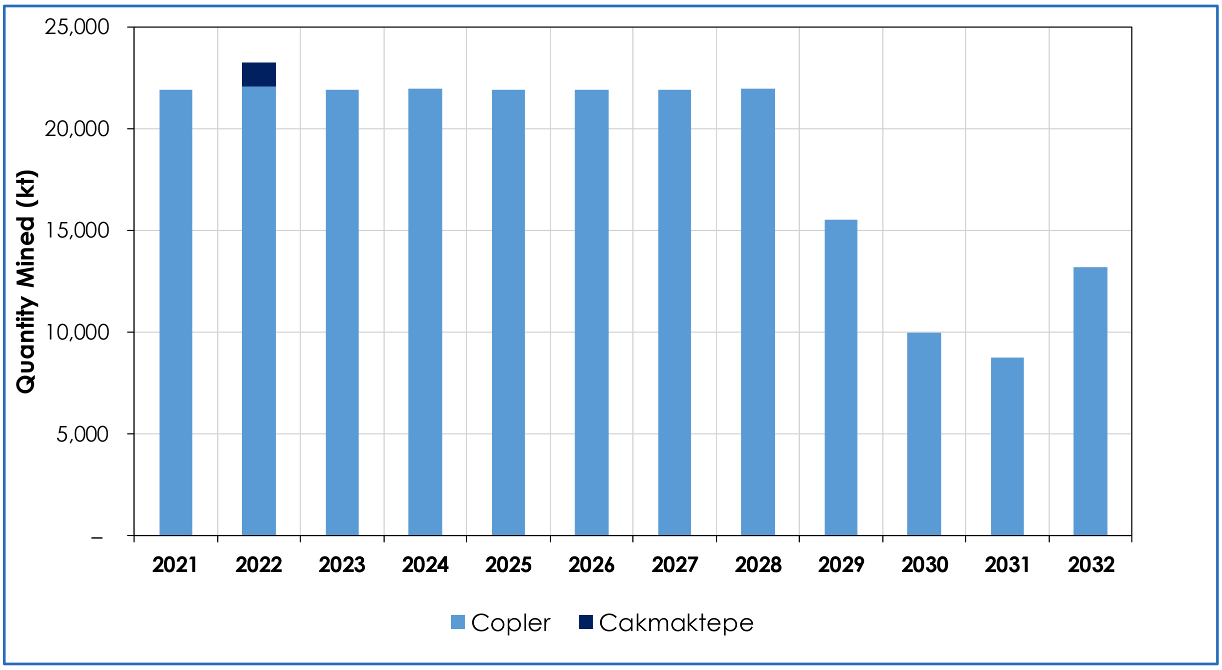
OreWin, 2020

Figure 16.18 Open Pit Mining – Çakmaktepe



OreWin, 2020

Figure 16.19 Open Pit Mining – Total Movement



OreWin, 2020

Table 16.13 Reserve Case Mining Schedule

Year	Total Tonnes (kt)	Oxide				Sulfide				Waste Tonnes (kt)
		Tonnes (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Tonnes (kt)	Au (g/t)	Ag (g/t)	SS (%)	
Q4'20	6,898	985	1.31	6.53	0.11	1,387	2.55	10.37	5.07	4,525
2021	21,900	1,205	1.13	3.77	0.11	6,966	2.36	9.13	3.96	13,728
2022	23,280	1,214	1.30	16.28	0.12	2,880	2.24	8.54	4.31	19,186
2023	21,900	938	1.04	1.97	0.16	3,810	2.30	2.44	3.68	17,152
2024	21,960	637	1.30	4.32	0.12	4,999	2.46	5.79	4.06	16,323
2025	21,900	345	1.35	3.97	0.13	5,285	2.33	4.53	4.01	16,270
2026	21,900	683	0.94	0.86	0.18	4,462	1.74	3.42	4.20	16,755
2027	21,900	309	1.10	2.55	0.09	4,174	2.03	4.66	4.97	17,418
2028	21,960	899	1.56	9.88	0.10	3,047	2.23	9.37	4.26	18,014
2029	15,547	89	0.98	3.76	0.07	894	1.76	3.70	4.94	14,564
2030	10,000	39	1.01	12.21	0.10	1,028	1.77	4.59	5.12	8,933
2031	8,757	14	1.46	5.42	0.06	1,800	1.78	4.09	5.05	6,944
2032	13,202	309	1.16	6.75	0.07	3,506	2.06	4.17	4.54	9,387
Total	231,105	7,668	1.22	6.51	0.12	44,238	2.18	5.83	4.29	179,200

Table shows mining schedule does not show processing or existing stockpile rehandle.

16.3.3 Processing Schedule

The processing schedule was balanced to meet the maximum build rates for the oxide heap leach pads, or available mill hours for the sulfide plant.

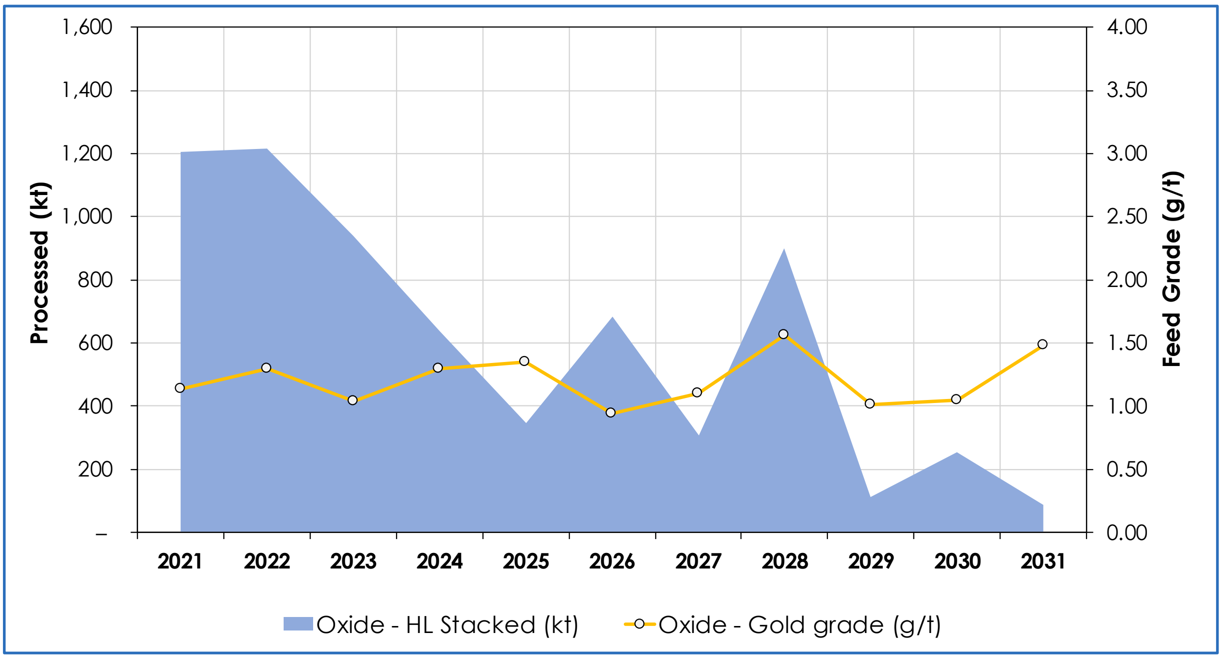
Sulfide ore production throughputs are limited dependent on ore tonnage, SS tonnage, and carbonate content, (expressed as C). The sulfide plant crusher / grinding circuit is limited to 400 tph, while the limitations on SS tonnage exist due to the consumption of oxygen by SS in the POX circuit and carbonate content to maintain an operable acid balance through the acidulation and POX circuits. The process facilities are limited by the amount of oxygen that can be provided to the POX process. Based on current performance, high-SS is unlikely to be a problem, and any higher material would be blended down using low-SS material. The carbonate:SS ratio will potentially be an issue with declining SS grades. The main issue currently appears to be a lack of SS in the feed, forming the justification for the flotation circuit. The flotation circuit upgrades the SS content into the autoclave feed and rejects carbonate.

In order to target the highest value material, the sulfide production schedule is therefore required to target the highest value material, while also balancing the plant throughput rates and required range of sulfide sulfur into the autoclave.

The Reserve Case production is predominantly from sulfide ore. The maximum oxide ore placed in any year is 1.2 Mt for a total production of 7.7 Mt. The oxide heap leach and sulfide plant processing schedules feed type, Au grade, and gold production are shown in Figure 16.20 through Figure 16.22. The production schedule is in Table 16.14.

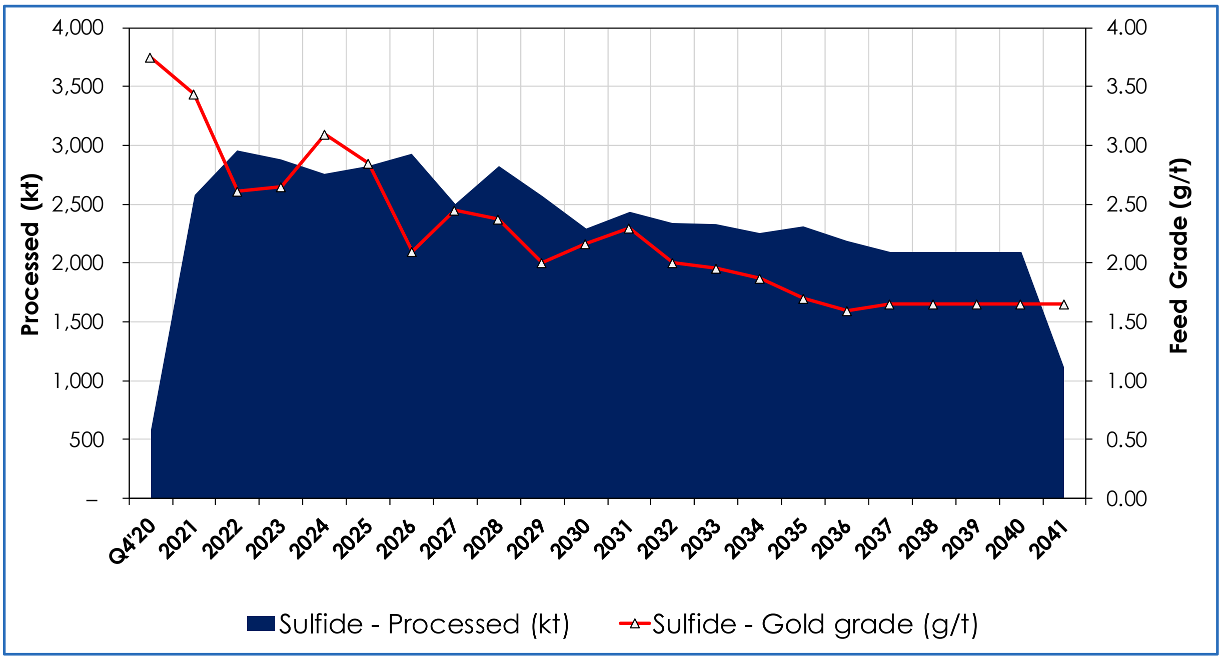
The Reserve Case production includes 7.7 Mt at 1.22 g/t Au oxide ore processed by heap leaching and 51.1 Mt at 2.24 g/t Au processed in the sulfide plant, total gold production is 3.6 Moz. All mining is completed by 2032, oxide heap leach stacking is completed by 2031, while sulfide processing will continue from stockpiles until 2041 for a 21 year mine life. The production schedule is for the period 1 October 2020 through 2041, the Mineral Reserves includes 403 kt at 2.47 g/t Au from the mine plan scheduled for September 2020.

Figure 16.20 Processing Schedule – Oxide Heap Leach



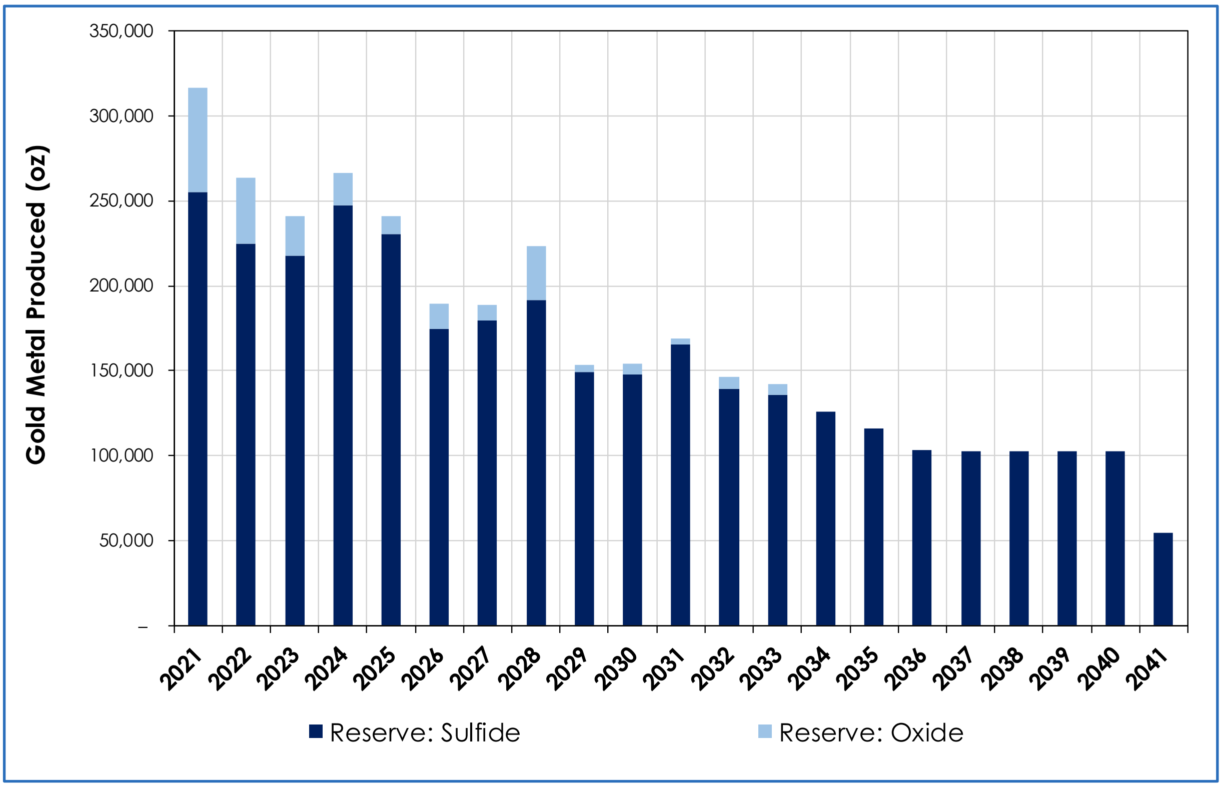
SSR Mining, 2020

Figure 16.21 Processing Schedule – Sulfide Plant



SSR Mining, 2020

Figure 16.22 Gold Production and Recovery



SSR Mining, 2020

Table 16.14 Production Schedule

	TOTAL	Year																					
		Q4'20	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042
Heap Leach																							
Stacked (kt)	7,668	985	1,205	1,214	938	637	345	683	309	899	112	253	86	–	–	–	–	–	–	–	–	–	–	–
Au Feed Grade (g/t)	1.22	1.31	1.13	1.30	1.04	1.30	1.35	0.94	1.10	1.56	1.01	1.05	1.48	–	–	–	–	–	–	–	–	–	–	–
Ag Feed Grade (g/t)	6.51	6.53	3.77	16.28	1.97	4.32	3.97	0.86	2.55	9.88	5.20	6.59	8.35	–	–	–	–	–	–	–	–	–	–	–
Cu Feed Grade (%)	0.12	0.11	0.11	0.12	0.16	0.12	0.13	0.18	0.09	0.10	0.08	0.07	0.09	–	–	–	–	–	–	–	–	–	–	–
Sulfide Plant																							
Plant Feed (kt)	51,084	585	2,574	2,961	2,884	2,761	2,828	2,926	2,507	2,821	2,572	2,295	2,435	2,339	2,332	2,254	2,310	2,192	2,097	2,097	2,097	2,097	1,118	–
Au Feed Grade (g/t)	2.24	3.75	3.44	2.61	2.65	3.09	2.85	2.10	2.45	2.37	2.01	2.17	2.30	2.00	1.96	1.87	1.70	1.60	1.66	1.66	1.66	1.66	1.66	–
Ag Feed Grade (g/t)	5.07	9.83	10.56	6.09	2.97	5.77	5.14	2.64	5.43	9.52	7.48	4.56	3.27	0.90	3.91	5.27	4.09	4.33	4.35	4.35	4.35	4.35	4.35	–
SS Feed Grade (%)	4.29	4.48	4.10	3.85	3.76	3.96	3.92	3.76	4.25	3.89	4.17	4.50	4.40	4.28	4.45	4.73	4.54	4.73	4.75	4.75	4.75	4.75	4.75	–
Recovered Metal																							
Gold Recovered (koz)	3,591	86	316	264	241	266	241	189	189	224	153	154	169	146	142	126	116	103	103	103	103	103	55	–
Silver Recovered (koz)	761	60	69	210	39	49	30	15	22	116	28	27	14	3	9	11	9	9	9	9	9	9	5	–
Copper Recovered (klb)	7	0.8	1.0	1.1	1.2	0.6	0.4	0.9	0.2	0.7	0.1	0.1	0.1	0.0	0.0	–	–	–	–	–	–	–	–	–

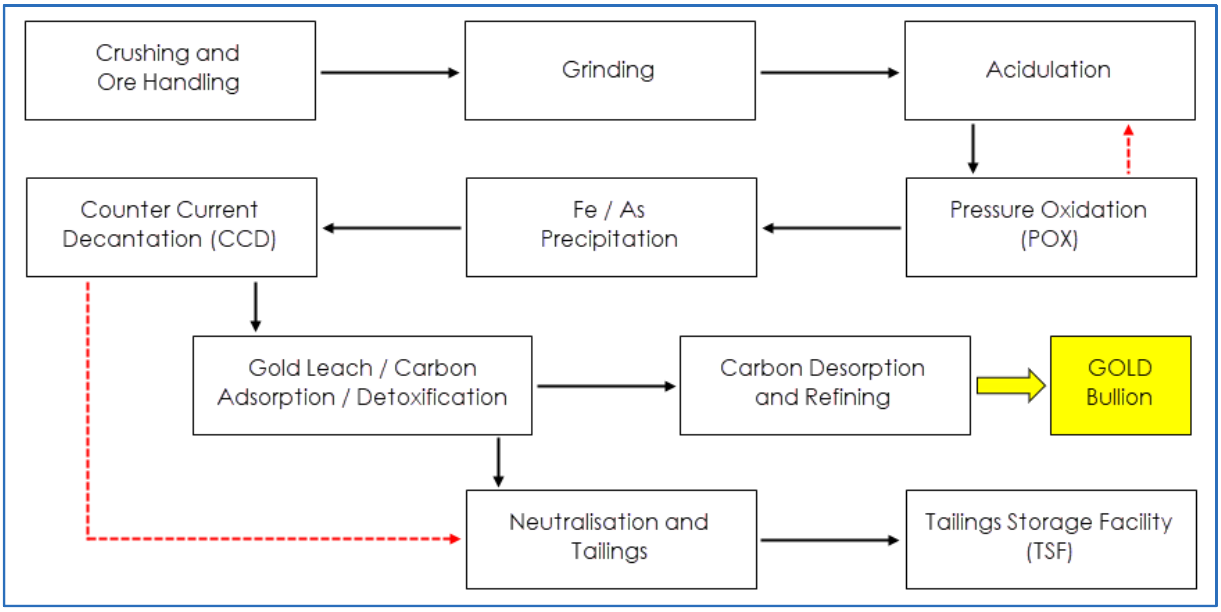
17 RECOVERY METHODS

17.1 Sulfide Ore Processing

The sulfide plant commenced commissioning in Q4'18. The basic flow sheet is shown in Figure 17.1 and comprises:

- Crushing and ore handling
- Grinding
- Acidulation
- Pressure oxidation
- Iron / arsenic precipitation
- Counter current decantation (CCD)
- Gold leach, carbon adsorption, and detoxification
- Carbon desorption and refining
- Neutralisation and tailings
- Tailings storage facility (TSF)

Figure 17.1 Çöpler Process Flow Sheet for Sulfide Plant



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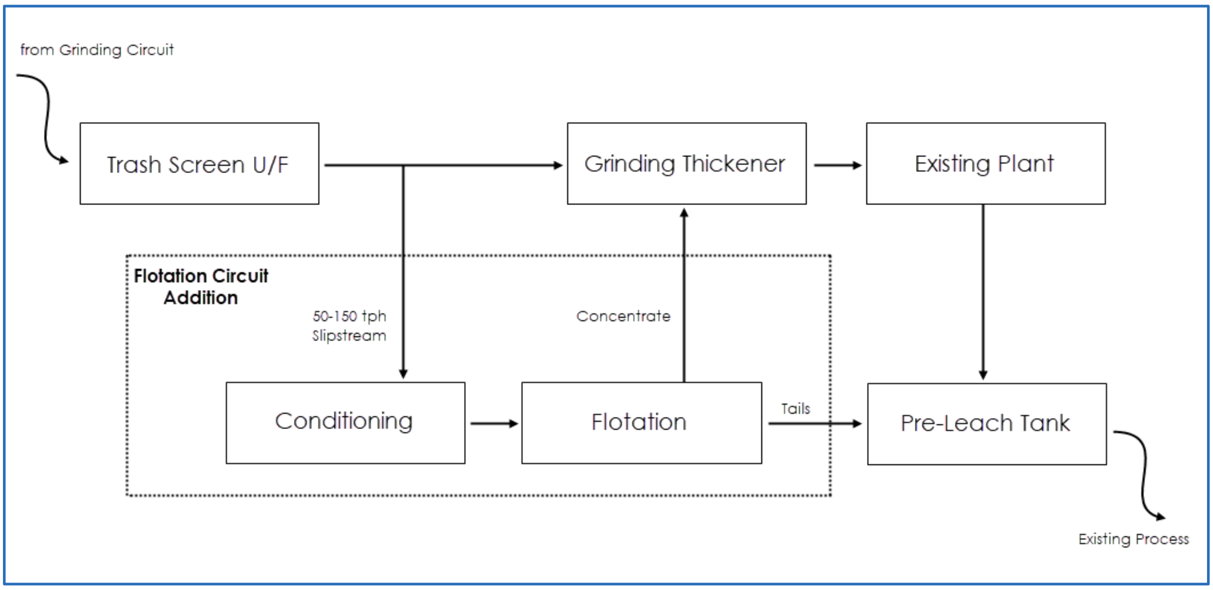
The incorporation of a flotation circuit into the existing sulfide plant is to upgrade sulfide sulfur to fully utilise grinding and POX autoclave capacity is under design and construction. This addition to the sulfide plant is incorporated between grinding and acidulation, as shown in Figure 17.2, by taking a bleed / slip stream from the grinding thickener feed, floating sulfides, and returning the sulfide concentrate to the grinding thickener to be combined with direct feed. Gold not recovered to flotation concentrate will report with flotation tails to the gold leaching and recovery circuit and combined with material process through the POX autoclave circuit to recover gold.

The flotation circuit will also reject carbonates to flotation tails, bypassing acidulation and POX, providing additional benefits in the acid balance through POX.

Figure 17.2 Flotation Block Flow Diagram



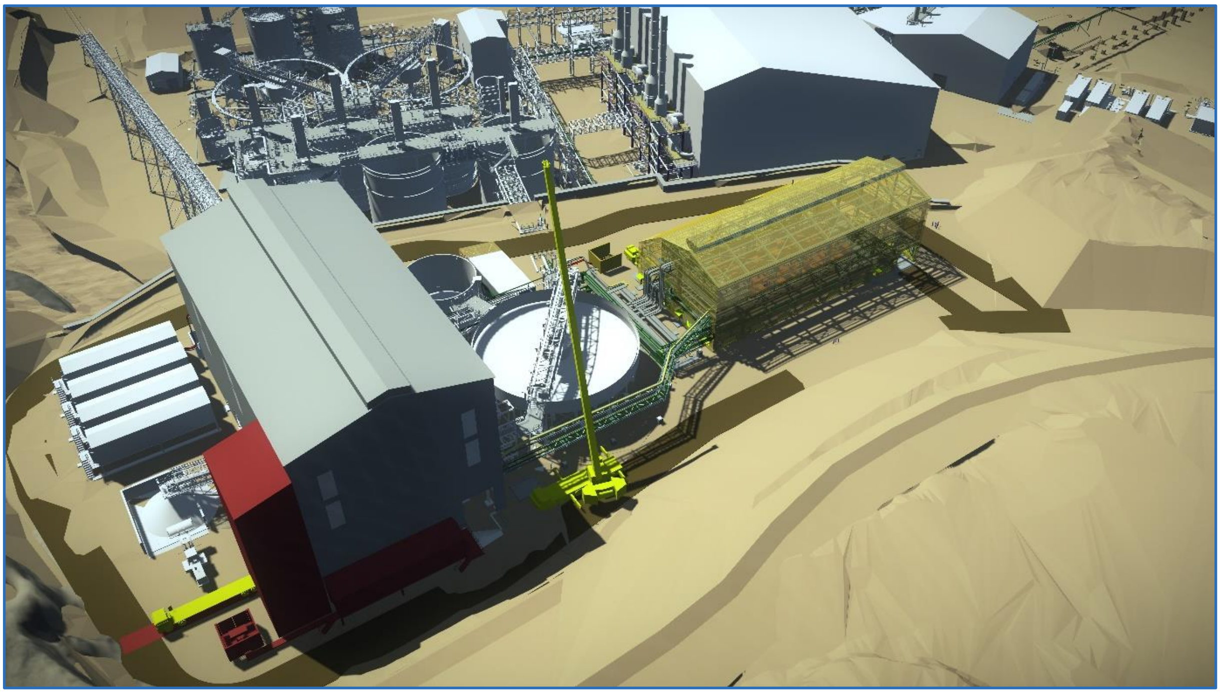
Anagold, 2020

The existing sulfide circuit, before the addition of flotation, has demonstrated additional latent capacity in throughput controlling sections of the circuit, crushing/grinding and autoclaves. The incorporation of flotation will allow the POX autoclaves to maximise throughput and sulfide sulfur oxidation capacity, utilising latent capacity in the process plant, in particular the grinding and pressure oxidation circuits. Fully utilising this latent capacity with the addition of a small flotation plant allows with minimal capital cost the increase in overall plant throughput.

The throughput from crushing and grinding was designed with a nominal volumetric capacity of 306 tph will increase up to a maximum of 400 tph. Additionally, the POX autoclave circuit has demonstrated it can process up to a maximum of 280 tph feed (two autoclave operation) and 13.75 tph sulfide sulfur, compared to design of 245 tph and 12.5 tph respectively. The limit of 13.75 tph sulfide sulfur is dictated by the capacity of the oxygen supply to effect oxidation of the sulfides. The flotation plant feed rate will be variable between 50–150 tph based on sulfide sulfur feed grade and the oxidation capacity of the POX autoclaves to oxidise sulfides. Operating performance of the autoclaves indicates that higher than design oxygen utilisations efficiencies are possible, which may allow greater than 13.75 tph sulfide sulfur to be treated. This oxygen utilisation efficiency along with increased oxygen availability is upside to the CDMP20 Reserve Case.

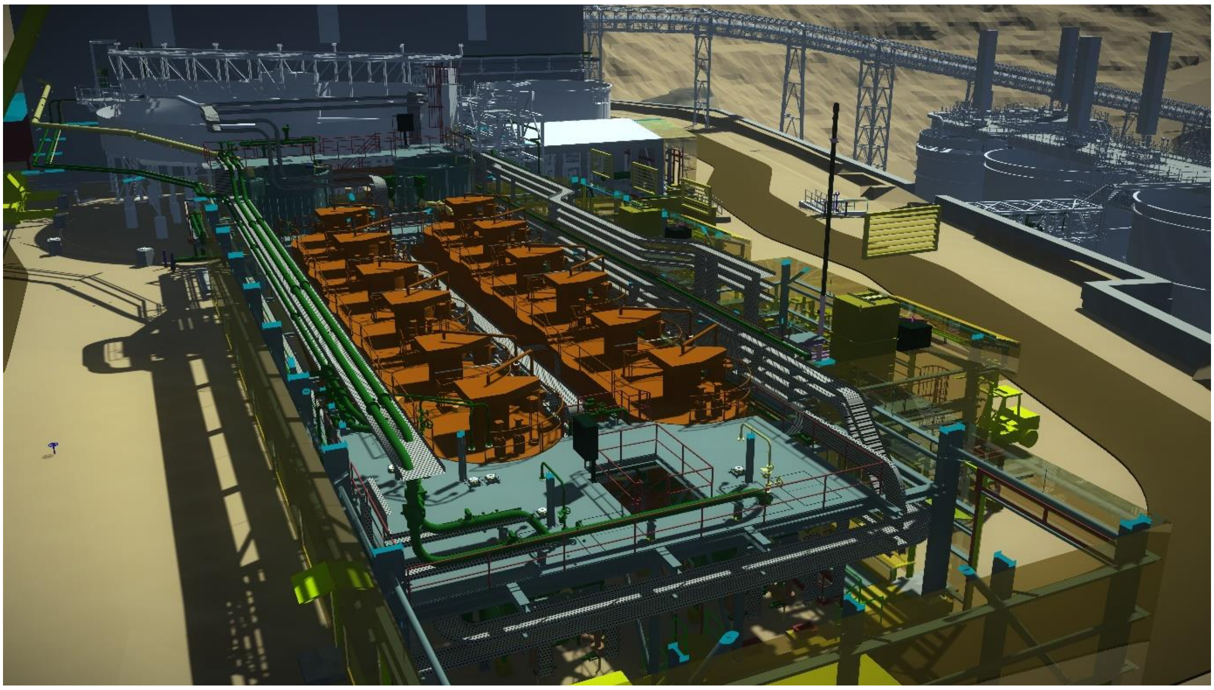
Figure 17.3 indicates the position of the flotation building, and Figure 17.4 shows the flotation circuit with the building not shown.

Figure 17.3 Flotation Circuit Building Location



Anagold, 2020

Figure 17.4 Flotation Circuit Location (building rendering not shown)



Anagold, 2020

17.1.1 Sulfide Plant Performance

The sulfide plant commenced commissioning in Q4'18. The commissioning period for the plant is considered to June 2019 followed by ramp up through to June 2020.

The operating performance is summarised in Figure 17.5 for throughput and recovery against the design basis, including allowances for commissioning ramp up. The ore supply to the plant during this period has consisted predominately of Manganese Diorite from historical stockpiles.

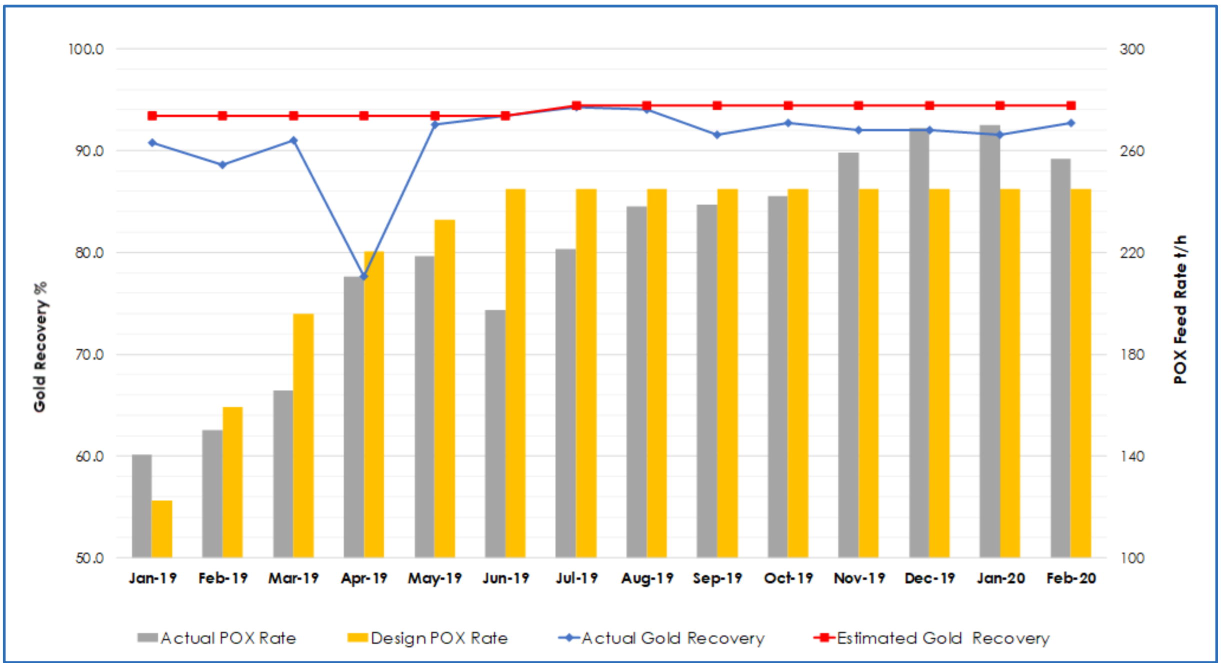
For the commissioning period, up to June 2019, both gold recovery and throughput have been lower than design, this is expected during commissioning. Typically, in a new plant, throughput rates ramp up more slowly than recovery. In April 2019 there was a total POX autoclave shut down for maintenance and inspections. During this period ore supply to the plant bypassed the POX circuit direct to gold leach which resulted in low gold recoveries. After start-up following the shutdown recovery continued to improve and throughputs increased progressively.

During the ramp up, from June 2019, gold recovery has approached the estimated recovery, the throughput has exceeded design since November 2019. The long-term POX autoclave throughput is expected to increase from a design of 245 tph to a maximum of 280 tph, dependent on sulfur grades, utilising the available oxygen supply for sulfide sulfur oxidation.

Further improvements have been implemented in mid-2020. The installation of oxygen to leach / CIP to supplement air to maintain sufficient oxygen levels for gold leaching has led to improved recoveries.

Ongoing studies and testwork is being undertaken to understand and improve plant performance.

Figure 17.5 Gold Recovery and Throughput Comparison

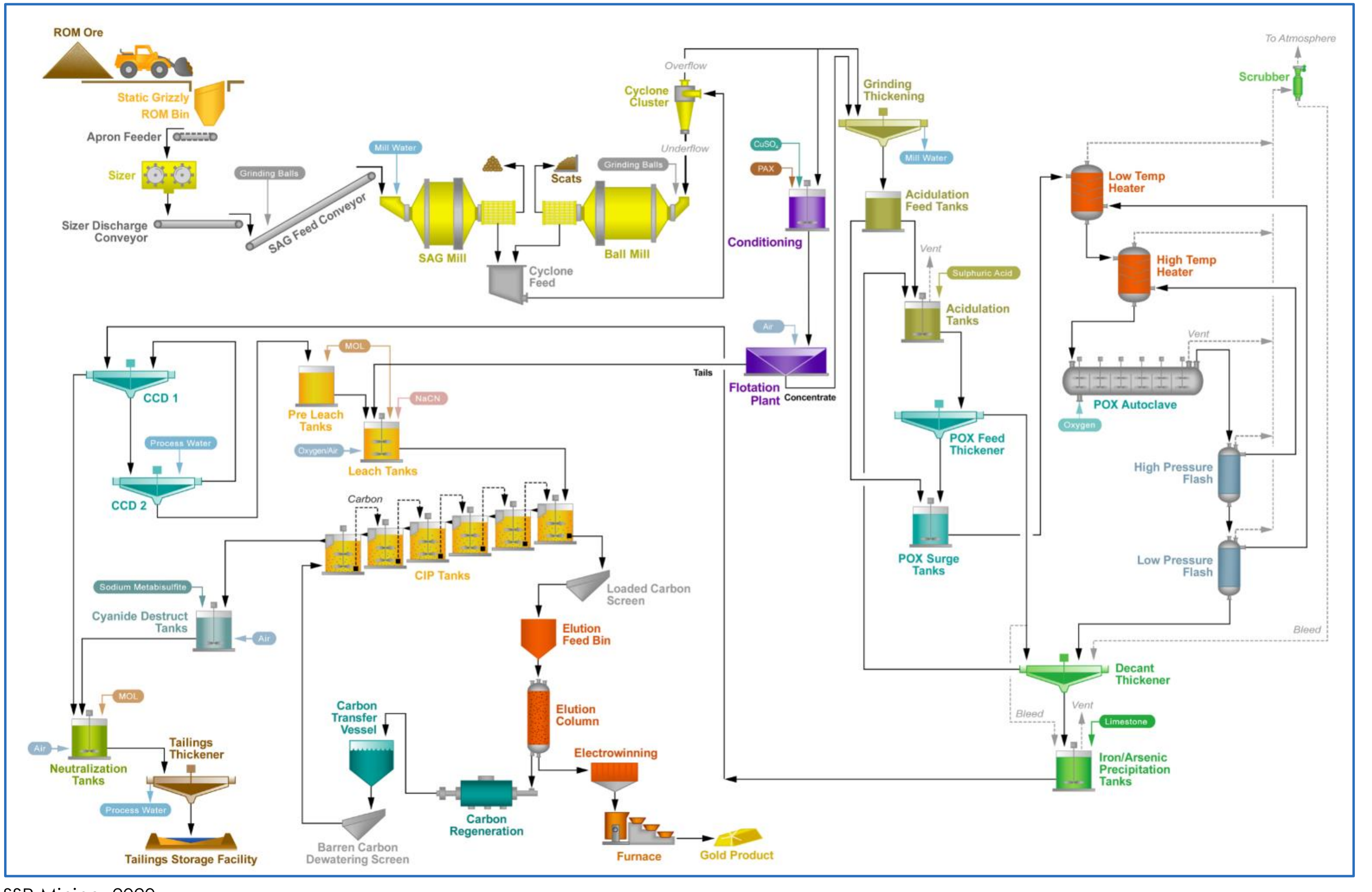


Anagold, 2020

17.1.2 Sulfide Plant Description

A detailed sulfide flow sheet is shown in Figure 17.6. The following description of the sulfide plant includes the existing operating circuits and the flotation circuit.

Figure 17.6 Process Flow Sheet for Sulfide Plant



SSR Mining, 2020

17.1.2.1 Crushing and Ore Handling

Haul trucks from the mine tip ore onto designated stockpile fingers. The ore is withdrawn from stockpiles by front end loader (FEL) and deposited into the ROM dump hopper. A static grizzly is fitted to the top of the ROM bin to remove coarse oversize.

ROM ore is reclaimed from the bin by the sizer apron feeder, which discharges material into the mineral sizer. The sizer is a tooth roll unit which crushes the ore from a feed top size of 500 mm to a nominal top size of 250 mm. Discharge from the sizer drops down a chute onto the sizer discharge conveyor.

The sizer teeth are configured in a manner to direct oversize rocks to one end where they pass through a spring-loaded oversize rejection gate and fall to a reject bunker. The crushed product is carried by the sizer product conveyor to the SAG mill feed conveyor. The SAG mill feed conveyor has a belt scale to monitor the ore flow to the SAG mill and this information is used to control the sizer apron feeder speed.

17.1.2.2 Grinding

The SAG milling stage consists of a high aspect SAG mill with water cannon pebble recycle. The SAG mill grinds the crushed ore to produce a discharge particle size distribution P_{80} of approximately 1,400 µm.

Large ore particles are retained in the SAG mill by the internal SAG discharge grate. Particles too large for ball milling are retained as oversize on the SAG mill trommel screen and this oversize is washed by trommel sprays. The trommel screen oversize is captured by a scoop on the trommel then dropped into a static central return tube from where it is projected back into the SAG mill using a high-pressure water cannon. Slurry that passes through the trommel screen discharges into the grinding cyclone feed pump box where it mixes with the ball mill discharge slurry and density control water.

Slurry collected in the grinding cyclone feed pump box from the SAG mill and ball mill is fed to the grinding cyclone cluster. The cyclones produce an overflow product with a P_{80} of 100 µm, which is screened to remove any trash (organic material, etc.) by the grinding trash screen. Coarse particles report to cyclone underflow, which is returned to the ball mill for further size reduction until it is fine enough to report to cyclone overflow and leave the circuit.

The slurry product from the grinding circuit, trash screen undersize, is currently thickened in a high-rate thickener and excess water reports to the thickener overflow for immediate re-use within the grinding circuit. The thickened slurry discharging from the thickener underflow is pumped to the grinding thickener underflow storage tanks.

To provide for the flotation circuit, a portion of the trash screen undersize, dependent on POX autoclave sulfide sulfur requirements, will be diverted to the flotation circuit where the remaining slurry continues to the thickener.

17.1.2.3 Flotation

A portion of the grinding trash screen undersize will be diverted to the flotation circuit and pumped to the conditioning tanks. This proportion, between 50 tph and 150 tph, will depend on SS feed grade and POX autoclave SS requirements. The flotation circuit can operate as a single or dual train, each train will have a maximum throughput of 75 tph.

The flotation plant consists of two equally sized conditioning tanks, in series, for copper sulfate, if required, and potassium amyl xanthate (PAX) conditioning with a nominal residence time of seven minutes each tank. From conditioning, the slurry is pumped to two equally sized flotation trains consisting of six 50 m^3 tank cells with a residence time of nominally 60 minutes at maximum throughput (75 t/h each). Frother dosing and supplemental collector dosing will occur down the trains in every second cell. The plant is designed to handle high mass pull to maximise sulfide recovery, with preference to high recovery over high selectivity.

The flotation concentrate is pumped to the grinding thickener feed mixing with slurry directly from the grinding circuit upgrading the sulfide sulfur material fed to the acidulation and POX circuit. The flotation tail is pumped to the gold leach tanks for recovery of gold present in the non-sulfidic portions of the ore.

17.1.2.4 Acidulation

The grinding thickener underflow storage tanks provide process surge and effectively decouple the upstream crushing, grinding and flotation, when operating, circuits from the downstream hydrometallurgical circuit. If the acidulation feed tanks reach their high-level limit then ore feed to the upstream circuits will be stopped. If the tanks are approaching their low-level limit then the upstream circuit feed rate can be increased to compensate.

The tanks are agitated for solids suspension and mixing and have a total residence time of 12 hours. Agitation achieves short term blending of the incoming feed from the upstream circuits and this provides a relatively slow-changing feed composition to the downstream hydrometallurgical circuit. Antiscalant can be added to these tanks if necessary, to reduce scale build-up in the downstream acidulation circuit.

The acidulation circuit uses recycled solution, containing free acid, from the decant thickener to leach the carbonate minerals in the ore. Supplemental concentrated sulfuric acid can also be added, when required, to meet total acid addition demand. The total acid addition targets nearly-complete destruction of acid soluble carbonates in the acidulation tanks. Acidulation is conducted in two reaction tanks. The acidulation tanks are agitated to disperse the slurry, acid and decant thickener overflow recycle throughout the tank and ensure the carbonates in the ore react with the acid in solution.

Depending on the ore type being processed the slurry from the grinding thickener underflow storage tanks is split between acidulation and the POX feed tanks. The proportion of this split is determined by how much carbonate in the feed material requires destruction to achieve the target of 22.5 g/L free acid content in the POX autoclave discharge slurry. This free acid level favours the formation of an iron mineral reaction product which exhibits better settling behaviour in downstream thickeners (hematite favoured over jarosite), while also reducing the potential for excessive CO_2 gas evolution and gypsum scaling in the POX autoclaves.

Additional concentrated sulfuric acid is added if required to maintain the targeted acid soluble carbonate destruction in the acidulation tanks. When there are low carbonate levels in the feed, and little or no acidulation is required, POX feed thickener overflow solution is recycled to the acidulation tanks (instead of decant thickener overflow solution) to limit the maximum concentration in the tanks to 30% solids.

Slurry overflows from acidulation tank 1 into acidulation tank 2 and then discharges into the POX feed thickener mix tank. Either of the acidulation tanks can be bypassed, if required. The diluted slurry from acidulation requires thickening prior to storage in the POX feed tanks. The POX feed thickener recovers excess solution and advances it to the decant thickener (as wash water) and/or to the iron / arsenic precipitation circuit (to maintain the water balance in the acidulation circuit) or recycles it to acidulation tank 1.

POX thickener underflow slurry is pumped to the POX feed thickener underflow surge tank. The storage in the surge tank allows blending in the correct proportions of the acidulated slurry with the un-acidulated grinding thickener underflow slurry in the POX feed tank to ensure the total level of acid soluble carbonates in the POX feed slurry is within target levels.

The decant thickener recovers acid (that is generated in the POX autoclaves) from the POX discharge slurry and recycles it to the acidulation circuit for carbonate destruction. The underflow slurry is pumped from the thickener to the iron / arsenic precipitation circuit by the decant thickener underflow pumps. Thickener overflow gravitates to the decant thickener overflow tank from where it is pumped to the acidulation tanks by the decant thickener overflow pumps. Solution is bypassed to the POX feed thickener overflow tank when processing low carbonate ores.

17.1.2.5 Pressure Oxidation

The POX feed surge tanks 1 and 2 are a common feed system that services both POX autoclave trains (T1 and T2). The tanks are agitated to mix / blend the incoming slurry and provide approximately 18-hours of slurry storage to minimise disruptions to the POX circuit. For simplicity, where only POX T1 is discussed in this document it is assumed that both T1 and T2 have identical configurations and controls.

Slurry is pumped to the POX trains 1 and 2 low temperature heaters by the POX heating feed pumps. The low-temperature (LT) heater receives incoming feed slurry and vent gas (predominantly steam) recovered from the LT flash vessel. The gas heats the slurry to approximately 95°C before being transferred to the high temperature (HT) heater. The steam in the gas condenses and any excess is vented to the wetted elbow of the POX T1 Venturi scrubber.

The HT heater receives slurry from the LT heater and vent gas (predominantly steam) recovered from the HT flash vessel. The gas heats the slurry to approximately 150°C before being pumped to the POX autoclave. The steam in the gas condenses and any non-condensing gases accumulate in the vapor space at the top of the vessel, prior to being vented.

Slurry is pumped to the autoclave by two pumping trains.

If one full autoclave train is offline, the remaining autoclave train can operate at 150% of normal capacity, provided both of its feed pumping trains are operating.

A horizontal multi-compartment autoclave is used to oxidise the sulfides in the ore at high temperature and pressure using gaseous oxygen. The oxidation of sulfide material in the autoclave generates heat and when the rate of heat generation exceeds that required to achieve the target temperature of 220°C quench water is added. Sufficient quench water is added to control the temperature to the target. The quench water is pumped through the same sparge pipe that introduces gaseous oxygen addition into the autoclave. There is one sparge pipe underneath each autoclave agitator.

A vent controls the pressure in the autoclave to prevent the water boiling. This pressure is called overpressure and results from the presence of gases such as oxygen, nitrogen and CO_2.

Slurry discharges from the autoclave through a severe service let down valve to the HT flash vessel. The HT flash vessel operates at a lower pressure than the autoclave and the resulting pressure drop for the discharge slurry entering the HT flash results in steam being flashed from the slurry. The flashing of steam cools the slurry to the equilibrium temperature corresponding to the pressure in the flash vessel.

Steam vented from the HT flash is sent to the HT heater to heat the feed to the autoclave, excess steam is vented to the venturi scrubber for treatment prior to discharge.

Slurry discharges from the HT flash vessel through a severe service let down valve to the LT flash vessel. The LT flash vessel operates at a lower pressure than the HT flash vessel, the resulting pressure drop for the discharge slurry entering the LT flash results in steam being flashed from the slurry. The flashing of steam cools the slurry to approximately 100°C at a pressure just above atmospheric. Slurry is forced from the HT flash vessel to the LT flash vessel by the pressure difference between the two vessels.

Steam vented from the LT flash is sent to the LT heater to heat the feed to the HT heater, excess steam is vented from the LT heater to the Venturi scrubber for treatment prior to discharge.

Steam, entrained slurry, together with gas, including carbon dioxide and unreacted oxygen vented from various points in the autoclave circuit, is scrubbed in Venturi scrubber to remove entrained acidic slurry droplets.

Demineralised water is used in the POX circuit for steam production and for seal water.

Flashed slurry is pumped from the LT flash vessel by decant thickener feed. The decant thickener was described previously and the decant thickener underflow is feed to iron / arsenic precipitation.

17.1.2.6 Fe/As Precipitation

Iron / arsenic precipitation uses limestone slurry addition to the decant thickener underflow slurry to neutralise the free acid and raise the pH to approximately 2.8, which removes ferric iron and arsenic from solution.

The decant thickener underflow duty pump transfers the thickener underflow slurry to iron / arsenic precipitation tank 1. Limestone is added for pH control, and low-pressure air is sparged into the tanks to oxidise any ferrous iron that may be present to ferric iron. The ferric ions combine with the residual arsenic, also leached in the POX circuit, and precipitate together as the pH of the solution is raised. Limestone reacting with the free acid generates carbon dioxide gas and gypsum.

The two iron / arsenic precipitation tanks normally operate in series. The treated slurry overflows from the second iron / arsenic precipitation tank to the CCD 1 Mix Tank.

The low-pressure air and CO_2 generated during the limestone neutralisation reactions rise above the slurry surface on top of the tanks and carry some entrained solution / slurry. These off-gases from the iron / arsenic precipitation tanks (1 and 2) are vented via the iron / arsenic precipitation tank fans 1 and 2 and fed to the iron / arsenic scrubber.

The iron / arsenic scrubber is a Venturi type scrubber. The off gases are cooled and scrubbed of the entrained solution / slurry in the scrubber. The clean gases are emitted to the atmosphere.

17.1.2.7 Counter Current Decantation

Counter current decantation (CCD) washes the iron / arsenic stage discharge slurry with process water using two stages of thickeners operating in counter current mode. The remaining soluble metals in solution exiting the iron / arsenic precipitation circuit are washed from the slurry and report to CCD 1 overflow. The slurry discharging from CCD 2 underflow has the soluble metals washed from the slurry to sufficiently low levels to feed into the cyanide leach circuit.

CCD thickener 1 overflow solution gravitates into the CCD thickener 1 overflow tank. The duty CCD thickener 1 overflow pump transfers the CCD thickener 1 overflow solution to the neutralisation circuit. The CCD thickener 1 underflow pump transfers the thickener underflow slurry to CCD 2 mix tank. Process water is added in the CCD 2 mix tank as wash solution to wash the solids. Diluted flocculant solution is added in the CCD 1 and 2 thickener feeds to aid in the settling of solids in the thickeners. Duty CCD thickener 2 underflow pump transfers the underflow slurry from the CCD thickener 2 to the pre-leach tank.

17.1.2.8 Cyanide Leach, Carbon Adsorption and Detoxification

The cyanide leach circuit consists of one pre-leach tank and two leach tanks. Slurry is received in the pre-leach tank from the duty CCD thickener 2 underflow pump and flotation tails. The pre-leach tank has a residence time of nominally 10 minutes and is used to raise the pH of the slurry to pH 10–11 prior to the slurry entering the leach tanks where cyanide is added for gold leaching.

The leach tanks have a total residence time of up to six hours and slurry flows through the leach tanks by gravity and discharges the final leach tank to enter the carbon adsorption circuit. The leach tanks operate at 30% solids concentration and have low pressure air and oxygen, from the Air Liquide oxygen plant, added to maintain sufficient oxygen in solution for gold leaching.

The carbon adsorption circuit consists of six agitated tanks with a total residence time of up to 12 hours. Each tank contains activated carbon to adsorb the leached gold contained in solution. Slurry flows by gravity from tank 1 to tank 6 and discharges into the detoxification circuit. Carbon flow is counter-current to slurry and therefore is transferred stage wise from tank 6 through to tank 1, using dedicated recessed impeller pumps. Each tank has interstage screens installed so that the carbon remains in each tank and does not follow the direction of the slurry flow.

Gold is loaded onto the carbon as it moves from tank 6 to tank 1 and reaches its maximum loading in adsorption tank 1. The loaded carbon is pumped from adsorption tank 1 to the loaded carbon screen where spray water on the screen washes the carbon prior to it entering the elution column for carbon desorption and recovery of gold through the refining circuit.

Slurry exiting adsorption tank 6 flows to the detoxification circuit where destruction of the residual cyanide contained in the slurry occurs. The detoxification circuit consists of one tank with a total residence time of one hour. Air and sodium metabisulfite are added to the circuit to destroy the residual cyanide down to a concentration of less than 5 ppm CNWAD. Residual copper in the slurry catalyses the cyanide destruction process.

17.1.2.9 Carbon Desorption and Refining

The carbon desorption method selected is a split AARL elution. A common stainless steel column is used for acid wash, cold cyanide strip for copper, when required, and a hot gold elution cycle to recover gold. The elution column is a 6 t column and is designed to handle the stripping of three carbon batches per day. Loaded carbon enters the elution column via the loaded carbon screen.

The first step of stripping the carbon is an acid wash using nitric acid solution to remove loaded impurities such as calcium. After the acid wash, a pre-soak solution is added to the elution column prior to commencement of the eluent recycle for initial stripping of copper, when required, followed by a hot elution cycle to strip gold from the carbon.

Pregnant eluate is collected in the pregnant eluate tank and pumped through electrowinning cells with gold metal plated out onto stainless steel cathodes. Smelting of gold recovered from the stainless-steel cathodes is conducted in the gold refinery.

Desorbed carbon from the elution column is regenerated through a horizontal diesel fired rotary kiln to remove organic material loaded onto the carbon.

17.1.2.10 Neutralisation and Tailings

Slurry from cyanide destruction and the CCD 1 thickener overflow solution are neutralised with lime to precipitate residual metals in solution. Air is added for the oxidation and removal of ferrous iron and manganese.

Normally the two neutralisation tanks operate in series. Discharge from the neutralisation feed box gravity flows into neutralisation tank 1 prior to overflowing into neutralisation tank 2. Discharge from neutralisation tank 2 gravitates into the tailings thickener mix tank.

The first neutralisation tank is equipped with a sodium metabisulfite addition system and this allows it to be used for the detoxification step, when the normal detoxification tank is bypassed for maintenance or descaling. Both neutralisation tanks can also be bypassed as required to allow for maintenance.

The discharge slurry from neutralisation flows by gravity into the tailings thickener mix tank before overflowing into the tailings thickener. Tailings thickener overflow water overflows directly into the process water storage tank. The underflow slurry from the tailings thickener is pumped to the agitated tailings tank. The discharge slurry from the tailings tank is pumped to a TSF on a continuous basis via the 4.3 km long tailings pipeline.

A schematic flow sheet of the process is shown in Figure 17.6 including the flotation circuit addition.

17.1.2.11 Tailing Storage Facility

The process tailings slurry is deposited into the TSF for final storage. Operators will alternate the location within the facility where the tailings are deposited to maximise the storage and dewatering within the facility.

In the TSF the solids compact and reject excess water which is recovered for recycling to the process plant. The controlled deposition of tailings at alternating locations around the perimeter of the TSF creates a pond that collects water, which decants from the tailings slurry as it settles and compacts. This decant water collected within the pond area is recycled to the process water system tank via the tailings water reclaim pumps.

The Tailings Storage Facility (TSF) is developed and constructed in stages. TSF 1 phase 3 is under construction in Q4'20, the development of TSF 1 is to phase 7. Construction and development of only TSF 1 will provide storage of tailings for up to 70.8 Mt, which is more than sufficient to accommodate the CDMP20 tailings to be produced. TSF 2 construction is not included in the mine plan but remains as an option for further expansions.

17.1.2.12 Reagents

There are ten major reagents used in the process plant, listed as follows:

- Oxygen
- Sulfuric acid
- Limestone
- Sodium hydroxide
- Flocculant
- Sodium metabisulfite
- Milk of lime
- Sodium cyanide
- Nitric acid
- Antiscalant

The flotation plant has the following main reagents:

- Frother
- Collector
- Copper Sulfate

All reagents are delivered in bulk tankers, containers or bags with storage on site. Any reagents that require dilution or mixing prior to use are prepared on site on a batch wise basis, as required. Oxygen is produced on site supplied from an Air Liquide owned and operated oxygen plant under a gas supply agreement. Additional oxygen can be delivered as liquid into on-site storage.

17.1.2.13 Utilities

The major utilities used in the process plant are as follows:

- Iron / arsenic low-pressure air
- CIP leach low-pressure air
- Plant air
- Instrument air
- Raw water
- Fire water
- Potable water
- Process water
- Diesel fuel

These utilities are reticulated throughout the process plant to their end user.

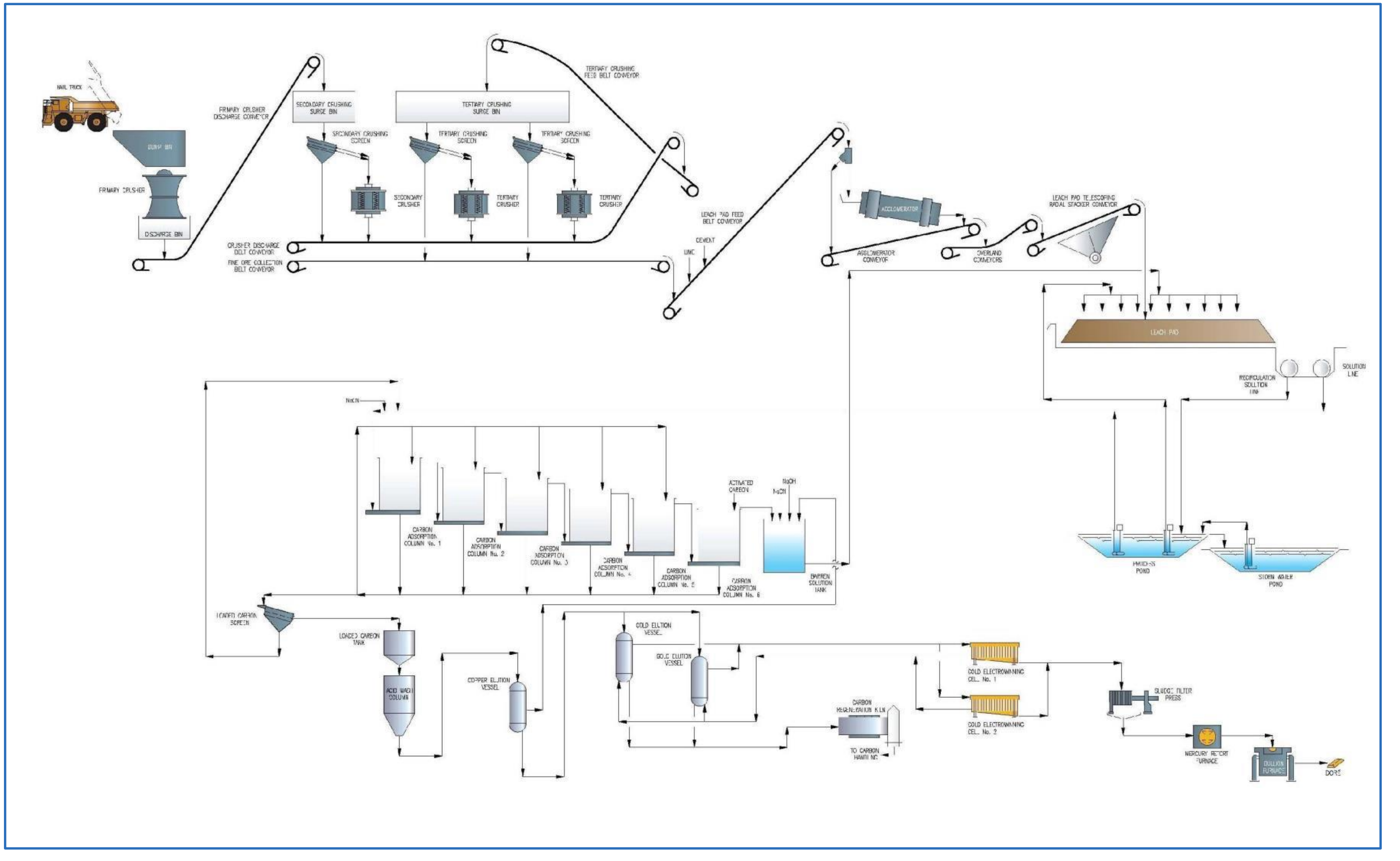
17.2 Oxide Heap Leach Processing

In the Reserve Case production is predominantly from sulfide ore, the maximum oxide ore placed in any year is 1.2 Mt for a total production of 7.9 Mt. The oxide heap leaching and associated facilities were commissioned in the second half of 2010 and initial gold production was achieved in Q4'10. The process was originally designed to treat approximately 6.0 Mtpa of ore by three-stage crushing (primary, secondary, and tertiary) to 80% passing 12.5 mm, agglomeration and heap leaching on a lined heap leach pad with dilute alkaline sodium cyanide solution. Gold is recovered through a carbon-in-column (CIC) system, followed by stripping of metal values from carbon, electrowinning and melting to yield a doré (containing gold and silver) suitable for sale. Control of copper in leach solutions is undertaken in a sulfidisation, acidification, recycling and thickening (SART) plant which also regenerates cyanide. The process flow sheet is summarised in Figure 17.7.

17.2.1 Oxide Heap Leach Performance

Since commissioning through the end of December 2019, an estimated 50.5 Mt of oxide ore was placed on the heap at an average grade of 1.37 g/t Au. At the end of December 2019, a total of approximately 1,670 koz had been produced as bullion.

Figure 17.7 Heap Leach Process Flow Sheet



Anagold, 2016

18 PROJECT INFRASTRUCTURE

18.1 Introduction

The facility infrastructure supports the mine, and process areas of oxide heap leach and sulfide plant. The existing infrastructure, and the tailings storage and heap leach pad area when the planned expansion are complete will be sufficient for the current Mineral Reserves. The infrastructure for the addition of flotation to the sulfide plant will be supported by the existing facility infrastructure with some components modified to meet the addition of the flotation circuit. The flotation circuit will be located within the sulfide plant foot-print adjacent to the grinding circuit building. The Reserve Case site plan is included as Figure 18.1.

The current leach pad consists of four phases designed to accommodate approximately 58 Mt of oxide ore heap with a nominal maximum heap height of 100 m above the pad liner. The additional two phases (5 and 6), with a capacity of 20 Mt are yet to be approved. The Heap leach pad area continues to be developed in advance of stacking area required.

The tailings storage facility (TSF) is developed and constructed in stages. TSF 1 phase 3 is under construction in Q4’20, the development of TSF 1 is to phase 7. Ongoing work in ensuring sufficient long-term capacity for storage of tailings has been undertaken. Studies by Anagold have determined, that the effect of the addition of the flotation plant to the sulfide plant circuit would result in an increase in the solids content and improvement in the final settled density based on an increase in the rate of tailings consolidation.

The TSF is developed and constructed in stages. TSF 1 phase 3 is under construction in Q4’20, the development of TSF 1 is to phase 7. Construction and development of TSF 1 will provide storage of tailings for up to 70.8 Mt, more than sufficient to accommodate the CDMP20 tailings to be produced.

A PFS level study (TSF 2) has been carried out that identifies approximately 13.4 Mt additional tailings storage capacity in a site adjacent to TSF 1, should it be required in the future.

18.1.1 Existing Infrastructure

The existing site infrastructure supporting the existing operation includes the following:

- Site security gate and guard station
- Site administration building
- Site warehouse
- Site assay laboratory
- Container or modular type offices
- Cyanide receiving and mixing system
- Site kitchens and eating areas
- Site single living dormitory with adjacent multi-purpose room
- Site family housing.

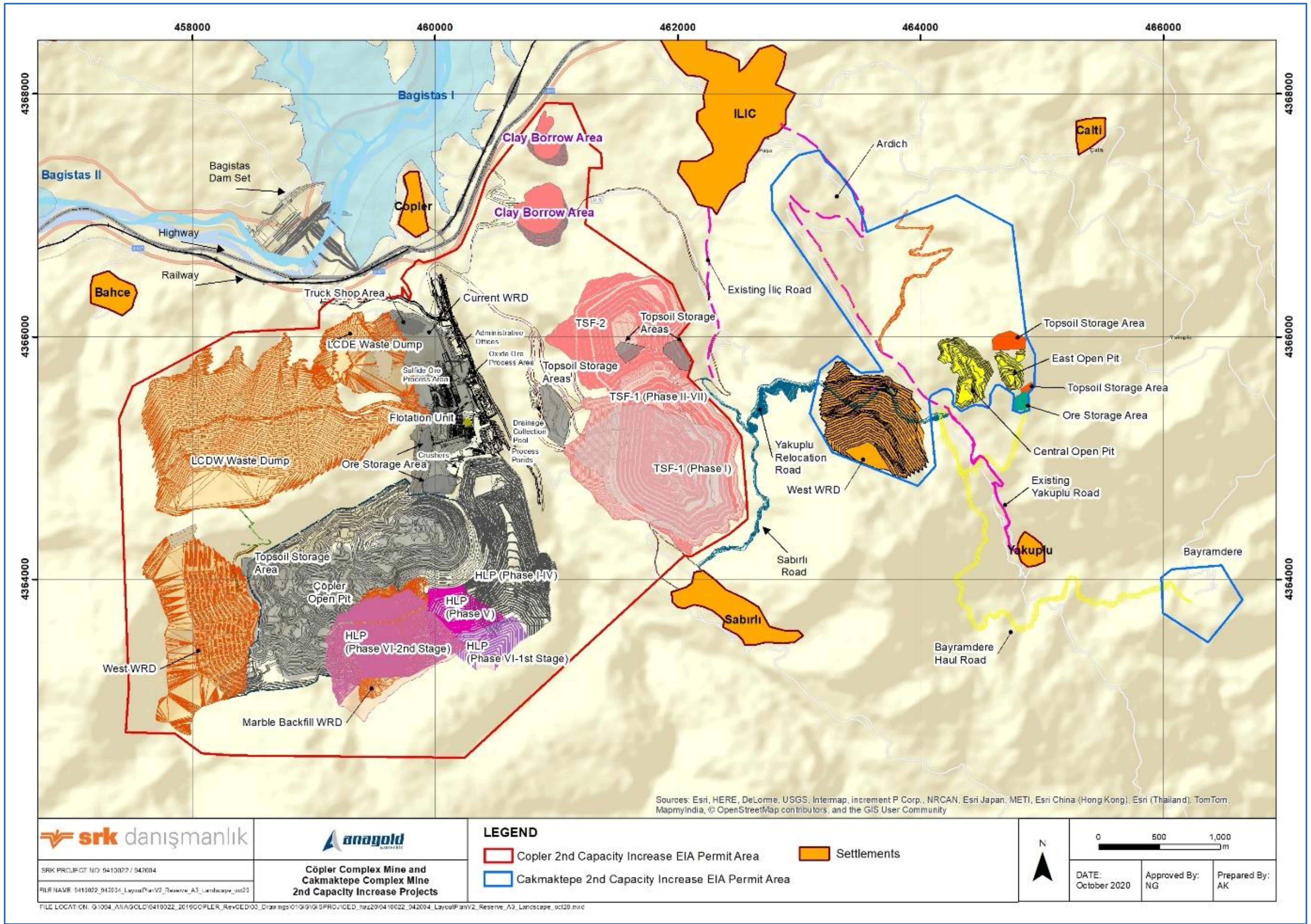
- Contractor (mining) dormitories, kitchens, and offices
- Site raw water wells, pumping system and storage tanks
- Site potable water treatment and distribution system
- Two sanitary wastewater collection and treatment systems
- Sulfide maintenance building
- Sulfide control rooms
- Combined oxide and sulfide gold refinery building
- Sulfide process buildings:
  - Grinding building
  - POX building
  - Carbon desorption building
- Tailings pump building
- Main control room and electrical building
- HV switchyard electrical building
- Crusher electrical building
- POX flocculant building
- Limestone building
- Potable water booster pump house
- Reagent building
- Tailings and process water pump house
- Plant and instrument air compressor building
- CCD electrical building
- Reagent dry storage
- Leach air compressor building
- Aw water pump building
- Lime slaking (MOL) building
- Fe/As air compressor building
- Emergency diesel generators building
- TSF reclaim electrical building
- TSF drainage tank electrical building
- TSF OD-UD pond electrical building
- CIP CCD ablutions block
- Pump shelters with monorails
- Carbon elution building – electrical room

- Raw water bores P/P house and electrical building
- Gatehouse
- Fire water pump house
- Community relations centre
- Raw water wells

18.1.2 Flotation Building

The flotation circuit will be an insulated engineered building. The building is equipped with an overhead crane for flotation cell and pump maintenance. Flotation reagent mixing and distribution are contained in a lean-to off the main flotation building.

Figure 18.1 Reserve Case Site Plan



Anagold, 2020

18.2 Site Water Management

18.2.1 Hydrology Background

The only perennial surface water in the vicinity of the Çöpler Mine is the Karasu River flowing in the northern and western part of the area. All other valleys are either ephemeral streams or dry valleys. The average flow rate of the Karasu River measured at the Bağıştaş / Karasu Gauging Station in the upper Euphrates Basin, is approximately 145 m³/sec, draining an area of 15,562 km². A hydroelectric dam (Bağıştaş -1 Dam) was built on the Karasu River downstream of the mine site. When the reservoir is at high levels the impoundment will extend into the very lower reaches of both the Çöpler and Sabırlı Creeks and the maximum inundation elevation will be 916.5 m as it is released into the spillway. The Çöpler and Sabırlı streambeds in the project area do not flow perennially. They both discharge into the Karasu River. The drainage area of the Sabırlı Creek is approximately 35 km² and that of the Çöpler Creek is approximately 10 km².

The project submitted a Five-Year Water Management Report in December 2019, prepared by SRK Danışmanlık ve Mühendislik A.Ş., as part of the EIA conditions. This report benchmarks the expected results with those achieved. Overall results achieved were generally as predicted. In 2020, as part of updating the EIA, further hydrogeology studies have been undertaken by SRK Danışmanlık ve Mühendislik A.Ş. The report has updated the surface water and hydrological models based on actual data over the operating period of the mine to improve the model.

18.2.2 Site-Wide Surface Water Hydrology

Existing mine site facilities are located primarily within the Çöpler and Sabırlı Creek watersheds immediately upstream of their confluence with the Karasu River. Site-wide surface water management for the included diversion facilities consist of a network of diversion channels and retention structures to minimise storm water run-on to the mine site facilities to prevent mine-impacted storm water run-off from exiting the site and discharging to the Karasu River.

The sub-basin areas, characterisation of the surface run-off conditions, and design rainfall data were used to construct the existing conditions hydrology model. The hydrology analysis utilised HEC-HMS software to develop estimates of the peak flow rates and volumes generated by the existing watersheds.

18.2.3 Surface Water Management Structures

Engineered surface water management structures are constructed to minimise effects of storm water run-off to critical mine facilities and to control the release of mine-impacted water to the environment. A combination of interim and permanent diversion channels and retention ponds are utilised to achieve these goals. Interim structures will be reclaimed at closure while permanent structures will remain in place post-closure. Other flood control structures were developed to control or direct runoff away from pit crests and are planned for runoff that does not discharge to surface water drainages or streams and therefore do not require lining. Sediment ponds to control runoff and sediment release are lined based on the EIA commitments. Interim diversion channels are designed to convey the 25-year storm event with 1.5 m of freeboard and the 100-year storm with no freeboard. Permanent diversion channels are designed to convey the 100-year storm with 0.5 m of freeboard. Lined sediment ponds are downgradient of the waste dumps and are sized to contain the 100-year run-off volume with an emergency spillway to safely discharge the peak flow. The TSF is designed to contain the volume generated by the 24-hour PMP within the operating freeboard.

18.2.4 Fresh Water Supply

Fresh water is supplied by existing wells to the site, supporting the operation. Figure 18.2 shows the location of the mine water extraction wells. An additional three wells were developed in 2018, wells WM-45, WM-46 and WM-47, to increase water supply for the project. Two raw water storage tanks support the demands of the heap leach and sulfide process equipment and the fire water requirements.

Figure 18.2 Mine Water Supply Well Locations

Anagold, 2020

18.2.5 Potable Water Treatment

The site is serviced by a potable water treatment system and distribution system. The system consists of multi-media filtration, carbon filtration, ultraviolet (UV) disinfection system (plus further softening and reverse osmosis for water used in the dining room), which directly feeds the site potable water distribution system.

18.2.6 Waste Management

Waste will be generated from multiple sources such as human waste, food spoilage, and process and maintenance wastes.

Hazardous wastes will be contained, packaged and disposed of in accordance with local, regional and national regulations. Non-hazardous wastes will either be buried on site or transported offsite to the appropriate processing site in accordance with local, regional, and national regulations.

18.3 Power to Site

The existing 154 kV line provides power to the mine and process plant. The following structures are associated with site power distribution:

- HV switchyard 154 kV
- Main electrical building
- Oxygen plant substation
- CCD electrical building
- Crushing electrical building
- Grinding electrical building
- Carbon elution electrical room
- TSF area electrical buildings
- Bore field area electrical building

18.4 Emergency Backup Power

Motors and loads for certain critical equipment and systems were identified as requiring power in the event of a utility outage. A load shedding scheme is applied to feed critical electrical users automatically in the event of a utility outage.

Generators are diesel fuelled with a minimum of eight hours of diesel storage based on generators operating under full load.

18.5 Communications

The Project uses networks for the DCS, PMS, the integrated process related and security CCTV system, security systems (access control / card reader), information technology (IT) and telephones and communication between the DCS and packaged control systems.

Single mode fibre and copper cabling is distributed within the sulfide plant area and selected buildings for the tailing pipeline and dam.

18.6 Site Roads

The Çöpler project has access provided via the main access road and sulfide plant roads.

Generally, site roads have an overall width of 6 m and provide everyday operational access for large trucks or facility access for site personnel vehicles. These roads are limited to a maximum grade of 9%. All roads are compacted hardstand surfaced with 100 mm wearing course and cross-sloped to provide positive drainage.

18.7 Plant Fire Protection System

A separate plant fire protection system is provided for the sulfide facility and will include the flotation building.

A combined sprinkler, hose reel and hydrant underground piping system is provided for the active fire protection of the facility.

A gas-based fire suppression system is used in the main control and electrical building.

18.8 Site Water Management

18.8.1 Hydrology Background

The only perennial surface water in the vicinity of the Çöpler Mine is the Karasu River flowing in the northern and western part of the area. All other valleys are either ephemeral streams or dry valleys. The average flow rate of the Karasu River measured at the Bağıştaş / Karasu Gauging Station in the upper Euphrates Basin, is approximately 145 m^3/sec, draining an area of 15,562 km^2.

A hydroelectric dam (Bağıştaş -1 Dam) was built on the Karasu River downstream of the mine site. When the reservoir is at high levels the impoundment will extend into the very lower reaches of both the Çöpler and Sabırlı Creeks and the maximum inundation elevation will be 916.5 m as it is released into the spillway.

The Çöpler and Sabırlı streambeds in the project area do not flow perennially. They both discharge into the Karasu River. The drainage area of the Sabırlı Creek is approximately 35 km^2 and that of the Çöpler Creek is approximately 10 km^2.

The project submitted a Five-Year Water Management Report in December 2019, prepared by SRK Danışmanlık ve Mühendislik A.Ş., as part of the EIA conditions. This report benchmarks the expected results with those achieved. Overall results achieved were generally as predicted.

In 2020, as part of updating the EIA, further hydrogeology studies have been undertaken by SRK Danışmanlık ve Mühendislik A.Ş. The report has updated the surface water and hydrological models based on actual data over the operating period of the mine to improve the model.

18.9 Heap Leach Facility

The Heap Leach includes the leach pad and collection ponds that consist of process ponds and a storm pond. The current leach pad consists of four phases and designed to accommodate approximately 58 Mt of oxide ore with a nominal maximum heap height of 100 m above the pad liner. The additional two phases, 5 and 6, with a capacity of 20 Mt are yet to be approved.

The heap is stacked in 8 m thick horizontal lifts at the natural angle-of-repose with intermediate benches to achieve an overall heap slope of 2H:1V.

18.9.1 Heap Leach Pad Development

The Heap Leach facility pad development is in six phases, and is in the same geographical area, adjacent to the Çöpler open pit as shown on Figure 18.1. The Heap Leach phases 1 to 3 are completed with the phase 4A pad constructed and, as at the report date, is being stacked and under leach with ore.

The remaining phases of pad development 4B, 5 and 6 are yet to be constructed and will have a combined capacity of 25.4 Mt.

Phase 4B pad development is underway, as of the report date.

The phase 5 (14.9 Mt capacity) is awaiting EIA approval for pad construction, expected December 2020. Work is underway, prior to pad development approval, on sub-base preparation with waste removal and back stability cuts in preparation of pad construction once approvals are obtained.

The phase 6 (5 Mt capacity) sits above phase 4B and 5 and will be the last to be constructed and stacked. Approvals and construction will be scheduled well in advance of being required for ore stacking and leaching.

18.10 Tailings Storage Facility

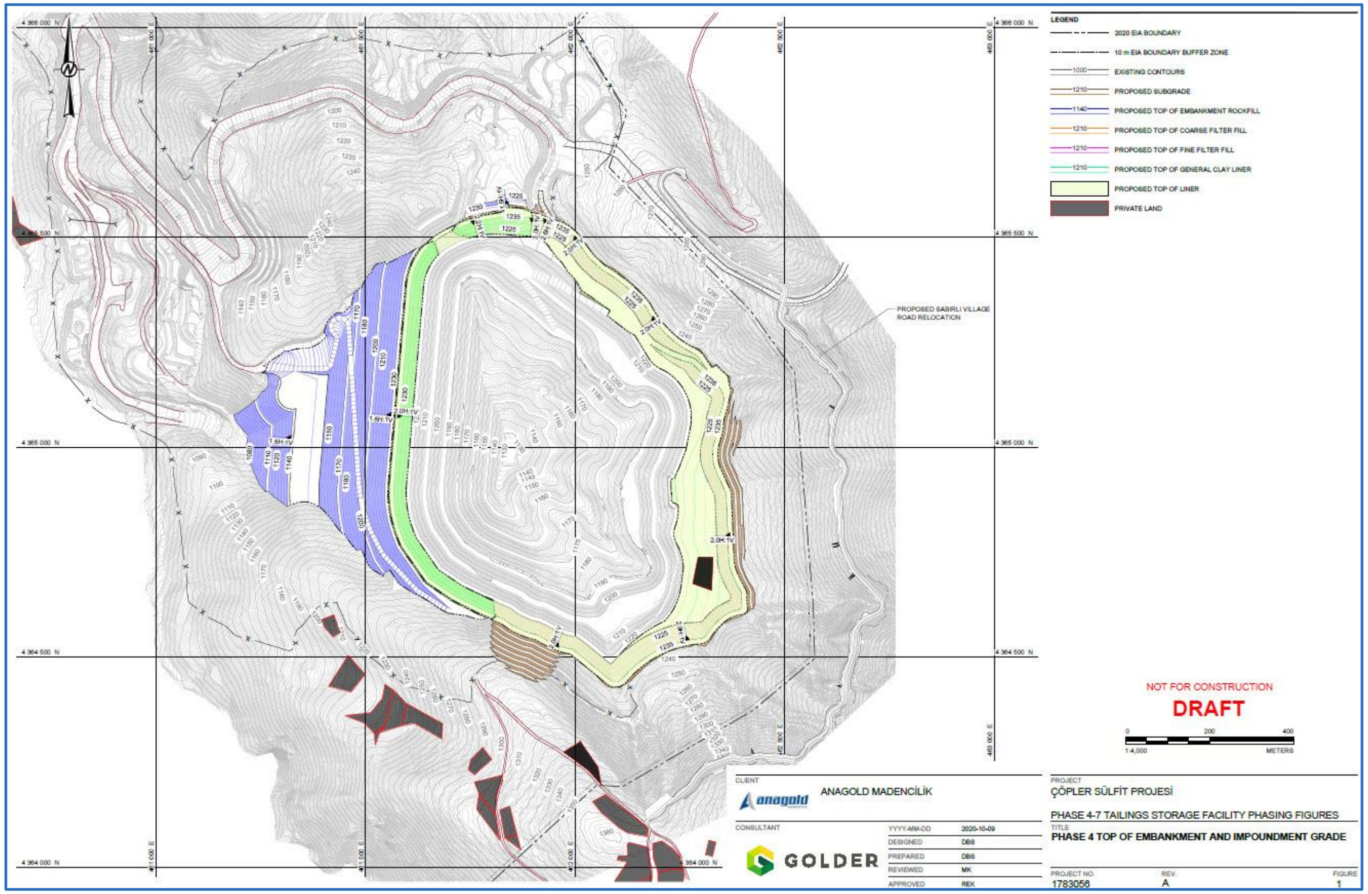
The existing Tailings Storage Facility (TSF) at the Çöpler mine was designed by Golder Associates Inc. (Golder) with support from Golder Associates Turkey, Ltd (Golder Turkey). The TSF initial design was developed to provide a capacity of 45.9 Mt through six phases with a crest elevation of 1,265 m. The TSF was permitted through submittal of a Turkish Design Application Report to the Turkish Ministry of Environment and Urbanization and subsequently approved based on the design through phase 5.

Anagold is advancing the development of the Çöpler Mine. Recently developed a prefeasibility level design for an additional TSF, referred to as TSF 2 in the valley adjacent and to the north of the existing TSF 1. Both TSF 1 and TSF 2 were included in the EIA submitted by Anagold in 2014. The current designs for TSF 1 and TSF 2 are within the 2014 EIA boundaries, except for a small portion of TSF 1, phase 7. Expansion beyond phase 3 of TSF 1 is currently limited by the pending construction and re-routing of a new road to Sabırlı Village as well as purchase of some small tracts of private land located within the phase 4 limits on the east side of the existing road to Sabırlı Village. Construction of the new Sabirli Village road is scheduled to begin in Q4'20. Anagold is also actively working to procure the private land.

Based on the prefeasibility design, TSF 2 has capacity for 13.4 Mt. To maximise capacity of TSF 1, phase 7 was developed as part of the design to a crest elevation of 1,280 m at a conceptual level and to support further planning, including planned updates to the site Environmental Assessment. Select engineering evaluation of phase 7 has been completed to support future planning including updated stability analysis, water balance, and consolidation modelling. Anagold's preference is to continue with development of TSF 1 phase 4 and to consider other options, if required depending on tailings capacity requirements, due to the higher capital costs related to construction of TSF 2 at this time. Without construction of TSF 2, TSF 1 alone provides for tailings capacity of up to 70.8 Mt through phase 7.

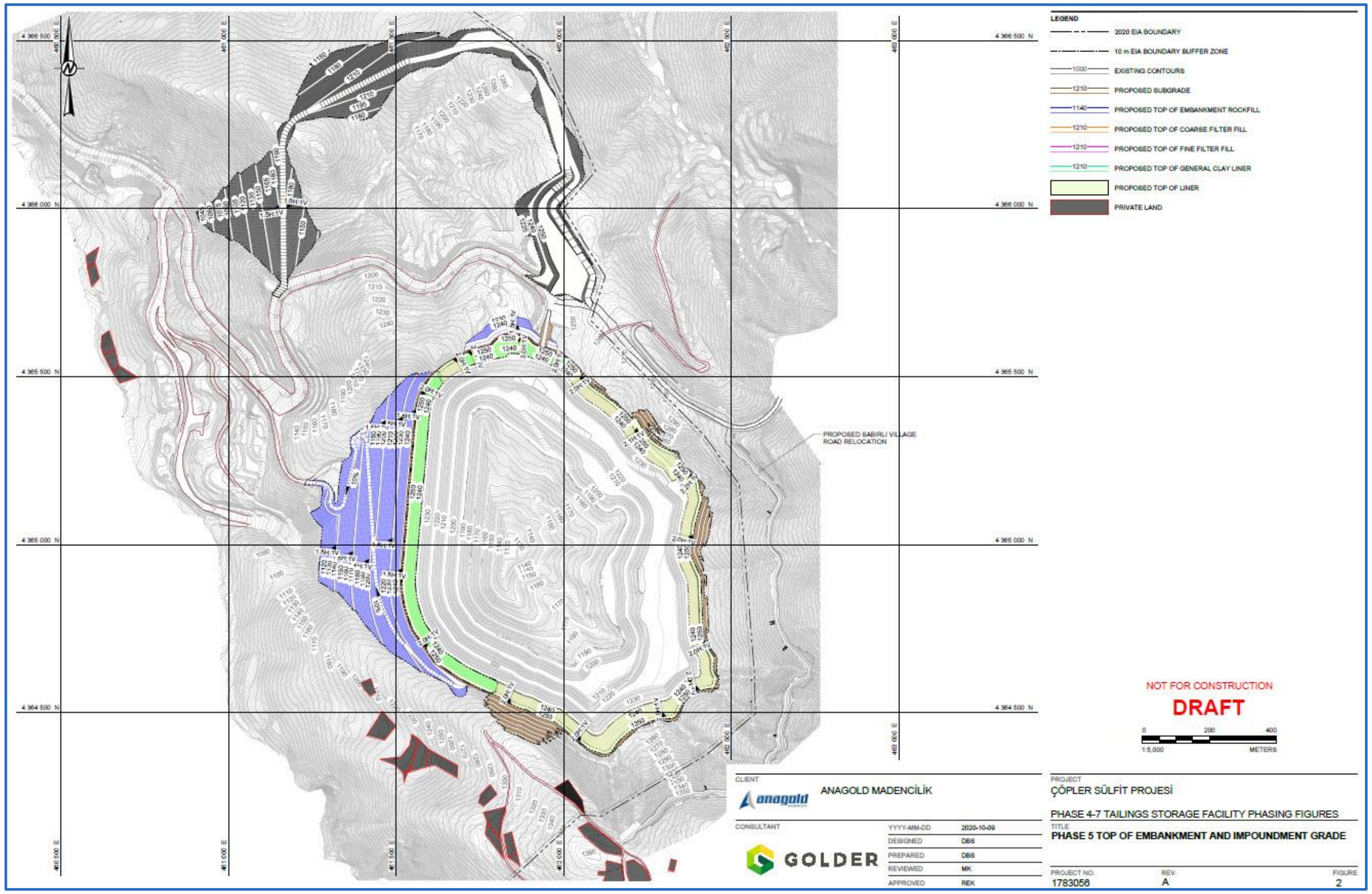
Figure 18.3 through to Figure 18.7 show the revised TSF 1 design for phases 4–7, and the TSF 2 design.

Figure 18.3 Phase 4 – Top of Embankment and Impoundment Grade



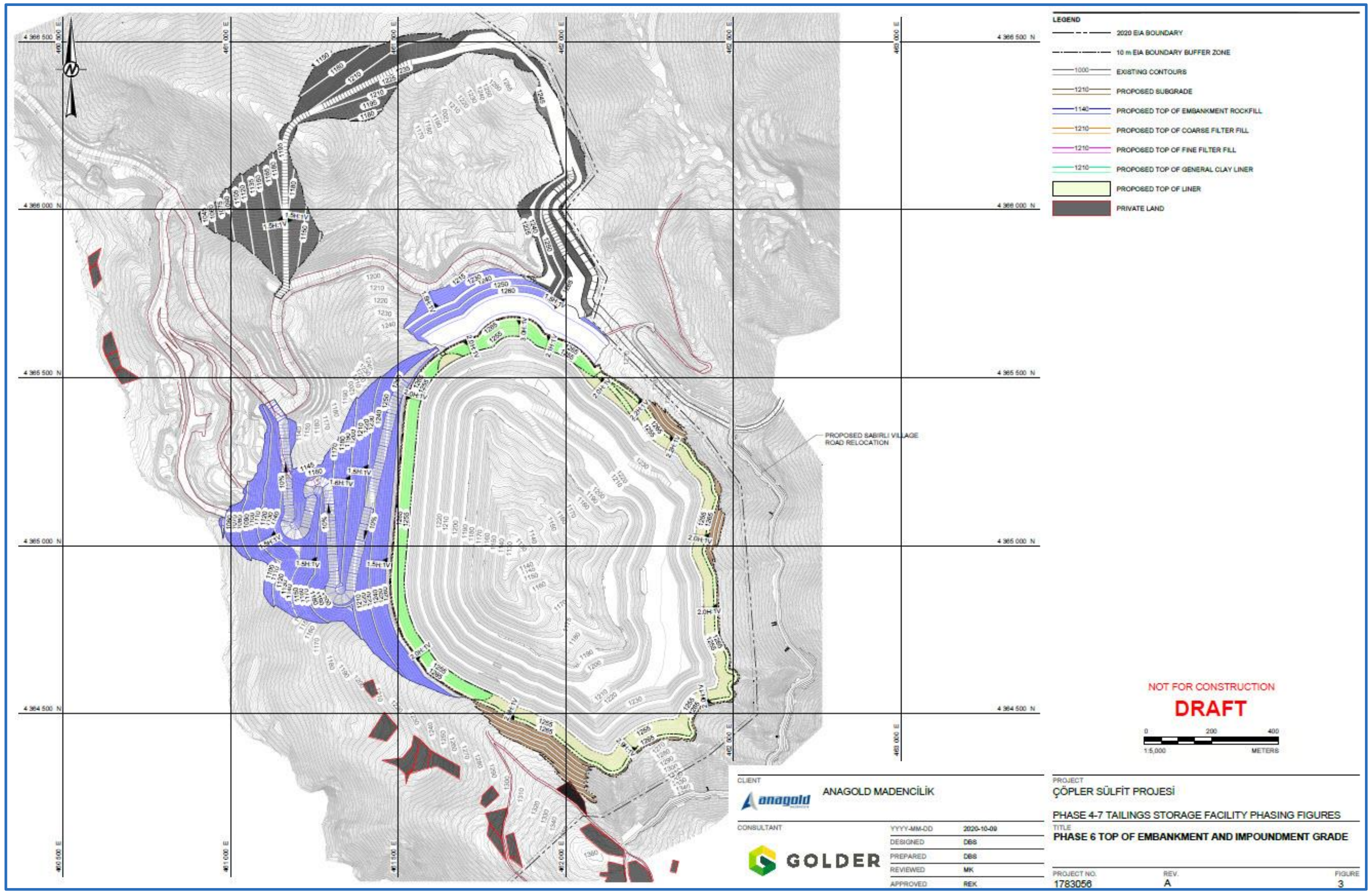
Anagold, 2020

Figure 18.4 Phase 5 – Top of Embankment and Impoundment Grade



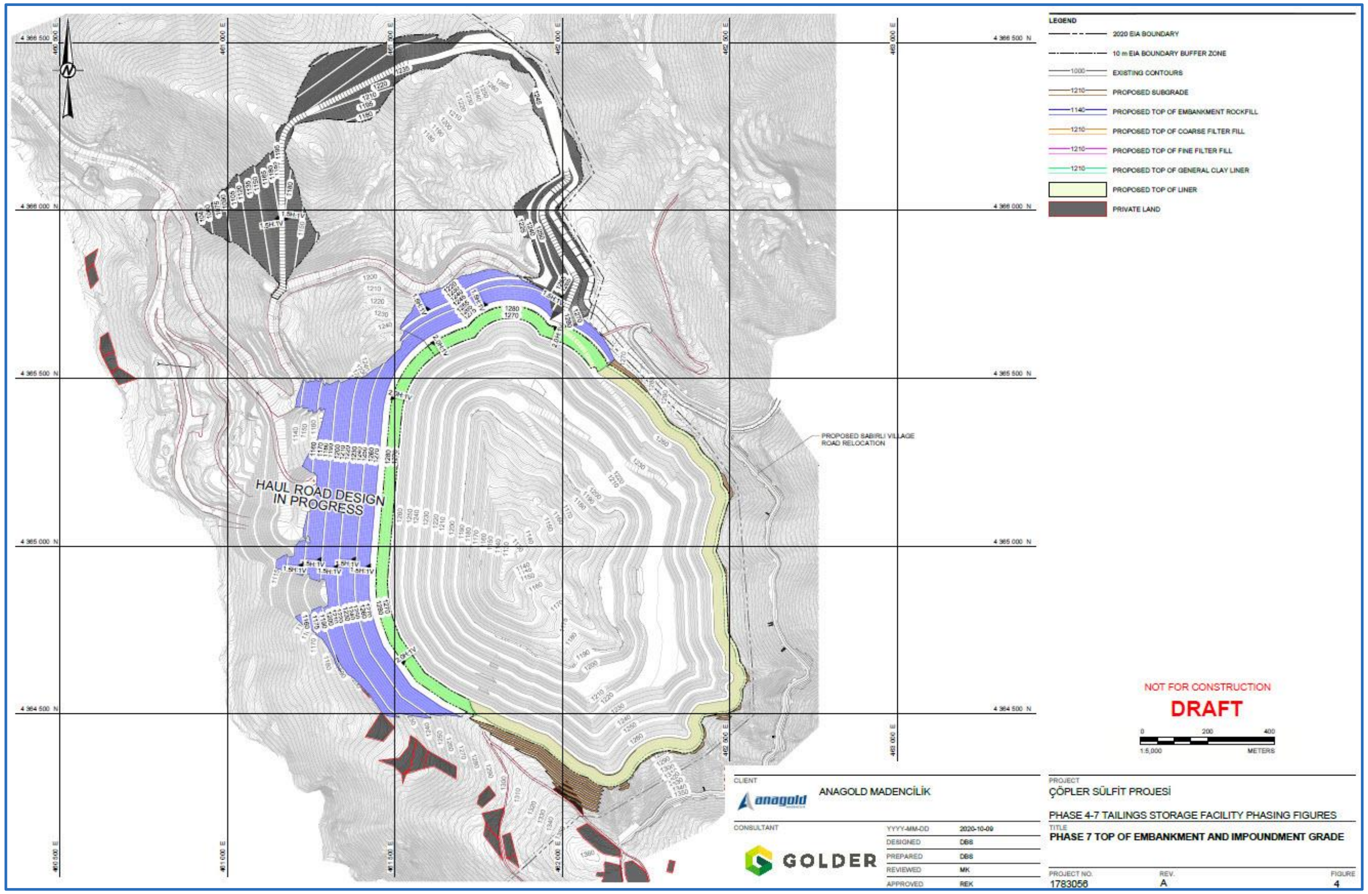
Anagold, 2020

Figure 18.5 Phase 6 – Top of Embankment and Impoundment Grade



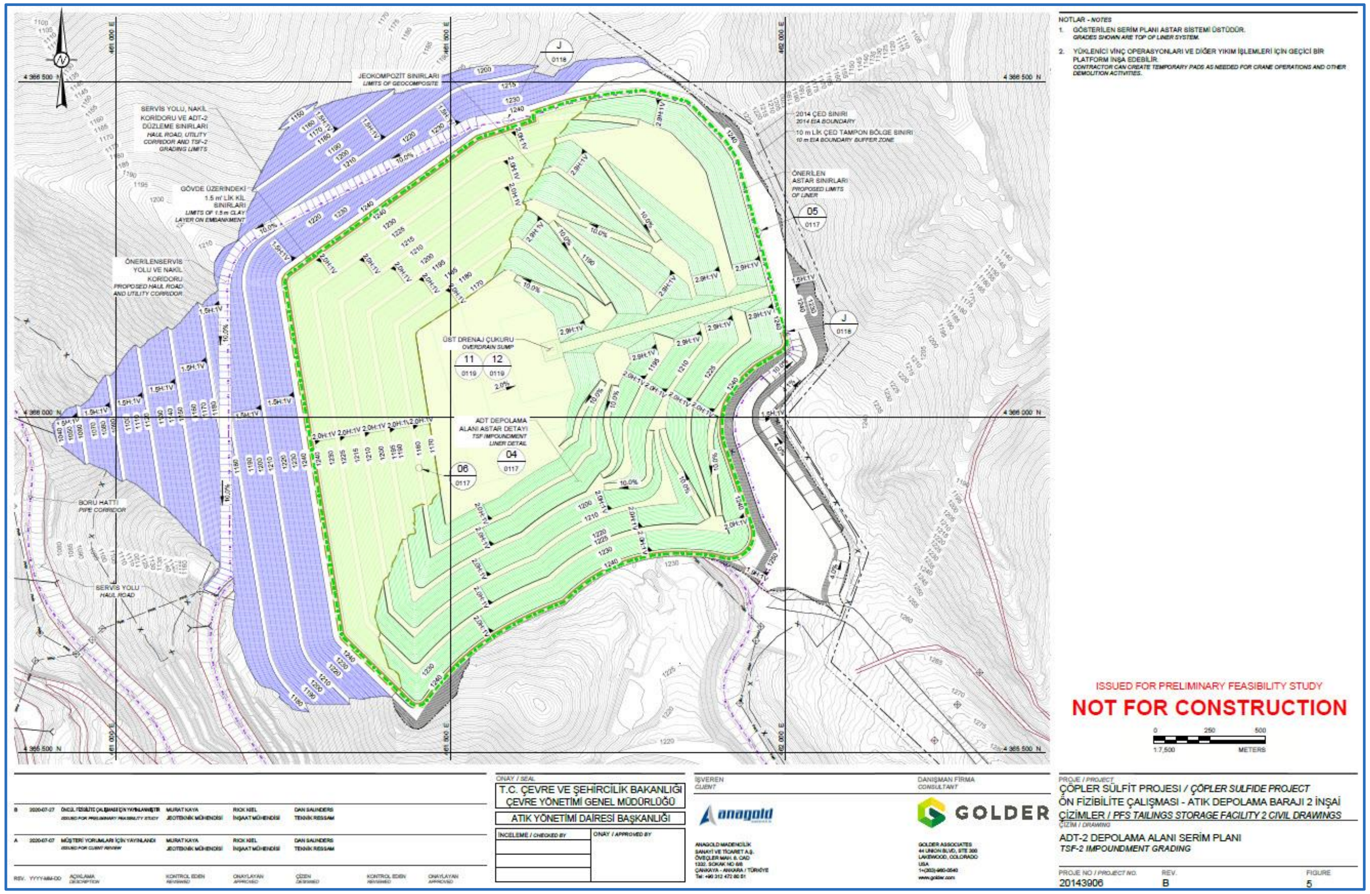
Anagold, 2020

Figure 18.6 Phase 7 – Top of Embankment and Impoundment Grade



Anagold, 2020

Figure 18.7 TSF 2 Impoundment Grading



Anagold, 2020

18.10.1 TSF Development and Summary of Current Operations

Construction of phase 1 of TSF 1 began in December 2016 and was completed in November 2018 with commissioning of the sulfide plant. Tailings were deposited initially from the emergency spigot and then typically from two to three spigots around the perimeter of the 1,190 m crest of the phase 1 embankment. The tailings initially have exhibited a solids content on the order of 24%. During the first two years of operations 4–5 m of water has been present over the top of the tailings surface. Reclaim water was managed by pumps on a rail-mounted sidehill reclaim system. The second raise, or phase 2 of TSF 1, was completed in April 2020 and construction of phase 3 is ongoing. The management of reclaim water has improved in the past year and currently the tailings surface is nominally 3 m below the top of the tailings water. A bathymetry survey was completed on 11 September 2020 and indicated a tailings average dry density of 0.68 t/m^3.

The reclaim water management system was converted to a conventional pontoon system accessible for maintenance from ramps constructed within the northern portion of the impoundment. Based on additional tailings testing completed in early 2020, the solids content of the tailings has improved to approximately 28% as a direct result of improved throughput stability at the sulfide plant, improvements to type of flocculants used and process control in the tailings thickener. As part of this tailings testing in early 2020, Golder evaluated the effect of the addition of the flotation plant to the sulfide circuit. The testwork indicated an increase in the solids content to 34% and improvement in the final settled density based on an increase in the rate of tailings consolidation.

18.10.2 Site Classification

The facilities are classified in accordance with the Canadian Dam Association (CDA) guidance (2013 Edition) as 'Significant' for the operational and post-closure phases. The 'Significant' classification is the second lowest in terms of risk with the Dam Classes being from least risk to greatest risk: Low, Significant, High, Very High, and Extreme.

18.10.3 Monitoring and Inspection

An Operational, Maintenance, and Surveillance (OMS) Plan was prepared by Golder with input and support from Anagold. The OMS Plan was prepared in accordance with the Turkish mining regulations (MoEU 2017) with additional guidance published by the Mining Association of Canada (MAC 2019). The OMS Plan is a 'living document' that is updated on an annual basis. In addition to providing the basic guidance for the management of process fluids, the OMS Plan does the following:

- Summarises the roles and responsibilities of Anagold personnel.
- Presents a description of the facility and pertinent design details.
- Provides maintenance and surveillance parameters and procedures.
- Outlines abnormal operating conditions.
- Details emergency preparedness and response protocols.
- Presents a conceptual closure plan.

The OMS Manual provides a documented framework for action, as well as a sound basis for measuring performance and demonstrating due diligence. It is intended to be a dynamic document that is reviewed and revised by site personnel and the Engineer of Record (EoR) on an annual basis and as operating conditions require. The OMS Manual includes a requirement for the annual dam safety inspection prepared by the EoR which includes a series of inspections at site that is documented in an annual Dam Safety Inspection Report. The first annual inspection for TSF 1 was conducted in Q4'19. The results of the inspection and data review indicated that the Çöpler TSF 1 is in good condition and operating in general accordance with the intended design of the facility. A review of the instrumentation indicated normal data trends and no unanticipated abnormal readings or 'triggering events' observed. Of the action items included in the report, none were considered serious in nature or otherwise a concern to the safety of the Çöpler TSF. The 2020 dam safety inspection is planned for Q4'20.

The TSF is inspected daily for signs of stress or damage. Daily and monthly operating data is collected on site and provided in a monthly report. The report estimates the settled solids volume in the TSF based on estimated bulk densities and provides for a comparison of actual tailings and water pool elevations compared to estimates made by Golder using data from the mine and tailings production plans and from the consolidation model that predicts settlement of the tailings. The difference between the actual tailings elevation and predicted elevations have shown close agreement generally less than 1 m.

In addition, members of the Anagold's HSSER team also inspect the TSF monthly. The TSF is subject to fortnightly external official audits by the Erzincan Provincial Environmental Directorate. The authorised hydraulic structures inspection company, Hidro Dizayn, is on site during construction at all times, on behalf of the Ministry of Environment and Urbanization of the Turkish Republic. The TSF design and engineering consultant is also on site during construction to ensure quality and conformance to design.

SSR Mining has established an Independent Tailings Review Board (ITRB), as per leading international best practices, to review tailings facilities as part of the review and oversight process. The ITRB reports directly to the senior management at a corporate level.

18.10.4 TSF Design

The TSF at Çöpler is a downstream, mass filled, dam. The technical specifications for the construction of the TSF conform with both Turkish national requirements and accepted good practice standards for tailings facilities, including; World Bank Standards, Canadian Dam Association Safety guidelines, and Mining Association of Canada (MAC) Guide to the Management of Tailings Facilities.

Both the TSF 1 and TSF 2 designs consist of fully lined impoundments, including a compacted earth and rockfill embankment. The TSF 1 and TSF 2 designs include the following primary components:

- A compacted earth and rockfill embankment with a zoned upstream granular filter protection system. Both facilities will have 1 m of freeboard under their crest elevations and are designed to contain the probable maximum precipitation (PMP) storm event. The downstream face of the ultimate embankments will be constructed at a composite slope of 1.7H:1V. The upstream face of the embankment will be constructed at a slightly shallower slope with slopes of 2.0H:1V to facilitate placement of the filter layers and liner system and a resultant composite slope on the order of 2.6H:1V after considering the operational benches. The filter layers and low-permeability soil layers are designed to be 1.5 m thick, as measured perpendicular to the slope. Measured horizontally, the layers are designed at 3.3 m wide each.
    - TSF 1 is a downstream raise construction which will consist of seven phases (six raises)
    - TSF 2, if constructed, is a downstream raise construction and is currently designed to be constructed in one phase.
- A composite liner system consisting of a 2 mm thick, double-sided, textured high-density polyethylene (HDPE) geomembrane and geosynthetic clay liner (GCL) over a low-permeability soil (i.e., clay) liner system that provides an equivalent protection to that provided by 5 m of a geologic barrier with k <10^{-9} m/s. A GCL is also substituted with low-permeability clay on select slopes steeper than 3H:1V as allowed by Turkish regulations.
- An impoundment gravity flow underdrain system for collection and monitoring of naturally occurring seeps and springs.
- An impoundment overdrain system for the collection and management of tailings seepage water through natural consolidation and drainage of excess process water.
- Perimeter roads and benches within and around the impoundment area for access and tailings distribution / reclaim water pipes.
- Tailings delivery and distribution system.
- Reclaim Systems.

18.10.5 Seismic Deformation Evaluation

The current deformation model provides the deformations under seismic loading conditions for a TSF 1 with 1,264 m crest elevation, which corresponds to phase 6 in the current design. Based on the average predicted deformations and the expected levels of liner strain, the TSF 1 phase 6 embankment is expected to remain stable when subjected to the design strong motion events. Simple deformation analysis by Bray and Travasarou (2007) was performed to assess the magnitude of earthquake induced movements on the phase 7 TSF 1 Embankment.

No deformation analysis was performed for TSF 2 considering it is a smaller dam and has a lower embankment height than TSF 1 and because of the similarities in design and foundation conditions. TSF 2 deformations are expected to be smaller than TSF 1 and in the acceptable deformation range as per the design criteria.

18.10.6 Tailings Consolidation and Capacity

Golder updated the tailings consolidation modelling to include the TSF 1 and TSF 2 joint operations and to account for the tailings characteristics obtained from 2020 laboratory tests on POX and Flotation tailings. The updated consolidation model also included the current mine plan. In the model, TSF 1 was first filled to elevation of 1,219 m (to the limits of phase 3 with a crest elevation of 1,220 m allowing for 1 m freeboard) and then tailings deposition was switched to TSF 2 and tailings in the TSF 1 was let to rest until TSF 2 is filled for a period of approximately 3.4 years. The rest period in TSF 1 increases the tailings density from 0.85 t/m^3 to 1.08 t/m^3 due to the natural consolidation and results with an average settlement on the order of 7 m which results in a capacity gain of 3.2 Mm^3 in TSF 1. The model results show that with the current mine plan and tailings characteristics TSF 1 and TSF 2 would have approximately 76 Mt and 14 Mt tailings capacity, respectively, over approximately 19.2 years of TSF 1 filling time.

The tailings tonnage estimate requires the sulfide plant feed to be adjusted to allow for the limestone added during processing for pH control. The limestone reacts with the acid to form gypsum. The applicable factor is 1.146. When the flotation plant commences operation in 2021 it will also directly contribute to the tailings placed.

Based on the updated consolidation analysis and assumptions on the mine plan, tailings characteristics, and operational plans as stated herein, approximately 90.6 Mt of tailings can be stored in TSF 1 and TSF 2 combined. The average dry tailings density expected at end of filling is 1.17 t/m^3 and 0.89 t/m^3 in TSF 1 and TSF 2, respectively.

18.10.7 TSF Schedule Assumptions

The key assumptions related to the ongoing construction and expansion of TSF 1 as follows:

- Phase 4:
  - There is a parcel of private land located east of Gully B that has not yet been purchased. if the private land cannot be purchased, contingent measures are in place to allow phase 4 to be constructed.
  - Construction of the new Sabirli Road is required for Phase 4 to be developed. Construction is planned to begin in Q4'20.
- Phases 5 to 7:
  - The design of the access roads and utility corridor for phases 5 to 7 considered construction of the haul road developed as part of the TSF 2 design which requires nominally 4 Mm^3 of rockfill. This design has been shown starting with phase 5. If TSF 2 were not developed, the access road and utility corridor could be further optimised depending on the extent of development. If only phase 5 were to be developed, the access could be provided by a much smaller ramp. If phase 6 and/or phase 7 are developed, then a route similar to that shown would be required.
- Schedule:
  - The current mine plan and schedule provides capacity within phase 3 through Q1'23 which generally requires that construction of phase 4, should start in 2021. As an alternative TSF 2 or another TSF could be constructed.
  - A partial development of phase 4 may provide additional time for consideration of other options however, construction of the new Sabirli Village road must be completed by Q1'23 in any event.

18.10.8 Further Work

There are opportunities that may offer significant reduction in capital costs with consideration of the following:

- Alternative TSF Considerations – The dam capacity to fill ratio for TSF 2 was approximately 1:25, which is significantly lower than TSF 1 due to the narrow and small valley where it is located. Several other options were identified in the CDMP20 Siting Study that would provide for reduced capital costs. Sites identified as TSF 4 and TSF 7 were determined to have dam capacity to fill ratios of 1:3.2 and 1:1.9, respectively based on conceptual designs only. Of the other sites considered in the CDMP20 Siting Study, TSF 4 was ranked second behind TSF 7 based on several environmental and social considerations namely due to its proximity and location with the Bağıştaş area, however, TSF 4, provides a significantly greater potential storage capacity with less fill required. TSF 7 would be highly visible to the Sabirli community but on the opposite side of Sabirli creek.
- Waste Rock Encapsulation – There are opportunities to consider encapsulation of potentially acid generating (PAG) waste rock within portions of the downstream embankment within the limestone. Field trials and studies are planned to be conducted in 2021 to evaluate this potential.

19 MARKET STUDIES AND CONTRACTS

19.1 Markets

The markets for gold and silver doré are readily accessed and available to gold producers. Currently, 100% of the gold and silver is delivered to the Istanbul Gold Refinery. Copper precipitate is currently produced from the SART plant and sold into local markets in Turkey. The sulfide plant does not currently include a copper circuit. Provisions have been made in the plant design to include the copper circuit in the future if market conditions warrant.

19.2 Contracts

Anagold contracts the mining operations to a Turkish mining contractor. The contract contains provisions for escalation / de-escalation of fuel prices, foreign exchange rates, haul grade and distance and Turkish inflation. The terms and prices for the mining contract are within industry standards for mining contracts.

20 ENVIRONMENTAL STUDIES, PERMITTING, SOCIAL AND COMMUNITY IMPACT

20.1 Environmental Studies and Material Impacts

The Çöpler mining and processing operations involve open pit mining from multiple pits, construction of multiple waste dumps to accommodate mined materials, processing of oxide ores and placement on a heap leach pad, and processing of sulfide ores with placement of tailings in a tails storage facility (TSF). These activities and facilities are carried out on treasury, pasture, and forestry lands, including some private lands.

In addition to the direct impacts on the involved lands, the operations impact on the surrounding lands and the local communities. Physical impacts may include changes to local surface and groundwater (including potential pollution), air quality impacts particularly from dust, and increased noise and vibration from mining and processing operations.

Operation of the Çöpler mining and processing facilities, and subsequent mining at Çakmaktepe, has been investigated and authorised by means of a series of EIAs, with positive decisions obtained from the Turkish Ministry of Environment and Urban Planning (MEUP). These EIA's include specific actions designed to address all material impacts of the mining and processing operations. Anagold has remained in compliance with all aspects of the EIA and operating permits throughout the history of the project.

The original 2008 EIA obtained on 16 April 2008 included three main open pits (manganese, marble contact, and main zones), five WRDs, a heap leach pad, a processing plant, and a TSF. The 2008 project description involved only the oxide resources.

The Çöpler project started its open pit and heap leach operation in 2010 and first gold was poured in December 2010. Additional EIA investigations have been submitted and approved, as required, to support on-going mining and processing operations, including:

- EIA to allow operation of a mobile crushing plant approved 10 April 2012.
- EIA to allow waste dump capacity expansion, oxide capacity expansion to 23,500 tpd and a SART plant approved 17 May 2012.
- EIA to allow the sulfide plant and heap leach area expansion approved 24 December 2014.
- EIA to allow the Çakmaktepe satellite pits expansion approved 26 January 2017.
- EIA to allow a Çakmaktepe capacity increase approved 9 August 2018.

In addition, pending EIA processes include:

- EIA to allow a second capacity expansion at Çöpler including heap leach pads 5 and 6, TSF expansion and operation of a flotation plant (process started December 2019, public hearing January 2020, in progress).
- EIA to allow Çakmaktepe second capacity increase to include initial mining from Ardich with EIA description file submittal expected in Q4'20.

Subsequent to the EIA positive decisions, additional permits and licences were required to be issued by government agencies consistent with the Turkish governing laws and regulations. These include land access permits (treasury, pasture and forestry); environmental permits and licences; workplace opening and operating permits; and licences and certificates. The status of project permits and operating licences is documented in Section 4 of this report.

In the period following the receipt of the 2008 EIA permit, Anagold has conducted further technical studies to supplement the Turkish EIA studies and to establish plans and procedures to manage potential project impacts and meet IFC requirements. Significant operational management plans established as a result of these prior and on-going studies include:

- Non-mining Wastes Management Plan
- Mining Waste Management Plan
- Water Resources Management Plan
- Biodiversity Management Plan
- Soil Management Plan
- Air Quality and Emissions Management Plan
- Mine Closure and Rehabilitation Plan
- Environmental Management System Framework
- Environmental Noise and Vibration Management Plan
- Hazardous Substances Management Plan
- Mine Closure Framework
- Resource Efficiency and Pollution Prevention Management Plan
- Cyanide Management Plan

20.2 Physical Features

The project site is located in a transition region between Central and Eastern Anatolian climates. The region has a continental climate, where summers are hot and dry, and winters are cold and relatively humid. Owing to the mountain ranges bordering Erzincan Province on all sides, the region has a milder climate than the neighbouring provinces.

The long-term annual average precipitation for the project site is 367 mm, including snow in the winter months. The annual average wind speed is 2.6 m/s. Maximum wind speeds are observed in spring. The prevailing wind direction is south.

The project site is located in a rural area with no significant commercial or industrial air pollution sources. Scattered slag piles and ore extraction sites remain from the former manganese mining operations.

The ambient air quality monitoring programme on site indicated that SO_2 and NO_2 levels, and particulate matter (PM10) and dust deposition levels in ambient air are well below the limit values defined in Turkish Air Quality Standards. Heavy metal concentrations in dust were well below the limit values defined by European Commission (EC), World Health Organisation (WHO) and Turkish standards.

The railway and the İliç-Kemaliye Road passing near the Euphrates River are the mobile sources of noise in the area. The Euphrates-Karasu River is the largest surface water body near the project; it borders the northern edge of the project area. Peak flow rates are observed in April and May following the snow melt and rainfalls. All other streams in the vicinity of the project area are intermittent, flowing between March–June.

The surface water quality within the site was investigated at various water sampling locations throughout the site. Water quality is classified from Class I (very good quality) to Class IV (highly polluted, poor quality water). Sampling has indicated Class IV water quality for Sabırlı and Çöpler Creeks, and Karabudak Stream. Similarly, the Euphrates-Karasu River is classified as a Class IV water resource. For all streams, metal concentrations, including aluminium, iron, copper and arsenic are high, especially in the drainage from Sabırlı and Çöpler creek catchments. Elevated metal concentrations in these catchments are attributed to natural metallic enrichment from the surrounding geology.

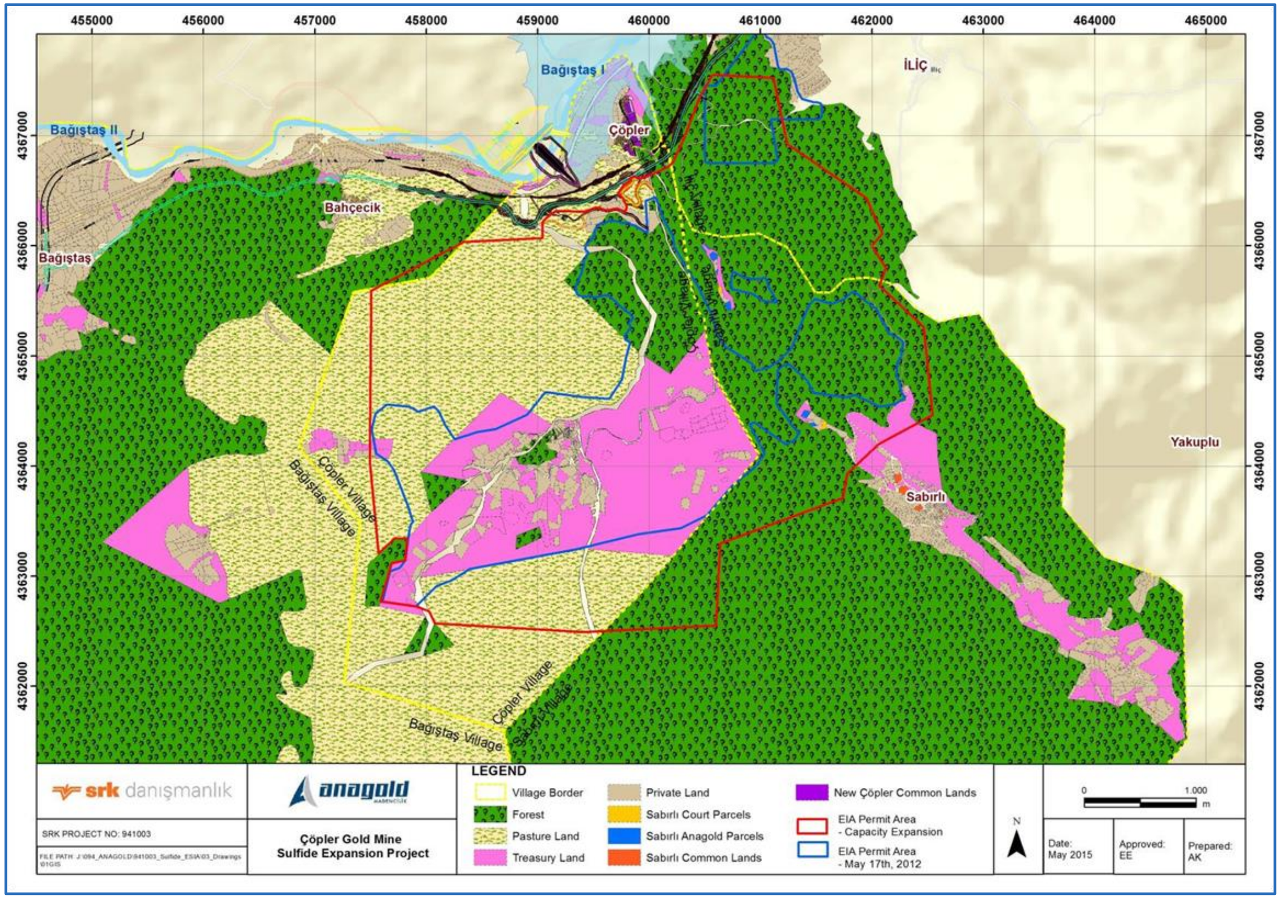
20.2.1 Land Use

The prevalent land use and cadastral information for the Project and its environs is presented in Figure 20.1. The land use patterns are based on maps produced by the General Directorate of Rural Services. As observed in Figure 20.1, most of the project area consists of pastureland, treasury, and forest. The Land Use Capability Classes (LUCC) for the project area and environs is given in Figure 20.2. Under the LUCC system, there are three main categories and eight classes (ranging between I and VIII). The first category covers Classes I through IV and describes lands which are suitable for cultivation and animal husbandry. This category has few limitations, except for Class IV, which requires very careful management because of its greater limitations. The second category covers Classes V through VII, which are unsuitable for cultivation, but which can support perennial plants when intensive conservation and development practices are applied. Under controlled conditions, this land may also support grazing and forestry. The soil type included in Class VII has severe limitations, preventing the growth of cultivated plants due to characteristics such as the formation of steep slopes (which are exposed to medium to severe erosion) and shallow soil layers, possessing stony, salty and sodic texture. As such their utilisation for agricultural purposes is very limited. The third category contains only the Class VIII, which is suitable only for wildlife, sports and tourism-related activities.

As shown in Figure 20.2, the project area has VI, VII and VIII types of LUCC. The land use types in the project area and its vicinity are:

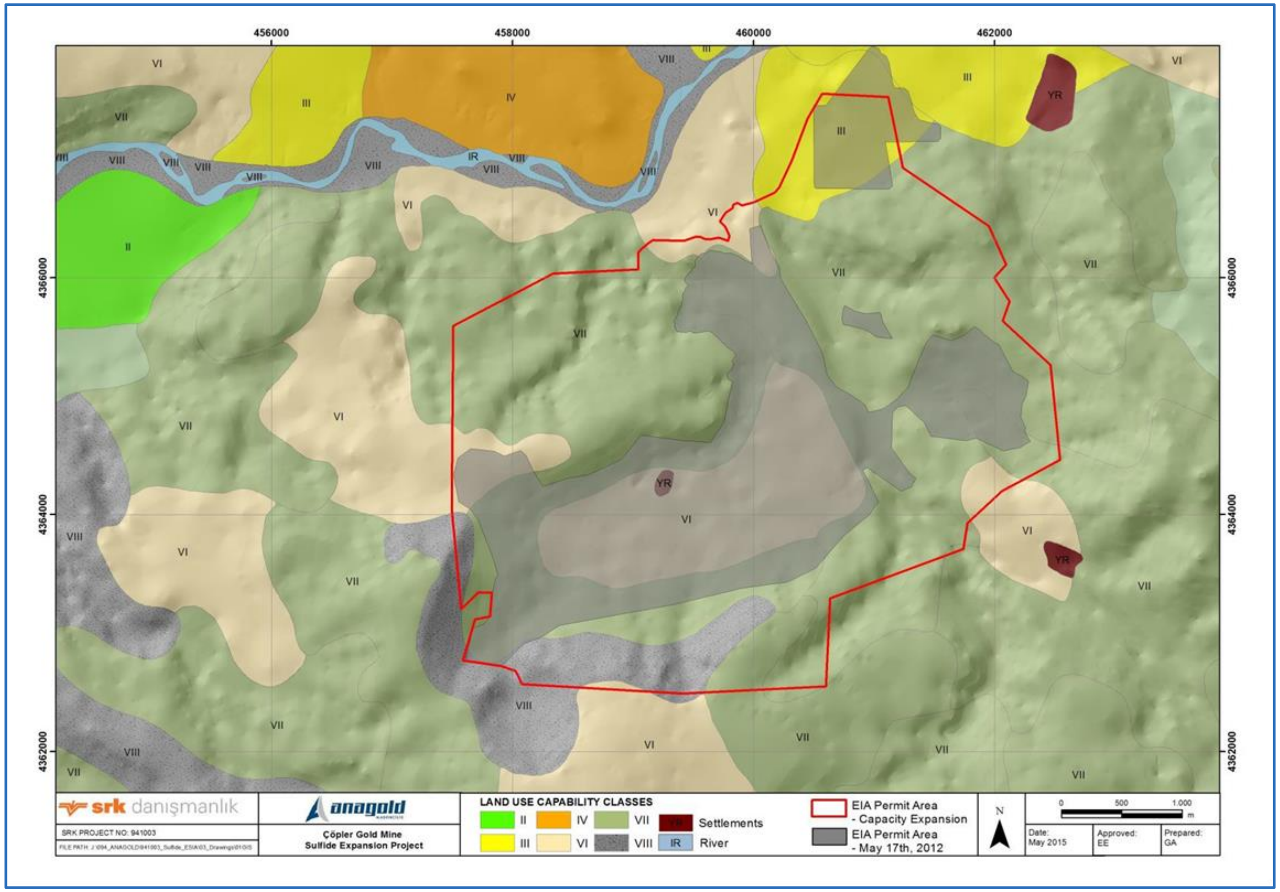
- Degraded forest lands and coppice
- Barren forest lands
- Agricultural lands
- Settlements

Figure 20.1 Current Land Use Types and Cadastral Map for the Çöpler Project



Anagold, 2016

Figure 20.2 Land Use Capability Classes (LUCC)



Anagold, 2016

The project area and surroundings are generally of low-land use capability and not suitable for agricultural activities. Although the agricultural activities are limited in the area, there are several small gardens which belong to the local villagers.

The forests in the area are under stress due to high grazing and illegal land use practices; pasture lands are used for the purpose of grazing, but it is illegal to use forestry lands for grazing. In general, the local soil has poor fertility due to its nature and elevation such that it only supports limited species of vegetation.

20.2.2 Biological Features

Floral species from the Irano-Turanian and Mediterranean phytogeographic regions are dominantly observed at the site. Most of the flora species are identified in the dry meadow habitats in the project area. Ruderal habitat (such as roadsides etc.) and rocky areas follow dry meadow habitats with respect to the floristic species diversity.

Flora and fauna surveys were conducted in the framework of the 2005–2007 EBS by specialists from Hacettepe University. Biodiversity of the site has been updated by the specialists from Gazi University and Hacettepe University via three seasonal surveys during 2011–2012. A Biodiversity Action Plan (BAP) was prepared and a BAP Report has been provided as an appendix of the ESIA Report for the Sulfide Expansion Project. The flora species were classified according to their thread status with respect to Turkish Red Data Book of Plants and the International Union for Conservation of Nature (IUCN) and European Red List (ERL) Categories and Criteria.

As a result of field surveys carried out within the Çöpler biodiversity study area a total of 328 Taxa were identified. Approximately 54 of these identified species are endemic and rare, and 21 out of 54 species are only known in the Province of Erzincan or other nearby provinces. There are four main vegetation types in the area namely: Quercus petraea subsp. pinnatiloba; Quercus libani and Quercus brantii forests; Irano-Anatolian steppe vegetation; and wooded steppes and rock habitat, while the rest of the site is designated for main mining activities. The faunal composition of the site is considered weak.

20.3 Social and Community Plans

The EIA studies are conducted according to the format stipulated by the Turkish EIA Regulation. The scope of the Turkish EIA studies differs from the scope of international Environmental and Social Impact Assessment (ESIA) studies (as established by the International Finance Corporation's (IFC)'s Environmental and Social Performance Standards), especially in terms of social impacts and public disclosure processes. While the social impact assessment and public disclosure processes are also parts of the Turkish EIA studies, they are treated less rigorously than in IFC standards.

SSR Mining has conducted further investigations to supplement the Turkish EIA studies, initially to support the original project establishment and, then subsequently, to monitor the social and community attitudes and the impacts of on-going mining operations on the adjacent communities. The fundamental data to assess social impact is derived from direct survey of the local community members in villages impacted by the mining operation. Significant (primary) surveys have included:

- Initial survey of 51 households in three villages (Sabirli, Bagistas and Dostal) presented collectively as part of the 2009 Çöpler Gold Project Social Impact Assessment (SIA) by KORA.
- Survey of 153 households in six villages (Çöpler, Bagistas, Bahcecik, Dostal, Yakuplu and Sabirli) presented individually performed by Middle East Technical University (January 2013).
- Survey of six villages performed by UDA Consulting (December 2014).
- SIA by SRK (2015).
- Survey by TANDANS Company (2017).
- Çöpler Mine phase 2 SIA Peer Review Report by Intersocial Company.

Anagold has considered the outcomes from the community surveys and SIA assessments as a key input to establish and monitor the social action plans associated with the project. These are also the basis to develop a strategic and planned approach to community investment and development programs. Some significant social and community plans and policies developed as a result of these investigations address the following:

- Community health and safety.
- Local employment
- Local procurement
- Community development fund (SKF)
- Donations
- Social investment and management funds
- Stakeholder engagement and community relations
- Environmental and social sustainability
- Health and safety
- Cultural Heritage
- Security Management
- Land access and resettlement

The performance and effectiveness of social and community plans are monitored, reviewed and updated, as required, to meet changing community needs and expectations.

20.4 Mine Closure

Mine rehabilitation and closure obligations are prepared and updated annually for the Çöpler project. Scheduling and costing of the closure tasks is made in accordance with the Anagold mine plan.

Cost estimates rely on data from mine operations including labour and equipment rates, material costs, groundwater well inventories, and electronic topography data.

Closure costs are estimated using the Standardised Reclamation Cost Estimator (SRCE). The SRCE is an industry standard tool developed to facilitate accuracy, completeness and consistency in the calculation of costs for mine site reclamation.

SRCE utilises lengths, areas, volumes, flow rates, quantities, etc., provided or estimated by the user (based on the reclamation or closure actions). Some actions require crews and fleets with productivities either provided by the SRCE default settings or those provided by Anagold to estimate the time it takes to perform the work. Where available, these times are then multiplied by labour and equipment rates provided by Anagold.

The Heap Leach Draindown Estimator (HLDE) model is another industry standard tool used for estimating heap leach pad draindown curves for reclamation bonding purposes. The HLDE inputs are derived from site-specific data.

20.4.1 Closure Cost Estimate Assumptions – Waste Rock Dumps

All slopes on the WRDs will be regraded to 2.5H:1V to prepare them for covering, scarification, and revegetation. The sequence of costs in the schedule corresponds to the assumption that reclamation will occur as soon as each WRD reaches final configuration.

Anagold plans to encapsulate all potentially acid-generating (PAG) waste rock within the WRDs as part of mining operations, leaving no PAG material on the surface or outer portions of the WRDs at closure. Therefore, although some PAG cells are currently exposed, costs for construction of a buffer layer encapsulating PAG waste rock are accounted under operational costs and no additional costs for mitigation of current configurations are included in the ARO estimates.

Per the EIA Report, waste rock management will be carried out to allow for the construction of a buffer layer to prevent degradation of seepage and these costs are accounted under operational costs. The seepage collection ponds active during the operations period will be reclaimed during closure. Seepage from the WRDs will not be monitored during closure and post-closure.

20.4.2 Closure Cost Estimate Assumptions – Pits

Berms will be constructed around the perimeter of the pit to discourage public access. There are no other physical reclamation measures assumed for the pit walls.

Rapid refilling of the pits with water is the preferred method for the western part of the pit. Costs for pit refilling by pumping flow of 66 litres per second (L/s) for four years are included in the ARO estimates.

Some PAG rock will remain exposed in the pit walls after formation of a pit lake; therefore, some reclamation work will be necessary to address the requirement (legal obligation) to cover remaining PAG materials exposed in the pit after mining ceases.

It is assumed that areas within the pit where PAG materials are exposed will be covered with 1 m of non-PAG (or non-acid generating – NAG) material. The PAG materials exposed within the pit walls are assumed to be located on gentle or nearly-flat slopes. Additional measures (e.g. reduction of pit wall slopes in exposed PAG areas to facilitate cover placement) are not taken into consideration at this time. No PAG cover will be required below the final pit lake elevation.

20.4.3 Heap Leach Pad

All slopes on the heap leach pads will be regraded to 2.5H:1V or flatter to establish a geotechnically stable closure configuration. Following regrading, the areas will be covered, scarified, and revegetated. The ARO estimates reflect the requirement per the EIA report that identifies 2–3 m of cover placement on the heap leach pad followed by growth medium placement after the reduction of heap and pond fluid inventory.

Although not a requirement in the EIA plan, there is a provision for extending half of the heap leach pad perimeter liner to contain heap material regraded beyond the existing liner during reclamation.

East and west buttresses are considered part of the heap leach pad area. The physical reclamation of this area by growth media placement and revegetation is included as a WRD.

The 2014 EIA discusses rinsing of the heap with fresh water with no subsequent fluid management. Rinsing of heap leach pads has been shown to be typically unnecessary and potentially detrimental to long-term chemical stability of gold heap leach.

Per the approach of the HLDE model mentioned above, heap drain-down will be initially managed for inventory reduction via recirculation and active evaporation, followed by active evaporation only. Active evaporation will continue until drain-down flows are reduced to a rate amenable to management with passive evaporation.

Following active solution management, when the heap drain-down flow rate decreases to a level where it can be managed exclusively within available emergency and process pond via passive evaporation, the two ponds will be converted to evapotranspiration (ET) cells. To convert process ponds to ET-cells, the ponds will require relining followed by backfilling with select material and revegetation.

Conversion costs are calculated based on experience from multiple Nevada sites.

In scheduling costs, the cost of construction of ET-cells is included at a time when drain-down rates reach a level that will allow fluid to be managed through the evapotranspirative capacity of ET-cells.

20.4.4 Tailings Storage Facility

Anagold submitted an EIA in 2014 that included TSF 1 and TSF 2. The current designs for TSF 1 and TSF 2 are within the 2014 EIA boundaries, except for a small portion of TSF 1 phase 7. TSF 1 phase 3 construction is in progress. The current mine plan only requires construction of TSF 1. Long-term management costs are included in the estimate and proportioned for the size of the TSF construction.

Reclamation of the LOM TSF includes the following actions:

- Reclamation of the TSF surface by placing a traffic layer and growth media followed by revegetation.
- Reclamation of the final TSF embankment.
- Fluid management including managing drainage from the TSF and removal of water ponding on the TSF surface due to consolidation of the tailings.

The estimate includes costs for placement of a traffic layer over the tailings material in addition to the growth media layer. The starter embankment is built at 1.5H:1V with the final embankment at 2.0H:1V. The costs of placing 1 m cover over the embankment are also included.

Costs are included for tailings fluid management crews, pumping for recirculation and forced evaporation as well as removal of the supernatant in the period soon after the TSF operations end.

20.4.5 Other

SRCE estimates costs to demolish buildings using productivities in conjunction with building volumes, wall areas, and slab volumes. Decontamination costs are included in the estimate for a decontamination crew to pressure-wash the plant site over a nominal number of weeks.

Production wells are assumed to be closed at the end of operation of the sulfide plant and monitoring wells are assumed to be abandoned at the end of the post-closure monitoring period.

20.4.6 Monitoring

The water quality and flow monitoring schedule during the operation, closure and post-closure monitoring period includes numbers of samples, frequencies, and durations for each closure phase. The monitoring locations include the groundwater monitoring wells around the heaps, WRDs, TSF and springs as well as pit lake water quality once the rapid filling begins.

20.4.7 Closure Planning

Closure planning costs are typical industry costs for development of closure plans and studies, reporting and preparation of closure designs and engineering.

20.4.8 Construction Management

Construction management costs include one supervisor during active reclamation. Costs are included for road maintenance, which will be carried out with a water truck and grader during active reclamation.

20.4.9 Human Resources

Closure personnel include a closure general manager, environmental manager, environmental technician, security, and surveyor for whom terminal benefits are included. Under the LOM schedule, the closure general manager would be present during the years of active reclamation and closure. Camp costs are included under general and administration costs.

For solution management, the cost of the heap drain-down management crew is assumed to be shared with those of the TSF.

20.4.10 Closure Schedule

The EOY 2020 closure is scheduled separately for the oxide and sulfide projects according to the mine plan and is consistent with the long-term management obligations expected for the TSF.

Heap drain-down management starts at the end of heap leaching operations in the mine plan. Ore will be sent to the leach pad until the end of 2030, although at a reduced rate after 2020. Management and reclamation on the heap will take place while other components of the Çöpler sulfide project continue to operate, with the active closure period starting after the end of deposition in the TSF.

20.4.11 Further Work

There may be an opportunity to utilise the heap drain-down solution in the sulfide circuit rather than disposing of it by forced evaporation, potentially reducing costs. This will require changes to the design of the evapotranspiration cells included in the current estimate.

Further studies and design work are required for the mitigation of PAG materials exposed in the pits to verify whether the proposed 1 m of non-PAG cover is practical and effective to implement.

The growth media inventory and expected amount to be recovered over the course of the project should be compared to the sum of the growth media requirements of the project facilities. Further work is required to determine the most sustainable revegetation covers to be employed.

20.5 Sustainability

SSR Mining aims to provide sustainability governance that not only meet or exceed the requirements of Turkish legislation, but also align with the expectations of ICMM (International Council of Mining & Metals) guidance and International Finance Corporation (IFC) Performance Standards, and the World Gold Council. The SSR Mining approach to policy development is to identify the most stringent standards and integrate them into project policy.

Çöpler project policies are supplemented by site-specific environmental and safety standards, management plans and procedures that are specifically tailored to the unique environmental and social challenges and permitting regulations of the site. These plans are certified to the requirements of international standards including ISO14001: 2015 and ISO45001.

SSR Mining maintains annual sustainability reporting for the project, the report is produced to be in accordance with GRI Standards. The last report was for 2019.

SSR Mining has a dedicated Environmental, Health, Safety and Sustainability (EHS&S) Committee. The EHS&S Committee oversees, monitors and reviews practice and performance in areas of safety, health, stakeholder relationships, environmental management and other sustainability issues.

Sustainability is also a key responsibility for group level executives and site teams. The approach to sustainability is underpinned by the principle of collective responsibility and a belief that every employee must contribute to our sustainability performance – particularly on issues of health and safety and reporting of incidents.

20.5.1 Stakeholder Engagement

At the Çöpler project, SS Mining has a wide-ranging stakeholder engagement program which sets out the ways in which SSR Mining engages with stakeholders and ensures regular communication with stakeholder groups.

During 2019 stakeholder consultations included meetings with shareholders, analysts, local communities, authorities, contractors, government representatives and trade union officials. Some of the key topics discussed included the Social Development Fund, exploration activities, cyanide awareness, local procurement and contracting opportunities and job creation.

The grievance mechanism is an important part of the SSR Mining local stakeholder engagement program and the overall governance of sustainability. The community grievance mechanism has been developed to meet the requirements of both Turkish regulations and the IFC Performance Standards. The mechanism is designed to be widely accessible and there are access points available throughout each of the five closest affected communities. There is also a dedicated access point for suppliers.

20.5.2 Health and Safety

Health and Safety Policy is guided by two key goals. First, to eliminate fatalities and serious injuries from our operations, and second, to continually reduce the number of minor injuries occurring on site. To fulfill these goals on the ground we implement:

- Robust systems and plans
- Risk assessment and controls
- Employee engagement
- Training

SSR Mining measure safety performance by tracking a range of leading and lagging safety indicators, the safety statistics reported also include exploration activities. All significant incidents are investigated and, based on findings, corrective action plans are developed to prevent recurrence.

20.5.3 Training and Development

The approach to the development of people is to strategically and continuously invest in staff training to ensure the business and operational needs both now and in the future are met. The development opportunities provided include technical skill development, leadership and business literacy skills, procedures and standards, and career development for staff. Çöpler has a specialized training centre with a capacity of 150 trainees.

SSR Mining carry out training and capability development programs for our neighbouring community. Training is directed to future roles with the project, while other training is focused on general skills development to enable people to seek gainful employment in other industries and locations throughout Turkey. This will help to broaden the economy and skills base in the Iliç District.

20.5.4 Industrial Relations

The workforce has no restrictions on union representation. Approximately 60% of the workforce at the Çöpler project are union members and have collective agreements in place. There have been no instances of industrial action.

20.5.5 Diversity and Inclusion

SSR Mining does not set diversity or gender quotas for the workforce. Personnel are appointed based on merit and have specific objectives in place to ensure that the candidate pools for any position available throughout the company are made up of a range of qualified and diverse candidates. Women are paid equal with men in similar positions. The SSR Mining Diversity Policy commits the project to provide:

- An environment in which all employees are treated with fairness and respect; and
- Equal access to opportunities - regardless of race, gender, sexual orientation and/or religious beliefs.

The approach to recruitment is to first look to local communities with appropriate skills. If unsuccessful, this is followed by recruiting from the wider region, followed by nationally, before finally looking internationally. The SSR Mining commitment to employing and developing local and national workers is reflected by the targets set for the Çöpler project:

- 90% of unskilled workers to be drawn from communities impacted and affected by SSR Mining operations
- 80% of semi-skilled worker to be drawn from impacted and affected communities
- 80% of skilled workers to be Turkish citizens

Suppliers are also encouraged to employ local workers whenever possible.

Local supply chains are preferred. Where supplier skills are lacking SSR Mining work with the suppliers to build capacity by providing training and mentoring.

20.5.6 Sustainable Community Development

The SSR Mining commitment to contribute to the development of local communities is set out in the Community Relations Policy, which has two clear goals: 1) to maximize the number of beneficiaries from the Mine affected settlements and 2) to foster long-term economic growth that is not dependent on the Mine.

Each year SSR Mining contribute to the development of local communities by making direct investments in community infrastructure and social programs. This spend is dedicated to four identified priorities:

- Improving access to education and academic opportunities for local communities;
- Creating long-term sustainable economic development for local communities;
- Improving local infrastructure; and
- Enhancing women's economic participation.

Alongside the direct investments SSR Mining also invest in the local community at the Çöpler project through our Social Development Fund (SDF). Launched in 2018, the SDF is an innovative partnership between the Mine and the community.

It aims to provide financial support to local entrepreneurs so they can set up or grow their own businesses that are not tied to the Mine as well as investing in a wide-range of social and community development projects. Projects are selected based on a set of development priorities agreed in consultation with the community and aligned with local government development plans and priorities. The priorities are reviewed and updated on a three-year basis.

The SDF is funded from contributions by SSR Mining of $2 for every ounce of gold produced by the Çöpler project, which links the benefits with the community to operational success. This investment is ringfenced solely to fund community projects. No money from the SDF is used for the management, monitoring and evaluation of projects. Social and economic benefits are delivered to the local community and country by way of creating jobs, procuring goods and services, making investment in community programs and infrastructure, and payment of taxes and royalties to local and national government.

20.5.7 Environmental Management

SSR Mining's commitment to responsible environmental management is set out in the Environmental Policy, which complies with in-country legislation, the IFC Performance Standards, and the Equator Principles. The Çöpler Environmental Management System (EMS) is certified to the international ISO14001: 2015 standard. The latest ISO14001: 2015 external audit was completed successfully in December 2019.

20.5.8 Water Risk

The Çöpler project is in a high desert region in Eastern Turkey near the culturally significant Euphrates River. All water used at Çöpler is governed by strict permitting rules regarding abstraction and discharge under Turkish regulations. The approach to water management is to use water as efficiently as possible and to only draw as much needed and allowed within permitted limits. All water abstract is groundwater. Water used on site is recycled and reused in the process plant. Water is not discharged to the environment.

20.5.9 Energy and Climate Change

All the electricity the Çöpler project uses is drawn from the Turkish national grid. Approximately 41% of Turkey's national grid capacity comes from hydropower stations. The treatment of sulfide ore requires a more energy and CO_2 intensive process than the oxide ore process that was previously the only ore treated at the Çöpler project. SSR Mining plan to use 2019, 269GWh, as the baseline year for electricity use and efficiency, and to set targets based on 2019. The Green House Gas emissions are published in the SSR Mining sustainability report.

20.5.10 Tailings Dam Management

Tailings produced by the Çöpler project are classified as Class II non-hazardous. All tailings are sent to a carefully engineered TSF. SSR Mining has procedures in place to ensure that all parts of the TSF life cycle from construction to closure align with international best practice standards.

The TSF at the Çöpler project is a downstream mass filled dam. It became fully operational during the final quarter of 2018 with the start-up of the sulfide plant. The technical specifications for the construction of the Çöpler project TSF conforms with both Turkish national requirements and accepted good practice standards for tailings facilities, including:

- World Bank Standards
- Canadian Dam Association Safety guidelines
- ICOLD (International Commission on Large Dams) Bulletins
- Turkish Hydraulic Works' Technical Codes
- Mining Association of Canada (MAC) Guide to the Management of Tailings Facilities.

The Çöpler project TSF has been designed to withstand significant earthquakes up to a magnitude of 7.5 on the Richter scale. Modelling showed that even in the most severe seismic event, the wall of the TSF will heave with minimal risk of altering facility location or strength. There are no communities living directly downstream of the Çöpler project TSF.

The TSF uses a combination of technology, regular inspections and external oversight and audits to monitor the Çöpler project TSF (see Section 18.10.3).

In addition to stability designs and monitoring, SSR Mining also have three groundwater monitoring wells in place both above and below the Çöpler project TSF, to monitor for signs of groundwater contamination. It was designed to meet the best in class requirements for Class-I (hazardous) waste, even though all tailings are classified Class-II (non-hazardous).

20.5.11 Water Management

The process of removing ore from the ground and extracting gold creates significant non-hazardous and some hazardous waste, which must be appropriately dealt with over the long- and the short-term. Ensuring all waste is responsibly dealt with is crucial to protecting the health of the local environment and neighbouring communities.

To ensure that all waste, whether hazardous or non-hazardous, is reduced and dealt with in a safe and responsible manner, the Çöpler project has a detailed and comprehensive waste management plan. This is underpinned by the goal to reduce the amount of waste generated and to maximize the proportion of waste sent for recycling.

The bulk of the waste created at the Çöpler project is waste rock. All the waste rock created by the Çöpler project is carefully disposed of in engineered waste rock dumps. The design and management of all waste rock dumps is overseen by geotechnical engineers to ensure they have safe slope angles, maximum structural stability and management of any potentially acid forming materials are conducted appropriately by mine operations and thus meet the requirements of Turkish national regulations, industrial best practices and the IFC Performance Standards.

20.5.12 Cyanide Management

The use of cyanide is a critical part of the gold mining process. However, if not handled correctly, cyanide can have significant impacts on both environmental and human health. The use of cyanide at the Çöpler project is governed both by the requirements of Turkish national laws and regulations and aligned with industrial best practice. All employees and contractors who handle, transport or dispose of cyanide are required to undertake specialized training in cyanide handling.

20.5.13 Biodiversity

The size, scale and location of mining operations means they can have a negative impact on local biodiversity. Failure to manage these risks and minimize the impacts on biodiversity could affect the social license to operate and reputation. The SSR Mining aim is to restore sites (both operational and exploratory) and repair any damage done to the extent practicable. To do this, detailed records of the full range of biodiversity present as part of feasibility studies of any project or expansion. These studies form the basis for a Biodiversity Action Plan (BAP). The BAP sets out how impacted ecosystems are to be restored to their original state (or as close as possible) at the time of closure. Both the Çöpler project, its associated TSF and prospects have Biodiversity Action Plans in place. SSR Mining also conduct biodiversity monitoring studies each quarter with experts from Gazi and Hacettepe Universities.

20.5.14 Air Quality

There is a potential for dust to be generated across many parts of the operation, including blasting, crushing and milling, and the movement of large vehicles on haul roads. Dust management is a key focus across all facets of the operation. Air quality and the presence of dust is an important factor for local communities and workers. Ensuring management air quality for workers and communities is an important part of environmental management. SSR Mining has put in place a dust management plan at the Çöpler project to minimize the levels of dust in the air and ensure they fall within Turkish and IFC guideline limits. There are several monitoring stations across site and in the local communities. These stations record levels of airborne particulate matter and dust fall out. The results from the monitoring stations are reported to the relevant national authorities, and to local communities.

21 CAPITAL AND OPERATING COSTS

Capital and operating cost estimates have been developed based on the current project costs, the mine and process designs, and discussions with potential suppliers and contractors. The estimated capital costs are to a feasibility level of accuracy and include a contingency of 10%.

21.1 Capital Costs

Growth capital costs in the Reserve Case includes costs for:

- Flotation circuit
- Heap leach phase 4B
- Road relocation, studies, and project management

Sustaining capital in the Reserve Case includes costs for:

- TSF
- Project team
- Technical services
- Administration
- Assay laboratory
- Mining
- IT
- Sulfide processing
- Oxide processing
- Environment
- Mineral / lands rights
- Health & safety
- Security
- Supply chain
- Reclamation

Capital costs assumptions to the end of 2021 and for the LOM are shown Table 21.1.

Table 21.1 Capital Costs

Description	Q4'20 and 2021 ($M)	Total LOM ($M)
Oxide		
Growth	29	29
Sustaining	4	9
Sulfide		
Growth	29	29
Sustaining	59	421
Site		
Reclamation	2	103
Working and Other	8	14
Total	131	605

21.2 Operating Costs

Operating costs were estimated based on current site cost performance and contract costs including actual operational costs for labour, consumables, contracts and the Anagold budget assumptions. The projected LOM unit operating cost estimate is summarised in Table 21.2 and the average costs are shown in Table 21.3.

Table 21.2 Average Operating Costs Unit Rates

Activity	Unit	Life-of-Mine Average Unit Cost
Mining	US$/t Mined	1.55
Rehandle	US$/t Rehandle	1.05
Processing – Heap Leach	US$/t HL processed	17.08
Processing – Sulfide	US$/t Sulfide processed	34.08
Site Support and Office	US$/t Ore processed	6.82

Table 21.3 Summary of Life-of-Mine Average Operating Costs

Cost	Total LOM ($M)	5-Year Average per year ($/t)	LOM Average per year ($/t)
Mining	371	9.59	6.32
Rehandle	62	0.67	1.05
Process	1,872	28.28	31.86
Site Support	462	10.11	7.86
Operating Costs	2,767	48.64	47.09

21.3 Mining Cost Summary

The mining costs were applied to the financial model as operating costs or capital costs. In the mining cost model, costs are broken down into specific areas including drill and blast, load and haul and rehabilitation.

Mining operations for the mine are currently contracted to a Turkish mining contractor. No capital cost is included for mining equipment or facilities. All such costs are built into the unit rate for mining operations included in the operating cost estimate.

Mining operating costs include:

- Drill and blast
- Load and haul
- Labour
- Dewatering
- Other indirects

Mining capital costs include:

- Fixed equipment
- Mobile equipment
- Office and supply
- Mine rehabilitation
- Studies

21.4 Processing and Infrastructure Cost Summary

The following has been included in the costs for processing:

- Oxide processing
- Sulfide processing
- Waste management
- TSF
- Utilities and services
- Reagents
- Plant infrastructure
- Plant mobile equipment

The following has been included in the capital costs for infrastructure cost estimates:

- Bulk services
- Site preparation
- Buildings and structures (new and refurbished)
- Communications
- IT hardware and software
- Security and access control
- Site costs
- Mobile equipment
- Services contracts
- Community support

The following has been included in the operating cost estimates:

- Plant consumables
- Crusher Consumables
- Screens
- Grinding media
- Filters
- Packaging plant bags
- Plant reagents
- Plant mobile equipment
- Plant maintenance
- Power

- Labour
- Production and dispatch
- Plant and infrastructure day work services
- Plant technical services
- Shift maintenance
- Laboratory service level agreement
- TSF water treatment

21.5 General and Administration Cost Summary

The General and Administrative (G&A) costs include costs not directly attributable to operational output such as the mining and processing operations. The following costs have been included in total G&A cost:

- Office and general expenses
- Site support costs
- Off-site Anagold offices
- Internal and external consultants
- Maintenance and inspection contracts
- Equipment and sundry
- Fuels and utilities
- Rentals and leases
- Insurance and insurance taxes
- IT hardware and software
- Personnel transport
- Communications
- Licences and land fees
- Labour
- Accommodation and messing
- Medical support
- Flights
- Light vehicles
- Environmental, community development and engagement
- Banking and audit fees
- Legal

22 ECONOMIC ANALYSIS

22.1 Economic Assumptions

The financial model was prepared using the Reserve Case production schedule, operating and capital assumptions on an annual basis. The assumptions for taxes and royalties were provided by SSR Mining.

22.1.1 Metal Prices

Metal prices were estimated after analysis of consensus industry metal price forecasts and metal prices used in other studies. The prices used for the economic analysis are shown in Table 22.1.

Table 22.1 CDMP20 Reserve Case Metal Price Assumptions

Metal	Units	Average	2020	2021	2022	2023	2024	2025	Long-Term
Gold Price	$/oz	1,658	1,850	1,965	1,835	1,745	1,645	1,585	1,585
Silver Price	$/oz	21.55	20.05	24.15	22.70	21.80	20.75	20.25	20.25
Copper Price	$/lb	2.95	2.70	2.90	2.90	2.95	3.00	3.05	3.05

22.1.2 Taxation

The Turkish government implemented a temporary rate increase from 20% to 22% for the periods of 2018-2020. From 2021 onwards, the effective tax rate is expected to return to 20%.

The CDMP20 economic analysis applies a corporate tax rate of 22% for Q4'20 and then the reduced 20% for 2021 onwards.

For tax purposes, a 20% accelerated depreciation rate is applicable for both the oxide and sulfide capital. The depreciation period is 10 years for general mining equipment, if not specifically defined by the tax office.

Investment incentive certificates are available for investments that promote economic development. Investment incentive certificates can be classified as strategic in specific circumstances and such certificates provide additional incentives. Anagold received a strategic incentive certificate for the sulfide process plant. An investment incentive certificate generates credits that offset corporate income taxes generated by the investment. The amount of investment credits generated from the investment incentive certificate is based on eligible capital expenditures. The investment credits generated by the strategic investment incentive certificate reduce the corporate tax rate to a minimum of 2% in a given tax period until the last quarter of 2023, thereafter it is assumed subsequent non-strategic investment incentive certificates will be available and the minimum rate will be 4%. Incentive tax credits can be carried forward to future tax periods indefinitely until exhaustion. Incentive tax credits and other tax pools are determined in the local currency, Turkish Lira, and subject to devaluation and revaluation as fluctuations against the US dollar occur. The cash flow model is prepared on a constant Turkish Lira basis.

VAT in Turkey is levied at 18% and the project is eligible for the Turkish exemptions for mining projects and mining equipment purchases. In the CDMP20 assumes the cash flows are not subject to VAT.

Import duties are not included in the capital cost estimate for mining related imported equipment because they are exempted in the incentive certificates.

22.1.3 Royalties

Under Turkish Mining Law, the royalty rate for precious metals is variable and tied to metal prices. The Çöpler project is subject to a mineral production royalty which is based on a sliding scale to gold price and is payable to the Turkish government. In September 2020 a presidential decree was issued, increasing the prescribed royalty rates by 25%.

Table 22.2 details the relevant prescribed royalty rates along with the revised rates following the September 2020 presidential decree. The royalties are calculated on total revenue with deductions allowed for processing and haulage costs of ore. As the Çöpler project produces by-products Silver and Copper as part of the process of treating gold ore, revenue from by-products is included in the total revenue used for royalty calculations.

The royalty rates outlined in Table 22.2 apply to sellers of raw ore. Royalty rates are reduced by 40% for ore processed in country, as an incentive to process ore locally. As the Çöpler project produces its gold doré on site, the Çöpler project is eligible for a 40% reduction to the royalty rate.

Table 22.2 Gold Royalty Rates

Metal Price ($/oz Gold)		Prescribed Royalty Rate (%)	Revised Royalty Rate (%)
From	To		
0	800	1.00	1.25
800	900	2.00	2.50
900	1,000	3.00	3.75
1,000	1,100	4.00	5.00
1,100	1,200	5.00	6.25
1,200	1,300	6.00	7.50
1,300	1,400	7.00	8.75
1,400	1,500	8.00	10.00
1,500	1,600	9.00	11.25
1,600	1,700	10.00	12.50
1,700	1,800	11.00	13.75
1,800	1,900	12.00	15.00
1,900	2,000	13.00	16.25
2,000	2,100	14.00	17.50
2,100	+	15.00	18.75

The Çöpler project effective LOM royalty rate based on the financial model metal price assumptions and applicable deductions is approximately 4.2%.

Other than the royalty payments, there are no other known back-in rights, payments, or other agreements and encumbrances to which the property is subject.

22.2 Reserve Case Economic Analysis Results

The Reserve Case production includes 7.7 Mt at 1.22 g/t Au oxide ore processed by heap leaching and 51.1 Mt at 2.24 g/t Au processed in the sulfide plant. Total gold production is 3.6 Moz. All mining is completed by 2032, oxide heap leach stacking is completed in 2031, and sulfide processing will continue from stockpiles until 2041. The Reserve Case shows an after-tax NPV at a 5% discount rate of $1.73 billion. The operation is cash positive in each year of the mine plan, therefore an IRR is not reported. The Reserve Case average all-in sustaining cost (AISC) is $945/oz gold.

The key production and economic analysis from the CDMP20 are shown in Table 22.3. The estimates of cash flows have been prepared on a real basis with a base date of Q4'20 and a mid-year discounting is used to calculate NPV. The economic analysis uses long-term metal price assumptions of $1,585/oz gold, $20.25/oz silver, and $3.05/lb copper. These prices are based on a review of consensus price forecasts from financial institutions and similar recently published studies.

All monetary figures have a base date of Q4'20 with no allowance for escalation and are expressed in US dollars (US$) unless otherwise stated. Production costs and AISCs are determined on a per ounce gold produced basis and do not consider the application of inventory movements or deferred waste stripping. Production costs do not equate to cash costs prepared under SSR Mining non-GAAP measures. AISCs do not equate to AISCs prepared under SSR Mining non-GAAP measures.

The start date for the Reserve Case economic analysis is 1 October 2020. The key results of the economic analysis are shown in Table 22.3. The NPV results for before and after-tax over a range of discount rates is shown in Table 22.4. Gold unit costs net of by products are shown in Table 22.5. Figure 22.2 shows the Reserve Case LOM cash flows.

Table 22.3 CDMP20 Reserve Case Results Summary

Item	Unit	Reserve Case
Oxide Processed		
Heap Leach Quantity	kt	7,668
Au Feed Grade	g/t	1.22
Sulfide Processed		
Quantity Milled	kt	51,084
Au Feed Grade	g/t	2.24
Total Gold Produced		
Oxide – Gold	koz	256
Sulfide – Gold	koz	3,334
Total – Gold	koz	3,591
Oxide – Gold Recovery	%	73%
Sulfide – Gold Recovery	%	91%
5 Year Average		
Average Gold Produced	kozpa	266
Free Cash Flow	$Mpa	224
Production Cost	$/oz gold	682
All-in Sustaining Costs	$/oz gold	865
Key Financial Results		
Production Cost	$/oz gold	748
All-in Sustaining Costs	$/oz gold	945
Site Operating Costs	$/t treated	47.09
After-Tax $NPV_{5\%}$	$M	1,733
Mine Life	years	21

5-Year annual average is for the period 1 January 2021 through 31 December 2025

Table 22.4 CDMP20 Reserve Case Before and After-Tax NPV

Discount Rate	Before-Tax NPV ($M)	After-Tax NPV ($M)
Undiscounted	2,397	2,306
5%	1,791	1,733
10%	1,434	1,393
12%	1,332	1,295

Table 22.5 CDMP20 Reserve Case Cash Costs

Costs per Ounce (Cash Basis)	Units	Reserve Case
Mining and Rehandle	$M	420
Process, Freight, and Refining	$M	1,633
Site Support	$M	400
Royalties	$M	232
Total Production Costs	$M	2,686
Production Cost	$/oz gold	748
Sustaining Capital	$M	430
Fixed Lease Payments	$M	201
Exploration – Sustaining	$M	14
Site G&A	$M	61
All-in Sustaining Costs	$M	3,392
All-in Sustaining Cost	$/oz gold	945

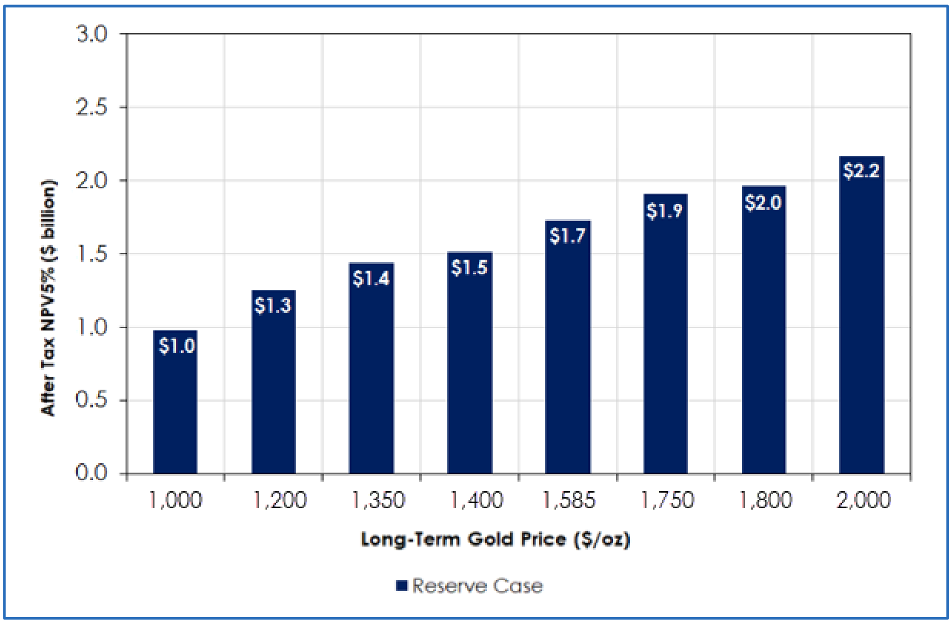
Process, Freight, and Refining includes by-product credits and excludes fixed lease costs
Royalties are calculated in the period incurred

The after-tax NPV sensitivity to metal price variation is shown in Table 22.6 and Figure 22.1 for gold prices from $1,000–$2,000/oz. Cost sensitivity is shown in Table 22.7.

Table 22.6 CDMP20 Reserve Case Gold Price Sensitivity

After-Tax NPV ($M)	Long-Term Gold Price ($/oz)							
Discount Rate	**1,000**	**1,200**	**1,350**	**1,400**	**1,585**	**1,750**	**1,800**	**2,000**
Undiscounted	1,066	1,514	1,833	1,943	2,306	2,578	2,667	2,989
5%	981	1,251	1,443	1,510	1,733	1,906	1,962	2,162
10%	906	1,080	1,204	1,247	1,393	1,509	1,546	1,680
12%	879	1,027	1,133	1,170	1,295	1,395	1,427	1,542
15%	842	961	1,045	1,074	1,175	1,256	1,282	1,376
18%	809	905	973	997	1,079	1,146	1,167	1,244
20%	788	872	932	953	1,025	1,084	1,103	1,171

Figure 22.1 CDMP20 Reserve Case Gold Price Sensitivity



OreWin, 2020

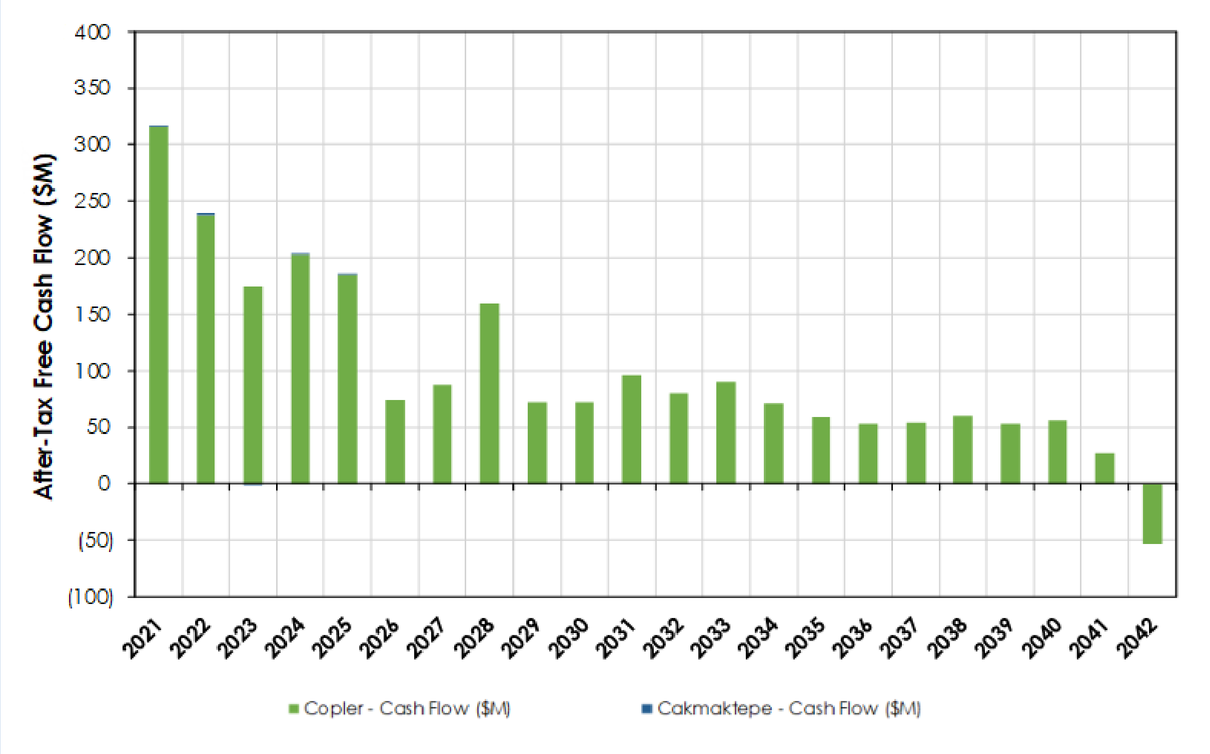
Table 22.7 CDMP20 Reserve Case Cost Sensitivity

			Change from Base $NPV_{5\%}$ ($M)				
Variable	**Units**	**Base Value**	**–20%**	**–10%**	**0%**	**10%**	**20%**
Capital Cost	$M	591	1,818	1,776	1,733	1,691	1,648
Mining Cost	$/t mined	1.55	1,782	1,758	1,733	1,709	1,684
Processing Cost	$/t treated	31.86	1,806	1,770	1,733	1,696	1,659
Site Operating Cost	$/oz	754	2,046	1,891	1,733	1,563	1,385

22.2.1 Project Cash Flow

The after-tax cash flow and average LOM AISC unit cost is shown in Figure 22.2. The revenue, operating cost and capital costs and net cash flow is tabulated in Table 22.8.

Figure 22.2 Reserve Case After-Tax Cash Flow



OreWin, 2020
Çakmaktepe cash flow occurs 2022–2024

Table 22.8 CDMP20 Reserve Case Cash Flow

Cash Flow Statement ($M)	TOTAL	Year																					
		Q4'20	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042
Heap Leach – Gold Revenue	450	41	120	71	40	31	17	23	14	51	7	10	5	10	10	–	–	–	–	–	–	–	–	–
Sulfide Plant – Gold Revenue	5,505	119	501	413	380	407	365	277	285	304	236	235	262	221	215	199	184	163	163	163	163	163	87	–
By-Product Revenue	51	5	8	11	6	4	2	4	2	6	1	1	1	0	0	0	0	0	0	0	0	0	0	–
Net Revenue	6,006	165	629	495	426	442	384	304	301	360	244	246	268	232	225	199	184	163	163	163	163	163	87	–
Realisation Costs																								
Freight and Refining	14	0	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0	0	0	–
Royalty Payments	252	–	29	41	26	19	19	14	9	10	13	7	7	9	7	7	6	5	4	4	4	4	5	2
Total – Realisation Costs	266	0	31	42	27	20	20	15	9	10	13	8	8	10	8	8	6	6	4	5	5	5	5	2
Operating Costs																								
Mining	358	8	30	39	35	33	32	33	34	34	27	18	15	20	–	–	–	–	–	–	–	–	–	–
Ore Rehandle	62	1	4	3	2	1	1	2	3	2	4	3	3	3	4	4	4	3	4	3	3	3	1	–
Processing – Heap Leach	131	11	19	20	17	12	7	12	7	16	2	6	2	–	–	–	–	–	–	–	–	–	–	–
Processing – Sulfide Plant	1,741	22	88	92	89	88	88	90	83	89	86	83	85	84	84	82	82	78	77	77	77	77	40	–
Site Support	400	6	42	30	30	30	27	27	27	27	23	19	19	13	13	9	9	9	9	9	9	9	9	–
Total – Site Operating Costs	2,692	48	182	184	173	164	156	165	153	167	141	130	125	120	100	94	94	90	89	89	89	89	50	–
Exploration	14	4	3	1	1	1	1	1	1	1	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Corporate Costs	61	1	5	5	5	5	5	5	5	5	4	4	4	4	0	0	0	0	0	0	0	0	0	–
Total – Operating Costs	2,767	54	190	190	180	171	161	171	159	173	145	133	128	123	101	95	94	90	90	90	89	89	50	–
Operating Surplus / (Deficit)	2,974	111	409	263	219	251	203	119	132	177	86	105	132	99	117	97	84	68	69	69	69	69	32	-2
Capital Costs																								
Growth	58	18	40	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Sustaining	430	17	46	24	40	40	12	42	42	12	12	29	29	12	12	12	12	12	12	6	6	–	–	–
Reclamation	103	–	2	0	3	2	0	0	–	–	–	0	2	4	11	11	10	1	1	1	1	1	1	51
Working and Other	–14	–4	–4	–2	–1	–1	–1	–0	–0	–0	–0	–	–	–	–	–	–	–	–	–	–	–	–	–
Total – Capital Costs	577	31	84	22	42	41	12	42	42	12	12	30	32	16	23	23	22	13	13	7	7	1	1	51
Net Cash Flow Before Tax	2,397	79	325	241	178	210	190	77	91	165	74	75	100	83	93	74	62	55	56	62	62	68	31	–53
Tax	91	2	7	4	3	7	6	3	3	5	2	3	4	3	3	3	2	2	2	2	9	12	5	–
Net Cash Flow After Tax	2,306	77	318	238	175	203	185	74	87	160	72	72	96	80	90	71	59	53	54	60	53	57	26	–53

Royalties are paid in the period after they are accrued

23 ADJACENT PROPERTIES

There are no adjacent properties that are applicable to the CDMP20.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 PEA Case Economic Analysis Results

A financial model was prepared using the PEA Case production schedule, operating and capital assumptions on an annual basis. The assumptions for taxes and royalties were provided by SSR Mining and are the same as the Reserve Case except that it is assumed that any capital expenditure incurred to mine and process the Ardich deposit will not qualify for the investment incentive credit regime.

Metal prices were estimated after analysis of consensus industry metal price forecasts and metal prices used in other studies. The prices used for the economic analysis are shown in Table 24.1. The PEA Case uses the royalty assumptions from the Reserve Case. The average PEA Case effective royalty rate is approximately 4.7%. Other than the royalty payments, there are no other known back-in rights, payments, or other agreements and encumbrances to which the property is subject.

Table 24.1 PEA Case Metal Price Assumptions

Metal	Units	Average	2020	2021	2022	2023	2024	2025	Long-Term
Gold Price	$/oz	1,644	1,850	1,965	1,835	1,745	1,645	1,585	1,585
Silver Price	$/oz	21.55	20.05	24.15	22.70	21.80	20.75	20.25	20.25
Copper Price	$/lb	2.95	2.70	2.90	2.90	2.95	3.00	3.05	3.05

The key results of the economic analysis are shown in Table 24.2. The start date for the PEA Case economic analysis is 1 October 2020.

The PEA Case production is 79.1 Mt at 2.13 g/t Au. The gold production in the PEA Case is 4.6 Moz. The increase in total production in the PEA Case is due to the addition of 20.3 Mt at 2.18 g/t Au from Ardich Mineral Resources. Like the Reserve Case, all mining is completed by 2032 in the PEA Case, oxide heap leach stacking is completed in 2031, while sulfide processing continues from stockpiles until 2042. The PEA Case shows an after-tax NPV at a 5% discount rate of $2.16 billion and the average AISC is $893/oz gold. The PEA Case is cash positive in each year of the mine plan.

The estimates of cash flows have been prepared on a real basis with a base date of Q4’20 and a mid-year discounting is used to calculate NPV. The economic analysis uses long-term metal price assumptions of $1,585/oz gold, $20.25/oz silver, and $3.05/lb copper. These prices are based on a review of consensus price forecasts from financial institutions and similar recently published studies.

The NPV results for before and after-tax for a range of discount rates is shown in Table 24.3. Gold costs net of by products are shown in Table 24.4. The after-tax NPV sensitivity to metal price variation is shown in Table 24.5 and Figure 24.1 for gold prices from $1,000–$2,000/oz. Cost sensitivity is shown in Table 24.6. Table 24.7 shows the PEA Case LOM cash flows.

Table 24.2 PEA Case Results Summary

Item	Unit	PEA Case
Oxide Processed		
Heap Leach Quantity	kt	25,008
Au Feed Grade	g/t	1.69
Sulfide Processed		
Quantity Milled	kt	54,073
Au Feed Grade	g/t	2.33
Total Gold Produced		
Oxide – Gold	koz	956
Sulfide – Gold	koz	3,691
Total – Gold	koz	4,646
Oxide – Gold Recovery	%	68%
Sulfide – Gold Recovery	%	91%
5 Year Average		
Average Gold Produced	kozpa	306
Free Cash Flow	$Mpa	249
Production Costs	$/oz gold	701
All-in Sustaining Costs	$/oz gold	886
Key Financial Results		
Production Costs	$/oz gold	726
All-in Sustaining Costs	$/oz gold	893
Site Operating Costs	$/t treated	42.87
After-Tax $NPV_{5\%}$	$M	2,164
Mine Life	years	22

5-Year annual average is for the period 1 January 2021 through 31 December 2025

Table 24.3 PEA Case Before and After-Tax NPV

Discount Rate	Before-Tax NPV ($M)	After-Tax NPV ($M)
Undiscounted	3,312	3,033
5%	2,310	2,164
10%	1,767	1,680
12%	1,617	1,543

Table 24.4 PEA Case Cash Costs

Costs per Ounce (Cash Basis)	Units	PEA Case
Mining and Rehandle	$M	727
Process, Freight, and Refining	$M	1,859
Site Support	$M	442
Royalties	$M	346
Total Production Costs	$M	3,374
Production Cost	$/oz gold	726
Sustaining Capital	$M	479
Fixed Lease Payments	$M	211
Exploration – Sustaining	$M	18
Site G&A	$M	67
All-in Sustaining Costs	$M	4,150
All-In Sustaining Cost	$/oz gold	893

Process, Freight, and Refining includes by-product credits and excludes fixed lease costs
Royalties are calculated in the period incurred

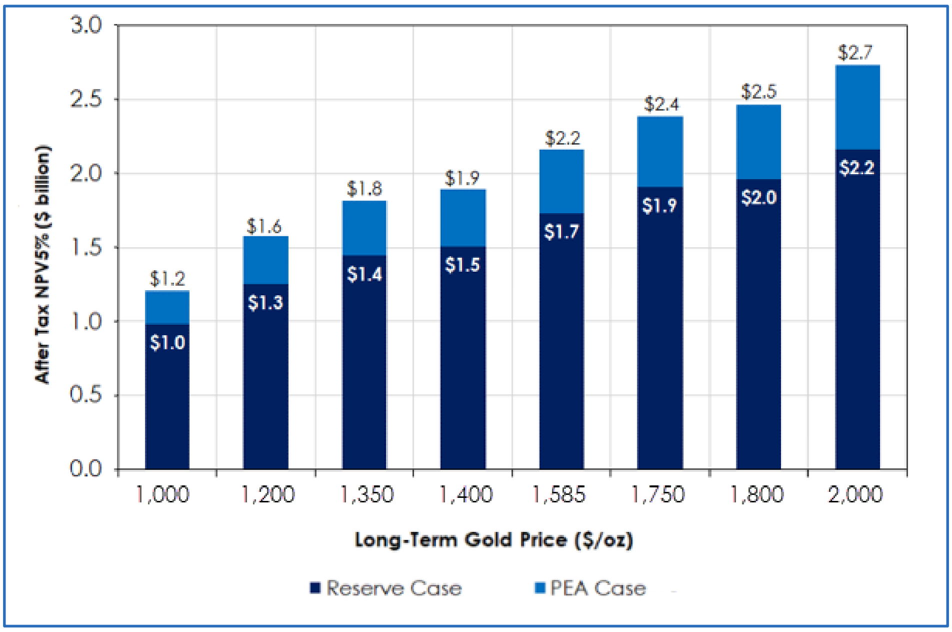
Table 24.5 PEA Case Gold Price Sensitivity

After-Tax NPV ($M)	Long-Term Gold Price ($/oz)							
Discount Rate	**1,000**	**1,200**	**1,350**	**1,400**	**1,585**	**1,750**	**1,800**	**2,000**
Undiscounted	1,481	2,090	2,470	2,598	3,033	3,404	3,528	3,967
5%	1,211	1,574	1,814	1,896	2,164	2,389	2,464	2,730
10%	1,050	1,285	1,446	1,500	1,680	1,827	1,876	2,050
12%	1,002	1,203	1,341	1,389	1,543	1,670	1,712	1,862

Table 24.6 PEA Case Mine Cost Sensitivity

			Change from Base $NPV_{5\%}$ ($M)				
Variable	**Units**	**Base Value**	**–20%**	**–10%**	**0%**	**10%**	**20%**
Capital Cost	$M	651	2,258	2,211	2,164	2,118	2,071
Mining Cost	$/t mined	1.67	2,247	2,206	2,164	2,123	2,081
Processing Cost	$/t treated	27.02	2,255	2,110	2,164	2,119	2,073
Site Operating Cost	$/oz	715	2,533	2,349	2,164	1,977	1,783

Figure 24.1 PEA Case Gold Price Sensitivity

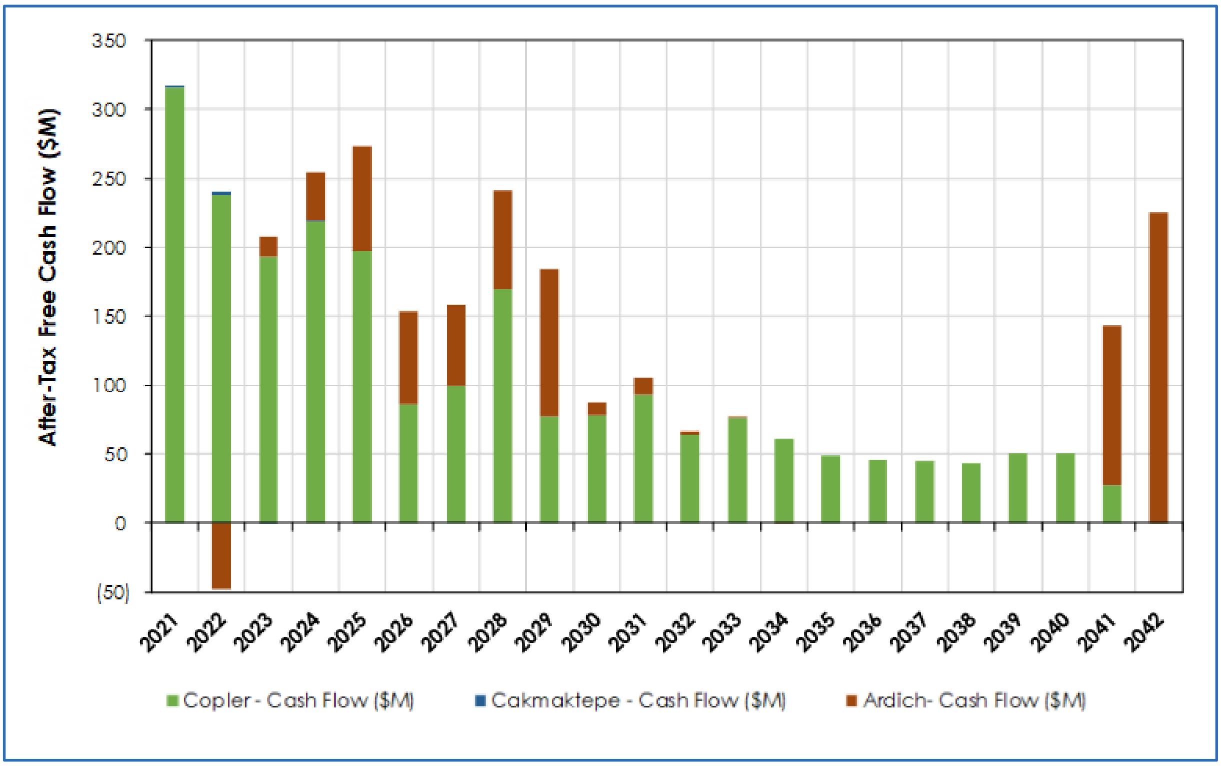


OreWin, 2020

The after-tax cash flow is shown in Figure 24.2. The revenue, operating cost and capital costs and net cash flow is tabulated in Table 24.7.

The estimates of cash flows have been prepared on a real basis with a base date of Q4'20 and a mid-year discounting is used to calculate NPV. All monetary figures have a base date of Q4'20 with no allowance for escalation and are expressed in US dollars (US$) unless otherwise stated. Production costs and AISCs are determined on a per ounce gold produced basis and do not consider the application of inventory movements or deferred waste stripping. Production costs do not equate to cash costs prepared under SSR Mining non-GAAP measures. AISCs do not equate to AISCs prepared under SSR Mining non-GAAP measures.

Figure 24.2 PEA Case After-Tax Cash Flow



OreWin, 2020
Çakmaktepe cash flow occurs 2022–2024

OreWin

Table 24.7 PEA Case Cash Flow

Cash Flow Statement ($M)	TOTAL	Year																							
		Q4'20	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043
Heap Leach – Gold Revenue	1,569	41	120	71	112	135	168	182	165	206	196	95	54	14	11	–	–	–	–	–	–	–	–	–	–
Sulfide Plant – Gold Revenue	6,070	119	501	413	380	407	365	277	285	304	236	235	262	221	215	199	184	163	163	163	163	163	272	380	–
By-Product Revenue	85	5	8	11	10	9	7	9	5	9	5	4	2	0	0	0	0	0	0	0	0	0	0	–	–
Net Revenue	7,723	165	629	495	503	551	539	468	455	518	437	334	318	235	226	199	184	163	163	163	163	163	272	380	–
Realisation Costs																									
Freight and Refining	18	0	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0	0	1	1	–
Royalty Payments	366	–	29	41	26	26	28	25	20	20	24	20	14	13	7	7	6	5	4	4	4	4	5	12	20
Total – Realisation Costs	384	0	31	42	27	27	30	26	22	22	25	21	14	13	8	8	6	6	4	5	5	5	5	13	20
Operating Costs																									
Mining	655	8	30	48	63	61	60	79	79	79	56	50	22	20	–	–	–	–	–	–	–	–	–	–	–
Ore Rehandle	72	1	4	3	3	2	2	3	3	3	5	4	4	3	4	4	4	3	4	3	3	3	2	2	–
Processing – Heap Leach	281	11	19	20	31	31	29	32	25	29	23	21	11	–	–	–	–	–	–	–	–	–	–	–	–
Processing – Sulfide Plant	1,855	22	88	92	89	88	88	90	83	89	86	83	85	84	84	82	82	78	77	77	77	77	78	77	–
Site Support	442	6	42	30	30	30	30	30	30	30	30	26	26	13	13	9	9	9	9	9	9	9	9	9	–
Total – Site Operating Costs	3,305	48	182	193	215	212	209	234	221	229	199	184	148	119	100	94	94	90	89	89	89	89	88	87	–
Exploration	18	4	3	1	1	1	1	1	1	1	1	1	1	1	–	–	–	–	–	–	–	–	–	–	–
Corporate Costs	67	1	5	5	5	5	5	5	5	5	5	5	5	4	0	0	0	0	0	0	0	0	0	0	–
Total – Operating Costs	3,390	54	190	199	222	219	215	241	227	235	206	190	154	125	101	95	94	90	90	90	89	89	88	87	–
Operating Surplus / (Deficit)	3,949	111	409	254	253	305	294	201	206	261	206	124	149	97	117	97	84	68	69	69	69	69	178	279	–20
Capital Costs																									
Growth	58	18	40	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Sustaining	479	17	46	62	40	40	12	42	42	12	12	29	29	12	12	12	12	12	12	12	6	6	–	–	–
Reclamation	114	–	2	0	3	2	0	0	–	–	–	0	2	6	12	12	12	–	2	3	1	1	1	1	54
Working & Other	–14	–4	–4	–2	–1	–1	–1	–0	–0	–0	–0	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Total – Capital Costs	637	31	84	60	42	41	12	42	42	12	12	29	32	18	24	24	24	12	14	15	7	7	1	1	54
Net Cash Flow Before Tax	3,312	79	325	194	212	263	282	159	165	249	194	94	118	79	94	73	60	56	55	54	62	62	177	278	–74
Tax	278	2	7	4	4	9	9	6	6	8	10	7	12	13	16	13	11	10	10	10	11	11	34	54	–
Net Cash Flow After Tax	3,033	77	318	191	208	254	273	153	159	241	184	87	106	67	77	60	49	45	45	43	51	50	143	224	–74

Royalties are paid in the period after they are accrued

24.2 PEA Case Mining

The Ardich deposit is a newly discovered deposit that is separate to the other deposits on the property. Drilling is continuing at the Ardich deposit and it is expected that the drilling will further define the Mineral Resource. The development of Ardich requires development of a new open pit that is 6 km from the current Çöpler pit and 1 km from the Çakmaktepe pit.

The PEA Case assumes that open pit mining is undertaken at Ardich using excavators and trucks and operated by a mining contractor, as is currently the case at the Çöpler pit. The Ardich production is primarily from oxide Mineral Resources. The pit has been split into five phases for production scheduling. phase 5 is mostly Inferred Mineral Resource and although it is close to the surface and next to phase 1, phase 5 has been delayed to the end of the Ardich schedule so that the influence of the Inferred Mineral Resource is reduced. The phase 5 area has been targeted for resource definition drilling to improve the confidence in the estimate of tonnes and grade in that area.

For the PEA Case, oxide ore is assumed to be treated on an expanded area of the existing Çöpler heap leach site. Ardich oxide is progressively stacked on the Çöpler heap leach pads as soon as mined. Ardich sulfide has been assumed to be placed in stockpile and treated once all the Çöpler sulfide feed has been processed. Figure 24.12 shows the timing of the Ardich material being processed, divided into low-sulfur oxide, high-sulfur oxide, and sulfide ore types.

The PEA Case is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

24.2.1 Ardich Pit Optimisation Inputs and Assumptions

The open pit limits for Ardich were identified to estimate potential production quantities and understand the characteristics of the open pit shells. Limits were determined by considering both physical and economic constraints to mining using pit optimisation software.

The mining cell model for Ardich was flagged with three different sulfur ranges for processing purposes, these are shown in Table 24.8.

Table 24.8 Ardich Process Types

Ore Type	S%
Low-Sulfur Oxide	<1
High-Sulfur Oxide	≥1 and <2
Sulfide	≥2

Material processing is based on similar criteria as the other mines where low-sulfur oxide and high-sulfur oxide material is processed via heap leach, whilst sulfide is sent to the sulfide plant. Processing recoveries vary depending on material type.

Table 24.9 outlines gold recovery for the low-sulfur oxide and high-sulfur oxide material.

Table 24.9 Ardich Oxide Heap Leach Gold Recoveries

Material Type	Unit	Low-Sulfur Oxide		High-Sulfur Oxide	
		Main	East	Main	East
Dolomite	%	73	55	58	45
Jasperoid	%	50	50	40	40
Listwanite	%	73	55	58	45

Sulfide POX recovery is calculated by the following formula:

$$\text{POX Gold Recovery} = a \times (1 - \text{EXP}(- b \times (\text{Au(g/t)} - c))) + d$$

Table 24.10 shows the POX gold recovery parameters used for Ardich sulfides.

Table 24.10 Ardich POX Gold Recovery Parameters

Material Type	a	b	c	d
Dolomite	96.7	1.2	–1.4	–1.0
Jasperoid	96.7	1.2	–1.4	–1.0
Listwanite	96.7	1.2	–1.4	–1.0

Economic assumptions for Oxide and Sulfide used in the optimisation process are shown in Table 24.11 and Table 24.12 respectively. A gold price of US$1,350/oz was for the analysis. There are no silver and copper grades in the Resource.

Table 24.11 Ardich Oxide Operating Costs

Parameter	Unit	Cost
Rehandle Cost	$/t	0.40
Processing – Fixed	$/t	3.05
Processing – Variable	$/t	5.44
G&A (Process and Site)	$/t	3.17
Ore Haulage	$/t	1.53
Mining Cost	$/t mined	1.61

Table 24.12 Ardich Sulfide Operating Costs

Parameter	Unit	Cost
Rehandle Cost	$/t	0.90
Processing – Fixed	$/t	8.32
Processing – Variable	$/t	19.10
Processing – Variable (SS)	$/t	2.68
G&A (Process and Site)	$/t	6.60
Ore Haulage	$/t	1.53
Mining Cost	$/t mined	1.61

24.2.2 Ardich Cut-off Grades

Internal Au cut-off grades (COGs) have been calculated for each of the material types based on the economic inputs and assumptions outlined in Section 24.2.1. Internal COGs have been used to calculate process quantities within the preliminary Ardich pit.

Table 24.13 Ardich Internal Au Cut-off Grades

Material Type	Cut-off Variable	Low-Sulfur Oxide		High-Sulfur Oxide		Sulfide	
		Main	East	Main	East	Main	East
Dolomite	Au g/t	0.39	0.52	0.49	0.63	1.11	1.11
Jasperoid	Au g/t	0.57	0.57	0.71	0.71	1.11	1.11
Listwanite	Au g/t	0.39	0.52	0.49	0.63	1.11	1.11

24.2.3 Ardich Pit Optimisation

Pit optimisation was completed using the July 2020 Ardich resource model. Mineral Resources classified as Measured, Indicated and Inferred were used in the optimisation. The pit optimisation work was used to generate pit shells that were then used for the pit phase designs in the PEA Case.

24.2.4 Ardich Pit Design

A preliminary ultimate pit design has been created for Ardich with the aim of determining indicative quantities and layouts. The design is based on a conventional open pit mining method assuming drill and blast, and excavators loading trucks mining 15 m benches in 5 m flitches. A minimum mining width of 20 m was applied to the design.

Table 24.14 shows preliminary design assumptions used for the ultimate pit design.

Table 24.14 Ardich Preliminary Pit Design Assumptions

Criteria	Unit	Value
Batter Angle	degrees	65.00
Berm Width	m	7.95
Bench Height	m	15.00
Road Width	m	15.00
Road Gradient	gradient %	9

Figure 24.3 shows the resulting preliminary ultimate pit design for Ardich.

The preliminary ultimate pit design was divided into five phases to allow schedule flexibility. These five phases are identified in Figure 24.4. Section views through the pit are shown in Figure 24.5 through Figure 24.8.

Figure 24.3 Ardich Preliminary Ultimate Pit Design

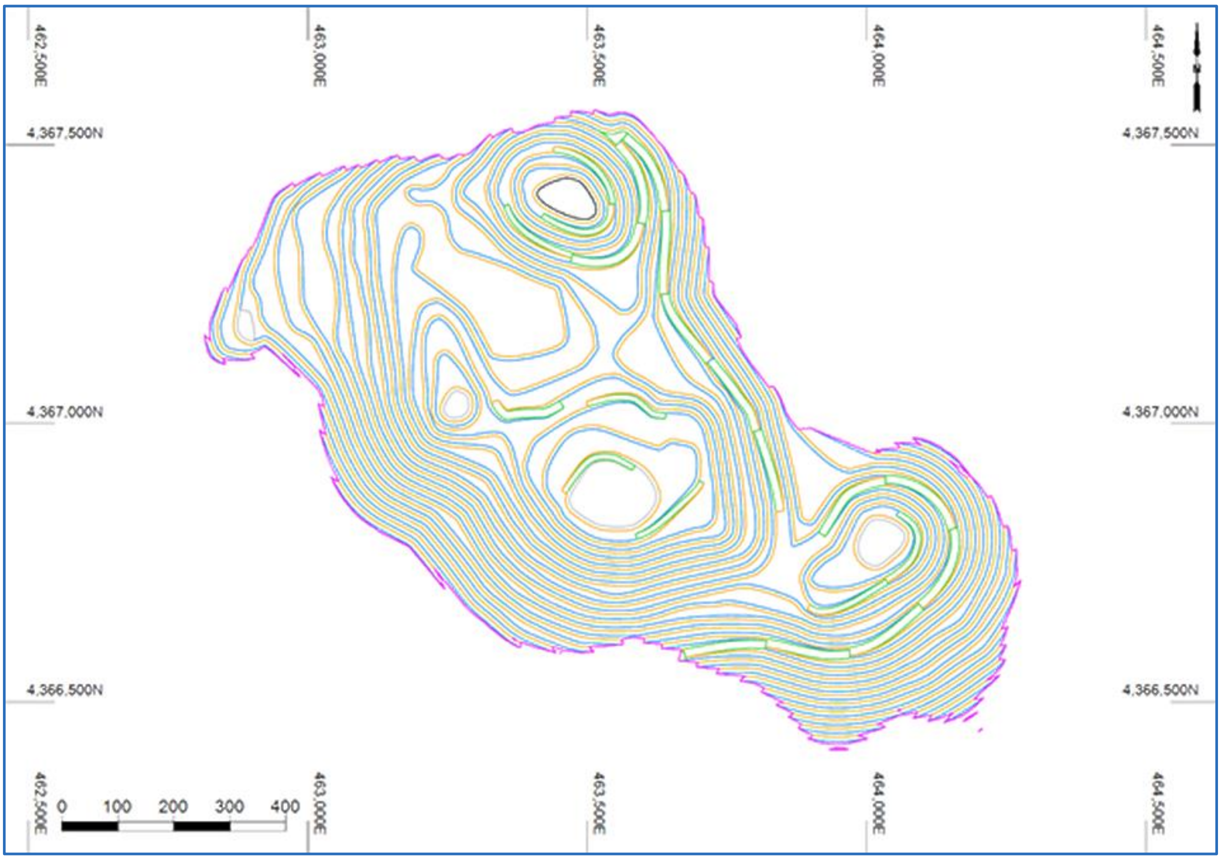


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Figure 24.4 Ardich Pit Phases

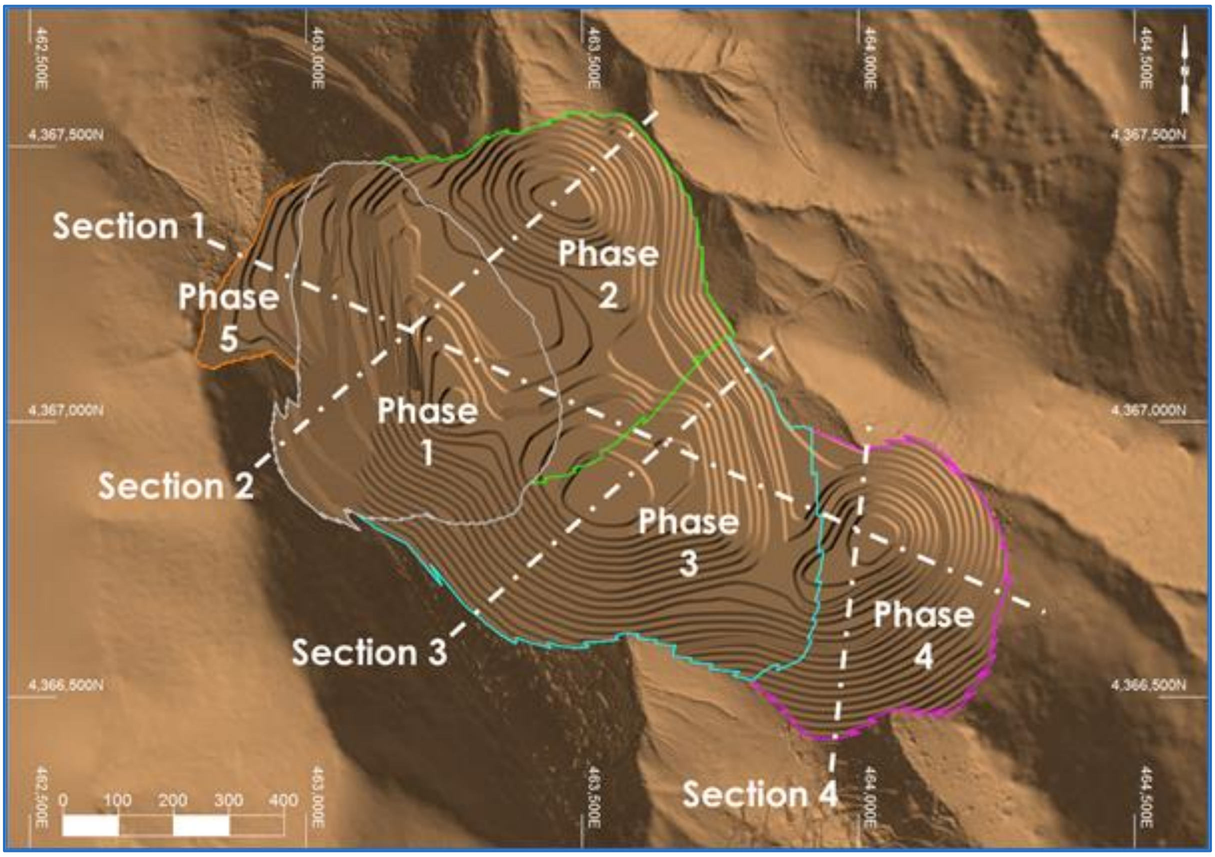

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Figure 24.5 Ardich Pit Phases (long-section 1, looking north-east)

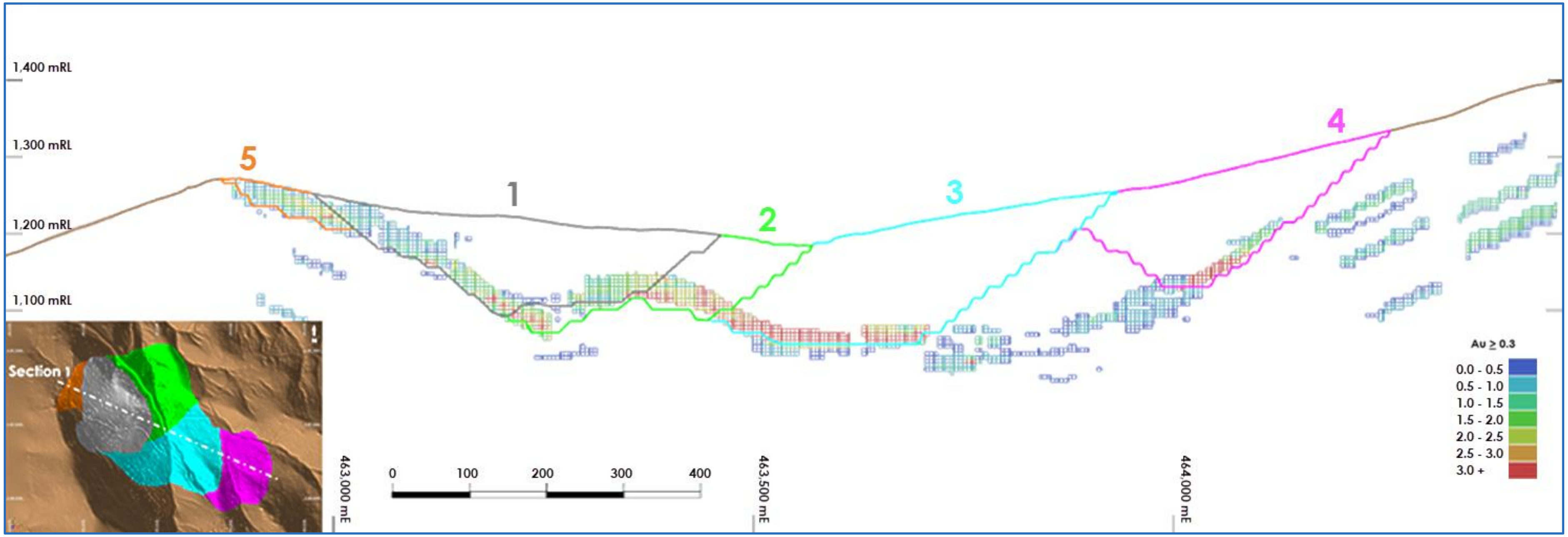

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Figure 24.6 Ardich Pit Phases (cross-section 2, looking north-west)

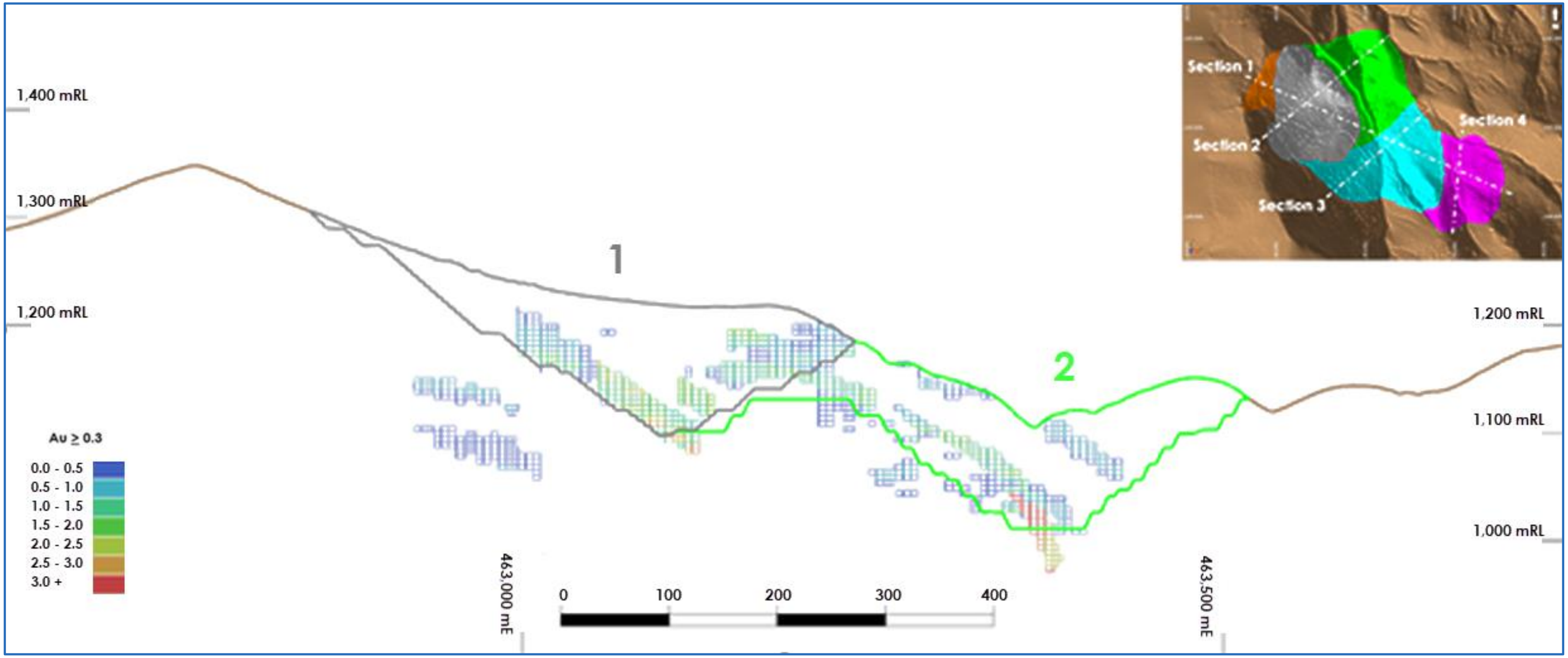

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Figure 24.7 Ardich Pit Phases (cross-section 3, looking north-west)

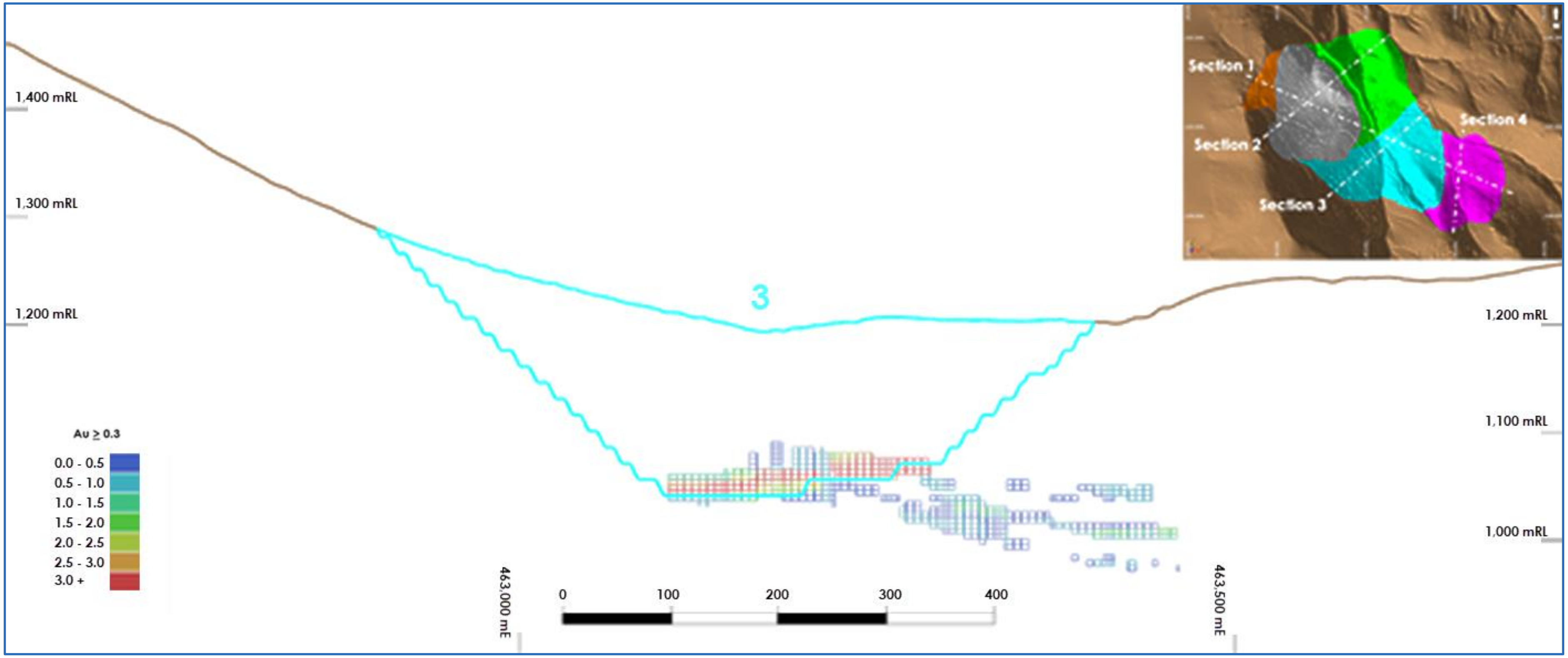

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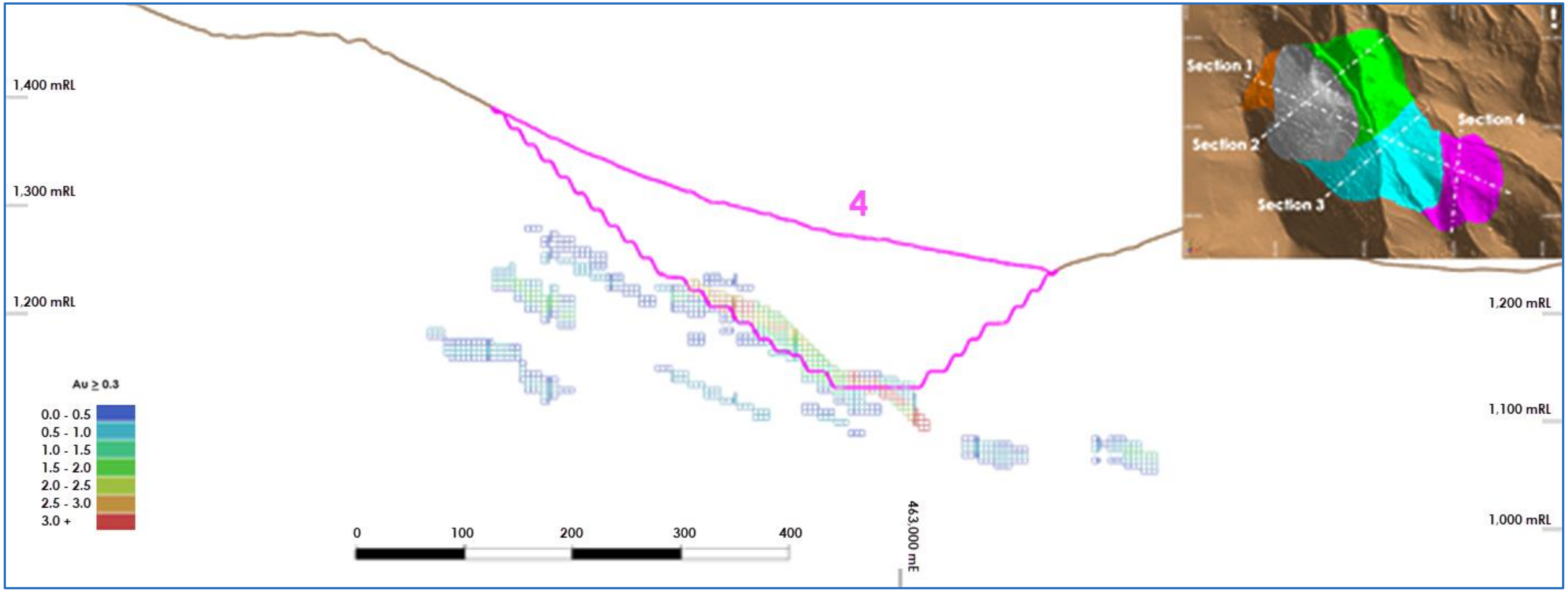
Figure 24.8 Ardich Pit Phases (cross-section 4, looking north-west)



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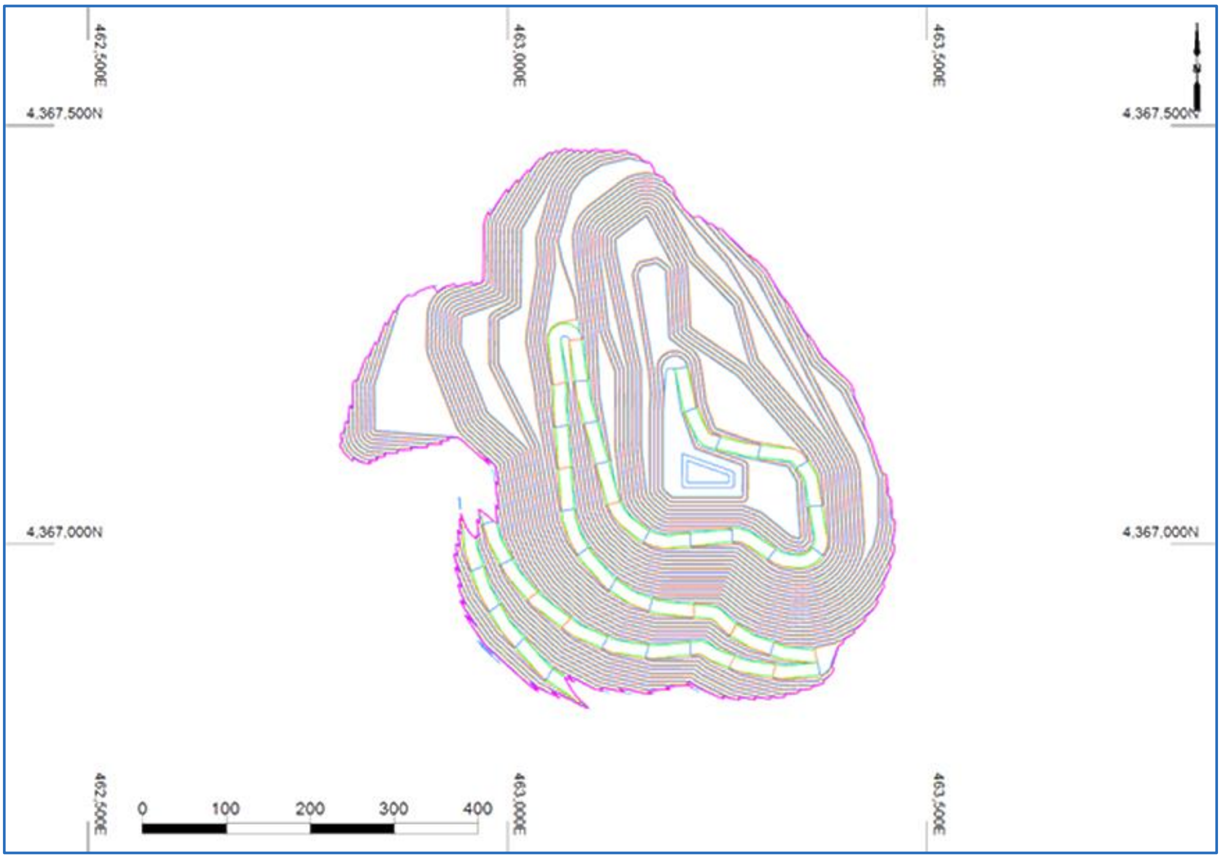
The pit design that was prepared by Anagold for the EIA extension submission is shown in Figure 24.9. The EIA pit design was split into phase 1 and phase 5.

Figure 24.9 EIA – Ardich Pit Design Phase 1 and Phase 5



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24.2.5 Ardich Pit Report

Table 24.15 shows the proportions of each Mineral Resource classification in the PEA Case pit. Table 24.16 shows the resulting process quantities contained within the Ardich pit, these are reported based on the internal COGs at a gold price of $1,350/oz. Process quantities include Inferred Mineral Resource material.

Table 24.15 PEA Case Material Classification

Classification	Unit	Total
Measured	%	14
Indicated	%	69
Inferred	%	17
Mineral Resources in PEA	%	100

Table 24.16 Ardich Process Pit Report

Classification	Tonnage (kt)	Au (g/t)	Contained Gold (koz)
Ardich – Oxide Low Sulfur			
Measured	2,426	1.37	107
Indicated	9,843	1.74	552
Total	12,269	1.67	659
Ardich - Oxide High Sulfur			
Measured	249	2.13	17
Indicated	2,021	3.01	196
Total	2,270	2.92	213
Ardich - Sulfide			
Measured	263	2.62	22
Indicated	2,111	3.82	259
Total	2,373	3.69	281
Ardich Total			
Measured	2,938	1.55	146
Indicated	13,975	2.24	1,007
Total	16,913	2.12	1,153
Inferred			
Ardich – Oxide Low Sulfur	2,309	1.87	139
Ardich - Oxide High Sulfur	493	2.68	42
Ardich - Sulfide	616	4.67	93
Ardich Total	3,419	2.49	274

24.3 PEA Case Production Schedule

For the PEA Case, Ardich oxide is assumed to be treated on an expanded area of the existing Çöpler heap leach site, progressively stacked on the Çöpler heap as soon as mined. Ardich sulfide has been assumed to be placed in stockpile and treated once all the Çöpler sulfide production has been processed.

Figure 24.10 shows the total movement for the PEA Case. The PEA case oxide and sulfide production schedules are shown in Figure 24.11 and Figure 24.12 respectively.

Oxide material from Ardich is introduced into heap leach production from 2022 and is a significant increase compared to the Reserve Case.

Sulfide feed production throughputs are limited dependent on ore tonnage and sulfide sulfur (SS) tonnage as in the Reserve Case. Sulfide material from Ardich is stockpiled and fed into the plant at the end of the project life.

The production scenario adopted for Ardich was to commence mining with a total movement limit of 15 Mtpa for the first three years. This was then increased to 25 Mtpa from the fourth year onwards. The schedule allows for pre-stripping of 4.6 Mt of waste material prior to the first year of mining in 2022.

Initial mining is to begin with phase 1. Mine phases are then staggered to allow for control of production and waste movements. All sulfide material is stockpiled and later fed to the sulfide plant based on plant capacity and material type. Oxide material quantities peak at 2,500 kt in 2024 and remain steady until 2026. Oxide material is placed directly onto the heap leach.

It is assumed that mining is carried out under the same contract agreement as the other SSR Mining mines in the area.

Figure 24.13 shows the mine schedule phases and associated start dates. Phase 5 contains a high proportion of Inferred material and has been delayed in the production schedule to reduce the impact of Inferred material on the mine schedule and Figure 24.14 shows a long-section through the pit phases.

Table 24.17 shows the PEA Case mining schedule, Table 24.18 shows the PEA Case process schedule and Table 24.19 shows the mining and process production for the Ardich Mineral Resource in the PEA Case.

Figure 24.15 shows material movement for the mine divided into waste mined, tonnages, and grades for oxide mined and sulfide mined. Figure 24.16 shows the mining timing of the process feed and waste mined, with process feed divided into individual phase areas. Figure 24.17 shows low and high sulfide oxide and sulfide processing.

Figure 24.18 shows material mined for processing by classification type. The majority of the Inferred material has been scheduled for mining towards the end of the mine life from 2028 onwards and sulfide ore is processed at the end of the project life (2041 and 2042).

Figure 24.10 PEA Case Mining Production

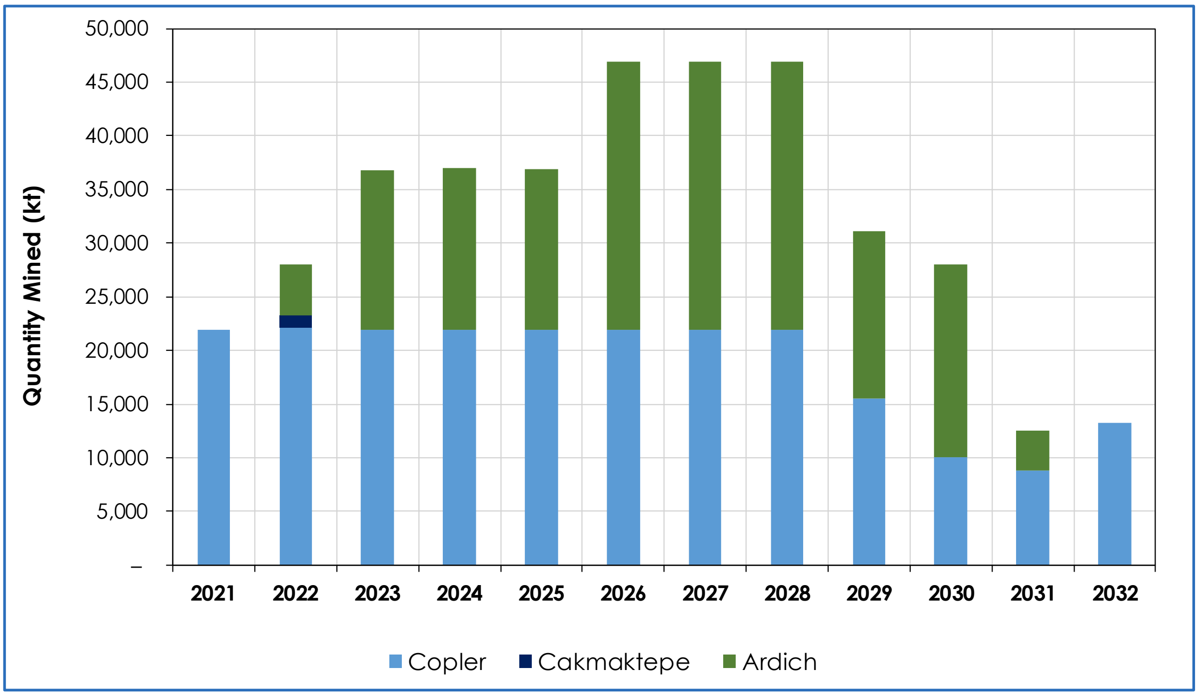

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Figure 24.11 PEA Case Oxide Production Schedule

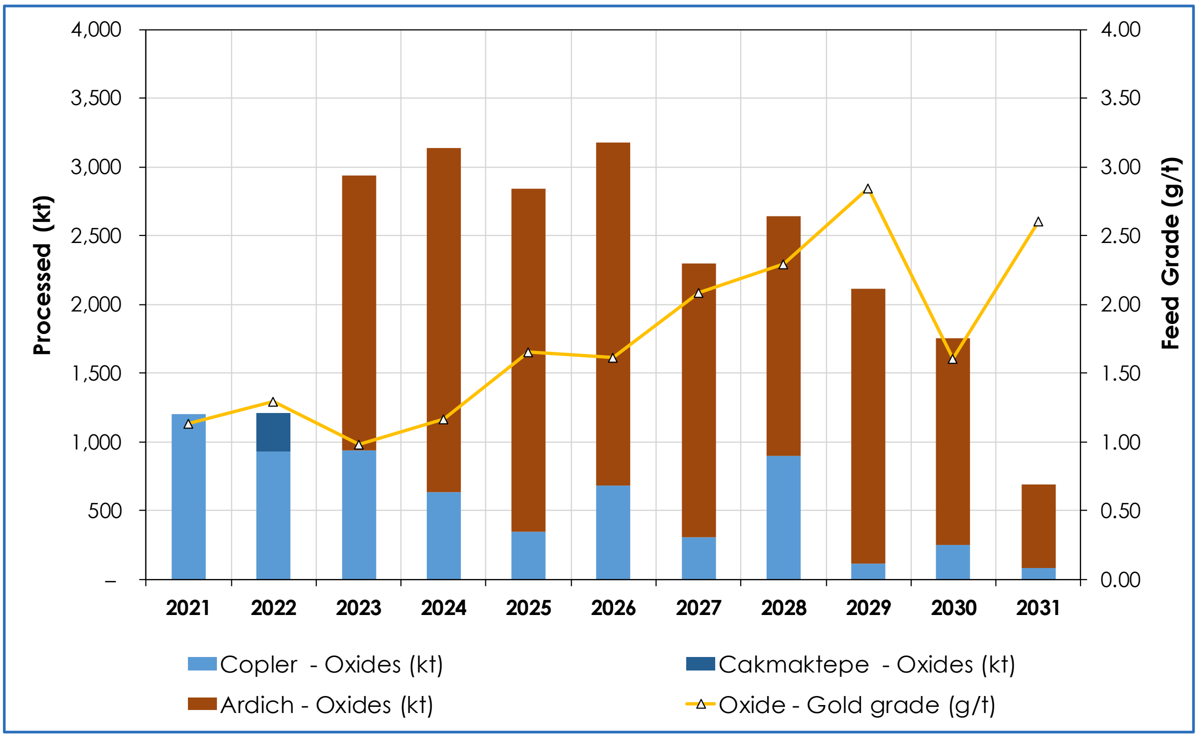

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Figure 24.12 PEA Case Sulfide Production Schedule

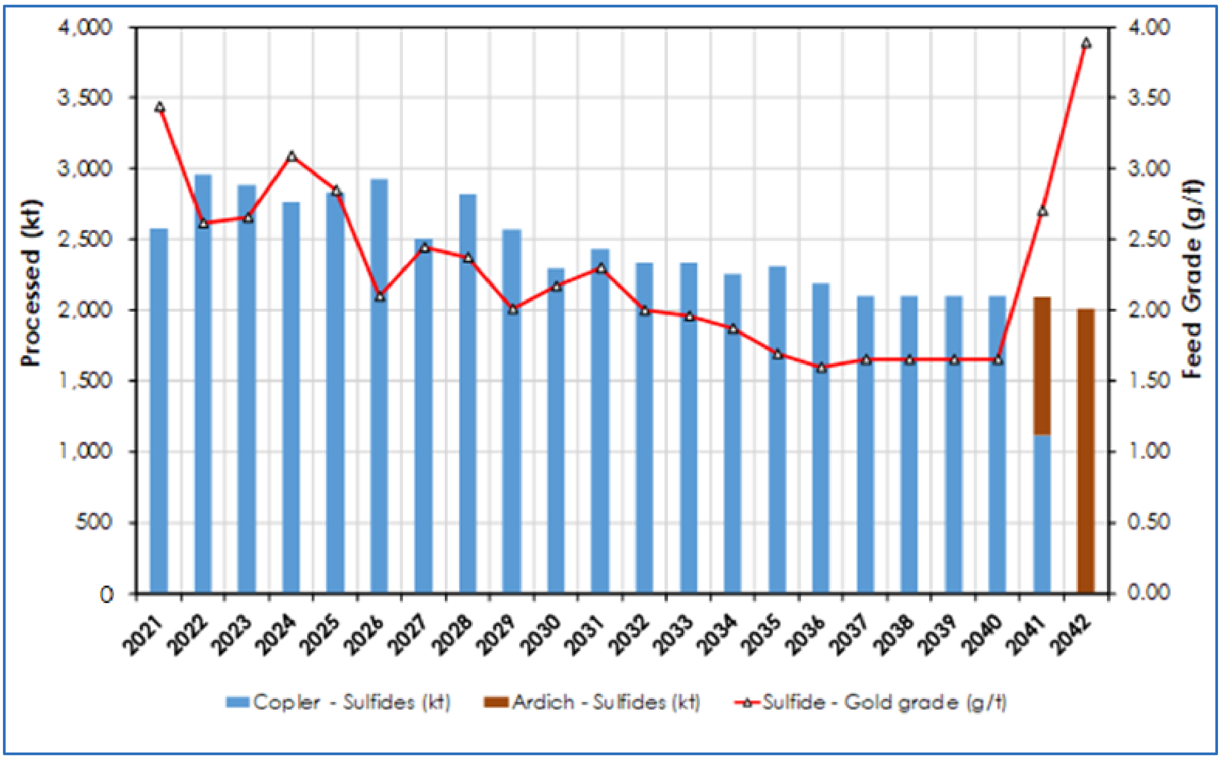

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Figure 24.13 Ardich Mine Schedule Phases

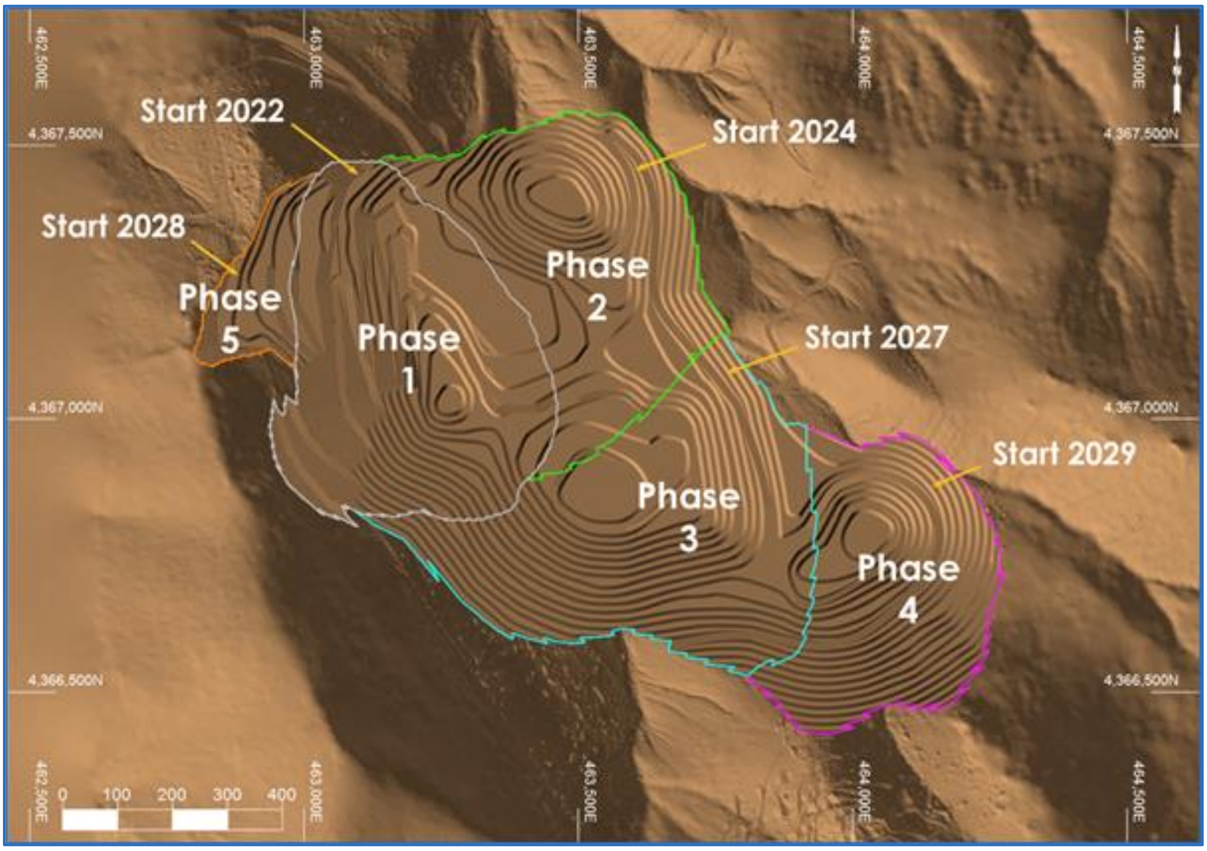

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Figure 24.14 Ardich Pit Phases (long-section 1, looking north-east)

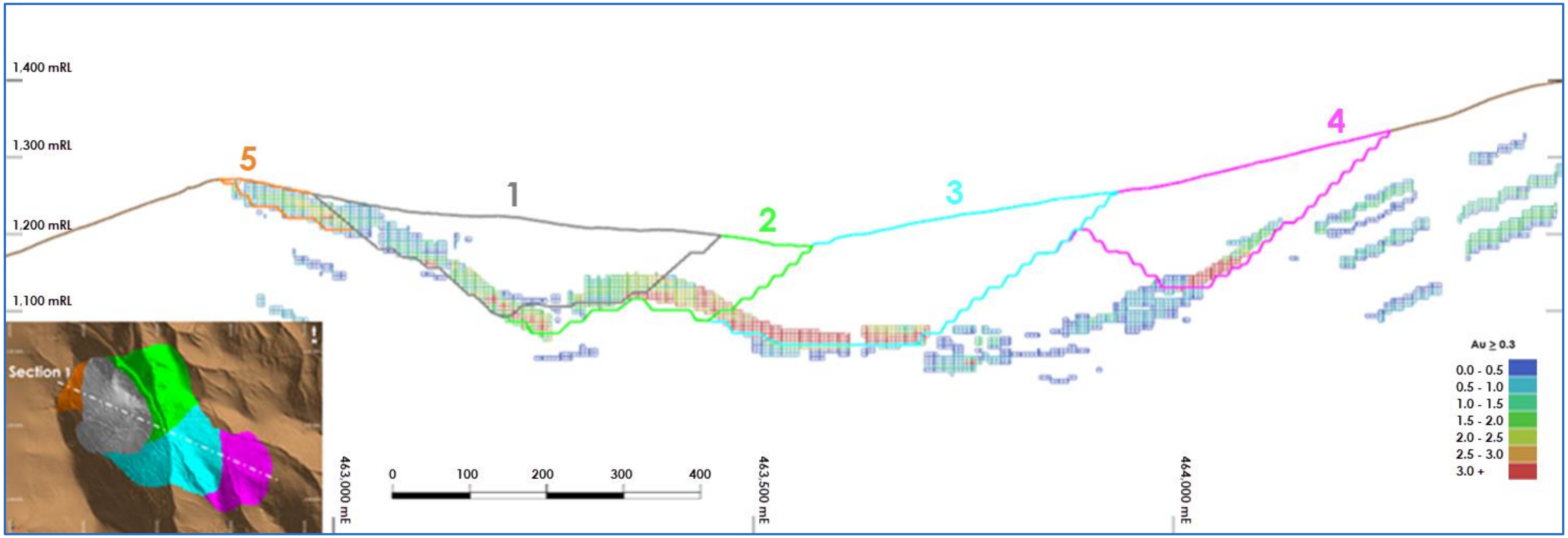

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Figure 24.15 Ardich Material Movement

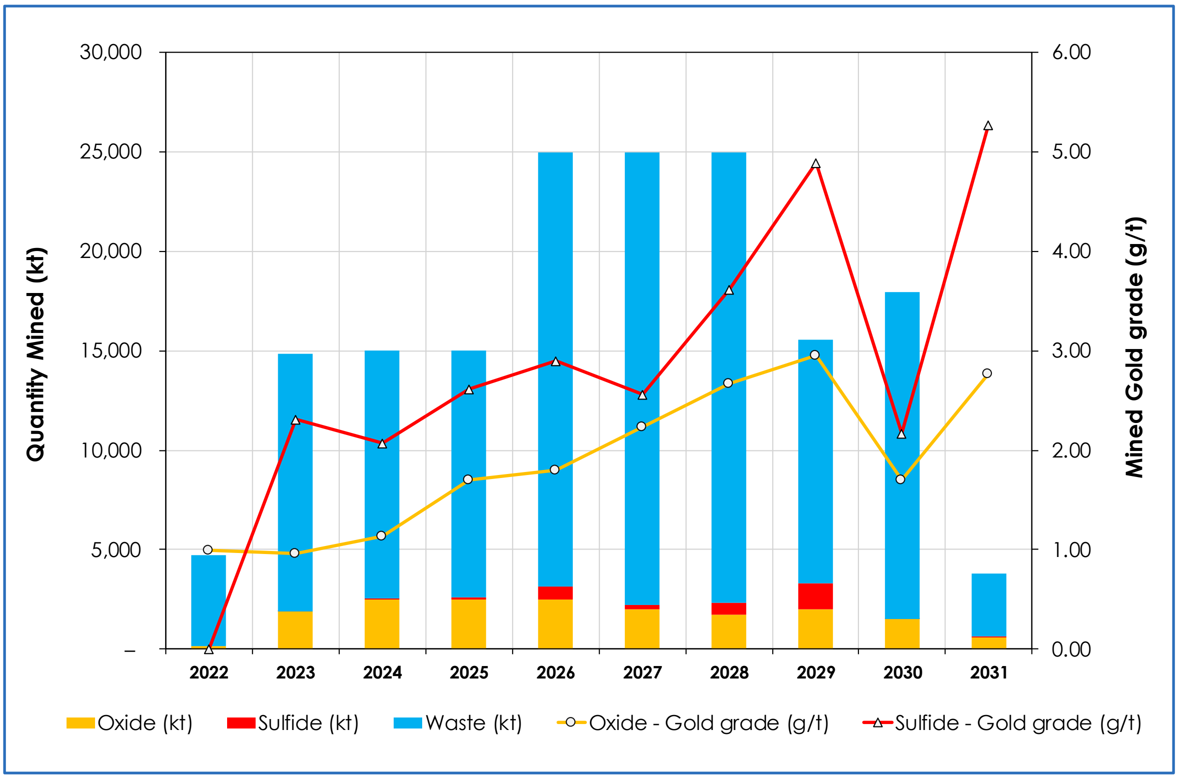

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Figure 24.16 Ardich Material Movement by Phase

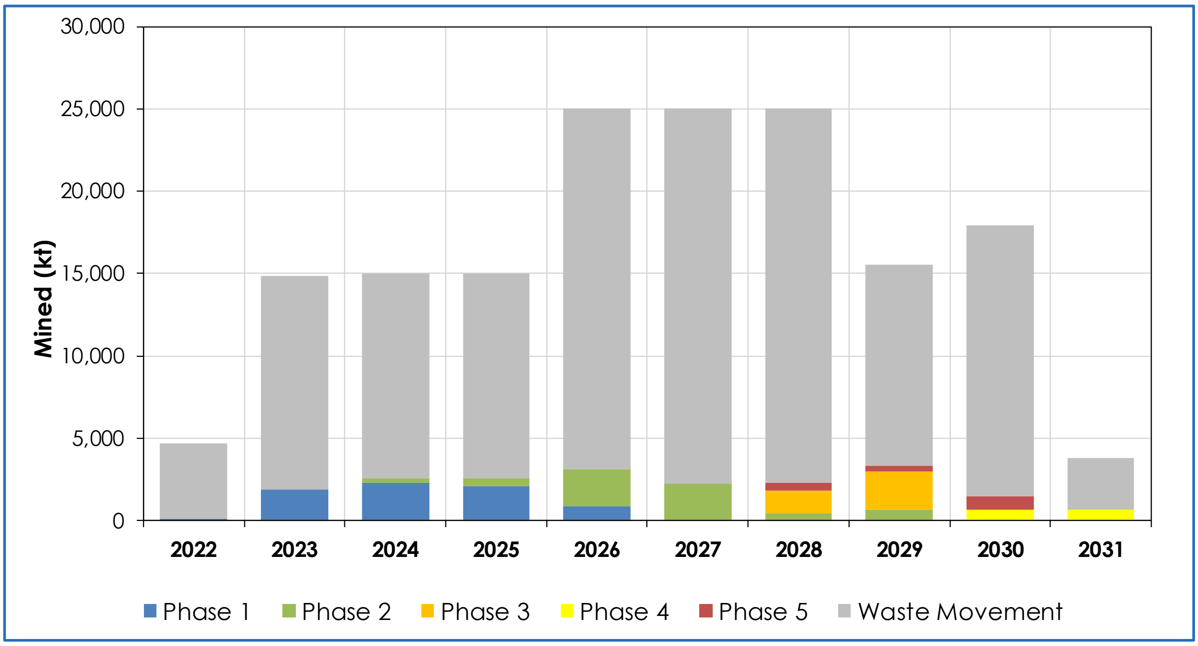

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Figure 24.17 Ardich Material Processed

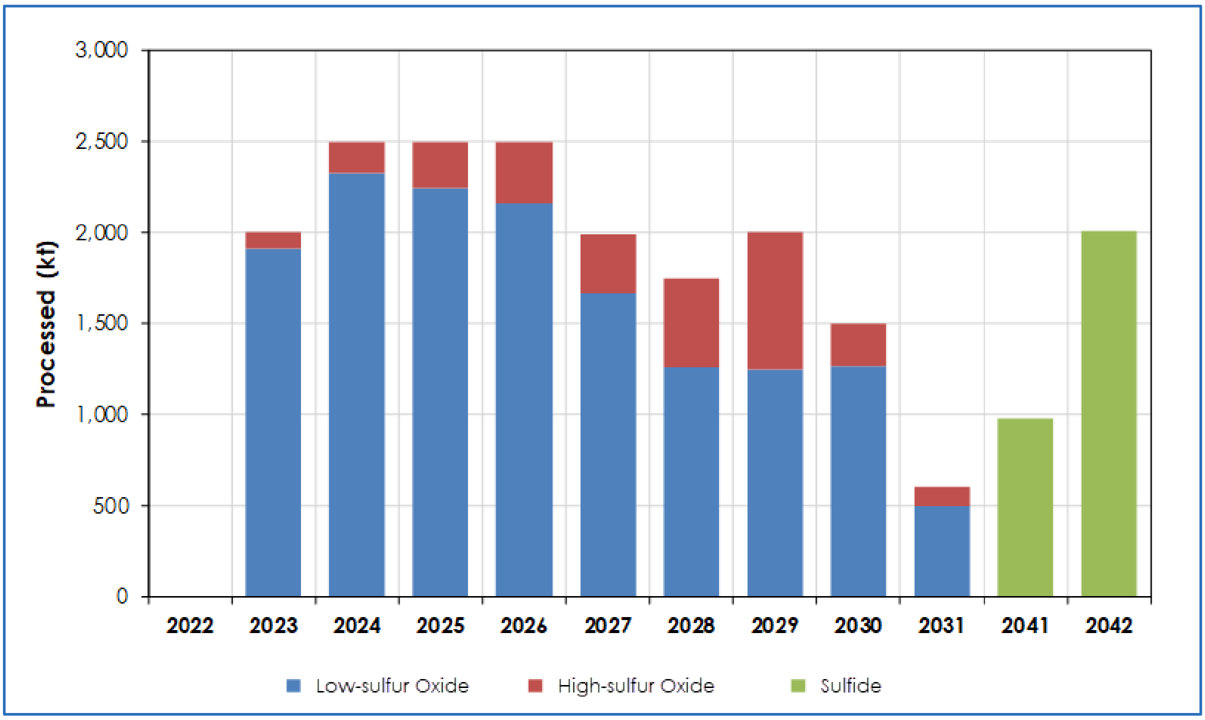

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Figure 24.18 Ardich Material Mined by Classification

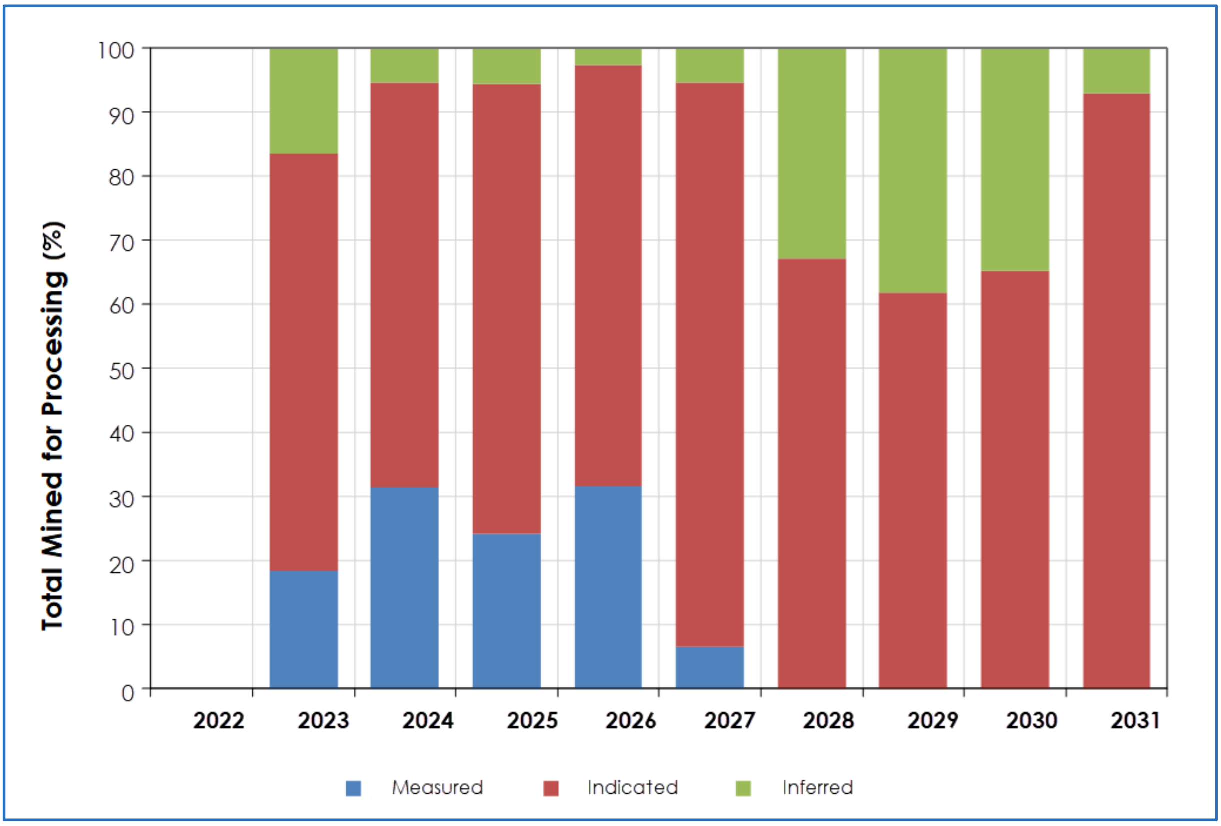

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Table 24.17 PEA Case Mining Schedule

Year	Total Tonnes (kt)	Oxide				Sulfide				Waste Tonnes (kt)
		Tonnes (kt)	Au (g/t)	Ag (g/t)	Cu (%)	Tonnes (kt)	Au (g/t)	Ag (g/t)	SS (%)	
Q4'20	6,898	985	1.31	6.53	0.11	1,387	2.55	10.37	5.07	4,525
2021	21,900	1,205	1.13	3.77	0.11	6,966	2.36	9.13	3.96	13,728
2022	27,976	1,349	1.27	14.65	0.11	2,880	2.24	8.54	4.31	23,746
2023	36,771	2,803	0.98	0.66	0.05	3,823	2.30	2.43	3.68	30,145
2024	36,960	3,137	1.17	0.88	0.02	5,056	2.45	5.73	4.05	28,767
2025	36,900	2,845	1.65	0.48	0.02	5,377	2.33	4.45	4.00	28,678
2026	46,900	3,183	1.62	0.18	0.04	5,104	1.88	2.99	4.20	38,613
2027	46,900	2,297	2.08	0.34	0.01	4,422	2.06	4.40	4.87	40,181
2028	46,960	2,645	2.29	3.36	0.03	3,621	2.45	7.88	4.26	40,695
2029	31,084	2,089	2.87	0.16	0.00	2,200	3.62	1.50	4.68	26,795
2030	27,964	1,541	1.69	0.31	0.00	1,032	1.77	4.57	5.11	25,391
2031	12,545	619	2.73	0.12	0.00	1,854	1.88	3.97	4.98	10,072
2032	13,202	309	1.16	6.75	0.07	3,506	2.06	4.17	4.54	9,387
Total	392,960	25,008	1.69	2.00	0.04	47,227	2.29	5.46	4.28	320,724

Table shows mining schedule does not show processing or existing stockpile rehandle

Table 24.18 PEA Case Process Schedule

Description	Units	Total	Year																					
			Q4'20	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042
Heap Leach Stacked	**kt**	**25,008**	**985**	**1,205**	**1,214**	**2,938**	**3,137**	**2,845**	**3,183**	**2,297**	**2,645**	**2,112**	**1,755**	**691**	**–**	**–**	**–**	**–**	**–**	**–**	**–**	**–**	**–**	**–**	**–**
Au Feed Grade	g/t	**1.69**	1.31	1.13	1.30	0.98	1.17	1.65	1.62	2.08	2.29	2.85	1.61	2.60	–	–	–	–	–	–	–	–	–	–	–
Ag Feed Grade	g/t	**2.00**	6.53	3.77	16.28	0.63	0.88	0.48	0.18	0.34	3.36	0.27	0.95	1.04	–	–	–	–	–	–	–	–	–	–	–
Cu Feed Grade	%	**0.04**	0.11	0.11	0.12	0.05	0.02	0.02	0.04	0.01	0.03	0.00	0.01	0.01	–	–	–	–	–	–	–	–	–	–	–
Gold Recovered	koz	**956**	22	61	39	64	82	106	115	104	130	123	60	34	9	7	–	–	–	–	–	–	–	–	–
Silver Recovered	koz	**511**	54	43	193	31	34	16	8	9	90	9	17	7	1	0	–	–	–	–	–	–	–	–	–
Copper Recovered	klb	**7**	0.8	1.0	1.1	1.2	0.6	0.4	0.9	0.2	0.7	0.1	0.1	0.1	0.0	0.0	–	–	–	–	–	–	–	–	–
Sulfide Plant Feed	**kt**	**54,073**	**585**	**2,574**	**2,961**	**2,884**	**2,761**	**2,828**	**2,926**	**2,507**	**2,821**	**2,572**	**2,295**	**2,435**	**2,339**	**2,332**	**2,254**	**2,310**	**2,192**	**2,097**	**2,097**	**2,097**	**2,097**	**2,097**	**2,011**
Au Feed Grade	g/t	**2.33**	3.75	3.44	2.61	2.65	3.09	2.85	2.10	2.45	2.37	2.01	2.17	2.30	2.00	1.96	1.87	1.70	1.60	1.66	1.66	1.66	1.66	2.70	3.89
Ag Feed Grade	g/t	**4.79**	9.83	10.56	6.09	2.97	5.77	5.14	2.64	5.43	9.52	7.48	4.56	3.27	0.90	3.91	5.27	4.09	4.33	4.35	4.35	4.35	4.35	2.32	–
SS Feed Grade	%	**4.28**	4.48	4.10	3.85	3.76	3.96	3.92	3.76	4.25	3.89	4.17	4.50	4.40	4.28	4.45	4.73	4.54	4.73	4.75	4.75	4.75	4.75	4.48	4.18
Gold Recovered	koz	**3,691**	64	255	225	218	247	230	175	180	192	149	148	166	140	136	126	116	103	103	103	103	103	171	240
Silver Recovered	koz	**250**	6	26	17	8	15	14	7	13	26	19	10	8	2	9	11	9	9	9	9	9	9	5	–
Total Feed	**kt**	**79,082**	**1,570**	**3,780**	**4,175**	**5,823**	**5,898**	**5,674**	**6,109**	**4,804**	**5,465**	**4,684**	**4,050**	**3,126**	**2,339**	**2,332**	**2,254**	**2,310**	**2,192**	**2,097**	**2,097**	**2,097**	**2,097**	**2,097**	**2,011**
Total Metal Recovered																									
Gold Recovered	koz	**4,646**	86	316	264	282	329	336	290	284	321	273	208	200	148	142	126	116	103	103	103	103	103	171	240
Silver Recovered	koz	**761**	60	69	210	39	49	30	15	22	116	28	27	14	3	9	11	9	9	9	9	9	9	5	–
Copper Recovered	klb	**7**	0.8	1.0	1.1	1.2	0.6	0.4	0.9	0.2	0.7	0.1	0.1	0.1	0.0	0.0	–	–	–	–	–	–	–	–	–

Table 24.19 Ardich Production Schedule

Description	Units	Total	Year																					
			Q4'20	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042
Total Mined for Processing	kt	20,331	–	–		2,014	2,557	2,592	3,141	2,237	2,320	3,305	1,506	660	–	–	–	–	–	–	–	–	–	–	–
Processed – Heap Leach	kt	17,341	–	–	–	2,000	2,500	2,500	2,500	1,988	1,746	2,000	1,502	605	–	–	–	–	–	–	–	–	–	–	–
Au Grade	g/t	1.89	–	–	–	0.96	1.14	1.70	1.80	2.24	2.67	2.95	1.70	2.76	–	–	–	–	–	–	–	–	–	–	–
Gold Recovered	koz	699	–	–	–	44	65	97	101	95	98	121	50	28	–	–	–	–	–	–	–	–	–	–	–
Processed – Sulfide	kt	2,990	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	979	2,011
Au Grade	g/t	3.89	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	3.89	3.89
Gold Recovered	koz	356	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	117	240
Waste Movement	kt	141,524	–	–	4,561	12,992	12,443	12,408	21,859	22,763	22,680	12,231	16,458	3,128	–	–	–	–	–	–	–	–	–	–	–
Total Movement	kt	161,855	–	–	4,696	14,871	15,000	15,000	25,000	25,000	25,000	15,537	17,964	3,788	–	–	–	–	–	–	–	–	–	–	–
Stripping Ratio (W:O) *	t	7.0	–	–	–	6.5	4.9	4.8	7.0	10.2	9.8	3.7	10.9	4.7	–	–	–	–	–	–	–	–	–	–	–

Ardich mining is complete in 2031. Production in 2041 and 2042 is from stockpiles

* Stripping Ratio reported as tonnes of waste per one tonne of plant feed mined

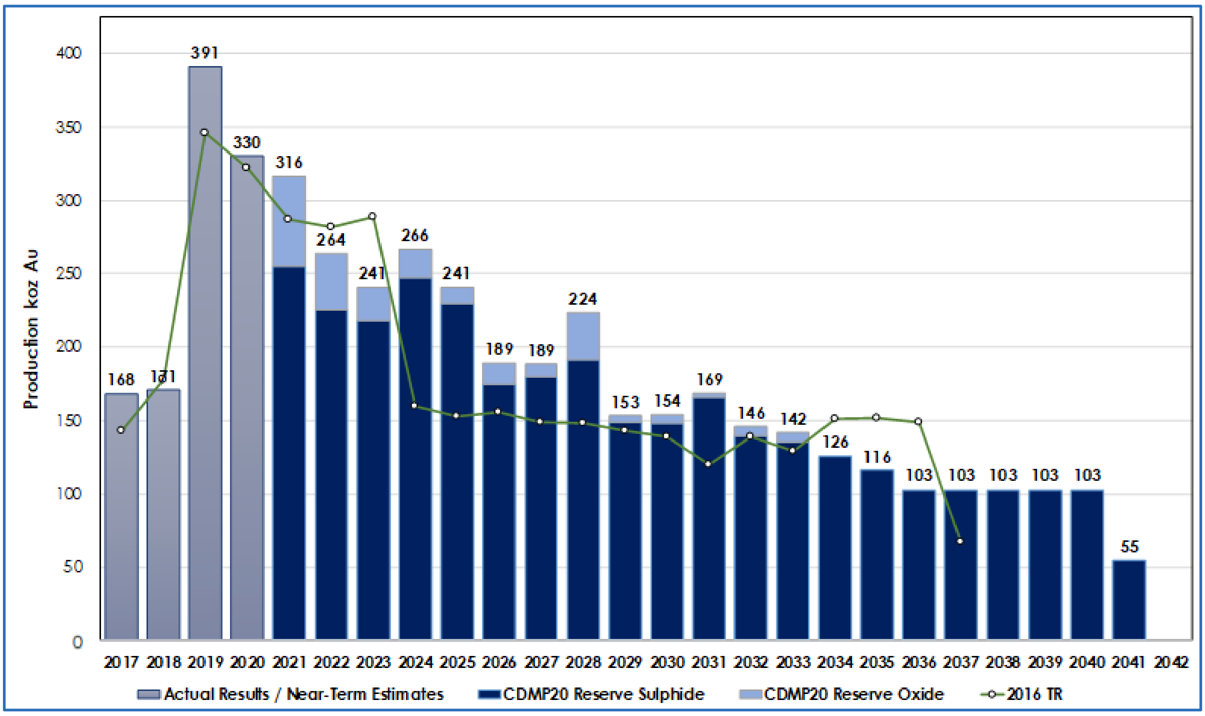
24.4 CDMP20 Comparison with 2016 Technical Report

A comparison of gold production in the CDMP20 cases and the 2016 Technical Report was prepared. Figure 24.19 details the Reserve Case gold production, the 2016 Technical Report, and actual / near-term estimates. The 2020 production is based on actual for Q1'20 through Q3'20 and a forecast estimate for Q4'20. Figure 24.20 shows the incremental change in gold production in the PEA Case from the addition of Ardich.

Actual gold production from the Çöpler project matched with the 2016 Technical Report for 2016 and 2017, while a large increase was experienced in 2019. Projections for 2020–2021 are again forecast to outperform the 2016 Technical Report gold production. The Reserve Case metal production is very similar to the 2016 Technical Report profile, with only a small dip expected in 2022–2023, gains in 2024–2033, and an extension to the tail, overall adding 0.69 Moz of total gold production relative to the 2016 Technical Report (2021–LOM). The PEA Case removes much of the 2022–2023 dip in gold production, and strongly outperforms the 2016 Technical Report from 2024 through 2031. The PEA Case also adds a further extension to the tail gold production, adding 1.06 Moz total gold relative to the Reserve Case (1.75 Moz relative to the 2016 Technical Report).

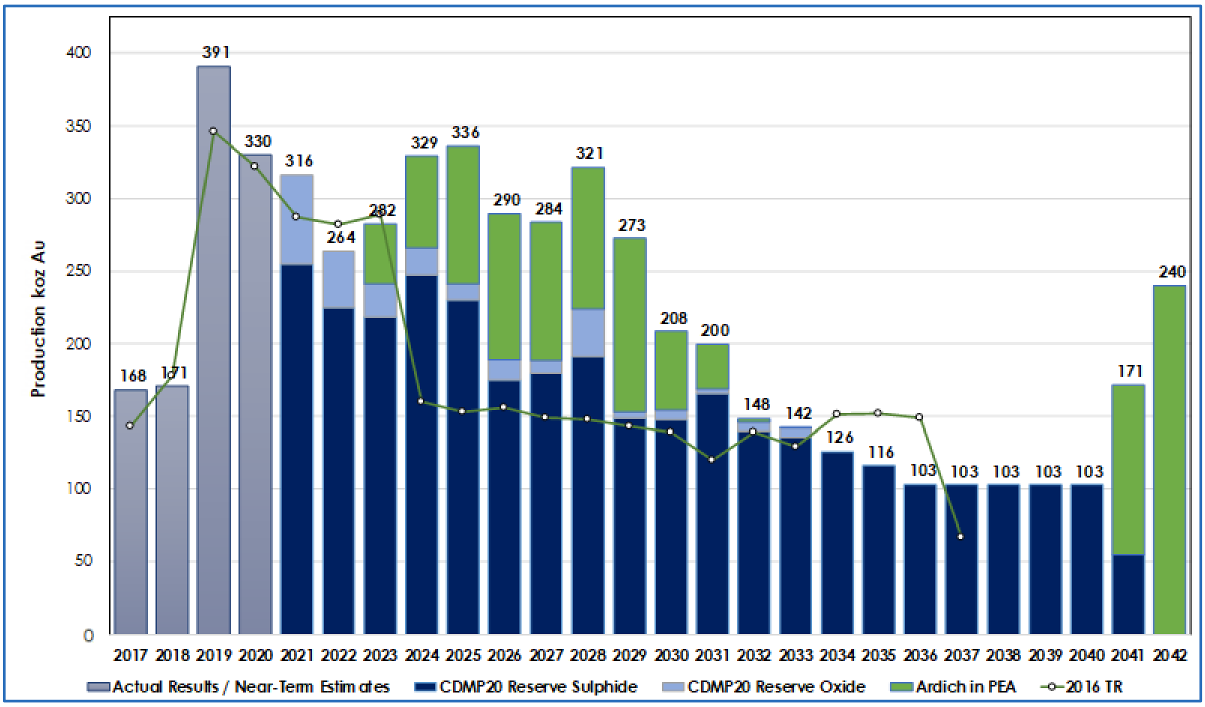
The PEA Case is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Figure 24.19 CDMP20 Reserve Case Gold Production



OreWin, 2020

Figure 24.20 CDMP20 PEA Case Gold Production



OreWin, 2020

25 INTERPRETATION AND CONCLUSIONS

Mineral Resources and Mineral Reserves in the CDMP20 meet the CIM Definition Standards on Mineral Resources and Reserves 2014 (CIM Definition Standards) and conform to the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

Significant factors that could materially affect the Mineral Resources and Mineral Reserve are:

- Environmental, Permitting Social and Community – the Çöpler project is subject to the laws and regulations of Turkey, the mine has a number of local communities that are nearby. In order to operate the mine, Anagold must maintain appropriate relations with all the authorities and stakeholders. Social, community and government relations are managed by Anagold and include programmes and engagement with the local communities and both local and national governments. Anagold has remained in compliance with all aspects of the EIA and operating permits throughout the history of the project.
- Seismic impacts – the Çöpler project is located in an area with a history of significant seismic activity that could negatively impact mining operations.
- Metal price impacts – gold is the primary revenue element and silver and copper are produced as by-products. The ore is mined at an elevated cut-off grade and low-grade ore is stockpiled for processing after mining is completed. The use of the elevated cut-off grade serves to mitigate the risks from periods of lower gold prices.
- Mining impacts – the mining equipment is suitable for a selective mining unit (SMU) of approximately 3 m x 3 m x 5 m. This allows for selectivity in mining and enhances the opportunities for blending the feed to the sulfide plant. The total mining rates in the CDMP20 mine plan are at 22.5 Mtpa. In the past, total mining rates of 36.5 Mtpa have been achieved, increasing the total mining rate may allow gold to be brought forward in the production schedule but will require additional stockpile storage areas.
- Geotechnical impacts – slope recommendations have significant impacts on the Mineral Reserve and the continued study will allow the Mineral reserves to be maximised.
- Processing impacts – the processing analysis in the Reserve Case includes incorporation of a flotation circuit into the existing sulfide plant to upgrade sulfide sulfur to fully utilise grinding and pressure oxidation (POX) autoclave capacity. Continued debottlenecking of the sulfide plant and optimisation of the flotation circuit when it commences operations may improve costs and recoveries, changing cut-off grades and impacting the Mineral Reserve.
- The addition of the flotation circuit to the sulfide plant requires new grade control protocols and a new associated stockpile strategy will be implemented to manage the required sulfide plant feed blend. It is likely that there will need to be a modification of the stockpiling cut-offs and procedures for both short-term and longer term blending, such as increasing the number of active mining areas, increasing the mining rate, and increasing the size or number of ROM stockpiles.

26 RECOMMENDATIONS

Key recommendations from the CDMP20 are:

- Continue to update and evaluate the Çöpler District Master Plan as the existing Mineral Resources and Mineral Reserves are updated and as new prospects are advanced.
- Re-design of Çöpler pits at updated metal prices.
- Geotechnical review and study of the re-evaluation of the re-designs.
- Optimisation of the sulfide flotation circuit, POX, and process operation.
- Metallurgical testwork on future oxide and sulfide ore sources.
- Optimisation of the oxide heap leach circuit.
- Optimisation of the mining rates to increase gold production.
- Stockpile reconciliation and management studies.
- Review and adapt the ore control and stockpiling strategies to maximise gold production.
- Continue drilling at Ardich.
- Geotechnical studies of Ardich.
- Reconciliation studies of Çöpler.
- Update Çöpler and Ardich resource models and estimates.
- Further study of PEA Case and advance to next stage of study:
  - Geotechnical studies
  - EIA and permitting
  - Blasting studies
  - Metallurgical studies

26.1 Mineral Resources

Specific recommendations related to the Mineral Resource are:

- Mineral Resource models should be updated on a campaign basis following the completion of planned drilling programmes. Where significant new data has been obtained (either exploration data, or production data), an annual model update roster should be adequate, but only required where warranted by the introduction of new data that has potential to result in a material change in the model (such as by significant modifications to the geological interpretation, or by substantial expansion of the dimensions of the mineralisation).
- The Çöpler model has not been updated since 2016. It is recommended that a new model be developed to incorporate the new exploration data obtained since that time, and to check interpretations relative to grade control data to help hone the interpretation.

- Continue drilling at Ardich.
- An update to the Ardich model is warranted given the quantum of new data that has been obtained since the most-recent update, and the status of the deposit as shown in the PEA Case.
- Both Çöpler and Ardich are geologically complex deposits with multiple metals that must be tracked along with oxidation type and lithological domains, further complicated by extensive structural disruption. Work on verifying and adjusting resource model domains and parameters should be continued to help facilitate a greater understanding of the deposits, hopefully resulting in improved resource estimates.
- Since the mineralisation locally follows the lithological contacts and structural features, using a search ellipse that follows these trends (dynamic anisotropy) should be evaluated in future models.
- An audit of the databases used to house exploration and grade control data should be undertaken on a reasonably regular basis (e.g. annually). This should include review of all related procedures, monitoring observance to the procedures, and spot checks of the database itself to identify errors and omissions.
- A comprehensive and consistent suite of assays should be collected routinely in exploration drilling. This should be formalised as a requirement across all exploration drilling. Estimation into the resource models should involve all components that may be of future interest.
- The routine collection of in-pit mapping data is encouraged as this information provides invaluable experiential knowledge to inform interpretations based on exploration data.
- Detailed scheduling and design of the sulfide ore stockpiles should be completed. Results from ongoing metallurgical testwork will assist in determining the optimal stockpiling strategy and in reconciliation success.
- Further refinement of the modelled carbonate and sulfide sulfur grades in the resource model should be completed.
- Further mapping and definition of the local and regional fault structures, alteration types, and other domains should be completed to reduce or realise geotechnical risk in the areas where these structures intersect the pit.

26.2 Mineral Reserves

Specific recommendations related to the Mineral Reserve are:

- Re-design of Çöpler pits at updated metal prices.
- Geotechnical review and study of the re-evaluation of the re-designs.
- Optimisation of float circuit POX and process operation including metallurgical testwork on Ardich and Çöpler.

- Review and monitor the stockpiling procedures and criteria to optimise the feed to the plant.
- Optimisation of the mining rates to increase gold production.
- Stockpile reconciliation and management studies.
- Geotechnical studies of Ardich.

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