Canadian National Instrument 43-101 Technical Report

Çöpler Mine
Technical Report

Erzincan Province, Turkey

Prepared by:

9635 Maroon Circle, Suite 300

Englewood, CO. 80112 USA

Report Effective Date: June 9, 2016

Dean David, FAusIMM, Amec Foster Wheeler Australia West

Richard Kiel, PE, Golder Associates Inc.

Mark Liskowich, P. Geo, SRK Consulting (Canada) Inc.

Jeff Parshley, CPG, SRK Consulting (U.S.) Inc.

John Marsden, PE, Metallurgium

Gordon Seibel, R.M. SME, Amec Foster Wheeler E&C Services Inc.

Dr. Harry Parker, PhD, R.M. SME, Amec Foster Wheeler E&C Services Inc.

Lisa Bascombe, MAIG, Mining Plus Pty Ltd

Robert Benbow, PE, Alacer Gold Corp.

Stephen Statham, PE, Alacer Gold Corp.

James Francis, MAIG, Anagold Madencilik

Sergei Smolonogov, RPGeo., Anagold Madencilik

IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report by Alacer Gold Corp (Alacer). Report contributors are Amec Foster Wheeler E&C Services Inc. and Amec Foster Wheeler Australia Pty Ltd (collectively Amec Foster Wheeler), Golder Associates Inc., SRK Consulting (U.S) Inc. and SRK Consulting (Canada) Inc. (collectively SRK), Metallurgium, and Mining Plus Pty Ltd (Mining Plus), (collectively the Consulting Engineering Firms). The quality of information, conclusions, and estimates contained within the sections prepared by the Consulting Engineering Firms is consistent with the level of effort involved in the Consulting Engineering Firms' various services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Alacer subject to the terms and conditions of its respective contracts with the Consulting Engineering Firms. Those contracts permit Alacer to file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to provincial and territorial securities legislation. Except for the purposes legislated under Canadian provincial and territorial securities law, any other use of, or reliance on, the contributor-prepared sections of this report prepared by Alacer or by the Consulting Engineering Firms by any third party is at that party's sole risk.

Mining Plus

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CERTIFICATE OF QUALIFIED PERSON

I, Lisa Bascombe of Perth Western Australia, Australia do hereby certify:

1. That I am Principal Geologist at Mining Plus Pty Ltd with a business address of 1 George Wiencke Drive, Perth Airport, Western Australia, 6105.

2. This certificate applies to the technical report titled "Çöpler Mine Technical Report, Erzincan Province, Turkey" dated June 9, 2016 with an effective date of June 9, 2016 (the "Technical Report").

3. That I am a member in good standing of the Australian Institute of Geoscientists (AIG), membership number 3520.

4. That I am a graduate of Macquarie University, New South Wales, Australia, graduating with BSc Geology in 1996.

5. That I have worked as an Exploration Geologist, Underground Mine Geologist, Senior Mine Geologist, Resource Geologist, Senior Resource Geologist, Senior Consultant Geologist and Principal Geologist for a total of 19 years.

6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that I am a "qualified person" for the purposes of NI 43-101.

7. That I, Lisa Bascombe have visited the Çöpler Project multiple times, the most recent of which was in March and April 2014 for a period of 30 days.

8. I am responsible for sections 1.6, 7.1 through 7.3, and 8 of the Technical Report.

9. I am not independent of Alacer Gold Corp. as described in Section 1.5 of NI 43-101.

10. I previously held the position of Senior Resource Geologist at Alacer and was responsible for the Çöpler Mineral Resource Estimation. I provided technical assistance to Çöpler's Exploration, Mine Geology and Mining departments. I was a Qualified Person for the Technical Report that was effective March 30, 2012.

11. I have read NI 43-101 and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

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

Signed this _7th__ day of __June____, 2016.

Original signed and sealed

____

Lisa Bascombe BSc, MAIG
Mining Plus

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CERTIFICATE OF QUALIFIED PERSON

I, Robert D. Benbow, PE do hereby certify that:

1. I am Senior Vice President Strategic Projects for Alacer Gold Corp., 9635 Maroon Circle, Suite 300, Englewood, Colorado, 80112, USA.
2. This certificate applies to the technical report titled "Canadian National Instrument 43-101 Technical Report – Çöpler Mine Technical Report" (the "Technical Report") with an effective date of April 30, 2016.
3. I am a licensed Professional Engineer in the State of Colorado (PE 0020633) and in the State of Nevada (PE 007677) in good standing. I graduated from the University of Texas at Austin with a Bachelor of Science in Civil Engineering degree in 1979 and from Regis University with a Master in Business Administration in 2002. I have worked as an Engineer and a Manager in the mining industry for a total of 43 years with relevant experience in precious and base metals in the United States and Turkey. During this time I have worked in the areas of mine design, mine scheduling, process design, mine construction and mine operations management. As per the definition of "qualified person" set out in the National Instrument 43-101, I certify that I am a "qualified person" as a result of my qualifications and experience.
4. My most recent inspection of the Çöpler mine property occurred from November 18 to 20, 2015. Prior to that time, I visited the mine on multiple occasions and served as the General Manager from September 2007 to August 2008 and as Vice President and Country Manager from August 2008 to August 2011.
5. I am responsible for Sections 1.1, 1.2, 1.17, 1.19.2, 1.20 through 1.22, 2, 4.4, 19, 21.9, 22.1 through 22.7, 23, 24.1, 25.7, 25.11 through 25.14, 26.8, 26.12 26.13 and 27 of the Technical Report.
6. I am not independent of the issuer as described in Section 1.5 of the Form 43-101-F1 Companion Policy.
7. My prior involvement with the property is limited to my term of employment with Alacer Gold Corp. and formally, Anatolia Minerals Development Corp. beginning in September 2007.
8. I have read NI 43-101 and Form 43-101-F1. The sections of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.
9. As of the aforementioned effective date, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: June 03, 2016

______"Signed and Sealed"_______

Robert D. Benbow, PE

CERTIFICATE OF QUALIFIED PERSON

I, Dean David, FAusIMM, am employed as Technical Director, Process with Amec Foster Wheeler Australia Pty Ltd (Amec Foster Wheeler).

This certificate applies to the technical report titled "Çöpler Mine Technical Report, Erzincan Province, Turkey" that has an effective date of 9 June, 2016 (the "technical report").

I am a Fellow of The Australasian Institute of Mining and Metallurgy (FAusIMM, membership number 102351). I graduated from The South Australian Institute of Technology (now University of South Australia) with a Bachelor of Applied Science in Metallurgy in 1982.

I have practiced my profession for 34 years. I have been directly involved in mineral processing research, operations, management and consulting, specializing in metallurgical testwork program design and review, comminution, classification, flotation, geometallurgy, beneficiation, dense media separation, and mine-mill optimization for projects in Australia, Asia–Pacific, Africa, and South America.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* ("NI 43–101").

I have not visited the Çöpler Mine.

I am responsible for Sections: 1.11.2, 1.15.2, 1.16.1, 1.19.1, 1.22; Section 2; Section 3; Sections 13.2 to 13.11; Section 17.2; Sections 18.1, 18.3 to 18.11; Sections 21.1 to 21.8 (excluding 21.3.1); Sections 25.4.2, 25.5, 25.10; Sections 26.5.2, 26.6 and 26.11, and Section 27 of the technical report.

I am independent of Alacer Gold Corp. as independence is described by Section 1.5 of NI 43–101.

I have no previous involvement with the Çöpler Mine.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 9 June, 2016

"Signed"

Dean David, FAusIMM.

9 June 2016 12381696DE 161 L16 Rev0

Loren Ligocki
Alacer Gold Corporation
9635 Maroon Circle, Suite 300
Englewood, Colorado 80112 USA

RE: CERTIFICATE OF AUTHOR – RICHARD E. KIEL

Dear Loren:

As a co-author of the "Çöpler Mine Technical Report, Erzincan Province, Turkey" that has a report effective date of 9 June 2016 (the "Technical Report") prepared by Alacer Gold Corp. ("Alacer"), 9635 Maroon Circle, Suite 300, Englewood, Colorado, USA, I, Richard E. Kiel, do hereby certify that:

1. I am a Principal and carried out this assignment for Golder Associates Inc., 44 Union Boulevard, Suite 300, Lakewood, Colorado 80228, USA, tel. (303) 980-0540, fax (303) 985-2080, e-mail rkiel@golder.com.
2. I hold the following academic qualifications: A. B.Sc., 1979, Geological Engineering, South Dakota School of Mines & Technology
3. I am a registered Member of the Society for Mining, Metallurgy, and Exploration (SME).
4. I am a registered professional civil engineer in California, Nevada, Colorado, and Wyoming.
5. I have worked as a civil and geological engineer in the minerals industry for 26 years.
6. I am familiar with NI 43-101 and – by reason of education, experience, and professional registration – I fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes 24 years as a consulting engineer on precious metals, base metals, and rare earth oxides, and 2 years as a geologist and engineer at an operating uranium mine. I have an additional 10 years of experience in a related industry (e.g., solid and hazardous waste management). I am qualified to prepare and review the engineering for the tailings storage facility and for geotechnical engineering aspects of the Çöpler Sulfide Project.
7. I am independent of Alacer as described in Section 1.5 of NI 43-101.
8. I have visited the property and inspected the Çöpler Project Area numerous times since 2012, the latest site visit being from 17 through 20 April 2016.
9. As of the date of this certificate, 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 this report not misleading.
10. I am responsible for the preparation of Sections 1.16.2, 2.5, 7.5, 16.3, 16.5.1, 18.2, 18.13-18.16, 21.3.1, 21.9.4, 25.6, 26.7, and 27 of the Technical Report.

Sincerely,

GOLDER ASSOCIATES INC.

Original Signed and Sealed

Richard E. Kiel, PE
Senior Geological Engineer

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I, Mark Liskowich, of Saskatoon, Saskatchewan, Canada do hereby certify:

1. That I am a professional Geologist employed as a Principal Consultant with SRK Consulting (Canada) Inc. at 205, 2100 Airport Drive, Saskatoon, Saskatchewan.
2. This certificate applies to the Technical Report titled "Canadian National Instrument 43-101 Çöpler Mine Technical Report, Erzincan Province, Turkey" dated June 9, 2016 (the "Technical Report").
3. That I am a member of the Association of Professional Engineers and Geoscientists of Saskatchewan.
4. That I am a graduate of the University of Regina. I graduated with a B.Sc (geology) degree in May 1989.
5. That I have practiced my profession within the mineral exploration, mining industry since 1989. I have been directly involved, professionally, in the environmental and social management of mineral exploration and mining projects covering a wide range of commodities since 1992 with both the public and private sector. My areas of expertise are environmental management, environmental auditing, project permitting, licensing, public and regulatory consultation.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that I am a "qualified person" for the purposes of NI 43-101.
7. That I, Mark Liskowich, visited the Copler project on September 21 and 22, 2010.
8. I am responsible for sections 1.3-1.5, 1.18, 4, 5, 6.1, 6.2, 201. To 20.12, 25.8 and 26.9 of the Technical Report.
9. I am independent of Alacer as described in Section 1.4 of NI 43-101.
10. I have had no prior involvement with the Çöpler Project.
11. I have read NI 43-101 and this Technical Report has been prepared in compliance with that instrument.
12. That as of the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed this 06 day of June 2016.

"original signed"

Mark W. Liskowich, P.Geo

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Phoenix, AZ 85028 E-mail: john@metallurgium.com

CERTIFICATE OF QUALIFIED PERSON

I, John Marsden, P. E., do hereby certify that:

1. I am President and Manager of John O. Marsden LLC, dba Metallurgium, 10645 N. Tatum Boulevard, Suite 200-550, Phoenix, Arizona, 85028, USA..
2. This certificate applies to the technical report titled "Canadian National Instrument 43-101 Technical Report – Çöpler Sulfide Expansion Project Feasibility Update" (the "Technical Report") with an effective date of June 9, 2016.
3. I am a Registered Member of the Society for Mining, Metallurgy & Exploration, Inc. (Registered Member Number 2029830) in good standing. I graduated from the Royal School of Mines, Imperial College of Science and Technology, University of London with a Bachelor of Science Engineering (BSc. Eng. Hons) degree in Mineral Technology in 1982. I have worked as a Metallurgical Engineer for a total of 34 years with relevant experience in mineral processing, extractive metallurgy and mineral technology for precious and base metals projects in the United States, Canada, South Africa, Zambia, Botswana, Chile, Peru, Brazil, Mexico, Australia, Indonesia, Democratic Republic of Congo, Cameroon, and Turkey. As per the definition of "qualified person" set out in the National Instrument 43-101, I certify that I am a "qualified person" as a result of my qualifications and experience.
4. My most recent inspection of the Çöpler mine property occurred from March 24 to 27, 2012.
5. I am responsible for Sections 1.11.1, 1.15.1, 2.5, 13.1, 17.1, 18.12, 25.4.1, and 26.5.1 of the Technical Report.
6. I am independent of the issuer as described in Section 1.5 of the NI 43-101.
7. My prior involvement with the property is limited to third party consulting services provided to Alacer Gold Corp. (previously Anatolia Minerals) as a Metallurgical Engineer beginning in July 2010.
8. I have read NI 43-101 and Form 43-101-F1. The sections of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.
9. As of the aforementioned effective date, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: June 5th, 2016

"Original signed and sealed"

John O. Marsden

CERTIFICATE OF QUALIFIED PERSON

I, Dr. Harry Parker, RM SME, am employed as a Consulting Geologist and Geostatistician with Amec Foster Wheeler E&C Services Inc. ("Amec Foster Wheeler").

This certificate applies to the technical report titled "Çöpler Mine Technical Report, Erzincan Province, Turkey" that has an effective date of 9 June, 2016 (the "technical report").

I am a Fellow of the Australian Institute of Mining and Metallurgy (FAusIMM #113051), and a Registered Member of the Society for Mining, Metallurgy and Exploration (#2460450). I am a Professional Geologist in California (#3402), in Arizona (#13317), and in Minnesota (#49606).

I graduated from Stanford University with BSc and PhD degrees in Geology in 1967 and 1975 respectively. I graduated from Harvard University in 1969 with an AM degree in Geology. I graduated from Stanford University with an MSc degree in Statistics in 1974.

I have practiced my profession for 49 years during which time I have been involved in the estimation of mineral resources and mineral reserves for various gold exploration projects and operating gold mines associated with intrusions. These include Colomac, NWT; Fort Knox, AK, Silangan, Philippines; Cripple Creek, Colorado; Lihir, PNG; Porgera, PNG.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* ("NI 43–101").

I visited the Çöpler Mine from May 5 to 11, 2014.

I am responsible for Sections 1.10, 1.12; Section 2.5; Section 3; Section 12; Section 14; Section 25.1; Sections 26.1 and 26.2, and Section 27 of the technical report.

I am independent of Alacer Gold Corp. as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Çöpler Mine during the preparation of this technical report, and I have previously co-authored the following technical reports on the Çöpler Mine:

- Bohling, R., Kiel, R., Armstrong, D., Liskowich, M., Parshley, J., Swanson, B., Seibel, G., Parker, H.M., Bascombe, L., 2014: Çöpler Sulfide Expansion Project Feasibility Study, Erzincan Province, Turkey: July 29, 2014.

- Bohling, R., Kiel, R., Birch, R.G., Liskowich, P.Geo., Parshley, J., Marsden, J., Seibel, G., Parker, H.M., Bascombe, L., Benbow, R., Statham, S., Francis, J., and Khoury, C., 2015: Çöpler Sulfide Expansion Project Feasibility Update Erzincan Province, Turkey, March 27, 2015.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 9 June, 2016

"Signed and stamped"

Dr Harry M Parker, RM SME.

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CERTIFICATE OF QUALIFIED PERSON

I, Jeffrey Vaughan Parshley, CPG do hereby certify that:

1. I am a Corporate Consultant for SRK Consulting (U.S.), Inc., 5250 Neil Road, Suite 300, Reno, NV, USA, 89502.
2. This certificate applies to the technical report titled “Çöpler Mine Technical Report, Erzincan Province, Turkey with an Effective Date of 9 June, 2016 (the “Technical Report”).
3. I graduated with a degree in B.A. in Geology from Dartmouth College in 1980. I am a Certified Professional Geologist of the American Institute of Professional Geologists. I have worked as a Geologist for a total of 36 years since my graduation from university. My relevant experience includes more than 25 years of mine permitting, closure and environmental studies in the U.S. and internationally.
4. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I have not personally visited the Çöpler Project site but relied on a site visit completed by Mr. Patric Lassiter, P.G., a co-author of the Technical Report.
6. I am responsible for the preparation of Sections 2.6, 20.13 through 20.18, 25.9 and 26.10 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is to have reviewed current the project closure liabilities each year since 2012.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th Day of June, 2016.

“Original Signed and Sealed”

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Jeffrey Vaughan Parshley

CERTIFICATE OF QUALIFIED PERSON

I, Gordon Seibel, RM SME, am employed as a Principal Geologist with Amec Foster Wheeler E&C Services Inc.

This certificate applies to the technical report titled "Çöpler Mine Technical Report, Erzincan Province, Turkey" that has an effective date of 9 June, 2016 (the "technical report").

I am a Registered Member of the Society for Mining, Metallurgy and Exploration (#2894840). I graduated from the University of Colorado with a Bachelor of Arts degree in Geology in 1980. In addition, I obtained a Masters of Science degree in Geology from Colorado State University in 1991.

I have practiced my profession for 33 years, during which time I have been directly involved in the development of resource models and mineral resource estimation for mineral projects in North America, South America, Africa, and Australia since 1991. I have previously estimated or audited gold Mineral Resources for Cripple Creek and Victor Gold Mining Company, Colorado; Spring Valley, Nevada; Soledad Mountain, California; Midas, Nevada; Callie, NT Australia; Conga, Peru; Donlin Creek, Alaska; Leeville, Nevada; Subika, Ghana and Ahafo North, Ghana

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* ("NI 43–101").

I visited the Çöpler Mine from May 5 to 11, 2014 and from June 6 to 10, 2015.

I am responsible for Sections 1.10, 1.12; Section 2.5; Section 3; Section 12; Section 14; Section 25.1; Sections 26.1 and 26.2, and Section 27 of the technical report.

I am independent of Alacer Gold Corp. as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Çöpler Mine during the preparation of this technical report, and I have previously co-authored the following technical reports on the Çöpler Mine:

- Bohling, R., Kiel, R., Armstrong, D., Liskowich, M., Parshley, J., Swanson, B., Seibel, G., Parker, H.M., Bascombe, L., 2014: Çöpler Sulfide Expansion Project Feasibility Study, Erzincan Province, Turkey: July 29, 2014.

- Bohling, R., Kiel, R., Birch, R.G., Liskowich, P.Geo., Parshley, J., Marsden, J., Seibel, G., Parker, H.M., Bascombe, L., Benbow, R., Statham, S., Francis, J., and Khoury, C., 2015: Çöpler Sulfide Expansion Project Feasibility Update Erzincan Province, Turkey, March 27, 2015.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 6 June, 2016

"Signed and stamped"

Gordon Seibel, RM SME.

Amec Foster Wheeler E&C Services, Inc.
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CERTIFICATE OF QUALIFIED PERSON

I, Sergei Smolonogov, BAppSc (Geology) do hereby certify that:

1. I am Geology Manager for Anagold Madencilik, Asagi Ovecler 8. Cd. 1332. Sk., No: 8/8 06460 Dikmen, Cankaya, Ankara Turkey.
2. This certificate applies to the technical report titled "Canadian National Instrument 43-101 Technical Report – Çöpler Mine Technical Report, Erzincan Province, Turkey" (the "Technical Report") with an effective date of June 9, 2016.
3. I am a Professional Geologist with 27 years' experience in the gold and base metals industry and a Registered Professional Member of the Australian Institute of Geoscientists (AIG #2456). I graduated from the University of Technology, Sydney, with a Bachelor of Applied Science (Hons.) Geology Degree in 1989. I have worked in exploration, resource and reserve development and mining of several mineralisation styles inclusive of epithermal gold deposits as developed at the Çöpler Mine and within the Çöpler District.
4. As per the definition of "qualified person" set out in the National Instrument 43-101, I certify that I am a "qualified person" as a result of my academic qualifications, recognized registered professional status (RPGeo#10174) and relevant work experience.
5. As Geology Manager I have worked at the Çöpler Mine and within the Çöpler District from May 1st, 2014.
6. I am responsible for Sections 1.7 to 1.9, 2.5, 7.4, 9, 10 and 11 of the Technical Report.
7. I am not independent of the issuer as described in Section 1.5 of the NI 43-101.
8. My involvement with the property is limited to my term of employment with Anagold Madencilik, as Geology Manager beginning May 1st, 2014.
9. I have read NI 43-101 and Form 43-101-F1. The sections of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned effective date, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: June 7th, 2016

"*Original signed and sealed*"

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Sergei Smolonogov, BAppSc (Geology), RPGeo, MAIG

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CERTIFICATE OF QUALIFIED PERSON

I, Stephen Statham, PE do hereby certify that:

1. I am Mining Services Manager for Alacer Gold Corp, 9635 Maroon Circle, Suite 300, Englewood, Colorado, 80112, USA.
2. This certificate applies to the technical report titled "Çöpler Mine Technical Report, Erzincan Province, Turkey" (the "Technical Report") with an effective date of June 9, 2016.
3. I am a licensed Professional Engineer in the State of Colorado (PE.0048263) in good standing. I am also a Registered Member of the Society of Mining, Metallurgy & Exploration (RM# 4140907) in good standing. I graduated from Virginia Polytechnic Institute and State University in 2006 with a Bachelor of Science in Mining and Minerals Engineering degree. I have worked as a Mining Engineer for a total of 10 years with relevant experience in mine planning for precious and base metals projects in the United States, Mexico, Australia, Indonesia, Democratic Republic of Congo, Ghana, and Turkey. During this time, I have worked in the areas of mine design, mine scheduling, Mineral Reserve estimation, and mine operations management. As per the definition of "qualified person" set out in the National Instrument 43-101, I certify that I am a "qualified person" as a result of my qualifications and experience.
4. My most recent inspection of the Çöpler mine property occurred from March 21 to April 7, 2016.
5. I am responsible for Sections 1.13, 1.14, 2.5, 6.3, 15, 16 (excluding 16.3 and 16.5.1), 25.2, 25.3, 26.3, and 26.4 of the Technical Report.
6. I am not independent of the issuer as described in Section 1.5 of the NI 43-101.
7. My prior involvement with the property is limited to my term of employment with Alacer Gold Corp. beginning in May 2013.
8. I have read NI 43-101 and Form 43-101-F1. The sections of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.
9. As of the aforementioned effective date, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: June 7th, 2016

"*Original signed and sealed*"

____

Stephen Statham, PE

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LIST OF ABBREVIATIONS

Above Mean Sea Level	amsl
Acidity	pH
Alternating current	AC
Ampere	amp
Atmosphere	atm
Average	avg
Barrel	bbl
Brake horsepower	bhp
Centimeter	cm
Centipoise	cP
Cubic meter	m3
Cubic meter per second	m^3/s
Cubic meter per hour	m^3/hr
Day	day
Days per week	day/week
Days per year (annum)	day/year
Decibel	dB
Degree	°
Degrees Celsius	°C
Direct current	DC
Dollar (US)	$
Euro	€
Gram	g
Grams per liter	g/L
Grams per tonne	g/t
Greater than	>
Hertz	Hz
Horsepower	hp
Hour	hr
Hours per day	hr/day
Hours per week	hr/week
Hours per year	hr/yr
Inside Diameter	ID
Joule	J
Kilo (thousand)	k
Kilogram	kg
Kilogram per cubic meter	kg/m3
Kilogram per hour	kg/hr
Kilogram per second	kg/s
Kilogram per meter square per second	kg/m2/s
Kilometer	km
Kilometer per hour	km/hr
Kilojoule	kJ
Kilopascal	kPa
Kilovolt	kV
Kilovolt-ampere	kVA
Kilowatt	kW
Kilowatt hour	kWhr
Kilowatt hours per short ton	kWhr/t
Kilowatt hours per year	kWhr/yr
Less than	<
Liter	L
Liters per minute	L/min
Mass percent	m%
Mega (million)	M
Megabyte	MB
Megavolt-ampere	MVA
Megawatt	MW
Meter	m
Meter per second	m/s
Meter squared	m^2
Micrometer (micron)	µm
Milligram	mg
Milligrams per liter	mg/L
Milliliter	mL
Millimeter	mm
Million	million or M
Million tonnes	Mt
Million tonnes per annum	Mtpa
Minute (plane angle)	'
Minute (time)	min
Month	month
Newton meter	Nm
Normal	N
Normal cubic meters per hour	N m^3/hr
Parts per billion	ppb
Parts per million	ppm
Pascal	Pa
Pascal second	Pa s
Percent	%
Revolutions per minute	rpm
Second (time)	sec
Selective Mining Unit	SMU
Specific gravity	SG
Standard deviation	SD
Square meter	m2
Thousand tonnes	kt
Tonne (metric)	t
Tonnes per day	tpd
Tonnes per hour	tph
Tonnes per year	tpy
Total dissolved solids	TDS
Total dynamic head	TDH
Total suspended solids	TSS
Troy oz	oz
Troy oz per short ton	opt
Turkish Lira	₤
Volume percent	v/v %
Volume/volume	v/v
Volt	V
Watt	W
Week	week
Weight percent	wt %
Weight/weight	w/w
Yard	yd
Year (annum)	yr

International System of Units (SI) PREFIXES

Power	Prefix	Symbol	Decimal Equivalent
10^{24}	yott-	Y	1,000,000,000,000,000,000,000,000
10^{21}	zeta-	Z	1,000,000,000,000,000,000,000
10^{18}	exa-	E	1,000,000,000,000,000,000
10^{15}	peta-	P	1,000,000,000,000,000
10^{12}	tera-	T	1,000,000,000,000
10^{9}	giga-	G	1,000,000,000
10^{6}	mega-	M	1,000,000
10^{3}	kilo-	k	1,000
10^{2}	hector-	h	100
10^{2}	deca-	da	10
10^{0}			1
10^{-1}	deci-	d	0.1
10^{-2}	centi-	c	0.01
10^{-3}	milli-	m	0.001
10^{-6}	micro-	µ	0.000 001
10^{-9}	nano-	n	0.000 000 001
10^{-12}	pico-	p	0.000 000 000 001
10^{-15}	femto-	f	0.000 000 000 000 001
10^{-18}	atto-	a	0.000 000 000 000 000 001
10^{-21}	zepto-	z	0.000 000 000 000 000 000 001
10^{-24}	yocto-	y	0.000 000 000 000 000 000 000 001

1.0 SUMMARY

1.1 Introduction and Scope of Work

Alacer Gold Corp. (Alacer or the Company) has prepared a Technical Report (the Report) on the Çöpler Mine and Çöpler Sulfide Expansion Project (collectively the Project), located in Turkey.

Alacer, listed on the Toronto Stock Exchange (TSX) and the Australian Stock Exchange (ASX) is a mid-tier gold producer and explorer with assets in Turkey. Alacer was formed following the merger of Anatolia Minerals Development Limited (Anatolia) and Avoca Resources Limited (Avoca) in February 2011.

The currently-operating Çöpler Mine is owned and operated by Anagold Madencilik Sanayi ve Ticaret Anonim Şirketi (Anagold). Alacer controls 80% of the shares of Anagold and Lidya Madencilik Sanayi ve Ticaret A.Ş. (Lidya), formerly Çalık Holdings A.Ş., controls 20%. The same ownership percentage interests apply to the Çöpler Sulfide Expansion Project (the Sulfide Expansion Project). Exploration tenures surrounding the Project are subject to joint venture agreements between Alacer and Lidya that have varying interest proportions. As noted earlier, Alacer Gold currently has an 80% stake in Anagold, and has a 50% stake in Kartaltepe Madencilik (Kartaltepe),

Co-contributors to the Report include Qualified Persons (QPs) from, in alphabetical order, Alacer, Amec Foster Wheeler E&C Services Inc., Amec Foster Wheeler Australia Pty Ltd (collectively Amec Foster Wheeler), Anagold Madencilik, Golder Associates Inc. (Golder Associates), John O. Marsden LLC (Metallurgium), Mining Plus Pty Ltd (Mining Plus), and SRK Consulting (Canada) Inc. and SRK Consulting (US) Inc. (collectively SRK).

Alacer completed a technical report titled *Çöpler Sulfide Expansion Project Prefeasibility Study* in May 2011. The prefeasibility study (PFS) found that the project was feasible and could be advanced to the feasibility study (FS) stage. A technical report titled *Çöpler Sulfide Expansion Project Definitive Feasibility Study, Revision B* was issued in August 2014, and found the project to be technically and financially feasible. In March 2015, a technical report tilted *Çöpler Sulfide Expansion Project Feasibility Update* was issued, updating the Mineral Resources, Mineral Reserves and other project-specific parameters. The later document provided the basis of a decision to advance the sulfide project to detailed engineering which is currently ongoing.

The intent of this Report is to update the Mineral Resource and Mineral Reserve estimates and the Sulfide Expansion Project status from the 2015 technical report. A material change in the Inferred Mineral Resource estimate has occurred since the year-end 2015 resource estimates, as initially published in Alacer's Management's Discussion and Analysis, dated February 8, 2016. This report was compiled to support the updated Mineral Resource estimates that were detailed in Alacer's news release dated 12 May, 2016, entitled *Alacer Gold Announces Çőpler Sulfide Project Approval* and to provide updated information on the current detailed engineering phase.

Alacer engaged Amec Foster Wheeler to conduct detailed engineering for the sulfide ore processing plant, and to provide procurement and construction management services. Golder Associates performed the design for the tailings storage facility (TSF) and is completing detailed design on site geotechnical and construction services. SRK Consulting, Metallurgium and Mining Plus are providing technical expertise specific to this Report.

The Mineral Resource estimate described in this Report are based on additional drilling conducted in 2015 and a new resource block model calibrated to production data.

Sulfide ore is currently being stockpiled for processing in the new pressure oxidation (POX) facilities currently scheduled to be constructed starting in mid-2016, and brought into production in the third quarter of 2018.

All units in this study are according to International Systems (SI) of units unless otherwise noted. All costs are in United States dollars and are based on fourth quarter (Q4) 2015 dollars unless otherwise noted.

The word “ore” in this report describes the mineralization to be delivered from the mine to the processing facilities and is used for material that has been estimated as Mineral Reserves as defined by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) 2014 Definition Standards.

1.2 Key Outcomes

Key outcomes from the Feasibility Study are summarized in Table 1-3, Table 1-4, and Table 1-5, included in later sections of this Summary.

- Measured and Indicated Mineral Resources for the open pit totals 100.4 Mt grading 1.93 g/t Au. Proven and Probable Mineral Reserves total 58.0 Mt grading 2.25 g/t Au.
- Planned POX process rate is 1.9 to 2.2 Mt/a, which will extend the mine life of Çöpler to 22 years with the operation forecasted to end in 2037.
- Commissioning of the sulfide process plant is scheduled to be completed by the end of second quarter 2018, with first gold pour in the third quarter 2018. The schedule allows for an 18-month ramp-up to achieve initial design capacity of 1.9 Mt through-put rate per year.

The Sulfide Expansion Project shows the following financials:

- Net Present Value (NPV) of US $728M
- An Internal Rate of Return (IRR) of 19.2%
- Payback period of 3.0 years.

1.3 Property Description and Location

The Çöpler Mine 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 center, İliç, (approximate population 2,600), is located about 6 km east of the Çöpler Mine.

Figure 1-1 Project Location Map

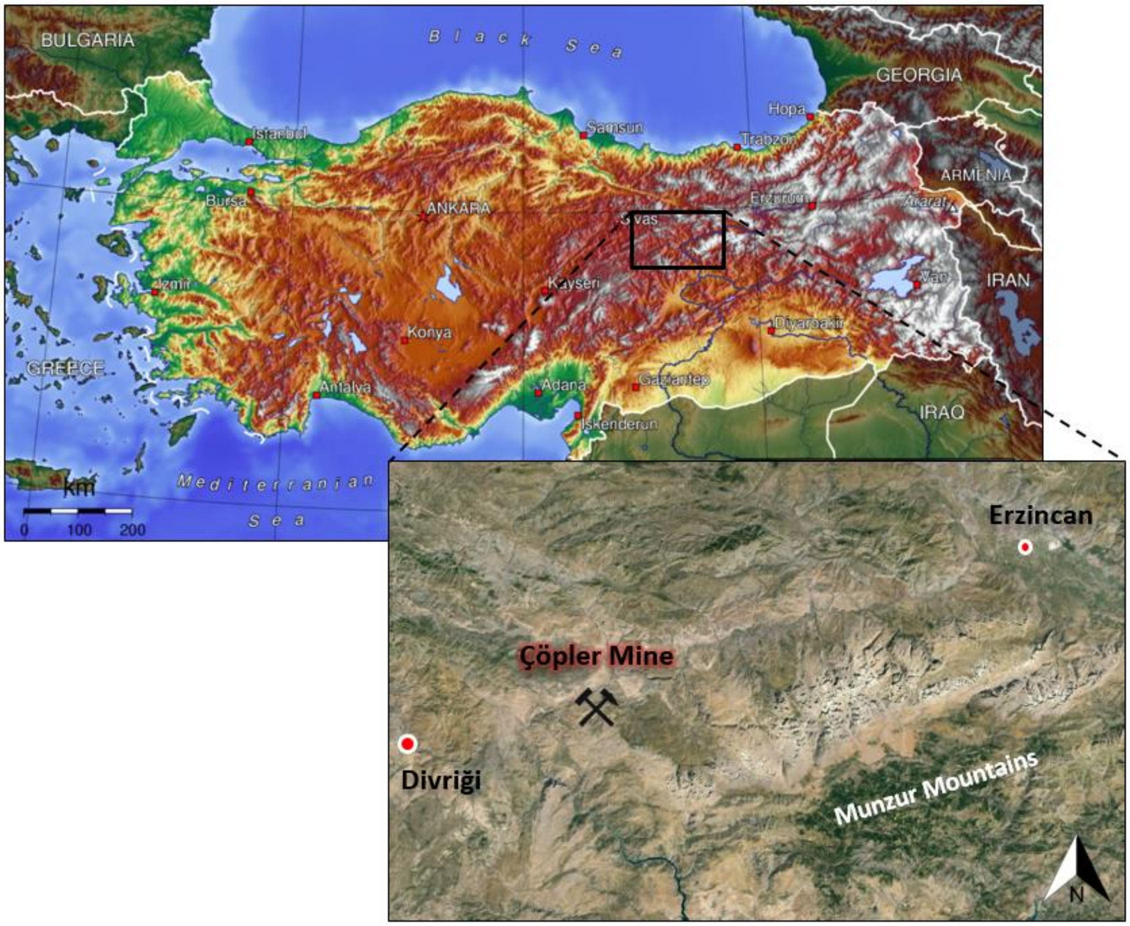


Figure prepared by Alacer, 2016.

There are seven granted licenses covering a combined area of about 16,573 ha. Mineral title is held in the name of Anagold.

Alacer holds sufficient surface rights to allow continued operation of the heap leach mining operation and has obtained the required surface rights to allow construction and operation of the Sulfide Expansion Project.

1.4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The mine is accessible by a maintained paved highway to the intersection of the mine access road, approximately 2 km from the mine. The mine access road is a well-maintained gravel road. The mine access road will be realigned as part of the Çöpler Sulfide Expansion Project.

The Project area is located in the Eastern Anatolia geographical district of Turkey. The climate is typically continental with wet, cold winters and dry, hot summers. The Çöpler mining area is accessed from the main paved highway between Erzincan and Kemaliye.

Mining operations are currently conducted year-round, and will continue to be a year-round activity when the Çöpler Sulfide Expansion Project is in operation.

1.5 History

The Turkish Geological Survey (MTA) carried out regional exploration work in the early 1960s that was predominately confined to geological mapping. During 1964, a local Turkish company started manganese mining, which continued until closing in 1973. Unimangan acquired the property in January 1979 and restarted manganese production, continuing until 1992.

In September 1998, Alacer's predecessor, Anatolia, identified several porphyry-style gold-copper prospects in east-central Turkey and applied for exploration licenses for these prospects. During this work, Anatolia identified a prospect in the Çöpler basin. This prospect and the supporting work was the basis for a joint venture agreement for exploration with Rio Tinto.

In January 2004, Anatolia acquired the interests of Rio Tinto and Unimangan. The property was under sole control of Anatolia until the joint venture agreement between Anatolia and Lydia was executed in August 2009.

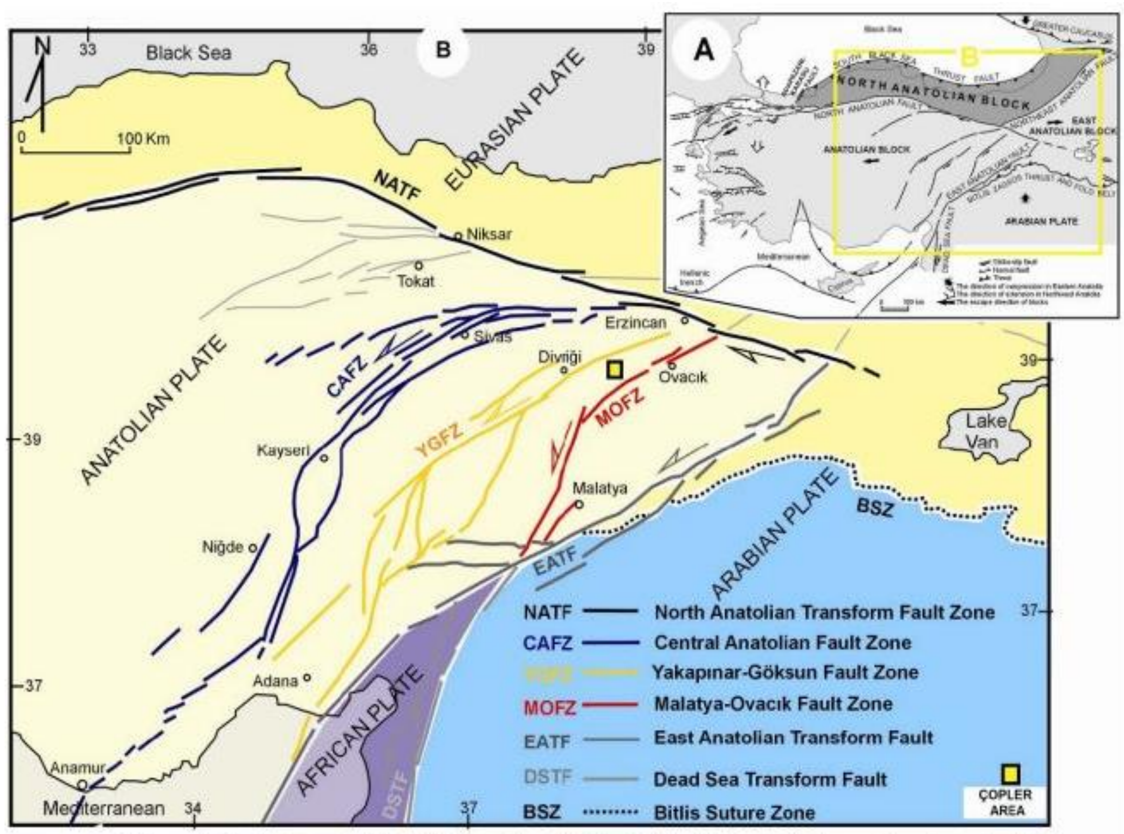
Anatolia merged with Avoca Resources Limited, an Australian company, to form Alacer Gold Corporation in February 2011. In October 2013, Alacer sold its Australian Business Unit.

Although the company will be referred to as Alacer, it may have been Anatolia at certain times referenced in the report.

1.6 Geological Setting and Mineralization

The Project is located near the north margin of a complex collision zone lying 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. Figure 1-2 illustrates the broad structural setting of the Anatolia region of Turkey. The Çöpler Mine is located between Divriği and Ovacık.

Figure 1-2 Structural Setting of Anatolia



Major neotectonic elements of Anatolia (from Elmas 2003), **B:** Simplified neotectonic map showing major tectonic escape-induced structures in eastern half of Turkey and adjacent areas (from Koçyiğit and Erol, 2001)

At Çöpler, gold, silver, and copper mineralization of economic interest occurs in a porphyry-related epithermal deposit, with most of the gold mineralization concentrated in three zones. The mineralization at Çöpler is present in five different forms:

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

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

1.7 Exploration

The primary exploration effort at Çöpler was completed by:

- Anatolia during 1998 and 1999 prior to entering into a joint venture with Rio Tinto.
- A joint venture between Anatolia and Rio Tinto from 2000 to 2004.
- Anatolia from 2004 to 2010.
- Anagold from February 2011 to date.

Initial exploration at Çöpler was directed at evaluating the economic potential for recovering gold by either heap leaching or conventional milling techniques from near-surface oxide mineralization.

A drilling program specifically designed to investigate the sulfides was commenced late in 2009 and completed early in 2010. Infill resource drilling has continued at Çöpler in an attempt to define extensions to the current resource and to collect additional information within the current resource boundary. Drill testing continues to date in order to better define both the oxide and sulfide portions of the deposit. In 2013, drilling occurred primarily in the western and northern portions of the Çöpler deposit, and in 2014 drilling focused on verification of existing mineralization through a twin hole program. Drilling in 2015 provided data coverage at depth in the Manganese pit, in-fill drilling in the Main pit and initial testing of low sulfur mineralization below the oxidation boundary. The majority of the drill meters in 2016 was on near-mine exploration projects. Drilling programs in 2016 also covered definition of the sulfide stockpile and testing of leachable material in the Main pit.

Exploration activities across the Yakuplu East, Yakuplu Southeast, Yakuplu North, Yakuplu Main and Bayramdere prospects have included geological mapping, geochemical sampling, geophysical surveys, and drilling.

Surficial mapping and geochemical soil sampling has continued in the wider district over the life of the Project.

1.8 Drilling

A significant amount of drilling has been undertaken at the Project in order to locate, test and define the mineralization and its extents, and to test exploration targets. A total of 1,125 reverse circulation (RC) drill holes (126.5 km), 734 diamond (DD) core holes (171.5 km) and 98 holes with mixed drilling methods have provided more than 297 km of drill sample in the vicinity of the Çöpler pit. Near-mine drilling on the exploration prospects includes 507 drill holes (55.2 km) of both RC and DD through April 2016.

The current drill hole spacing at surface is a nominal 50 m by 50 m; however, infill drilling to 25 m by 25 m has occurred over the majority of the drilled areas.

1.9 Sampling Method, Approach and Analyses

From 2004 to late 2012, samples were prepared at ALS İzmir, Turkey and analyzed at ALS Vancouver, Canada. From late 2012 to 2014, samples were prepared and analyzed at ALS İzmir, Turkey. Samples in 2015 and 2016 were prepared and analyzed at the SGS Laboratory in Ankara, Turkey.

SGS Ankara 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.

SGS and ALS are specialist analytical testing service companies that are independent of Alacer.

Samples provided to SGS in 2015 were analyzed for gold using SGS method FAA303 which uses a 30 g pulp for fire assay and measurement by atomic absorption spectroscopy (AAS). The gold detection limits are 0.01 g/t to 100 g/t. SGS method FAG303 using a gravimetric finish was also included when the gold content was found to be above 3 g/t.

From 2004 to end of 2014, samples sent to ALS were analyzed for gold using the ALS method Au-AA25 that comprises a fire assay of a 30 g pulp sample followed by measurement of gold grades using AAS. The lower and upper gold detection limits are 0.01 g/t and 100 g/t respectively. Samples with returned gold grades above the upper detection limit are re-analyzed using the gravimetric method Au-GRA21.

Analysis of 33 other elements is accomplished through the ALS method ME-ICP61 which involves a four-acid (perchloric, nitric, hydrofluoric and hydrochloric acid) sample digest followed by measurement of element grades by inductively coupled plasma – atomic emission spectroscopy (ICP-AES). Silver, copper, lead, zinc and manganese are among the 33 elements analyzed by this method.

1.10 Data Verification

Data verification was conducted during compilation of technical reports on the Project from 2003 to 2012. None of the verification programs identified material issues with the supporting data.

In 2014, Amec Foster Wheeler conducted a database audit and review of available quality assurance and quality control (QA/QC) data to ensure the data were of sufficient quality to support resource estimation. The database audit covered data collected from 2000 to December 2013.

Amec Foster Wheeler was unable to validate collar and down-hole survey data because Alacer was unable to provide copies of the original documents. Scans of original drill logs (lithology, RQD and bulk density) were compared to values contained in the database. Rio Tinto operated a drill program from 2000 to 2003; samples from this program were submitted to OMAC Laboratories Limited (OMAC), a certified laboratory that was independent of Rio Tinto. Assay results from early drill holes (2000 to 2003) assayed by OMAC were unable to be obtained at the time of the audit. OMAC drilling represents 6% of the total meters drilled at the time of the database extract for the resource estimate. Amec Foster Wheeler used statistical methods to validate the 2000 to 2003 data against the ALS data and found the data to be comparable. Assay results from 2004 to 2013 were obtained from ALS. Amec Foster Wheeler electronically compared assay results (gold, copper, silver, arsenic, iron, manganese, sulfur and zinc) to the database.

A set of witness samples were collected in 2014 from blast hole cuttings that were submitted to both the Çöpler site laboratory and to ALS. The mean of ALS results is 8% higher than the mean of the results provided by the Çöpler site laboratory. If the result from one high-grade sample (above 4 g/t gold) is removed from the comparison, the mean ALS gold grade is only 3% higher than the mine site laboratory. In Amec Foster Wheeler's opinion this is acceptable agreement between the two laboratories.

In 2015, Amec Foster Wheeler reviewed the Çöpler deposit database as of July 15, 2015 in order to verify the data were of sufficient quality to support Mineral Resource estimation of gold, copper and silver for the Çöpler deposit. This audit focused on the 121 drill holes totalling 12,959.8 m completed since the previous audit.

Amec Foster Wheeler validated collar and downhole survey data against the original documents. Amec Foster Wheeler compared original drill logs for lithology and rock quality designation (RQD) to values contained in the database. Density data were supplied on a separate Excel spreadsheet and were compared to the original logs. Assay results from 2014 and 2015 were obtained directly from ALS and SGS. Amec Foster Wheeler electronically compared assay results (gold, copper, silver, iron, manganese, sulfur) to the database. Available QA/QC data were evaluated to ensure the assay data are suitable to support resource estimation. A list of samples and data to be reviewed and checked was forwarded to Alacer as a result of the audits. A number of recommendations were also made, and included:

- As silver contributes 0.4% to the overall economics, Amec Foster Wheeler recommends adding a single silver certified reference material (CRM) within the expected grade range.
- An additional CRM to monitor sulfur assays at the sulfur grade used to define the oxide/sulfide boundary should be considered.

In Amec Foster Wheeler's opinion, the data contained in the Alacer database is of sufficient quality to support Mineral Resource estimation.

1.11 Metallurgical Testwork

1.11.1 Heap Leaching Testwork

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

The heap leaching facilities were commissioned in late 2010 and have operated continuously since that time.

Heap leaching process gold recovery assumptions have been updated to reflect actual performance of the operation between September 2010 and December 2015. The gold recovery assumptions for oxide ore are summarized in Table 1-1. Material that was previously considered within a transition zone adjacent to the oxidation boundary is not currently considered to be suitable for heap leach feed.

Table 1-1 Gold Recovery Assumptions for Heap Leaching of Material in the Çöpler Oxide Zone

Oxide Ore Type	Manganese	Marble	Main	Main East	Main West	West
Marble	78.4	75.7	68.6	78.4	75.7	75.7
Metasediments	66.8	66.8	66.8	66.8	66.8	66.8
Gossan	71.2	65.1	71.2	71.2	65.1	65.1
Diorite	71.2	62.3	71.2	71.2	62.3	62.3
Mn Diorite	71.2	62.3	71.2	71.2	62.3	62.3

1) *Table units are recovery percentages.*

Sulfide material (containing ≥ 2% sulfide sulfur content) is not suitable for treatment by the heap leaching process, and therefore no gold recovery assumptions are provided for this material.

The original gold recovery assumptions have been updated during operations.

The recovery assumptions listed in Table 1-1 consider heap leaching of ore crushed to 80% passing 12.5 mm, agglomerated with lime and moisture to achieve consistently high quality agglomerates, and placed on a lined heap leach pad for treatment. The general process flowsheet is shown in Figure 13-1.

The gold recovery assumptions provided in Table 1-1 represent a positive adjustment of 1.0476 applied to the original (2008) assumptions, reflecting the results of additional metallurgical testing and the results of the heap leach production model performance and calibration.

1.11.2 POX Testwork

1.11.2.1 Historical Testwork

RDi performed several sulfide processing scoping level investigations for Alacer in the period 2006 to 2009. SGS Lakefield Research Limited (SGS) conducted a two-phase program on sulfide samples in 2009 and 2010 to support the pre-feasibility study (PFS) completed by Samuel Engineering (Samuel, 2011). A quantitative evaluation of minerals by scanning electron microscopy (QEMScan) mineralogy study on six oxide and three sulfide samples was performed by AMMTEC Limited (AMMTEC) in December 2008.

The historical work completed at both RDi and SGS evaluated typical sulfide processing options including direct cyanidation, flotation, cyanidation of flotation concentrates and flotation tailings, POX coupled with cyanidation, and roasting coupled with cyanidation.

Diagnostic leaching testwork carried out by RDi indicated that only 11% to 30% of the gold content in the sulfide samples is amenable to whole-ore direct cyanidation. It was evident that 60% to 80% of the gold content was intimately associated with sulfide minerals, and it would only be possible to release this gold for recovery by cyanidation using a pyrite oxidation step.

The RDi scoping studies showed the most effective pre-treatment method for the ore was POX, which promised greater than 90% gold extraction. Flotation of pyrite (and minor chalcopyrite) recovered a large amount of the gold, but the concentrates were low grade with relatively high mass pulls, and gold recovery was low. Testwork also found flotation concentrates and tailings did not leach well using cyanide, even after being finely ground.

The scoping test program on new samples by SGS in 2009 sought to verify the findings of RDi, and begin to develop the metallurgical flowsheet. Results from the flotation testwork were consistent with the RDi tests, demonstrating that it was not feasible to make either a saleable copper concentrate or saleable sulfide concentrate.

The refractory nature of the Çöpler sulfide mineralization to direct cyanidation was confirmed. POX testing successfully oxidized 90% to 99% of the sulfide content and provided gold extractions consistently in the range of 90% to 96%. Roasting was able to oxidize the contained sulfide minerals; however, gold was not fully liberated for cyanidation, yielding gold cyanidation extractions around 79%.

SGS completed a second phase of metallurgical testing in 2010, to support a PFS using POX followed by cyanidation. The flowsheet continued to achieve superior gold extractions when compared to alternative treatment options. Included in the evaluation were ultra-fine grinding followed by direct cyanidation and Albion oxidation followed by cyanidation.

SGS demonstrated that the SO_2/air process destroyed cyanide remaining in POX leach residues. Consistent with previous testwork, limestone neutralized the POX solution phase and, subsequently, sodium hydrosulfide (NaHS) successfully precipitated copper.

1.11.2.2 Mineralogy

In December 2008, Alacer had QEMScan precious metals search (PMS), trace mineral search (TMS), and energy dispersive spectra signal (EDS) mineralogy analyses performed on three sulfide samples by AMMTEC. Samples of diorite, metasediments (MTS), and massive pyrite mineralization were analyzed. The results indicated that the gangue is composed mainly of quartz (31%), micas/clays (27%) and feldspars (21%). The sulfide mineralization consists of pyrite, arsenopyrite, chalcopyrite and sphalerite.

AMTEL Ltd. (AMTEL) analyzed a sample of sulfide ore (composite MC4) and showed that sulfide minerals contain most of the gold. The majority of the sulfide gold is present in a submicroscopic form. Arsenopyrite has the highest content of submicroscopic gold, followed in turn by pyrite and marcasite. Metallic gold accounted for 14% of the gold in the sample, and this is consistent with conventional direct cyanidation extracting only 17% of the gold. Only an additional 10% of the gold was extracted using ultra-fine grinding (P80 of 5 µm) and cyanidation. The mineralogical work conducted by AMTEL confirmed that gold recovery requires either whole ore pre-oxidation or flotation.

1.11.2.3 Flowsheet Determination Testwork

The PFS process flowsheet design, a POX circuit followed by copper and gold recovery circuits, used criteria developed from the 2009 and 2010 SGS metallurgical test program.

Alacer developed and implemented a metallurgical test program with Hazen Research Inc. (Hazen) in early 2012 to support the 2014FS. Alacer personnel identified and shipped samples representing the rock types hosting sulfide mineralization to Hazen in Golden, Colorado. Hazen prepared the samples and conducted the majority of the FS testwork. The program aimed to determine appropriate operating conditions for the POX circuit and the subsequent process operations. Hazen completed multiple batch testwork campaigns and multiple pilot plant campaigns under the banners Campaign 1 through Campaign 4. Additional testwork was conducted by third-party consultants and vendors, using samples generated by Hazen.

The first objective of the Hazen campaigns was to develop a feasible POX process followed by copper recovery and conventional cyanidation of POX residue for the recovery of gold. The second objective, predominantly achieved by continuous pilot testing, was to develop metallurgical data to support completion of a FS.

The test campaigns incorporated variability testing of spatially-diverse samples from the deposit and head grade variability within the mineralization types. The campaign results allowed development of recovery models, selection of major equipment, and the estimation of reagent consumptions.

The Hazen campaigns covered the following areas:

- Head characterization of Campaigns 1 through 4 and Variability Study (VS) VS1 and VS2.
- Comminution testing.
- Direct cyanidation.
- POX testing.
- Hot cure testing.
- Iron arsenic precipitation.
- Metal sulfide precipitation (MSP) (for copper recovery).
- Solid-liquid separation.
- Tailings filtration.
- Bulk cyanidation and carbon kinetics.
- Cyanide destruction and environmental testing.
- Sulfide feed stock variability testing.
- Flotation testing.

Campaign 4 results provided the fundamental basis for the flowsheet.

SGS Lakefield Oretest in Perth, Western Australia conducted additional pilot testing (Campaign 5) during 2015 at the direction of Alacer.

The Campaign 5 testwork utilized various composite samples that represent the first 3 years' operation and LOM blend that resulted in changes to the acidulation area and changes in thickener design. Analysis of gold recovery results on variability samples confirmed that, in laboratory conditions, it is possible to recover between 96 and 98% of the gold (depending on ore type) at expected head grades and using design operating conditions to achieve almost complete oxidation of pyrite. On average, only 14.6% of the silver was recovered.

Analysis of the results provided a recovery model for use in economic analysis. Additional discounts have reduced the calculated recoveries allowing for commissioning, solution losses in the counter-current decantation (CCD) stage and for operation on a single autoclave (rather than two autoclaves) at high throughput rates.

1.12 Mineral Resource Estimates

The Mineral Resource model was constructed by Loren Ligocki, SME Registered Member (RM SME), Alacer's Resource Geologist and full-time employee of Alacer, and Gordon Seibel, RM SME, a Principal Geologist with Amec Foster Wheeler. The Mineral Resource estimates were reviewed by Dr. Harry Parker, RM SME, Consulting Mining Geologist and Geostatistician with Amec Foster Wheeler. Gordon Seibel and Dr. Harry Parker are the Qualified Persons for the Mineral Resource estimate. Mineral Resources were classified using the criteria set out in the 2014 Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards for Mineral Resources and Mineral Reserves (the 2014 CIM Definition Standards).

The resource estimation method was designed to address the variable nature of the epithermal structural and disseminated styles of gold mineralization while honoring the bi-modal distribution of the sulfur mineralization that is critical for mine planning (material with sulfur < 2% is sent to the heap leach while material with sulfur grades ≥ 2% will be 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 (Manganese, Main, Marble and West) to reflect the different trends of the mineralization that commonly follow structures and/or the 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 mineralization within an expanded mineralized wireframe generated in the commercially-available software, Leapfrog. A probabilistic envelope was generated within the expanded gold shape to define the limits of the economic mineralization. The Leapfrog 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 mineralization. Two Au PACK models were constructed. The first (low-grade) model was applied to < 2% S material that can be processed by heap leaching, and the second (high-grade) model was later applied to ≥ 2% S material to be processed by the POX plant.

Geology, exploratory data analysis (EDA), composite grade comparisons and other checks were performed to develop the parameters used to build the models. Once constructed, the gold models were calibrated to past production categorized by total sulfur content (< 2% S and ≥ 2% S material) and mining area. Mineral Resources were classified to each block based on drill hole density and data quality.

Mineral Resources were assessed for reasonable prospects for eventual economic extraction by reporting only material that fell within a Lerchs-Grossmann (LG) conceptual pit shell using metal prices of $1,400/oz for gold and $21.00/oz for silver. Due to process design changes for the proposed POX plant, copper was not included in the LG calculation. Key parameters are summarized in Table 1-2.

Table 1-2 Summary of Key Parameters Used in Lerchs-Grossmann Conceptual Pit Shell

Description	Element	Minimum	Maximum
Heap Leach Recovery	Au	62.3%	78.4%
	Ag	24.6%	37.8%
	Cu	3.5%	15.8%
POX Recovery	Au	94%*	94.0%
	Ag	3.0%	3.0%
	Cu	~	~
Mining Cost per tonne mined	---	$1.90	$1.90
Process Costs Heap Leach per tonne	---	$5.24	$9.87
Process Costs POX per tonne	---	$33.40	$33.40
Site Support per tonne processed	---	$3.50	$3.50
Internal Au Cutoff - Heap Leach	---	0.25	0.40
Royalty	---	2%	2%
Inter Ramp Slope RQD≤15	---	25 degrees	52.5 degrees
Inter Ramp Slope RQD>15	---	40 degrees	52.5 degrees

1. POX costs assume 5,000 tonne per day production rate
2. An Au cut-off of 1.00 g/t was applied to all sulfide material
3. * Au recovery is the average percent over the life of mine

Mineral Resources are reported inclusive of Mineral Reserves, and have been tabulated by resource classification and oxidation state in Table 1-3. Mineral Resources are presented on a 100% basis.

Table 1-3 Mineral Resource Tabulation by Resource Classification and Oxide State

Mineral Resource Statement for the Çöpler Deposit (As of December 31st, 2015)							
Gold Cut-off Grade (g/t)	Material Type	Resource Category Material	Tonnes (x1000)	Au (g/t)	Ag (g/t)	Cu (%)	Contained Au (oz x 1000)
Variable	*Oxide*	Measured	-	-	-	-	-
		Indicated	24,959	1.04	3.19	0.13	836
		Stockpile - Indicated	148	0.87	-	-	4
		Measured + Indicated	**25,106**	**1.04**	**3.17**	**0.13**	**840**
		Inferred	20,863	0.83	6.40	0.13	557
1.0	*Sulfide*	Measured	-	-	-	-	-
		Indicated	70,151	2.12	5.94	-	4,771
		Stockpile - Indicated	5,102	3.67	-	-	602
		Measured + Indicated	**75,253**	**2.22**	**5.53**	-	**5,373**
		Inferred	12,739	1.99	12.00	-	814
Variable	*Stockpiles*	Indicated	5,250	3.59	-	-	606
Variable	***Total***	Measured	-	-	-	-	-
		Indicated	100,359	1.93	4.95	0.03	6,213
		Measured + Indicated	**100,359**	**1.93**	**4.94**	**0.03**	**6,213**
		Inferred	**33,602**	**1.27**	**8.52**	**0.08**	**1,371**

1. Mineral Resources have an effective date of December 31, 2015. Gordon Seibel and Harry M. Parker, both SME Registered Members, and Amec Foster Wheeler employees, are the Qualified Persons responsible for the Mineral Resource estimates. The Mineral Resource model was prepared by Messrs. Gordon Seibel and Loren Ligocki.
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 Alacer 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 (S < 2%) and sulfide material (S ≥ 2%) based on sulfur content.
5. The Mineral Resources meet the reasonable prospects for eventual economic extraction by reporting only material within a Lerchs-Grossmann (LG) conceptual pit shell. The following parameters were used: assumed throughput rate of 1.9 to 2.2 Mt/a; variable metallurgical recoveries in oxide including 62.3–78.4% for Au, 24.6–37.8% for Ag, 3.5–15.8% for Cu; metallurgical recoveries in sulfide including 94% for Au, 3% for Ag, no recovery for Cu; mining cost of $1.90/t; process cost of $5.24–$9.87/t leached and $33.40/t through the POX; general and administrative charges of $3.50/t; 2% royalty payable; inter-ramp slope angles that vary from 25–52.5º. Metal price assumptions were $1,400/oz for gold, $21.00/oz for silver, with copper excluded.
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 are rounded to the nearest thousand tonnes; grades are rounded to two decimal places.
9. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.

1.13 Mineral Reserve Estimates

Alacer currently operates a heap leach operation at the Çöpler mine with a production rate of approximately 6.0 Mt of oxide ore per annum with an average remaining life-of-mine (LOM) grade of 1.13 g/t Au. Heap leach operations are expected to continue through 2022 with production rates diminishing in 2018 as the mine transitions into the sulfide mineralization. All mining at Çöpler is undertaken by conventional open pit mining techniques. At present, all mining activities related to the extraction of material from the pits is being conducted by a contractor, retained by Alacer. It is anticipated the sulfide mineralization will also be exploited by conventional open pit methods, and that contractor mining will continue to be utilized.

Through the process of pit optimization and limitations on tailings disposal capacity, the Çöpler pit design and stockpiles delineates 18.0 Mt of oxide ore and 40.0 Mt of sulfide ore. The total LOM tonnage mined from the beginning of 2016 is 277.6 Mt with a strip ratio of 4.25 (waste/ore).

The pit design consists of 16 phases that first target oxide ore and then target sulfide ore in a manner that maximizes cash flow and efficiencies in the mine-to-mill interface. The final pit will be spread out over 2.7 km from west to east, 1.1 km from north to south with a maximum depth of 295 m below the original ground topography.

The commercially-available MineSight Schedule Optimizer tool was used to schedule the extraction of ore from the mine, with the objective of maximizing the net present value (NPV) within the constraints of production tonnages, metallurgical blend requirements, and mining operational efficiencies. The first scheduling period was started as of January 1, 2016, using the end of year December 31, 2015 surveyed topography for the mine. The scheduling interval was on a monthly basis through 2016, on a quarterly basis from 2017 through 2020, and thereafter on an annual basis for the remainder of the mine life. Prior to the commissioning of the sulfide mill, all sulfide ore is shipped to one of three sulfide ore stockpiles. The three sulfide ore stockpiles will be used for low-grade (1.5 – 3.2 g/t Au), medium-grade (3.2 – 4.0 g/t Au), and high-grade (4.0 g/t Au and higher) sulfide ore. The mill is scheduled to be in production through 2037, when it will exhaust the remainder of the low-grade sulfide ore contained in stockpile. All mining activities (oxide and sulfide) will cease in 2023, and the remaining mine life to 2037 is based on re-handle of stockpile material.

A resource block model, completed by Amec Foster Wheeler and Alacer in February 2016, was used as the basis for detailed economic pit optimization using the commercially available Geovia Whittle Version 4.4.1 pit optimization software. This software, in conjunction with economic, metallurgical, and geotechnical criteria, was used to develop a series of economic pit shells that formed the basis for design and production scheduling.

On the basis of metallurgical testwork and trade-off studies, the Mineral Reserve estimates are based on the following process routes:

- Heap leach of all oxide ore.
- Whole ore POX of all sulfide ore.

This Technical Report is based on the continued use of a mining contractor. The contractor supplies all personnel, equipment, and facilities required to perform the entire mining operation. Alacer will incur additional costs associated with the supervisory, engineering, and grade control functions.

All costs mentioned in Section 16.0 are used as the basis of the Mineral Reserve estimate and may not reflect cost metrics used for financial analysis based on the timing of the cost estimate and the differences in allocation of various site support costs

The Mineral Reserves for the Çöpler gold deposit have been estimated by Alacer as summarized in Table 1-4. Mineral Reserves are presented on a 100% basis.

Mineral Reserves are quoted as of December 31, 2015. Oxide Mineral Reserves use a calculated internal gold cut-off grade (excluding mining cost) ranging from 0.30 g/t Au to 0.45 g/t Au, while sulfide Mineral Reserves use a gold cut-off grade of 1.50 g/t Au.

Table 1-4 Mineral Reserves for the Çöpler Gold Deposit

Mineral Reserves for the Çöpler Mining area deposit (As of December 31st, 2015)						
Reserve Category Material	**Tonnes (x1000)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Contained Au Ounces**	**Recoverable Au Ounces**
Proven - Oxide In-Situ	-	-	-	-	-	-
Probable - Oxide In-Situ	17,836	1.13	3.53	0.13	650,000	494,000
Probable - Oxide Stockpile	148	0.87	-	-	4,000	3,000
Total - Oxide	**17,984**	**1.13**	**3.50**	**0.13**	**654,000**	**497,000**
Proven - Sulfide In-Situ	-	-	-	-	-	-
Probable - Sulfide In-Situ	34,879	2.63	7.23	-	2,944,000	2,829,000
Probable - Sulfide Stockpile	5,102	3.67	-	-	602,000	579,000
Total - Sulfide	**39,982**	**2.76**	**6.30**	-	**3,546,000**	**3,408,000**
Proven - Oxide + Sulfide + Stockpile	-	-	-	-	-	-
Probable - Oxide + Sulfide +Stockpile	*57,965*	*2.25*	*5.44*	*0.04*	4,200,000	3,905,000
Total - Oxide + Sulfide	***57,965***	***2.25***	***5.44***	***0.04***	***4,200,000***	***3,905,000***

1. Mineral Reserves are not diluted.
2. Full mine recovery assumed.
3. Average Heap Leach Au recovery for all rock types is estimated at 76.0% and for Pressure Oxidation (POX), 96.1%.
4. Numbers may not add up due to rounding.
5. The Mineral Reserves were developed based on mine planning work completed in March 2016 and estimated based on End of December, 2015 topography surface.
6. A calculated gold internal cut-off grade was applied to Oxide Heap Leach Mineral Reserves using the equation: Xc = Po / (r * (V-R)) where Xc = Cut-off Grade (gpt), Po = Processing Cost of Ore (USD/tonne of ore), r = Recovery, V = Gold Sell Price (USD/gram), Refining Costs (USD/gram). A gold cut-off grade of 1. 50 g/t was used for Sulfide Pressure Oxidation Ore.
7. Mineral Reserves are based on US$ 1,250/Oz Au Gold Price.
8. The Mineral Reserves were estimated by Stephen Statham, PE (Colorado License #PE.0048263, SME 4140907RM) of Alacer, a qualified person under NI 43-101 and JORC guidelines.

Mineral Reserves have been classified using the 2014 CIM Definition Standards.

The Mineral Reserves disclosure presented in Table 1-4 were estimated by Stephen Statham, PE, RM SME, who is a full-time employee of Alacer.

The mine plan developed in this report is based on Proven and Probable Mineral Reserves only. There is upside opportunity for the Project if some or all of the Inferred Mineral Resources can be upgraded to higher-confidence categories with additional infill drilling and supporting studies.

1.14 Mining Methods

All mining at Çöpler will be undertaken by conventional open pit mining techniques used for hard-rock truck-and-shovel operations. Contractor mining will be retained for the LOM.

1.15 Process Plants

1.15.1 Oxide Ore Heap Leach Processing

Construction of a heap leach facility was undertaken from 2008-2010, and the first gold pour was achieved in the fourth quarter of 2010. The process was designed to treat approximately 6.0 Mtpa of ore by three-stage crushing (primary, secondary and tertiary) to 80% passing 12.5 mm, agglomeration (with cement and water) 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 using a high-temperature, pressure elution process, and electrowinning, retorting and melting of the resulting product to yield a doré (containing gold and silver) suitable for sale. Carbon is regenerated using acid washing and reactivation in a rotary kiln, and the carbon is recycled back to the CIC system. Subsequent to commissioning of the plant, a sulfidization-acidification-recovery-thickening (SART) plant has been constructed and commissioned to remove copper from the leaching solution and to regenerate cyanide. The SART process operates intermittently, on an as-needed basis. The process flowsheet is summarized in Figure 1-3.

Since commissioning through the end of December 2015, an estimated 1,734 koz ounces of gold has been placed on the heap, contained within approximately 35.2 Mt of ore at an average grade of 1.52 g/t Au (0.049 oz/t). At the end of December 2015, 1,078 koz ounces had been produced as bullion. It is noted that approximately 25% of the material placed onto the leach pad between 2010 and the end of 2014 was placed as run-of-mine ore (no crushing or agglomeration).

Figure 1-3 Process Flowsheet for Heap Leach

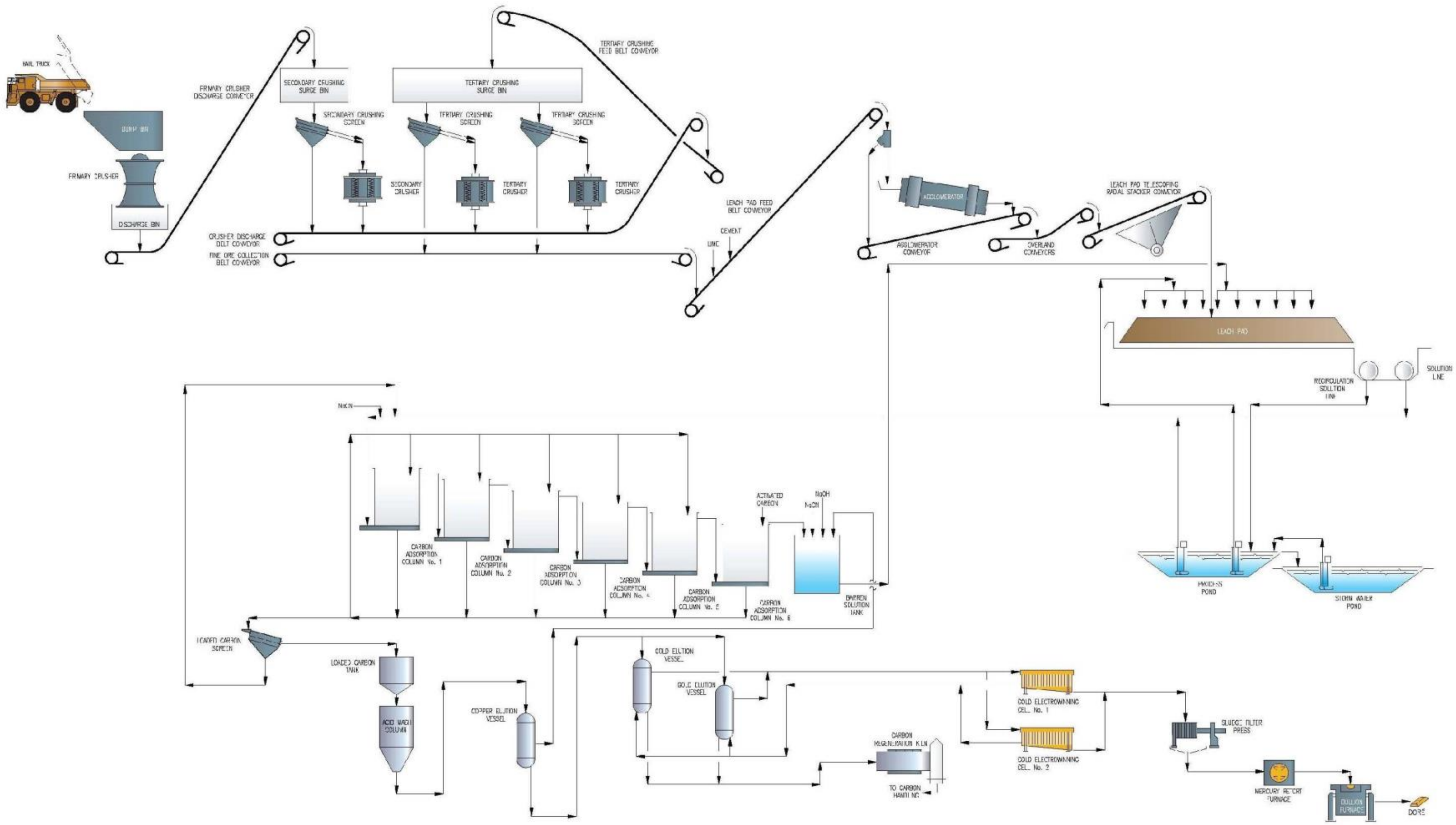


Figure prepared by Alacer, 2016.

1.15.2 POX Processing

The Çöpler Sulfide Expansion Project is designed to treat 1.9 to 2.2 Mtpa of sulfide ore, from which gold-silver doré will be produced.

Run-of-mine (ROM) sulfide process feed stock will be transported by haul trucks to the sulfide process stockpile. Sulfide process feed stock will be deposited in specified areas in the process stockpile according to sulfide feed blending parameters. The POX circuit was designed to run within a specific range of feed parameters. In order to feed the POX system a consistent blend meeting these parameters, front-end loaders will be used to deliver sulfide process feed stock from the various areas of the stockpile to the primary crusher according to blending parameters.

Ore will be fed to a ROM bin protected with a static grizzly and an apron feeder at the base of the bin feeds a primary sizer. During major sizer maintenance the crusher will be removed to be worked on off-line and will be replaced with a 300 mm square static grizzly. Finer than average ore will be deliberately selected for feeding at these times. The primary crushed ore (or 300 mm grizzly undersize) will directly feed a semi-autogenous grinding (SAG) mill. Due to the fine (and potentially sticky) nature of the ROM ore, there is no SAG mill feed stockpile. The SAG mill will be fitted with a discharge trommel screen and the screen oversize will be recycled directly to the mill using a water-jet trumpet return, centrally located in the trommel.

SAG screen undersize will feed the ball mill that will operate in closed circuit with a cyclone cluster. The grinding circuit product, cyclone overflow, will be screened to remove tramp oversize, then it will be thickened in the grinding circuit thickener. The thickener underflow slurry will be pumped to the acidulation feed tanks.

A partial acidulation circuit was adopted where one fraction of the grinding thickener underflow will be acidulated fully and the remainder will bypass acidulation and will be sent directly to the POX feed tanks. Slurry will be acidulated using recycled acid from the decant thickener overflow, and supplemented with fresh sulfuric acid if required. The acidulated slurry stream portion will be pumped to the POX feed thickener with most of the thickener overflow pumped to the decant thickener. Excess thickener overflow will be bled to the iron/arsenic precipitation tank as needed. The thickened acidulated slurry will be pumped to the POX feed surge tank to join the unacidulated slurry and decouple the thickener system from the autoclaving system.

Slurry will be pumped from the POX feed surge tank to the low-temperature heaters. Slurry will be heated using steam generated in the low-temperature flash tank. The low-temperature heated slurry will be pumped to the high-temperature heater and mixed with steam from the high-temperature flash tank. The hot slurry will be pumped from the high-temperature heater to the autoclaves at the required POX system operating pressure.

The autoclave circuit will consist of two horizontal autoclaves operating in parallel. The slurry will flow through the baffled chambers of the autoclaves and will be reacted with oxygen gas at each of the agitators. The autoclaves are designed to operate at 220°C, 3,150 kPa.g and provide 60 minutes of residence time, each with half the plant flow going to each unit. Treated slurry will exit the last vessel through the pressure letdown system consisting of a high-pressure and a low-pressure flash vessel.

The depressurized hot slurry will be combined with the POX feed thickener overflow and thickened in the decant thickener. The thickened slurry will be pumped to the iron/arsenic precipitation system. The thickener overflow will be recycled to the acidulation circuit to minimize fresh acid addition.

The iron/arsenic precipitation system will consist of two agitated tanks in series. Limestone will be added, raising the slurry pH to form a stable iron arsenate precipitate.

The treated slurry from the iron/arsenic precipitation system will be pumped to the two-stage CCD thickener system to remove dissolved copper from the gold-bearing solids. This step is required to limit copper consumption of cyanide and copper loading onto activated carbon. Washed slurry from CCD2 will be pumped to the pre-leach tank, the first step of the cyanidation circuit. The CCD1 overflow will be pumped to the tailings neutralization tanks. Provision has been made in the plant layout for future recovery of a saleable copper product from the CCD1 overflow.

Lime will be added to the washed slurry from CCD2 in the pre-leach tank. Lime raises the slurry pH to about 10.5 prior to feeding the two-stage cyanide leach tanks. Sodium cyanide will be added in the leach tanks to dissolve virtually all the gold and a small amount of the silver from the oxidized solids. The leached slurry will feed a six-stage carbon-in-pulp (CIP) gold recovery system.

In the CIP tanks, the solubilized precious metals will load onto activated carbon that will be mixed with the leached slurry in each tank. Slurry will flow continuously from tank to tank through carbon screens, which will retain the carbon in each tank. Loaded carbon will be removed from the first CIP tank and pumped to the new adsorption-desorption-recovery (ADR) plant.

A new ADR facility and refinery will be provided to strip gold and silver from the loaded carbon, producing a pregnant solution for feeding an electrowinning system. Electrowinning will convert dissolved gold and silver to metal form ahead of producing doré bars. The new ADR plant and refinery will be equipped with air emissions control equipment to scrub the gas being vented to meet Turkish air emission limits. Stripped carbon will be reactivated using a carbon kiln and reused in the CIP circuit.

CIP tailings will be processed in a cyanide destruction circuit utilizing SO_2/air treatment technology. The system will reduce the slurry cyanide concentration to meet Turkish discharge regulations. The detoxified slurry will be pumped to the tailings neutralization circuit.

The detoxified CIP tailings will be combined with the CCD1 overflow where milk-of-lime slurry will be added to raise the pH to precipitate manganese and magnesium, stabilizing the slurry in the neutralization tanks. The neutralized slurry will flow to the tailings thickener. The thickener underflow will be pumped to the tailings holding tank. The tailings will be pumped from the holding tank through the tailings pipeline to the tailings storage facility. Tailings thickener overflow will be pumped to the process water tank for reuse in the process.

A pumping system will be provided in the tailings storage facility (TSF) to reclaim decanted water and return the water to the process water tank.

Reagent systems will be provided to mix and deliver the required reagents to the various addition point in the process.

Utility systems including compressed air, steam generators, and water distribution systems will be provided to service the process systems.

A schematic flowsheet of the process is presented in Figure 1-4.

Figure 1-4 Çöpler Sulfide Process Schematic

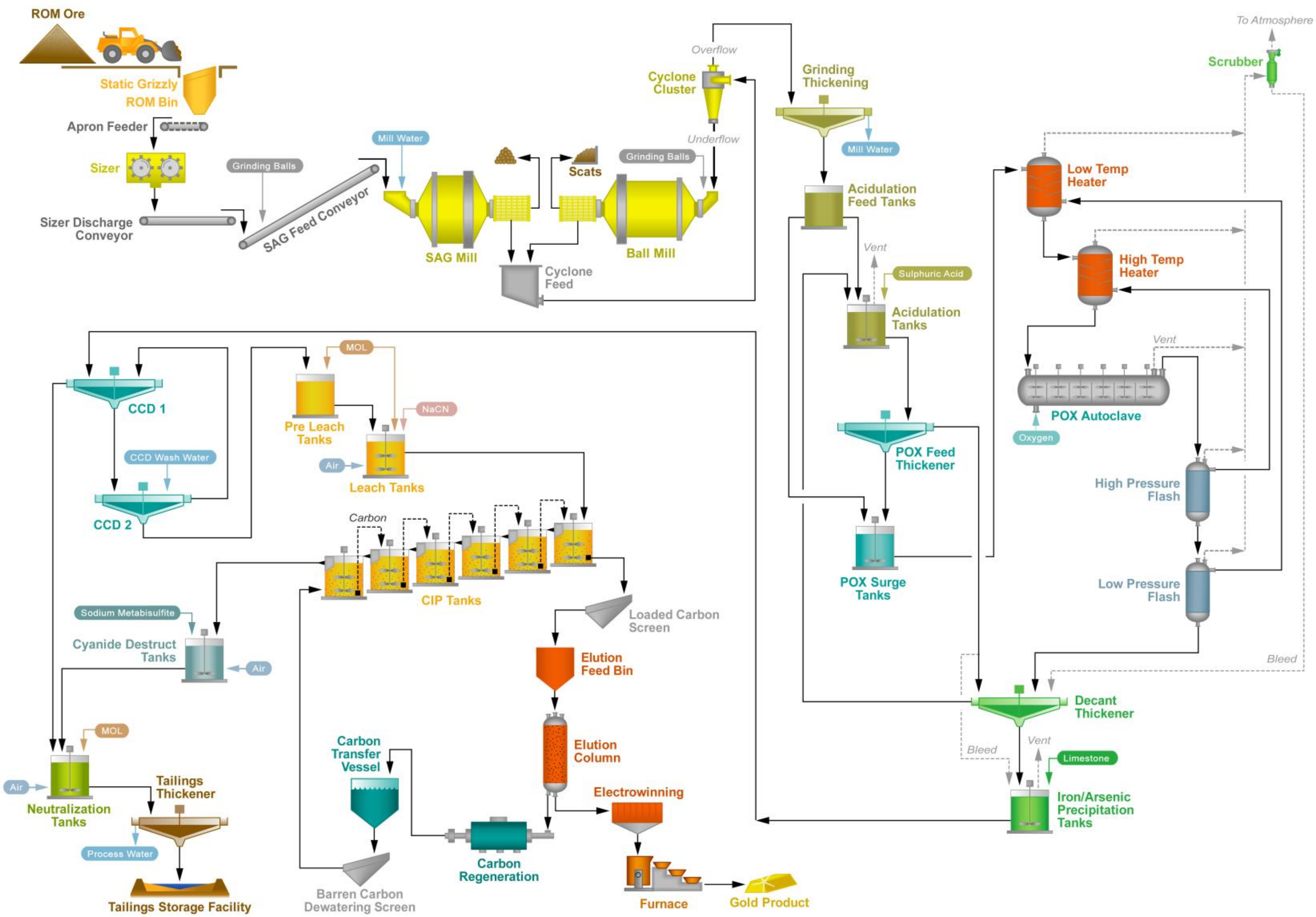


Figure prepared by Amec Foster Wheeler, 2016.

1.16 Project Infrastructure

1.16.1 Infrastructure

Infrastructure required for the heap leach operation is in place and no additional infrastructure is required for the heap leach activities for the remainder of the mine life.

The infrastructure for the Sulfide Expansion Project will be partially supported by the existing facility infrastructure. Some of the existing infrastructure will adequately support the new facility, while other components will be modified to meet the design criteria of the overall mine. The majority of the infrastructure for the Sulfide Expansion Project will be new.

The planning and design of new infrastructure was developed to suit the available area and to provide the required resources at the site. Consideration was given to the topography, geotechnical information, space constraints and economical process flow requirements during construction and operation. All aspects of the design reflect the compliance to applicable Turkish national codes and local codes.

The new infrastructure requirements include power supply, buildings, water and sewage, communications, site roads, plant fire protection system, and plant lighting system.

1.16.2 Tailings Storage Facility

The TSF for the Sulfide Expansion Project has been designed to provide containment for up to 45.9 Mt of mill tailings. The tailings will be pumped to the fully-lined tailings impoundment over an approximate 20-year mine life. Approximately 6,293 tpd of tailings will be pumped at a slurry density of 28% by weight from the tailings thickener to the TSF.

The Sulfide Expansion Project will make use of the same TSF location proposed in 2007, with an increase in overall height of the embankment crest from 1,224 m to 1,264 m amsl to accommodate the increased mass of tailings anticipated in the current mine plan.

The TSF design includes a rockfill embankment with downstream raise construction, an impoundment underdrain system, a composite liner system, and an overdrain system.

1.17 Market Studies and Contracts

1.17.1 Markets

The markets for gold and silver doré are international and generally robust but variable, depending on supply and demand.

Currently, 50% of the gold and silver from the Çöpler heap leach operations is delivered to METALOR Technologies S.A in Switzerland. The remaining 50% is delivered to the Istanbul Gold refinery. It is expected that sale of gold recovered from the Sulfide Expansion Project will be similar to the current arrangement.

Due to low copper prices, a decision has been made to remove the copper circuit in the POX plant design. Provisions have been made in the plant design to include the copper circuit in the future should copper prices improve. Copper precipitate is currently produced from the SART plant and sold into local markets in Turkey.

1.17.2 Contracts

Anagold contracts the mining operations to a Turkish mining contractor. The contract term expires on February 1, 2017. The contract contains provisions for escalation/de-escalation for 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.

Anagold has entered into a contract with Amec Foster Wheeler for engineering, procurement and construction management for the Sulfide Project. The Company has or will enter into a number of additional contracts for earthworks, oxygen supply and construction services in connection with the construction of the Sulfide Project.

1.18 Environmental and Permitting

The EIA permitting process for the Sulfide Expansion Project started on April 07, 2014 and ended by receiving the "EIA Positive Statement" on December 24, 2014. The EIA permit serves as a construction permit. The forestry land use permits for the construction of the Çöpler Sulfide Expansion Project were obtained on 20 April, 2016.

The EIA permitting for the Çöpler gold mine for the oxide ore was completed in April 2008 with the issuance of an EIA positive certificate. All of the operation permits have already been obtained for the oxide resources. These are: explosive storage permit, permit for water abstraction from groundwater sources, EIA positive 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 started on April 7, 2014 and was completed with the receipt of an "EIA Positive Statement" on December 24, 2014. In addition to EIA approval, other permits required for the Sulfide Expansion Project involve an expanded workplace opening permit, additional operating permits and land acquisition permits for forest areas and pasturelands, etc.

An Environmental Impact Assessment (EIA) study was completed in 2008 for the heap leach operation assuming processing of oxide ores. The project description for the 2008 EIA included three main open pits, five waste rock storage areas (WRSAs), a heap leach pad, a processing plant, and a TSF. The 2008 project description involved only the oxide resources.

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

- EIA permit dated April 10, 2012 for the operation of a mobile crushing plant.
- EIA permit dated May 17, 2012 for capacity expansion involving (i) increasing the operation rate to 23,500 tpd; (ii) increasing the Çöpler WRSA footprint area; (iii) adding a SART plant to the process in order to decrease the cyanide consumption due to high copper content in some ores.

The EIA studies were conducted according to the format stipulated by the Turkish EIA Regulation. In the period following the receipt of the 2008 EIA permit, Alacer conducted additional studies to supplement the Turkish EIA study and subsequently meet International Finance Corporation (IFC) requirements. These studies involved a Resettlement Action Plan (RAP) for the Çöpler village, a socio-economic baseline study for the Çöpler village, a human rights assessment study, an Environmental Management Plan, and a biodiversity study.

SRK Danışmanlık ve Mühendislik A.Ş. (SRK) was retained by Alacer to undertake the Çöpler Sulfide Project Environmental and Social Impact Assessment (ESIA) study for permitting and possible financing purposes. The Stakeholder Engagement Plan (SEP) and the Social Impact Assessment (SIA) for the Sulfide Expansion Project was prepared and reported in May 2015.

The Çöpler Sulfides Expansion Project ESIA process did not identify any fatal-flaw impacts, due to the limited nature of sensitive environmental and human receptors, and the existing disturbed nature of the site.

1.19 Capital and Operating Costs

1.19.1 Capital Costs

Capital costs were updated during the detailed engineering phase. The update reflects the decision to adopt two horizontal autoclaves in the current process design over the vertical autoclave arrangement that was envisaged in earlier designs, updated material quantities, updated equipment pricing and revised construction direct and indirect cost estimates.

The initial capital cost estimate was based on the scope of work as outlined in the facilities description and Work Breakdown Structure (WBS).

The estimate is considered to have an accuracy of +10% / -5%. The total estimated initial capital cost to design, procure, construct and start-up the facilities as of April 1, 2015 is $743.7 million, including owner's costs. The initial capital required for the TSF starter embankment is $30.7 million. Total LOM capital for the TSF is estimated at $291.6 million. This includes initial and sustaining capital costs for the TSF. Table 1-5 summarizes the estimated initial capital costs.

Table 1-5 Overall Initial Project Capital Cost Summary

PROJECT AREA	USD $M
1000 - Process Plant	269.9
2000 - Process Plant Utilities & Services	74.1
3000 - Tailings Storage Facility	30.7
4000 - Support Infrastructure & Temp Facilities	101.0
5000 - Engineering, Procurement & Construction Management	93.6
6000 - Start-up & Commissioning	10.3
8000 - Owner's Costs	87.1
9000 - Provisions (incl. Growth Allowance & Contingency)	76.9
TOTAL	**743.7**

The estimate is expressed in fourth-quarter 2015 United States dollars.

Mining operations are currently contracted to an outside party and this arrangement is expected to continue during the foreseeable future. Therefore, 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.

Costs incurred prior to 1 April 2015 are considered to be sunk costs.

1.19.2 Operating Costs

Operating costs are expressed in Q4 2015 U.S. dollars with no allowance for escalation. The projected LOM unit operating cost estimate is summarized in Table 1-6.

Table 1-6 Summary of Life-of-Mine Average Operating Costs

Activity	Unit	Life of Mine Average Unit Cost
Mining	per tonne mined	1.50
Rehandle	per tonne ore rehandled	1.12
Heap Leach Processing	per tonne HL ore processed	8.09
POX Processing	per tonne POX ore processed	31.80
Site Support and Offsite	per tonne ore processed	5.83

The LOM all-in operating costs per gold ounces are summarized in Table 1-7.

Table 1-7 Summary of All in Cash Costs Net of By-Products

Costs per Ounce (Cash Basis)	Units	Amount
Cash Operating Costs (C1)	**US$/oz**	**563**
By-Product Credits (Ag, Cu)	US$/oz	(9)
Cash Operating Costs net of By Products (C1)	**US$/oz**	**554**
Royalties	US$/oz	17
Total Cash Costs (C2)	**US$/oz**	**570**
Sustaining Capital	US$/oz	74
All In Sustaining Costs (AISC)	**US$/oz**	**645**
Sulfide Preproduction Capital	US$/oz	183
Reclamation	US$/oz	17
All In Costs (AIC)	**US$/oz**	**844**

Reported as Unit Cost per Ounce. Negative costs indicated in this table reflect the positive revenue from the silver and copper by-product sales that are deducted from the operating cash costs. Totals may not sum due to rounding.

Sulfide Processing Costs

The process operating costs for the Sulfide Expansion Project were estimated from first principles. They were calculated assuming 19 full years of operation for the POX plant. Operating costs were based on metallurgical testwork, the mine plan, Alacer compensation/benefit guidelines, and recent supplier quotations for consumables. Consumables included in the operating costs include spare parts, repair supplies, wear liners, grinding media and screen components. Alacer has elected to capitalize autoclave vessel refractory replacement in the years following the initial start-up, and these are not part of the operating costs but are included in sustaining capital.

The copper recovery circuit has been eliminated from the process flowsheet due to low copper prices. This has resulted in approximately a $3/t reduction in operating costs. Reagent costs have been updated to Q4 2015 US dollars based on recent quotes and foreign exchange rates.

LOM average sulfide processing costs for the project are shown in Table 1-8. Costs are shown on a $/tonne sulfide ore processed, $/oz of gold recovered by the sulfide process, and the average total sulfide circuit operating cost in million $/year.

Table 1-8 Life-of-Mine Sulfide Processing Costs by Cost Component

Item	$/t Sulfide	$/oz Sulfide	Annual Cost, $M
POX Processing - Labour	4.50	53	9.5
POX Processing - O_2 Plant Fixed	4.05	48	8.5
POX Processing - O_2 Plant Variable	3.41	40	7.2
POX Processing - Reagents	9.23	108	19.4
POX Processing - Fuel Oil	0.92	11	1.9
POX Processing - Electrical	4.52	53	9.5
POX Processing - Maintenance Materials	3.89	46	8.2
POX Processing - Large Mobile Equipment	0.22	3	0.5
POX Processing - Laboratory	0.89	10	1.9
POX Processing - Commissioning	0.17	2	0.4
Total Sulfide Processing Costs	**31.80**	**373**	**66.9**

Note: Totals may not sum due to rounding.

1.20 Economic Analysis

Information in this sub-section includes forward-looking information. Readers are requested to view the cautionary statements in Section 2.2 regarding information that is forward-looking. Actual results may differ from those presented in this sub-section.

A financial analysis for the Sulfide Expansion Project was carried out using an incremental or differential cash flow approach. Cash flow models were developed for the Sulfide Expansion Project with the oxide heap leach as well as for the oxide heap leach alone without the sulfide project. A differential cash flow was calculated between the two sets of cash flows to determine the financial benefit of the sulfide project. The IRR and NPV using a discount rate of 5% were calculated using this differential cash flow. The financial analysis was performed using the following key assumptions:

- The base case gold, silver and copper prices are $1,250/oz, $18.25/oz and $2.75/lb respectively.
- Cash flows begin on January 1, 2016 and end on December 31, 2046.
- The cash flows take into account depreciation, cash taxes, changes in working capital, and tax credits.
- Commissioning is expected at the end of second quarter 2018 with sulfide gold production to begin in the third quarter of 2018.

- Unless noted otherwise, all cost and sales estimates are in constant Q4 2015 U.S. dollars with no escalation factors taken into account.

Table 1-9 provides a summary of the NPV, IRR and payback period using a 5% discount rate.

Table 1-9 Financial NPV, IRR, and Payback Period

Description	Unit	Amount
METAL PRICES		
Gold Price LOM	US$/oz	1,250
Silver Price LOM	US$/oz	18.25
Copper Price LOM	US$/lb	2.75
PROJECT CASH FLOWS		
Sulfide and Oxide Projects	US$M	1,577
Oxide Project	US$M	94
Project Differential Cash Flow	US$M	1,483
PROJECT FINANCIALS		
NPV at 5% of Differential Cash Flows	US$M	728
IRR of Differential Cash Flows	%	19.2
Payback on Sulfide Project Cash Flow (from Start of Sulfide Production)	years	3.0

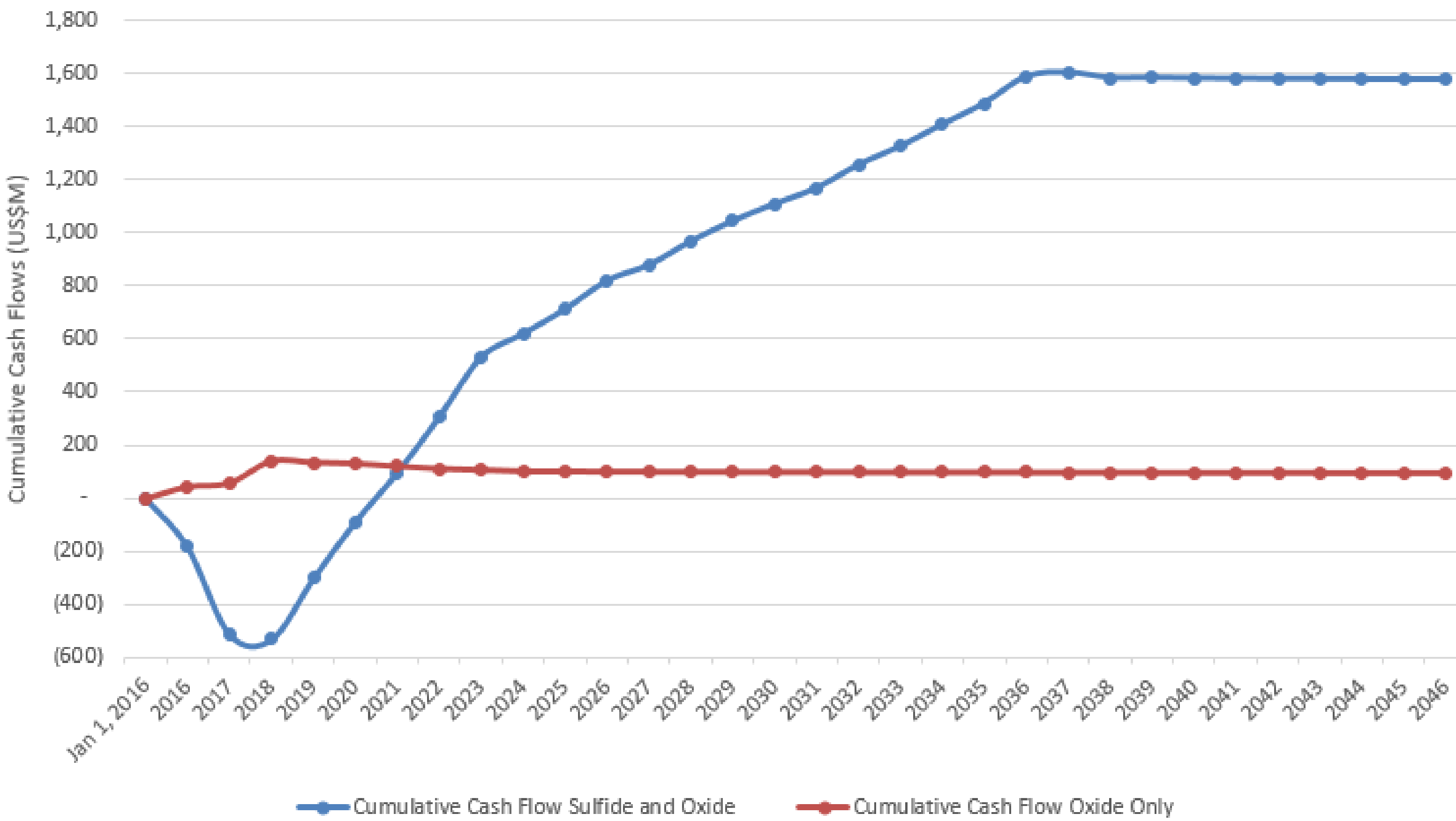
The project payback period, based on the cash flow for the combined sulfide processing and heap leach operation, is 3.0 years following the startup of the POX plant.

LOM cash flows for the Project, the oxide heap leach only case and the differential cash flow between the two are shown in Table 1-10.

Table 1-10 Sulfide Project with Oxide Heap Leach Cash Flow

Year	Gold Price ($/oz)	Copler Sulfide Project and Oxide Heap Leach ($M)	Oxide Heap Leach Only ($M)	Cash Flow Differential ($M)
2016	1,250	(184)	44	(228)
2017	1,250	(328)	14	(342)
2018	1,250	(20)	81	(101)
2019	1,250	229	(5)	234
2020	1,250	213	(4)	217
2021	1,250	185	(10)	195
2022	1,250	210	(8)	218
2023	1,250	225	(4)	229
2024	1,250	89	(5)	94
2025	1,250	93	(1)	93
2026	1,250	107	(1)	108
2027	1,250	59	(1)	60
2028	1,250	89	(1)	90
2029	1,250	77	(0)	77
2030	1,250	64	(0)	64
2031	1,250	60	(0)	60
2032	1,250	87	(0)	88
2033	1,250	69	(0)	70
2034	1,250	83	(0)	83
2035	1,250	80	(0)	80
2036	1,250	100	(0)	101
2037	1,250	15	(0)	16
2038	1,250	(20)	(0)	(20)
2039	1,250	1	(0)	1
2040	1,250	(3)	(0)	(2)
2041	1,250	(1)	(0)	(1)
2042	1,250	(1)	(0)	(1)
2043	1,250	(1)	(0)	(1)
2044	1,250	(1)	(0)	(1)
2045	1,250	(0)	(0)	(0)
2046	1,250	(0)	(0)	(0)

Figure 1-5 Cumulative Cash Flows for Sulfide Project with Oxide Heap Leach and for the Oxide Heap Leach Only



The sensitivity analyses for NPV and IRR are shown in Figure 1-6 and Figure 1-7 respectively when the gold price, operating cost (Opex), capital costs (Capex) costs, sulfide gold grade and Turkish lira exchange rate assumptions vary.

Figure 1-6 Incremental NPV at 5% Sensitivities

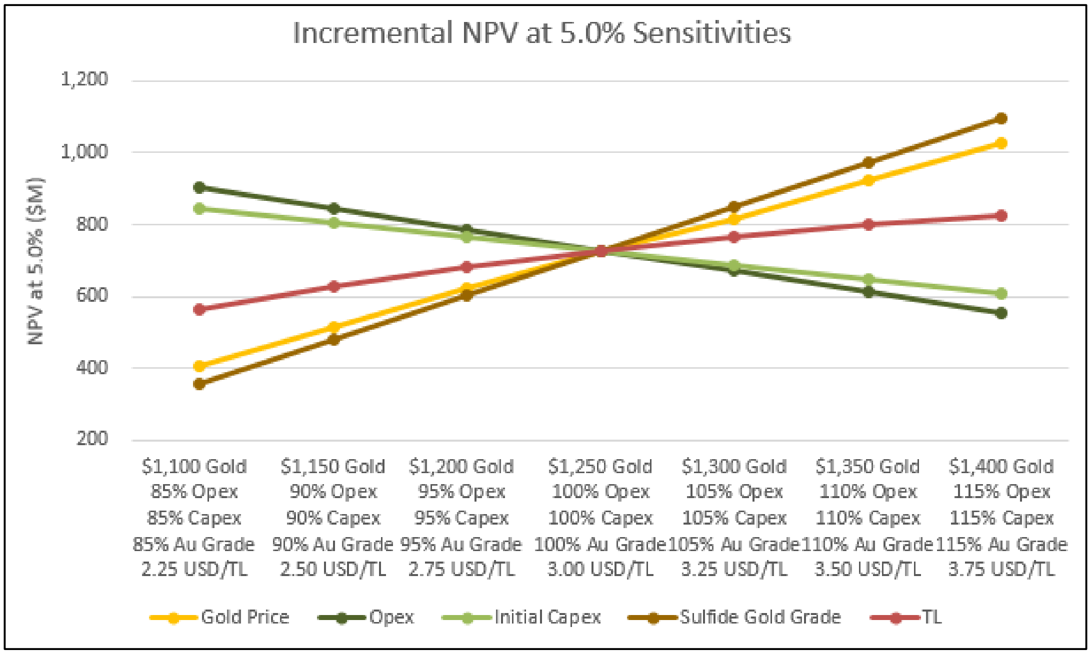
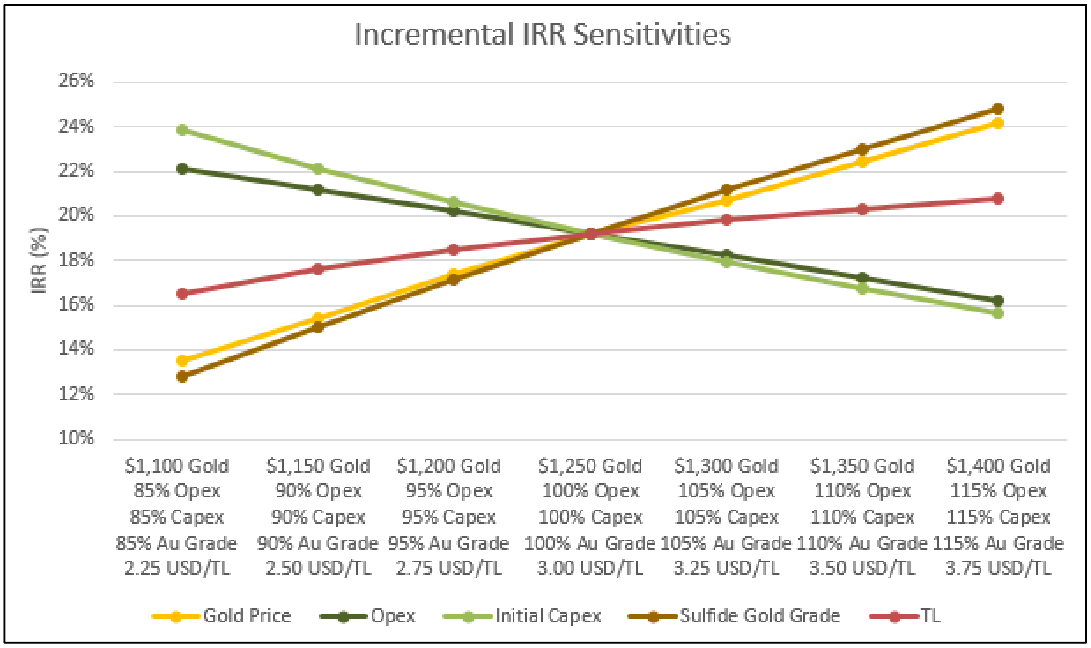


Figure 1-7 Incremental IRR Sensitivities



Figures prepared by Alacer, 2016.

USD = US$; TL = Turkish Lira; Opex = operating cost; Capex = capital cost.

1.21 Interpretation and Conclusions

Under the assumptions presented in the Report, the currently-operating heap leach operation shows positive economics. The Sulfide Expansion Project is shown to be economically and technically feasible and that the Project should move to construction.

There are no project execution issues identified at this time that could jeopardize the success of the Project.

1.22 Recommendations

Key recommendations by area:

Recommendations made by Amec Foster Wheeler for the drill database:

- Differences noted in the ALS and SGS assays should be corrected in the Datashed master database.
- Anagold should follow QA/QC protocol on lab checks, reassay when outside of acceptable range, and increase blank sample submission.

Recommended as part of the next phase of engineering and design associated with the Project:

- Detailed scheduling and design of the sulfide ore stockpiles should be completed. Results from ongoing metallurgical test work will assist in determining the optimal stockpiling strategy.
- Further refinement of the modeled carbonate and sulfide sulfur grades in the resource model should be completed.
- A detailed pit dewatering and depressurization plan should be designed and implemented to account for the increased depths of mining activities through the sulfide phases of the pit design.

Recommendations for metallurgy and mineral processing identified during the FS engineering:

- It is recommended that an effective heap leach production model be maintained and that the model be calibrated at least annually against actual gold production from the heap leaching facilities.
- Sulfide sulfur content in heap leach feed materials, as well as column and IBRT feed materials should be measured routinely and correlated against gold extraction.
- Perform a study of tailings disposal optimizing slurry disposal and examine slurry disposal versus dry tailings to meet project closure and reclamation requirements.

Some of the recommendations from the ESIA report are:

- An Integrated Water Management Plan will be developed for the Çöpler Mine. The management plan will enable the detailed assessment of process water use and water management during the operation phase as well as planning for the closure lake formation. Integrated water management report will be prepared every 5 years in the light of the estimations stated at the EIA report for the closure and the post-closure period, and will be submitted to the General Directorate of State Hydraulic Works.

- A monitoring program will be conducted in accordance with the commitments in EIA report and reported to the Ministry of Environment and Urban Planning, to the General Directorate of State Hydraulic Works.
- When the project enters the construction phase, and throughout the remaining life of the project, stakeholder engagement will also include:
    - Reporting on the Environmental and Social Management Plan (ESMP) and relevant supporting management plans; and
    - Opportunities for stakeholders to respond to the information received

Additional recommendations for the project are included in Section 26.0.

2.0 INTRODUCTION

Alacer Gold Corp. (Alacer or the Company) has prepared a Technical Report (the Report) on the Çöpler Mine and Çöpler Sulfide Expansion Project (the Project), located in Turkey.

The Çöpler Mine is owned and operated by Anagold Madencilik Sanayi ve Ticaret Anonim Şirketi (Anagold). Alacer controls 80% of the shares of Anagold and Lidya Madencilik Sanayi ve Ticaret A.Ş. (Lidya), formerly Çalık Holdings A.Ş., controls 20%. The same ownership percentage interests apply to the Çöpler Sulfide Expansion Project (the Sulfide Expansion Project).

Exploration tenures surrounding the Project are subject to joint venture agreements between Alacer and Lidya that have varying interest proportions. As noted earlier, Alacer Gold currently has an 80% stake in Anagold, and has a 50% stake in Kartaltepe Madencilik (Kartaltepe).

The Report has been prepared in support of updated Mineral Resource estimates disclosed in the Alacer news release dated 12 May, 2016, entitled Alacer Gold Announces Çőpler Sulfide Project Approval and to provide updated information on the current detailed engineering phase.

Co-contributors to the Report include Qualified Persons (QPs) from, in alphabetical order, Alacer, Amec Foster Wheeler E&C Services Inc., Amec Foster Wheeler Australia Pty Ltd (collectively Amec Foster Wheeler), Anagold Madencilik, Golder Associates Inc. (Golder Associates), John O. Marsden LLC (Metallurgium), Mining Plus Pty Ltd (Mining Plus), and SRK Consulting (Canada) Inc. and SRK Consulting (US) Inc. (collectively SRK).

2.1 Scope of Work

The Report presents the progress by Alacer in advancing the Sulfide Expansion Project. The Project is currently in detailed engineering, and procurement of long lead-time equipment has commenced.

A pre-feasibility study (PFS) titled *Çöpler Sulfide Expansion Project Prefeasibility Study* was completed in May 2011. The PFS found that the project was feasible and could support advancement to feasibility-level evaluation. Alacer published a report titled *Çöpler Sulfide Expansion Project Definitive Feasibility Study – Revision B* (FS) in August 2014 and found the project technically and financially feasible. In March 2015, a Technical Report tilted *Çöpler Sulfide Expansion Project Feasibility Update* was issued updating Mineral Resources, Mineral Reserves and other project specific parameters. The later document provided the basis of a decision in April 2015 to advance the sulfide project to detailed engineering, which is currently ongoing.

The Sulfide Expansion Project scope-of-work includes the process plant from run-of-mine stockpile and primary crushing to tailings discharge and tailings storage facility. Also included are a new absorption, desorption and refining (ADR) facility, a new warehouse/maintenance shop, and utilities including standby power generation and power distribution. Additional permanent office facilities will be provided as part of the Project scope. The existing administration facilities will continue to be used for owner personnel supporting the existing heap leach, and as space allows, part of the personnel required for support of the Sulfide Expansion Project.

Both oxide and sulfide material are contained within the Sulfide Expansion Project gold resource. All material that is heap leachable will be processed in the existing heap leaching plant (HLP) facilities.

On the basis of metallurgical test results the PFS and the FS, a whole-ore pressure-oxidation (POX) process was selected as the basis for the Project. The sulfide material will be processed in the new mill and POX facilities. The new processing facilities will be constructed adjacent to the existing heap leach facilities.

The processing facility for the Sulfide Expansion Project will have a throughput rate of 1.9 to 2.2 Mt/a. The average estimated gold production from the Sulfide Expansion Project will be 156,000 troy ounces per year.

The proposed tailings dam is designed with a capacity of 45.9 Mt to support the Sulfide Expansion Project.

POX process feed is currently being stockpiled in anticipation of the construction of the new sulfide processing facilities.

2.2 Forward-Looking Information

Except for statements of historical fact relating to Alacer, certain statements contained in this Report constitute forward-looking information, future oriented financial information, or financial outlooks (collectively “forward-looking information”) within the meaning of Canadian securities laws. Forward-looking information may be contained in this document and other public filings of Alacer. Forward-looking information often relates to statements concerning Alacer’s future outlook and anticipated events or results and, in some cases, can be identified by terminology such as “may”, “will”, “could”, “should”, “expect”, “plan”, “anticipate”, “believe”, “intend”, “estimate”, “projects”, “predict”, “potential”, “continue” or other similar expressions concerning matters that are not historical facts.

Forward-looking information includes statements concerning, among other things, preliminary cost reporting in this Report, production, cost and capital expenditure guidance; ability to expand the current heap leach pad, development plans for processing sulfide ore at Çöpler; results of any gold reconciliations; ability to discover additional oxide gold ore, the generation of free cash flow and payment of dividends; matters relating to proposed exploration, communications with local stakeholders and community relations; negotiations of joint ventures, negotiation and completion of transactions; commodity prices; Mineral resources, Mineral reserves, realization of Mineral Reserves, existence or realization of mineral resource estimates; the development approach, the timing and amount of future production, timing of studies, announcements and analysis, the timing of construction and development of proposed mines and process facilities; capital and operating expenditures; economic conditions; availability of sufficient financing; exploration plans; receipt of regulatory approvals and any and all other timing, exploration, development, operational, financial, budgetary, economic, legal, social, regulatory and political matters that may influence or be influenced by future events or conditions.

Such forward-looking information and statements are based on a number of material factors and assumptions, including, but not limited in any manner to, those disclosed in any other of Alacer’s filings, and include the inherent speculative nature of exploration results; the ability to explore; communications with local stakeholders and community and governmental relations; status of negotiations of joint ventures; weather conditions at Alacer's operations, commodity prices; the ultimate determination of and realization of mineral reserves; existence or realization of mineral resources; the development approach; availability and receipt of required approvals, titles, licenses and permits; sufficient working capital to develop and operate the mines and implement development plans; access to adequate services and supplies; foreign currency exchange rates; interest rates; access to capital markets and associated cost of funds; availability of a qualified work force; ability to negotiate, finalize and execute relevant agreements; lack of social opposition to the mines or facilities; lack of legal challenges with respect to the property of Alacer; the timing and amount of future production and ability to meet production, cost and capital expenditure targets; timing and ability to produce studies and analysis; capital and operating expenditures; execution of the amended credit facility; ability to draw under the credit facility and satisfy conditions precedent including execution of security and construction documents; economic conditions; availability of sufficient financing; the ultimate ability to mine, process and sell mineral products on economically favorable terms and any and all other timing, exploration, development, operational, financial, budgetary, economic, legal, social, regulatory and political factors that may influence future events or conditions. While we consider these factors and assumptions to be reasonable based on information currently available to us, they may prove to be incorrect.

You should not place undue reliance on forward-looking information and statements. Forward-looking information and statements are only predictions based on our current expectations and our projections about future events. Actual results may vary from such forward-looking information for a variety of reasons including, but not limited to, risks and uncertainties disclosed in Alacer's filings at **www.sedar.com** and other unforeseen events or circumstances. Other than as required by law, Alacer does not intend, and undertakes no obligation to update any forward-looking information to reflect, among other things, new information or future events.

2.3 Qualified Persons

The QPs for the Report are:

- Robert Benbow, PE, Alacer Gold.
- Stephen Statham, PE, Alacer Gold.
- Dean David, FAusIMM, Amec Foster Wheeler.
- Gordon Seibel, R.M. SME, Amec Foster Wheeler.
- Dr. Harry Parker, R.M. SME, Amec Foster Wheeler.
- Sergei Smolonogov, RPGeo., Anagold Madencilik.
- Richard Kiel, PE, Golder.
- John Marsden, PE, Metallurgium.
- Lisa Bascombe, MAIG, Mining Plus.
- Jeff Parshley, CPG, SRK Consulting.
- Mark Liskowich, P. Geo, SRK Consulting.

2.4 Effective Dates and Declaration

The following effective dates pertinent to the Report are:

- Database close out for Mineral Resource estimation: 15 July 2015
- Date of updated Mineral Resource estimate: 31 December 2015
- Date of Mineral Reserve estimate: 31 December, 2015
- Date of Forestry Permit approval required to fully construct the sulfide plant and tailings storage facility: 20 April 2016
- Date of supply of updated information on detailed engineering: 30 April, 2016

No material changes have occurred with respect to the Project between the 30 April 2016 date of last information on the engineering design and the filing date, therefore, the effective date of the technical report is considered to be that of the filing date, and is 9 June, 2016

2.5 Site Visits and Scope of Personal Inspection

Mark Liskowich inspected the Çöpler Project Area from September 21 to 22, 2010. The visit was to perform a site inspection of the environmental management of Phase 1 of the existing heap leach project. This site visit was also to gather information to be able to review and qualify the environmental sections of the Pre-Feasibility technical report (Samuel, 2011).

Dr. Harry Parker visited the project site from May 5 to 11, 2014. During the site visit, Dr. Parker inspected the open pit and selected drill core, reviewed cross sections, and reviewed reconciliation of production to Mineral Resource model depletions. He also inspected the head sampler, reviewed blast hole sampling, reviewed the sample preparation and visited the onsite assay laboratory.

Gordon Seibel visited the project site from May 5 to 11, 2014 and June 6 to 10, 2015. During the site visit Mr. Seibel inspected the open pit and selected drill core, reviewed cross sections, and reviewed reconciliation of production to Mineral Resource model depletions. He also inspected the head sampler, reviewed blast hole sampling, reviewed the sample preparation and visited the onsite assay laboratory. In addition, Mr. Seibel verified the locations of selected drill hole collars, visited the ALS and SGS laboratory, and collected witness samples.

John Marsden completed a visit to the Çöpler site from March 24 to 27, 2012. The purpose of the site visit was to view the existing operation and facilities, evaluate the locations for processing and ancillary facilities, view core samples to understand the geology as it relates to metallurgy, meet with Anagold employees and verify other information.

Robert Benbow is the Senior Vice President Strategic Projects and was General Manager of the Çöpler Mine in 2007. Mr. Benbow served as Vice President and Country Manager of Alacer's Turkish Business Unit from August 2008 to August 2011. Mr. Benbow has visited the site on numerous occasions in conjunction with his duties with Alacer, the latest being November 18 through 20, 2015.

Stephen Statham is an Alacer employee and has visited the project site on multiple occasions; most recently from March 21 to April 7, 2016. Site visits have included review of mine plans, mine design, pit slope geotechnical conditions, blasting conditions, operating strategy, and technical transfer of knowledge.

Richard Kiel has visited the site on numerous occasions since 2012 in conjunction with his duties as Golder's project manager, the latest being April 18-20, 2016. The visits prior to start of construction included a detailed review of the planned sulfide plant, WRSAs, TSF haul road, and TSF areas, as well as review of existing geological, geotechnical, and geophysical information. Site visits since the 3rd quarter of 2015 have included a review of ongoing civil earthworks construction activities in the process plant area and in support of future planning.

Lisa Bascombe has visited the site multiple times. Reviews of the Çöpler exploration drilling, logging and sampling systems and procedures were undertaken. The most recent site visit was in March 2014 for 30 days.

Sergei Smolonogov is the Çöpler Geology & Exploration Manager with on-going duties at the mine site and across the Çöpler District. Mr. Smolonogov is a full-time employee of Anagold.

2.6 Information Sources

The reports and documents listed in Section 2.6 (Previous Technical Reports), Section 3.0 (Reliance on Other Experts) and Section 27.0 (References) of this Report were used to support the preparation of the Report. Additional information was sought from Alacer personnel where required.

The following Amec Foster Wheeler personnel provided specialist input to Mr. Dean David:

- Karel Osten, Process Consultant, Amec Foster Wheeler, provided hydrometallurgical oversight for the study and viewed the Campaign 5 Pilot testwork. Karel provided the hydrometallurgical data, interpretations and designs summarized in the Report.
- Yavuz Atasoy, Principal Process Engineer, Amec Foster Wheeler, visited the Çöpler site from 19-27 November 2015 and provided data on stockpiled sulfide ore that is summarized in the Report.

The following SRK Consulting personnel provided specialist input to Mr. Jeff Parshley:

- Patric Lassiter, associate, visited the site October 17-20, 2012, and provided the discussion on closure approach and the LOM closure cost estimate for the 2012 study.
- Filiz Toprak, consultant, visited the site October 17-20, 2012, and provided the LOM closure cost estimate in 2012 and subsequent updates.

The following Golder personnel provided specialist input to Mr. Richard Kiel:

- Dale Armstrong, Senior Hydrogeologist with Golder, visited site from March 24 to 26, 2012, as a representative of Golder's hydrogeology team. The visit included a detailed review of the geologic and hydrogeologic setting of the project area, review of existing geological, hydrogeological, and geophysical information. In addition, the site visit included planning for future hydrogeologic field investigations to acquire additional data for use in the groundwater flow modelling phase of the project and preparation of information necessary to obtain permits as required for additional mine expansions.
- Mr. Mark Birch, a registered Geologist/Hydrogeologist in Washington, Registration #1308 carried out a detailed review of the geologic and hydrogeologic setting of the project area, and review of existing geological,

hydrogeological, and geophysical information. Mr. Birch provided a review of changes to the mine plan and hydrogeologic model results relative to those changes which were made to the mine plan since the initial groundwater model was developed in 2012.

- Mr. Alan Hull, Principal Seismic Hazard specialist, visited the site in the fall of 2013 and provided a review of the existing probabilistic seismic hazard assessment, and conducted a site specific study of faults in the area, specifically for the Zirayet Tepe fault in support of confirmation of the seismic design parameters used by the civil design team.

The following Alacer personnel provided specialist input to Mr. Benbow:

- Roy Kim, Vice President Business Development, developed the financial model for the oxide and sulfide case and the oxide only case from which financial metrics were derived.
- Victor Ketcham, Metallurgical General Manager, provided operational and operating cost data used in developing pressure oxidation operating costs.

2.7 Previous Technical Reports

The following technical reports have been filed on the Çöpler Project:

- Watts, Griffis and McQuat Limited, 2003: Update of the Geology and Mineral Resources of the Çöpler Prospect, May 1, 2003.
- Independent Mining Consultants, Inc., 2005: Çöpler Project Resource Estimate Technical Report, October 19, 2005.
- Marek, J.M., Pennstrom, W.J., Reynolds, T., 2006: Technical Report Çöpler Gold Project Feasibility Study, May 30, 2006.
- Marek, J.M., Moores, R.C., Pennstrom, W.J., Reynolds, T., 2007: Technical Report Çöpler Gold Project, March 2, 2007 as amended 30 April 2007.
- Easton, C.L., Pennstrom, W.J., Malhotra, D., Moores, R.C., Marek, J.M., 2008: Çöpler Gold Project East Central Turkey Preliminary Assessment Sulfide Ore Processing, February 4, 2008.
- Marek, J.M., Benbow, R.D., Pennstrom, W.J., 2008: Technical Report Çöpler Gold Project East Central Turkey, December 5, 2008 (Amended and Restated; supersedes 11.07.2008 version).
- Altman, K., Liskowich, M., Mukhopadhyay, D.K., Shoemaker, S.J., 2011: Çöpler Sulfide Expansion Project Pre-Feasibility Study, March 27 2011.
- Altman, K., Bascombe, L., Benbow, R.D., Mach, L., Shoemaker, S.J., 2012: Çöpler Resource Update, Erzincan Province Turkey, March 30 2012.
- Bascombe, L., Swanson, B., Bair, D., Mach, L., Benbow, R.D., and Altman, K., 2013: Technical Report on the Çöpler Mineral Resource Update, Erzincan Province, Turkey, March 28, 2013.
- Bohling, R., Kiel, R., Armstrong, D., Liskowich, M., Parshley, J., Swanson, B., Seibel, G., Parker, H.M., Bascombe, L., 2014: Çöpler Sulfide Expansion Project Feasibility Study, Erzincan Province, Turkey: July 29, 2014.

- Bohling, R., Kiel, R., Birch, R.G., Liskowich, P.Geo., Parshley, J., Marsden, J., Seibel, G., Parker, H.M., Bascombe, L., Benbow, R.D., Statham, S., Francis, J., and Khoury, C., 2015: Çöpler Sulfide Expansion Project Feasibility Update Erzincan Province, Turkey, March 27, 2015.

3.0 RELIANCE ON OTHER EXPERTS

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 Alacer and legal experts retained by Alacer for this information through the following document:

- Biçer, İ., 2016: Mining Title Opinion of Turkish Legal Counsel: letter addressed to Anagold, from the legal firm, Baycan Hukuk Bürosu, April 26, 2016, 8 p.

This information is used in Section 4 of the Report. The information is also used in support of the Mineral Resource estimate in Section 14, the Mineral Reserve estimate in Section 15, and the financial analysis in Section 22.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Çöpler Mine 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 center, İliç, (approximate population 2,600), is about 6 km east of the Çöpler Mine.

Figure 4-1 illustrates the location of the project within the country of Turkey, and indicates the deposit's proximity to surrounding communities.

The Project centroid is situated at about 459,977 E and 4,364,422 N, and has an approximate elevation of 1,161 m amsl.

The mining operation is located 900 m southwest of the İliç district center, 650 m south of the Bahçe neighborhood, 250 m south of the Çöpler village, and 180 m north of the Sabırlı village, and remains within the license areas numbered 847, 49729 and 20067313 (Figure 4-2) which have been granted by the General Directorate of Mining Affairs (GDMA).

Figure 4-1 Location of the Project

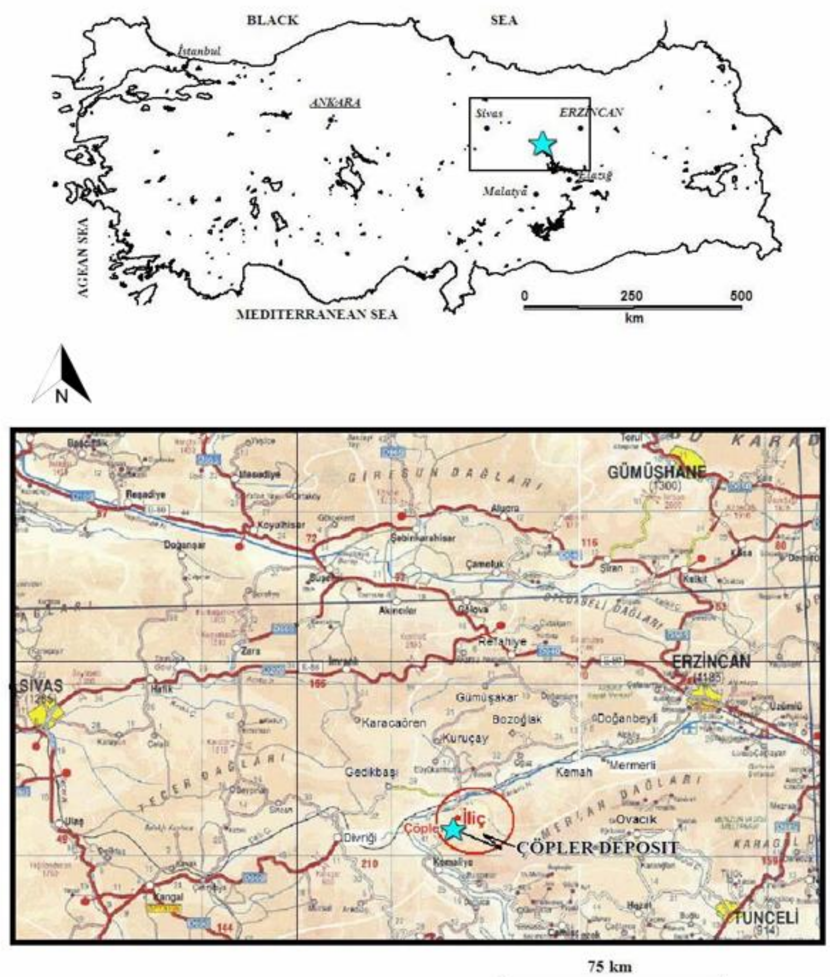


Figure prepared by Alacer, 2010.

Figure 4-2 Çöpler Mine License and Surrounding Licenses (UTM Grid)

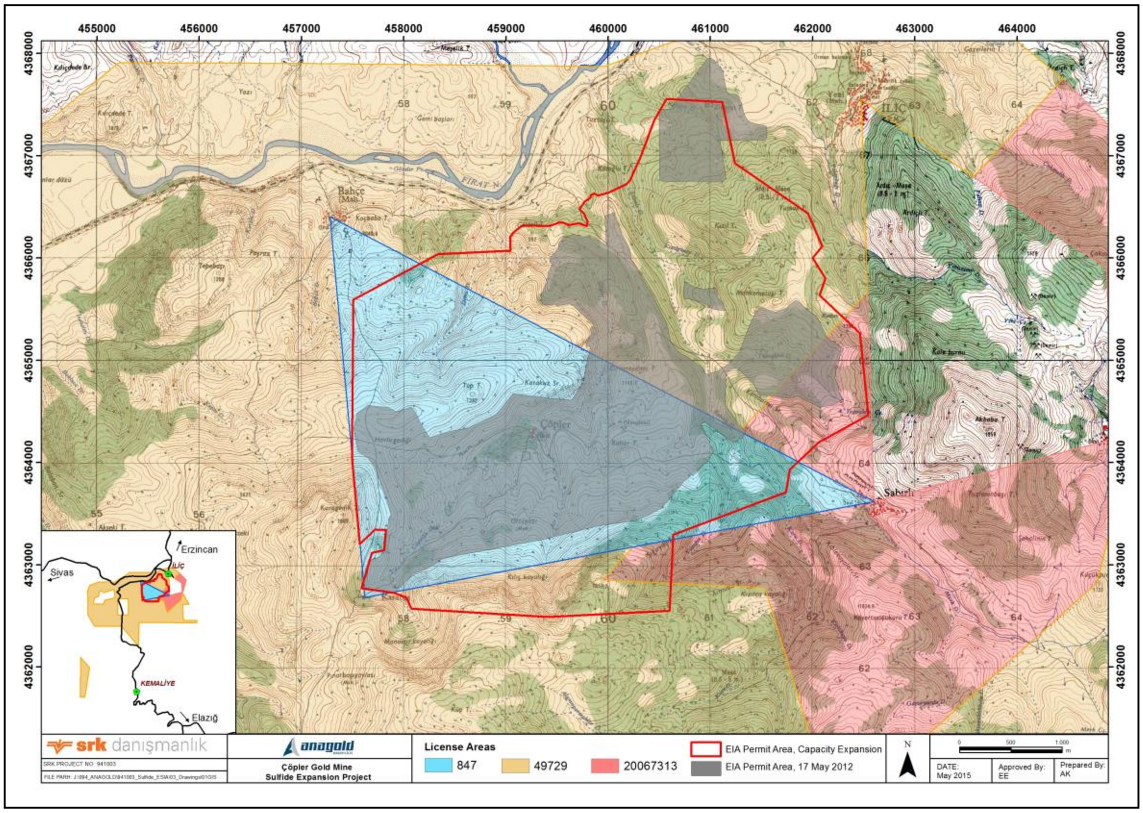


Figure prepared by SRK, 2016.

The currently permitted Environmental Impact Assessment (EIA) boundary incorporates 1,686 ha, whereas the footprint of the mine units covers a combined 969 ha.

4.2 Mineral Tenure

There are seven granted licenses (Table 4-1) covering a combined area of about 16,573 ha. The major license boundaries are shown in Figure 4-3. Mineral title is held in the name of Anagold.

The granted licenses include two borrow pit licenses (76817 and 76818) that fall within the main Çöpler license.

The Çöpler Mine and associated infrastructure are hosted within the triangular-shaped 257/847 concession.

Yakuplu East, Yakuplu North and Yakuplu Main prospects are all on Kartaltepe license 1054. Yakuplu Southeast prospect is on Anagold license 20067313. Bayramdere prospect is on Kartaltepe license 7083. All three of the licenses are operation licenses.

Table 4-1 Granted Licenses

City	Town	Village	License or Permit Type	No	Registration No	Commencement date	Expiry Date	Area (ha)
Erzincan	İliç		Operation License 4. Group (Copper)	847	1027313	06.11.1986	06.11.2026	941.92
Erzincan	İliç	Çöpler	Operation License 4. Group	49729	2384036	08.03.2017	08.03.2017	13747.31
Erzincan	İliç	Sabırlı	Operation License 4. Group	20067313	3095732	16.02.2012	16.02.2022	1184.91
Erzincan	İliç	Ortatepe	Operation License 4. Group (Gold)	50237	2386272	21.03.2008	21.03.2018	600
Erzincan	İliç		Operation License I-b Group (Brick –Tile Clay)	76817	3201587	15.07.2009	15.07.2019	49.32
Erzincan	İliç		Operation License I-b Group (Brick –Tile Clay)	76818	3201588	15.07.2009	15.07.2019	49.09

Figure 4-3 Tenure Layout Plan

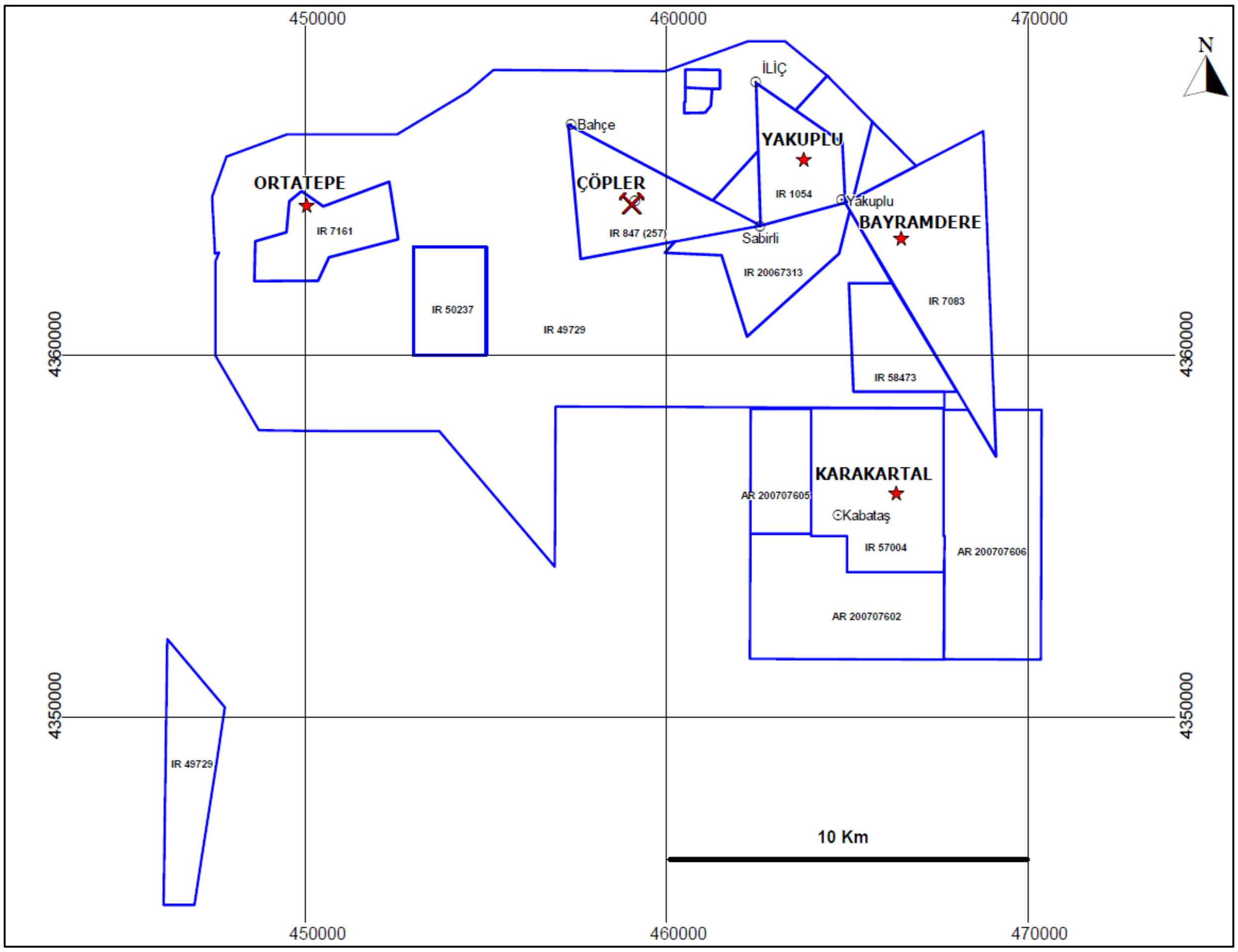


Figure prepared by Anagold, 2016. Black circles are villages.

Anagold was advised that some of the licenses (847; 49729; 50237, 76817, and 76818) may include stone quarry sites within the license area and has agreed not to conduct any activity within such sites. Some licenses (847; 49729; 20067313; 50237, 76817, and 76818) may cover raw material permits that have issued to state institutions and entities and these entities have the right to conduct activities as per such permits. Two licenses (49729 and 76817) require specific permits to operate as they fall within the Bağiştaş 1 Hydro electrical Power Plant Project Area.

Legal opinion provided to Alacer indicates that charges and administrative expenses due to the Turkish Ministry of Energy and Natural Resources, Directorate General of Mining Affairs (MIGEM) have been paid, and all licenses were in good standing as of the 26 April, 2016 legal opinion date.

Anagold has also been issued with the mine operating permits shown in Table 4-2.

Table 4-2 Operating Permits

City	Town	Village	License or Permit Type	No	Registration No	Commencement Date	Expiry Date	Permit Area (ha)
Erzincan	İliç		Operation Permit 4. Group (Copper-Gold- Silver- Mercury)	847	1027313	06.11.1986	06.11.2026	941.92
Erzincan	İliç		Operation Permit 4. Group (Manganese)	847	1027313	06.11.1986	06.11.2026	941.92
Erzincan	İliç	Çöpler	Operation Permit 4. Group (Copper-Gold- Silver- Molymbdenu)	49729	2384036	08.03.2017	08.03.2017	589.67
Erzincan	İliç	Sabırlı	Operation Permit 4. Group (Gold)	20067313	3095732	16.02.2012	16.02.2022	91.28
Erzincan	İliç	Ortatepe	Operation Permit 4. Group (Gold)	50237	2386272	21.03.2008	21.03.2018	18.07
Erzincan	İliç		Operation Permit I-b Group (Brick –Tile Clay)	76817	3201587	15.07.2009	15.07.2019	6.68
Erzincan	Iliç		Operation Permit I-b Group (Brick –Tile Clay)	76818	3201588	15.07.2009	15.07.2019	49.09

4.3 Surface Rights

Alacer currently holds sufficient surface rights to support the heap leach mining operations and the proposed Sulfide Expansion Project.

4.4 Royalties and Encumbrances

See Section 22.0 for more information on royalties and other financial impacts to the property. 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.

4.5 Environmental Liabilities

There are no known existing environmental liabilities for the Çöpler Project, except for Alacer's obligation for ultimate reclamation and closure. See Section 20.0 for information on closure and associated costs.

4.6 Permits

The EIA permitting process for the Sulfide Expansion Project started on April 07, 2014 and ended by receiving the "EIA Positive Statement" on December 24, 2014. The EIA permit serves as a construction permit. The forestry land use permits for the construction of the Çöpler Sulfide Expansion Project were obtained on 20 April, 2016 and the remaining permits required will be obtained by upgrading/amending the existing oxide operation permits. Operational environmental permits are obtained within two years of the start of mine operation. The private land acquisition and pasture land permitting processes continues and most of the operational permits have are already been obtained.

The EIA permitting for the Çöpler gold mine for the oxide ore was completed in April 2008 with the issuance of an EIA positive certificate. All of the operation permits have already been obtained for the oxide resources. These are: explosive storage permit, permit for water abstraction from groundwater sources, EIA positive 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 started on April 7, 2014 and was completed with the receipt of an “EIA Positive Statement” on December 24, 2014. In addition to an EIA approval, other permits required for the Sulfide Expansion Project involve an expanded workplace opening permit, additional operating permits and land acquisition permits for forest areas and pasturelands, etc.

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

- EIA permit dated April 10, 2012 for the operation of mobile crushing plant.
- EIA permit dated May 17, 2012 for the capacity expansion involving:
    - (i) increasing operation rate to 23,500 tpd.
    - (ii) increasing Çöpler WRSA footprint area.
    - (iii) adding a sulfidization-acidification-recovery-thickening (SART) plant to the process in order to decrease the cyanide consumption due to the high copper content of the ore.

The EIA positive decision provides the legal permit to construct the Sulfide Expansion Project.

Other permits required for the Sulfides Expansion Project are:

- A workplace opening permit needs to be obtained from provincial directorate prior to the startup of the business.
- The EIA permit acts as a temporary permit for the construction of the mine. Within one year after start of the operation, however, a Temporary Environmental Operation License application has to be made to the Turkish Ministry of Environment and Urban Planning (MEUP). Following the application, compliance testing via an accredited laboratory is conducted for mine emissions and discharges and then “Final Environmental Operation License” is issued. The environmental licenses are managed by the MEUP and cover all aspects of the environment, including, but not limited, to waste water discharge, air pollutant emissions, noise, solid waste, hazardous waste.

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 Report.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Çöpler Mine is accessed from the main paved highway between Erzincan and Kemaliye. The highway passes 3 km north of the nearby village of İliç where it crosses the Karasu River via a bridge. From İliç there is an additional 4.5 km of graded dirt road to reach the Çöpler mining site. Work has been completed to upgrade sections of road from the bridge to just east of the İliç railway station and to construct a road bypassing İliç to the Project site. This roadwork provides improved access to the mine and for construction equipment.

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 (refer to Figure 4-3). The railway line connects the site with Ankara and the west as well as with sea ports to the north on the Black Sea, and to the south on the Mediterranean Sea. Overnight passenger sleeper cars are available to and from Ankara.

The reservoirs of the Bağıştaş I & II Hydroelectric Power Plants (HEPP) are 350 m and 1,800 m away from the Çöpler mine site, respectively. The embankment of Bağıştaş I Dam covered some portion of the existing highway, railroad, and railroad station so these were relocated before dam construction was completed. Construction routes for the railroad and highway were located between the new Çöpler village and mine site. The current mine access road will be connected to the relocated road. 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 Erzincan, Erzurum, Malatya, Elazığ and Sivas. Driving from these cities to the Project site takes about 2 to 4 hours on paved highways. Driving from Ankara to the site takes about 8 hours.

The process plant area is essentially bounded by longitudinal lines E460,500 to E 460,000 (east to west) and latitude lines N4,366,000 to N4,364,750 (north to south).

5.2 Local Resources and Infrastructure

The town of İliç has a population of approximately 2,600 inhabitants and is located 6 km northeast of the site. The town has a hospital, schools, municipal offices, 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, along the Karasu River, some wheat farming. Additionally, there is some light manufacturing and grain milling performed in İliç.

The workforce for the Alacer exploration programs has primarily consisted of residents drawn from the local communities of Çöpler, İliç, and Sabırlı.

Turkish telecommunications are good and up to European standards. High speed, fiber-optic internet access is installed at the mine site.

Electrical power at 380V, 50Hz, is available in İliç and at the mine site, but the line capacity is not sufficient to handle the industrial loads required by the Sulfide Expansion

Project. A 40 km, 154kV power line from the substation at Divriği to the mine site has been installed, is currently operational and will provide sufficient electrical supply.

Sufficient water supply exists to support the heap leach operation. Ground water resources have been identified about 2 km north of the Project site near the Karasu River and two production water wells have been constructed as a replacement for the production wells which remained under the highway and railway relocation routes due to the constructions of the Bağıştaş I Dam and HEPP and the Bağıştaş II regulator on the Karasu River. The reservoir for the Bağıştaş I Dam has risen to within 35 m to 50 m from the perimeter of the new Çöpler settlement.

Fresh water is being supplied by three existing wells to the site at a rate of 100 L/sec. Additional wells can be drilled if required to support the Sulfide Expansion Project.

Further information on Project infrastructure is included in Section 18, and 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 FS report.

The Project area is located in the Eastern Anatolia geographical district of Turkey. The climate is typically continental with wet, cold winters and dry, hot summers. In winter, the night-time temperature can drop to minus 25º C although the average is usually a few degrees below freezing. The July temperature frequently exceeds 40º 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. The average yearly precipitation in the region was recorded at 366.6 mm, with a maximum of 610 mm and a minimum of 210 mm. Snowfall is common during the mid-November to February period, but with little accumulation, if any. Snow depth assessments are based on the Divriği State Meteorological weather station, located 41 km west of the Project site, which shows maximum snow pack depths at about 200 mm for 1985.

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 station in 2004 ranges from 15 to 25 m/s with variable directions mainly from the north, south and east.

Mining operations are currently conducted year-round. It is expected that the Sulfide Expansion Project will also have year-round operations.

5.4 Physiography

The Çöpler Mine is located in a broad east-west oriented valley at an altitude of 1,100 to 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 deposit. These mountains are at the western end of the Munzur range that rises to more than 3,300 m between Ovacık and Kemah.

The region is sparsely vegetated with semi-arid brush and scrub trees.

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

- Latitude: 39º 25’ North.
- Longitude: 38º 32’ East
- Elevation: 1,150 m amsl.
- Frost depth: 50 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: +40ºC.
- Minimum design temperature: - 25ºC.
- Annual rainfall: 367 mm.
- Maximum snowfall depth: 500 mm (estimated).
- Design maximum rainfall: 24 hours, 76 mm.

The surface rights sufficiency for the Project is discussed in Section 4.

6.0 HISTORY

6.1 General History

The Çöpler area has seen gold and silver mining that dates back at least to Roman times, and possibly earlier, with historic bullion production estimated at about 50,000 ounces of gold. A copper-rich slag pile of approximately 25,000 t is located at the western edge of the district and is believed to be waste from ancient bullion production. Although the district contains copper mineralization, 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 byproduct copper production.

The Turkish Geological Survey (MTA) carried out regional exploration work in the early 1960s that was predominately confined to mapping. During 1964, a local Turkish company started manganese mining that produced about 73,000 t of manganese ore until closing in 1973. Unimangan acquired the property in January 1979 and restarted manganese production the same year producing about 1,000 to 5,000 tpa of ore until 1992. Total production from the Manganese Mine Zone, during this period, is estimated to have been 15,000 t of ore at a grade of between 43%and 51% Mn.

The Çöpler prospect was first identified by the predecessor company of Alacer, Anatolia Minerals Development Ltd (Anatolia; a Rio Tinto subsidiary) in 1998 as part of a literature review of Turkish mineral properties and as a follow-up of a gossan investigation program in the district. In September 1998, Anatolia identified several porphyry-style gold-copper prospects in east-central Turkey and applied for an exploration license totaling over 100,000 ha covering 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 mineralized granodiorite, intruded metasediments 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, Anatolia and Rio Tinto explored and drilled the Çöpler deposits and developed Mineral Resource estimates in three mineralized zones: the Main, Manganese, and Marble Zones. In January 2004, Anatolia acquired Rio Tinto's joint venture interest and the interest of Unimangan. The property was under Anatolia's sole control until the joint venture with Lydia was executed in August 2009.

Anatolia merged with Avoca Resources Limited, an Australian company, to form Alacer Gold Corporation in February of 2010.

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

6.2 Exploration and Development History

Work completed by Alacer to date 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 polarization (IP), time domain IP and controlled source audio-frequency magneto-tellurics (CSAMT) surveys; a regional helicopter-borne geophysical survey; RC and core (DD) drill programs, acquisition of satellite imagery, mining technical studies, geotechnical and hydrogeological studies, environmental baseline studies, studies in support of Project permitting, metallurgical testwork and metallurgical studies, and condemnation evaluations.

Exploration of the Çöpler area has been conducted by Anatolia and then Alacer since September 1998. The principal exploration technique has been RC and diamond core drilling, conducted in a number of campaigns starting in 2000. Initially, exploration was directed at evaluating the economic potential of the near-surface oxide mineralization for the recovery of gold by either heap leaching or conventional milling techniques. This program was successful in demonstrating that heap leaching was commercially viable. Gold production commenced in December 2010, and gold is presently being produced from the property by this method.

The Sulfide Expansion Project is planned to produce gold via POX methods.

6.3 Production History

Modern gold production at the Çöpler mine commenced in 2010 as a heap leach operation producing an average of 19,500 tonnes per day. As of January 1, 2016 over 35 Mt of oxide ore at an average grade of 1.57 g/t has been delivered to the heap leach pad for gold recovery. The Çöpler mine has produced over 1 Moz since 2010 of which 890,000 oz are attributable to Alacer. Table 6-1 details the annual production figures for the Çöpler Mine.

Table 6-1 Annual Production Summary for Çöpler Gold Mine

	Total	2009	2010	2011	2012	2013	2014	2015
Oxide Ore Mined - Tonnes	**35,327,447**	19,504	1,560,444	7,443,854	7,036,221	6,673,520	6,474,401	6,119,503
Sulfide Ore Mined - Tonnes	**5,143,632**	-	-	-	190,024	1,345,882	1,788,127	1,819,599
Waste Mined - Tonnes	**106,379,851**	142,754	8,317,871	11,371,206	18,071,316	20,683,286	22,959,588	24,833,830
Oxide Ore Grade (g/t)	**1.57**	-	1.05	1.54	1.61	1.90	1.69	1.21
Sulfide Ore Grade (g/t)	**3.71**	-	-	-	4.16	4.94	3.72	2.75
Gold Ounces Produced - Total	**1,078,341**		512	185,418	188,756	271,063	227,927	204,665

When encountered, sulfide ore is currently stockpiled for processing in the POX circuit once that circuit has been commissioned.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Geological Setting

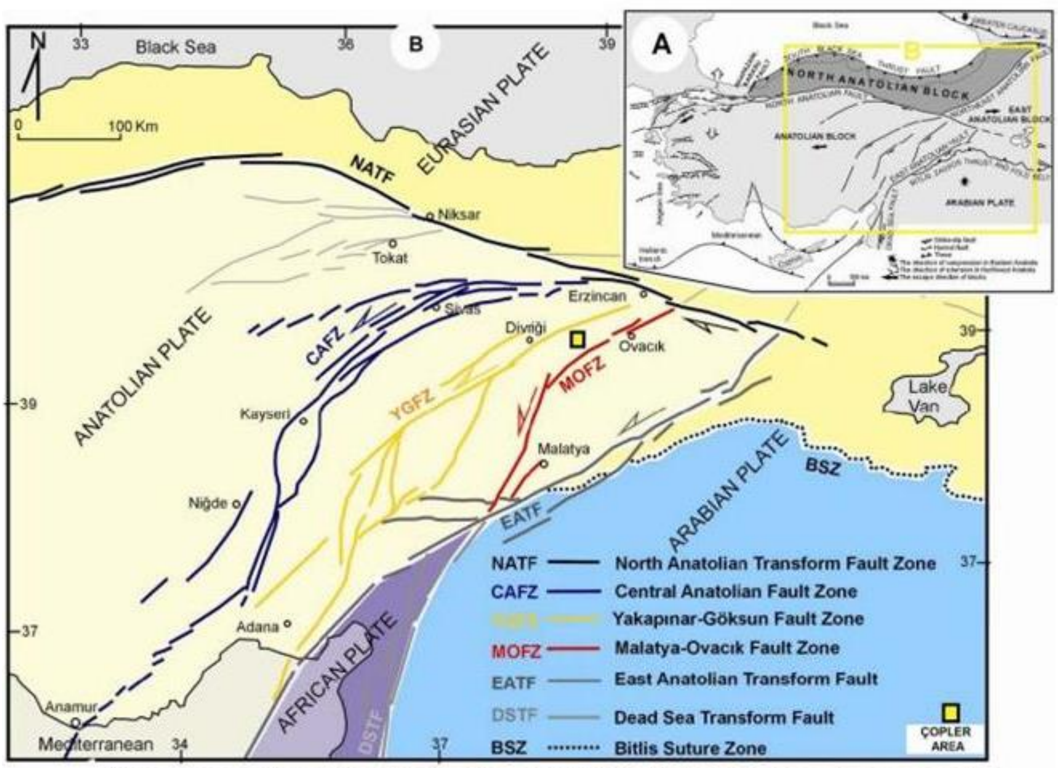
The following discussion of the geology and mineralization of the Çöpler deposit was derived principally from a report completed in August 2010 by Firuz Alizade, formerly Vice President Exploration of Alacer and current General Manager of Polimetal Madencilik, an exploration JV between Alacer and Lydia.

7.1.1 Regional Geology

The Çöpler Mine is located near the north margin of a complex collision zone lying between the Pontide Belt/North Anatolian Fault, the Arabian Plate and the East Anatolian Fault (Figure 7-1). 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.

The Project location is highlighted by the small yellow square between Divriği and Ovacik in Figure 7-1.

Figure 7-1 Structural Setting of Anatolia



Major neotectonic elements of Anatolia (from Elmas 2003). **B:** Simplified neotectonic map showing major tectonic escape-induced structures in eastern half of Turkey and adjacent areas (from Koçyiğit and Erol, 2001)

7.1.2 Property Geology

The Çöpler mining area is centered on a composite diorite to monzonite porphyry stock that has been emplaced into metasediments and limestone-marbles of the Munzur Formation. The intrusive unit is believed to be late Cretaceous to Eocene in age. The lower Permian, limestone turbidite sequence has been metamorphosed to metasediments and is overlain by massive porcellanous limestone that has been altered close to the intrusion by both contact metamorphism and hydrothermal solutions. The relationship between all three principal rock types, which is illustrated in Figure 7-2, is often complex and has not yet been fully defined.

Figure 7-2 Contact between Munzur Formation Limestone and Çöpler Intrusive-Metasediment Complex, Looking West

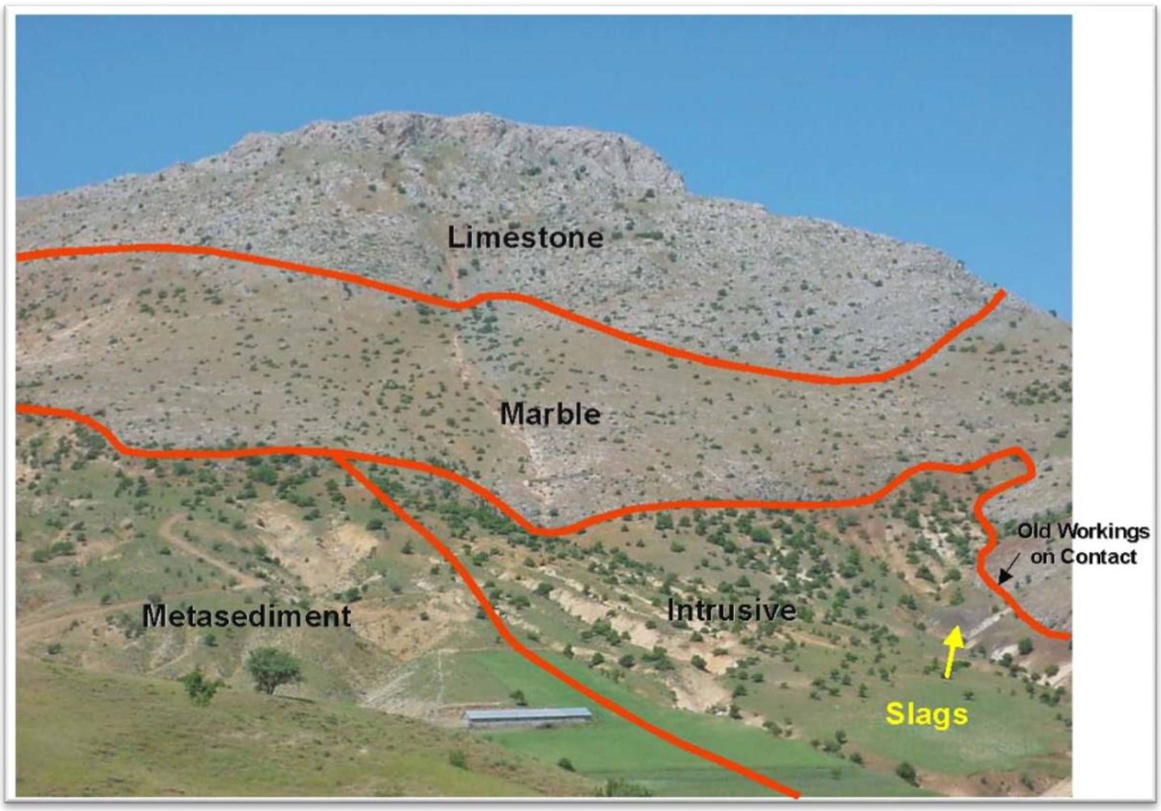


Figure sourced from Bohling et al., 2014.

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 Mine intrusion. 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 argillically-altered rock are evident on a centimeter scale in outcrop and drill core.

It is possible that there are several intrusive phases but, if so, they have been obscured by alteration, comprising either potassic in the porphyry core or argillic and advanced argillic in association with the epithermal mineralization. The age of the Çöpler intrusion is thought to be Eocene. The evidence for this is not conclusive, although Eocene conglomerates on the northeast side of the property show a similar style of alteration.

The primary control on the location of the Çöpler intrusion appears to have been the metasediment-carbonate contact. The contact of the Çöpler intrusion has a roughly rectilinear shape, suggesting control by pre-existing east-northeast trending faults, and by a set of north-northwest trending fractures. The north-northwest 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-northwest trending fracture control referred to above.

A pronounced ground magnetic anomaly is centered on the core of the porphyry, that has been modeled as a stock-like intrusion dipping steeply towards the south, and reflects the potassically-altered core of the porphyry system. In addition, there are a number of dikes and intrusive apophyses; most notably, a brecciated and strongly clay-altered intrusion centered on the Manganese Mine Zone.

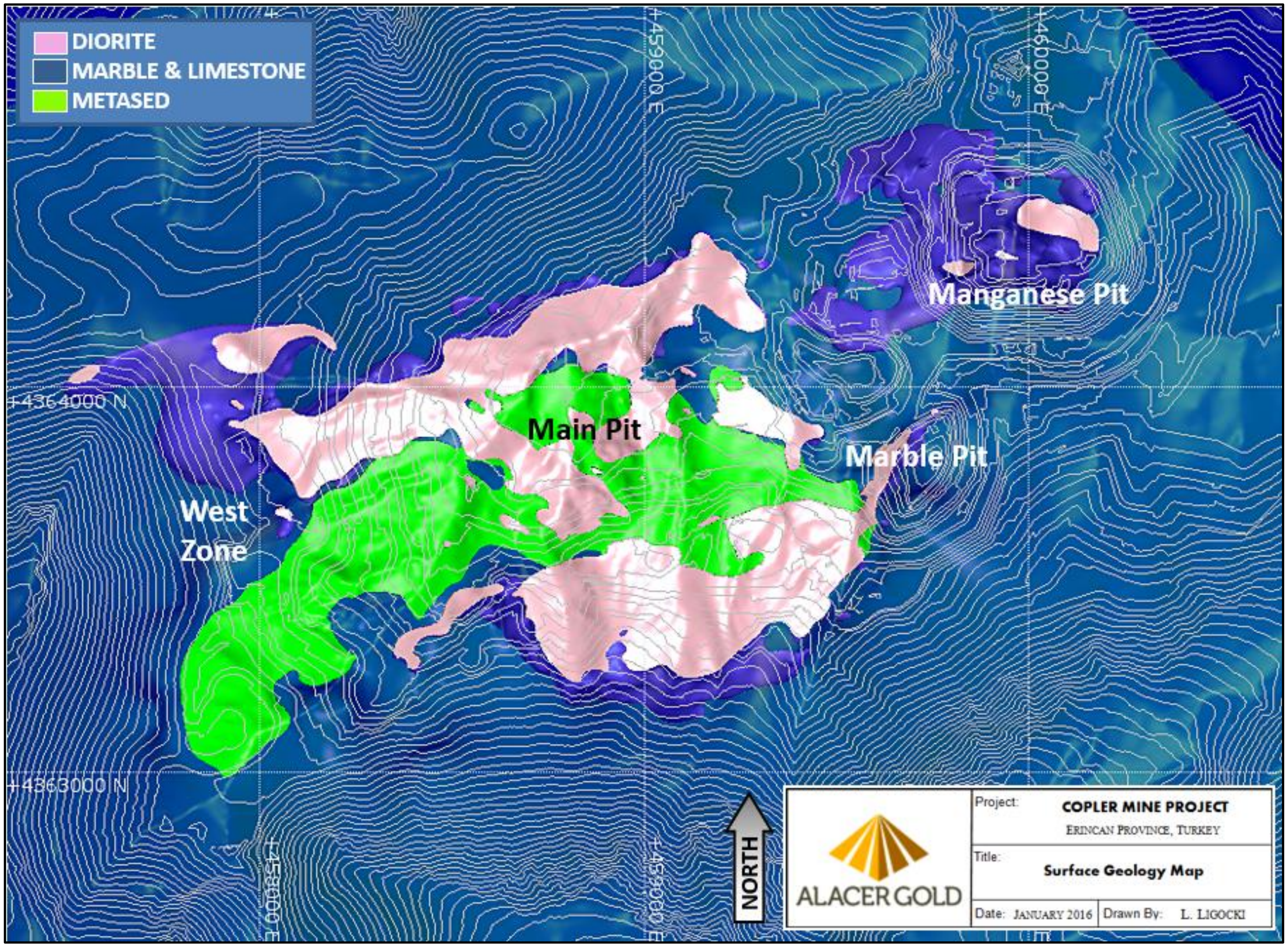
Two parallel east-northeast striking faults spaced roughly 300 to 500 m apart cross the project area, and are identified as the Çöpler North and Çöpler South faults. The faults transect all rock units in the Project area and may have provided the locus for the intrusive events.

The Çöpler North fault is believed to be a low angle thrust fault passing through the Manganese Mine Zone; however, it can only be traced for 200 m to the west-southwest, where it is lost in the marble near the old Çöpler village. Further to the west the fault is expressed as:

- An inferred faulted contact between metasediment and marble northwest of the old Çöpler Village.
- As a straight metasediment-intrusion contact trending west-southwest.
- As a prominent lineament in marble on the southwest side of the intrusion.

The Çöpler South fault is a high-angle fault forming the metasediment/marble contact southeast of the old Çöpler village, which can be traced to the east-northeast through the northern part of the Marble Contact Zone. Northeast and northwest-striking faults exist between the two major faults reflecting the regional stress field that provided further ground preparation for hydrothermal mineralization. There are diorite intrusive units below surface within the Manganese Mine and Marble Contact Zones that do not crop out on the surface map shown in Figure 7-3. The mineralization within those zones is proximal to and associated with the diorite intrusions. Additional contact metamorphic rocks in the form of jasperoids occur locally at contacts between the intrusions and calc-silicates.

Figure 7-3 Local Geology (Alacer Geological Map)



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

7.2 Mineralization

Epithermal gold mineralization at Çöpler occurs within structurally-controlled zones of stockwork and sheeted veins hosted by a Tertiary diorite intrusion and an older metasediment complex, and as contact-type mineralization along the intrusive-metasediment fault contact with the Munzur Formation limestones. The epithermal mineralization may be related to porphyry copper-style mineralization that has been intersected by several of the drill holes.

Gold mineralization at Çöpler exhibits five principal styles:

1. Stockwork quartz-veined metasedimentary rocks and diorite with disseminated marcasite, pyrite, arsenopyrite and tennantite-tetrahedrite. Oxidation has resulted in the formation of goethitic/jarosite assemblages hosting fine-grained gold (Main Zone).
2. Clay-altered, brecciated and carbonatized diorite with rhodochrosite veinlets and disseminated marcasite, pyrite, realgar, orpiment, tennantite-tetrahedrite, other sulfosalts, sphalerite and galena (Manganese Mine Zone).
3. Massive marcasite-pyrite replacement bodies along marble and faulted contacts (Main Zone, Main Zone East, Main Zone West, Marble Contact Zone, West Zone and Manganese Mine Zone).
4. Massive jarositic gossan (Marble Contact Zone, Main Zone Contacts).
5. Massive manganese oxide (Manganese Mine Zone).

Oxidation of the above mineralization types has resulted in the formation of gossans, massive manganese oxide and goethitic/jarositic assemblages hosting fine-grained free gold.

The Çöpler mining area can be sub-divided into six deposits. The mineralization occurrences within each area are summarized in the following subsections.

7.2.1 Main Zone

The Main Zone lies in the west portion of the Project area and occupies a footprint of approximately 750 m north to south by 1,000 m east to west. Typical depths of mineralization range from surface to +200 m in depth. Disseminated quartz-pyrite-arsenopyrite epithermal veinlets are primarily hosted in diorite and metasediments with some marble-hosted mineralization on the eastern margin of the zone. Oxidation has occurred, and oxide mineralization occurs from near surface to depths of approximately 40 m, with the thickest development over ridges and thinning in the intervening valleys.

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

7.2.2 Manganese Mine Zone

The Manganese Mine Zone occupies the eastern end of the Çöpler mining area. The zone is approximately 650 m wide from north to south by approximately

650 m in the east to west direction. The pre-mining surface expression of this area consisted predominately of marble. A moderately-sized intrusion of diorite occurs sub-surface. A large proportion of the Manganese Mine Zone mineralization is associated with the contact between this diorite and the surrounding marble. Mineralization ranges from surface to approximately 400 m deep.

Free gold mineralization occurs in the marble with minimal associated sulfides. Disseminated quartz-sulfide mineralization occurs in clay-altered and brecciated diorites as well as locally carbonate-altered diorite. Moderate volumes of massive sulfide pyrite mineralization occur within the Manganese Mine Zone. It appears that “leachable” mineralization is a combination of free gold in marble and supergene oxidized mineralization in both marble and diorite. Leachable oxide mineralization occurs to over 200 m in depth.

7.2.3 Main Zone East

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

7.2.4 Marble Contact Zone

The Marble Contact Zone occurs in the southeastern portion of the Project area and is associated with a northeast-striking fault contact between marble on the east and metasediments and intrusions on 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 mineralizing fluid flow has been considerable. The width of the Marble Contact Zone is approximately 350 m, and the strike length is 300 m in an east-northeasterly direction. The depth of mineralization ranges from surface to approximately 160 m.

Mineralization occurs as both disseminated sulfides in veinlets and massive sulfide along the marble contact. Oxidation has occurred along the northeast structure resulting in greater depths of oxidized mineralization than in the Main Zone.

7.2.5 West Zone

The West Zone occupies the westernmost portion of the Project area and is located at the contact between the basement metasedimentary rocks and the overlying limestone, where a large-scale northeast-trending fault is located. Mineralization is present within veinlets containing disseminated sulfides, massive sulfide and oxidized gossan. The West Zone has a strike length of approximately 700 m in a northeasterly direction and is approximately 150 m wide. Multiple, narrow, mineralized zones are present sub-parallel to the faulted contact, and occur to a depth of approximately 150 m below surface.

7.2.6 Main Zone West

Main Zone West is located in the northwest corner of the Project area at the contact between diorite, marble and the basement metasedimentary units. The mineralization is hosted within narrow gossans located at the contact, and in sub-parallel veinlets containing disseminated sulfides within the marble and metasedimentary rocks. Main Zone West has a strike length of approximately 750 m and is approximately 75 m wide.

7.3 Structural Geology

Northeast to east-trending structures dominate the Çöpler Project. The variable northeast-trending Çöpler North and South faults are the most important of the structures crossing the entire property. At least three jasperoid bodies have formed along the Çöpler South Fault, and ground preparation for both the eastern stockwork quartz veinlet zone (in the metasedimentary units) and the western stockwork quartz veinlet zone (in diorite porphyry) is most likely related to the fault.

Numerous small jasperoid bodies are related to an east-west lineament that intersects the Çöpler fault. There is at least one other fault sub-parallel to the Çöpler North Fault that controls manganese mineralization approximately 1 km northeast of Çöpler. Copper oxide mineralization in granodiorite porphyry and quartz monzonite porphyry in the northwest corner of the prospect appears to be related to shear zones on this northwesterly trend.

7.4 Prospects and Targets

Primary targets explored for near surface oxide gold potential around the Çöpler mine include the Yakuplu and Bayramdere prospects, Figure 7-4. These prospects are located approximately 6 km northeast from Çöpler mine operations, and are at various stages of exploration evaluation. Demirmağara is a gold/copper prospect located about 7 km southwest of the Çöpler mine. No current exploration activity is planned for Demirmağara.

Provisional geological models have been constructed for Yakuplu East, Yakuplu Southeast and the Bayramdere prospects, based on 2015 drill program results. Yakuplu North was drilled in 2015 and continues to be evaluated in 2016. Data collected to date include magnetic geophysical surveys, surface and wall mapping, rock and soil sampling, channel sampling and drilling.

Projects include:

- Yakuplu: 80:20 or 50:50 ownership with Kartaltepe Madencilik depending upon location. The geology includes ophiolites, recrystallized limestone and diorite intrusions. Gold mineralization is associated with various stages of faulting cross-cutting and juxtaposing ophiolites, diorites and limestones. Gossan bodies occur near-surface and vary in thickness. Gold mineralization is often within, and proximal to, the gossan bodies. The area contains several small open pits from historic iron ore mining. The current primary metal of interest is gold with anomalous copper.
- Demirmağara: 80:20 ownership. A gold-copper zone has been identified along structures. Gold mineralization appears within thin fractures hosted by fine-grained diorite. Copper oxide mineralization is represented by malachite and azurite. The diorite contains disseminated pyrite without copper. A total of 26 drill holes for 2,909.2 m has been collected from a number of separate targets across the Demirmağara prospect. Demirmağara is the least explored prospect in terms of oxide potential.

- Bayramdere: 50:50 ownership. Gold mineralization has been identified along limestone, ophiolite and diorite contact zones. A total of 104 drill holes for a total of 10,039 m of drilling has been completed. As with Yakuplu, gold mineralization is associated with, but not exclusive to, gossans. Mineralization has been defined in a number of flat-stacked horizons that have been historically mined for iron ore where they are exposed in the hillside. Potential remains to extend the deposit westwards, with drilling in 2014 establishing the limits to the mineralization to the north, east and south.

Figure 7-4 Surrounding Çöpler Prospects

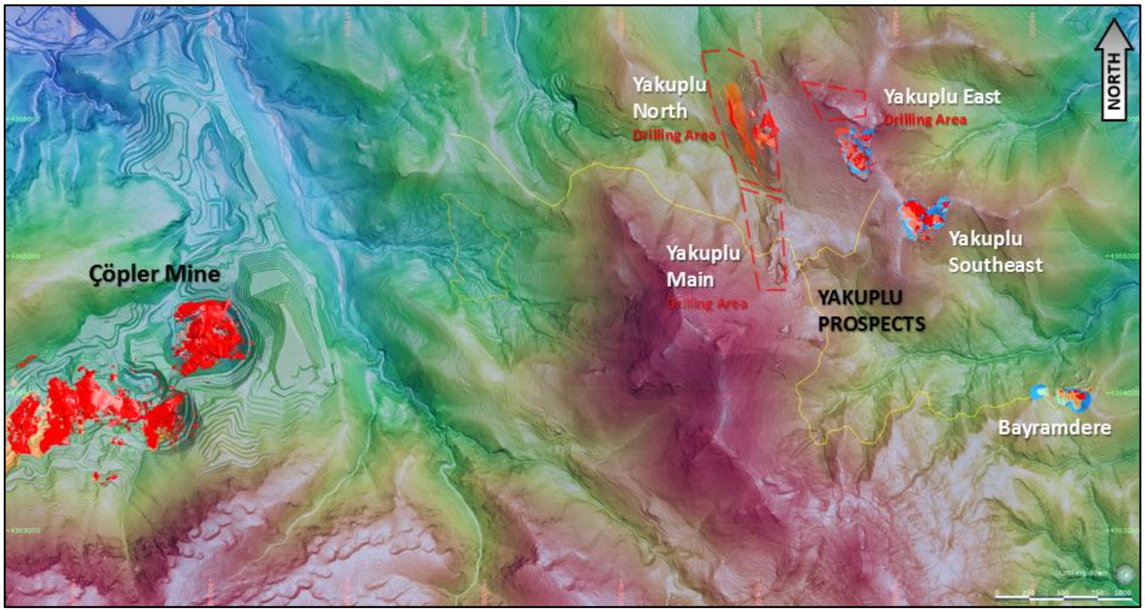


Figure prepared by Anagold, 2016.

Additional drilling and project work continues with further testing of oxide potential at Yakuplu. The work plan will include both diamond and RC drilling containing:

- A mix of shallow RC exploration.
- Shallow diamond development drilling.
- Deeper RC precollar with diamond tail drilling.

If mineralization can be identified that could support Mineral Resource estimation, this material could represent supplementary oxide feed for the Çöpler Mine.

7.5 Hydrogeology

The following discussion of the hydrogeology of the Çöpler mine area is based on the hydrogeological report by Golder that was completed in September 2013 (Golder, 2013b).

Golder conducted a hydrogeological investigation of the Sulfide Expansion Project area in 2012-2013 with the investigation and supporting modeling efforts designed to advance the current understanding of the groundwater system of the mine expansion area. Site selection of additional monitoring wells and piezometers was based on the proposed facility designs and locations in mid-2012, inferred geologic controls (bedrock fracture and fault complexes), and both up-gradient and down-gradient positions for the proposed mining complex. The discussion below summarizes the earlier work, and provides up to date information from the investigations.

7.5.1 Existing Data Evaluation, Field Investigation, Hydrogeologic Conceptual Model Update

Hydrogeologic investigations for the Çöpler site have been conducted by prior investigators over the past several years. These studies were reviewed during Golder's hydrogeologic investigation. During the field investigation phase of Golder's program, particular attention was paid to regional and local fault systems, limestone and marble potential karst development, hydrothermal and supergene alteration assemblages derived by the mineralizing systems, and the construction and development of the Bağıştaş I Dam and reservoir.

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

- Munzur Limestone: Gray to blue-gray, fine-grained to recrystallized 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.
- Metasediments: 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. Serpentinaztion 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 argillites 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.

7.5.2 Monitoring Well Installation Program

The 2012 monitoring well installation program was designed to provide groundwater level data, aquifer characteristics, and general lithology permeability values. The program consisted of the installation of 13 monitoring wells, as shown in Table 7-1.

Table 7-1 Listing of the Groundwater Monitoring Wells (GMW)

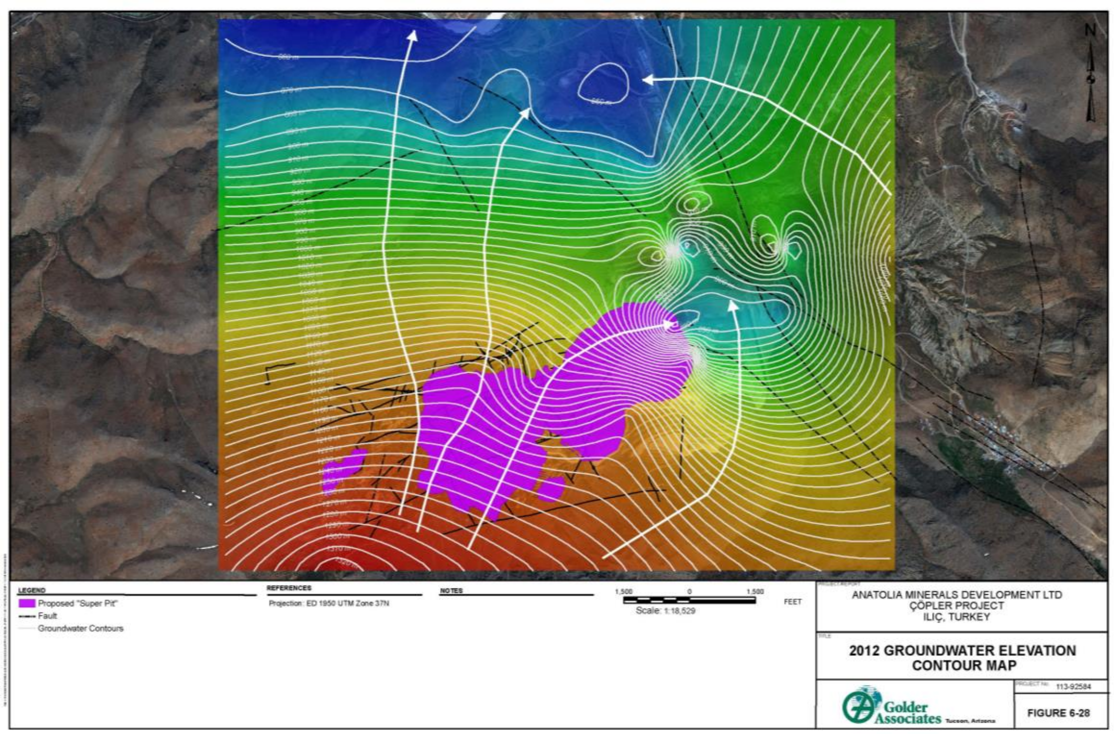
Well ID	Depth (m)	Formation	Location	Aim
GMW-02	150	Limestone	Downstream of TSF 1	Aquifer characterization
GMW-03	122	Alluvium, Granodiorite	Downstream Sabırlı Creek Valley	Aquifer characterization, replaced well WM-14
GMW-05	274	Limestone	Southeast of Open pit	Reaching Pit Bottom
GMW-09	55	Limestone	Downstream Çöpler Creek	Limestone characterization
GMW-10	384	Metasediments	Southwest of Super Pit, upstream	Metasediments characterization
GMW-13	202	Diorite	Inside Super Pit	Diorite characterization
GMW-14	198	Limestone, Diorite	Inside Super Pit	Reaching Pit Bottom, characterization
GMW-16	102	Limestone	North of Mine Complex, Near Karasu River	Limestone characterization
GMW-21	438	Limestone	South of current leach pad	
GMW-24	67	Limestone	Downstream Çöpler Creek	Replacing GMW-09, Limestone characterization
GMW-25	120	Limestone	Downstream North Waste Dump, Çöpler Creek	Replacing WM-03, Limestone characterization
GMW-30	270	Limestone	Inside Manganese Pit	Reaching Pit Bottom
GMW-31	138	Conglomerate, Fault zone, Limestone	Sabırlı Creek	WM-08 Nested well, characterization, Sabırlı fault investigation

Most monitoring wells were designed for aquifer testing and therefore are not typical groundwater monitoring well designs. The aquifer testing was conducted in seven new and six existing, retrofitted monitoring wells. Table 7-2 contains the monitoring well testing descriptions. Aquifer test data was analyzed with Aqtesolv and HydroBench to derive hydraulic conductivity and aquifer transmissivity estimates. These data were incorporated into the groundwater flow model. Groundwater elevation data collected from these new monitoring wells and additional recent groundwater elevations from existing wells were used to generate a groundwater elevation map, Figure 7-5. The general groundwater gradient is from the south to the north with the Karasu River as the major receiving body of water. Three additional monitoring wells will be added to allow monitoring in the vicinity of the planned TSF, with two monitoring wells installed downgradient and one monitoring well installed up gradient as required by the regulations. The installation of the new monitoring wells is planned for third quarter 2016 in advance of the TSF construction.

Table 7-2 Listing of the Groundwater Monitoring Wells (GMW)

Well ID	Formation	Aquifer Testing
GMW-02	Limestone	Dry
GMW-05	Limestone	Dry
GMW-09	Limestone	Falling head, Rising head
GMW-10	Metasediments	Rising head
GMW-13	Diorite	Falling head, Rising head
GMW-14	Limestone	Slug test
GMW-16	Limestone	Falling head, Rising head
GMW-24	Limestone	Falling head, Rising head
GMW-25	Limestone	25lps discharge rate produced insignificant drawdown.
GMW-30	Limestone	Falling head, Rising head
GMW-31	Sabırlı fault	Dry

Figure 7-5 Groundwater Elevations for the Çöpler Site, November 2012 Data

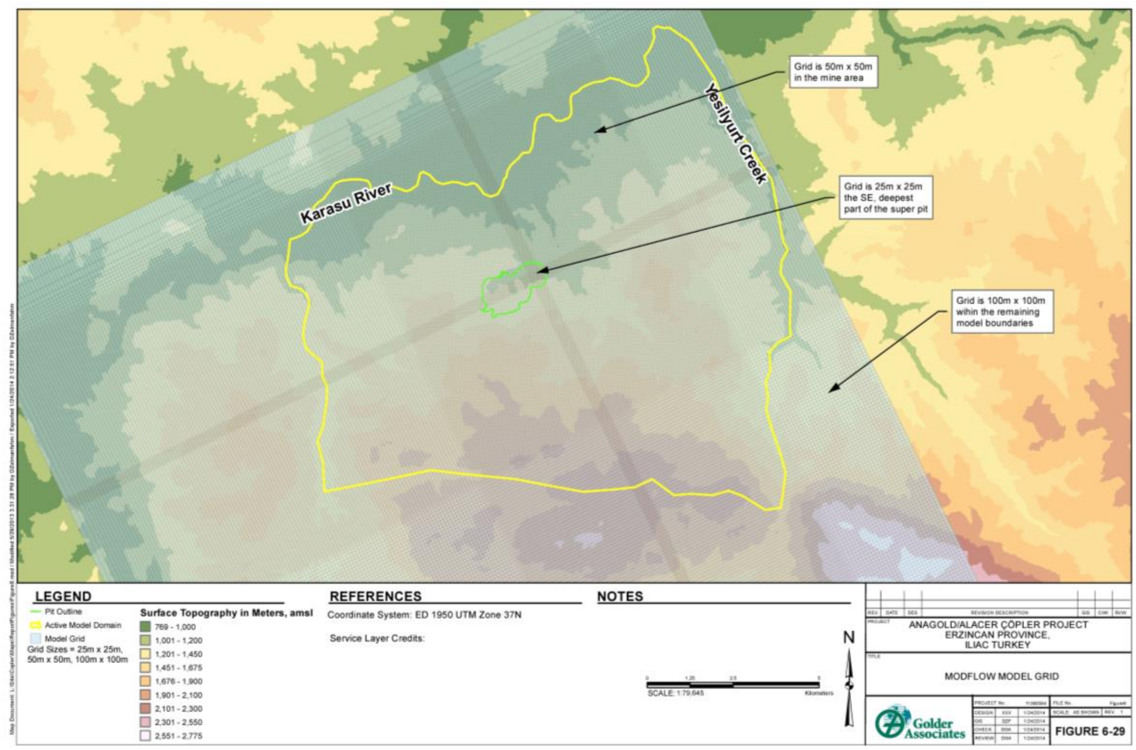


7.5.3 Groundwater Flow Model

The numerical groundwater flow model was completed in September 2013. The modeling software MODFLOW SURFACT was used as the modeling platform in order to deal with the various complexities of the groundwater system including perched groundwater. Golder's hydrogeological study included characterization of the Munzur limestone aquifer in the Çöpler area.

The numerical groundwater flow model was constructed based on the local geology provided by Alacer and published information on the regional geology. Model limits were selected based on hydrologic controls including major faults, hydrologic divides, and the Karasu River. Figure 7-6 depicts the model extent and the model grids.

Figure 7-6 Project Area with Various Model Grids



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 programs, perched groundwater conditions were reported above the clay-altered intrusions. It is anticipated that the perched groundwater is present in restricted areas and the volume of water held in storage as perched groundwater is unknown.

Water balance assumptions are presented in Table 7-3. The reported rates for spring discharges are highly variable and additional measurements were not possible due to the construction of the Bağıştaş I Dam.

A water balance by Ekmekci and Tezcan (2007) of the area estimates that 6% of precipitation recharges the aquifer. The Karasu River is the major perennial surface water feature and the erosional base of the region, and therefore groundwater flow at the Çöpler site is generally toward the river.

Groundwater elevations at the Çöpler site range from 1,328.5 m at Well GMW-10 at the southern end of the site to 864.7 m at Well GMW-09 at the northern end of the site. Observations of cavernous features (karst) during borehole 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.

Table 7-3 Reported Water Budget Values Used in Initial Modeling Stages

Water Balance Estimate						Reference
Precipitation	384.3	mm	0.3843	m	mean annual precipitation 1970-2011 at Divrigi	Golder Associates, 2012
	378.8	mm	0.3788	m	mean annual precipitation 1975-2006 at Erzincan	Ekmekci and Tezcan, 2007
	381.55	mm	*0.38155*	m	average of Divrigi and Erzincan annual precipitation	
Recharge	6	%			of total precipitation recharges aquifer	Ekmekci and Tezcan, 2007
Runoff	14	%			of total precipitation becomes runoff	Ekmekci and Tezcan, 2007
ET	80	%			of total precipitation is lost to ET	Ekmekci and Tezcan, 2007
Karasu River	152,600	lt/s	13,184,640	m^3/d	Average of all monthly average flow rates between 1969 and 2007	Site Data
Springs[1]	125	lt/s	10,800	m^3/d	Estimate of total spring discharge, including Gozeler	Ekmekci and Tezcan, 2007
	45	lt/s	3,888	m^3/d	Estimate of Gozeler Spring, March 2006	Ekmekci and Tezcan, 2007
	12	lt/s	1,037	m^3/d	Estimate of Gozeler Spring "wet season", April 2006	Ekmekci and Tezcan, 2007
	150	lt/s	12,960	m^3/d	Estimate of total spring discharge including Gozeler, Nov 2006	Ekmekci and Tezcan, 2007
	7.2	lt/s	622	m^3/d	Estimate of seep discharge, other than Gozeler (total of 14 springs and 17 seeps)	Ekmekci and Tezcan, 2007
Water Use	27	lt/s	2,333	m^3/d	Estimate of pumping from wells WM-17 and WM-18	
Water Balance Estimates Applied to Model Domain						
	137,740,796	m^2			GIS measurement of area of model domain	
	52,555,001	m^3/yr	143,986	m^3/d	Average annual total precipitation in model area (Area*382 mm/yr)	
Recharge	3,153,300	m^3/yr	8,639	m^3/d	Average annual recharge in model area (Area*382 mm/yr*6%)	
ET	42,044,000	m^3/yr	115,189	m^3/d	Average ET in model area (Area*382 mm/yr * 80%)	

[1]Construction of the hydroelectic dam on the Karasu River prevented the possibility for field measurements of spring discharge during this study.

Geologic cross-sections were constructed and then digitized into Leapfrog Hydro software to create a three-dimensional (3D) geologic model of the study area. This geology model was subsequently imported into the numerical groundwater flow model as zones of hydraulic conductivity. Geologic model construction in Leapfrog consisted of creating separate smaller geologic models bounded by major faults and then combining these smaller models into the final regional 3D geologic model. Figure 7-7 illustrates the construction of models and the final blended regional model. Once the models were joined, the MODFLOW grid was imported into Leapfrog and the 3D geologic model was interpolated into MODFLOW layers and cells.

Hydraulic properties were then assigned to each hydrogeologic unit. The purpose of the numerical model was to confirm the conceptual understanding of the hydrogeologic system and to estimate impacts to the system from further development of the open pit.

Figure 7-7 Individual Geologic Models Created in Leapfrog

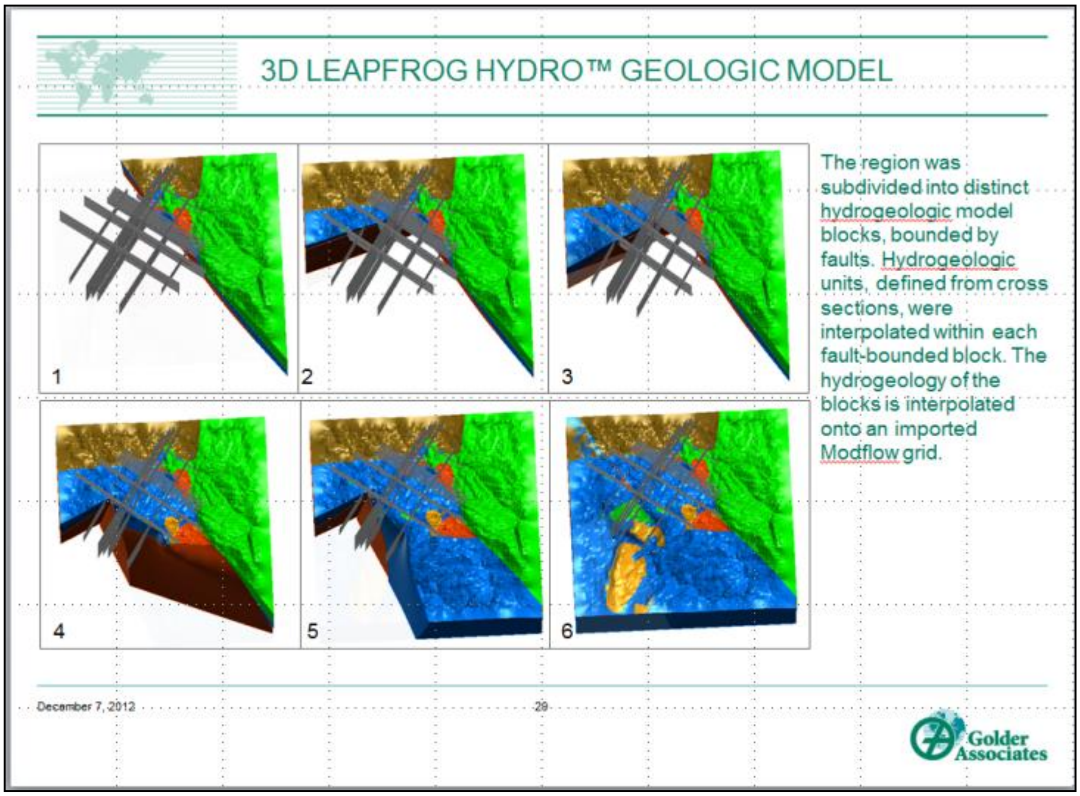


Figure prepared by Golder, 2016.

7.5.4 Pit Lake Development

Earlier studies have predicted the formation of pit lakes at various stages of mining. Golder's hydrogeologic 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 ARD work being conducted by SRK Turkey are being used to predict pit lake water quality.

Revisions to the pit design since the groundwater model was constructed and calibrated in 2012 show that the minimum pit elevation (895 m) will be 20m higher than the minimum pit elevation simulated in the model (875 m). Additionally, the area on the north side of the pit and portions of the southern and southeastern pit will be mined to a lower elevation than is 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 potential magnitude of the increase or impact.

8.0 DEPOSIT TYPES

The setting, alteration mineralogy and mineralization characteristics of the mineralization within the Project are somewhat consistent with an intermediate sulfidation epithermal system as defined in Hedenquist et al., (2000). Some deposits with mostly low-sulfidation characteristics with respect to their alteration mineral assemblages have sulfide ore mineral assemblages that represent a sulfidation state between that of high-sulfidation and low-sulfidation deposits. Such deposits tend to be more closely spatially associated with intrusions, and Hedenquist et al., (2000) suggest the term 'intermediate sulfidation' for these deposits.

Intermediate-style epithermal systems are typically hosted in arc-related andesitic and dacitic rocks. Mineralization is silver- and base metal-rich, and associated with Mn-carbonates and barite. Sulfide assemblages in intermediate-style epithermal systems typically comprise tennantite, tetrahedrite, hematite–pyrite–magnetite, pyrite, chalcopyrite, and iron-poor sphalerite. Quartz can be massive or display comb textures. Sericite is common as an alteration mineral, but the adularia, more typical of low sulfidation systems, is rare to absent.

Exploration programs that have used epithermal-style deposits as the geological model target have shown success in the Çöpler area, having discovered the Çöpler deposit and a number of prospects.

Drill intercepts have been logged that show features that may be indicative of porphyry–style mineralization, and a porphyry model is also applicable as an exploration geological model target.

9.0 EXPLORATION

9.1 Çöpler Exploration

The primary exploration effort at Çöpler was conducted by:

- Anatolia during 1998 and 1999 prior to entering into a joint venture with Rio Tinto.
- A joint venture between Anatolia and Rio Tinto from 2000 to 2004.
- Anatolia from 2004 to 2010.
- Alacer from February 2011 to date.

Drilling continues to better define both the oxide and sulfide portions of the deposit. 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 mineralization through a twin hole program. Drilling in 2015 provided data coverage at depth in the Manganese pit, in-fill drilling in the Main pit and testing of low sulfur mineralization below the oxidation boundary.

9.1.1 Surface Mapping and Sampling

As outlined within Section 6.0, the initial reconnaissance exploration was completed in the early 1960's by MTA.

Exploration by Anatolia commenced in 1998 and resulted in the discovery of several porphyry style gold-copper deposits in east-central Turkey. Shortly after that time, the joint venture with Rio Tinto resulted in an extensive drill hole exploration program at Çöpler.

Surface mapping and sampling has been undertaken over the life of the Project, culminating in a detailed geologic map of the Çöpler valley.

Geological mapping is used in support of exploration vectoring, exploration activities, infrastructure locations, mine planning and environmental monitoring.

9.1.2 Geophysics

Ground and airborne geophysical surveys were conducted at Çöpler from mid-2000 until the end of 2006. Rio Tinto and company geophysicists carried out ground magnetic, complex resistivity/ IP, time domain IP and CSAMT surveys. Fugro Airborne Surveys Ltd. carried out a regional helicopter-borne survey in 2002 that included the Çöpler area.

Rio Tinto field staff carried out quality control, processing and inversion of most of the data, the exception being the CSAMT data, which was processed by Rio Tinto personnel in Bristol, England. Zonge Engineering, of Tucson, Arizona, USA, also carried out some of the geophysical data inversions.

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

After Rio Tinto withdrew from the project in 2004, Alacer geoscientists continued the IP and resistivity surveys with large size dipole (100 m) survey lines and infill survey lines.

Details of the geophysical surveys undertaken at the Çöpler project area are tabulated in Table 9-1.

Table 9-1 Çöpler - Geophysical Survey Details, Life of Project

Survey	Date	Area	Array	Type	Line Direction	Line space (m)	Dipole (m)	Total (line km)
ZongeIP_2000	August-September 2000	Main Zone	Dipole-Dipole	Time/Frequency (CR) Domain	approx N-S	Variable 100-200	Variable 75-100	12.9
ScintrexIP_2001	September 2001	Mn Mine Zone	Dipole-Dipole	Time Domain	N-S	200	50	3.7
ZongeIP_2002	February-March 2002	Mn Mine Zone	Dipole-Dipole	Time Domain	N-S	75	50	19.3
ZongeIP_2002	April 2002	NW saddle	Dipole-Dipole	Time Domain	N-S	75	50	10
ScintrexIP_2002	August-September 2002	Main Zone	Dipole-Dipole	Time Domain	E-W	75	50	30.2
Zonge CSAMT	February 2002	Main Zone	LL	Scalar	NW-SE		50	
Ground magnetic	August-September 2000	Main Zone		hand set GPS/mobile	N-S	100 and 25		46.8
Ground magnetic	September 2001	Mn Mine Zone/Marble Contact Zone		High accuracy GPS/walkmag	E-W	25		48.3
Airborne magnetic	June 2001	All areas		Airborne	N-S	approx 125		52.1
ScintrexIP_2006	May 2006	Main Zone	Dipole-Dipole	Time Domain	E-W	75	50	30.2
ScintrexIP_2006	December 2006	Main Zone/West Zone	Dipole-Dipole	Time Domain	N-S	75	100	18.5
ScintrexIP_2006	December 2006	Main Zone/West Zone	Dipole-Dipole	Time Domain	NS-EW	150	50	4
ScintrexIP_2005	June 2005	Mn Mine Zone/Marble Contact Zone	Dipole-Dipole	Time Domain	NS-EW	150	100	14

9.2 Near Mine Exploration

Exploration activities across the Yakuplu East, Yakuplu Southeast, Yakuplu North, Yakuplu Main and Bayramdere prospects have included geological mapping, geochemical sampling, geophysical surveys, and drilling.

9.2.1 Geological Mapping

Mapping of the Yakuplu prospects and Bayramdere has been ongoing since early reconnaissance mapping by Rio Tinto exploration geologists in 2000. Figure 9-1 provides a summary of recent mapping campaigns.

Figure 9-1 Areas of Field Mapping and Sampling in 2014 and 2015

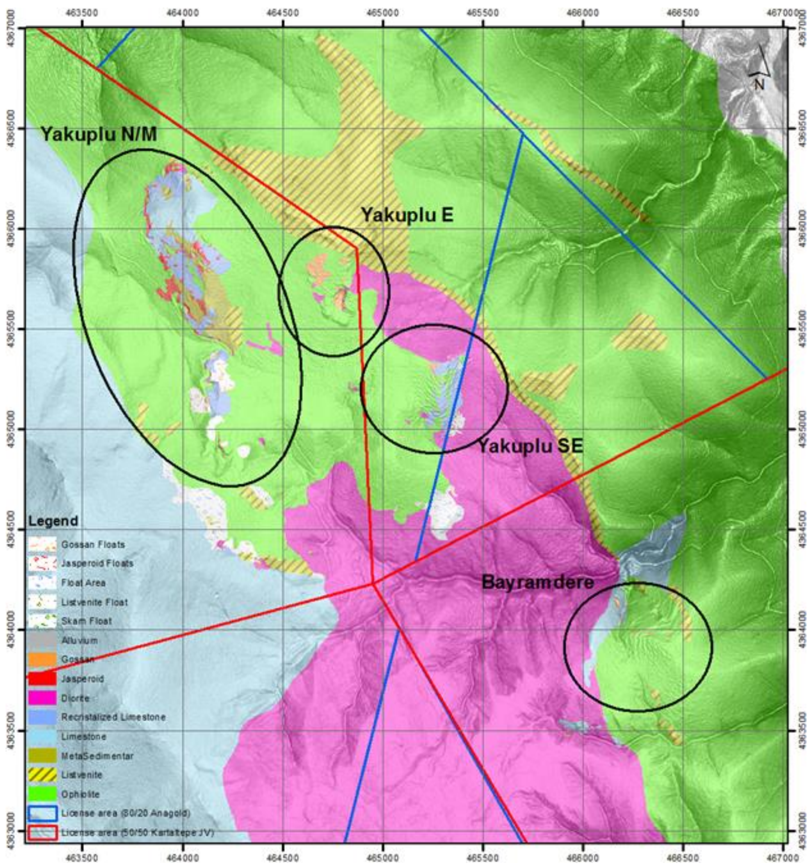


Figure prepared by Anagold, 2016

Mapping in 2014 and 2015 focused on deposit-scale surface geology definition at a scale of 1:1000, reducing to 1:500 scale for Yakuplu East, Yakuplu Southeast, Yakuplu North and Bayramdere. This has been made possible by the establishment of a network of drill access roads and drill pads cut into the sides of hills and ridges. All of these areas have been mapped including surrounding outcrops. Mapping included the collection of lithological, alteration, mineralization and structural data. In areas of drill access road development and drill sites, systematic 1 m interval rock chip sampling and assaying followed mapping. Remapping and sampling of all historic iron ore workings at Yakuplu North, Yakuplu East and Yakuplu Main was also completed. Mapping and field sampling assay data was interpreted in plan view, with subsequent correlation with sectional drilling data.

Results of the mapping programs have been used in further elucidating the general geological setting in the area of the prospects.

9.2.2 Geochemical Sampling

Geochemical sampling programs included stream sediment, rock chip and soil sampling. Most of the geochemical sampling campaigns across the Yakuplu and Bayramdere prospects occurred since 2010. Geochemical sampling was used to vector into areas of alteration or mineralization that could support more detailed investigations.

9.2.2.1 Stream Sediment Sampling

Regional stream sediment sampling was carried out from 2000 to 2003 by Rio Tinto. The Yakuplu and Bayramdere prospects were identified from five sampling sites capturing sediments shedding off the now identified prospect areas at a much higher elevation. Figure 9-2 identifies the locations of the discovery drainage sites versus the position of the prospects.

Figure 9-2 Yakuplu and Bayramdere Prospect Sediment Sampling Sites – 2000 to 2003

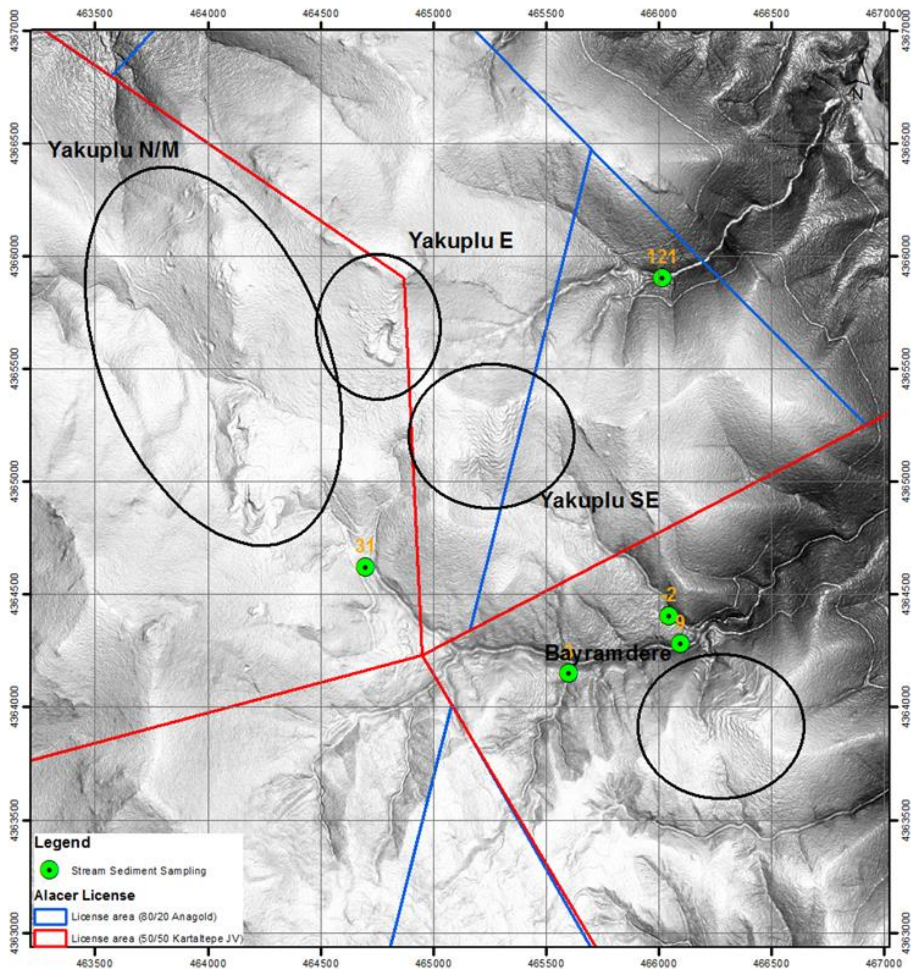


Figure prepared by Anagold, 2016.

9.2.2.2 Rock Chip Sampling

A total of 5,102 rock chip samples have been collected from across the Yakuplu and Bayramdere prospects since 2000. Rock chip sampling has been the most representative surface sampling as a result of poor soil development and abundant fresh rock exposure in a mountainous terrain. Recent (2014 and 2015) high volume rock chip sampling has been generated by routine mapping and sampling of newly established drill access roads, drill pads and historic iron ore workings. Increased rock chip sampling in 2014 and 2015 was also part of the process of reducing the scale of mapping from 1:1000 to 1:500 across areas that were considered to represent drill targets (Yakuplu E, Yakuplu SE, Yakuplu N and Bayramdere).

Figure 9-3 Yakuplu and Bayramdere Prospect Rock Chip Sampling Sites – 2000 to 2015

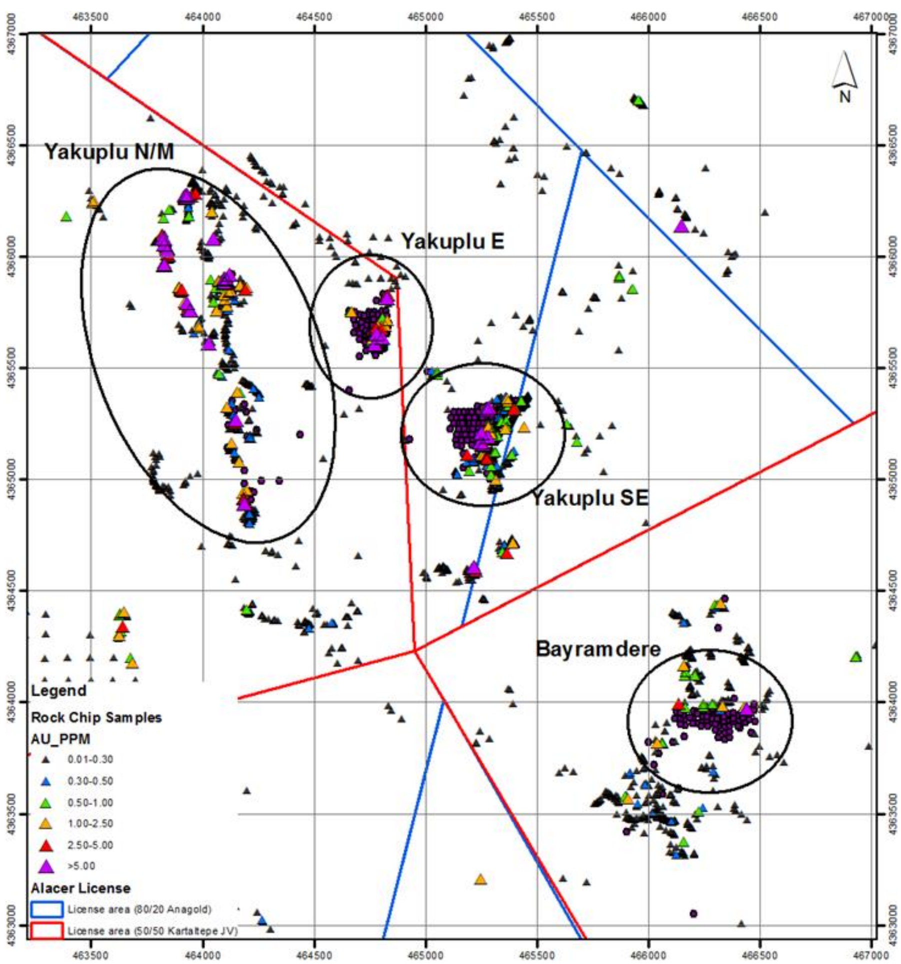


Figure prepared by Anagold, 2016.

9.2.2.3 Soil Geochemical Sampling

Rio Tinto completed targeted soil sampling as part of regional geochemical reconnaissance across tenements, with early targets being potentially mineralized listvenite capped faults. Systematic soil sampling commenced as of

2011 with Alacer Gold targeting to achieve a 200 m x 200 m soil sample coverage across all tenements. Full 200 m x 200 m regional soil sampling coverage was achieved over the Yakuplu and Bayramdere prospects. At Yakuplu N, soil sampling was reduced down to 100 m x 100 m and in select areas 50 m x 50 m spacing.

9.2.3 Remote Sensing and Satellite Imagery

In 2015, in an effort to improve survey coverage and the accuracy of survey, PhotoSat Information Ltd was contracted to provide license wide (312 km^2) high resolution satellite imagery coverage. PhotoSat provided imagery taken on August 8, 2015. The imagery was to a 50 cm resolution, stereo and non-stereo. Additional to the tenement wide imagery, Photosat also provided ortho-corrected topographic contouring to 1 m spacing and imagery colored by elevation over a 68 km^2 area covering the Çöpler Mine, Yakuplu and Bayramdere areas.

Figure 9-4 2015 Satellite Imagery over the Yakuplu and Bayramdere Prospects

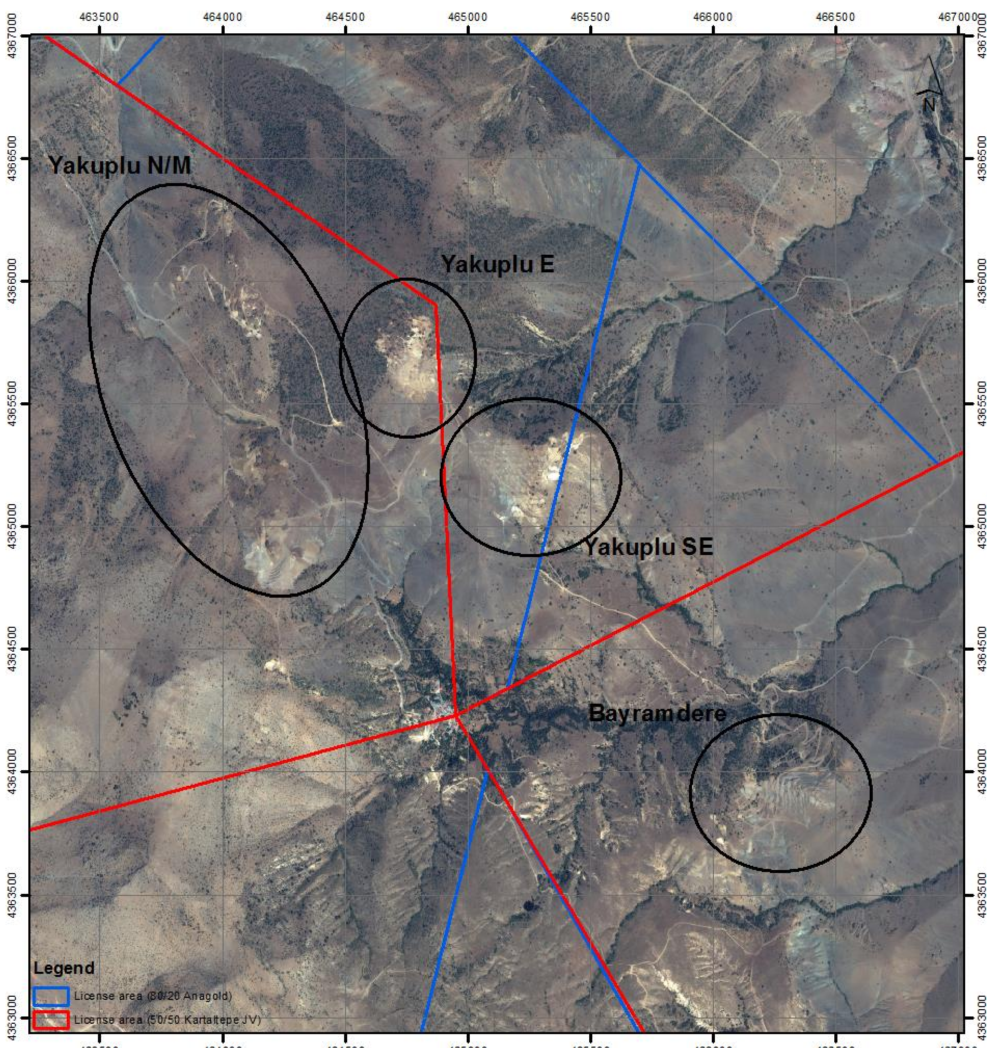


Figure prepared by Anagold, 2016.

10.0 DRILLING

Drilling has been completed on both a deposit and exploration reconnaissance scale.

Exploration drilling is summarized in Table 10-1 and collar locations are as indicated in Figure 10-1.

Table 10-1 Exploration Drilling through April 2016

Drill Prospect	Meters Drilled
Bayramdere	10,155.7
Yakuplu South East	12,912.7
Yakuplu East	10,849.9
Yakuplu North	21,275.0
Total	**55,193.3**

Figure 10-1 Project-wide Drill Collar Location Plan

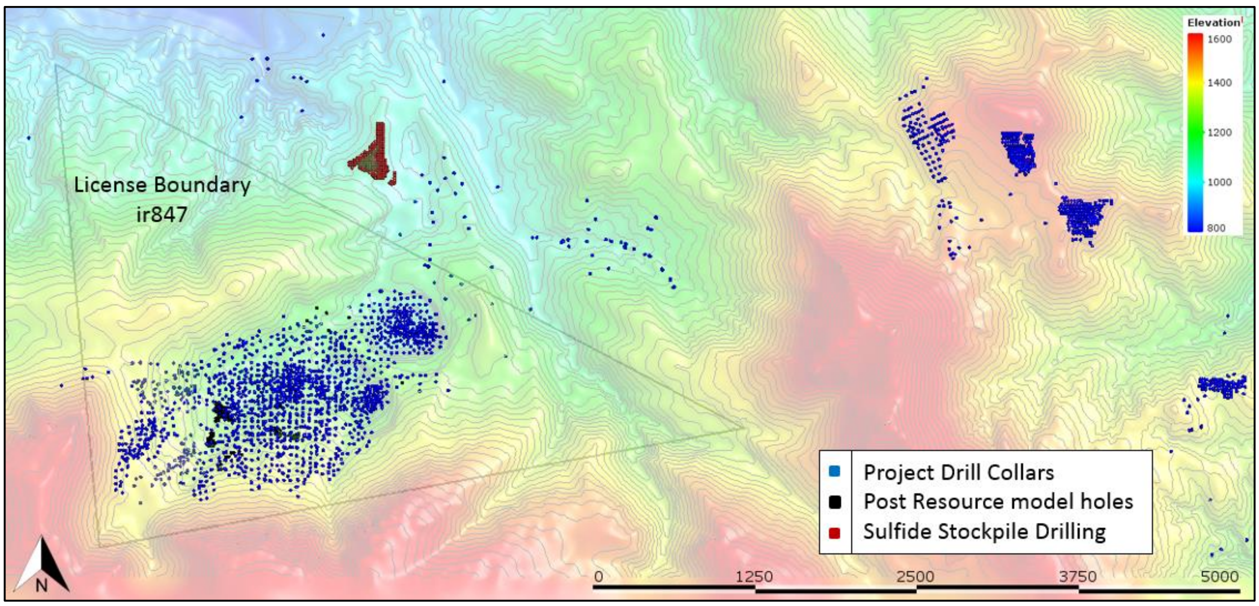


Figure courtesy of Alacer, 2016.

The Çöpler deposit has been tested by RC and DD drilling. The drilling statistics for drill holes utilized in this Mineral Resource update for the Çöpler deposit are presented in Table 10-2.

Typically the drill hole spacing at surface is a nominal 50 m by 50 m; however in some areas the drill spacing has been reduced to 25 m by 25 m (Figure 10-2).

Step-out drilling at the Çöpler deposit has defined most of the lateral boundaries of mineralization. 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 programs have occurred since 2007. Drilling in 2014 focused on ore zone confirmation with a twin hole program.

Development drilling continued in 2015 by improving sample coverage at depth in the Manganese pit and along structural boundaries in the Main pit.

Table 10-2 Çöpler Drilling by Method, through the 15th of July 2015

Hole Type	Number of Holes	Total Meters Drilled
DD	734	169,626.6
RC	1,125	125,938.6
Other	98	2,233.0
TOTAL	**1,957**	**297,798.2**

In addition to the drilling of in-situ mineralization, a stockpile drill program began in December 2015 to confirm sulfide stockpiled ore grade, grade distribution and mineralogy. Collar locations for this drilling were included in Figure 10-1. Drilling post close-out date for the Mineral Resource estimate is summarized in Table 10-3, and represents all drilling completed to April 30, 2016.

Table 10-3 Drilling Completed Post Database Closeout Date

Drill Prospect	Meters Drilled
Çöpler Pit	3,597.1
Sulfide Stockpile	4,200.5
Total	**7,797.6**

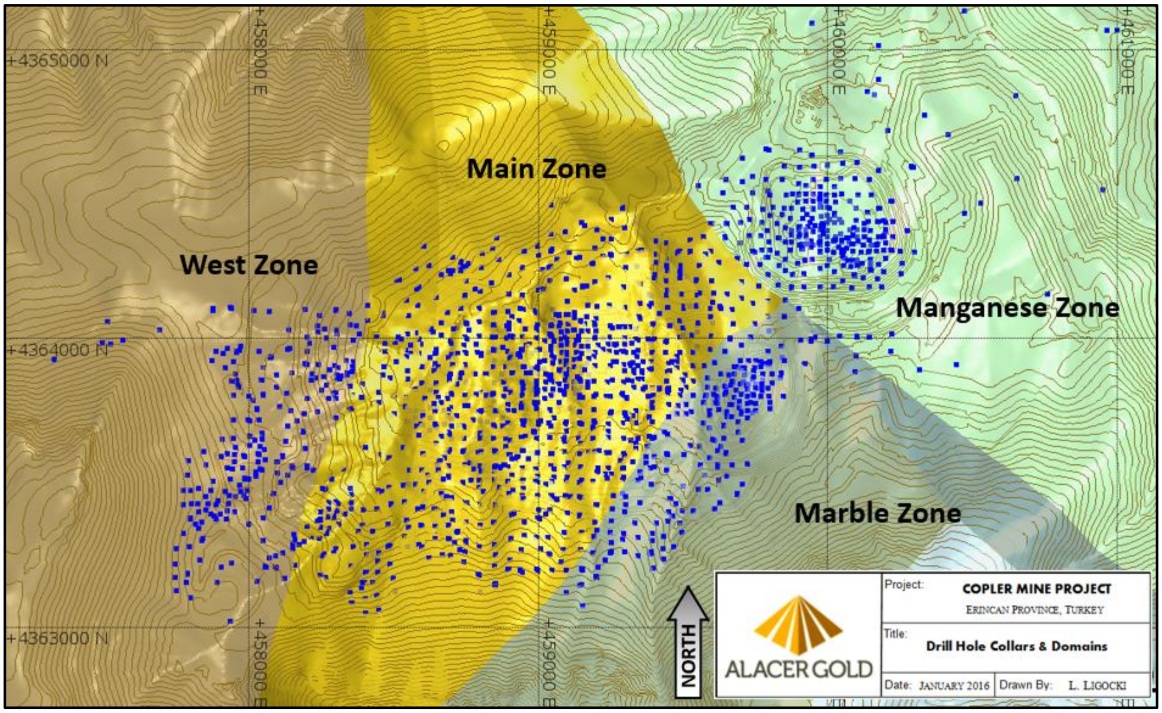
A total of 161 holes were drilled on 20 x 20 m spacing from the top of the sulfide stockpile. Total drilled meters for this program was 4200.5 m. Samples have been submitted to the SGS laboratory in Ankara with results pending. The intention is to construct a model of gold grade and process parameters to inform stockpile feed.

A total of 94 RC holes were drilled in early 2016 to infill and provide more information on the distribution of oxide mineralization in the Main Zone. A total of 2194.0 m of drilling was completed at a drill spacing of 20 by 20 m across four areas. At the time of Report effective date, samples had been submitted to the SGS lab in Ankara with results pending. The intention is to use this data to supplement site knowledge of oxide mineralization and provide short-term mine planning support. Results will be integrated with routine ore control work.

10.1 Collar Location Coordinate Systems

The database for the Mineral Resource estimate contains geological and assay information from 1,957 drill holes, distributed across the deposit as shown in Figure 10-2. The Çöpler Mine uses the European 1950 (E1950) datum, which is a Turkish Government requirement. The Çöpler deposit is located in UTM6 zone 37N of the E1950 coordinate system. Drill collars are surveyed by the mine surveyors in the E1950 UTM3 coordinate system and then converted to E1950 UTM6 before making them available to Exploration personnel. The conversion from UTM3 to UTM6 is -1746 m in Y (Northing) and +17 m in X (Easting). There is no rotation, scaling or change in elevation between the E1950 UTM3 and E1950 UTM6 systems.

Figure 10-2 Drill Hole Collar Locations and Domains (E1950 UTM6)



10.2 Collar and Downhole Surveys

Drill hole collars are surveyed by the Çöpler Mine surveyors using a Topcon differential global positioning system (DGPS) instrument. The data is provided to the Senior Exploration Geologist who makes them available for loading into the Alacer corporate database. Approximately 4% of the drill holes have planned collar locations, rather than surveyed collar data.

Down-hole surveys are currently collected for all drill holes. Prior to 2009, 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 to the Project. Drill contractors upgraded to a Reflex – EZ Trac tool for down-hole survey data collection through the end of 2014. For the drilling completed during 2015, gyroscopic (gyro) methods of down-hole survey were applied. A micro-electro mechanical-systems (MEMs) and high accuracy (HA) north-seeking gyro probe manufactured by Reflex, was supplied and used by Well Force International contractors. Using this method, survey measurements were taken every 10 m down-hole, and then 20 m up hole, providing quality assurance and quality control (QA/QC) data for each survey. The gyro survey method commenced at drill hole CRC1030 and CDD593.

The depth of the surveys varies between drill holes and is dependent on the depth and angle of the drill hole. Approximately 50% of the drilling is near vertical.

Representative drill sections with drill traces are included in Section 14.5. These sections show the relationship of the angled drilling to the mineralization and illustrate areas of lower and higher-grades.

Descriptions of the geological logging and sampling are included in Section 11.

Core recovery is discussed in Section 14.

11.0 SAMPLE PREPARATION, ANALYSIS AND SECURITY

From 2004 to late 2012, samples were prepared at ALS İzmir, Turkey and analyzed at ALS Vancouver, Canada (ALS Vancouver); collectively ALS. From late 2012 to 2014, samples were prepared and analyzed at ALS İzmir, Turkey. Samples in 2015 were prepared and analyzed at the SGS laboratory in Ankara, Turkey (SGS Ankara). Umpire analysis was completed by ACME Mineral Laboratories (ACME) in Ankara, Turkey.

SGS Ankara 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 group, globally certified to ISO9001:2008.

SGS and ALS are specialist analytical testing service companies and are independent of Alacer.

Rio Tinto operated a drill program from 2000 to 2003. Samples from this program were submitted to OMAC Laboratories Limited (OMAC) in Loughrea, Ireland. ALS assumed ownership of OMAC in 2011.

Rio Tinto instigated detailed sampling and QA/QC programs for RC and DD that have been in use since the first drill program. The QA/QC program was retained by Anagold, although the insertion rates have been modified for some of the later programs.

Anagold operates an onsite laboratory for assay of production samples. The onsite laboratory is not independent and not certified, and is not used for exploration samples.

11.1 Sample Collection

11.1.1 Reverse Circulation Sample Collection

Historical RC drilling was completed with a 4.5 inch to 4.75 inch (11.4 cm to 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 under-flow in large reinforced plastic bags.

RC drilling in 2015 was completed with a nominal $5^1/_4$ inch face sampling hammer with center-sample return to a side mounted sampling system. The sampling system consisted of a cyclone providing 1 m samples to a rotary cone splitter. The rotary cone sample splitter was adjusted to maintain a representative sample volume. RC chip samples were collected in calico bags weighing 3-5 kg for analysis and representative 1 m sub-samples were placed into chip box trays (each chip box holding 10 m) as a geological record. Reject samples were collected in PVC bags and stored in a bag farm for 6 months in case re-logging, duplicate sampling, metallurgical sampling or follow-up QA/QC was needed.

The Çöpler Mine drilling is generally above the water table, particularly in the Manganese pit and Marble Contact Zones; thus wet holes are not a particular problem for RC drilling in those areas. The water table is closer to the surface in the northern portion of the Main Zone, and for that reason, the preferred drilling method in this zone is DD.

Typically, the RC sample passes up the drill pipe and through the sample hose into the cyclone where it drops into a large plastic bag. Prior to 2015, RC samples were split using a Jones splitter. Several stages of splitting were needed to reduce the sample size down to approximately 1 kg of sample. RC sample splits are now completed via a cone splitter within the cyclone of the drill rig. The 1 kg sample is collected in a calico bag and becomes the sample submitted to the laboratory for analysis. All sample bags are clearly numbered and labeled with the drill hole name and sample number.

The rig sampler sieves a small portion of remaining residual sample from the large plastic bag and places it in a plastic tray, in order to generate a sample for logging. The plastic chip trays are also photographed.

Any remaining sample is returned to the large plastic bag which is transferred to the sample storage and core sawing facility located immediately north of the Administration office at the mine site for storage.

The RC sample preparation procedures at the Çöpler Mine are as indicated in Figure 11-1.

Figure 11-1 Sample Preparation Procedures

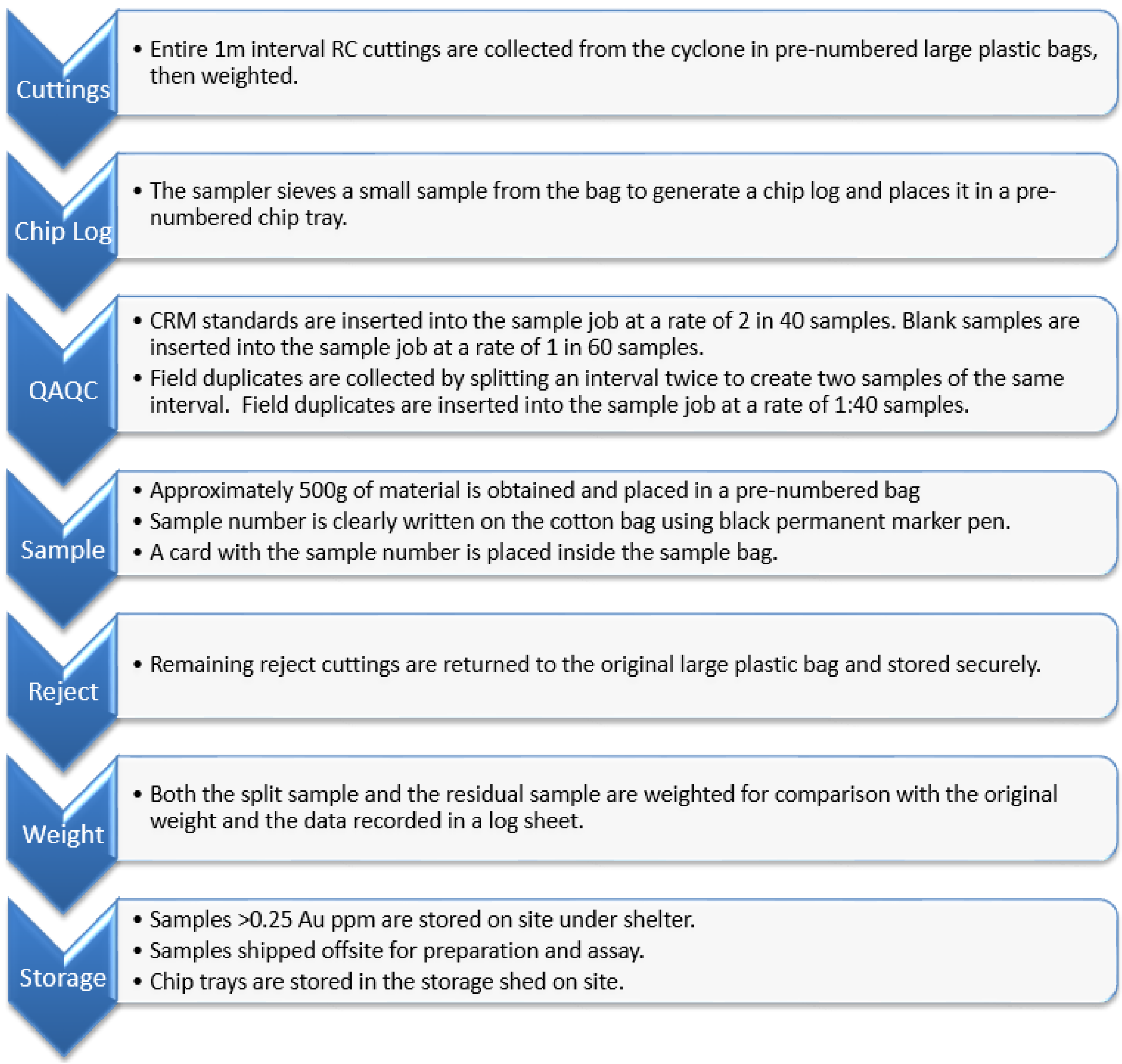


Figure prepared by Alacer, 2016.

QA/QC samples are collected during the sampling process. Certified Reference Materials (CRMs) are inserted into each sample job at a rate of two CRMs in every 40 samples (1:20 insertion rate). Blank samples are inserted into each sample job at a rate of one blank in every 60 samples (1:60 insertion rate). Field duplicate samples are collected by splitting an RC sample twice to collect two independently numbered samples of the same interval. Field duplicates are collected and inserted into the sample job at a rate of 1:40 samples. In 2015, duplicate insertion rates were increased to 1:20.

11.1.2 Diamond Drilling Core Sample Collection

DD has generally utilized NQ or HQ diameter core, as defined by the Diamond Core Drill Manufacturers Association. HQ core has a nominal diameter of 63.5 mm while NQ has a nominal size of 47.6 mm. Approximately 90% of the core drilled at Çöpler is HQ. Some drill holes are started with HQ and are reduced in size to NQ later in the hole.

Drill core is boxed at the rig by the driller and transported to the sample preparation facility on site for logging by Anagold staff. All core is digitally photographed and logged at the core shed. Minor geotechnical data, such as rock quality designation (RQD) and the percentage of solid core, is recorded together along with core recovery.

Competent drill core 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 half the sample. Half the core is placed in a sample bag and half is returned to the core tray. Sample numbers are assigned and sample tags are placed in the sample bags and recorded in the master sample list by down-hole interval. Sample intervals are typically one meter down-hole.

QA/QC samples are collected routinely during the sampling process. CRMs are inserted into each sample job at a rate of 1:20. Blank samples are inserted into each sample job at a rate of 1:60. Field duplicate samples are 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 1:40 samples. In 2015, the field duplicate insertion rate was increased to 1:20.

The DD sampling protocol at the Çöpler site is as indicated in Figure 11-2.

Figure 11-2 Core Sampling Protocol

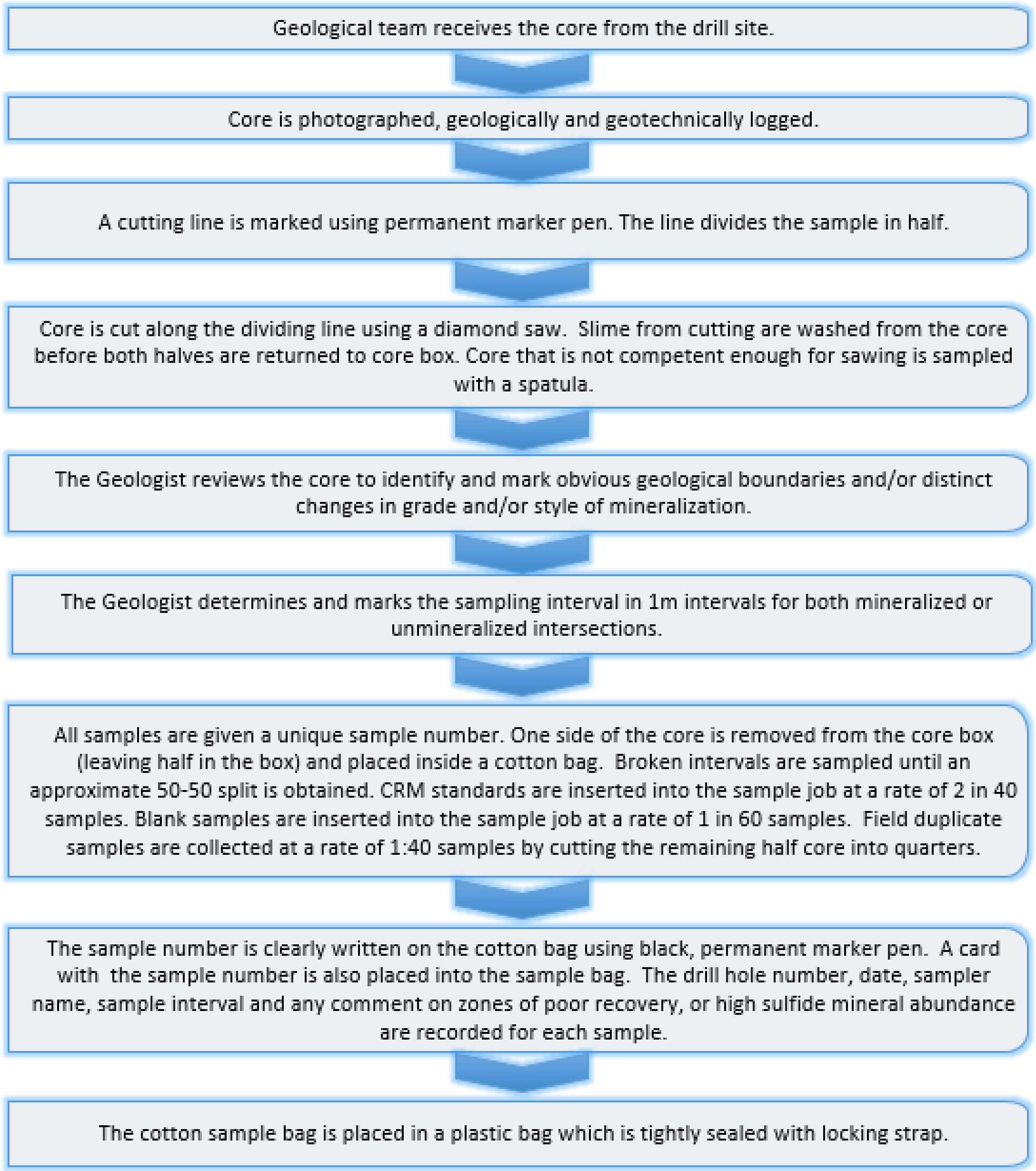


Figure prepared by Alacer, 2016.

11.1.3 Drill Hole Logging and Data Collection

All drill holes are logged for detailed geological information such as rock type, alteration, mineralization, veining and structure using defined Anagold geological codes and logging formats.

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

All geological data are recorded onto hard-copy logs and then transcribed into text files using data-loading templates, ready for loading into the Corporate relational SQL database.

The SQL drilling database, for both exploration and mine production, is managed by the Anagold geology team located at the Çöpler Mine. Support from personnel in the Ankara office occurs on a regular basis and is organized by Anagold management at the mine site.

11.2 Sample Preparation

11.2.1 RC Sample Preparation

The majority of historical RC sample preparation was completed at the ALS preparation facilities in İzmir, Turkey. From late 2012 through the end of year 2013, pulp samples weighing approximately 150 g were sent to ALS Vancouver. All samples in 2014 were generated and analyzed by ALS İzmir, Turkey. In 2015, samples were sent to SGS Ankara for preparation and assay. The procedures used by SGS Ankara are detailed below in Figure 11-3.

Figure 11-3 RC and Diamond Core Sample Preparation Procedure for SGS Ankara

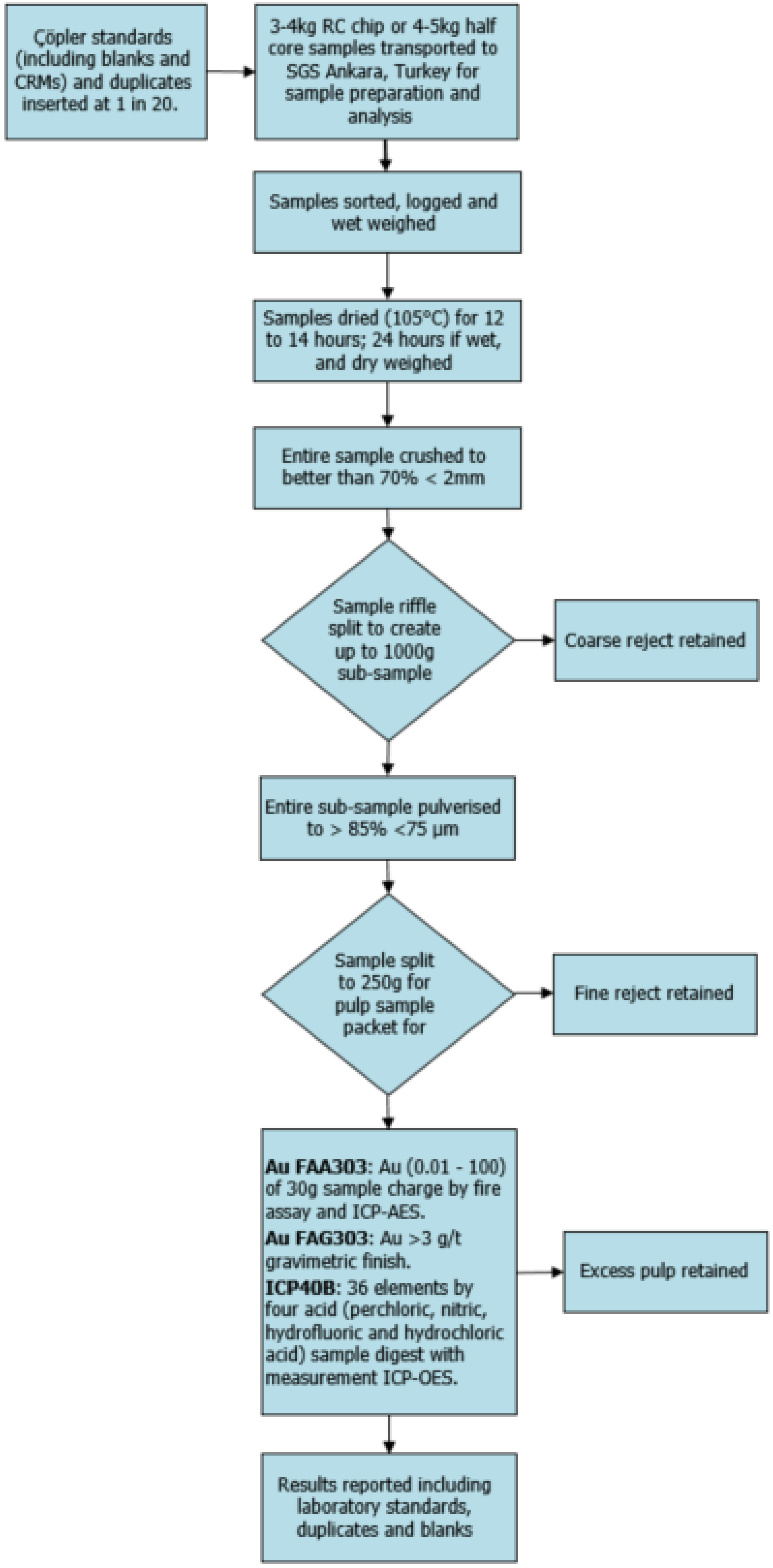


Figure courtesy SGS, 2016

11.2.2 DD Sample Preparation

The majority of historical DD sample preparation was completed at the ALS İzmir. From late 2012 through the end of year 2013, pulp samples weighing approximately 150 g were sent to ALS Vancouver. All samples in 2014 were generated and analyzed by ALS İzmir. Then in 2015, samples were sent to SGS Ankara for preparation and assay. The SGS Ankara procedures used are detailed in Figure 11-3.

11.3 Sample Analysis

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

Analysis of an additional 33 elements was performed using the ALS 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). Silver, copper, lead, zinc and manganese are among the 33 elements analyzed by this method.

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

A 36 element analysis was performed at SGS Ankara with ICP40B method which involves a four acid digest followed by measurement of element grades by inductively coupled plasma –atomic emission spectroscopy (ICP-OES).

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 being conducted. The core storage facility is enclosed by a fence and gate that is locked at night and when the geology staff is absent. The samples were transported to ALS İzmir and SGS Ankara by commercial carrier.

11.5 QA/QC Procedures

A detailed QA/QC protocol was implemented by Rio Tinto at Çöpler. This protocol is still currently in use, although the insertion rates have been amended. The Project QA/QC program has historically consisted of a combination of QA/QC sample types that are designed to monitor different portions of the sample preparation and assaying process.

Blanks consist of non-mineralized samples that are submitted in order to identify the presence of poor sample preparation practices. Prior to 2015, blank samples comprised prepared pulp samples obtained from commercial vendors. Commencing in 2015, the pulp samples were switched to a coarse quartz material that would allow for better monitoring of sample contamination. Because pulp blanks are not crushed or pulverized they are of limited value. Blank samples have been inserted routinely into all sample batches. If a blank returns an assay grade above an acceptable limit, contamination from a previous mineralized sample has occurred at either the crushing or pulverization stage. The first sample in a drill hole is typically a blank, after which blanks are inserted into the sample batch at a nominal rate of 1 in 60 samples. This insertion rate does not meet with industry practice, which is typically 1 in 20 samples.

Certified reference materials (CRM) samples are inserted into sample submissions in order to monitor and measure the accuracy of the assay laboratory results. CRMs have been inserted into sample submissions at a nominal rate of 1 in 30 samples at Çöpler. The frequency was increased from 3% to 5% in 2015. A number of different CRMs have been selected for use at varying gold and copper grades over the life of the Project. The pulp blanks utilized by Anagold are capable of determining 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.

Field duplicates are used as a means of monitoring and assessing sample homogeneity and grade variability. They enable the determination of bias and precision between the 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 quarters and submitting one of the quarters of core as the field duplicate. RC field duplicates are generated by splitting the RC sample twice to create two samples of the same interval. Field duplicates have been historically and continue to be submitted at a nominal rate of 1 in 40 samples.

Rio Tinto undertook a small program of coarse reject duplicate and pulp duplicate analyses on samples during the 2000 to 2003 drilling programs; however, this program has not been undertaken since.

11.6 Opinion on Adequacy

Sergei Smolonogov of Anagold is of the opinion that the sample preparation, sample security and analytical procedures utilized are appropriate for support of Mineral Resource and Mineral Reserve estimates, and for mine planning purposes.

12.0 DATA VERIFICATION

12.1 Data Verification in Support of Technical Reports

Data verification was conducted during compilation of technical reports on the Project from 2003 to 2012 (refer to the list of reports in Section 2). None of the verification programs identified material issues with the supporting data.

12.2 Çöpler Drilling

In 2014, Amec Foster Wheeler conducted a database audit and review of available quality assurance and quality control (QA/QC) data to ensure the data are of sufficient quality to support resource estimation. The database audit covered data collected from 2000 to December 2013, which included 1,462 drill holes.

Amec Foster Wheeler could not validate collar and down-hole survey data because Alacer was unable to provide copies of the original documents. Scans of available original drill logs (lithology, RQD and bulk density) were compared to values contained in the database. Rio Tinto operated a drill program from 2000 to 2003, samples from this program were submitted to OMAC. Assay results from early drill holes (2000 to 2003) assayed by OMAC were unable to be obtained at the time of the audit. Assay results from 2004 to 2013 were obtained from ALS. Amec Foster Wheeler electronically compared the ALS assay results (gold, copper, silver, arsenic, iron, manganese, sulfur and zinc) to the assay results in the database.

In 2015, Amec Foster Wheeler reviewed the Çöpler deposit database as of July 15, 2015 in order to verify the data are of sufficient quality to support Mineral Resource estimation of gold, copper and silver for the Çöpler deposit. This audit focused on the 121 drill holes (12,959.8 m) completed since the previous audit.

Amec Foster Wheeler validated collar and down-hole survey data against the original documents. Amec Foster Wheeler compared original drill logs for lithology and RQD to values contained in the database. Density data were supplied on a separate Excel spreadsheet and were compared to the original logs. Assay results from 2014 and 2015 were obtained directly from ALS and SGS. Amec Foster Wheeler electronically compared assay results (gold, copper, silver, iron, manganese, sulfur) to the database. Available QA/QC data were evaluated to ensure the assay data are suitable to support resource estimation.

12.3 Collar Location

The 2014 audit indicated Alacer has not retained the original collar survey documentation provided by the mine site survey department. Additionally, collar locations cannot be confirmed because about one-third of the drill hole collars have been mined away (either benched or buried).

Amec Foster Wheeler performed a field check of drill collars and recorded the locations of 38 drill hole collars during a site visit in May 2014. Collar coordinates for these holes were collected in the field using a hand-held GPS. The mine site survey crew also collected the locations of the same points with the Topcon instrument. The locations of two holes (CRC490 and CRC775) differed by more than 10 m compared to the Alacer database. It was recommended that Alacer site staff revisit the locations of these holes and resolve the differences.

Because Alacer lacks original surveyor's records, Amec Foster Wheeler recommended Alacer re-survey the remaining drill hole collars, update the current database, and archive the survey coordinates. A qualified surveyor should sign and date the surveyed coordinates and this documentation should be added to the drill hole folders.

In 2015, Amec Foster Wheeler audited collar locations for 111 of 121 drill holes. Minor differences (to the second decimal point) were noted between the original data and the database, but none of these differences would affect the Mineral Resource estimate.

12.4 Down-hole Surveys

The 2014 audit was unable to review the down-hole surveys because Alacer did not have the original films, records, or documents available.

Amec Foster Wheeler recommended Alacer initiate a procedure to retain the down-hole survey data as they are collected. This information should be reviewed by the responsible geologist, then be signed, dated and added to the drill hole folder.

Amec Foster Wheeler also recommended that Alacer apply the proper magnetic declination correction of 5.6°E rather than the 3.0°E correction currently being applied. The declination correction has varied from 4.5°E in 2000 to 5.6°E in 2014. The correction applied should be based on the year the data were collected.

The 2014 audit indicated approximately 32% of the holes have a recorded down-hole survey, while the remaining holes used the planned drill azimuth and inclination. Amec Foster Wheeler compared the actual end-of-hole location for 245 drill holes 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 block dimension of 10 m x 10 m x 5 m; however, Amec Foster Wheeler recommends that all core holes with lengths greater than 300 m should be surveyed down the hole.

Anagold now uses an external contactor for down-hole measurements on all holes drilled.

In 2015, Amec Foster Wheeler audited down-hole survey data for 94 of 120 drill holes that had been surveyed down-hole since the audit in 2014. Drill hole CDD589 did not have a down-hole survey. As with the collar surveys, minor differences (to the second decimal point) were noted for the azimuth and dip data between the original data and the database, but none of these differences would materially affect the Mineral Resource estimate. The dip values for drill hole MET003 have been imported incorrectly; however, the impact of this error is considered to be minor as the length of the hole is only 80 m.

Amec Foster Wheeler utilized a proprietary computer program (Kinkcheck) to check for excess deviation in the holes. A 5° deviation over a distance of 30 m was set as the maximum deviation allowed. Five drill holes (CDD238, CDD426, CDD435, CRC198 and MET715A) were flagged as having intervals exceeding the allowable deviation. These intervals were provided to Alacer who reviewed and corrected the down-hole survey data. A check on the corrected database indicated that no drill holes contained excessive deviation.

12.5 Geology Logs, Density Logs and RQD Logs

In the 2014 audit, Alacer was not able to provide all the requested geology, geotechnical and density logs to support the audit due to missing drill logs. Amec Foster Wheeler recommended Anagold attempt to locate original logs for the missing holes. For current and future holes, Amec Foster Wheeler recommended the Anagold Senior Geologist review, sign and date the final logs. At this time, Anagold Chief or Senior geologist sign logging forms after completion.

In 2015, Amec Foster Wheeler audited the geology logs for 11 of the 121 (approximately 9%) drill holes added to the database since the audit in 2014. No material errors were identified.

Density data were supplied in a spreadsheet "Copler_Resdev_BD_18-04-2015.xlsx" and were compared to scanned images of the original logs. Bulk density is determined based on the wax coated immersion method. Ten data entry errors were observed, typically resulting in minor changes to the density value. Two errors caused a significant change to the density value; however, due to the large number of density data available, these errors will not materially impact the resource estimate. Amec Foster Wheeler has supplied a list of the corrections to Alacer staff.

Amec Foster Wheeler reviewed the input of the RQD data into the database for five core drill holes (CDD584, CDD593, CDD603, CDD613 and MET001) which represents about 10% of the new drill holes containing RQD data. No material errors were identified.

12.6 Assays – 2000 to 2003

Assay laboratory certificates for drilling prior to the year 2004 were not available. Rio Tinto conducted the drilling program, and samples were submitted to OMAC. ALS assumed ownership of OMAC in 2011. Since the Amec Foster Wheeler audit, Anagold obtained the electronic records from OMAC, however, the laboratory certificates where not with ALS.

Amec Foster Wheeler used statistical methods (histograms and quantile-quantile plots) to validate the OMAC data against the ALS data and found the data to be compare well. A divergence at approximately 4 g/t Au seen in the QQ plot is explained by the inclusion of a few higher-grade composites. These composite grades were confirmed by Anagold drilling in the vicinity. OMAC drilling represents 6% of the total meters drilled at the time of the database extract for the resource estimate.

12.7 Assays – 2004 to 2015

In the 2014 audit, Amec Foster Wheeler received ALS assay results as *.csv* and Excel spreadsheets for the period 2004–2013. The results for gold, silver, arsenic, copper, iron, manganese, total sulfur and zinc were extracted and compiled into an Access database. These results were compared to the values contained in the *.csv* file supplied by Anagold. The results of the comparison are presented in Table 12-1.

Table 12-1 ALS Assay Audit Summary

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 manganese is due to conversion of manganese oxide (MnO) assays to manganese-only values. Amec Foster Wheeler used a conversion factor of 0.7745 to convert MnO assays to manganese values. It did not appear, however, that a constant conversion factor was applied to the values in the Anagold database.

Amec Foster Wheeler compared gold (and cyanide-soluble gold assays if available), silver, arsenic, copper, iron, manganese, total sulfur and zinc from 995 samples analyzed by ALS and noted only one error. The database contains a copper assay of 1.0% for sample number 333474 rather than the correct value of 1.074% (refer to ALS certificate IZ140478).

Amec Foster Wheeler compared gold (both fire assay and cyanide soluble), silver, arsenic, copper, iron, manganese, total sulfur and zinc from 11,228 samples analyzed by SGS and noted 53 errors for gold and nine errors for copper. A list of sample numbers, assay values and associated SGS certificates was sent to Alacer staff for review and to be used to update the database.

The current detection limit for SGS procedure (ICP40B) for silver is 2 g/t. Amec Foster Wheeler recommends employing an analytical method such as GE ICM40B which would provide a detection limit of 0.02 g/t Ag. Samples with a silver grade over 75 g/t should be re-assayed with a procedure using a four-acid digestion and AAS finish.

In Amec Foster Wheeler's opinion, the current assay data are of sufficient quality to support resource estimation.

12.8 Amec Foster Wheeler Witness Samples

In 2014, Amec Foster Wheeler collected 10 witness samples obtained from blast hole cuttings which were then submitted to both the Çöpler site laboratory and to ALS. The mean of the ALS gold assay results is 8% higher than the mean of the results provided by the Çöpler site laboratory. If the result from one high-grade sample (above 4 g/t Au) is removed from the comparison, the mean ALS gold grade is 3% higher than for the mine site laboratory. In Amec Foster Wheeler's opinion this is acceptable agreement between the two laboratories. Amec Foster Wheeler included one CRM with the submissions; the result from this CRM indicates acceptable performance by the assay laboratories. The results from the witness samples are stated in Table 12-2.

Table 12-2 Amec Foster Wheeler Witness Sample Results

Au_ppm fire assay				
Blast Hole Sample	Copler Ore Control	Copler Blind	ALS1	ALS2
C2-1240-002-345	0.36	0.40	0.44	0.50
C2-1240-002-346	0.99	0.93	1.02	1.05
C2-1240-002-347	0.99	0.96	0.99	0.98
C2-1240-002-348	1.17	1.19	1.25	1.17
C2-1240-002-349	4.35	4.50	4.81	5.26
C2-1240-002-387	0.33	0.30	0.29	0.25
C2-1240-002-388	0.13	0.14	0.14	0.13
C2-1240-002-389	0.38	0.35	0.27	0.30
C2-1240-002-390	0.72	0.69	0.71	0.73
C2-1240-002-391	0.71	0.67	0.81	0.70
Average Value	1.01	1.01	1.07	1.11

12.9 Quality Assurance Quality Control (QA/QC) Results

Amec Foster Wheeler evaluated the available QA/QC data to ensure the assay data were suitable to support Mineral Resource estimation.

12.9.1 Screen Analyses

As part of the 2014 audit, Amec Foster Wheeler reviewed 1,724 crusher screen test results obtained from 387 ALS certificates reporting the percent passing a 2 mm screen. All but eight samples exceeded the specification of 70% passing 2 mm. A review of 3,945 pulverizer screen test results was made from 750 ALS certificates for material passing a 75 µm screen. There were 443 samples (11%) did not meet the specification of 85% passing 75 µm. There is a marked improvement in pulverization starting about July 2013. Amec Foster Wheeler is unsure of the cause of this improvement.

There were very few ALS screen test results from 2014, but Amec Foster Wheeler reviewed the 2015 crusher and pulverizer screen test results from SGS. All of the 681 crusher screen test results met the specification of 70% passing 2 mm; and only one of the 680 pulverizer screen test results failed to meet the specification of 85% passing 75 µm.

12.9.2 Certified Reference Material (CRM)

Rio Tinto’s review of the CRM results from the samples submitted to OMAC (2000 to 2003) indicated that acceptable accuracy was achieved by OMAC: for 632 out of 651 gold standards and blanks used, Au analyses for 97% fell within the ± 2 standard deviation accepted range.

CRM results between 2004 to 2006 were not provided.

In 2014, Amec Foster Wheeler was provided with CRM results for the period 2007 to 2013. During this period Alacer had used over 50 CRMs and property standards in their QA/QC program. These were inserted at a rate of 5%. The CRMs were obtained from Rock Labs, Geostats and Gannett Holdings, who are recognized suppliers of such samples. The property standards were generated from material collected at the Çöpler site itself. The property standards do not have sufficient round robin results to qualify as certified standards, and were not included in Amec Foster Wheeler’s review.

Despite apparent mislabeling of some CRM samples, Amec Foster Wheeler noted the overall relative bias for the CRMs from this period is within 5% and is of the opinion that the assay accuracy is sufficient for Mineral Resource estimation.

Anagold has used over 11 different CRMs since 2013. These are inserted at a frequency of 5%. However, three CRMs are primarily used to monitor assay accuracy, these are: OREAS152b, OREAS502b and OREAS504b (obtained from Ore Research and Exploration P/L located in Australia). Amec Foster Wheeler noted the overall relative bias for these CRMs is within 5% and concludes the assay accuracy is sufficient for resource estimation.

Amec Foster Wheeler recommends inserting an additional CRM near the oxide cutoff grade of 0.30 g/t Au. As silver is a very small contributor to the project economics adding a single silver CRM would be sufficient. An additional CRM to monitor total sulfur assays should be added at the sulfur grade (2%) used to define the oxide/sulfide boundary.

12.9.3 Blank Samples

Rio Tinto did not note any issues with sample contamination at OMAC.

The 2014 audit reviewed the results from 2,437 blank samples from 10 blank material sources blindly inserted into drill sample submissions. Although the results indicate that there is likely some carry-over contamination of gold, the amount of contamination is not sufficiently high to materially affect Project assay results; hence Amec Foster Wheeler concludes there is no significant risk to the resource estimate.

In 2015, Amec Foster Wheeler reviewed the results from 264 blank samples from two blank material sources blindly inserted into drill sample submissions. Based on these sample results, there does not appear to be any indication of sample contamination. However, based on the sample number of the blank samples, it appears only 1 in 60 samples is submitted as a blank. This is well below industry-leading practices which use a submission rate of 1 in 20 samples.

Amec Foster Wheeler recommends that Alacer commence inserting blank samples at an insertion rate of1 in 20.

12.9.4 Field Duplicates

During 2000 and 2003, Rio Tinto submitted both coarse reject and pulp reject duplicate samples. They noted an issue possibly due to coarse gold in the coarse rejects. The pulp reject duplicates showed excellent agreement.

Amec Foster Wheeler used the oxide cutoff grade of 0.30 g/t gold for assessing the precision of the gold assays. The 90th percentile absolute relative distance (ARD) for the core duplicates (2009 to 2013) with grades exceeding 0.3 g/t Au is ±55%. The 90th percentile ARD for the core duplicates (2014 to July 2015) with grades exceeding 0.3 g/t Au is ±60% while the RC duplicates have an 90th percentile ARD of ±36%.

Precision for gold is somewhat poorer than Amec Foster Wheeler's target level of ±30%; however, in Amec Foster Wheeler's experience gold assays often do not meet this threshold unless the mineralization is of the Carlin disseminated-type and the mineralization has maximum gold particle sizes less than 5 µm with the gold commonly well dispersed throughout the drill core.

Amec Foster Wheeler used 20 g/t Ag (10 times the Ag detection limit) to assess the precision for Ag assays. The 90th percentile ARD is ±53% and ±40% for core and RC duplicates respectively, but there are very few data supporting this conclusion. Silver has a very minor contribution to the Project economics, and therefore this is not considered to be a material issue.

Amec Foster Wheeler used 0.10% S (10 times the sulfur detection limit) to assess the precision for the sulfur assays. The 90th percentile ARD for sulfur both core and RC duplicates is ±30%.

Amec Foster Wheeler finds the assay precision is likely adequate for Mineral Resource estimation, based on the core/RC duplicates. The precision of silver assays may be improved by using an analytical method with a lower detection limit, however Ag contributes less than 1% to the Project economics, and therefore the precision of the silver assays is not considered to be material.

12.9.5 Check Assays

Based on the 2004 report by Rio Tinto for the 2000 to 2003 drilling, 403 check samples of prepared coarse reject material and 203 samples of fine reject material were submitted for check gold (± copper and silver) assays at OMAC, ALS and Bondar Clegg. This was carried out as a quality control review of both the sample preparation at ALS Izmir and also the accuracy of analyses at OMAC. Rio Tinto stated they found excellent agreement between intra-laboratory duplicate fire assay gold analyses carried out at OMAC, and inter-laboratory analyses between OMAC, ALS, and Bondar Clegg.

It does not appear that check samples were submitted from 2005 to 2009, or from 2011 to 2014. Historic pulp and sample reject material prior to 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 programs. These samples were submitted to ACME for analysis. Anagold was unable to supply the check assay results; thus Amec Foster Wheeler has commented on results stated in a report written by Georgi Magaranov, dated 6 April, 2010. Both pulp rejects and field duplicates were submitted as check samples, and results from the pulp rejects are discussed in the report.

Based on 111 results, the gold assays from ALS between the 2009 and 2010 time period were biased 6% high compared to ACME for the RC holes. Based on 51 results, the gold assays from ALS are biased 8% higher than ACME for the core drill holes.

The Magaranov report does not state whether CRMs were included with the samples submitted to ACME; thus Amec Foster Wheeler is unable to comment further on the differences noted.

Amec Foster Wheeler recommends a random selection of 5% of the available samples from 2013 be collected and submitted for check assaying. Suitable CRMs and blanks should be included with these samples with a minimum insertion rate of 5%.

In 2015 Anagold submitted 318 samples to Bureau Veritas Commodities Canada Ltd. (BV) as check samples. This submission included 301 check samples (pulps), 11 CRMs and six blank samples. A review of the gold, silver, copper and total sulfur results indicates that SGS is biased 2.2% low for gold, 6.9% low for silver and 11% high for sulfur compared to BV. There was no bias noted for the copper results.

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

12.10 Discussion

During the validation process in 2014, Anagold was unable to provide collar documentation, down-hole survey documentation, and a significant number of drill logs were not able to be provided. Amec Foster Wheeler recommends that Anagold attempt to obtain as many historical logs as possible and implement procedures to ensure current data are collected and stored in a series of folders. Ideally, data for each drill hole would be stored in an individual folder. For current and future holes, the Anagold Senior Geologist should review, sign and date the final logs. Revisions and updates should also be dated and signed.

Assay laboratory certificates for Rio Tinto drilling from 2000 to 2003 are not available. Electronic assay values from the laboratory were obtained; however, assay certificates were not retained by OMAC. Amec Foster Wheeler validated these Rio Tinto results against ALS assay results from adjacent or nearby drill holes. The electronic assay values obtained from the laboratory should be compared to the values contained in the database.

ALS provided assay results (2004 to 2014) for gold, silver, copper, iron, manganese, total sulfur and zinc. These data were compared to the values contained in the Anagold database. Amec Foster Wheeler provided Anagold with a summary of the differences for review and correction by Anagold's database staff. These should be assessed and corrections verified prior to preparing future Mineral Resource estimates.

Amec Foster Wheeler compared gold (both fire assay and cyanide soluble), silver, copper, iron, manganese, total sulfur and zinc from 11,228 samples analyzed by SGS in 2015 and noted 53 errors for gold and nine errors for copper. A list of sample numbers, assay values and associated SGS certificates was sent to Alacer staff for review and to be used to update the database.

Amec Foster Wheeler evaluated available QA/QC data to ensure the assay data were suitable to support Mineral Resource estimation.

Amec Foster Wheeler reviewed 1,724 crusher screen test results obtained from ALS certificates (387 certificates) for material passing -2 mm. All but eight samples exceeded the specification of 70% passing -2 mm.

Amec Foster Wheeler reviewed 3,945 pulverizer screen test results obtained from 750 ALS certificates for material passing 75 µm. A total of 443 samples (11%) did not meet the specification of 85% passing -75 µm. For the period from July 2013 to the end of 2013, there is an abrupt improvement in pulverization. Amec Foster Wheeler is unaware of the reason for this.

There were very few ALS screen test results from 2014, but Amec Foster Wheeler reviewed the 2015 crusher and pulverizer screen tests results from SGS. All of the 681 crusher screen test results met the specification of 70% passing 2 mm; and only one of the 680 pulverizer screen test results failed to meet the specification of 85% passing 75 µm.

The Anagold QA/QC program includes CRMs, blanks, preparation duplicates and field duplicates and is acceptable according to industry standards. The following improvements could be made:

- QA/QC results should be monitored on a regular basis during a drilling program and the laboratory asked to follow up on samples that are outside the acceptable range.
- Anagold should add an additional CRM (~0.3 g/t) to monitor gold assays and another CRM to monitor silver assays. In addition, Anagold should add additional CRMs to monitor sulfur assays near the current oxide/sulfide threshold of 2% S.
- As mining progresses into dominantly sulfide material, the CRMs should be changed to sulfide-based CRMs.
- Based on the sample number of the blank samples, it appears only 1 in 60 samples is submitted as a blank. Amec Foster Wheeler recommends that Alacer commence submitting 1 in 20 samples as a blank.
- Anagold should follow QA/QC protocol by sending 5% of the samples to a secondary laboratory for check analysis. Samples should be sent on a regular basis, and not at the end of the drilling program.
- Anagold should modify their procedures for insertion of QA/QC samples (blanks, CRMs, pulp duplicates) and selection of check samples to ensure an insertion rate of 1:20 is maintained.
- The current detection limit for SGS procedure (ICP40B) for silver is 2 g/t. Amec Foster Wheeler recommends employing an analytical method such as GE ICM40B which would provide a detection limit of 0.02 g/t Ag.
- Ensure pulp duplicate data are collected and added to the database. Amec Foster Wheeler was unable to review pulp duplicate results.
- Drill samples from when the property was managed by Rio Tinto were sent to OMAC. Anagold should compare original assay values to those stored in the database.

12.11 Opinion on Adequacy

Amec Foster Wheeler is of the opinion that the QA/QC supports the information in the database, and that the database can be used for Mineral Resource estimation.

Risks and opportunities that may affect the Mineral Resource statement are as follows:

- During the validation process, Anagold was unable to provide collar documentation prior to 2015, down-hole survey documentation prior to 2015, and a significant number of drill logs were not able to be provided.
- A single CRM to quantify silver assay accuracy should be added.
- Amec Foster Wheeler finds the assay precision is adequate for resource estimation based on the core/RC duplicates.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Historical Testwork – Oxide Ore Heap Leaching

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

The heap leaching facilities were commissioned at the Çöpler Mine in late 2010, and have operated continuously since that time. Operations were continuing at the Report effective date.

13.1.1 Column Leach Testing - RDi

All testwork performed by RDi was on non-agglomerated crushed ore samples. Recovery rates for all column tests were generally fast with rapid leaching (by column standards) occurring in the first six days followed by a slow leaching component for the remainder of the time allowed. The slowest initial leaching rate was observed in the diorite ore from the Main Zone.

Column leach tests were carried out at three crush sizes: 80% passing 25 mm, 12.5 mm, and 6.4 mm, with most of the work performed with 12.5 mm material. Marble ores from the Manganese Zone showed a relatively small decrease, 0.05 g/t Au, in residue grade at the finer crush size. Re-crushing residues from column tests having a crush size of 12.5 mm to 6.4 mm and re-leaching them improved recoveries by an average of 5%. From the few tests run at 25 mm, recoveries were shown to be lower than seen from the 12.5 mm crushed-size material.

Good correlation was found in the plot of recovery against head grade for marble ore in the Manganese Zone.

Recoveries of 60-75% for marble and diorite ore are indicated from the tests at a low average cyanide consumption of between 0.5 and 0.6 kg/t NaCN.

13.1.2 Column Leach Testing – AMMTEC

Column leach testing was performed on individual oxide ore types for marble, metasediments, gossan, diorite and manganese diorite. The results indicated gold extractions ranging from 55% for gossan to 88% for marble with an average of 79%. This work supported scaled up commercial gold recovery in the range of 75%.

13.1.3 Cyanide Soluble Copper

Cyanide soluble copper trends from analytical tests, carried out on drill core composite samples prepared from nominally 7-10 m interval of core are summarized as follows:

- Marble lithology had the lowest total copper and cyanide soluble copper. Average soluble copper was 6 to 14% of the total copper for all three deposits, when leached at high temperature and with high cyanide-strength solutions. Total range of averages for marble lithology in the three deposits were approximately 300–700 ppm Cu. For column tests, average copper extraction in marble ores was 7%.

- Cyanide-soluble copper in other non-marble lithologies was significantly higher with extraction averages in the range of 10 to 50%. Total copper range of averages for non-marble lithologies was approximately 500–7,000 ppm Cu (0.05–0.7 percent).
- Extreme variability in copper to solution was evident, ranging from 1 to 40% in marble lithology and 1 to 70% in non-marble lithologies.

A relatively strong correlation of cyanide-soluble copper with total copper was observed for all ore types. The relationship can be described by an algorithm using regression analysis, where there is sufficient data.

13.1.4 Agglomeration Tests for the Heap Leach Process

Preliminary agglomeration test work was performed by Kappes-Cassiday Laboratories in Sparks, Nevada. Different additions of cement were used to determine optimum cement addition for a variety of ores.

13.1.5 Copper Carbon Loading and Stripping

Tests were carried out to determine the anticipated copper carbon loading and the ability to remove copper from the carbon using a cold cyanide-strip method. Results indicated:

- Copper carbon loading could be minimized by increasing the cyanide concentration of the solutions prior to adsorption.
- Copper loading on carbon was less than 3% with an initial copper feed grade of 8,450 g/t.
- Stripping of copper from carbon was highly successful at ambient temperatures with a 5% cyanide solution removing over 90% of the copper from the carbon, and less than 0.6% of the gold. Copper stripping was essentially complete in six hours.

13.1.6 Heap Leach Gold Recovery

The heap leaching process gold recovery assumptions have been updated to reflect actual performance of the operation between September 2010 and December 2015. The gold recovery assumptions for oxide ore are summarized in Table 13-1. Material that was previously considered within a transition zone adjacent to the oxidation boundary is not considered to be suitable for heap leach feed.

Table 13-1 Gold Recovery Assumptions for Heap Leaching of Material in the Çöpler Oxide Zone

Oxide Ore Type	Manganese	Marble	Main	Main East	Main West	West
Marble	78.4	75.7	68.6	78.4	75.7	75.7
Metasediments	66.8	66.8	66.8	66.8	66.8	66.8
Gossan	71.2	65.1	71.2	71.2	65.1	65.1
Diorite	71.2	62.3	71.2	71.2	62.3	62.3
Mn Diorite	71.2	62.3	71.2	71.2	62.3	62.3

Sulfide ore (material with >2% sulfide sulfur content) is not suitable for treatment by the heap leaching process and therefore no gold recovery assumptions are provided for this material.

The original gold recovery assumptions were developed by Mr. William Pennstrom in 2008, based on the results of column leach and bottle roll testing performed by RDi between 2005 and 2008. These recovery assumptions were revised and updated by Metallurgium for the current Mineral Resource and Mineral Reserve estimates 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 Mine between July 2011 and December 2015.
- Development of an Excel-based heap leach production model by KCAA (Perth, Australia) which was calibrated against actual gold production data at the Çöpler Mine from start-up of the operation in late 2010 through end of September, 2015.

The results of the column leach tests on monthly composite samples of heap leach feed range from 46% to 95% with an average of 76%. The results of the bottle roll tests on monthly composites of heap leach feed range from 40% to 92% with an average of 73%.

The recovery assumptions listed in Table 13-1 consider heap leaching of ore crushed to 80% passing 12.5 mm, agglomerated with lime and moisture to achieve consistently high quality agglomerates, and placed on a lined heap leach pad for treatment. The general process flowsheet is shown in Figure 13-1

The gold recovery assumptions provided in Table 13-1 represent a positive adjustment of 1.0476 applied to the original (2008) assumptions, reflecting the results of additional metallurgical testing and the results of the heap leach production model performance and calibration.

Figure 13-1 Process Flowsheet for Heap Leach

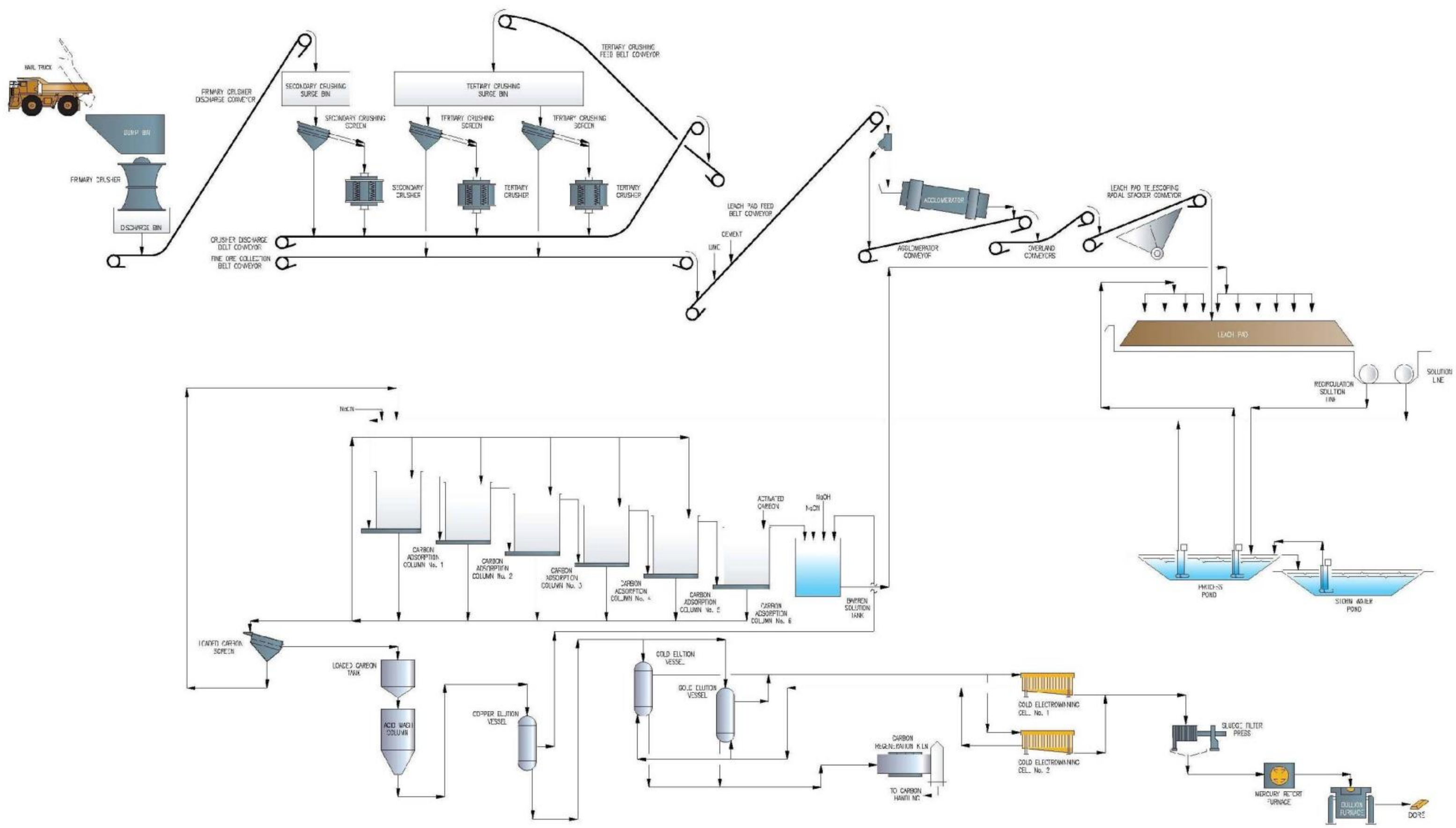


Figure prepared by Alacer, 2016.

13.2 Historical Testwork – Sulfide Ores

Historical testing for Alacer was conducted on samples from the sulfide resource in several phases. RDi performed several sulfide processing scoping-level investigations from 2006 to 2009. A two-phase program on sulfide resource samples was conducted at SGS Lakefield Research Limited (SGS) in 2009 and 2010 to support a PFS completed by Samuel Engineering (Samuel, 2011). A QEMScan (quantitative evaluation of minerals by scanning electron microscopy) mineralogy study on six oxide and three sulfide 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, 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 FS.

Initial metallurgical testwork carried out by RDi indicated that 11% to 30% of the gold content in the Çöpler sulfide material, as demonstrated by diagnostic leaching, may be amenable to whole-ore cyanidation. About 60% to 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 concentrates and tailings did not leach well using cyanide, even after being finely ground.

13.2.1 Mineralogy

In December 2008, Alacer had AMMTEC Ltd make QEMScan precious metals search (PMS), trace mineral search (TMS), and energy dispersive spectra signal (EDS)mineralogy analyses performed on three sulfide resource samples. Analyses were performed on samples of diorite, metasediments (MTS), and massive pyrite rock types.

The findings from the 2008 QEMScan analyses indicated that the gangue mineralization in the sulfide resource is composed mainly of quartz, micas/clays and feldspars (displaying relative abundances of approximately 31%, 27%, and 21%, respectively). The sulfide mineralization 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 is not part of the flowsheet, 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% sulfide sulfur. Recoveries of gold and sulfur to concentrate were 72.7% and 90% respectively. Flotation tailings assayed 0.68% Au and 0.48% sulfide sulfur.

The detailed mineralogical analysis is summarized in Table 13-2 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 submicroscopic 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.

Table 13-2 Gold Deportment in Flotation Separated Streams

Form & Carrier of Au	Con	Tails
Assayed Grade	10.187 ±0.167	0.837 ±0.028
Free/liberated gold grains		
• >40µm	0.106	0.004 *
• 5-40µm	0.346	0.003
• <5µm	0.871	0.146
Exposed Associated Gold Grains		
• free sulphides +5µm	0.350	0.018
• -5µm		
• rock-sulphide composites	0.125	0.052
• rock particles	0.021	0.035
Enclosed Associated Gold Grains		
• free sulphides +5µm	0.977	0.007
• -5µm	0.292	0.029
• rock-sulphide composites	0.338	0.023
• rock particles	0.014	0.031
Submicroscopic Gold		
• free sulphides +5µm	4.156	0.020
• -5µm	1.244	0.157
• associated sulphides	1.605	0.304
Total **(mineralogically accounted)**	**10.444** (102.5%)	**0.829** (99.0%)

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

Arsenopyrite was the sulfide mineral found to have the highest contained gold grade, averaging 123 ppm by one measure and 182 ppm by a second. Gold in pyrite was more than an order of magnitude lower than arsenopyrite and averaged 7 ppm. Marcasite, a mineral similar to pyrite chemically, carried 17.8 ppm Au. Of the gold contained in sulfides, 50% was found to be in arsenopyrite, 25% was in pyrite and 20% in marcasite.

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

13.2.2 Direct Cyanidation

Hazen performed direct cyanidation carbon-in-leach (CIL) tests at various grind sizes with no pretreatment 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 program.

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

13.2.3 Flotation Tests

Alacer conducted a flotation investigation at FLSmidth, in Salt Lake City, Utah, in 2013 to determine the potential to make a gold bearing sulfide concentrate that could be sold or processed by cyanidation as an alternative to POX. A series of flotation tests using various reagent schemes were performed.

The results from the flotation program show gold recoveries to flotation concentrate ranged from 55% to about 80%, with gold grades ranging from about 9 to 15 g/t. Weight recovery to concentrate was high, ranging from 10 to 30% of the flotation feed mass. Consistent with previous flotation investigations, the tests indicated that flotation gold recoveries will generally be low, even with high weight recovery to the flotation concentrate.

Two cyanidation tests were conducted, one on a flotation concentrate ground to a P80 of 7 µm, and the other on flotation tailings. The concentrate leach test gave a gold extraction of only 36.6%. This is consistent with previous testing and confirms that concentrate leaching would not be an attractive process for gold recovery. The tailings leach test gave a gold extraction of only 15.5% and was consistent with previous testing.

13.3 Testwork - Comminution

The comminution properties for the three major ore domains (metasediment, main diorite and manganese diorite) have been measured during all testwork stages, and the full set of results was analyzed by Amec Foster Wheeler to develop design parameters. The principal comminution characteristics that Amec Foster Wheeler relies on for grinding circuit design are the JK ore competence (Axb or drop weight index (DWI)), the Bond ball milling work index (BWI) and the Bond abrasion index (Ai). Competence drives semi-autogenous grind (SAG) mill selection, BWI drives ball mill selection and Ai is used to estimate media and mill liner consumption rates in the operating cost (Opex) calculations.

The results are summarized in Table 13-3. Note that although the SPI (SAG power index) values are shown in the table, they were not used in Amec Foster Wheeler's evaluations.

Table 13-3 Summary of Comminution Test Results

Ore Domain	Testwork Phase	BWI kWh/t	Ai g	DWI kWh/m^3	Axb -	SPI Minutes
Metasediments	PFS	15.5	0.238	6.45	41.6	
	DFS1-3	13.1	0.1801	3.68	72.4	
	DFS4	13	0.2584	3.04	84.6	76.7
	DFS4	12.6	0.2178	3.4	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	161.3
	DFS4	13		6.19	43	80.6
	DFS4	12				59
	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	0.1591	4.75	54.9	26.2
Manganese Diorite	PFS	13.4	0.033	2.69	104	
	DFS4	9.1				3.8
	DFS4	15.4	0.238	4.8	54.4	56.6
	DFS4	10.4				5.8
	DFS4	11.3		1.22	205.3	10.3
	DFS5	14.6	0.149	4.01	64.8	8.5
Main Diorite	PFS	13.1	0.063	1.7	145	
	DFS1-3	10.3	0.0498	1.58	164.4	
	DFS4	14	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	2.1	122.5	19.1
	DFS4	12.3		3.4	80	20.9
	DFS4	13.4	0.223	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

For each measurement (except Axb, which is strongly related to DWI) the cumulative ranking distributions (S-curves) have been plotted. The ball mill work index results are plotted in Figure 13-2. The BWI design value is used to select the ball mill for the circuit.

Figure 13-2 S-Curve for Bond Ball Mill Work Index (BWI)

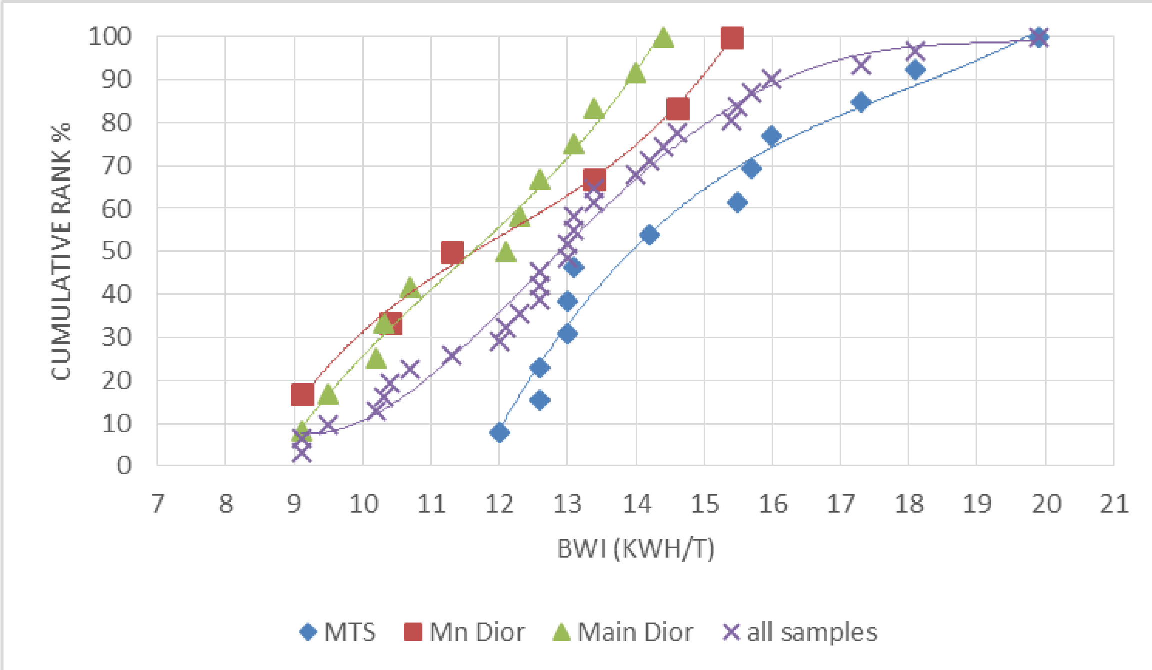


Figure prepared by Amec Foster Wheeler, 2016.

The curves show that the MTS is the hardest of the main ore types, and that the two diorite domains are softer and have similar ball mill grindability characteristics. The "all samples" curve shows that the 80th percentile of the measured samples is about 15 kWh/t, and the average grindability is 13 kWh/t. Overall, Çöpler ore is classed as slightly softer than average with respect to ball mill grinding. The coefficient of variation (COV = SD / Average) is 20%, a relatively high value for BWI. The high variability value, even with a modest amount of blending, will translate to variable recirculating loads, and variable grind sizes for leach feed. For design purposes, a mine plan weighted average of 80th percentile values of the ore types was used, giving a slightly more conservative value of 15.6 kWh/t.

The SAG mill competence results (DWI) are plotted in Figure 13-3. SAG mill competence is used in the selection of the SAG mill for the circuit.

Figure 13-3 S-Curves for Competence as measured by SMC test and Drop Weight Index (DWI)

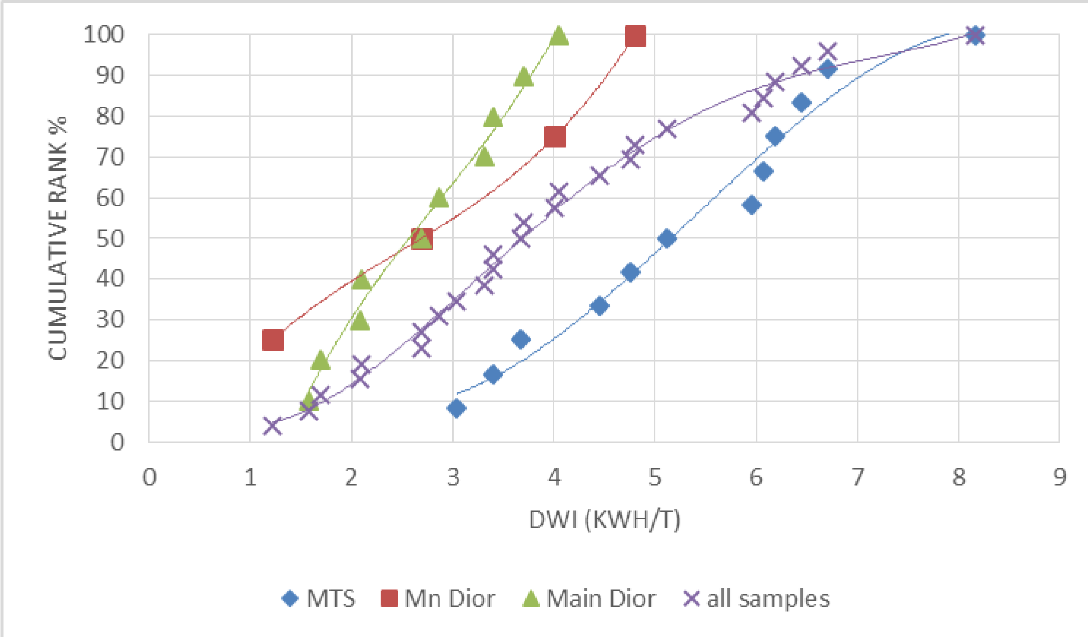


Figure prepared by Amec Foster Wheeler, 2016.

Again, MTS is the hardest (most competent) domain of the three, and the two diorite samples are similar. It could be argued that the manganese diorite is more competent than the main diorite, but there are not sufficient manganese diorite samples to make this judgement. Overall, the average DWI is 3.7 kWh/m^3, which places the ore in the low competence category. The 80th percentile competence value is 5.3 kWh/m^3, close to the average value for ores. The coefficient of variation (COV) for competence is 45%, a relatively high value and indicative of high potential for variable grinding circuit throughput, even with a modest amount of blending. For design purposes, a mine plan weighted average of 80th percentile values of the ore types was used giving a slightly more conservative value of 5.37 kWh/m^3.

The Bond Abrasion index results (Ai) are plotted in Figure 13-4. The abrasion index is used to estimate consumption of steel grinding media in SAG and ball mills and to estimate liner wear in crusher and mills.

Figure 13-4 S-Curve for Bond Abrasion Index (Ai)

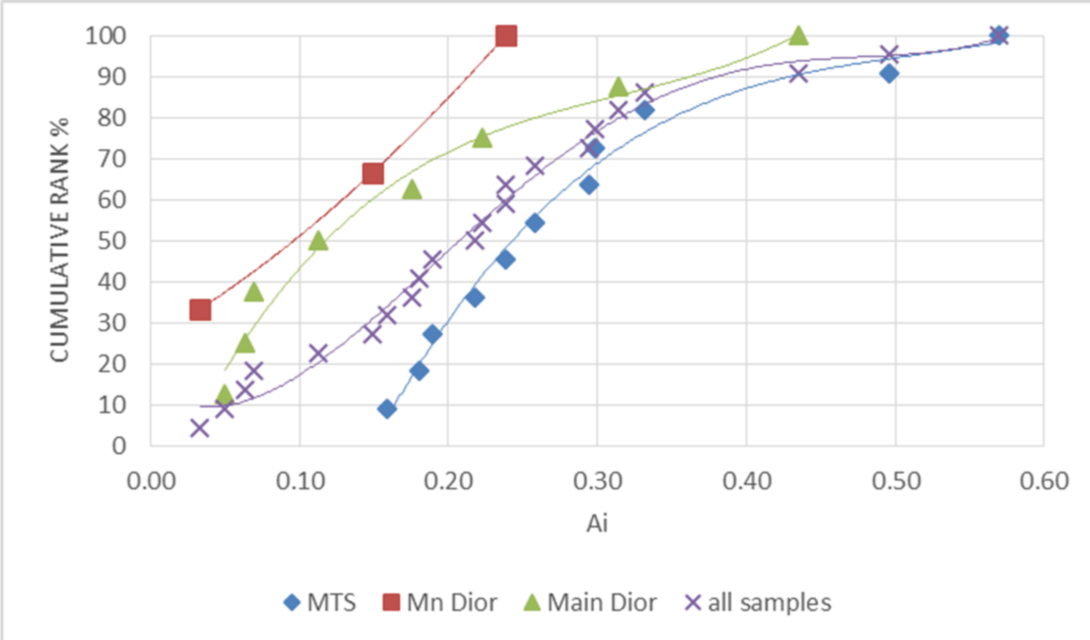


Figure prepared by Amec Foster Wheeler, 2016.

Again, MTS is the worst of the ores with the highest abrasion indices on average. It appears that main diorite is worse than the manganese diorite, but there are insufficient manganese diorite samples tested to make this judgement.

As the Ai value is used to estimate wear rates over a year it is not appropriate to use an upper value for design calculations such as an 80th percentile (0.31). The average is the appropriate value to use in these calculations, a value of 0.23. At this level the Çöpler ore has a low abrasion rating, and this results in a relatively low annual consumables (liners and grinding balls) cost.

About 10% of the samples are very abrasive (>0.5), and this may result in short-term, high consumption rates.

The SPI results are shown in Figure 13-5. SPI is used in an alternative proprietary method of designing SAG mills and is included here for completeness. These values were not used by Amec Foster Wheeler.

Figure 13-5 S-Curve for SAG Power Index (SPI)

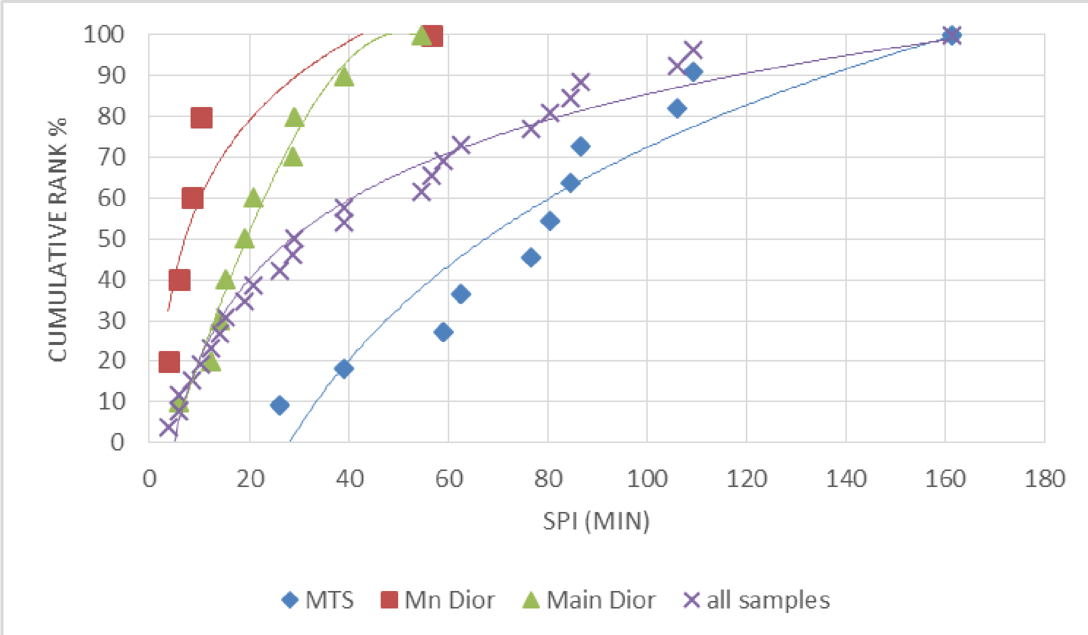


Figure prepared by Amec Foster Wheeler, 2016.

In this instance, manganese diorite is definitely ranked softest, followed by main diorite and the hardest, once again, is MTS. The average SPI value is 30 minutes and the 80th percentile is 80 minutes. The very high COV for this measure of 86% is part of the justification for it not being used in the design.

13.4 Testwork - POX

Three continuous pilot plant programs have been conducted for the Sulfide Expansion Project; the first two programs at Hazen Research, Inc. (Hazen) comprising a total of four test campaigns, and the third program at SGS Lakefield Oretest, Perth (SGS Perth). Three campaigns were completed during the first pilot plant program with the first campaign commencing in February 2012. The second pilot program incorporating one campaign, was conducted in December 2012. The third pilot program, 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 program included the following continuous circuits: acidulation, POX autoclave, hot cure (HC), primary neutralization (PN), six-stage counter-current decantation (CCD) and mixed sulfide precipitation (MSP). Ore preparation (grinding), cyanidation, activated carbon gold recovery, cyanide destruction, tailings neutralization, and final tailings production were all completed on a batch basis.

Campaign 1 during the first pilot plant program 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 Alacer to be most representative of early commercial plant operation.

The following conditions were targeted in Campaign 3, Run 27:

- Feed ore: 20% manganese diorite, 80% Master Composite 2 at a P80 grind size of 100 µm
- Acidulation extent: 40% carbonate decomposition
- Acidulation temperature: 220°C
- Acidulation retention time: 60 minutes
- Feed slurry % solids: 35 % w/w
- Total non-condensable overpressure: 100 psi (680 kPa)

The purpose of the second continuous pilot plant program was to conduct additional pilot plant studies in support of a feasibility study completed by Jacobs Engineering on processing a sulfide ore from the Çöpler gold mine in Turkey. As part of this Campaign 4 was used to obtain key process information to populate the process design criteria document.

The following primary objectives were set for Campaign 4:

- To operate the pilot plant for a prolonged period at the optimum conditions for the autoclave as established in the previous pilot testing (Campaigns 1-3), specifically based on the conditions tested during the last run (Run 27) of Campaign 3:
    - Acidulation extent: 38-42% (sufficient for 15-20 g/L free acid in the autoclave discharge)
    - Autoclave temperature: 220°C
    - Autoclave retention time: 60 minutes
    - Feed slurry % solids: 35% w/w
    - Total non-condensable overpressure: 100 psi (680 kPa).
- To obtain solid-liquid separation (SLS) data on slurry samples collected during steady-state operations of the pilot plant.
- To characterize the hot cure unit operation to determine whether this process is effective for the conversion of jarosite to hematite, possibly improving the solid-liquid separation properties of the POX residue.
- To produce final tailings samples for rheological characterization to assist in specifying and sizing the tailings pumping and piping system.
- To operate the sulfide precipitation circuit to achieve at least 95% copper removal and evaluate the option of significantly reducing the retention time in the circuit by trialing an inline reactor.
- To evaluate the effect of increasing oxygen overpressure in the autoclave on the formation of hematite rather than jarosite.
- To provide head samples for cyanide leaching and detoxification testwork at McClelland Laboratories (McClelland).

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 neutralization (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 one-quarter drill 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, cation exchange capacity (CEC) swelling-clay analysis, and samples for initial and detailed chemical analyses. The VS2 work was completed as part of Hazen Project 11677.

After selecting sample material for 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:

- Meta-sediments (28.5%)
- Main 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 program comprised a single pilot plant campaign, Campaign 5, which was conducted at SGS Perth during August/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.

A summary of the Campaign 4 and 5 testwork programs is discussed below, highlighting how the key process design criteria numbers were developed.

13.5 Campaign 4 - Hydrometallurgical Testwork

The data required for design of several unit processes including solids thickening, slurry rheology though the process, and copper precipitation, were not totally developed in Campaigns 1 to 3 due to the focus on determining the best POX operating conditions. This resulted in non-steady state operation in the pilot processes downstream of the POX autoclave, and reduced the amount of sample available for solids/liquid separation testing.

Campaign 4 had the following objectives:

- To confirm pilot plant operation of the proposed FS flowsheet using selected design conditions
- To develop thickener sizing data
- To determine the effectiveness of the hot cure circuit on jarosite decomposition
- To provide samples for tailings slurry rheology testing

- To provide samples for cyanidation and adsorption testing at McClelland Laboratories
- To investigate tailings cyanide destruction.
- To prove and optimize the sulfide precipitation circuit for greater than 95% Cu removal.
- To examine the effect of increasing autoclave oxygen overpressure on the jarosite formation.

13.5.1 Acidulation

The pilot acidulation circuit comprised four agitated tanks maintained at a nominal temperature of 65°C. Concentrated sulfuric acid, recycle solution (decant thickener overflow) and fresh ore were added into tank 1 targeting complete acidulation and a pH of 2.0 in tank 2. Fresh ore was added into tank 3 to achieve the overall target acidulation extent of approximately 40% and a pH of 3.0 in tank 4. The pilot acidulation circuit incorporated the recycle of autoclave discharge thickener (THK2) overflow, initially using a synthetic solution before switching to decant thickener overflow. Table 13-4 summarizes the results of the acidulation circuit during Campaign 4, consisting of five 12 hour periods.

Table 13-4 Campaign 4 Acidulation Operating Conditions and Results

Description	Units	Period				
		Acid MB 1	Acid MB 2	Acid MB 3	Acid MB 4	Acid MB 5
Residence Time	minutes	97	122	140	105	111
Tank 2 Acidulation	%	94.3	95.9	96.3	96	96.1
Acid addition						
- Recycle	kg/t	105.7	58.3	54.8	26.3	36.5
- Fresh acid	kg/t	115.4	60	56.2	48.8	45.2

13.5.2 POX

The Campaign 4 POX program consisted of the following three runs:

- Run 28 at target conditions with 16.7% manganese diorite.
- Run 29 with 20% manganese diorite at the same conditions as Run 28.
- Run 30 with the same feed as Run 29 at a higher oxygen overpressure.

Acidulated ore was blended with unacidulated ore to achieve an overall acidulation target for the autoclave feed. All of the runs generally operated without serious upsets. The Campaign 4 POX circuit operating conditions and results are summarized in Table 13-5.

Table 13-5 Campaign 4 POX Circuit Operating Conditions and Results

Description		Run 28			Run 29				Run 30	
Mass Balance Period	Units	MB1	MB2	MB3	MB1	MB2	MB3	MB4	MB1	MB2
Acidulation Extent	%	42			38				38	
POX Temperature	°C	220			220				220	
Feed Solids Content	% w/w	37.9	36.9	36.9	36.7	36.9	36.5	37.7	36.8	36.5
Residence Time	min	63	62	62	62	66	65	62	61	62
Oxygen Overpressure[1]	kPa	347	335	331	367	537	395	392	534	731
Feed Solids										
Copper	% w/w	0.165	0.19	0.175	0.177	0.174	0.186	0.172	0.169	0.165
Carbonate[2]	% w/w	1.85	1.8	1.85	2	2	2.05	1.95	2.1	2.15
Sulfide Sulfur	% w/w	4.28	4.16	4.03	4.2	3.86	3.7	4.5	4.88	3.98
Sulfur to carbonate ratio	-	2.3	2.3	2.2	2.1	1.9	1.8	2.3	2.3	1.9
Extraction/Oxidation										
Copper	%	83.2	91.6	87.4	88.6	89.1	88.6	87.6	89.8	91.1
Carbonate	%	97.3	97.3	97.3	97.5	97.5	97.6	97.5	97.6	97.6
Sulfide Sulfur	%	94.5	97.9	96.1	97.1	97.9	97.1	98	98.6	98.7
POX Discharge Solution (Before Flashing)[3]										
Free Acid	g/L	22.5	24.6	23.3	22.1	21.1	21.3	22.1	20.6	21.1
Total Iron	g/L	1.2	1.43	1.35	1.41	1.31	1.19	1.59	1.03	1.11
Ferrous	g/L	0.4	0.25	0.25	0.27	0.27	0.33	0.37	0.16	0.16
Ferric to Ferrous ratio	-	2	4.7	4.4	4.2	3.9	2.6	3.3	5.4	5.9
Arsenic	g/L	0.11	0.12	0.11	0.1	0.11	0.1	0.11	0.1	0.1
Discharge Solids										
Total Sulfur	% w/w	3	2.94	3.05	2.99	3.06	2.89	3.16	3.05	3.1

1) *Calculated from "O_2 and non-condensable overpressure, psi" and "Exhaust O_2 concentration (dry gas), mol% O_2".*
2) *Carbonate grades expressed as C(CO_2-based) in the Hazen report*
3) *Concentrations expressed as % w/w in the Hazen report*

The following conclusions can be drawn for the process design criteria from these results:

- Autoclave residence time of 60 minutes and a temperature of 220°C resulted in:
  - Sulfide sulfur oxidation extents of greater than 94%, with the majority of runs greater than 97%.
  - Copper extractions of 83% to 91%.
  - Carbonate decomposition extents of greater than 97%.
- Autoclave nominal discharge free acid concentration of 22.5g/L was maintained throughout the run.
- Total sulfur grades in the autoclave discharge ranged between 2.89 and 3.16% w/w.
- Increasing the oxygen overpressure during Run 30 had the following effects:
  - Increased the sulfide oxidation extent and possibly copper extraction
  - Decreased the discharge total iron and ferrous concentrations
  - No noticeable effect on carbonate decomposition

- No noticeable effect on discharge total sulfur grade, which is an indicator of changes to the amount of jarosite precipitated
- No noticeable effect of the discharge solution concentrations of free acid or arsenic.

Using the detailed mass balances provided in Appendix D02 of the MC4 Hazen report, Amec Foster Wheeler inferred the sulfide oxidation kinetic profile. This profile was adjusted to allow for the differences between the pilot and full scale autoclave configuration, as presented in Figure 13-6. In addition to the sulfide oxidation profile, the results from these campaigns were also used to specify the rest of the POX chemistry, including ferrous oxidation, ferric hydrolysis, arsenic precipitation and aluminum solubilization.

Figure 13-6 Campaign 4 POX Sulfide Oxidation Kinetics Profile

Figure prepared by Amec Foster Wheeler, 2016.

13.5.3 Hot Cure

Jacobs incorporated the hot cure (HC) circuit into the Campaign 4 flowsheet to determine whether jarosite compounds formed in the autoclave would be decomposed or converted to hematite. The circuit comprised four tanks maintained at 90°C for a nominal residence time of 240 minutes. The operating conditions and results for the Campaign 4 HC circuit are provided in Table 13-6.

Table 13-6 Campaign 4 HC Circuit Operating Conditions and Results

Description	Units	HC MB 1	HC MB 2	HC MB 3	HC MB 4	HC MB 5	HC MB 6	HC MB 7	HC MB 8
Residence Time	min	272	258	262	379	273	277	295	301
Digestion Extent									
Total Sulphur	%	2.3	2.1	-2	2.5	-2.6	-2.4	3.3	3.2
Iron	%	2.3	2.2	1.5	2.4	1.1	1.8	2.6	2.3
Arsenic	%	2.9	3	1.9	1.7	1.8	1.9	3.1	3.5

The results from the Campaign 4 HC circuit indicate the following:

- Solubilization extents for iron and arsenic were 1.1-2.3% and 1.7-3.5% respectively, possibly indicating partial digestion of ferric arsenate (scorodite).
- The digestion of total sulfur ranged between 0 and 3.3% (relative). From this, a minor digestion of solid sulfate sulfur (either jarosite or basic iron sulfate) can be inferred.

Based on the minor solid sulfate sulfur solubilization extent and the partial digestion of arsenic, the HC circuit was not incorporated into the full scale flowsheet.

13.5.4 Iron/Arsenic Precipitation

The key objective of the iron/arsenic precipitation circuit was to precipitate ferric iron and arsenic from solution using limestone, prior to CCD thickening. The feed to the iron/arsenic precipitation circuit was underflow from the hot cure discharge thickener (THK2). The operation of the iron/arsenic precipitation circuit was split into six 12 hour periods. The operating conditions and results for the iron/arsenic precipitation circuit from Campaign 4 are shown in Table 13-7. Note that the addition of ferric sulfate solution was not required as there was sufficient soluble ferric in the feed to the circuit to promote the precipitation of ferric arsenate.

Table 13-7 Campaign 4 Iron/Arsenic Precipitation Operating Conditions and Results

Description	Units	PN MB 1	PN MB 2	PN MB 3	PN MB 4	PN MB 5	PN MB 6
Temperature	°C	65	65	65	65	65	65
Tank Slurry pH	-	2.9	2.9	2.8	2.8	2.6	2.8
Residence Time	min	160	122	121	127	139	139
Feed Solution[1]							
Total Iron	mg/L	1700	1540	1470	1340	1060	942
Ferrous	mg/L	970	893	918	806	458	405
Ferric	mg/L	730	647	552	534	602	537
Arsenic	mg/L	98	91	82	67	65	60
Copper	mg/L	765	736	675	596	445	388
Aluminium	mg/L	919	905	855	747	535	456
DischargeSolution[1]							
Total Iron	mg/L	918	880	853	855	604	416
Ferrous	mg/L	915	901	619	903	495	397
Ferric	mg/L	3	0	234	0	109	19
Arsenic	mg/L	11	15	15	10	9	2
Copper	mg/L	702	758	662	642	439	374
Aluminium	mg/L	543	787	726	652	483	377
Precipitation Extents[2]							
Total Iron	%	47.4	44.3	43.4	37.9	45.1	57.4
Ferrous	%	8.1	1.7	34.3	0	0	5.4
Ferric	%	99.6	100	58.7	100	82.6	96.6
Arsenic	%	89.1	83.9	82.2	85.5	86.7	96.8
Copper	%	10.6	0	4.4	0	5	7
Aluminium	%	42.5	15.2	17.2	15.1	13.1	20.2

1) *Solution composition is expressed as % w/w in the Hazen Campaign 4 report*

2) *Extents calculated by Amec Foster Wheeler*

The results from the Campaign 4 iron/arsenic circuit pilot testing indicate the following:

- Arsenic was precipitated by greater than 80% for the entire run
- Virtually all of the ferric iron was precipitated from solution
- The ferrous oxidation extent was typically less than 8%

The solubility of both copper and aluminum at these conditions is approximately 600 mg/L.

13.5.5 Sulfide Precipitation

A continuous sulfide precipitation circuit for the recovery of copper was tested during Campaign 4. CCD 1 thickener overflow containing aqueous copper was fed to a static mixer where sodium hydrosulfide (NaHS) was added to precipitate copper sulfide. An aging coil was installed on the discharge of static mixer providing additional residence time, before discharging into two agitated tanks in series. Tank discharge was thickened and a portion of the thickener underflow was recycled to the static mixer, providing seed material to promote particle growth. The operating conditions and results from the sulfide precipitation circuit are provided in Table 13-8.

Table 13-8 Campaign 4 Sulfide Precipitation Circuit Operating Conditions and Results

Description	Units	MSP MB 1	MSP MB 2	MSP MB 3	MSP MB 4	MSP MB 5	MSP MB 6
Temperature	°C	50	62	61	61	61	61
Tank Slurry pH	-	4.4	4.8	4.8	4.5	4.5	4.3
Residence Time (Static Mixer + Tanks)	min	99	64	62	80	94	93
Solution Copper Concentration							
MSP Feed	mg/L	256	262	332	315	316	293
MSP Discharge	mg/L	2180	990	800	430	30	40

The results represent averages across nominally 12 hour operating periods and do not clearly demonstrate the effective operation of the sulfide precipitation circuit. Therefore, the design of the full scale copper precipitation circuit was primarily based on Amec Foster Wheeler’s experience with other comparable commercial scale installations. However, after reassessing the economic viability of the circuit, it was excluded from the full scale flowsheet.

13.5.6 CIL and Cyanide Destruction

Samples of CCD 3 underflow were taken throughout Campaign 4 for conditioning (slurry neutralization), cyanide leaching and carbon adsorption batch testing. CIL tests were performed at 25% solids under ambient conditions, maintaining a cyanide concentration of 2 g/L (as NaCN). The retention times for conditioning and leaching were 1.5 and 24 hours respectively. A summary of the CIL tests conducted during Campaign 4 is shown in Table 13-9.

Table 13-9 Campaign 4 CIL Test Summary (By Hazen)

Test No	Au Head Assay (g/t)		Consumption (kg/t)		Au Extraction % (products basis)	Cu in Leach Residue (%)
	Direct	Calculated	$Ca(OH)_2$	NaCN		
1	2.5	2.45	11.79	2.76	96	0.01
2	2.4	2.39	9.08	2.02	96	0.01
3	2.5	2.41	15.37	2.12	96	0.009
4	2.5	2.23	10.06	2.31	96	0.01
5	2.7	2.67	11.25	2.56	96.4	0.011
6	2.7	2.65	8.62	2.44	96.3	0.008
7	2.3	2.22	7.88	2.77	95.6	0.006
8	2.5	2.47	7.07	2.42	96	0.007
9	2.5	2.4	10.97	2.89	96	0.007

Gold extractions were consistently between 95 and 96%, with lime consumptions of 7-15 kg/t, and NaCN consumptions of 2-3 kg/t. Copper in the CIL leach residues ranged from 0.007 to 0.011%, compared with 0.164% Cu in the MC4 feed ore, indicating that about 95% of the copper was extracted across the entire process.

At the conclusion of Campaign 4, a sample of CCD3 underflow discharge was collected and sent to McClelland for cyanide leaching and gold adsorption testwork. The cyanide leaching residue sample was then delivered to the Cyanco Corporation laboratory (Cyanco) to conduct cyanide destruction testing using the INCO cyanide destruction process. The cyanide leaching and gold adsorption and testwork was reported separately to the cyanide destruction testwork. McClelland submitted the final detoxified tailings sample to Western Environmental Testing Laboratory for synthetic precipitation leaching procedure (SPLP) extraction and extract analyses to categorize the impounded tailings.

Results from the reports indicated:

- Gold recoveries of 92.8 to 93.7% were achieved
- The initial target pulp density of 40% w/w solids was found to cause severe pulp viscosity problems. Therefore, the target was reduced to 33% w/w solids.
- The optimum cyanide concentration (as NaCN) for cyanide leaching of the POX residues was 0.40 g/L (maintained during leaching).
- Increasing the cyanide leach NaCN concentration beyond 0.40 g/L did not increase the overall gold extraction or the gold extraction kinetics.
- Cyanide consumption increased with increased cyanide leach concentration.
- “Extremely high” lime requirements were observed, caused by adjusting the pH of the CCD 3 underflow pulp from 2.8 to 10.5. Over 95% of the lime required was added during the initial pulp pH adjustment, with the remaining 5% added during leaching.

- The carbon adsorption rate was relatively rapid for a pulp carbon concentration of 1.0 g C/L pulp, and adsorption was effectively complete after 8 hours of carbon contact.
- A maximum loading of 2,610 mg Au/kg C was achieved, although this was at ambient temperature, not at the design operating temperature of approximately 43°C.
- Six stages of carbon contactors can produce an adequate barren solution gold concentration (<0.01 mg/L).
- The optimum INCO detoxification conditions for treating CIL residue the to reduce the CN_{WAD} concentration to 5 ppm, were determined to be:
  - A sodium metabisulfite addition rate of 4 g SO_2 per g CN_{WAD}
  - Copper sulfate addition to achieve 25 ppm Cu^{2+}
  - A retention time of 2 hours.
- Based on the SPLP extract analysis, the impounded tailings would be categorized as “non-hazardous” by the United States Environmental Protection Agency (USEPA).

These results were used to support the FS. However, during review as part of detailed engineering, various inconsistencies were noted such as the temperatures and slurry densities used for testing. The design of the circuit was updated and further supported by additional leach, adsorption and rheology testwork performed as part of the next test program, Campaign 5.

The sodium metabisulfite addition rate recommended by McLelland/Cyanco for the detox circuit was adopted for the FS. However, the addition rate was reduced to 1 mole SO_2 per mole CN_{WAD} after further testing during Campaign 5.

13.5.7 Tailings Neutralization

The tails neutralization circuit was designed to precipitate residual metal from solution prior to thickening and tails disposal. The target operating conditions for the tails neutralization circuit were a total residence time of 180 minutes and a slurry pH of between 10 and 10.5. The summary operating conditions are shown in Table 13-10.

Table 13-10 Tails Neutralization Operating Conditions

Period	Units	TN MB 1	TN MB 2	TN MB 3
Temperature	°C	60	59	59
Tank Slurry pH	-	10.6	10.5	10.4
Residence Time	minutes	152	146	164
TN Feed Solution				
Iron	mg/L	394	465	541
Ferrous	mg/L	391	495	543
Arsenic	mg/L	5	6	7
Copper	mg/L	108	161	113
Silicon				
Aluminium	mg/L	632	571	623
Manganese	mg/L	5050	4810	5140
Sodium	mg/L	100	146	173
Calcium	mg/L	586	487	615
Magnesium	mg/L	2410	2140	2370
Mercury	mg/L	0.005	0.005	0.005
Sulfur Total	mg/L	7900	7140	8000
pH	-	2.6	2.7	2.6
TN Discharge Solution				
Iron	mg/L	1	12	16
Ferrous	mg/L	0	0	111
Arsenic	mg/L	0	1	2
Copper	mg/L	0	6	11
Silicon		10	10	10
Aluminium	mg/L	1	6	13
Manganese	mg/L	0	30	50
Sodium	mg/L	45	63	64
Calcium	mg/L	650	675	613
Magnesium	mg/L	0	10	30
Mercury	mg/L	0	0	0
Sulfur Total	mg/L	490	570	740
pH	-	10.9	10.5	10.5

13.5.8 Solid-Liquid Separation Tests

Samples of slurries and dilution waters were collected from the Campaign 4 pilot plant run by Hazen personnel and were provided to four thickener vendors and one solids-liquid testing consultant, Pocock Industrial (Pocock). The four thickener vendors were contracted to perform solid-liquid separation testwork to develop thickener sizing design parameters using their respective test procedures and sizing protocols, which could then be used for thickener sizing. Pocock was contracted to conduct thickener sizing tests and to test and provide design data for the filtration of the process final tailings stream.

Testing was conducted for the following duties:

- Grinding thickener
- POX feed thickener
- Decant thickener
- CCD thickeners
- Tailings thickener
- Copper precipitation thickener

Each of the five firms performing solid-liquid separation testing were provided with target design criteria as shown in Table 13-11 to measure and compare the results of the vendor tests.

Table 13-11 Vendor Thickener Design Criteria

Design Criteria for Thickener Sizing	Units	Grinding	POX Feed	Decant	CCD	Tailings
Operating Temperature	°C	44.5	70	73	64	50
Solids Feed Rate	t/h	245	245	243	254	284
Solution Density	g/L	1	1.02	1.02	1.02	1
Solids Density	g/L	2.87	2.83	2.63	2.62	2.58
Pulp pH	-	7 - 8	1 - 3	1 - 2	2.5 – 3.5	10.5
Feed Percent Solids	% w/w	28.6	13.9	13.5	33.2	31.5
Target Underflow Percent Solids	% w/w	55	35	45	40	40 - 45
DFS PDC Underflow Percent Solids	% w/w	45	45	40	40	37

The test results showed that the process streams could be thickened, but in several cases the target underflow percent solids were not achieved. Additionally, the results were inconsistent across the vendors without an obvious pattern or bias.

After commencing work on the Çöpler Project, Amec Foster Wheeler updated the target underflow densities after a review of the relevant testwork with the revised target design criteria shown in Table 13-12. Three design scenarios were considered for each duty:

- Nominal case (NOM), based on the nominal mass balance flow rate and underflow density
- Design case 1 (DSN 1), based on the design underflow density
- Design case 2 (DSN 2), based on the maximum solids throughput.

Table 13-12 Revised Thickener Design Conditions

Duty		Grinding Thickener			POX Feed Thickener			Decant Thickener		
Parameter	Units	NOM	DSN 1	DSN 2	NOM	DSN 1	DSN 2	NOM	DSN 1	DSN 2
Solids	t/h	245	245	282	245	245	282	240	240	276
U/F Solids	% w/w	45	50	45	45	48	45	40	45	40
Duty		CCD Thickener			Tails Thickener			CuS Thickener		
Parameter	Units	NOM	DSN 1	DSN 2	NOM	DSN 1	DSN 2	NOM	DSN 1	DSN 2
Solids	t/h	253	253	291	282	282	325	6.9	6.9	7.9
U/F Solids	% w/w	40	43	40	34	36	34	15	20	15

Amec Foster Wheeler requested the preferred equipment supplier review the project thickener sizes based on the Campaign 4 testwork program and the revised design criteria. The sizes agreed with the vendor are shown in Table 13-13.

Table 13-13 Revised Thickener Sizing

Thickener Duty	Flux, $t/m^2.h$	Diameter, m
Grinding	0.5	27
POX Feed	0.35	32
Decant	0.35	32
CCD	0.3	35
Tails	0.22	44
CuS	0.01	32

Additional thickener testwork was conducted by Outotec in August/September 2015 as part of the pilot plant Campaign 5 completed at SGS Perth. The outcomes from this testwork are discussed in Section 13.7.7.

13.6 Campaign 5 Comminution Testwork

As part of the Campaign 5 testwork undertaken at SGS Perth, additional samples of the dominant ore types making up the feed to the plant (main diorite, metasediments and manganese diorite) were subjected to a comminution testwork program designated CP100. A comparison of these results with the FS 80th percentile design values are provided in Table 13-14.

Table 13-14 Comparison of DFS and Campaign 5 Comminution Parameters

Data Set		Metasediments		Manganese Diorite		Main Diorite	
Parameter	Units	DFS 80th percentile	CP100	DFS 80th percentile	CP100	DFS 80th percentile	CP100
Bond Ball Mill Work Index	kWh/t	17	16	13.8	14.6	13.1	14.4
Bond Abrasion Index		0.364	0.159	0.197	0.149	0.296	0.176
Ore SG		2.7	2.60*	2.6	2.61*	2.64	2.57*
Minnovex SPI	min	106.7	26.2	28.8	8.5	33.2	6.5
SMC Parameters							
Axb		41.6	54.9	67.2	64.8	79.6	63.6
ta		0.41	0.55	0.76	0.65	0.9	0.64
Mia	kWh/t	18.8	15.3	13.2	13.4	11.4	13.6
DWI	kWh/m^3	6.45	4.75	3.96	4.01	3.35	4.05

**Apparent SG determined during SMC testing on rock specimens that may contain closed voids*

Comments on the comparative differences between data sets is provided below:

- Metasediments – the CP100 sample exhibited significantly lower ore competency and hardness than the FS 80th percentile values. The abrasivity was also much lower in the CP100 sample.
- Manganese diorite – the CP100 sample was slightly more competent and harder than the DFS 80th percentile values. The abrasivity was similar for both samples.
- Main diorite – the CP100 sample was considerably tougher and harder than the DFS 80th percentile values. The abrasivity was significantly lower for the CP100 sample.

Comparing results in this manner does not allow a clear conclusion to be drawn on the effect on mill capacity as feed will always be blended. A more useful approach is to use the values in prescribed ore blends under consideration, namely the May 2015 “LOM blend” and the “Early Years” blend. A comparison of calculated parameters in proportions assigned to these blends is provided as Table 13-15.

Table 13-15 Comparison of Ore Blend Comminution Parameters

Data Set		May 2015 LOM Blend		Early Years Blend	
Parameter	Units	DFS 80th percentile	CP100	DFS 80th percentile	CP100
Blend Composition					
Metasediments	%	60.2	60.2	32.1	32.1
Manganese Diorite	%	25.2	25.2	49.3	49.3
Main Diorite	%	14.6	14.6	18.6	18.6
Total	%	100	100	100	100
Bond Ball Mill Work Index	kWh/t	15.6	15.4	14.7	15
Bond Abrasion Index		0.312	0.159	0.269	0.157
Ore SG		2.67	2.6	2.64	2.6
Minnovex SPI	min	76.4	18.8	54.7	13.7
SMC Parameters					
Axb		49.9	58.3	57.5	61.1
ta		0.51	0.58	0.61	0.61
Mia	kWh/t	16.5	14.5	14.8	13.9
DWI	kWh/m^3	5.37	4.46	4.65	4.26

Based on the above comparison, the CP100 based feed blend has comparable ore hardness and slightly lower ore competency than the LOM blend based using the FS 80th percentile parameters. Abrasivity to liners and grinding media is also expected to be slightly lower with the CP100 data set. Overall, the CP100 data does not indicate that any significant deviation to the current design is required. The slight hardness increase seen in the "Early Years" blend is expected to be handled by the design margin adopted in the ball mill selection.

13.7 Campaign 5 Hydrometallurgical Testwork

Campaign 5 piloting testwork was conducted by SGS Perth during August 2015. The objectives of Campaign were:

- To investigate the performance of composite and individual lithologies at "high" and "low" acidulation extents using the current flowsheet
- To provide confirmatory data for detailed design including thickener sizing

Additional acidulation batch testing was performed after the main pilot run to improve the understanding of the acidulation chemistry.

Two composite and three lithology samples were tested during Campaign 5 including; Master Composite 5 (MC5), Master Composite 6 (MC6), Metasediments (MTS), Main Diorite (Main D), and Manganese Diorite (Mn D). MC5 and MC6 represent the first three years of operation and a LOM blend respectively. The gold, sulfide sulfur and carbonate head assays for the five samples that were tested during Campaign 5 are provided in Table 13-16.

Table 13-16 Campaign 5 Head Assays

Sample	Au	S^{2-}	CO_3	$S^{2-} : CO_3$
	ppm	%	%	-
Master Composite 5 (MC5)	5.50	4.45	5.45	0.82
Master Composite 6 (MC6)	5.81	5.07	4.39	1.15
Metasediments (MTS)	6.44	4.19	4.95	0.85
Main Diorite (Main D)	5.14	2.99	4.95	0.60
Mn Diorite (MnD)	5.03	6.72	6.60	1.02

The MC5 composite was prepared prior to the start of the acidulation testwork but the MC6 composite was prepared part way through the run using excess sample. The MC6 composite was seen as a link to the MC4 sample for Campaign 4. The compositions were provided to SGS Perth by Anagold (Table 13-17).

Table 13-17 Composite Sample make-up

Sample	MC5	MC6 (net)	MC6 (actual)
	%	%	%
Metasediments (MTS)	25	59.6	46.8*
Main Diorite (Main D)	25	14.8	2.0*
Mn Diorite (MnD)	50	25.6	-
LA MC5	-	-	35.3*
MC5	-	-	15.9

** These samples were acidulated at "low" acid conditions prior to blending. For the purpose of this table no net mass gain or loss has been assumed.*

13.7.1 Acidulation

The acidulation circuit was piloted ahead of the POX and the downstream hydrometallurgical circuits. The circuit treated each of the samples except for MC6 which was prepared after acidulation. Each acidulation sample was treated at "high" and "low" acidulation conditions to evaluate their response to complete and partial acidulation. The nominal operating conditions for the acidulation circuit were:

- Feed P80 grind size of 105 µm
- Feed slurry density of 45% w/w solids
- Operating temperature of 65°C
- Retention time of two hours.

Mildly acidic synthetic solution was added to the first two tanks in series to represent recycled decant thickener overflow together with fresh acid to provide the target acid to ore ratio. The composition of the synthetic solution and the acid to ore ratio for each feed blend was based on Metsim mass balances, however acid addition was adjusted during each run based on analytical data to achieve the target carbonate decomposition (acidulation extent). The results of the profiles taken of the acidulation circuit during the Campaign 5 piloting are provided in Table 13-18.

Table 13-18 Campaign 5 Acidulation Piloting Results

Test Feed	Units	1	2	3	4	5	6	7
		MTS	MTS	MTS	MTS	Main D	Main D	Main D
Conditions	-	High	Low	Low	Low	High	Low	Low
Acid to Ore ratio	kg/t	50	42	13	15	149	40	31
Terminal pH	-	3.38	2.17	4.94	5.07	0.97	3.28	3.34
Acidulation Extent	%	78.9	84.4	55.4	39.5	97.1	85.6	94.2
Test Feed	Units	8	9	10	11	12	13	-
		MC5	MC5	MC5	MC5	Mn D	Mn D	-
Conditions	-	High	High	Low	Low	High	Low	-
Acid to Ore ratio	kg/t	74	15	20	17	106	3	-
Terminal pH	-	1.17	1.57	3.45	3.84	1.15	5.95	-
Acidulation Extent	%	95	58	47.1	87.4	34	95	-

Generally, the “low” acidulation runs resulted in a terminal higher slurry pH and tended to overshoot the carbonate decomposition target. Conversely, the “high” acidulation runs required more acid than predicted by the Metsim models using the assumed FS chemistry.

To further understand the acidulation chemistry, three batch acidulation tests were performed after the Campaign 5 piloting. Various acid to ore ratios were tested including those without fresh acid addition. The results from the three batch tests are summarized in Table 13-19.

Table 13-19 Campaign 5 Batch Acidulation Test Results

Test Feed	Units	1	2	3	4	5
		MC5	MC5	Main D	Main D	Main D
Acid to Ore Ratio	kg/t	0	20	0	15	25
pH	-	6.13	5.06	6.19	4.54	3.9
Acidulation Extent	%	31	52	35	60	73

The chemistry inferred from the results of the batch acidulation tests suggested that the precipitation of dissolved metal ions, such as iron, arsenic, aluminum, copper and zinc in the synthetic decant thickener overflow contributed to the acid balance and carbonate decomposition. This was more noticeable at lower acid addition ratios with higher terminal slurry pH, explaining why the carbonate acidulation targets were consistently exceeded during the low acidulation runs. The assumed acidulation chemistry was subsequently updated to incorporate partial precipitation of iron, aluminum and copper.

13.7.2 POX

The pilot testing of the POX circuit was conducted as a series of nominally 12 hour periods. Test conditions included 40, 50 and 60 minute retention time at 220°C, with oxygen overpressures ranging between 250 and 400 kPa. The operating conditions and results are summarized in Table 13-20. Testing was performed in a 24L autoclave with four agitated compartments.

Table 13-20 Campaign 5 POX Circuit Operating Conditions and Results

Sample	Units	MTS				MC6			
Profile Number		**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**
Feed Solids Content	% w/w	37.5	36.6	37.6	40.6	37.2	35.7	37.4	37
Residence Time	min	42	61	61	42	41	45	62	54
Oxygen Overpressure [1]	kPa	706	603	586	586	706	653	696	696
Feed Solids									
Carbonate	% w/w	3.17	3.22	3.37	3.37	4.14	3.98	5.7	3.76
Sulphide Sulfur	% w/w	3.61	4.66	4.49	4.65	5.26	4.85	5.09	5.07 [2]
Sulphide Sulfur Oxidation [3]	%	71.6	94.4	86.4	74.9	81	84.9	92	91.5
Discharge Solution (Before Flashing)									
Free Acid	g/L	12.3	18.1	16.5	12.7	16.5	19.4	20.9	19.1
Total Iron	mg/L	1310	676	847	1400	1190	1280	1110	976
Ferrous	mg/L	1290	279	447	1100	772	637	402	335
Discharge Solids Total Sulfur	% w/w	1.96	2.27	1.9	1.84	2.05	2.03	2.44	2.3
POX Discharge Th. UF Solids [4]	% w/w	45.6	46.2	42.5	*41.5*	47.7	*45.9*	42.4	*37.5*

Sample	Units	MC5			Mn D			Main D	
Profile Number		**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**
Feed Solids Content	% w/w	38.3	38.4	37.4	31.4	33.5	34	39.7	40.7
Residence Time	min	51	62	43	42	61	54	60	41
Oxygen Overpressure	kPa	652	733	731	451	451	541	760	762
Feed Solids									
Carbonate	% w/w	3.66	5.15	3.34	5.75	7.5	5.5	1.76	1.5
Sulphide Sulfur	% w/w	5.33	6.11	5.36	6.69	6.96	6.42	3.87	3.96
Sulphide Sulfur Oxidation	%	97.6	97.9	95.5	96.4	96.3	96.3	98.7	95.2
Discharge Solution (Before Flashing)									
Free Acid	g/L	26.7	26.1	26.4	22.6	28	29.7	21.8	25
Total Iron	mg/L	766	680	1150	1010	869	1500	540	1070
Ferrous	mg/L	85	74	145	117	163	64	44	234
Discharge Solids Total Sulfur	% w/w	2.72	3.21	3.03	3.01	2.55	2.68	2.74	2.41
POX Discharge Th. UF Solids	% w/w	44.4	41.8	47.1	45.3	47.6	42	42.5	48

1) *Calculated from "O_2 and non-condensable overpressure, psi" and "Exhaust O_2 concentration (dry gas), mol% O_2".*

2) *Sulfide grade for period 8 reported as 0.02% by SGS. Value shown is the average of the periods 7 and 10.*

3) *Sulfide sulfur oxidation calculated by Amec Foster Wheeler using silicon as a tie component.*

4) *Values in italics taken from the Daily Met Report*

Sample	Units	MC5			Main D	MTS	Mn D
Profile Number		**18**	**20**	**21**	**22**	**23**	**24**
Feed Solids Content	% w/w	38	35	34	39.5	36	26.2
Residence Time	min	47	50	61	65	55	50
Oxygen Overpressure	kPa	729	817	821	825	794	807
Feed Solids							
Carbonate	% w/w	0.23	0.21	0.31	0.34	0.59	0.53
Sulfide Sulfur	% w/w	6.42	6.02	6.08	3.81	4.24	8.31
Sulfide Sulfur Oxidation	%	99.2	99.3	99.5	99.2	98.5	99.1
Discharge Solution (Before Flashing)							
Free Acid	g/L	27.4	27.8	28.7	28.9	25.8	25.7
Total Iron	mg/L	1750	1600	1510	907	777	702
Ferrous	mg/L	97	54	42	56	67	67
Discharge Solids Total Sulfur	% w/w	3.62	3.84	3.83	2.75	2.85	4.22
POX Discharge Th. UF Solids	% w/w	*37.2*	37.2	40.4	33.3	43.8	40.8

Low sulfide sulfur oxidation levels were achieved during the first six profile periods. This was traced to operation of the autoclave agitators at tip speeds considerably lower than would be used in a commercial autoclave, coupled with an oxygen addition profile which was weighted too heavily towards the first compartment. These two conditions were rectified, and from profile 7 onwards high sulfide sulfur oxidation extents were consistently achieved, although the oxygen overpressure was mostly higher than the target of 350 kPa due to operational issues. Consequently, the Campaign 4 sulfide oxidation kinetics were retained as the basis of the design.

A reasonably strong correlation between discharge free acid concentration and discharge solid total sulfur grade was noted, indicating increased amounts of jarosite formation at higher discharge free acid concentrations. Higher autoclave discharge free acid concentrations were also observed to have a dramatically negative impact on thickener underflow densities in downstream circuits. This relationship is discussed further in the solid-liquid separation vendor testing Section 13.7.7.

13.7.3 Iron/Arsenic Precipitation

The iron/arsenic pilot circuit consisted of two aerated agitated tanks in series heated to 65°C, providing a nominal 2 hr retention time. Limestone slurry was dosed to control the pulp to pH 2.5. Arsenic precipitation extents of greater than 90% were consistently achieved, even for the later profiles where the discharge pH decreased to less than 2.3. The higher residual concentrations of arsenic in the later profiles were likely related to higher feed arsenic concentrations. Ferric sulfate addition was not necessary to provide the target iron to arsenic ratio of greater than 4:1. Figure 13-7 illustrates the performance of the pilot iron/arsenic precipitation circuit.

Figure 13-7 Campaign 5 Iron/Arsenic Circuit Performance

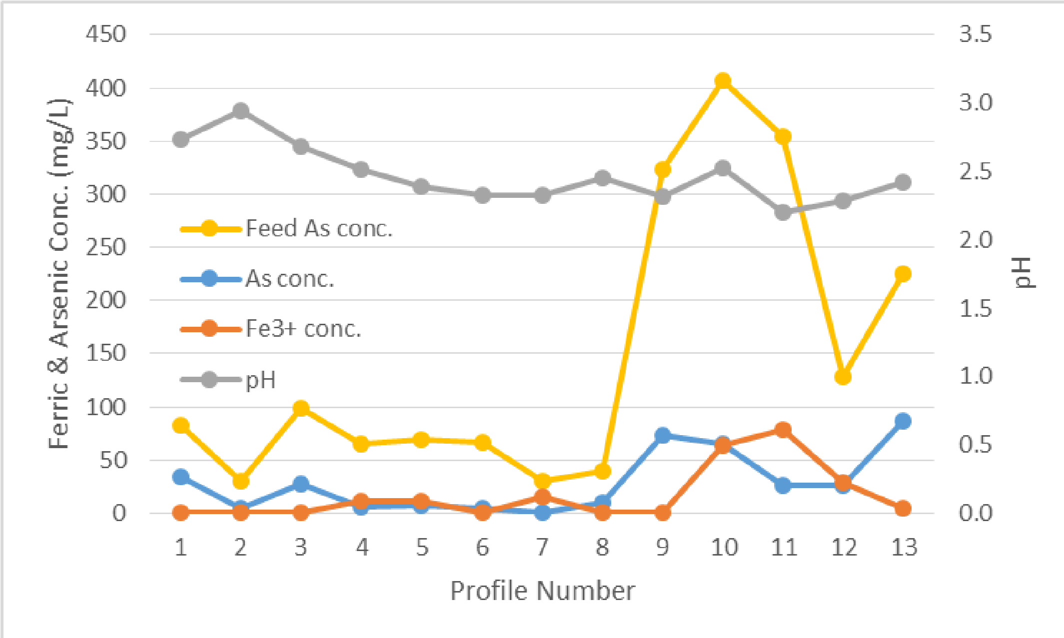


Figure prepared by Amec Foster Wheeler, 2016.

13.7.4 Copper Precipitate

Copper was recovered from CCD overflow in the copper precipitation circuit using sodium hydrosulfide (NaHS) to precipitate copper sulfide. The circuit comprised of two agitated tanks in series heated to 60°C, incorporating a recycle of copper sulfide precipitate to provide nucleation sites to improve crystal growth. Coagulant (SNF FL-4440) was added prior to the addition of flocculant (BASF Rheomax 1040). Barren solution was dosed with ferric sulfate to remove residual hydrogen sulfide in solution. The performance of the copper precipitation circuit is summarized in Table 13-21.

Table 13-21 Campaign 5 Copper Precipitation Circuit Operating Conditions and Results

Profile Number	Units	1	2	3	4	5	6	7
NaHS:Cu Ratio	g/g	-	4461	3242	0.8	1.58	1.47	1.24
Tank 2 pH	-	10	8.9	4.8	2.8	2.7	3.1	2.5
Seed Recycle	g/L	-	5.1	8.7	9.4	4.1	2.8	7.4
Feed Cu Conc.	mg/L	18.5	5	150	227	121	196	96.1
Tank 2 Cu Conc.	mg/L	0.67	BDL	BDL	30	1.42	1.04	23.3
Cu Precipitation	%	96.4	100	100	85.4	98.8	99.5	75.6
Filter Cake Solids	% w/w	65.1	31.6	23.4	33.5	25.7	46	54.9

Stable operation of the circuit was achieved after profile 3 was taken. Copper precipitation extents of approximately 99% were obtained during profiles 5 and 6 with NaHS to copper ratios of 1.58 and 1.47 respectively (equivalent to 179 and 167% of the stoichiometric requirement for copper in the feed). The solids content of the filter cake appeared to still be improving, reaching 54.9% solids by the end of the run.

After the Campaign 5 piloting testwork, the copper precipitation circuit was excluded from the full scale flowsheet.

13.7.5 Cyanide Leaching and Gold Adsorption

Seven bulk batch tests aligning with periods when certain samples were treated in the autoclave circuit were used for cyanide leaching and gold adsorption. Each batch was heated to 43°C in an agitated tank, neutralized with hydrated lime slurry to pH 10 and leached with cyanide for a nominal duration of 24 hr. After four hours of leaching, activated carbon was added at a concentration of 20 g/L. The results of bulk batch cyanide leaching and gold adsorption tests are provided in Table 13-22. The lower than design gold extractions achieved on the MTS and MC6 low-acidulation samples (profiles 1 and 2, at 91.5% and 91.8%, respectively) are believed to have been caused by the low sulfide sulfur oxidation levels achieved in the autoclave while producing these samples (Pressure Oxidation profiles 1 through 6). The low gold extraction of 84.9% reported for the Main Diorite low acidulation sample is believed to be an outlier caused by an assaying error on the final tail sample, as samples taken 4 hr and 16 hr into the leach yielded 95.3% and 95.7% extraction, respectively.

Table 13-22 Campaign 5 Batch Cyanide Leaching and Adsorption Operating Conditions and Results

Sample		MTS	MC6	MC5	Mn D	Main D	MC5	Mn D/ Main D/ MTS
Profile Number	**Units**	**1**	**2**	**3**	**4**	**5**	**6**	**7**
Acidulation Extent		Low	Low	Low	Low	Low	High	High
Feed Solids	% w/w	30	26	26.5	23	22.5	23.5	26.5
$Ca(OH)_2$ addition	kg/t	15.7	18	15.4	14.7	15.2	24.2	16.7
NaCN addition	kg/t	2.5	2.5	2.2	2	2.4	2.9	2.2
Solid Au Grade								
Feed	ppm	6.65	5.46	5.07	7.6	5.5	5.57	5.68
Tails	ppm	0.57	0.45	0.19	0.2	0.83*	0.18	0.25
Loaded C	ppm	97.5	98	94.8	85.4	76.9	82	90.5
Gold Extraction	%	91.5	91.8	96.2	97.4	84.9**	96.8	95.7

** This is believed to be either an assay error, sample error or due to contamination*

*** Possible error with solids samples taken at 4 hr and 16 hr being 93.5% and 95.7% extraction*

In addition to the bulk batch cyanide leaching and gold adsorption tests, smaller scale batch tests were performed on 800 g samples of autoclave discharge solids from every test period. Selected leach kinetics results from these tests were modelled to predict gold extraction and the optimum leach retention time for the full scale circuit.

Carbon adsorption tests were performed for several ore types and composites at different carbon to slurry ratios and at three temperatures, including the expected operating temperature of 43°C. Fleming and Nicol constants were derived from the experimental data, as were the gold loading factors and the expected carbon loadings. These were then used to model the adsorption circuit to predict the expected soluble loss, derive the circuit carbon concentration and carbon movement which was in turn used to size the elution circuit.

13.7.6 Cyanide Destruction

Tailings from the cyanide leaching batch tests were treated in a continuous cyanide destruction circuit to remove weak acid dissociable (CN_{WAD}) cyanide. The circuit consisted of two agitated tanks in series with air spargers, sodium metabisulfite (SMBS) addition and pulp pH control via the dosage of hydrated lime slurry. The second tank was bypassed and the circuit operated with a single tank for profiles 5 through 8. Copper sulfate addition was found to be unnecessary. Table 13-23 summarizes the conditions and results from the continuous cyanide destruction pilot circuit.

Table 13-23 Campaign 5 Cyanide Destruction Circuit Operating Conditions and Results

Profile Number	Units	1	2	3	4	5	6	7	8
Retention Time [1]	mins	90	87	98	50	42	37	42	42
Discharge Solids	% w/w	25.6	30	31	25.8	25.9	21.5	25	24.9
Discharge pH	-	9.5	9.5	9.4	9.7	10.2	9.7	9.5	9.6
SMBS Addition	g/g CN_{WAD}	20.6	4.9	15.3	4.6	3	3.2	5.2	6.3
	mole SO_2/ mole CN_{WAD}	5.62	1.33	4.19	1.25	0.83	0.89	1.41	1.72
CN_{Total} Feed [2]	mg/L	118	66	170	150	211	210	293	85
CN_{Total} Discharge	mg/L	BDL	BDL	BDL	BDL	BDL	28	BDL	BDL
CN_{Free} Feed	mg/L	55.2	9.41	101	82.8	118	119	149	44.1
CN_{Free} Discharge	mg/L	BDL	0.08	BDL	0.19	0.3	5.1	BDL	0.33
CN_{WAD} Feed	mg/L	87.5	42	133	126	105	177	268	81
CN_{WAD} Discharge	mg/L	BDL	BDL	BDL	BDL	BDL	19	BDL	BDL
SCN Feed	mg/L	168	217	83	96	45	70	58	58
SCN Discharge	mg/L	168	247	77	93	47	69	57	55

1) Retention Time is per tank.
2) SGS defines CN_{WAD} to include HCN, CN_{Free} and cyanide that is weakly bound up in metal complexes. CN_{Total} represents all of the cyanide in solution including that which is tightly bound up in metal complexes, but excluding thiocyanate and cyanate.

These results confirm both the design cyanide destruction circuit configuration of a single tank with 60 min residence time and the design conditions of an SMBS addition rate of 1 mole SO_2 per mole CN_{WAD} to achieve a discharge CN_{WAD} concentration of <5 mg/L CN_{WAD}. However, the tests showed that the conditions were not effective for destroying thiocyanate (SCN).

13.7.7 Final Neutralization

The final neutralization circuit included two stages; the first was treatment of copper precipitation filtrate with ferric sulfate using an inline mixer to remove residual hydrogen sulfide, and the second combined treated copper precipitation filtrate with detoxified slurry followed by neutralization at 50°C to a pH of approximately 10, using hydrated lime slurry. The conditions and results of final neutralization pilot circuit are provided in Table 13-24.

Table 13-24 Campaign 5 Final Neutralization Circuit Operating Conditions and Results

Profile Number	Units	1	2	3	4	5	6	7
Ferric Addition	g Fe^{3+} / m^3 Cu Filtrate	403	161	226	136	14	2	262
Retention Time *	minutes	158	103	109	107	36	40	37
Discharge pH	-	10.1	11.2	9.7	10.4	10	9.7	13.1
Fe Precipitation	%	100	100	100	100	100	100	100
Mg Precipitation	%	91	99	87	98	96	92	100
Mn Precipitation	%	100	100	100	100	100	100	100
Profile Number	**Units**	**8**	**9**	**10**	**11**	**12**	**13**	**15**
Ferric Addition	g Fe^{3+} / m^3 Cu Filtrate	178	196	178	183	155	128	55
Retention Time	minutes	95	97	96	108	101	106	116
Discharge pH	-	11.1	11.4	10.9	10.4	10.9	10.8	10.5
Fe Precipitation	%	100	100	100	100	100	100	100
Mg Precipitation	%	99	100	99	98	99	99	98
Mn Precipitation	%	100	100	100	100	100	100	100

**Total from both tanks*

Residence time in the circuit was reduced to 36 and 40 minutes during the last three profiles and precipitation of the contained metals remained excellent. This supports the design residence time of 18 minutes per tank for the final neutralization circuit.

13.7.8 Solid-Liquid Separation Testing

By the time of the Campaign 5 solid liquid separation testwork review a decision had already been made to maintain thickener sizing apart from the grinding thickener as part of the review of the Campaign 4 testwork accepting any process implications in terms of performance. A summary of the design conditions and thickener performance is shown in Table 13-25. Note that the flux is based on the design throughput for each thickener.

Table 13-25 Updated Thickener Sizing

Thickener Duty	Flux, $t/m^2.h$	Diameter, m
Grinding	0.5	28
POX Feed	0.35	32
Decant	0.35	32
CCD	0.3	35
Tails	0.22	44

A solid liquid separation equipment vendor was engaged to perform solid-liquid separation testwork on various samples produced during the Campaign 5 piloting. A summary of the results is shown in Table 13-26.

Table 13-26 Updated Thickener Design Conditions and Vendor Data

	Duty		Grinding		POX Feed		Decant	
	Parameter	Units	DSN 1	DSN 2	DSN 1	DSN 2	DSN 1	DSN 2
Design	Solids	t/h	245	306	245	282	240	276
	Underflow	% w/w	50	45	48	45	45	40
Vendor	Underflow	% w/w	51.5	51	52	51	43	43

	Duty		CCD		Tails	
	Parameter	Units	DSN 1	DSN 2	DSN 1	DSN 2
Design	Solids	t/h	253	291	282	325
	Underflow	% w/w	43	40	28	28
Vendor	Underflow	% w/w	40	40	28	28

The testwork outcomes are as follows:

- The grinding thickener settling flux rate is consistent with the rate achieved during the Campaign 4 testwork program. The slight increase in grinding thickener diameter (from 27 m to 28 m) is to allow for a short term increase in through-put through the grinding circuit when treating soft ore.
- POX feed thickener tests were conducted on four samples at both “high” and “low” acidulation extents (eight tests overall). The underflow densities that were achieved ranged between 36% and 52% w/w solids based on settling fluxes of 0.10 t/m².h to 0.35 t/m².h, with the samples from the “low” acidulation tests not meeting the minimum underflow density of 45% w/w solids.
- Based on this finding, the acidulation flowsheet was changed from partial acidulation of the whole POX feed, to full acidulation of a portion of the POX feed, thickening of acidulation discharge and recombining with the remainder of the (unacidulated) POX feed.
- The expected underflow density for the decant thickener exceeded the nominal mass balance requirement but fell short of the design requirement.
- The expected underflow density for the CCD thickeners matched the nominal mass balance requirement but fell short of the design requirement, which had already been reduced on the basis of the Campaign 4 testing.
- For the decant and CCD thickeners Anagold have accepted that underperformance will be addressed by measures such as increasing wash water and reagent addition.
- The expected underflow density for the tails thickener was significantly lower than that design which was subsequently revised from 34% w/w to 28% w/w. The tails pumping and water circuits were updated accordingly.

13.8 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 intentions.

13.8.1 Gold Recovery

The gold recovery results of the acceptable tests are plotted in Figure 13-8, Figure 13-9 and Figure 13-10, together with an appropriate recovery model curve in each instance.

Figure 13-8 Metasediment Gold Recovery Results and Model

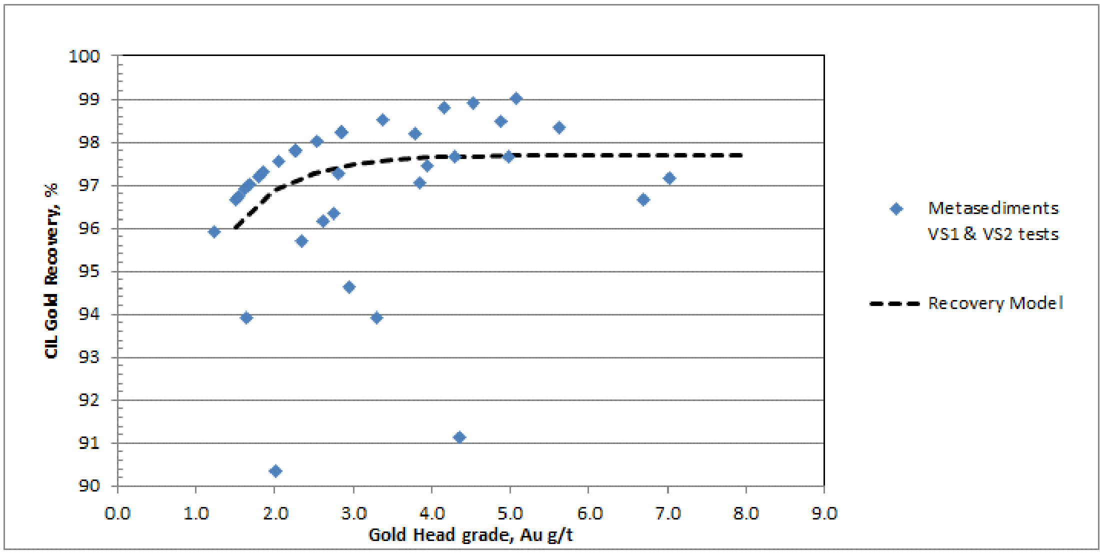


Figure prepared by Amec Foster Wheeler, 2016.

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 13-9 Main Diorite Gold Recovery and Model

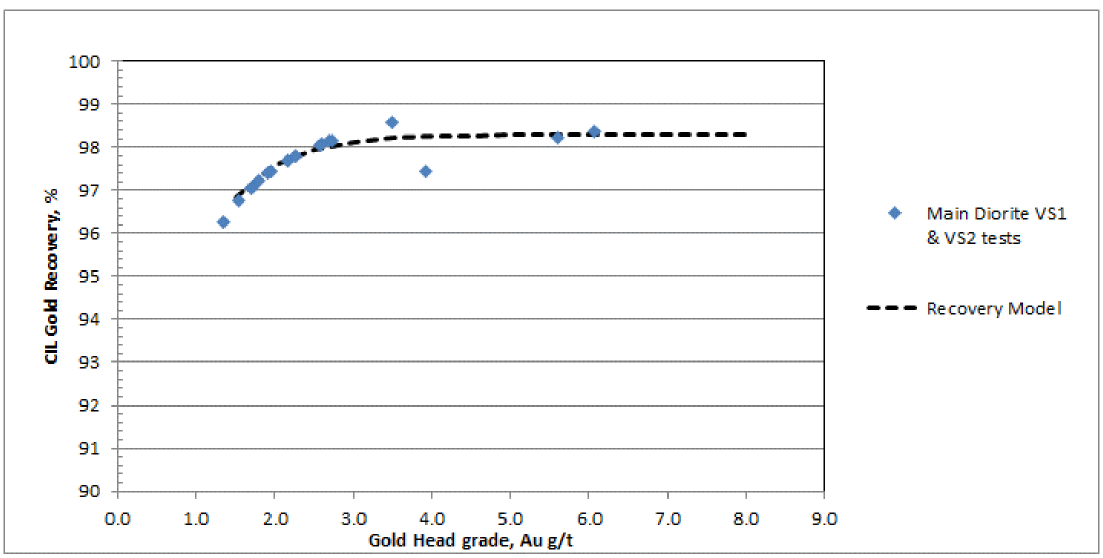


Figure prepared by Amec Foster Wheeler, 2016.

Note that the above two figures have 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 gold grade was below the limit of detection and a set tails grade, equal to half the limit of detection was assigned for calculation purposes.

Figure 13-10 Manganese Diorite Gold Recovery and Model

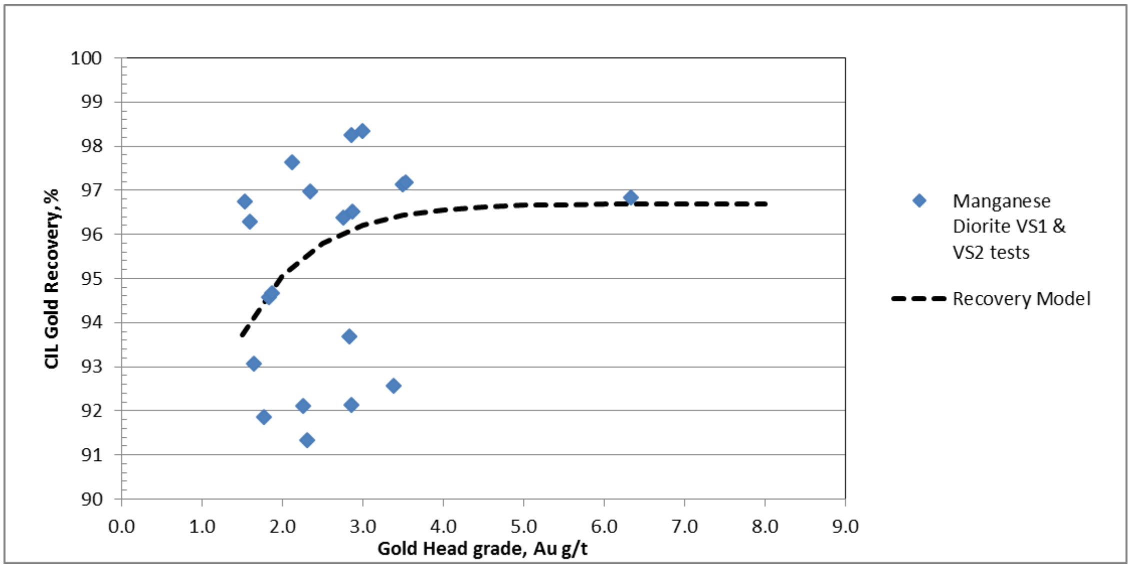


Figure prepared by Amec Foster Wheeler, 2016.

The recovery model was prepared by Amec Foster Wheeler is represented by the equation:

Au Recovery = A * (1 – exp (-b * (Gold Head Grade in g/t – c)))

This equation has not been fitted to the data by using an error minimization technique. Instead, the parameters have been manually adjusted to provide a reasonable representation of the trend in the available data. Note that parameter A is the only one of the three that has a direct process meaning. “A” represents the maximum recovery the equation can generate and the function tends to this value as an asymptote.

The parameters used to generate the curves in Figure 13-8, Figure 13-9 and Figure 13-10 are shown in Table 13-27, together with the set of weighted-average LOM model parameters (based on the proportion of ore types).

Table 13-27 Au Recovery Model Parameters

Material Type	A	B	C
Metasediment	97.7	1.4	-1.4
Main Diorite	98.3	1.4	-1.5
Mn Diorite	96.7	1.2	-1.4
LOM	**97.94**	**1.4**	**-1.4**

The LOM gold recovery model, together with all the accepted data, is shown in Figure 13-11. Note that the majority of the low recovery points are for manganese diorite ore. They do not have a significant influence on the LOM because manganese diorite ore is only a small proportion of the feed.

Figure 13-11 All Ores LOM Gold Recovery and Model

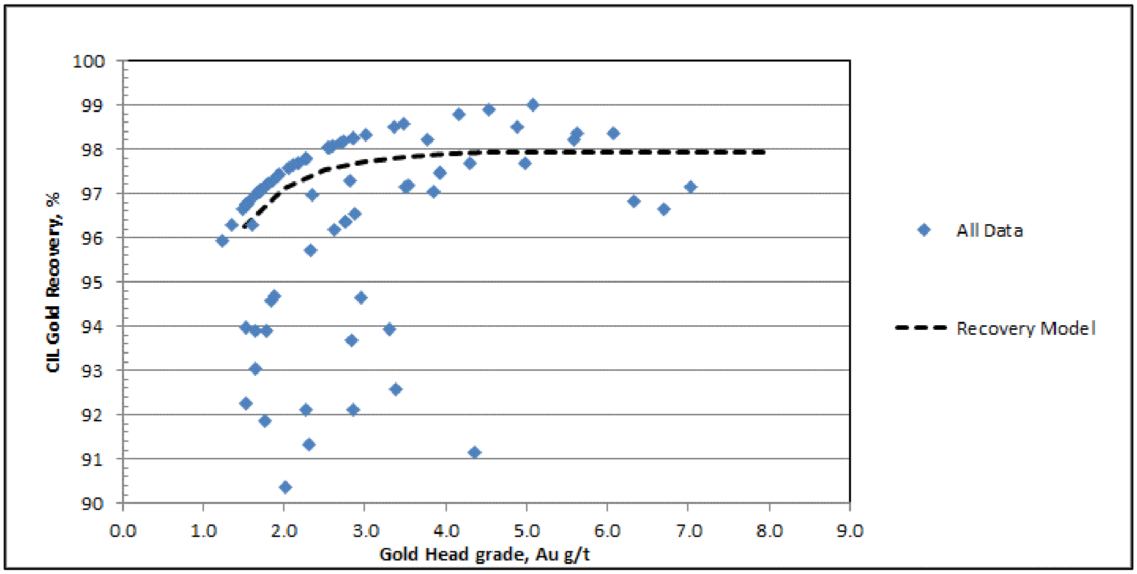


Figure prepared by Amec Foster Wheeler, 2016.

The most concerning aspect of the recovery model is the existence of low recovery results for metasediments (Figure 13-8) that are not explainable by the available data and have little influence on the model. Metasediment is the dominant ore type and an understanding of the reasons for recovery loss, especially if it is found to be predictable or avoidable, would be valuable to the Project.

13.8.2 Silver Recovery

The silver recovery pattern is much less clear than gold because silver is not released by the oxidation process. The plot of silver recovery vs silver head grade for all tests included in the gold analysis is shown in Figure 13-12.

Figure 13-12 Silver Recovery vs Head Grade, All Tests

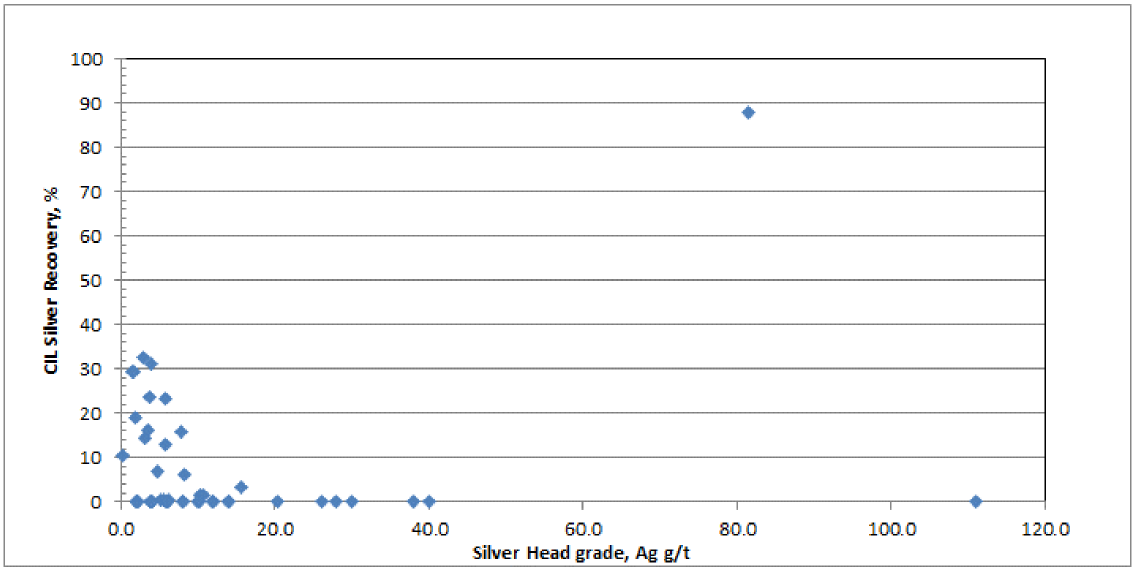


Figure prepared by Amec Foster Wheeler, 2016.

In many instances there was negligible silver recovery recorded and most of these were from the VS1 sample set. Consequently only the VS2 data was used for silver and this set is shown in Figure 13-13.

Figure 13-13 Silver Recovery vs Head Grade, All Tests

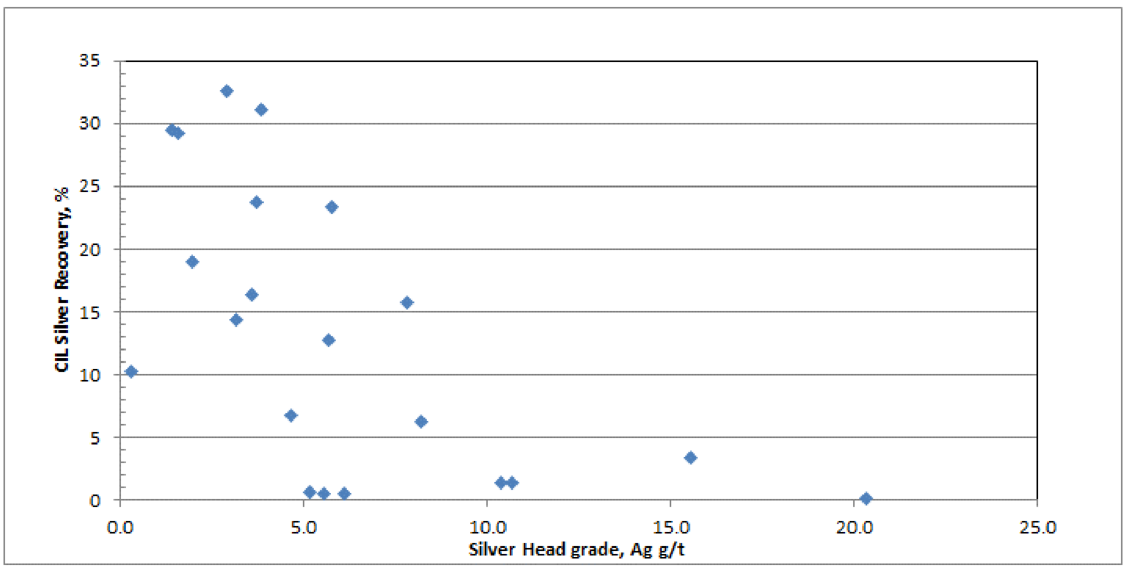


Figure prepared by Amec Foster Wheeler, 2016.

The oretype weighted average silver recovery calculates to 14.6%.

13.8.3 LOM Recovery Expectations

Sulfide recoveries using the equation from Section 13.8.1 (gold) and fixed base recovery or 14.6% from Section 13.8.2 (silver).

The commissioning and operational loss allowances in Table 13-28 have been made on top of the base recoveries.

Table 13-28 Commissioning and Operational Loss Allowances

Recovery Corrections	Au	Ag
Commissioning 2018	-2%	-2%
Ramp-up 2019	-1%	-1%
Solution Losses (LOM)	-1%	-1%
Single Autoclave operation (LOM)	-0.3%	0%

This leads to the total corrections for recovery as indicated in Table 13-29.

Table 13-29 Total Recovery Corrections

Recovery Corrections	Au	Ag
2018	-3.3%	-3%
2019	-2.3%	-2%
2020 to LOM	-1.3%	-1%

This gives the recovery predictions based on the mine plan that are illustrated in Figure 13-14.

Figure 13-14 LOM Recovery Predictions

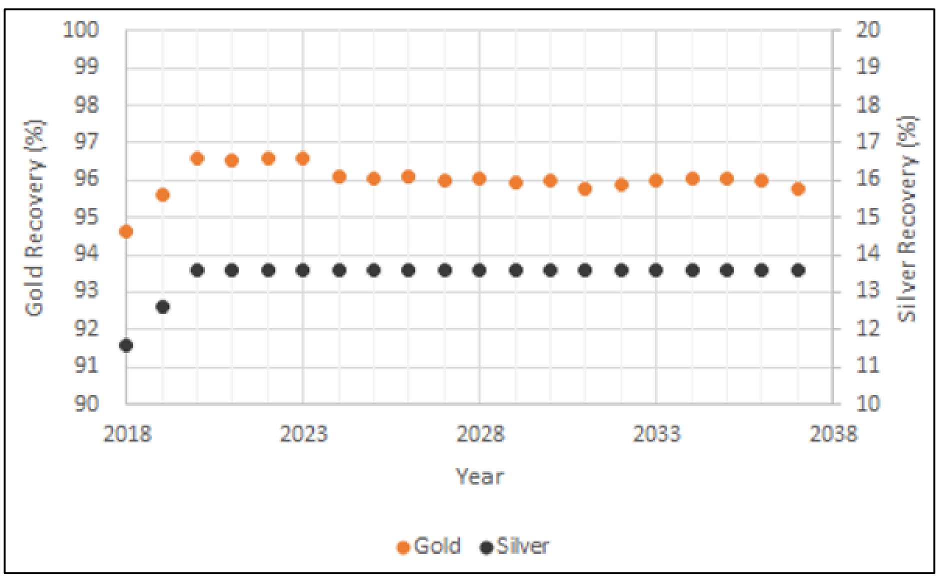


Figure prepared by Amec Foster Wheeler, 2016.

13.9 Variability

The POX metallurgical variability test program (batch testing) was conducted on samples representing each of the main ore types (metasediment, main diorite and manganese diorite) and representing the full grade spectrum, in terms of gold, silver, copper, sulfide sulfur, and carbonate for each type. The flowsheet development testing, at both batch and pilot scale, was conducted on orebody composites representing early plant operation, as defined at the time of sample selection. The sulfur levels, which are critical for POX process development, were similar to mine plan sulfide sulfur levels. Gold levels were closer to the long-term gold grades rather than the high grades expected in the early operating years.

The orebody has been well represented in the test program. However, the understanding of plant feed in the early operational years is complex because millions of tonnes of sulfide ore have been, and continue to be, stockpiled and will be used for plant feed in the early operation. Although the complexity introduces some risk, it is mitigated by the flexibility of being able to select feed from numerous locations around the stockpile and by blending with ore directly mined from the pit.

13.10 Deleterious Elements

No deleterious elements are expected in the final gold-silver doré product from site.

13.11 Mineral Processing and Metallurgical Discussion

A large amount of testwork has been performed on Çöpler sulfide ore across a number of campaigns and as a result the processes used have been shown to be reasonably robust. However, late circuit changes (such as split acidulation) have reintroduced some uncertainty, especially in the area of POX thickener performance. It is recommended that testwork continue with the aim of reducing risk in areas of the flowsheet that have recently changed.

A large amount of sulfide ore has already been mined and stockpiled and this will form a significant proportion of the plant feed in the first few years of operation. Characterization of the stockpiled ore from a chemical perspective is needed to allow effective ore blending to occur. Ore blending is needed primarily to achieve autoclave feed that is consistently within the target ranges around which the autoclaves have been designed.

The ore is typically soft and will be easy to grind in the comminution circuit. However, the hardest ores, if treated as a majority of the feed blend, are expected to grind at less than the target autoclave feed rate of 245 t/h and are best blended with soft feed. There is expected to be good opportunity for achieving throughput rates well above average and this will ensure the plant will be unconstrained by comminution throughout its life.

The reasons behind low gold recovery results when all design conditions have been met, especially for metasediment ore, should be investigated further.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The Çöpler open pit resource model was constructed by Loren Ligocki, Alacer Resource Geologist and Gordon Seibel, R.M. SME and Principal Geologist with Amec Foster Wheeler, Inc. The Mineral Resource estimate was reviewed by Dr. Harry Parker, Consulting Mining Geologist and Geostatistician for Amec Foster Wheeler. Dr. Harry Parker and Gordon Seibel are the QPs for the resource model and Mineral Resource estimate.

The Amec Foster Wheeler QPs consider that the mineral resource models and Mineral Resource estimates derived from those models are consistent with the 2014 Canadian Institute of Mining Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (2014 CIM Definition Standards) and were performed in accordance with the relevant CIM Best Practice Guidelines (2003).

Traditional Mineral Resource modeling methods are commonly undertaken by manually constructing wireframes around the economic mineralization. Such methods are labor intensive, time consuming, and difficult to update with additional drilling or changing cut-off grades. Due to these concerns, a hybrid gold model was constructed to define the geometry of the gold mineralization and to calibrate the resource estimate to production data. Steps for the gold modeling process included:

- Wireframe gold mineralization using commercially-available Leapfrog software by interpolating assay values in the drill holes. This step used structural trends to guide the shape of the wireframes along known geologic features within the deposit. Mineralized trends commonly follow lithologic contacts, such as the diorite/marble contact in the Manganese pit, and structural features identified by surface mapping. A total of 15 trends across the deposit were used to produce a 3D solid. Trends were developed using the geologic model, pit mapping and blast hole data.
- Potentially economic gold mineralization was then estimated using probability assigned constrained kriging (PACK) and then trimmed using the 3D solid. PACK first uses a probabilistic model or envelope to define the limits of the potentially economic mineralization. The blocks and drill hole composites within these indicator envelopes were then used for mineral resource model estimations. The PACK was designed to prevent economic grades inside the indicator envelope from being smeared into the waste and restricts low grade material outside the estimated indicator envelopes from diluting the mineralized material inside the envelope. PACK has the advantage of being easily updated with changing economic parameters, addition of new data, new geological interpretations, and can be calibrated to include estimated block grades that in aggregate match historic production. Details on PACK modeling parameters are provided in Section 14.3.
- For the Çöpler gold resource estimations, the PACK parameters used to construct the indicator envelopes were calibrated so the estimated tonnes and grades approximated the historic production data. The calibrations were performed by area, material type and time period so the calibrations could be studied and evaluated in detail.

Copper and silver were estimated into the gold block model using ordinary kriging, while arsenic, manganese, iron and zinc were estimated using inverse distance squared (ID2) interpolation.

14.2 Key Assumptions/Basis of Mineral Resource Estimate

For the Çöpler model, lithology shapes and domain boundaries (Figure 14-1) were first constructed using Leapfrog, version. 2.2.1, and then imported into Vulcan, version 9.1.3. software. Metal, density and RQD estimations, as well as oxidation definitions and block classification were added in Vulcan. The combined Vulcan model was then supplied to the Alacer mining department for mine planning.

Figure 14-1 Geographic Domains with Drill Hole Collar Locations

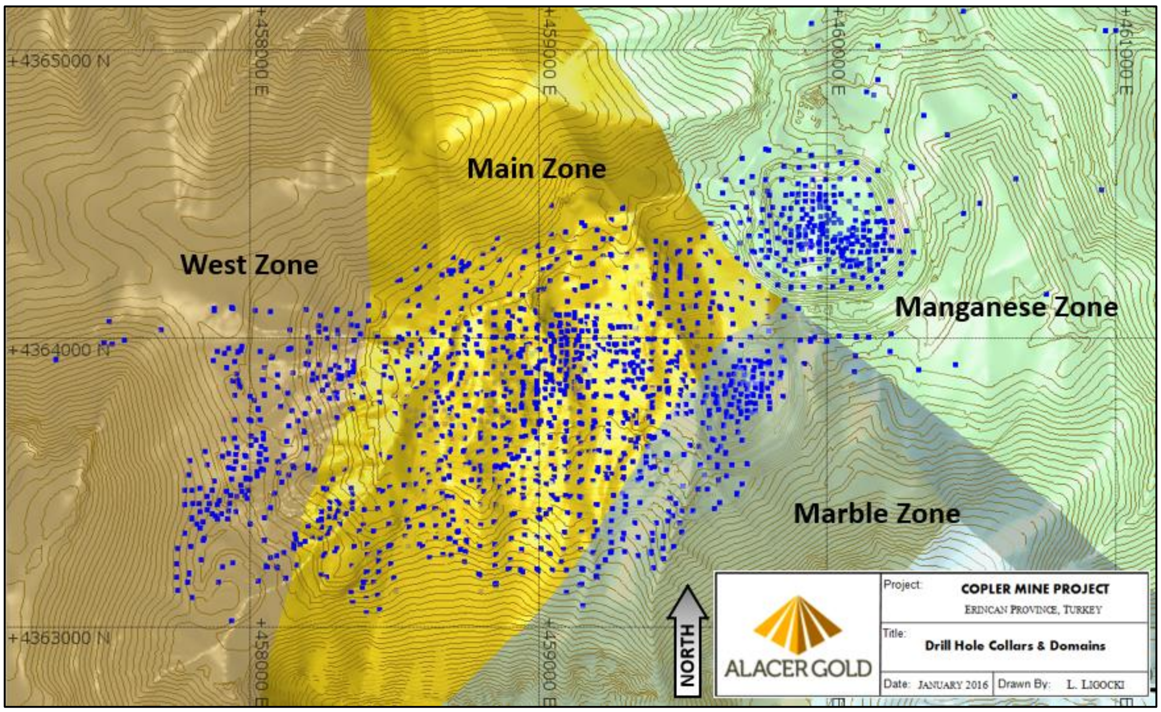


Figure courtesy of Alacer, 2016

The estimation methods were designed to address the variable nature of the epithermal structural and disseminated styles of Au mineralization while honoring the bi-modal distribution of the sulfur mineralization and oxide-sulfide boundary used to define the material types that is critical for mine planning. The modeling methods were designed so the

a) Mineral Resources could be updated with additional drilling, and

b) Changes in cut-off grades could be recalibrated using up-to-date production data.

Although silver and copper were estimated and used in the mining studies, the model design focused on the gold mineralization as it is the dominant economic contributor.

Since no obvious correlations were observed between gold and total sulfur, they were domained and estimated separately. Since gold showed little correlation with lithology it was domained by mining areas (Manganese, Main, Marble, West) which reflects the different trends of the mineralization that commonly follow structures and lithological contacts.

Total sulfur percentage is the main criterion used to delineate between “oxide” and “sulfide” material types. Oxide material (S percentages <2%) is processed using a heap leach method using a gold cut-off grade of approximately 0.3 g/t, and the sulfide material (S percentage ≥ 2%) is stockpiled for the proposed POX plant using an approximated gold cut-off grade of 1.5 g/t.

Total sulfur shows a good correlation with lithology and exhibits a bimodal distribution with a distinct inflection point at 2% S. The 2% S break point also agrees well with a 1.0% pyrite visual percentage break point logged in the drill holes.

As a result, sulfur was modeled using oxide and sulfide sub-domains within each lithology, and gold PACK models were constructed separately for oxide and sulfide within each lithology. The lower-grade oxide model used a 0.3 g/t Au indicator threshold to reflect the lower gold cut-off grade, and the second higher-grade sulfide model used a 1.5 g/t Au threshold to reflect the higher gold cut-off grade used for sulfide material. It is worth noting that while a 1.5 g/t Au threshold is used for the higher-grade indicator, a 1.0 g/t cutoff is used for resource calculation. The 1.0 g/t reflects the likelihood of reducing the cutoff at a higher gold price.

The gold models were then reconciled to historic production data and the modeling parameters were adjusted to best match the historic data. Mineral Resource categories were applied to each block based on drill hole density and data quality.

14.3 Base Indicator Model

In order to constrain the Vulcan model into a reasonable file size, blocks were only generated within the 3D solid. The shell follows the original topography approximately 30 m above and extends beyond drilling by 300 m. This allows blocks at the corners of the square model to be excluded, reducing model size by about 40% without impacting the generation of the resource cone. Block model parameters are provided in Table 14-1. The 10 x 10 x 5 m block size was selected, as the horizontal X and Y directions are approximately one half the average drill hole spacing and the 5 m height of the blocks matches the current mining bench height. The Mineral Resource Model has an implicit SMU of 5 x 10 m x 5 m.

Table 14-1 Block Model Parameters for the Çöpler Resource Model

Axis	Minimum (m)	Maximum (m)	Range (m)	Block Size (m)	Number of Blocks
X	457,100	461,100	4,000	10	400
Y	4,362,500	4,365,100	2,600	10	260
Z	400	1,750	1,350	5	270

Two gold PACK models were constructed, a lower-grade model for oxide material (< 2% S) and a higher-grade model for sulfide material (≥ 2% S). For the oxide or low-sulfur PACK indicator model, an indicator threshold of 0.3 g/t Au was selected which approximates the current gold cut-off grade for low-sulfur material. For the high sulfur PACK indicator model, a 1.5 g/t Au threshold was selected as it approximates the gold cut-off grade being used for the Mineral Reserves having high-sulfur content.

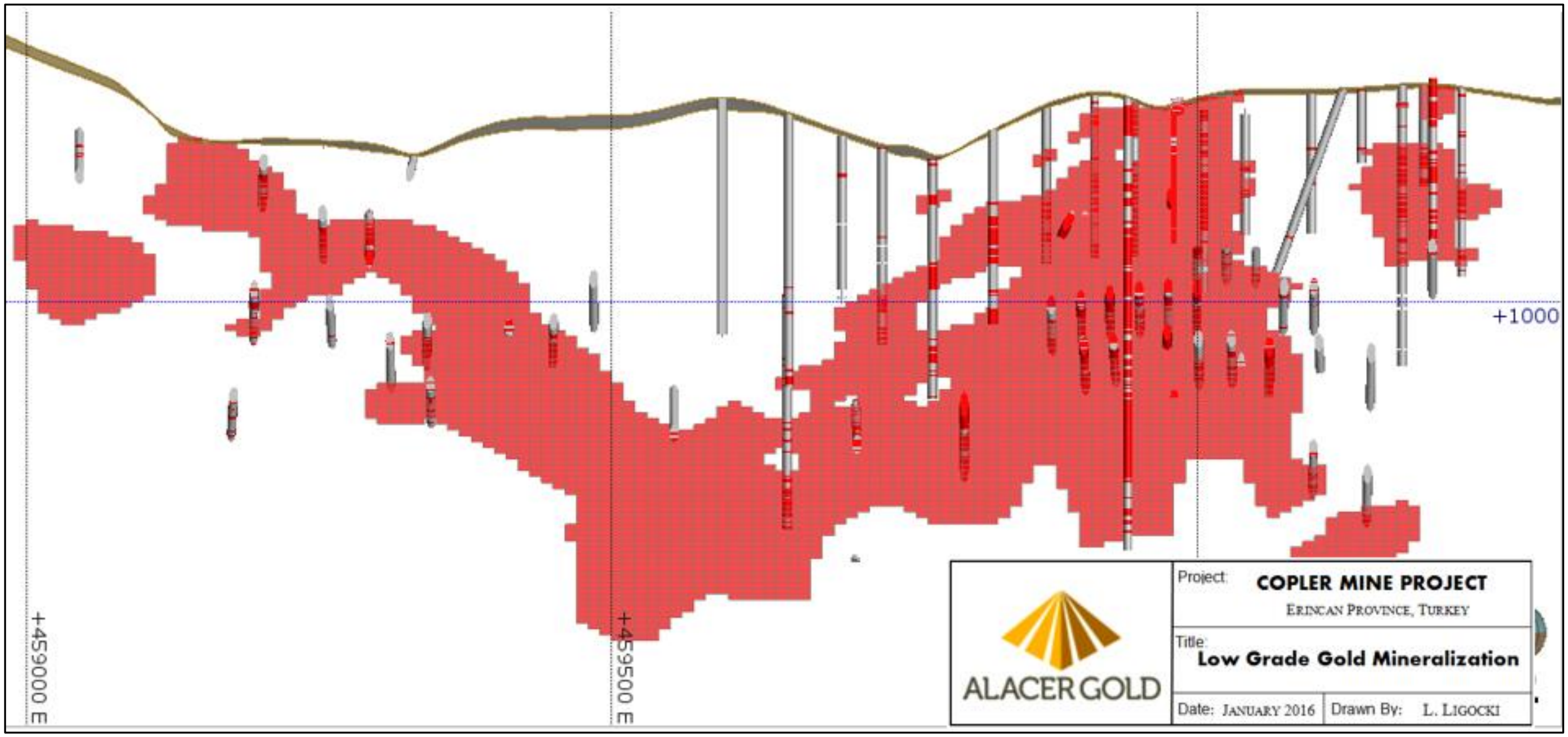
Au intervals were first composited to 10 m down-the-hole lengths and then assigned Au indicator values. The sulfur indicator used 5 m composites for the estimate. Both the gold and sulfur indicator values were estimated into the base model using an ID2 estimate with parameters shown in Table 14-2. In making an indicator model, composites below the threshold are assigned 0 and composites at or above the threshold are assigned a 1. The estimated indicators represent a weighted average of the composite indicators, and have values between 0 and 1. An example cross section is shown in Figure 14-2 of the lower-grade model blocks against drilling with Au grades above a 0.3 g/t cut-off colored red. Exploratory data analyses (EDA) and capping studies were performed on samples within this envelope. A PACK threshold has not been applied to the blocks in Figure 14-2.

Table 14-2 Gold and Sulfur Indicator Estimation Parameters

	Model Variable	Estimate Method	Pass	Samples		Max # Per Hole	Search Distance (m)		
				Min	Max		X	Y	Z
AU INDICATOR	ind1	ID2	1	3	20	2	40	40	30
	ind1	ID2	2	3	20	2	60	60	40
	ind1	ID2	3	2	20	2	150	150	75
	ind2	ID2	1	3	20	2	40	40	30
	ind2	ID2	2	3	20	2	60	60	40
	ind2	ID2	3	2	20	2	150	150	75

	Model Variable	Estimate Method	Pass	Samples		Max # Per Hole	Search Distance (m)		
				Min	Max		X	Y	Z
S INDICATOR	s_ind	ID2	1	3	12	2	40	40	40
	s_ind	ID2	2	3	12	2	60	60	60
	s_ind	ID2	3	2	12	2	160	160	160

Figure 14-2 Cross Section of Low-Grade Economic Mineralization, Section 4,364,400N, looking North



1. Figure courtesy of Alacer, 2016
2. Red blocks show the extents of the interpreted Leapfrog gold mineralization
3. Drill holes are colored by Au grade, red intervals display Au ≥0.3 g/t

14.4 Domains

The base indicator model was divided into the four domains for resource estimation that follow the four separate mining areas (Manganese, Main, Marble and West) using wireframes constructed in Leapfrog Geo. The boundary between the Manganese and the Main domains was selected to lay between the different diorite intrusive events. The boundaries for the Marble domain were selected along one of the arms of the diorite intrusion associated with a zone of higher-grade mineralization. The boundary direction follows the northeast-southwest trend of the mineralization. The extension of this boundary includes a larger, but dissimilar diorite intrusive that carries minor gold mineralization along its contact with the metasediments. Refer to Figure 14-1 for a plan view of the domain boundaries.

The top of the domain boundaries were trimmed to the original topography. The extents of the domain boundaries exclude exploration drill holes to the far north and east of the main resource area. Figure 14-3 shows the spatial relationships between the four domains used in the mineral resource models to the July 2015 mined pits.

Figure 14-3 Resource Model Domains and Block Extents

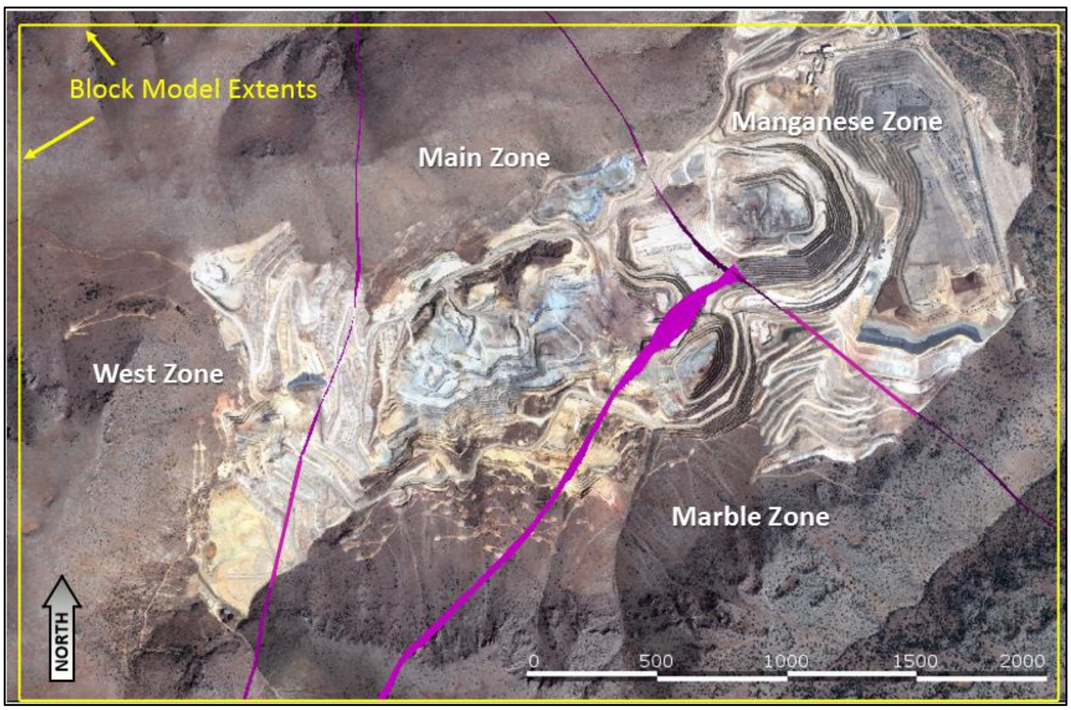


Figure courtesy of Alacer, 2016

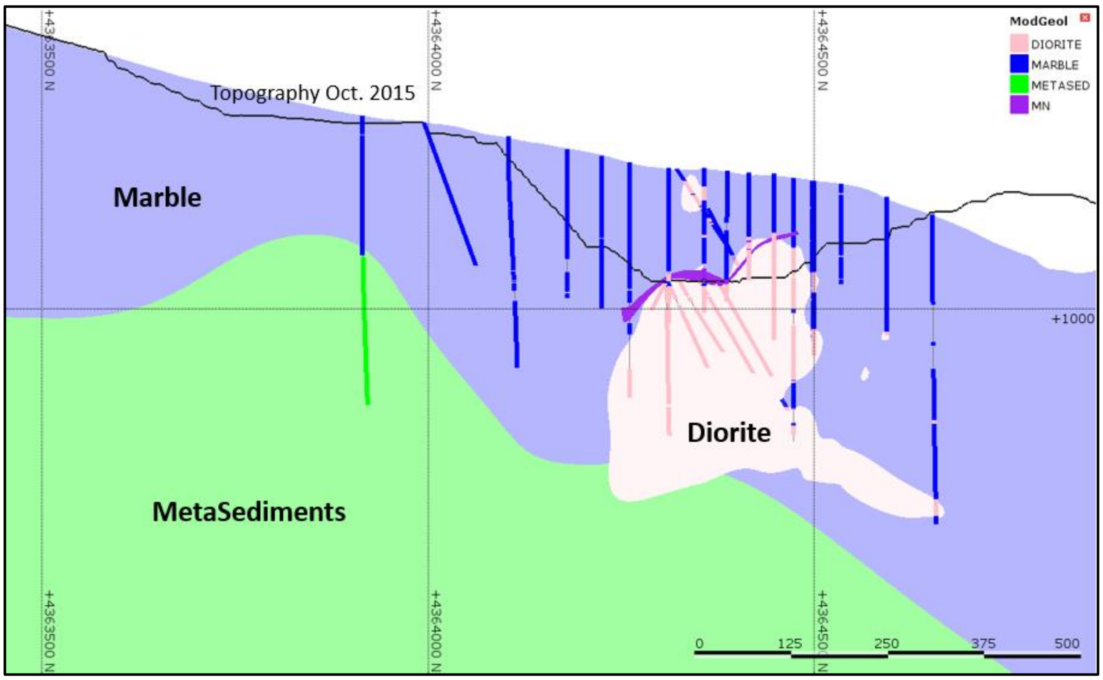
14.5 Geological Model

Geological wireframes were constructed for the four main geological units: marble, diorite, metasediments, and manganese-rich zones. The wireframes were generated using Leapfrog Geo software and geological data collected by site geologists. Drill data and surface mapping were interpolated into 3D solids that represent the major rock types. This process included generating contact surfaces used to define the division boundaries that represent the geological faults and lithologic contacts. This method allows for rapid regeneration of solids for all rock types. New or modified information can be added to the existing model without reworking digitized sections.

Surface mapping was used to provide indicative contact locations in areas of sparse drilling. In areas where the two data sets did not match, priority was given to the drill hole data. The model was adjusted in the Manganese open pit after referencing the blast hole information. Blast hole data were not used to generate the geologic model, but provided guidance when modeling exploration holes drilled through zones of wide-spaced drilling and in areas with missing drill data.

Construction of the geologic model was made within a defined boundary, sufficiently large enough to cover areas of interest for block modeling. Typical cross-sections illustrating the lithology wireframes are shown Figure 14-4 and Figure 14-5.

Figure 14-4 Lithology Model – Manganese Domain, Section 459,900E, Looking West



1. The Manganese domain contains a zone of manganese oxide, shown in purple
2. Figure courtesy of Alacer, 2016

Figure 14-5 Lithology Model – Marble Contact Zone, Section 459, 700E, Looking West

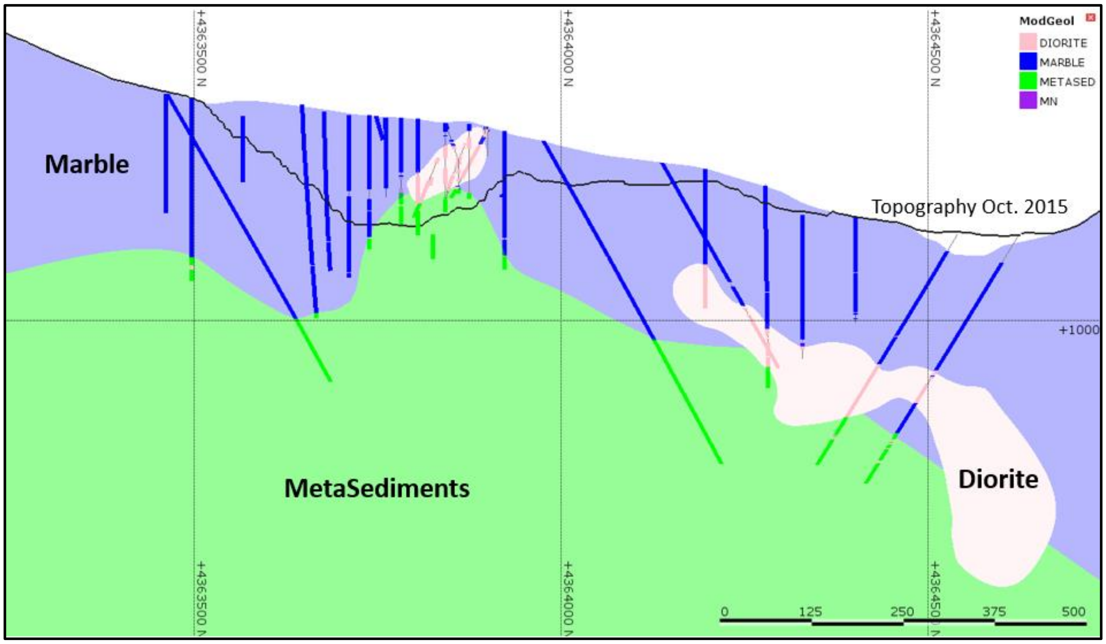


Figure courtesy of Alacer, 2016

14.6 Data Summary

The cut-off date for exporting the drill holes from the database to be used in the resource model was July 15th, 2015. The area contained 1,957 drill holes with a total of 297,798.2 m of drilling. Of this, a total of 1,880 drill holes have collar coordinates within the extents used to construct the block model. These data were used for statistical analysis and the preliminary indicator model. In general, the drill hole spacing ranged from 5 to 60 m and averaged about 20 m. Most holes are either vertical or inclined at 60 degrees. About 2% of the drill holes had missing assays that were set to a null value, and not used in the statistics or mineral resource estimation.

14.7 Exploratory Data Analyses (EDA)

14.7.1 Summary Statistics

A mix of sample lengths was submitted to the laboratory for assay analysis for both DD and RC 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 drill data set was composited to 1 m intervals to provide equal support. For grade estimations, the samples were composited into 5 m down-hole composite intervals to match the mining bench height. Table 14-3 and Table 14-4 summarize the statistics for the key elements of samples located within the interpreted 3D solid. One meter composites for gold were capped at 40 g/t to limit skewing the overall mean. The 40 g/t cap applied to 89 of the more than 243,000 composites.

Table 14-3 General Gold Statistics by Geology (based on capped 1 m composites)

Lithology	Count	Min	Max	Mean	Std Dev	Variance	CV
Diorite	73,458	0.005	40.00	0.70	2.11	4.46	3.00
MetaSeds	97,085	0.005	40.00	0.59	1.56	2.43	2.62
Marble	71,995	0.005	40.00	0.41	1.75	3.06	4.22
Mang Zone	1,219	0.010	40.00	4.04	5.73	32.84	1.42
Diorite < 2%S	30,312	0.005	40.00	0.53	2.29	5.26	4.32
MetaSeds < 2%S	31,838	0.005	40.00	0.36	1.45	2.11	4.04
Marble < 2%S	70,208	0.005	40.00	0.39	1.66	2.75	4.28
Mang Zone < 2%S	853	0.010	40.00	4.52	5.65	31.93	1.25
Diorite ≥ 2%S	43,146	0.005	40.00	0.83	1.97	3.86	2.38
Metaseds ≥ 2%S	65,247	0.005	40.00	0.71	1.59	2.54	2.25
Marble ≥ 2%S	1,787	0.005	40.00	1.50	3.76	14.11	2.50
Mang Zone ≥ 2%S	366	0.020	40.00	2.92	5.77	33.28	1.97

Table 14-4 Key Element Statistics (based on 1 m composites)

Metal	Count	Min	Max	Mean	Std Dev	Variance	CV
Gold g/t	245,124	0.005	40.00	0.59	1.86	3.46	3.14
Silver g/t	245,122	0.250	1500.00	2.10	13.61	185.24	6.47
Copper %	239,969	0.001	22.80	0.08	0.23	0.05	2.81
Sulfur %	245,122	0.005	50.00	2.24	2.65	7.05	1.18
Arsenic ppm	244,888	2.500	81644	586	1494	2232780	2.55
Mn ppm	245,055	2.500	630000	2023	7787	60637504	3.85

14.7.2 Histograms

A histogram of sulfur percent and gold grade using 1 m composites shows a bimodal distribution in Figure 14-6. Oxide is defined as material ≤ 2% sulfur and sulfide is > 2% sulfur.

Figure 14-6 Bimodal Distribution of Sulfur

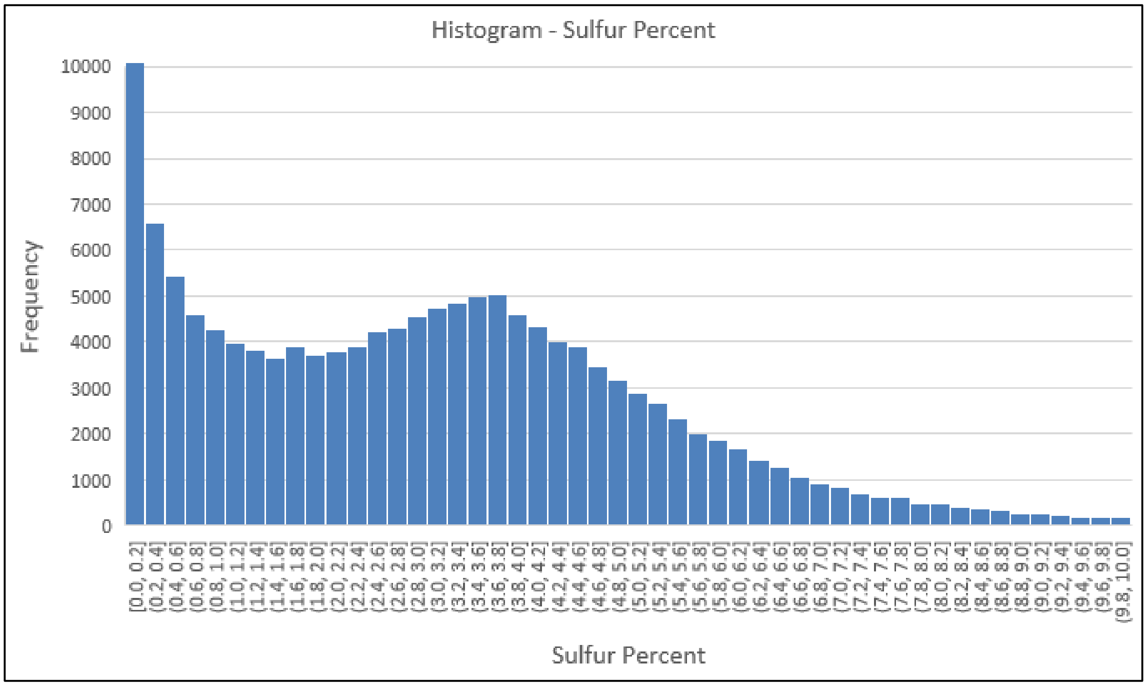


Figure prepared by Alacer, 2016

14.7.3 Boxplots

Boxplots were categorized by lithology. Examples for gold and sulfur are shown in Figure 14-7 and Figure 14-8.

The red diamond is the mean grade; the horizontal line in the box is the median; the lower grey box is the 2nd quartile, with the upper blue box being the 3rd quartile. The vertical line indicates the range of grades within each category. Box plots used 1 m drill composites (cmp).

Figure 14-7 Boxplot of 1 m Au composites Categorized by Lithology

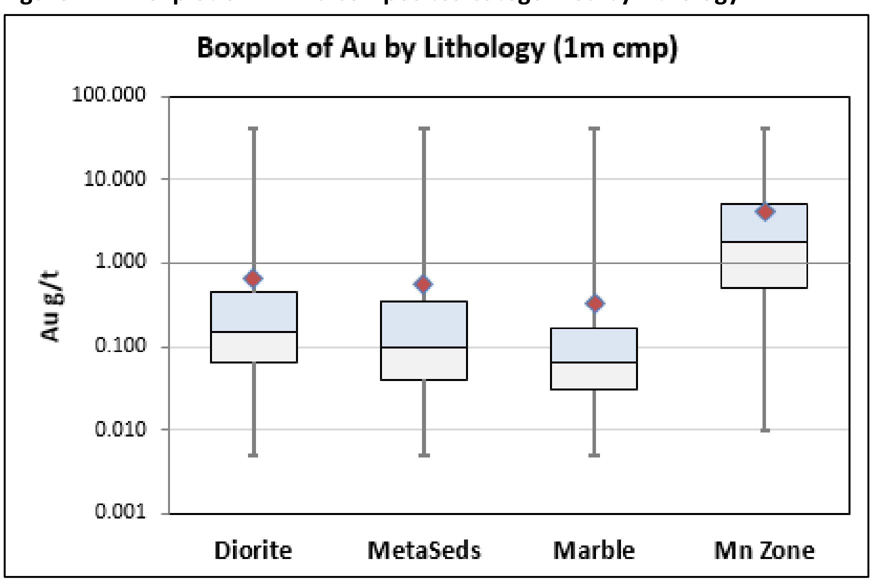
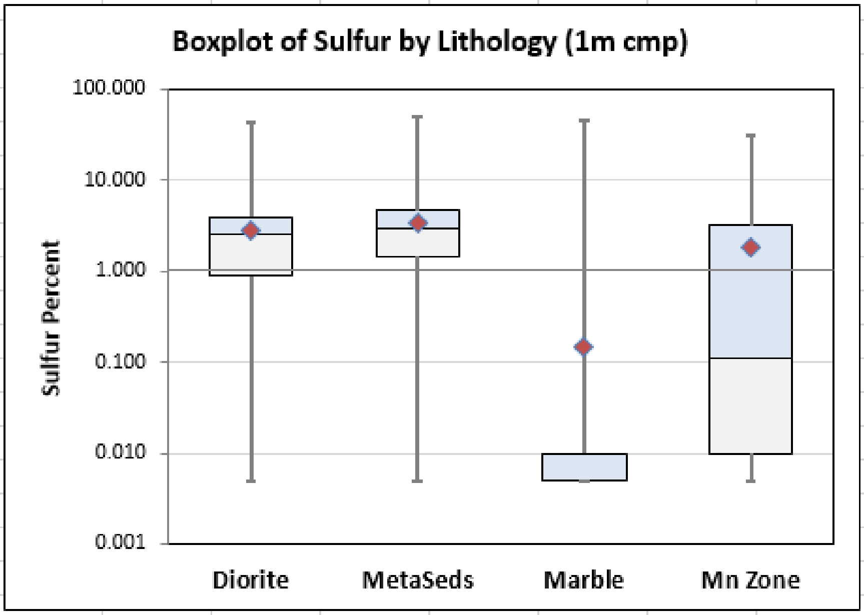


Figure 14-8 Boxplot of 1m S composites Categorized by Lithology



Figures prepared by Alacer, 2016

Key findings from the boxplots are as follows:

- Mean gold grade statistics are similar for diorite, metasediments and marble but higher in the manganese zone. When viewing the data spatially, however, the higher-grade gold mineralization commonly occurs along the lithological contacts especially along the manganese / diorite contact in the manganese domain.
- Mean silver grades were similar for diorite and metasediments, but lower in marble and higher in the manganese zone.
- Mean copper grades varied between lithologies, but in general are higher in the diorite and metasediments.
- Mean gold grades in diorite, metasediments and the manganese zone are higher within the upper oxide material. Marble carries a higher mean grade within the sulfide material.
- Distinctively different sulfur populations were observed for each lithology (although each lithology hosts both low and high-sulfur mineralization) suggesting that sulfur should be domained by lithology for estimation. This approach was taken on the current model.
- Arsenic showed similar mean grades for diorite, metasediments and manganese zone, but had lower mean grades in the marble.

14.7.4 Correlation Coefficients

Correlation coefficients and scatterplots of the elements with the higher correlations were constructed. Correlation coefficients are summarized in Table 14-5.

Table 14-5 Correlation Coefficients using 5m composites

	aufa	*ag_ppm*	*cu_pct*	*as_ppm*	*fe_pct*	*mn_ppm*	*zn_ppm*	*s_pct*
aufa	1.00							
ag_ppm	0.27	1.00						
cu_pct	0.09	0.05	1.00					
as_ppm	0.41	0.17	0.03	1.00				
fe_pct	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_pct	0.05	0.07	0.18	0.23	0.26	0.02	0.14	1.00
	Scale	0.00	0.20	0.40	0.60	0.80	1.00	
		No Correlation				Direct Correlation		

Key findings from the correlation coefficients are as follows:

- There is moderate correlation between
  - gold and arsenic
  - copper and iron
- Minor correlations occur between:
  - gold and silver
  - silver and arsenic
  - silver and manganese

Although a correlation probably exists between gold and sulfur on a mineralogical level as suggested by correlation between gold and arsenic and observed presence of arsenopyrite (FeAsS), this correlation is probably masked by the much larger episode of non-auriferous sulfide mineralization. This suggests that it is reasonable to model silver, copper, zinc, arsenic and manganese using the gold envelopes.

A scatter plot of gold g/t to arsenic is shown in Figure 14-9.

Figure 14-9 Scatter plot of Gold to Arsenic - 5 m composites

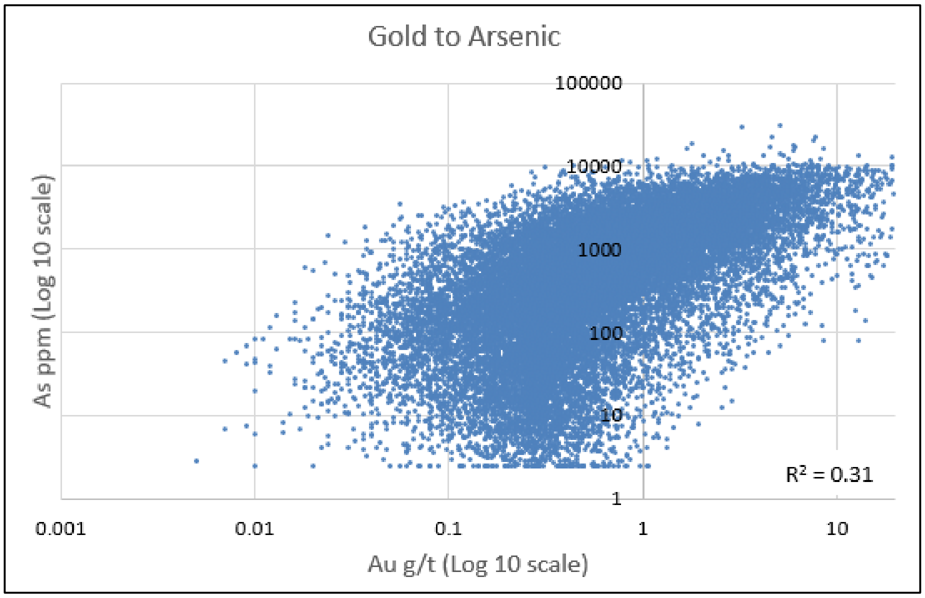


Figure prepared by Alacer, 2016

14.8 Core Recovery

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

- No correlation exists between any of the elements and core recovery.
- There is no obvious increase or decrease in gold grade with lower core recovery.

In 2014, two nearest-neighbor (NN) models were constructed to quantify the influence of the drill hole assays with low 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%.

14.9 Twin Holes

Twenty-three twin hole comparisons were made between various combinations of DD holes and RC holes for gold, sulfur and copper. 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 summarized with the following results:

- The average RC gold 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 copper, little difference was noted between the DD holes and RC holes, but the grades were very low.
- The PQ holes showed about a 6% higher grade, but the data set is limited

In conclusion, the twin hole comparisons agree well.

14.10 Contact Plots

Contact plots were constructed for the different combinations of lithological contacts and categorized by material residing within the upper oxide or lower sulfide portion of the deposit. The oxide and sulfide boundary used for the plots was defined by the interpreted oxidation surface based on visual logging. An example of a contact plot of gold across the diorite–metasediments contact for sulfide material is shown in Figure 14-10.

Figure 14-10 Contact plot of Au across the Diorite–Metasediments Contact for Sulfide Material

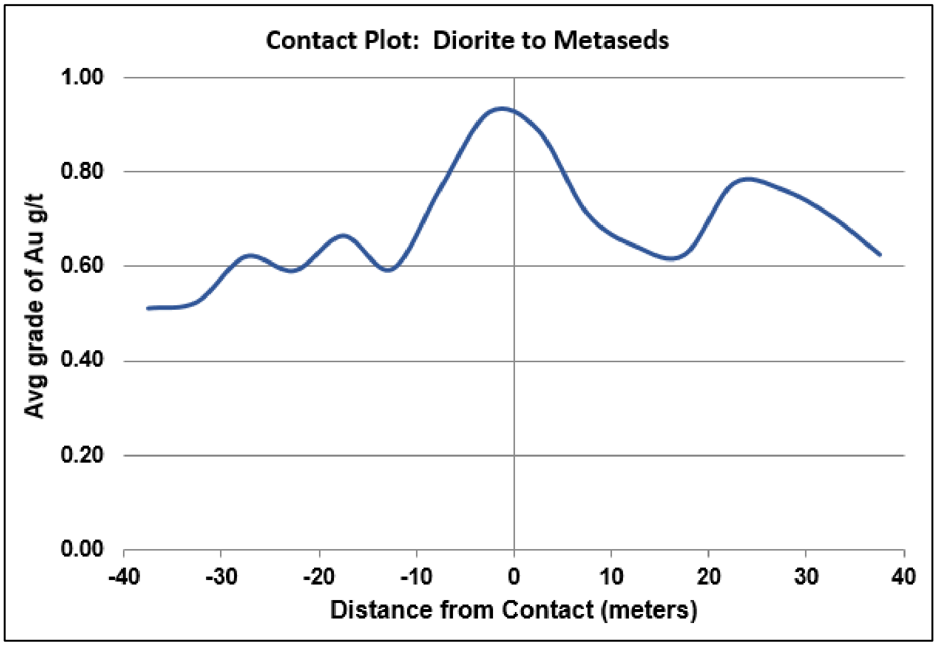


Figure courtesy Alacer, 2015

In general, no hard contacts were observed for gold, and Figure 14-11 shows that the higher-grade Au mineralization commonly occurs along the lithologic contacts. This is shown in cross-sections that show higher-grade gold mineralization commonly occurring along the lithologic contacts and indicates that the gold mineralization should not be modeled separately by lithological domains.

Figure 14-11 Gold Assays above 1 g/t at Lithology Contacts, Section 459,400E, Looking West

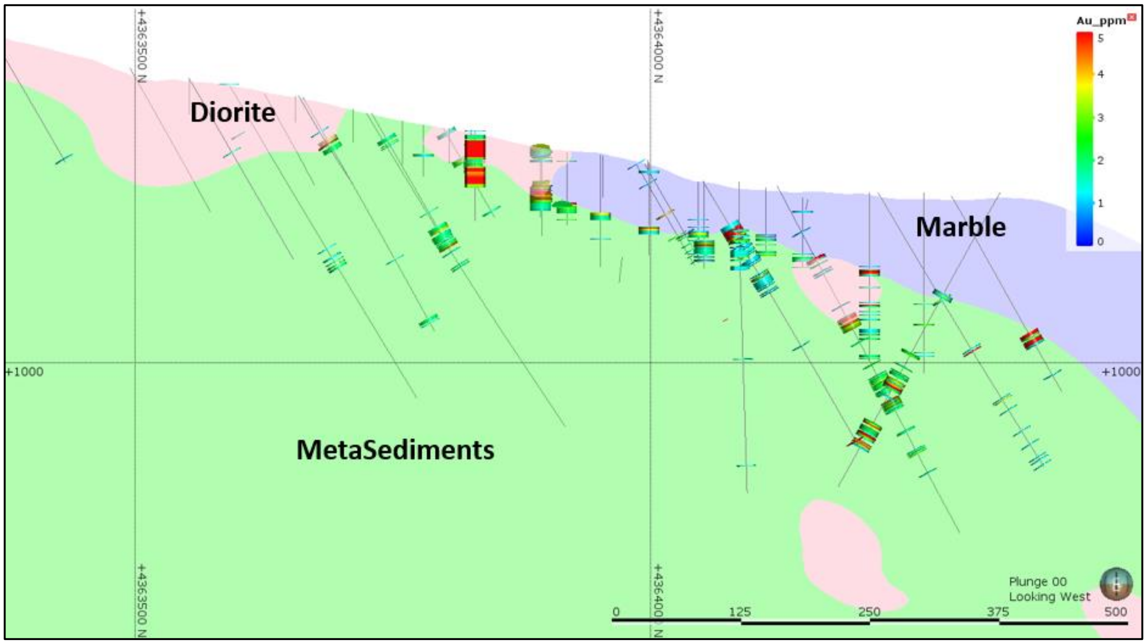


Figure courtesy of Alacer, 2014

14.11 Capping (Top Cutting)

In mineral deposits having skewed distributions, it is not uncommon for 1% of the highest assays to disproportionately account for over 20% of the total metal content in the resource model. Although these assays are real and reproducible, they commonly show little continuity, and add a significant amount of uncertainty to the mineral resource estimate.

Since high-grade material is not usually drilled to a suitable spacing to verify its spatial limits, the very high-grade assays should be constrained during mineral resource estimation to minimize the high risk of this material and local grade overestimation. One way to minimize the influence of these samples is to apply a top cut or cap grade to the assays before compositing and mineral resource estimation.

To determine an appropriate capping grade, capping studies were performed for each of the domains categorized by < 2% S and ≥ 2% S. The capping studies performed were:

- Looking for kinks or discontinuities in cumulative log probability plot (CLPP).
- Decile analysis. (Parrish, 1997)
- Quantifying the number of high-grade samples lying in close proximity to each other (DIST).

Results for each method were compared and a capping threshold was selected. Capping was performed on the 1 m composites before compositing into the 5 m composites used for the mineral resource estimations. Gold was studied and capped by domain and low / high-sulfur material. Capping thresholds for silver, copper, sulfur, arsenic, and iron, manganese and zinc were applied globally. For some of the variables, such as arsenic, the value selected was partially due to the upper limit of the assay method used. In this case, the 10,000 ppm arsenic value was an artificial break due to the recording of this value in the database when the maximum assay threshold was exceeded. This practice was not used throughout the life of the project, and therefore values above the 10,000 ppm arsenic threshold were included in the database.

As a result, the model of the elements with assays that exceed the range of the analytical method should be used with caution. The capping thresholds applied before compositing are summarized in Table 14-6 and Table 14-7.

Table 14-6 Capping Limits for Au (g/t)

Domain #	Domain Name	Metal	Sulfur %	CAP g/t
1	Manganese	Au	< 2	**18**
1	Manganese	Au	≥ 2	**18**
2	Main	Au	< 2	**16**
2	Main	Au	≥ 2	**14**
3	Marble	Au	< 2	**30**
3	Marble	Au	≥ 2	**25**
4	West Zone	Au	< 2	**16**
4	West Zone	Au	≥ 2	**14**

Table 14-7 Capping Thresholds Applied Globally

Element	Cap
Ag_ppm	300
Cu_pct	5
S_pct	20
C_pct	13
As_ppm	30,000
Fe_pct	50
Mn_ppm	100,000
Zn_ppm	60,000

14.12 Drill Hole Compositing

Composites used for Mineral Resource estimations were prepared by first compositing to 1 m down-the-hole intervals to provide equal support for the capping studies. Gold composites were capped at 40 g and used for EDA and capping studies. The 1 m capped intervals were then composited into 5 m down-the-hole intervals for additional EDA studies and mineral resource estimation. The composites were not broken across lithological contacts, or domain boundaries. The 5 m interval was selected as it matches the mining bench height. Statistics of the 5 m gold composites used for mineral resource estimation are summarized in Table 14-8.

Table 14-8 Drill Hole Composite Statistics for Au by Domain (based on 5 m capped composites)

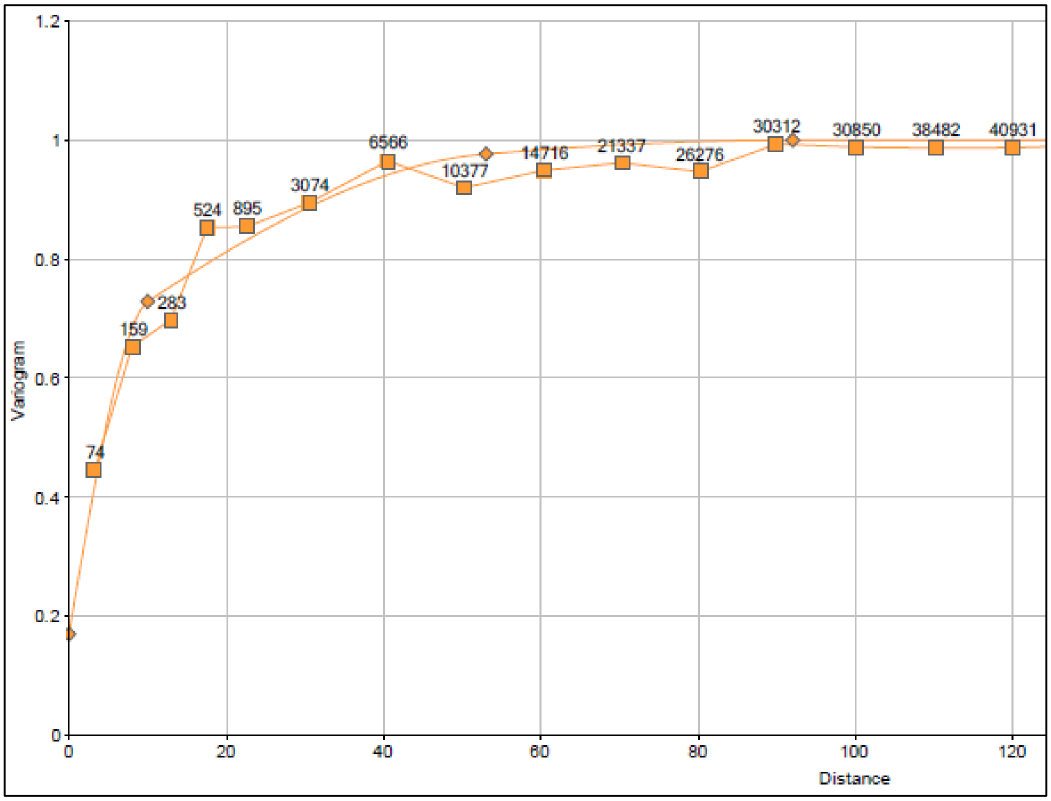
Domain	Count	Min	Max	Mean	Std Dev	Variance	CV
Mng Zone	9,597	0.005	18.00	0.74	1.60	2.55	2.16
Main Zone	31,200	0.005	15.58	0.54	1.09	1.19	2.01
Marble Zone	4,598	0.005	30.00	0.76	2.60	6.76	3.40
West Zone	4,212	0.005	12.69	0.24	0.69	0.47	2.88
Diorite < 2%S	6,115	0.005	17.04	0.58	1.48	2.18	2.54
MetaSeds < 2%S	13,941	0.005	15.58	0.31	0.80	0.65	2.61
Marble < 2%S	3,696	0.005	30.00	0.76	2.70	7.27	3.56
Mang Zone < 2%S	2,534	0.005	12.69	0.26	0.83	0.70	3.16
Diorite ≥ 2%S	3,482	0.007	18.00	1.02	1.75	3.06	1.72
Metaseds ≥ 2%S	17,259	0.005	15.32	0.73	1.25	1.55	1.70
Marble ≥ 2%S	902	0.005	22.03	0.79	2.16	4.67	2.74
Mang Zone ≥ 2%S	1,678	0.005	5.08	0.20	0.36	0.13	1.80

14.13 Variography

The EDA showed that the trends of the gold mineralization followed lithologic contacts and structures which vary by domain. As a result, variograms (correlograms) were calculated for gold, silver and copper composites for each domain categorized by < 2% S and ≥ 2% S.

The directions of the anisotropy axes were first determined by creating multi-directional variograms, variogram models, and visual observation of the tabular shaped trends of the mineralization. After the anisotropy had been determined, three directional variograms were calculated and modeled in each of the three primary anisotropic directions. Since the low and high-sulfur domain variograms showed similar structures, with the low-sulfur domain variogram structures better defined, the low-sulfur domain variograms were used for the Mineral Resource estimation. An example modeled gold variogram for the Main domain is shown in Figure 14-12. Variogram parameters are summarized in Table 14-9.

Figure 14-12 Example Gold Variogram for Main Domain



1. Figure courtesy of Amec Foster Wheeler, 2015
2. Azimuth = 147, Inclination = 0
3. Orange line through the diamonds represents the model

Table 14-9 Variogram Parameters Used in the Mineral Resource Estimation

Domain - Element	Azim / Inclin	Axis	Nugget	Structures C1/C2/C3	Ranges a1/a2/a3
Mn Domain - Ag	302 / 0	X	0.17	0.27 / 0.54 / 0.02	51 / 82 / 250
	212 / 52	Y	0.17	0.27 / 0.54 / 0.02	19 / 82 / 235
	32 / 38	Z	0.17	0.27 / 0.54 / 0.02	36 / 73 / 112
Mn Domain - Au	302 / 0	X	0.12	0.55 / 0.32 / 0.01	43 / 88 / 222
	212 / 52	Y	0.12	0.55 / 0.32 / 0.01	20 / 63 / 235
	32 / 38	Z	0.12	0.55 / 0.32 / 0.01	30 / 40 / 112
Mn Domain - Cu	302 / 0	X	0.21	0.26 / 0.36 / 0.17	22 / 58 / 234
	212 / 52	Y	0.21	0.26 / 0.36 / 0.17	12 / 74 / 100
	32 / 38	Z	0.21	0.26 / 0.36 / 0.17	48 / 52 / 95
Main Domain - Ag	147 / 0	X	0.37	0.24 / 0.26 / 0.13	22 / 68 / 196
	57 / 10	Y	0.37	0.24 / 0.26 / 0.13	22 / 63 / 192
	237 / 80	Z	0.37	0.24 / 0.26 / 0.13	9 / 31 / 124
Main Domain - Au	147 / 0	X	0.17	0.47 / 0.26 / 0.10	10 / 53 / 92
	57 / 10	Y	0.17	0.47 / 0.26 / 0.10	12 / 74 / 192
	237 / 80	Z	0.17	0.47 / 0.26 / 0.10	12 / 48 / 196
Main Domain - Cu	147 / 0	X	0.17	0.41 / 0.25 / 0.17	18 / 93 / 300
	57 / 10	Y	0.17	0.41 / 0.25 / 0.17	26 / 65 / 182
	237 / 80	Z	0.17	0.41 / 0.25 / 0.17	10 / 33 / 300
Marble Domain - Ag	210 / 0	X	0.19	0.30 / 0.51 / 0.0	48 / 83 / 242
	120 / 50	Y	0.19	0.30 / 0.51 / 0.0	72 / 90 / 192
	300 / 40	Z	0.19	0.30 / 0.51 / 0.0	33 / 80 / 200
Marble Domain - Au	210 / 0	X	0.06	0.82 / 0.08 / 0.04	52 / 77 / 106
	120 / 50	Y	0.06	0.82 / 0.08 / 0.04	21 / 53 / 121
	300 / 40	Z	0.06	0.82 / 0.08 / 0.04	21 / 55 / 200
Marble Domain - Cu	210 / 0	X	0.26	0.47 / 0.27 / 0.00	27 / 114 / 186
	120 / 50	Y	0.26	0.47 / 0.27 / 0.00	49 / 99 / 121
	300 / 40	Z	0.26	0.47 / 0.27 / 0.00	31 / 83 / 250
West Domain - Ag	50 / 0	X	0.24	0.03 / 0.25 / 0.48	17 / 50 / 106
	320 / 65	Y	0.24	0.03 / 0.25 / 0.48	20 / 108 / 140
	140 / 25	Z	0.24	0.03 / 0.25 / 0.48	9 / 53 / 105
West Domain - Au	50 / 0	X	0.20	0.11 / 0.68 / 0.01	17 / 42 / 91
	320 / 65	Y	0.20	0.11 / 0.68 / 0.01	24 / 40 / 115
	140 / 25	Z	0.20	0.11 / 0.68 / 0.01	32 / 48 / 105
West Domain - Cu	50 / 0	X	0.08	0.54 / 0.35 / 0.03	46 / 300 / 400
	320 / 65	Y	0.08	0.54 / 0.35 / 0.03	42 / 194 / 300
	140 / 25	Z	0.08	0.54 / 0.35 / 0.03	200 / 500 / 500

14.14 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 stockpile for the proposed POX plant. EDA showed that sulfur should be modeled separately in each of the four main geologic units (marble, diorite, metasediments and manganese zone). The sulfur estimate proved to be very sensitive, as minor changes in the estimation parameters causes the reclassification of material from high to low-sulfur and vice versa. The change in the sulfur classification 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 lithologic domain, a sulfur indicator was generated using a discriminator of 2% total sulfur. To accomplish this, a sulfur indicator field was created in the drill data set. A sulfur indicator of 0 was used if the total sulfur percentage was between 0 and 2. A value of 1 was used in the sulfur indicator field when the total sulfur percentage was ≥ 2%. This was modeled using NN and ID2 estimation methods. ID2 will give indicator estimates between 0 and 1. The number of blocks above and below 2% sulfur was first determined using the NN model, and the ID2 indicator estimate was calibrated against the NN model so the proportion of low and high-sulfur material honored the NN proportions. Sulfur indicator estimate thresholds that honored the results of the NN estimation for low-sulfur/high-sulfur proportions were:

- Diorite = 0.50
- Metasediments = 0.51
- Marble = 0.26
- Manganese zone = 0.36

To soften the low/high sulfur boundary, the sulfur indicator estimate used to select the composites were adjusted by slightly raising the maximum indicator estimate for the oxide estimate and lowering the minimum indicator estimate for the sulfide estimate. The 5 m sulfur composites were used to interpolate sulfur grades modeled into the low-sulfur and high-sulfur domains for each geological unit using an inverse distance weighted to the second power (ID2) method. All searches were isotropic, and the sulfur model was not constrained by the gold 3D solid. This means sulfur was also estimated in the waste rock for waste rock characterization.

Sample indicator estimate limits by lithology and material type are summarized in Table 14-10, estimation parameters are summarized in Table 14-11, and a typical cross section is shown in Figure 14-13.

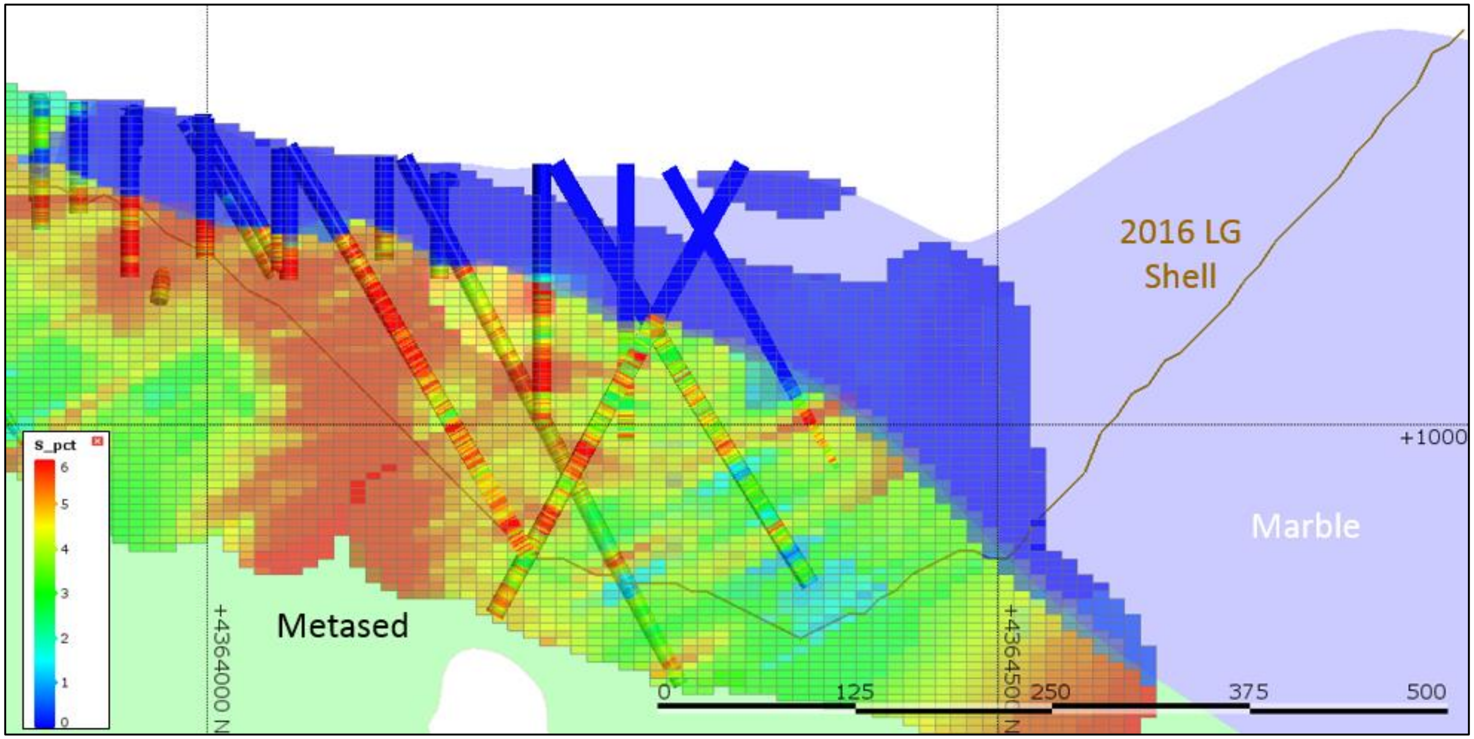
Table 14-10 Sulfur Block and Sample Indicator Estimate Limits

Estimating	Lithology	Sulfur Ind Blocks		Sulfur Ind Samples	
		Min	Max	Min	Max
Low S	Diorite	0.00	0.50	0.00	0.58
	Metased	0.00	0.51	0.00	0.59
	Marble	0.00	0.26	0.00	0.30
	Manganese	0.00	0.36	0.00	0.40
High S	Diorite	0.50	1.00	0.40	1.00
	Metased	0.51	1.00	0.40	1.00
	Marble	0.26	1.00	0.21	1.00
	Manganese	0.36	1.00	0.29	1.00

Table 14-11 Sulfur Estimation Parameters

Model Variable	Estimate Method	Pass	Samples Min	Samples Max	Max # Per Hole	Search Distance (m) X	Search Distance (m) Y	Search Distance (m) Z
s_pct	ID2	1	3	12	2	40	40	40
s_pct	ID2	2	3	12	2	60	60	60
s_pct	ID2	3	2	12	2	160	160	160

Figure 14-13 Cross-Section of Sulfur Model, Section 459,400E, Looking West



1. Figure courtesy of Alacer, 2016
2. The distinct breaks in sulfur grades occur along the lithological contacts.

14.15 Gold and Other Metal Models

A total of nine elements, gold, silver, copper, sulfur, carbon, zinc, iron, arsenic and manganese were estimated. Gold, copper and silver were estimated using ordinary kriging (OK) and the remaining elements were estimated using the ID2 method. Zinc, iron, arsenic and manganese were restricted to the mineralized gold envelopes, as these elements are only used for material-type classification.

Estimation parameters used in the PACK model are shown in Table 14-13. All blocks were estimated using a block discretisation of 3 x 3 x 1.

The volume of the PACK mineralized envelope was calibrated to past production by:

Creating a production block model:

- Constructing a 3 x 3 x 5 m block model in the areas that have been mined
- Populating the 3 x 3 x 5 m blocks with the ore control tonnes and grades estimated from blast hole assays by site
- Tabulating ore control tonnes and grade from January 2014 through October 2015.

Building an indicator model and estimation of gold grade:

- The low-sulfur gold estimates were constructed using an indicator model defined using a 0.3 g/t Au discriminator. First a low-grade indicator field was established in the drill composite file. If the composite gold grade was < 0.3 g/t, the low-grade indicator field was set to 0; if the composite Au grade was ≥ 0.3 g/t, the low-grade indicator was set to 1. The low-grade indicator was then estimated into all the blocks, and blocks with an estimated low-grade indicator greater than 0.30 were selected to define the mineralized envelope. The estimated low-grade indicator values in the block model were then tagged back into the composites and only composites within the mineralized envelope were used to estimate the gold grade, Figure 14-14.
- Similarly, a high-grade gold estimate was constructed using a high-grade indicator model using the same method except the gold discriminator was increased from 0.3 to 1.5 g/t to reflect the higher cut-off required for processing the material through the POX plant. The high-grade gold estimate also uses a 0.3 estimated discriminator threshold as the boundary limits.
- The low-grade gold estimates were applied to those blocks with estimated sulfur grades < 2%, and the high-grade gold estimates were applied to those blocks with estimated sulfur grades ≥ 2% S. Figure 14-15 shows the blocks with sulfur grades higher than 2% within the indicator model. Figure 14-16 shows the estimated gold grades after combining the two indicator estimates.

Calibrating the PACK model:

- The PACK model parameters were then adjusted so the gold ounces in the PACK model approximates the gold ounces reported from the ore control; this is explained further in Section 14.18.4. The calibrations were categorized by domain and by oxide/sulfur material types, Table 14-12.
- After the gold ounces were calibrated by domain and material type, blocks with estimated gold grades below the selected indicator threshold were set to waste.

Table 14-12 Selected Gold Indicator Values for Oxide material by Domain

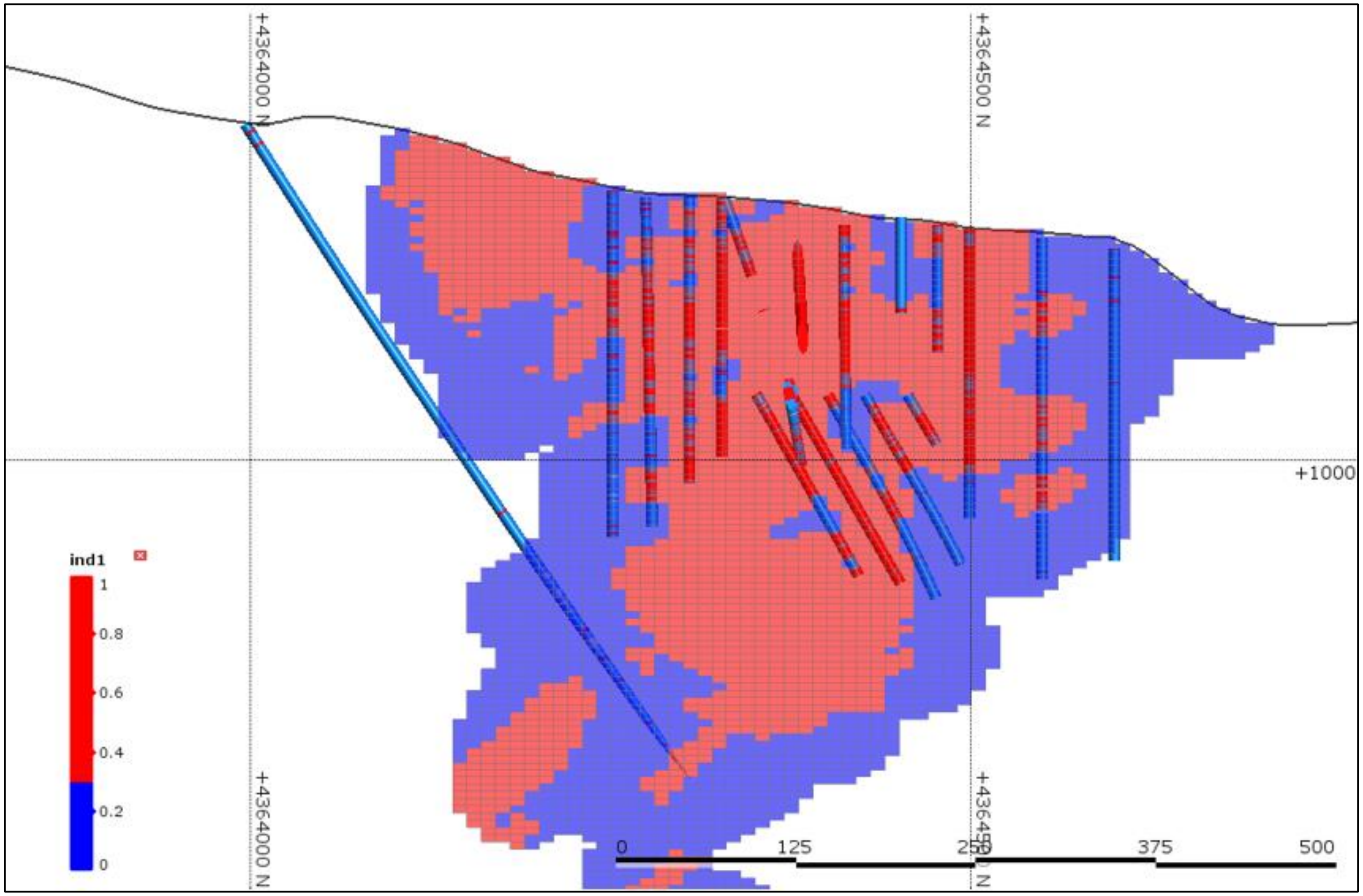
Indicator Domain	Pit	Estimated Indicator Threshold
1	Manganese	0.30
2	Marble	0.35
3	Main	0.56
4	West	0.47

Sulfide production data showed higher grades and lower tonnes across all domains when compared to the resource estimate, therefore an indicator threshold above 1.5 was not applied to reduce sulfide tonnes in the resource model.

Various periods of production data were reviewed with data from the 2014 and 2015 time-period used for model calibration. Production ore tonnes totaled 11.3 Mt for oxide and 3.0 Mt for sulfide. The Marble, Manganese and Main pit were the majority of the production tonnes with limited mining in the West pit.

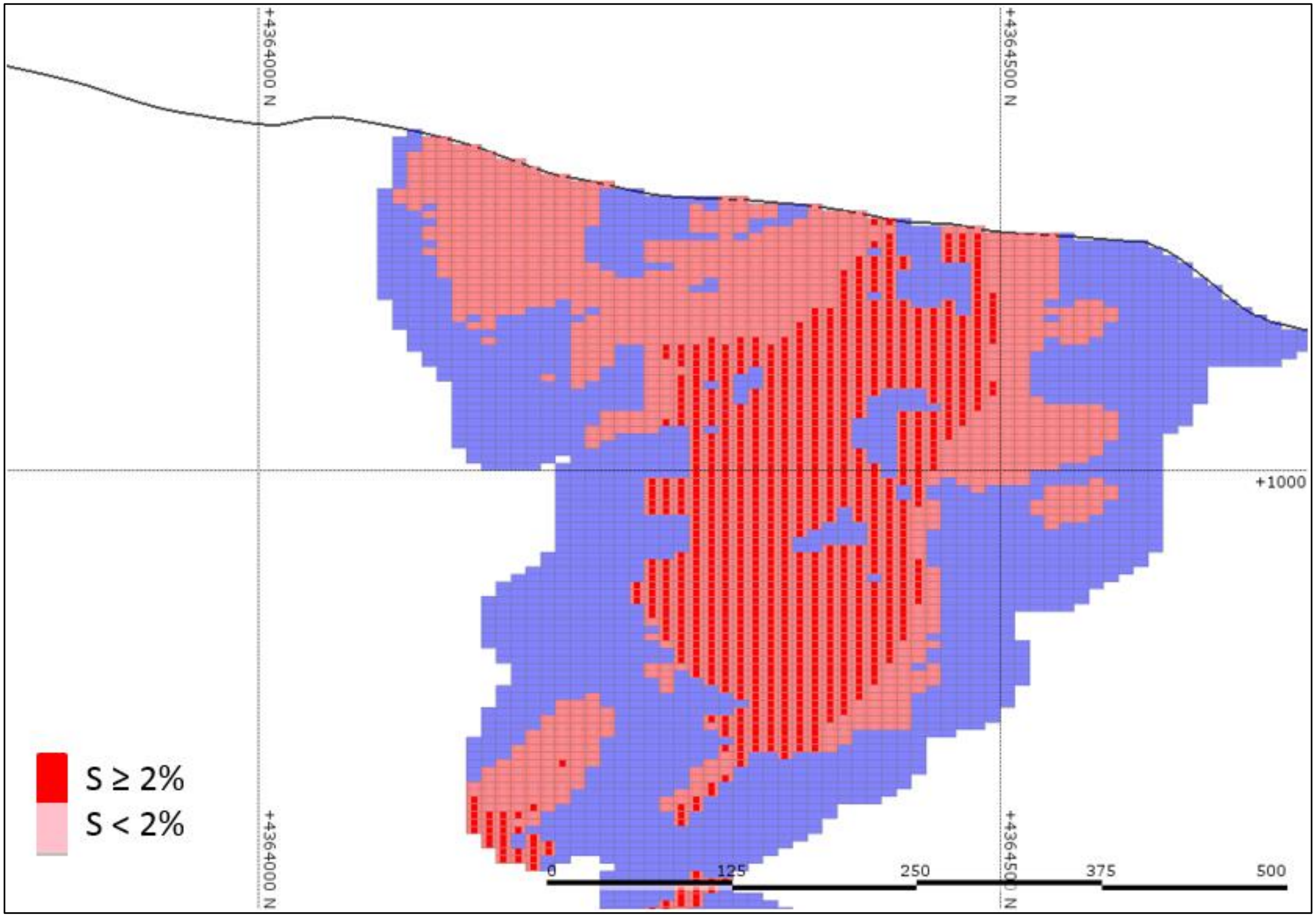
Estimated gold grades in blocks with an indicator estimate less than the indicator estimate threshold were set to a waste grade of 0.001 g/t using a block calculation script.

Figure 14-14 Geometry of the Gold Indicator PACK Model within Au 3D solid; Section 460,000E, looking West



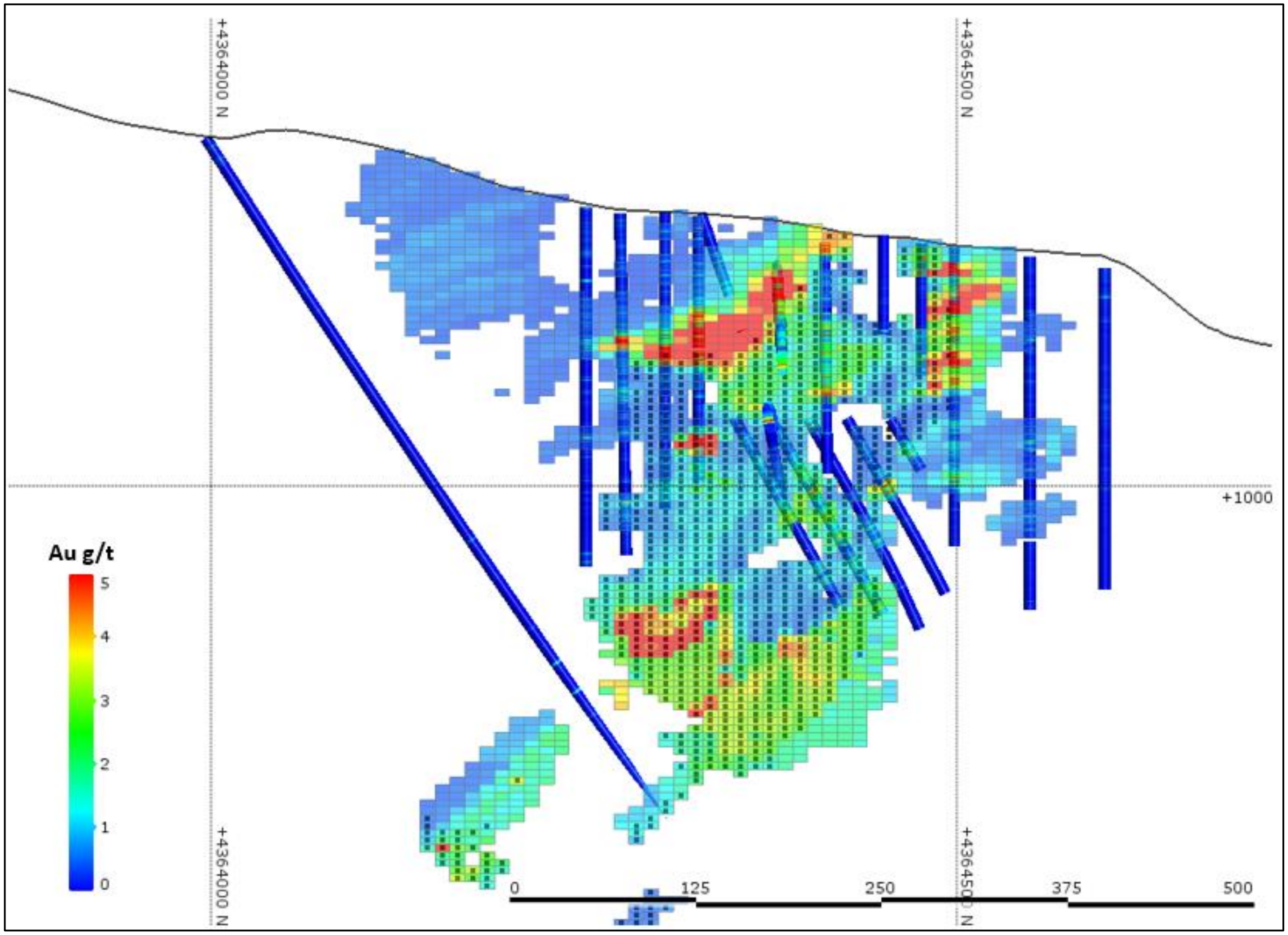
1. Figure courtesy of Alacer, 2016
2. Blocks are colored by estimated indicators, blue is indicator < 0.30, red is indicator ≥ 0.30
3. Drill holes are colored by gold grade, blue is Au < 0.3 g/t, red is Au ≥ 0.3 g/t

Figure 14-15 Geometry of 2% Sulfur Blocks within Gold Indicator PACK Model; Section 460,000E, looking West



1. Figure courtesy of Alacer, 2016
2. Blocks are colored by indicators, blue is indicator < 0.30, red is indicator ≥ 0.30
3. Within the indicator area, blocks are colored by sulfur (S) grade, pink is S < 2.0%, red is S ≥ 2.0%

Figure 14-16 Combined Indicator and Sulfur PACK Gold Estimates; Section 460,000E, looking West



1. Figure courtesy of Alacer, 2016
2. Drill holes and model use same color legend
3. Black dots indicate blocks with S ≥ 2.0%

Table 14-13 Summary of the Estimation Parameters Used in the PACK Estimations

Domain / Element	Azim / Inclin	Search Pass 1		Search Pass 2		Search Pass 3	
		Distance	Min / Max	Distance	Min / Max	Distance	Min / Max
All Domains	90 / 0	40	3 / 12	60	3 / 12	160	2 / 12
Sulfur	0 / 0	40	3 / 12	60	3 / 12	160	2 / 12
	0 / -90	40	3 / 12	60	3 / 12	160	2 / 12
Mn Domain	302 / 0	40	3 / 12	60	3 / 12	150	2 / 12
Low-grade Au, Ag, Cu	212 / -52	40	3 / 12	60	3 / 12	150	2 / 12
	32 / -38	30	3 / 12	40	3 / 12	80	2 / 12
Mn Domain	302 / 0	40	3 / 12	60	3 / 12	150	2 / 12
High-grade Au	212 / -52	40	3 / 12	60	3 / 12	150	2 / 12
	32 / -38	30	3 / 12	40	3 / 12	80	2 / 12
Main Domain	147 / 0	40	3 / 12	60	3 / 12	150	2 / 12
Low-grade Au, Ag, Cu	57 / -10	40	3 / 12	60	3 / 12	150	2 / 12
	237 / -80	20	3 / 12	30	3 / 12	75	2 / 12
Main Domain	147 / 0	40	3 / 12	60	3 / 12	150	2 / 12
High-grade Au	57 / -10	40	3 / 12	60	3 / 12	150	2 / 12
	237 / -80	20	3 / 12	30	3 / 12	75	2 / 12
Marble Domain	210 / 0	40	3 / 12	60	3 / 12	150	2 / 12
Low-grade Au, Ag, Cu	120 / -50	40	3 / 12	60	3 / 12	150	2 / 12
	300 / -40	20	3 / 12	30	3 / 12	75	2 / 12
Marble Domain	210 / 0	40	3 / 12	60	3 / 12	150	2 / 12
High-grade Au	120 / -50	40	3 / 12	60	3 / 12	150	2 / 12
	300 / -40	20	3 / 12	30	3 / 12	75	2 / 12
West Domain	50 / 0	40	3 / 12	60	3 / 12	150	2 / 12
Low-grade Au, Ag, Cu	320 / -65	40	3 / 12	60	3 / 12	150	2 / 12
	140 / -25	20	3 / 12	30	3 / 12	75	2 / 12
West Domain	50 / 0	40	3 / 12	60	3 / 12	150	2 / 12
High-grade Au	320 / -65	40	3 / 12	60	3 / 12	150	2 / 12
	140 / -25	20	3 / 12	30	3 / 12	75	2 / 12

14.16 Çöpler Resource Classification

Resources were classified using a common industry and Amec Foster Wheeler internal guideline that Indicated Mineral Resources 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. At this level, the drilling is usually sufficiently close-spaced enough to permit confirmation (Measured) or assumption of continuity (Indicated) between points of observation. For the Çöpler model, a drill hole spacing study was performed to determine the nominal drill hole spacing required to classify material as Indicated.

Confidence limits were calculated on a single block that represents one month's POX production (based on 1.9 Mt/yr). The confidence limits, a review of continuity on sections and plans, and an assessment of data quality were used to determine minimum drill hole spacing by domain. A spacing of 40 by 40 m in Marble, 50 by 50 m in the Manganese and West zone, and 60 x 60 m in the Main zone was required to meet the requirements for Indicated. An 80 by 80 m spacing was required for Inferred in all domains. Blocks with a drill hole spacing that was greater than 80 m were not classified. The classification was then smoothed to remove the isolated blocks with a different classification than the surrounding blocks.

No blocks in the model were classified as Measured Mineral Resources due to the following:

- Reconciliation for sulfide material to date has shown mined to model variances are greater than 15% (positive) over annual periods. To date, the observed variances are not fully understood. Verification of quantity and grade are currently being checked through a sulfide stockpile drill program.
- Documentation of the historic collar locations and down-hole surveys are not available.
- Verification of the blast hole database used to calibrate the model against the site Labware LIMS system has not been completed. The site laboratory was audited by Tramecon in May 2015. The audit report contains gaps in QA/QC protocol and out of date lab laboratory standard operating procedures (SOPs).
- Additional sampling and assay analysis are being obtained through the stockpile drill program for sulfide sulfur, carbon, copper, silver, and manganese grades.

The resulting classification shows the majority of the deposit can be classified as Indicated (green) with Inferred blocks (blue) forming a halo around the Indicated mineralization, Figure 14-17.

Figure 14-17 Plan view of Resource Classification

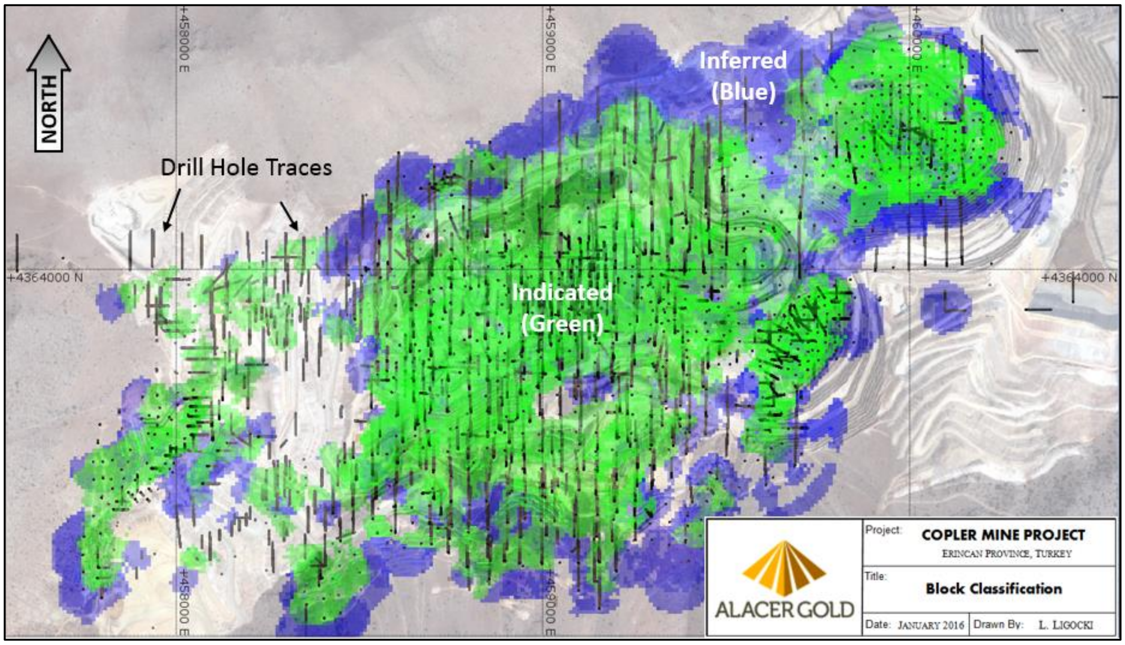


Figure courtesy of Alacer, 2016

14.17 Density Model

To minimize the block model file size and increase its display functionality, a bounding solid was used to limit block generation within the block model extents. This reduced the number of blocks by approximately 50% by removing the unnecessary corners of the model and blocks above the original topography. The extents of the model were designed so the limits of the model would not restrict the Lerchs-Grossmann (LG) optimizations.

Skarn and gossan shapes for the model area were updated but not included into the block model due to limited volume and thickness in relation to individual block size. The prior modeling nomenclature for rock type numbering was retained to keep scripting and downstream processes the same for engineering purposes.

Waste stockpiles through August 2015 were included and flagged as “dump” in the block model with an assigned density of 1.80 t/m^3. Blocks for this material were included for LG runs and financial consideration. The leach pad east of the Manganese pit was also included in the model to restrict LG optimizations.

14.17.1 Density Model Construction

Density measurements were performed on representative diamond drill core by the site exploration geologists using the wax coated water displacement method. Results were then sent to the Anagold database manager where they were loaded into the corporate database. Density measurements from the recent 2015 drilling were appended to the density data used in the previous resource model for statistical analysis and modeling.

Density values were assigned to the block 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.

The density samples were then flagged by depth using wireframe solids for the three depth categories. The fourth category (greater than 60 m) was considered as the default, and no solids were constructed for this category.

Density data for the Çöpler Project were then reviewed spatially and statistically. Density values that fell outside the expected upper and lower density limits were considered to be outliers and removed, Table 14-14.

Table 14-14 Upper and Lower Density Limits by Rock Type

Rock Type	Lower limit	Upper limit
diorite	1.7	3.5
manganese	*none*	none
marble	1.7	3.5
metaseds	1.7	3.5

Note: units in t/m^3

The data were plotted by depth below the original topographic surface, and categorized by rock type. The average density was then calculated in 20 m depth bins below the original topographic surface. Based on the statistical analysis, density values were assigned by rock type and depth to the resource model. Density values are plotted by depth for the diorite in Figure 14-18 and for the metasediments in Figure 14-19. Since very little change in density with depth was noted for the marble, a single density value was applied to all blocks in the resource model coded as marble.

In total, 5,678 density measurements were used to estimate density. Since the majority of the measurements were taken in the diorite, marble and metasediments, the densities for these units are considered to be more reliable than the resulting manganese density value used. These data reflect the observed geology showing that the diorites and metasediments are more weathered near 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 summarized in Table 14-15.

Figure 14-18 Diorite Density Values by Depth below Surface

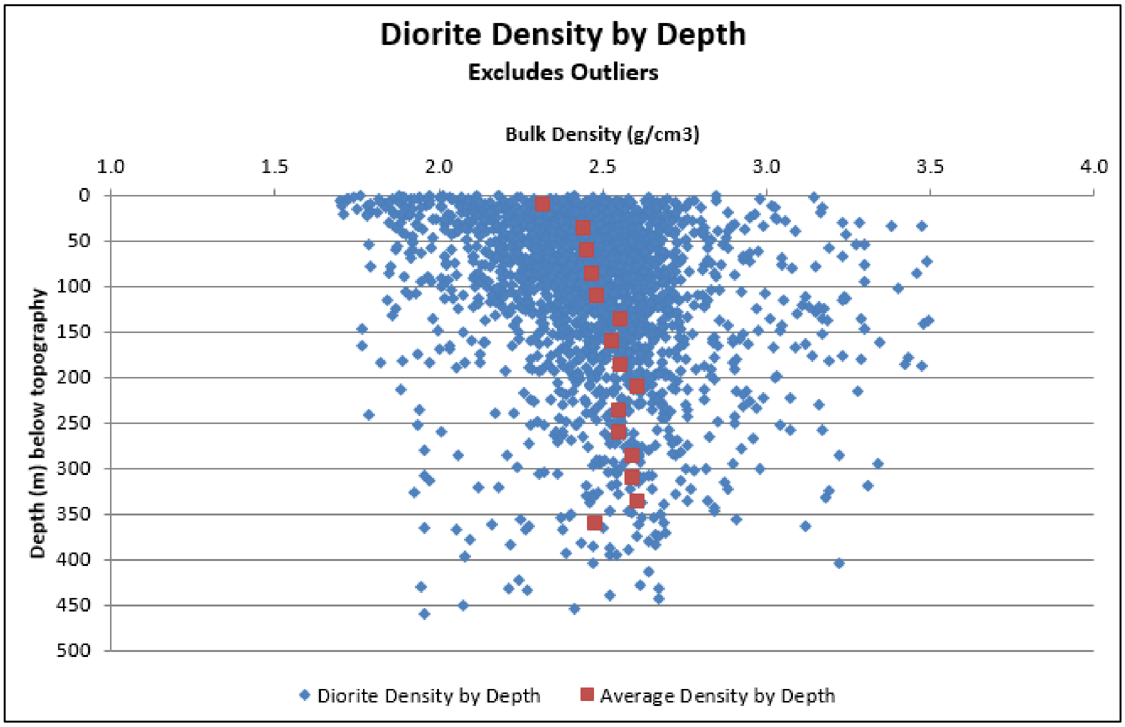


Figure courtesy of Alacer, 2015

Figure 14-19 Metasediments Density Values by Depth below Surface

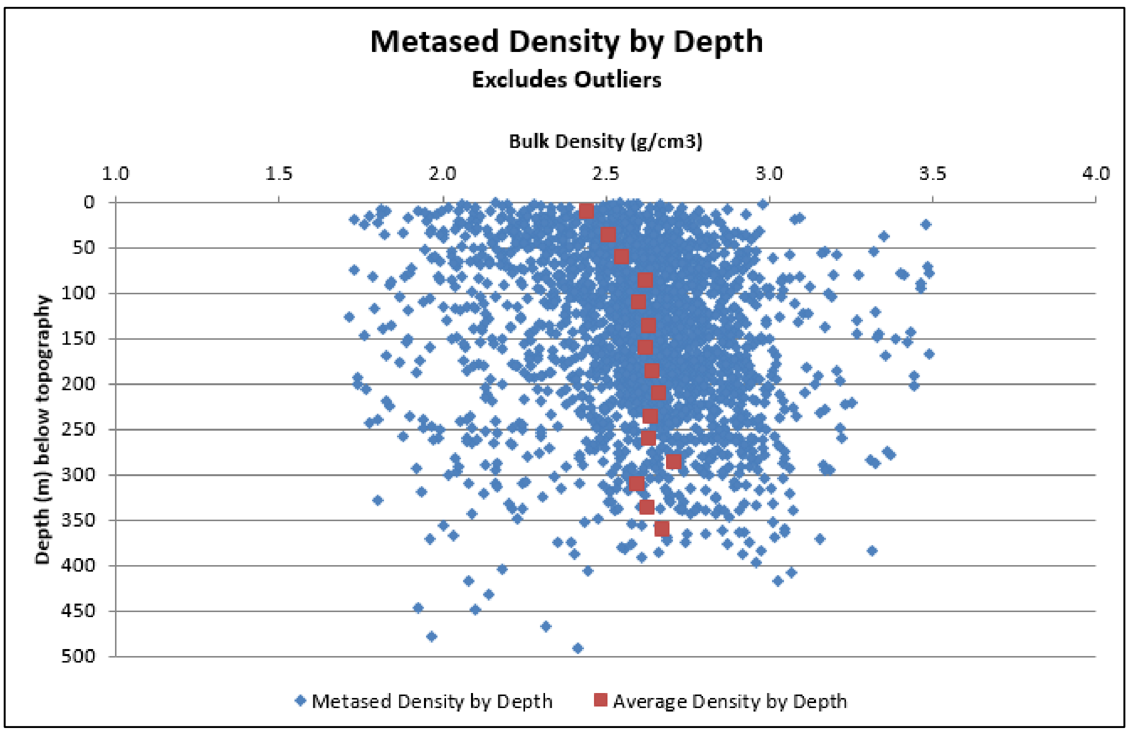


Figure courtesy of Alacer, 2015

Table 14-15 Density Values Assigned to the Block Model by Rock Type and Vertical Depth

Rock Type	Depth (m)	# of Samples	Density (g/cm^3)
Diorite_1	0 - 20	111	2.22
Diorite_2	20 - 40	173	2.42
Diorite_3	40 - 60	155	2.44
Diorite_4	60+	1653	2.50
Marble	all	1099	2.57
Metasediments_1	0 - 20	86	2.38
Metasediments_2	20 - 40	209	2.51
Metasediments_3	40 - 60	219	2.54
Metasediments_4	60+	1769	2.63
Manganese rich	all	23	2.63

14.18 Model Validation

Block classification from the Datamine model were added to the Vulcan grade, density and oxidation models. The combined Vulcan model was submitted to Alacer's mine engineering department for construction of the resource LG shell, Mineral Resource tabulation, and Mineral Reserve estimates.

Model validations include:

14.18.1 Visual

The estimated gold grades in the model were compared to the composite grades by visual inspection in plan views, N-S cross sections, and E-W cross sections. In general, the model and composite grades compared well.

14.18.2 Global Bias

The block model was checked for global bias by comparing the average gold, silver, copper, and sulfur grades (with no cut-off) from the model (OK/ID2 grades) with means from NN estimates for Indicated blocks. The NN estimator produces a theoretically unbiased (declustered) estimate of the average value when no cut-off grade is imposed and provides a good basis for checking the performance of different estimation methods. In general, an estimate is considered acceptable if the bias is at or below 5%. Table 14-16 shows the bias results on a global basis.

Table 14-16 Global Bias by Metal and Domain

	Element	OK Model	NN Model	Rel. Diff
TOTAL	Au Oxide	0.745	0.751	-0.7%
	Au Sulfide	1.045	1.005	3.9%
	Silver	3.399	3.341	1.8%
	Copper	0.029	0.029	0.9%
	Sulfur	2.995	3.018	-0.8%
Domain 1	Au Oxide	1.151	1.147	0.3%
	Au Sulfide	1.571	1.482	6.1%
	Silver	5.631	5.530	1.8%
	Copper	0.019	0.018	3.3%
	Sulfur	2.286	2.292	-0.2%
Domain 2	Au Oxide	0.514	0.513	0.1%
	Au Sulfide	0.961	0.930	3.4%
	Silver	2.887	2.819	2.4%
	Copper	0.034	0.034	1.0%
	Sulfur	3.287	3.314	-0.8%
Domain 3	Au Oxide	1.448	1.503	-3.6%
	Au Sulfide	1.802	1.751	2.9%
	Silver	2.174	2.190	-0.7%
	Copper	0.028	0.027	1.0%
	Sulfur	1.423	1.417	0.4%
Domain 4	Au Oxide	0.612	0.662	-7.4%
	Au Sulfide	0.428	0.421	1.6%
	Silver	3.573	3.809	-6.2%
	Copper	0.018	0.019	-4.0%
	Sulfur	2.296	2.350	-2.3%

Domain 1 = Manganese, Domain 2 = Main, Domain 3 = Marble, Domain 4 = West

Domain 4 shows the highest variance for oxide, however with the limited amount of drilling, ore material and scheduled mining in this area, the variances were not considered critical for operations.

14.18.3 Local Bias

Local trends in the grade estimates (swath checks, also called drift analysis) were performed by plotting the mean values from the NN estimate versus the kriged results for Indicated blocks in east-west, north-south and vertical directions. Swath plots by direction are shown in Figure 14-20, Figure 14-21, and Figure 14-22. The black dashed line is the grade of the nearest-neighbor model; the blue line is the grade of the kriged model.

The swath grade profile plots help assess the local mean grades and are used to validate grade trends in the model. Although the global comparisons agree well, the swath plots illustrate the existence of slight local differences between the NN and kriged model grades.

Figure 14-20 Gold Grade Trend Plot by Easting

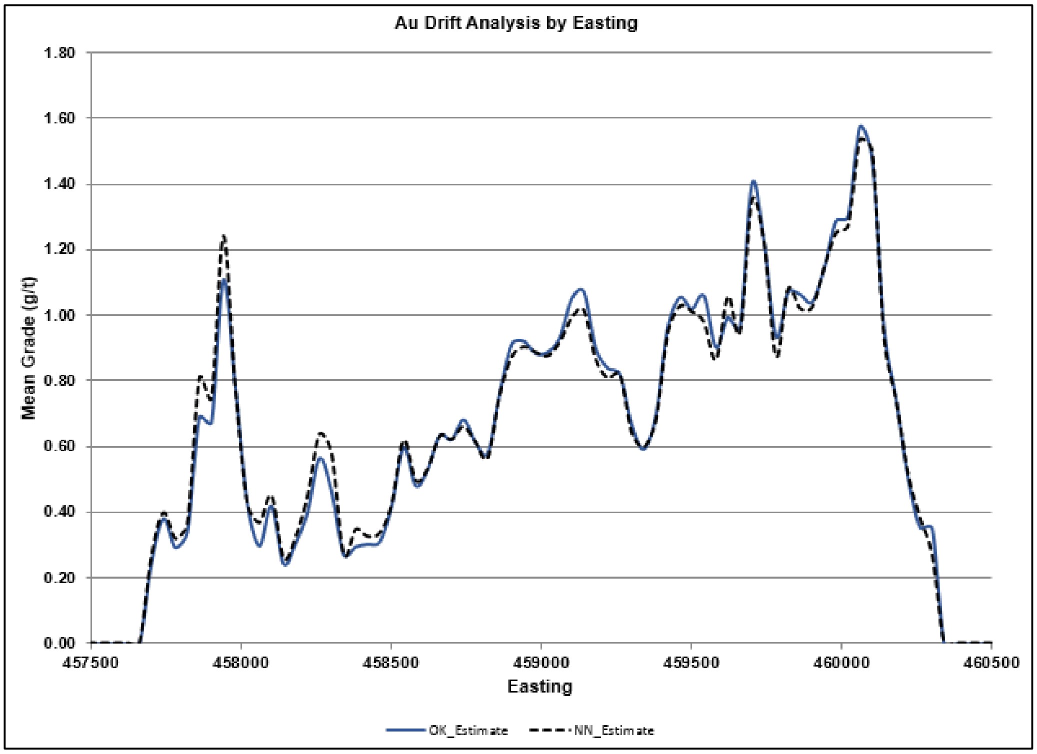


Figure courtesy of Alacer, 2016

Figure 14-21 Gold Grade Trend Plot by Northing

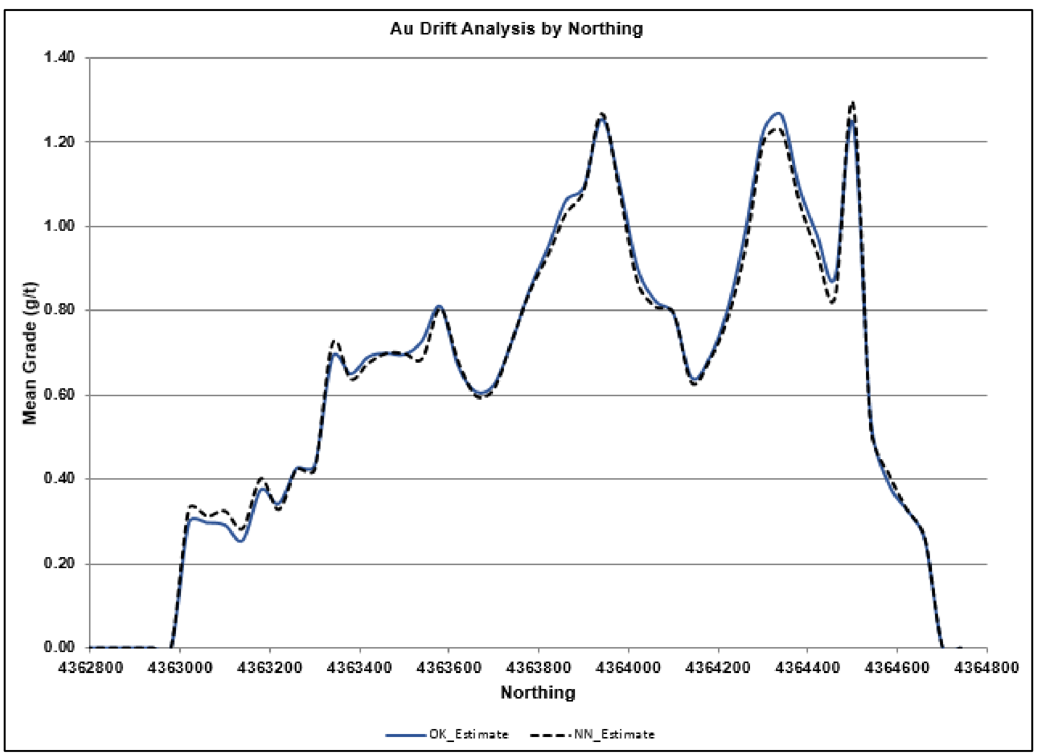


Figure courtesy of Alacer, 2016

Figure 14-22 Gold Grade Trend Plot by Elevation

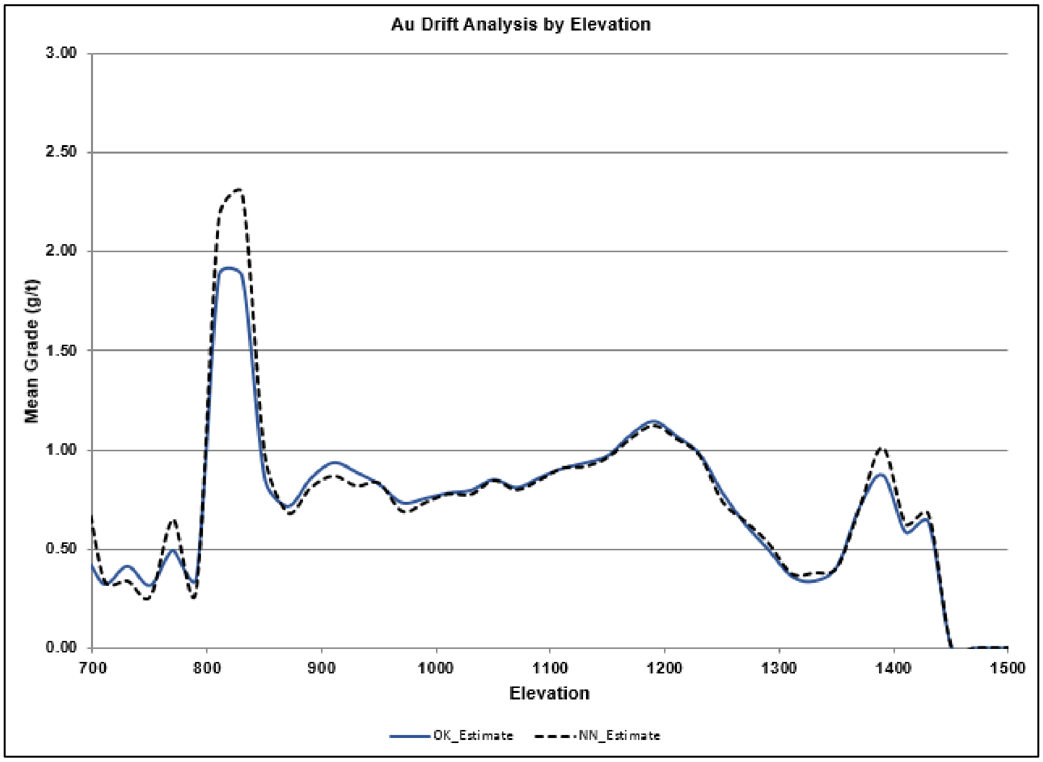


Figure courtesy of Alacer, 2016

14.18.4 Change of Support

Tonnes, grade and resulting gold ounces from the resource models were calibrated against mine production data. Comparisons between the resource model and ore control dig line model were performed by material type, mining area and time period. Indicator thresholds were modified for oxide material to minimize the variance between predicted resource model ounces and estimates using blast hole production data. The calibration step assumes historic mining practices will closely follow future mine operations.

The resource model calibration process involved:

1. Reporting resource model ore tonnes, gold grade and sulfur percent within each mining area. Mine production ore tonnes, gold grade and material type was tracked by mining area through grade control.
2. Tabulating material type (oxide or sulfide) above cutoff within each mining area. The estimated indicator threshold by block, ranging from 0 to 1, was included for all material.
3. Increasing the indicator threshold by individual mining domain to obtain similar contained ounces in the resource model when compared to the grade control / production data.

Mining domains were used in the resource model to allow the calibration of ore tonnes according to the mined pits. Mine domain boundaries were generated based on mine design and are not the same as resource model domains. The separate mine domains were used to calibrate estimated gold ounces with production information. Mining domains are shown in Figure 14-23.

Figure 14-23 Mining Domains used for Calibration of Gold Ounces

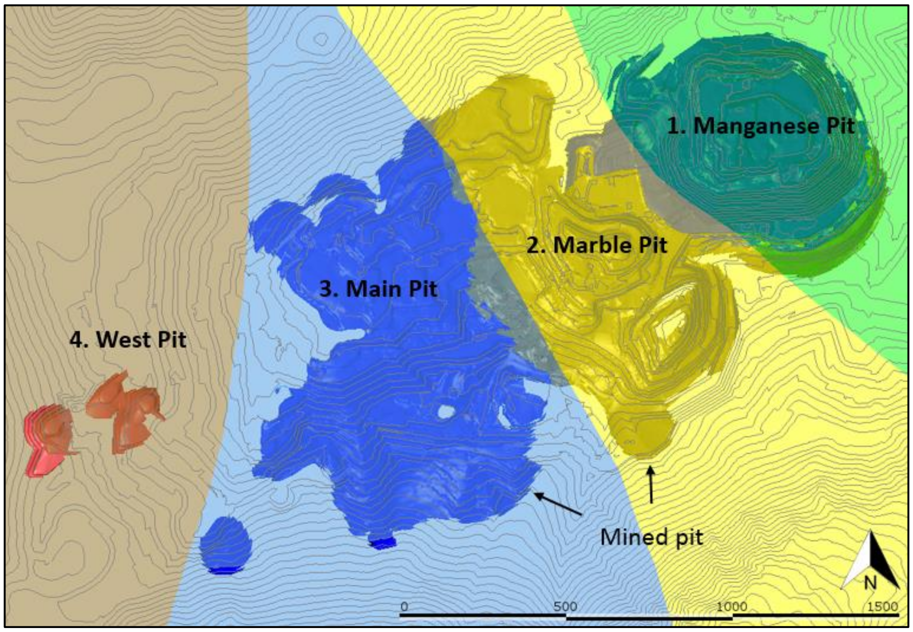


Table 14-17 Relative Difference of Gold between Ore Control and the Resource Model

Ore Type	Pit	Relative Difference % Tonnes	Relative Difference % Au Grade	Relative Difference % Au Cont. Ounces
Oxide	Marble	(0.1%)	1.4%	1.3%
	Manganese	6.5%	(4.7%)	1.5%
	Main	6.7%	(5.3%)	1.0%
	West	12.0%	(11.5%)	(0.9%)
	Oxide Total	**5.1%**	**(3.9%)**	**1.0%**
Sulfide	Marble	(21.1%)	14.3%	(9.8%)
	Manganese	(4.4%)	40.6%	34.4%
	Main	(2.4%)	1.0%	(1.4%)
	West	(2.5%)	(5.4%)	(7.7%)
	Sulfide Total	**(5.5%)**	**15.7%**	**9.4%**
Grand Total		**2.7%**	**1.4%**	**4.1%**

Positive percentages indicate ore control is higher than the resource model.

Table 14-17 shows the relative difference of the ore control data when compared to the resource model. Adjustment of the indicator allowed for an overall variance on contained Au ounces of 1% for oxide and 9.4% for sulfide.

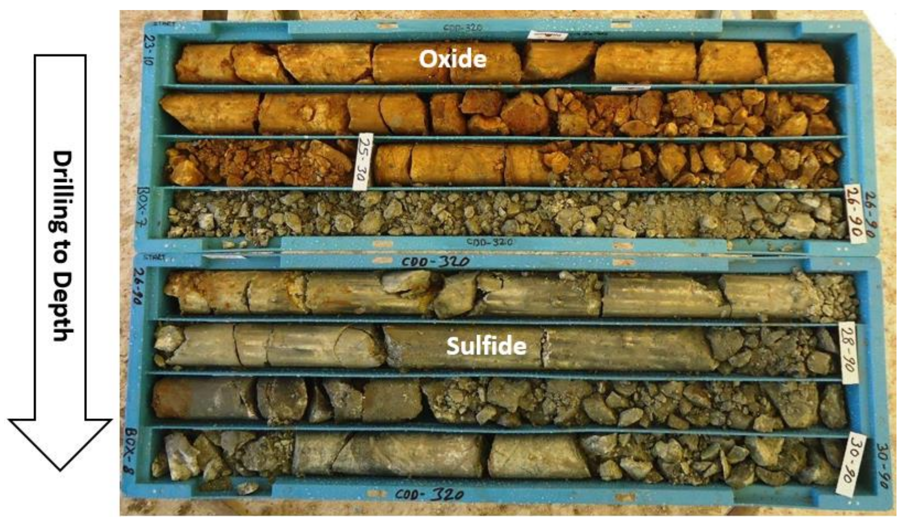
In the oxide domain, ore control has higher tonnage at slightly lower grade indicating the resource model is slightly under smoothed. The opposite appears to be true in the sulfide domain. The resource model may not be capturing the higher grade, short range structures seen in ore control with elevated gold grades.

14.19 Oxidation Model

The oxidation model reflects oxidation due to surficial weathering and/or oxidation resulting from the manganese alteration. "Oxide" or low sulfur material (S < 2%) can be processed by heap leaching while "sulfide" or high- sulfur material (S ≥ 2%) is stockpiled for the POX plant.

The low/high-sulfur criteria were then refined using the color codes and pyrite percentages recorded in the drill hole logs. Review of the drill logs showed relatively sharp color changes from orange–brown to grey–black (Figure 14-24), and a wireframe was constructed at this color change. The wireframe was further refined using the visually-estimated pyrite recorded in the drill hole database. Near-surface material is highly oxidized 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-gray color boundary within 5 m, but locally deviated vertically up to 10 m. The 5 m variance is considered within the accuracy of the data, as it reflects the composite sample length and the mining bench height.

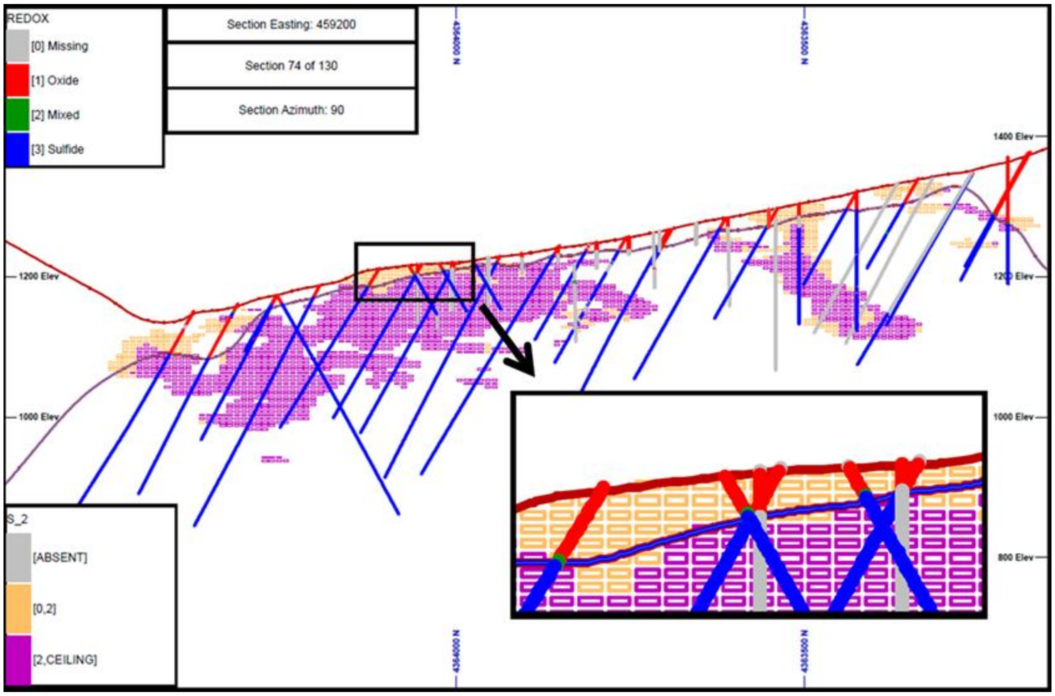
Figure 14-24 Drill Core Photograph Showing Color Change from Oxide to Sulfide



1. Figure courtesy Alacer, 2015
2. From drill hole CDD320 located in the southern portion of the Main pit. The image shows drill core from 23-30.9 meters down the hole with the change in color occurring at about 26 m.

The resulting oxide-sulfide boundary was compared to the sulfur model, Figure 14-25, and showed that the < 2%, ≥ 2% S domains matched the oxide–sulfide boundary reasonably well although there are local areas of material with sulfur grades < 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.

Figure 14-25 Model of the Oxidation Boundaries



Top left legend oxidation codes for drill holes

Bottom left legend for sulfur <2% and ≥2% for block model

Figure prepared by Amec Foster Wheeler, 2015

Open pit mining in the Main pit has reached the oxide/sulfide boundary. Blast hole data that contains both gold fire assays (AuFA) and gold cyanide leach assays (AuCN) show that the gold recovery significantly decreases below this boundary. This implies there is low sulfur material below the oxide/sulfide boundary that has not oxidized and hence lower recoveries are obtained by cyanide leaching. As a result, the oxide/sulfide boundary is used in the Main pit to delineate material types. In the Manganese and Marble pits, however, the estimated sulfur content is used to delineate material.

In the eastern portion of the Çöpler Mine, 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. Figure 14-26 is a cross-section showing the variance in depths of the oxidation profiles.

Figure 14-26 Oblique Section of the Oxidation Boundary, looking northwest

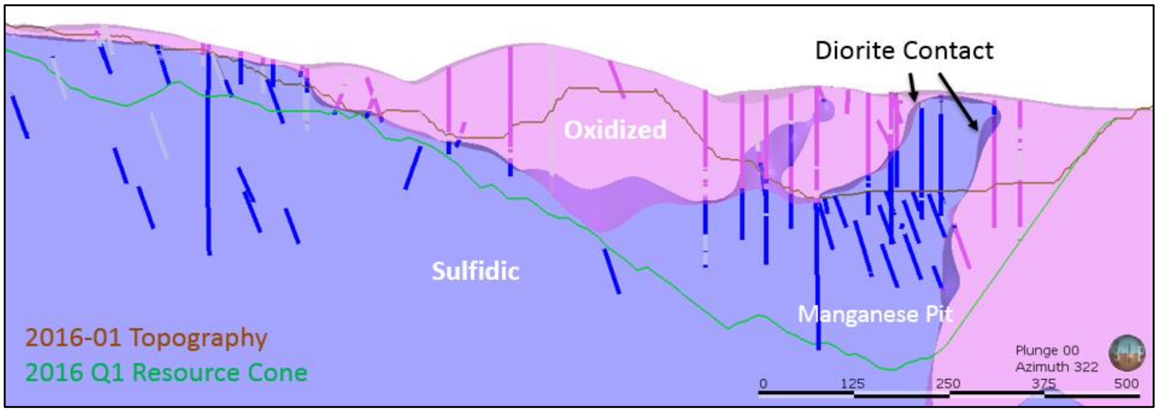


Figure courtesy Alacer, 2016

Note: In the Manganese pit, the oxidation profile model generally follows the diorite contact to depth.

The purple color shows the undulation of the geologic contact over the 25 m section thickness.

14.20 Assessment of Reasonable Prospects of Eventual Economic Extraction

Mineral Resources were shown to meet reasonable prospects for eventual economic extraction criteria by reporting only material that was contained within a LG conceptual pit shell using metal prices of $1400/oz for gold and $21.00/oz for silver with the parameters summarized in Table 14-18. These parameters are the same parameters as those used to define the Mineral Reserve pit with the exception of the metal prices. The sulfide material processing cost is the calculated cost for processing the Mineral Resource through a 1.9 to 2.2 Mt/a mill.

Table 14-18 Summary of Key Parameters Used in Lerchs-Grossmann Conceptual Pit Shell

Description	Element	Minimum	Maximum
Heap Leach Recovery	Au	62.3%	78.4%
	Ag	24.6%	37.8%
	Cu	3.5%	15.8%
POX Recovery	Au	94%*	94.0%
	Ag	3.0%	3.0%
	Cu	~	~
Mining Cost per tonne mined	---	$1.90	$1.90
Process Costs Heap Leach per tonne	---	$5.24	$9.87
Process Costs POX per tonne	---	$33.40	$33.40
Site Support per tonne processed	---	$3.50	$3.50
Internal Au Cutoff - Heap Leach	---	0.25	0.40
Royalty	---	2%	2%
Inter Ramp Slope RQD≤15	---	25 degrees	52.5 degrees
Inter Ramp Slope RQD>15	---	40 degrees	52.5 degrees

** Au recovery is the average percent over the LOM.*

14.21 Mineral Resource Tabulation

Mineral Resources are reported inclusive of Mineral Reserves, and have been tabulated by resource classification and material type in Table 14-19. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The overall tonnage and grade estimate has decreased for oxide material from the previously-reported estimate in 2015. Changes include the removal of the transitional zone in the Main pit, increase in the sulfide indicator threshold from 1.0 to 1.5 g/t to track the sulfide cut-off grade, and use of a soft boundary across the high/low indicator domain. Other factors that contributed include mining depletion and changes to the resource LG shell parameters constraining the resource estimate. New drilling and updated search orientations had minimal contributions.

Table 14-19 Mineral Resource Table by Classification and Oxide State

Mineral Resource Statement for the Çöpler Deposit (As of December 31st, 2015)							
Gold Cut-off Grade (g/t)	Material Type	Resource Category Material	Tonnes (x1000)	Au (g/t)	Ag (g/t)	Cu (%)	Contained Au (oz x 1000)
Variable	*Oxide*	Measured	-	-	-	-	-
		Indicated	24,959	1.04	3.19	0.13	836
		Stockpile - Indicated	148	0.87	-	-	4
		Measured + Indicated	**25,106**	**1.04**	**3.17**	**0.13**	**840**
		Inferred	20,863	0.83	6.40	0.13	557
1.0	*Sulfide*	Measured	-	-	-	-	-
		Indicated	70,151	2.12	5.94	-	4,771
		Stockpile - Indicated	5,102	3.67	-	-	602
		Measured + Indicated	**75,253**	**2.22**	**5.53**	-	**5,373**
		Inferred	12,739	1.99	12.00	-	814
Variable	*Stockpiles*	Indicated	5,250	3.59	-	-	606
Variable	***Total***	Measured	-	-	-	-	-
		Indicated	100,359	1.93	4.95	0.03	6,213
		Measured + Indicated	**100,359**	**1.93**	**4.94**	**0.03**	**6,213**
		Inferred	**33,602**	**1.27**	**8.52**	**0.08**	**1,371**

1. Mineral Resources have an effective date of December 31, 2015. Gordon Seibel and Harry M. Parker, both SME Registered Members, are the Qualified Persons responsible for the Mineral Resource estimates. The Mineral Resource model was prepared by Messrs. Gordon Seibel and Loren Ligocki

4. Mineral Resources are reported inclusive of Mineral Reserves; Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability

5. Mineral Resources are shown on a 100% basis, of which Alacer owns 80%

6. In the Main pit, oxide is defined as material above the interpreted oxide surface. Material in the Main pit beneath the oxide surface is classified as sulfide. In the Manganese and Marble pit, material type is determined by sulfur content: material with estimated S grades < 2% are oxide, and material estimated with S grades ≥ 2% are sulfide.

7. The resources meet the reasonable prospects for eventual economic extraction by reporting only material within a Lerchs-Grossmann conceptual pit shell. The following parameters were used: assumed throughput rate of 1.9 to 2.2 Mt/a; variable metallurgical recoveries in oxide including 62.3–78.4% for Au, 24.6–37.8% for Ag, 3.5–15.8% for Cu; metallurgical recoveries in sulfide including 94% for Au, 3% for Ag; mining cost of $1.90/t; process cost of $5.24–$9.87/t leached and $33.40/t

through the POX; general and administrative charges of $3.50/t; 2% royalty payable; inter-ramp slope angles that vary from 25–52.5º. Metal price assumptions were $1400/oz for gold, $21.00/oz for silver, no copper credit.

8. Reported Mineral Resources contain no allowances for unplanned dilution, or mining recovery.
9. Tonnage and grade measurements are in metric units. Contained gold is reported in troy ounces
10. Tonnages are rounded to the nearest thousand tonnes; grades are rounded to two decimal places
11. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content

In the Main pit, the oxide / sulfide boundary follows surficial oxidation. This boundary is abrupt and therefore a transition zone was not used. Low-sulfur material below the oxide / sulfide boundary in the Main pit has reduced gold leach recovery, often not profitable on the leach pad. However, in the Manganese pit and Marble pit, the oxide boundary closely follows the diorite intrusion which is near vertical. Gold recovery follows total sulfur content rather than the oxide surface so the 2% sulfur cut was used to determine material type in these areas.

The majority of the inferred oxide material is located at depth within the Manganese pit. An increase in gold price will capture some of this material, however the heap leach facility location restricts the expansion of the LG cone to the east and therefore to depth.

Figure 14-27 and Figure 14-28 respectively show the distribution of the tonnage and grade within the LG shell for oxide and sulfide. The lower cut-offs in the grade/tonnage curves represents the lower limit for the mineralization to meet reasonable prospects of eventual economic extraction.

Figure 14-27 Distribution Curve for Oxide Material Classified as Indicated Mineral Resources within the 2016 Lerchs-Grossmann Conceptual Pit Shell

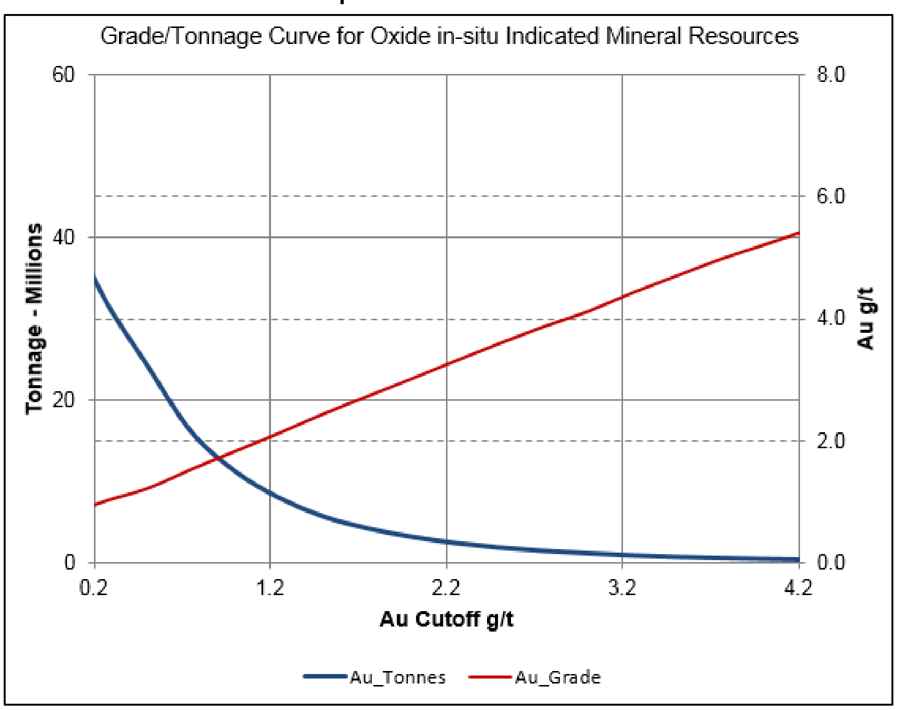


Figure courtesy Alacer, 2016

Figure 14-28 Distribution Curve for Sulfide Material Classified as Indicated Mineral Resources within the 2016 Lerchs-Grossmann Conceptual Pit Shell

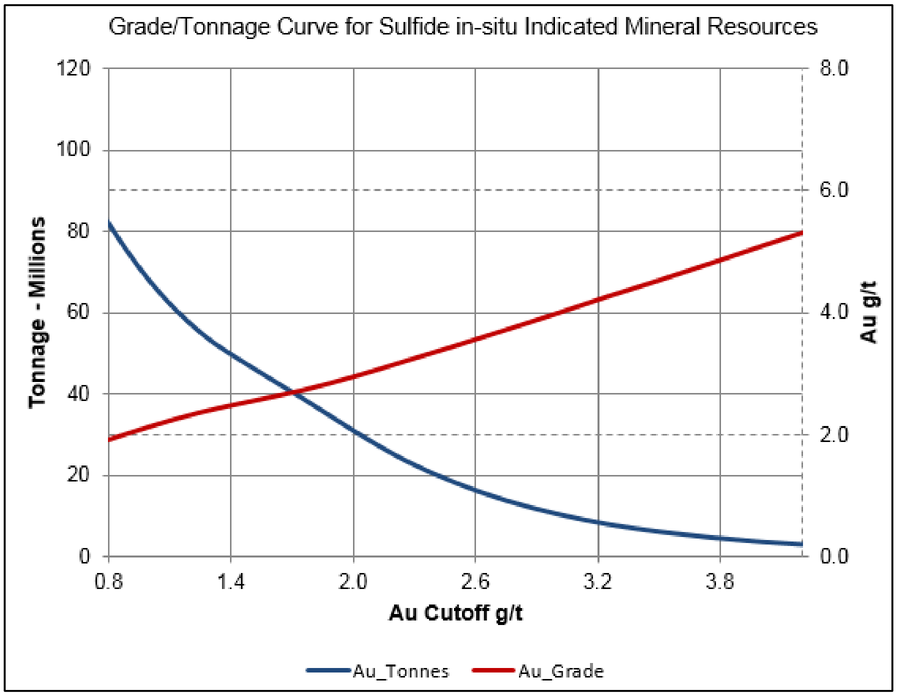


Figure courtesy Alacer, 2016

14.22 Risks and Opportunities

Risks and opportunities that may affect the Mineral Resource statement are as follows:

- Çöpler is a geologically complex deposit with multiple metals that must be tracked along with oxidation type and lithologies. As studies are completed the information is used to improve the resource estimation causing changes to overall oxide and sulfide tonnes and grade.
- Since the gold mineralization locally follows the lithological contacts, applying an overall trend to the mineralization may not reflect the local variability of the lithological contacts. Implementing a search ellipse that follows these contacts (dynamic anisotropy) may provide better local estimates.
- Additional studies should be performed including quantifying what percent of the sulfur is derived from sulfate minerals.
- Reported tonnages and grades depend on the cut-off grade that will vary with changing metal prices, costs, metallurgical recoveries, exchange rate assumptions and the sulfur threshold used to delineate oxide and sulfide.
- Changes to the LG shell used to constrain the estimate will affect the reported tonnages and grades.
- The model was reconciled against past production using the blast hole database, which was not audited.

- Visual comparison between drill holes and the blast holes have shown that small, narrow mineralized zones identified by the blast holes are not always delineated in the exploration drill hole data set. In addition, a blast hole to drill hole comparison study showed that Au grade in the Manganese pit is higher in blast holes than in exploration holes.
- All risks associated with the data quality issues reported in Section 12.0 will be risks in the resource model.
- The current resource model assumes that low-grade oxide material below the redox boundary in the Main pit will be sent to waste. There is an upside opportunity if this material can be sent to the heap leach pad rather than being treated as waste.
- There is upside if Inferred Mineral Resources can be converted to Indicated with further drilling. The value that might be added should be estimated and might be persuasive in deciding to embark on an infill drilling program.

There are no other known factors or issues that materially affect the estimate other than normal risks faced by mining projects in terms of environmental, permitting, taxation, socio-economic, marketing and political factors.

15.0 MINERAL RESERVE ESTIMATES

Çöpler Mineral Reserves Estimate

The Mineral Reserves for the Çöpler gold deposit have been estimated by Alacer as summarized in Table 15-1.

Mineral Reserves are quoted as of December 31st, 2015. Mineral Reserves to be processed through the heap leach use a calculated gold cut-off excluding mining costs, while sulfide Mineral Reserves use a gold cut-off of 1.50 g/t.

Table 15-1 Mineral Reserves for the Çöpler Gold Deposit

Mineral Reserves for the Çöpler Mining area deposit (As of December 31st, 2015)						
Reserve Category Material	**Tonnes (x1000)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Contained Au Ounces**	**Recoverable Au Ounces**
Proven - Oxide In-Situ	-	-	-	-	-	-
Probable - Oxide In-Situ	17,836	1.13	3.53	0.13	650,000	494,000
Probable - Oxide Stockpile	148	0.87	-	-	4,000	3,000
Total - Oxide	**17,984**	**1.13**	**3.50**	**0.13**	**654,000**	**497,000**
Proven - Sulfide In-Situ	-	-	-	-	-	-
Probable - Sulfide In-Situ	34,879	2.63	7.23	-	2,944,000	2,829,000
Probable - Sulfide Stockpile	5,102	3.67	-	-	602,000	579,000
Total - Sulfide	**39,982**	**2.76**	**6.30**	-	**3,546,000**	**3,408,000**
Proven - Oxide + Sulfide + Stockpile	-	-	-	-	-	-
Probable - Oxide + Sulfide +Stockpile	*57,965*	*2.25*	*5.44*	*0.04*	4,200,000	3,905,000
Total - Oxide + Sulfide	***57,965***	***2.25***	***5.44***	***0.04***	***4,200,000***	***3,905,000***

1. Mineral Reserves are not diluted.
2. Full mine recovery assumed.
3. Average LOM Heap Leach Au recovery for all rock types is estimated at 76.0% and for Pressure Oxidation (POX), 96.1%.
4. Numbers may not add up due to rounding.
5. The Mineral Reserves were developed based on mine planning work completed in March 2016 and estimated based on end of December, 2015 topography surface.
6. A calculated gold internal cut-off grade was applied to Oxide Heap Leach Mineral Reserves using the equation: Xc = Po / (r * (V-R)) where Xc = Cut-off Grade (gpt), Po = Processing Cost of Ore (USD/tonne of ore), r = Recovery, V = Gold Sell Price (USD/gram), Refining Costs (USD/gram). A gold cut-off grade of 1.50 g/t was used for Sulfide Pressure Oxidation.
7. Mineral Reserves are based on US$ 1,250/Oz Au Gold Price.
8. The Mineral Reserves were estimated by Stephen Statham, PE (Colorado License #PE.0048263, SME 4140907RM) of Alacer, a qualified person under NI 43-101 and JORC guidelines.

The 2014 CIM Definition Standards define Proven Mineral Reserves as “the economically mineable part of a Measured Mineral Resource” and Probable Mineral Reserves as “the economically mineable part of an Indicated Mineral Resource, and in some circumstances a Measured Mineral Resource.” These criteria have been applied to the Mineral Reserves estimate reported in Table 15-1.

The Mineral Reserves disclosure presented in Table 15-1 were estimated by Stephen Statham, PE, RM SME, who is a full-time employee of Alacer.

The mine plan developed in this report is based on Probable Mineral Reserves only (no Proven Mineral Reserves exist as no Measured Mineral Resources were estimated). There may be opportunity to upgrade some of the Inferred Mineral Resources to higher confidence categories with additional infill drilling and supporting studies.

15.1 Mine Production Schedule

For the Çöpler mine production schedule, the MineSight Schedule Optimizer (MSSO) tool was used to schedule the extraction of ore from the mine, with the objective of maximizing the NPV of the project within the constraints of production tonnages, metallurgical blend requirements, and mining operational efficiencies. The mine schedule is comprised of detailed sequencing of 16 pit phase designs as described in Section 16.2 of this document. The phase designs are scheduled in a way that ensures production targets are operationally achievable.

15.1.1 Schedule Periods

The first scheduling period is started as of January 1, 2016 using the end of year December 31, 2015 surveyed topography for the mine. The scheduling interval is on a monthly basis through the year 2016, quarterly basis from year 2017 through year 2020, and annual basis for the remainder of the mine life.

15.1.2 Schedule Throughput Rates

All throughput rates are reported inclusive of all availability and utilization factors on a full 365/366 day per year calendar. Total mine production is limited to an annual average of 100,000 tpd with throughputs as low as 95,000 tpd during the wet winter months, and as high as 105,000 tpd during the dry summer months. These throughput objectives are supported by current mining rates. Additionally, mining rates are limited based on vertical advance and bench configuration in order to ensure that the schedule is achievable.

Oxide ore heap leach production rates are limited to a maximum of 20,000 tpd. This throughput objective is supported by past production rates which have averaged between 18,000 and 22,000 tpd.

Sulfide ore POX production throughputs are limited dependent on ore tonnage and sulfide sulfur tonnage. Limitations on sulfide sulfur tonnage exist due to the consumption of oxygen by sulfide sulfur in the POX circuit. The process facilities are limited by the amount of oxygen that can be provided to the POX process. Both ore tonnage and sulfide sulfur tonnage limitations are subjected to a ramp-up curve starting in 2018 at 2,242 tpd ore and 116 tpd sulfide sulfur. By the third quarter 2019, production reaches nameplate capacity at 5,152 tpd ore and 248 tpd sulfide sulfur. Through 2020 operational efficiencies are realized in order to improve throughput. Beginning in 2021 debottlenecking optimizations are incorporated to bring the maximum production throughput up to 6,027 tpd ore and 261 tpd sulfide sulfur.

15.1.3 Stockpiling

Oxide ore stockpiling is not considered during the long-range mine planning. However, small stockpiles do exist directly adjacent to the crusher and are accounted for in Mineral Reserve estimates and mine scheduling for the first period scheduled.

Prior to the commissioning of the sulfide mill all sulfide ore is shipped to one of three sulfide ore stockpiles. The three sulfide ore stockpiles will be used for low-grade (1.50 – 3.2 g/t Au), medium-grade (3.2 – 4.0 g/t Au), and high-grade (4.0 g/t Au and higher) sulfide ore. The mill is scheduled to be in production through 2037 when it will exhaust the remainder of the low-grade sulfide ore contained in stockpile. During operation, a majority of high-grade sulfide ore mined and some medium-grade sulfide ore mined will be placed directly on the run-of-mine (ROM) pad adjacent to the crusher where it will be rehandled by a loader and placed into the crushing system. Stockpiles are scheduled on the basis that the last material placed in stockpile will be the first material rehandled out of stockpile; known as last-in, first-out (LIFO).

15.1.4 Recovery Assumptions

Gold is considered to be the only materially important saleable product produced by the Çöpler Mine. However, the mine plan does consider recoverable silver and copper value when optimizing production sequencing and throughput.

Oxide ore heap leaching produces gold, silver, and copper from the stacked ore. Recoveries are variable by rock type and by pit region. Oxide ore recoveries used for Mineral Reserve reporting are shown in Table 15-2.

Table 15-2 Oxide Ore Processing Recoveries

Pit Region	Rock Type	Processing Recovery		
		Au	Ag	Cu
Manganese	Limestone/Marble	78.4%	27.3%	3.5%
Marble	Limestone/Marble	75.7%	34.0%	3.5%
Main	Limestone/Marble	68.6%	24.6%	3.5%
Main East	Limestone/Marble	78.4%	27.3%	3.5%
Main West	Limestone/Marble	75.7%	34.0%	3.5%
West	Limestone/Marble	75.7%	34.0%	3.5%
Manganese	Metasediments	66.8%	32.5%	13.8%
Marble	Metasediments	66.8%	32.5%	13.8%
Main	Metasediments	66.8%	32.5%	13.8%
Main East	Metasediments	66.8%	32.5%	13.8%
Main West	Metasediments	66.8%	32.5%	13.8%
West	Metasediments	66.8%	32.5%	13.8%
Manganese	Diorite	71.2%	37.8%	15.8%
Marble	Diorite	62.3%	32.0%	15.8%
Main	Diorite	71.2%	37.8%	15.8%
Main East	Diorite	71.2%	37.8%	15.8%
Main West	Diorite	62.3%	32.0%	15.8%
West	Diorite	62.3%	32.0%	15.8%

The POX mill will produce gold and silver from the processed ore. The LOM average gold recovery is 96.1% and silver recovery is 13.5%. Gold recovery is calculated based on the gold head grade being fed into the mill. The gold recovery equation is shown in Figure 15-1.

Figure 15-1 Gold Recovery Equation for Reserve Estimate (Where HG_{Au} is gold head grade in gpt)

$$Au\ Recovery\ (\%) = 97.94 \times \left(1 - e^{-1.4 \times (HG_{Au} + 1.4)}\right)$$

Additionally, a recovery discount is applied throughout the mine life in order to account for operational inefficiencies. During the first two quarters of operation, a 3.3% recovery reduction is applied. During quarters three through six, a 2.3% recovery reduction is applied. And during the remainder of the operation, a 1.3% recovery reduction is applied.

Silver recovery for sulfide ore is estimated to be 11.6% during the first two quarters of operation, 12.6% during quarters three through six, and 13.6% during the remainder of the operation.

15.1.5 Cut-off Grade

Cut-off grades have been based exclusively on gold grade, as the additional silver and copper that is extracted at Çöpler is considered to be non-material and has limited effect on the profitability of the mine.

Because the oxide ore is nearing depletion at the Çöpler Mine, there is little value in mining oxide ore at an elevated cut-off grade. In the past it made economic sense to mine to an elevated cut-off grade in order to maximize the grade of material being processed through the heap leach facility on an annual basis. However, because oxide ore production does not meet processing throughput limits, there is no benefit to routing ore using an elevated cut-off grade. As a result, an internal cut-off grade (excluding mining cost) is applied using the equation shown in

Figure 15-2.

Figure 15-2 Internal Cut-off Grade Equation

$$Cutoff\ Grade\ (Au\ gpt) = \frac{Processing\ Cost\ (\frac{USD}{tonne\ ore})}{\left(Recovery \times \left(Au\ Sell\ Price\ \left(\frac{USD}{gram}\right) - Refining\ Cost\ (\frac{USD}{gram})\right)\right)}$$

The above equation results in a cut-off grade of 0.30 g/t for limestone oxide ore and 0.45 g/t for metasediment and diorite oxide ore.

Sulfide ore NPV benefits by the use of an elevated cut-off grade. An elevated cut-off grade is used over three separate grade bins; low-grade (1.50 – 3.2 g/t Au), medium-grade (3.2 – 4.0 g/t Au), and high-grade (4.0 g/t Au and higher) sulfide ore.

The lower limit for sulfide ore grade is first optimized using various LG pit shell optimizations which are constrained by cut-off grades ranging from break-even to 2.0 g/t. The NPV of each of these pit shells is compared and the cut-off grade which maximizes NPV is considered for use as the lower boundary cut-off. In this case, cut-offs ranging from 1.5 g/t to 1.7 g/t provided the best NPV value to the Çöpler deposit. From this point, pit designs are generated and the cut-off grade is again re-evaluated in regard to ultimate pit reserves, total processing tonnage limitations, cash flow objectives, and NPV. After this evaluation, a lower cut-off grade of 1.5 g/t was selected. The medium-grade cut-off was selected on the basis that it is desirable to maximize the feed grade through year 2023. Therefore, the grade was set so that enough tonnes would be available to fulfil production requirements through 2023. The high-grade cut-off was selected on the basis that maximum NPV could be realized by processing approximately three years' worth of material at the highest grade possible.

15.1.6 Schedule Results

The Çöpler Mine currently operates a heap leaching facility into which oxide ore is mined, crushed, and stacked onto an existing heap leach pad. Oxide ore stacking will continue through 2023, when the mine is scheduled to cease mining operations. Following 2017, oxide ore production will decrease as the oxide resource is depleted. Oxide ore production beyond 2018 will be produced as a byproduct of sulfide ore extraction.

A sulfide Mineral Resource also exists at the Çöpler Mine. The sulfide ore is currently being mined as a byproduct of oxide ore extraction. This sulfide ore is stockpiled according to grade in anticipation of future processing. As of December 31, 2016, 5.1 Mt tonnes of sulfide ore, averaging 3.67 g/t Au, has been stockpiled. Sulfide ore production is planned to begin in the third quarter of 2018.

A map showing the progression of the mining schedule is shown in Figure 15-3 below. A tabular representation of the mine production schedule is shown in Table 15-3 below. End of year mining progress maps are shown in Figure 15-4 through Figure 15-12 for years 2015 through 2023.

Figure 15-3 Annual Mining Schedule Map

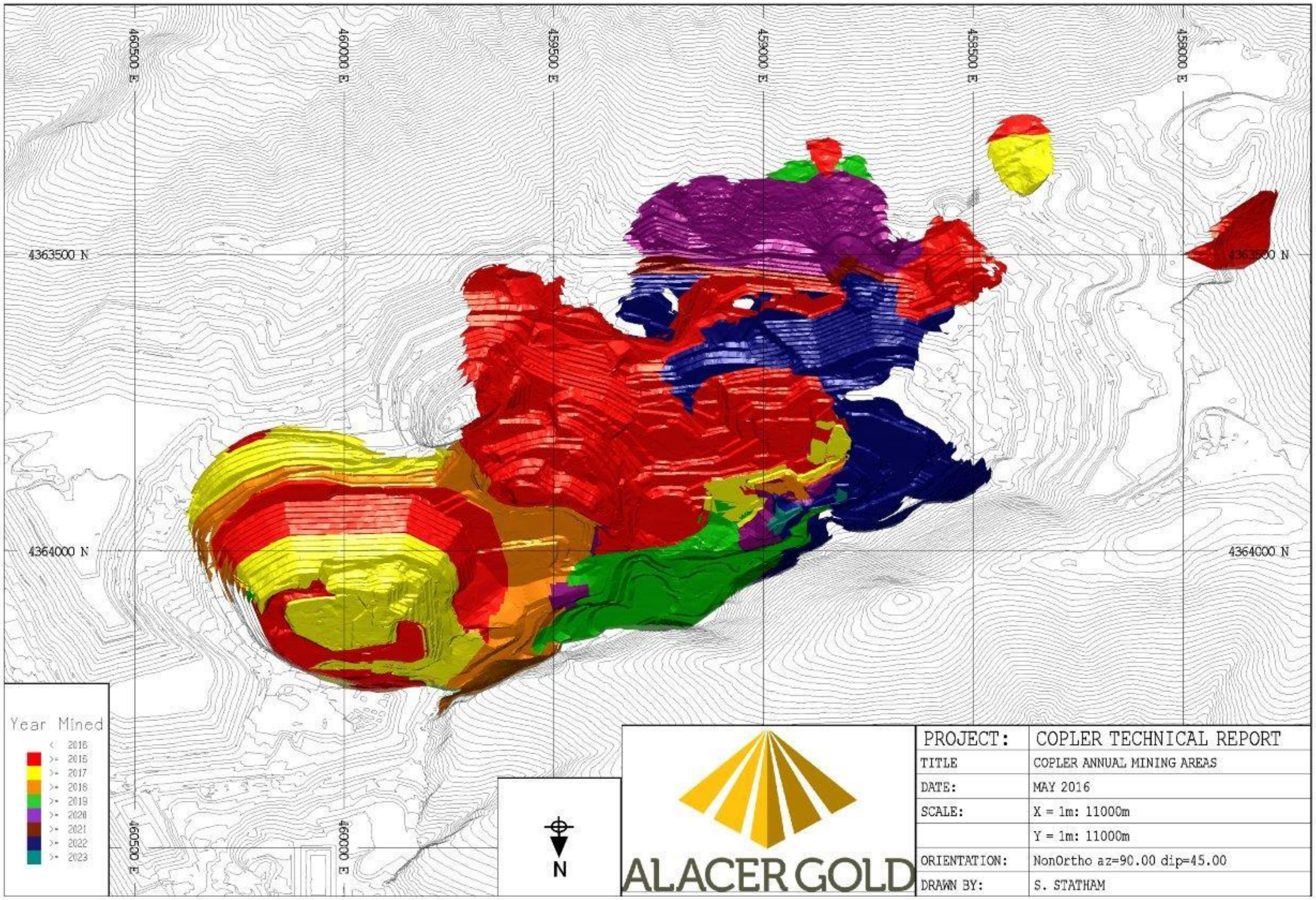


Table 15-3 Çöpler Mine Production Schedule

Period	Oxide Ore		Sulfide Ore		Totals	
	Ore Processed (Tonnes x1000)	Ore Grade (Au gpt)	Ore Processed (Tonnes x1000)	Ore Grade (Au gpt)	Waste Mined (Tonnes x1000)	Total Mined (Tonnes x1000)
2016	4,378	1.38	-	-	29,318	36,838
2017	6,269	0.99	-	-	25,704	34,828
2018	1,255	1.15	636	5.04	31,908	34,950
2019	3,493	0.92	1,862	4.76	29,545	36,244
2020	1,183	1.30	1,911	4.60	31,395	36,402
2021	1,308	1.34	2,200	3.71	29,083	36,005
2022	96	1.31	2,200	3.91	29,104	34,675
2023	-	-	2,200	4.23	18,699	27,677
2024	-	-	2,206	2.35	-	-
2025	-	-	2,200	2.26	-	-
2026	-	-	2,200	2.30	-	-
2027	-	-	2,200	2.20	-	-
2028	-	-	2,206	2.18	-	-
2029	-	-	2,200	2.10	-	-
2030	-	-	2,047	2.20	-	-
2031	-	-	1,969	1.99	-	-
2032	-	-	2,176	2.07	-	-
2033	-	-	1,911	2.19	-	-
2034	-	-	2,166	2.26	-	-
2035	-	-	2,200	2.23	-	-
2036	-	-	2,206	2.19	-	-
2037	-	-	1,086	2.00	-	-
Total	**17,984**	**1.13**	**39,982**	**2.76**	**224,757**	**277,620**

Figure 15-4 End of Year 2015 Mining Progress

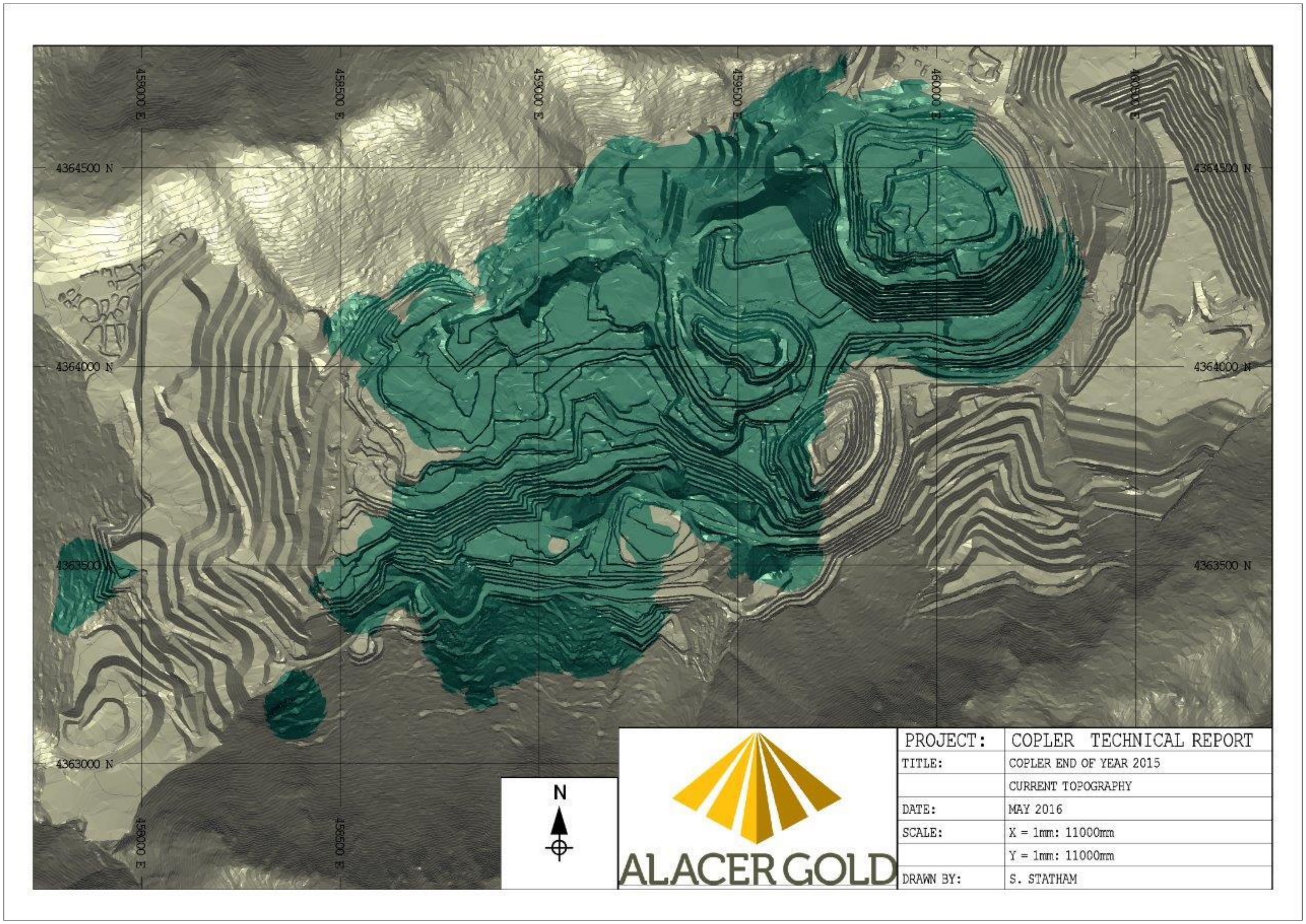


Figure 15-5 End of Year 2016 Mining Progress

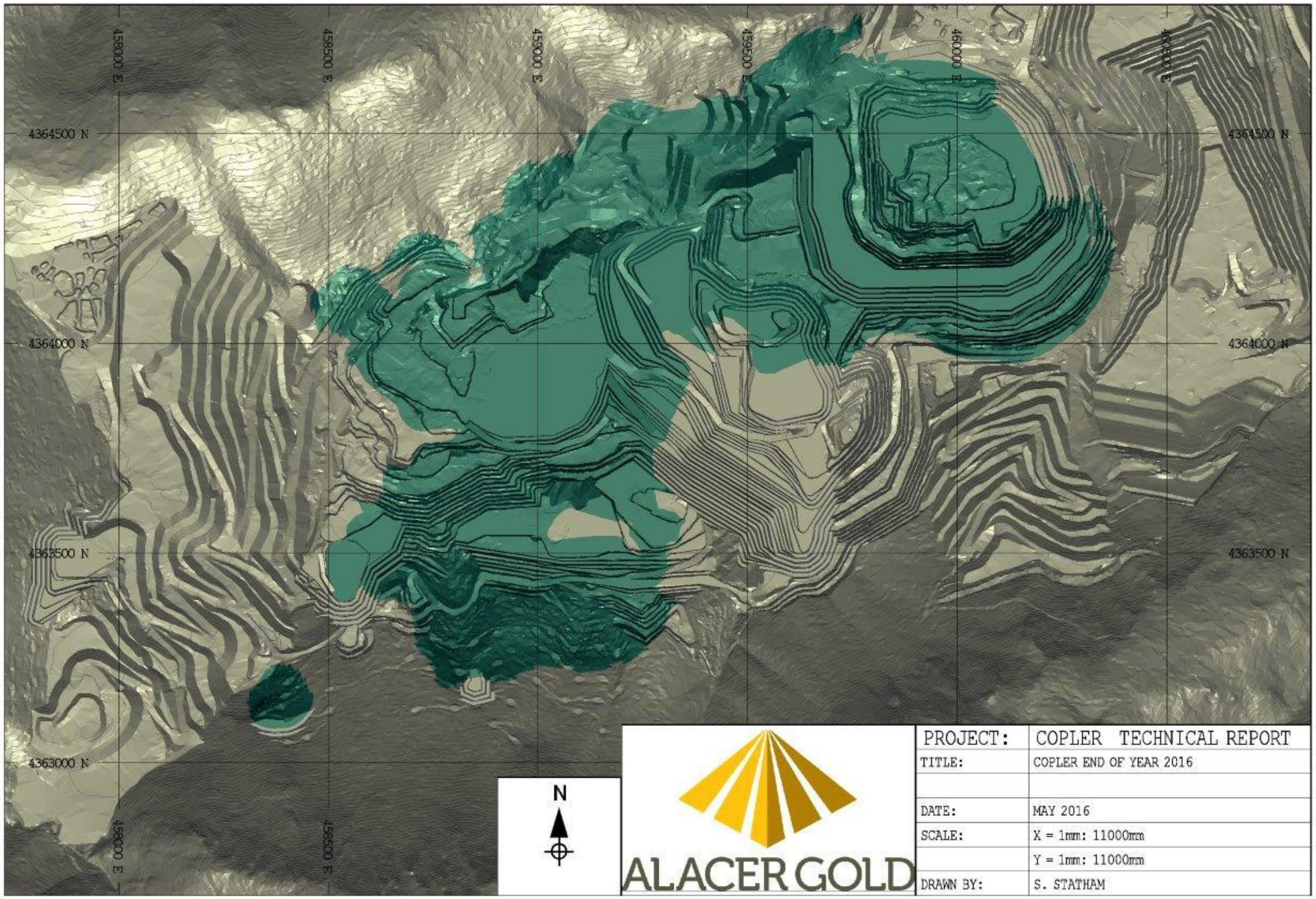


Figure 15-6 End of Year 2017 Mining Progress

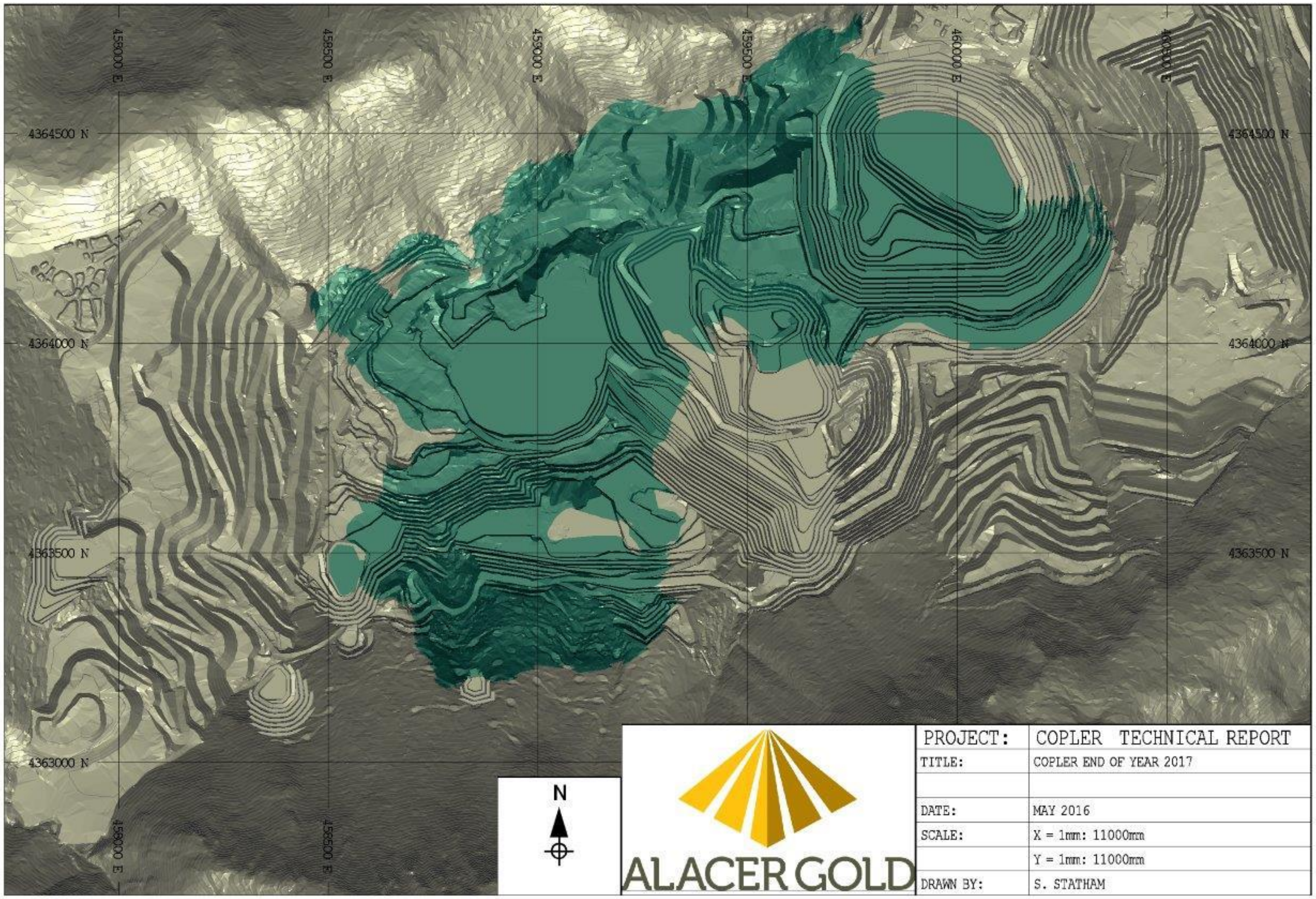


Figure 15-7 End of Year 2018 Mining Progress

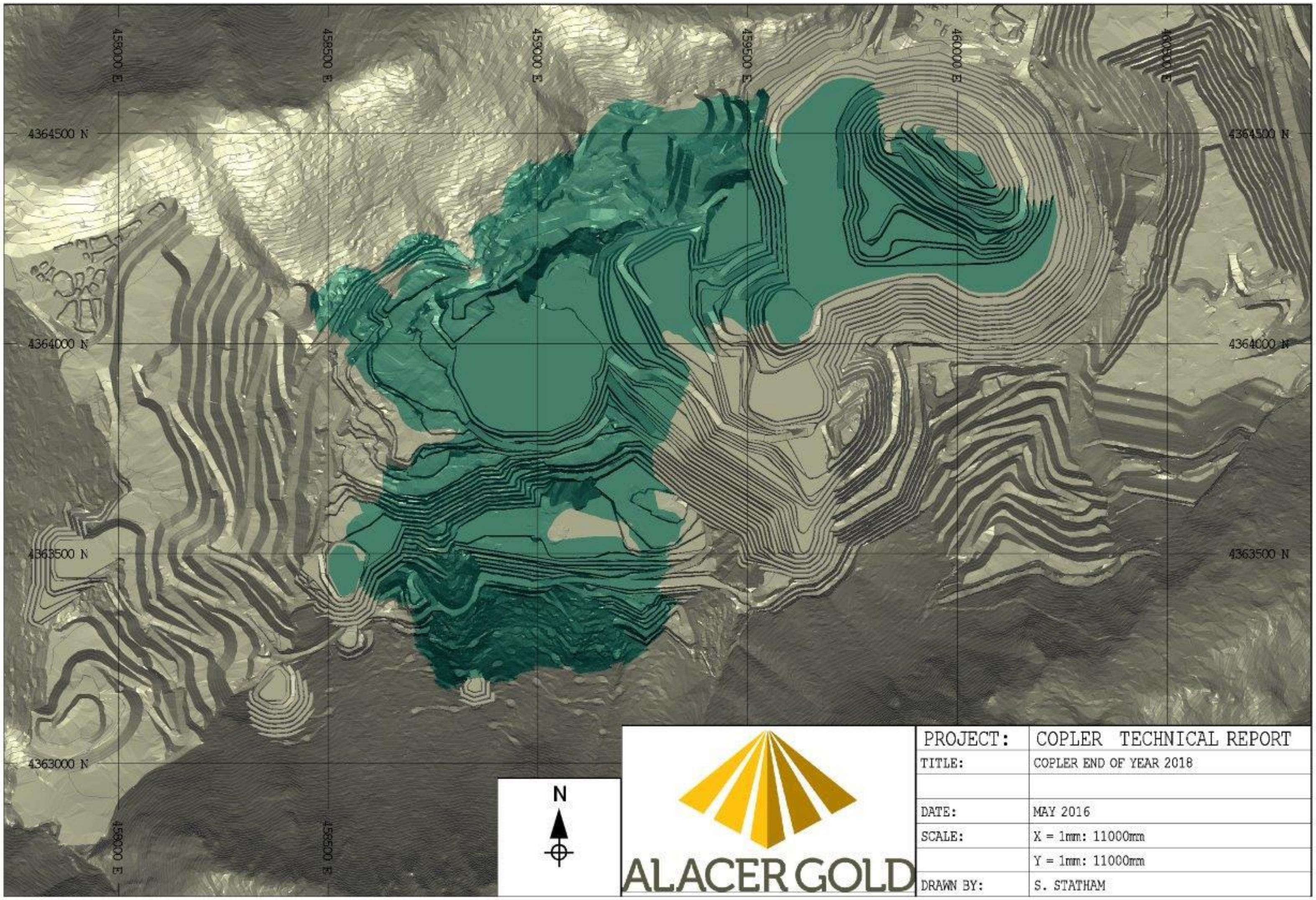


Figure 15-8 End of Year 2019 Mining Progress

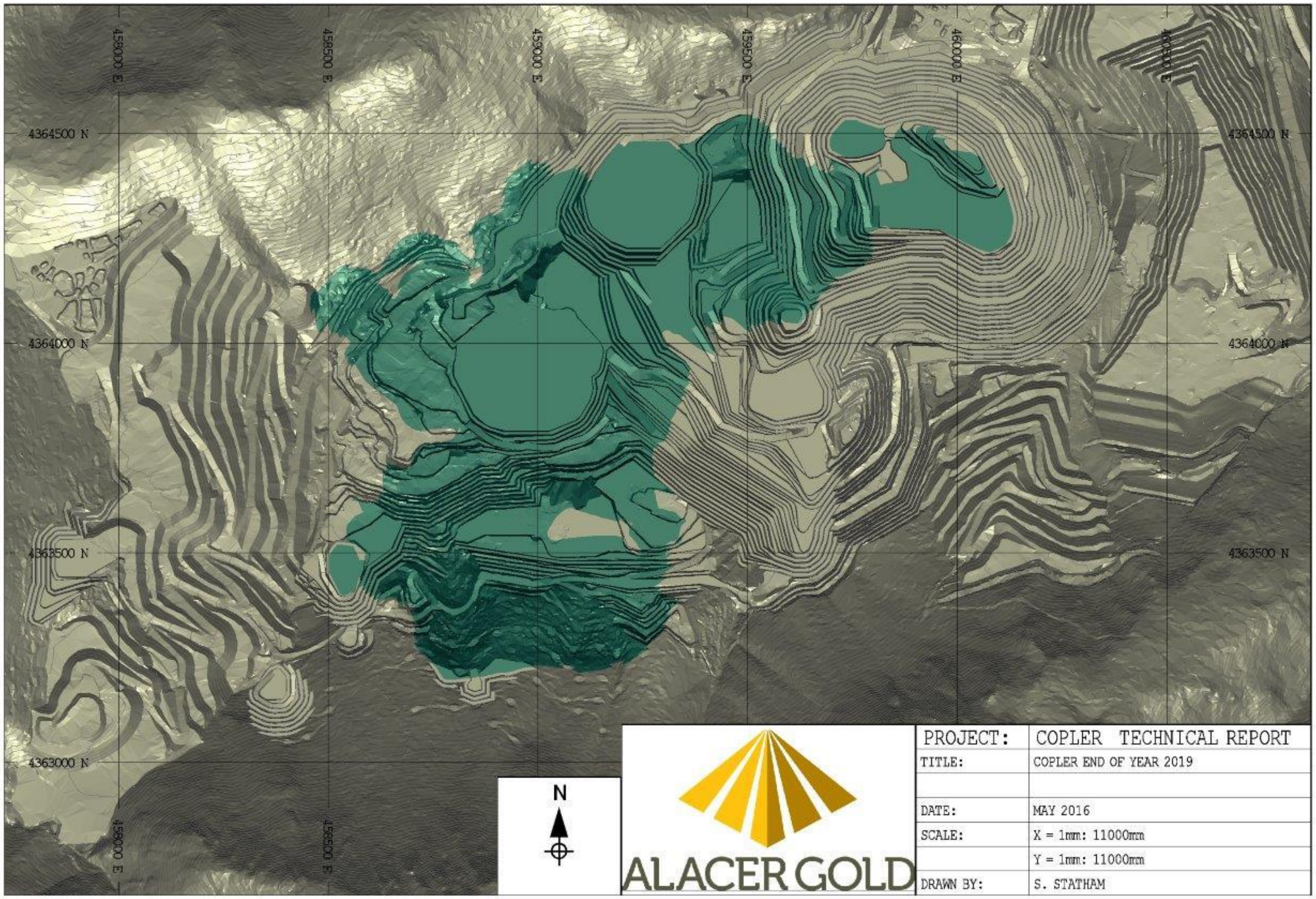


Figure 15-9 End of Year 2020 Mining Progress

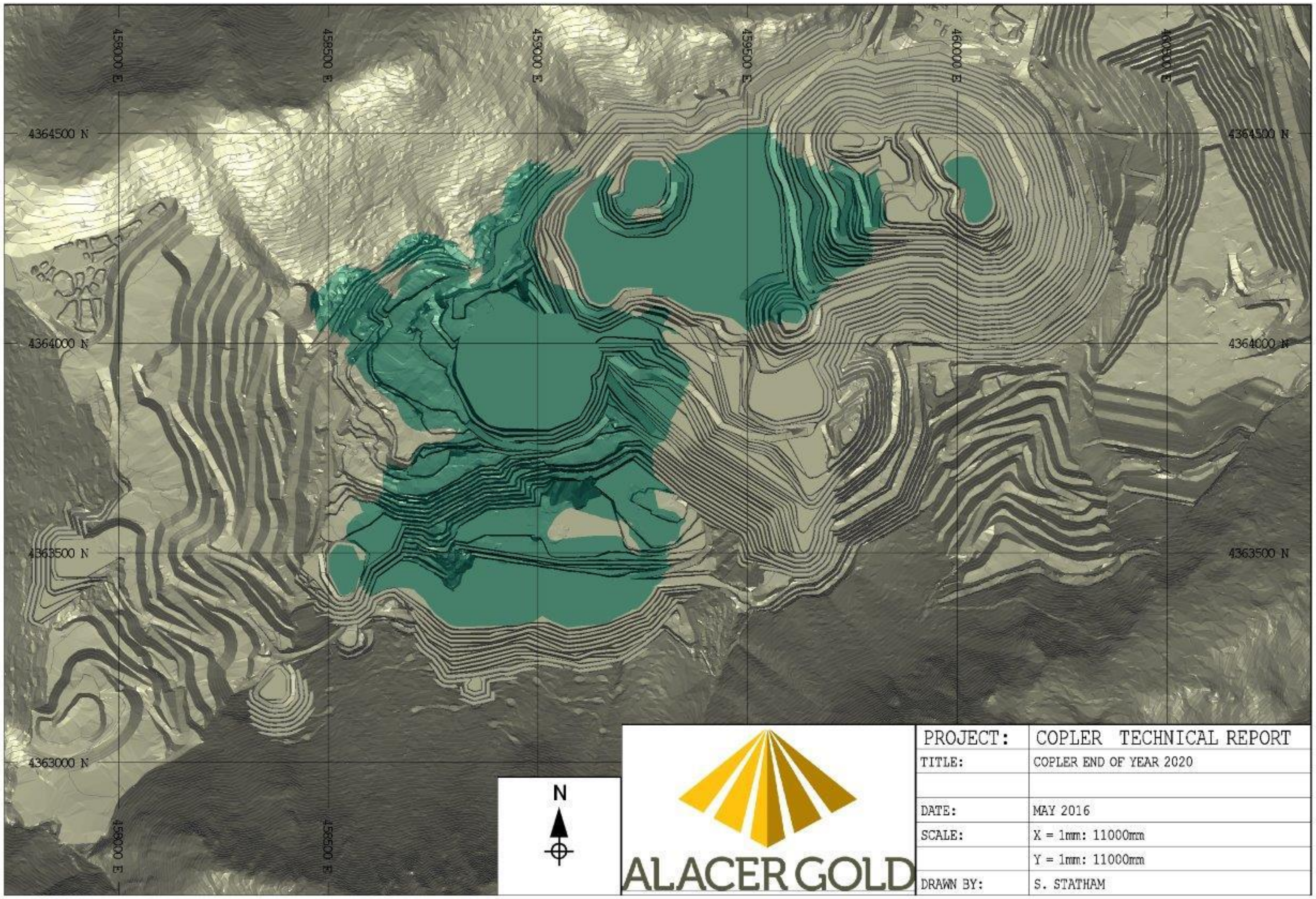


Figure 15-10 End of Year 2021 Mining Progress

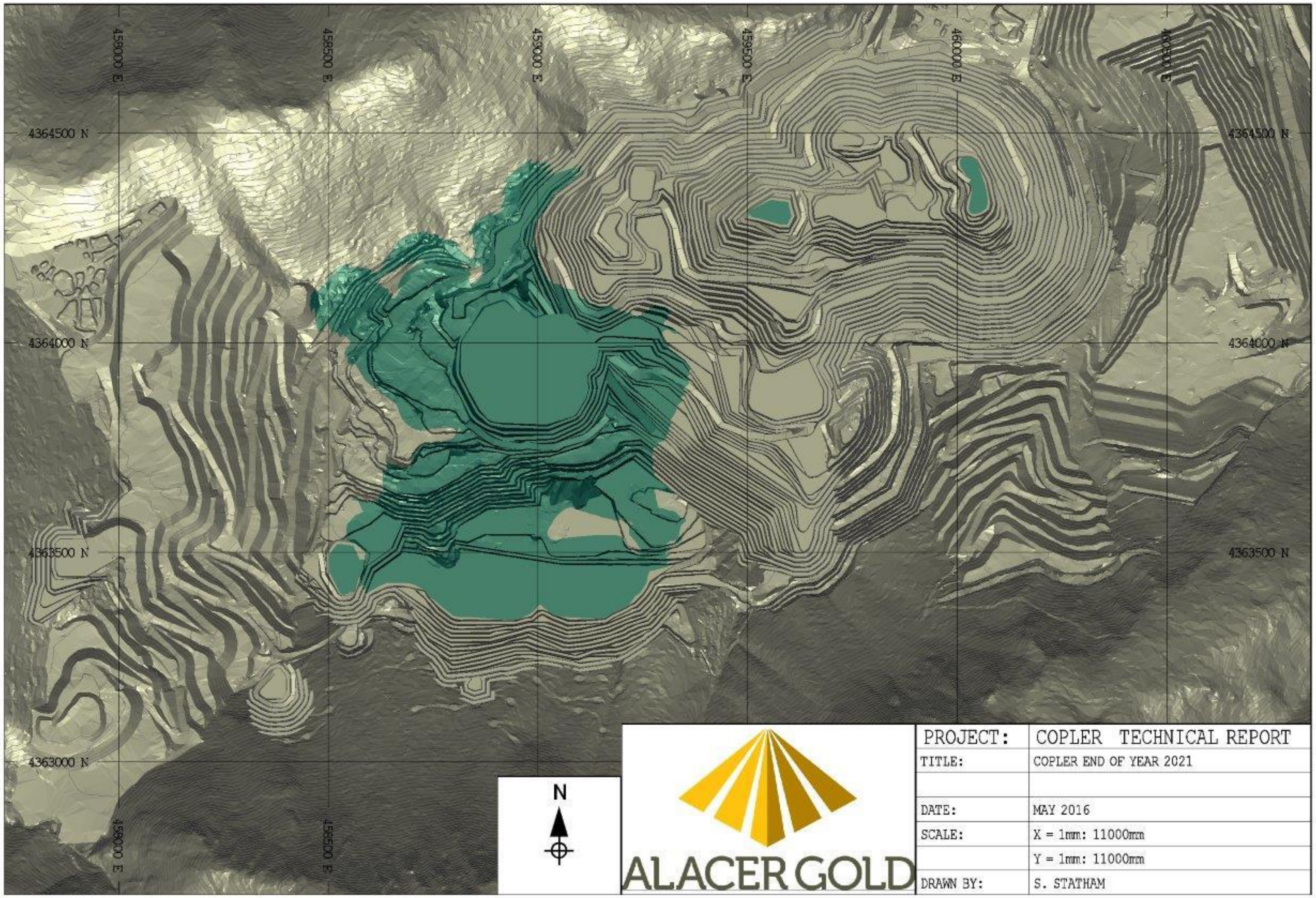


Figure 15-11 End of Year 2022 Mining Progress

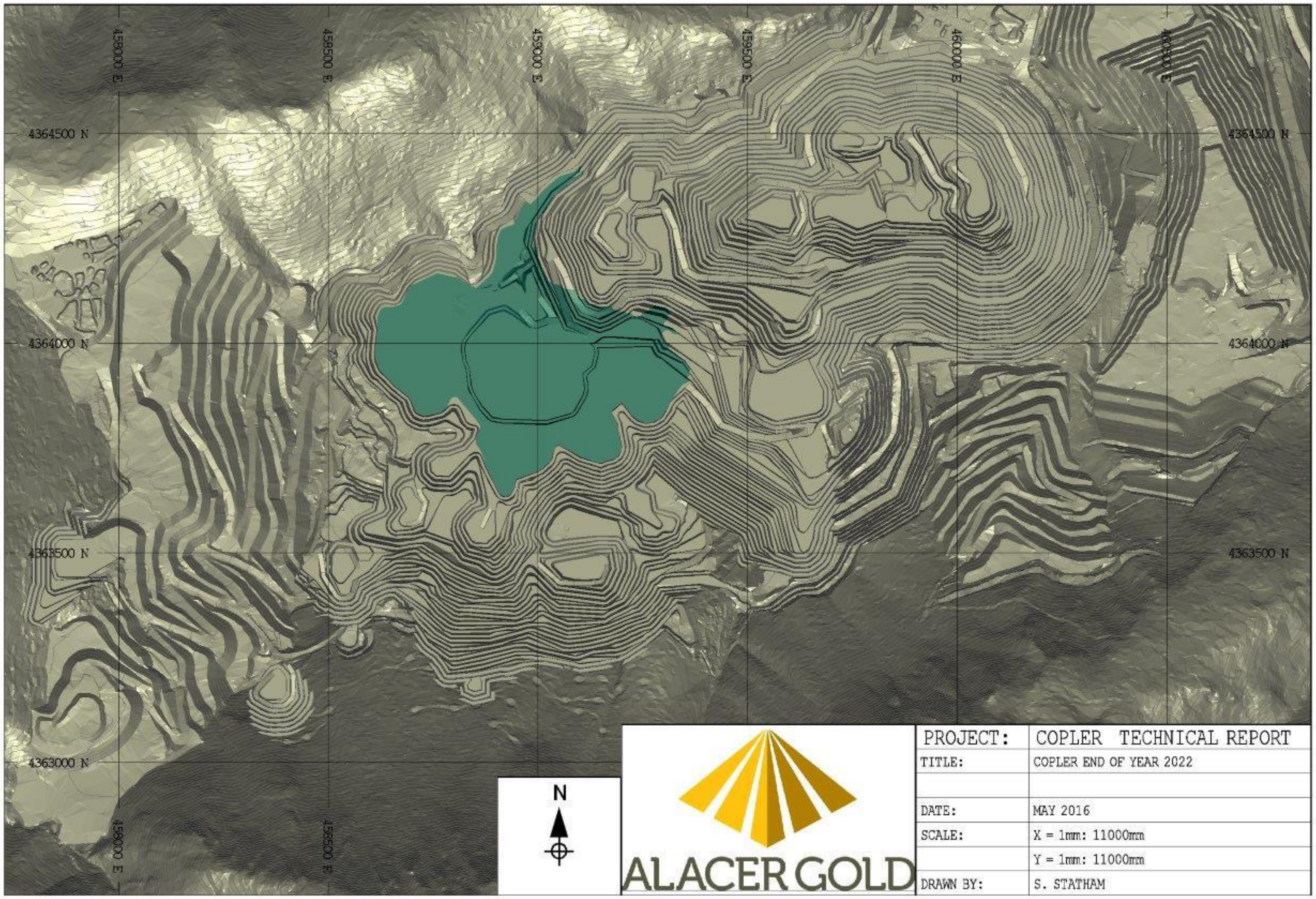
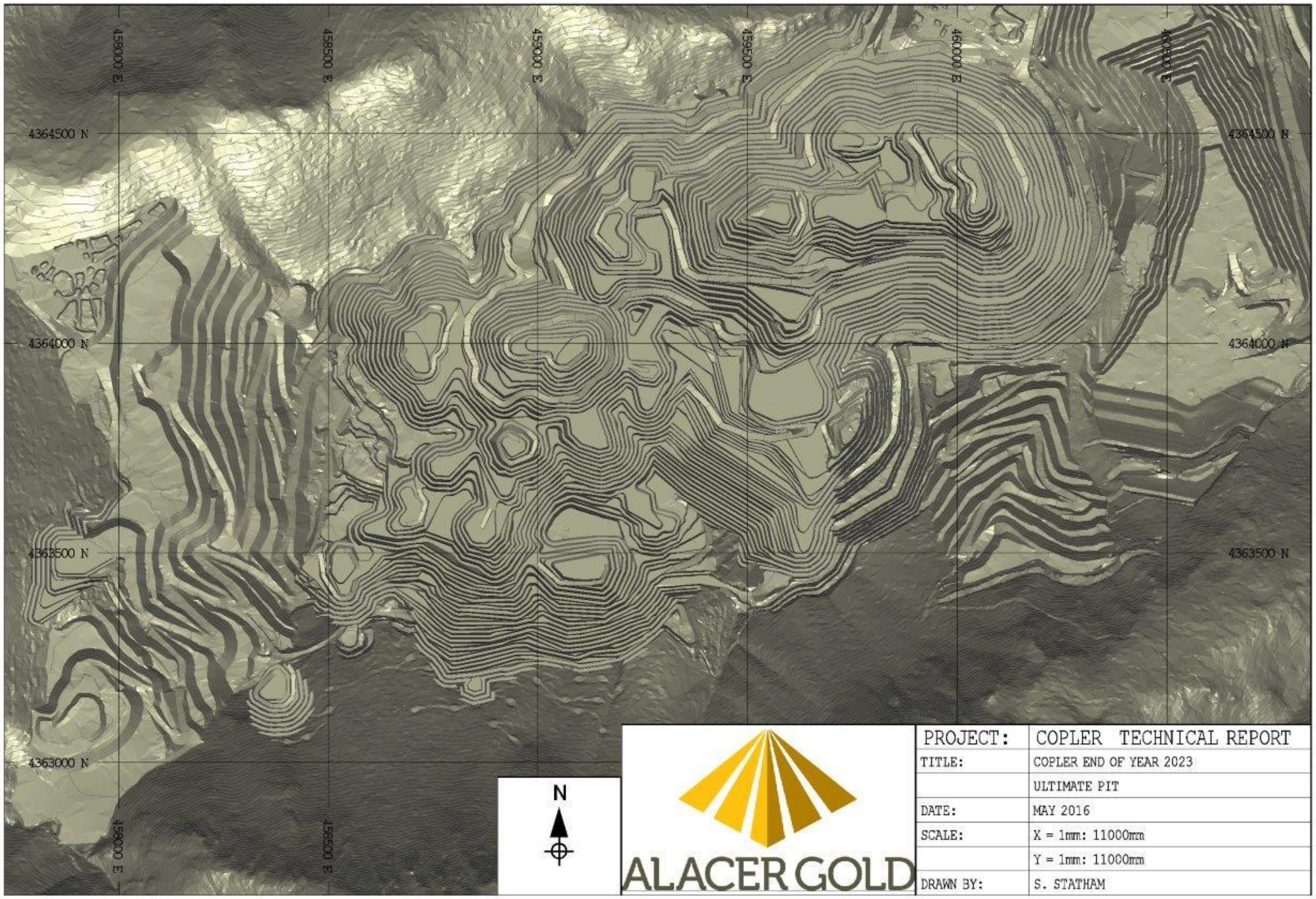


Figure 15-12 End of Year 2023 (Ultimate Pit) Mining Progress



15.2 Risks and Opportunities

There exist several risks and opportunities that could affect the above Mineral Reserve statement and the viability of the mine plan. The known potential risks to the Çöpler Mineral Reserves are:

- Political risks – both local and national.
- Social risks – To have a successful mining operation a company must have a social license to operate.
- Environmental risks – The Çöpler Mine is subject to a changing regulatory environment. The Çöpler Mine must also remain within compliance of current environmental regulations.
- Fuel cost risk – Turkey is highly dependent on foreign supplied petroleum and fuel cost makes up a large portion of the overhead cost of operating a mine. An increase in fuel costs could have a negative impact on the economic viability of the Çöpler Mine.
- Geotechnical risks – Open pit mines are susceptible to highwall and stockpile failures resulting in injury, equipment loss, and/or the abandonment of all or part of a designed pit. Additionally, the Çöpler Mine is located in an area with a history of significant seismic activity that could negatively impact mining operations.
- Gold price risk - Volatility in the price of gold can have an impact on current pit limit economics.
- Contract mining cost risk – A contract will need to be negotiated between Anagold and the mining contractor for the purposes of the Sulfide Expansion Project to secure mining services through the end of the anticipated mine life.
- Haulage distance – As the mine limits expand, waste storage areas will increase in height resulting in increased haulage distance. Additionally, the sulfide ore requires a large amount of stockpile space due to blending requirements. As blending requirements are further refined it may become necessary to develop more space for sulfide ore stockpiles, causing the waste material to be hauled a further distance than anticipated, having a negative impact on economics.
- Ability to selectively blend from ore stockpiles – sulfide and carbonate will require blending from different parts of the pit and stockpiles to meet a required metallurgical threshold for average grade and, in the case of POX feed, ranges of acceptable chemical compositions for species like sulfide sulfur and carbonate. The ability to provide proper blend from the stockpiles and mine will be important to optimize mill operations and maximize gold recovery. The ability to do this requires detailed grade control and stockpile management systems that have not been fully developed at this time.

In addition to potential risks to the Sulfide Expansion Project, there also exist opportunities for the mine to increase performance and achieve better than expected results. The opportunities include:

- Mining rate – The Çöpler Mine has achieved past production rates that exceed 100 ktpd. This allows Anagold the opportunity to increase the scheduled mining rate if required or deemed advantageous for increasing cash flow or overall economics of the project.
- Mining selectivity – The equipment used for the extraction of ore at the Çöpler Mine has been proven to achieve an operational SMU of approximately 3 m x 3 m x 5 m. This SMU is significantly smaller than the 10 m x 10 m x 5 m block size used to estimate the resource block model to determine grade and mine dilution. This allows for a high level of selectivity for material mined as ore, allowing for potential increases in head grade and/or a decrease in tonnes processed. Additionally, ore control operations are able to sample blast holes that are spaced at 3.75m with 3.25m burden which allows for further selectivity when designing ore control boundaries.
- Geotechnical opportunity – Improved lithological definition could allow for an increase in slope angles in specific alteration zones. Currently these zones are not well defined and are therefore assigned a minimum slope angle. If the definition of the alteration zones can be improved this would result in improved slope parameters allowing for improved ore extraction with reduced waste stripping ratios.
- Exploration potential – There exists opportunity at the Çöpler Mine for the conversion of Inferred Mineral Resources to higher confidence categories through additional infill drilling and supporting studies. Additional upside potential may result if some of the exploration targets can be upgraded with drilling and studies to resource estimates that can eventually be converted to Mineral Reserves
- Mill tailings capacity – The current mine schedule is limited by capacity of the TSF. If an additional TSF were to be constructed, or if settled densities of the tailings were improved, the mine design would have increased opportunity for optimization.
- Underground Mining – Underground mining has not been considered at this time but there remains the potential that high grade zones beneath the open pit may be amenable to underground extraction methods. This would be particularly beneficial as a mill feed source to supplement processing of medium and low grade sulfide ores.

15.3 Conclusions and Recommendations

A credible Mineral Reserve exists within the confines of the designed open pit presented within this Report. The design is well suited for open-pit mining operations by conventional mining equipment. The production schedule is readily achievable and the mining operation will continue in the same manner as the existing oxide operation at the Çöpler Mine.

The following items are recommended as part of the next phase of engineering and design associated with the project. These recommendations are:

- 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.

- Further refinement of the modeled carbonate and sulfide sulfur grades in the resource model should be completed.
- Further mapping and definition of alteration types and zones should be completed so that improved pit slope angles can be realized and geotechnical risk can be reduced.
- A detailed pit dewatering and depressurization plan should be designed and implemented to account for the increased depths of mining activities through the sulfide phases of the pit design.
- Further mapping and definition of the local and regional fault structures should be completed to reduce or realize geotechnical risk in the areas where these structures intersect the pit.
- Pit designs should be further optimized for haulage requirements, blend scheduling, and backfill potential.
- Conduct limit equilibrium analysis for both static and pseudo-static cases using the FS pit design. The purpose of the analysis is to determine a factor of safety (FOS) to quantify the risk of open pit wall failure in various areas of the Project.

16.0 MINING METHODS

Mining operations for the Sulfide Expansion Project will be conducted using the established conventional open-pit mining techniques already in place for the oxide mining operations. There is no expectation that the mining rate or equipment demands will increase.

This Report is based on the assumption of continued use of a mining contractor. The contractor will supply all personnel, equipment, and facilities required to perform the entire mining operation at a current LOM average cost of US$1.71/t of total material mined. Alacer will incur additional costs of US$0.19/t associated with the supervisory, engineering, and grade control functions.

The above costs are used as the basis of the Mineral Reserve estimate and may not reflect cost metrics used for financial analysis because of the timing of the cost estimate and the differences in allocation of various site support costs.

16.1 Whittle Pit Shell Optimization

A resource block model completed by Amec Foster Wheeler and Alacer in February 2016 was used as the basis for detailed economic pit optimization using Geovia Whittle Version 4.4.1 pit optimization software. This software, in conjunction with economic, metallurgical, and geotechnical criteria, was used to develop a series of economic pit shells. These pit shells formed the basis for design and production scheduling.

On the basis of metallurgical testwork and trade-off studies, the following processes were selected for the processing of ore:

- Heap leach of all oxide ore
- Whole-ore POX of all sulfide ore

16.1.1 Mining and Processing Economics

The following costs are used as the basis of the Mineral Reserve estimate and may not reflect cost metrics used for financial analysis based on the timing of the cost estimate and the differences in allocation of various site support costs.

Mining costs are based on current contract rates that have been agreed upon with the existing on-site contractor. Current contract mine rates are $3.950/m^3 of material moved, or $1.50/t. Anagold is charged an additional fuel surcharge based on the haulage distance and fuel price over the life of the contract. Alacer has estimated this cost to be an additional $0.21/t mined due to extended haulage distances over the life of the Project. Alacer expects to incur an additional $0.19/t mined for mine administration, engineering, ore control, pit geology, and survey control activities. The total mining cost for operations at the Çöpler Mine is estimated at $1.90/t of material mined.

Oxide ore processing operating costs have been estimated by the Alacer metallurgy group based on 2014 and 2015 actual costs and previous testwork. Additionally, $3.50/t and $0.33/t of ore has been added to total oxide ore processing costs for site general and administrative (G&A) and sustaining capital respectively. These costs are shown in Table 16-1.

The sulfide POX ore processing operating cost of $33.40/t used in pit optimizations are from previous operating cost estimates. Additionally, $3.50/t and $4.53/t of ore have been added to total sulfide ore processing costs for site G&A and sustaining capital respectively. Processing costs used for pit optimization are shown in Table 16-1. Costs attributable to gold sales used in the Whittle optimization are shown in Table 16-2.

All processing costs are reported as US dollars per tonne of ore processed. All mining related costs are reported as US dollars per tonne mined.

Table 16-1 Mining and Processing Costs for Whittle Optimization

Material	Mining Cost	Rehandle Cost	Crushing Cost	Fixed Processing Cost	Variable Processing Cost	SART Cost	Total Processing Cost ($/t)	Sustaining Capital	G&A (Process and 90% of Site)	Whittle Ore Cost
Limestone/Marble - Oxide Ore	$ 1.895	$ 0.320	$ 2.027	$ 1.760	$ 1.086	$0.048	**$ 5.241**	$ 0.334	$ 3.502	**$ 9.077**
Metasediments - Oxide Ore	$ 1.895	$ 0.320	$ 2.027	$ 1.760	$ 4.752	$0.429	**$ 9.288**	$ 0.334	$ 3.502	**$ 13.124**
Diorite - Oxide Ore	$ 1.895	$ 0.320	$ 2.027	$ 1.760	$ 4.879	$0.881	**$ 9.867**	$ 0.334	$ 3.502	**$ 13.703**
Manganese Diorite - Oxide Ore	$ 1.895	$ 0.320	$ 2.027	$ 1.760	$ 4.879	$0.881	**$ 9.867**	$ 0.334	$ 3.502	**$ 13.703**
ALL Sulfide Ore	$ 1.895	$ 0.900		$ 15.770	$ 16.730		**$ 33.400**	$ 4.533	$ 3.502	**$ 41.435**

Table 16-2 Costs Attributable to Gold Sales

2016 Optimization Metals Prices - Basis	
Au Sell Price/oz	$1,250.00
Ag Sell Price/oz	$ 17.00
Cu Sell Price/lb	$ 2.80
Selling Costs/oz	$ 8.54
Royalty (%)	2.0%

16.1.2 Processing Recovery

Processing recoveries are based on test data and further described Section 13.0.

Oxide ore recovery is variable by rock type.

Table 16-3 details the recoveries used for oxide ore in the Whittle optimization process. At the time of pit optimization, sulfide ore gold recovery was estimated by the recovery equation shown in Figure 16-1. Sulfide ore silver recovery was estimated to be 3.0%. Subsequently, the gold recovery equation has been refined and changed. This change in gold recovery is noted in Section 15.1.4.

Figure 16-1 Gold Recovery Equation (where HG_{Au} is the gold head grade)

$$Gold\ Recovery\ (\%) = \left\{\frac{HG_{Au} - \left[0.0285 * \ln\left(HG_{Au} + 1 + \frac{HG_{Au}}{0.028}\right)\right]}{HG_{Au}} - 0.01\right\} * 100$$

Table 16-3 Whittle Optimization Recoveries – Oxide Ore

Pit Region	Rock Type	Processing Recovery		
		Au	Ag	Cu
Manganese	Limestone/Marble	78.4%	27.3%	3.5%
Marble	Limestone/Marble	75.7%	34.0%	3.5%
Main	Limestone/Marble	68.6%	24.6%	3.5%
Main East	Limestone/Marble	78.4%	27.3%	3.5%
Main West	Limestone/Marble	75.7%	34.0%	3.5%
West	Limestone/Marble	75.7%	34.0%	3.5%
Manganese	Metasediments	66.8%	32.5%	13.8%
Marble	Metasediments	66.8%	32.5%	13.8%
Main	Metasediments	66.8%	32.5%	13.8%
Main East	Metasediments	66.8%	32.5%	13.8%
Main West	Metasediments	66.8%	32.5%	13.8%
West	Metasediments	66.8%	32.5%	13.8%
Manganese	Diorite	71.2%	37.8%	15.8%
Marble	Diorite	62.3%	32.0%	15.8%
Main	Diorite	71.2%	37.8%	15.8%
Main East	Diorite	71.2%	37.8%	15.8%
Main West	Diorite	62.3%	32.0%	15.8%
West	Diorite	62.3%	32.0%	15.8%

16.1.3 Pit Slope Angle

In April 2015, Golder Associates (Golder) completed a pit slope optimization study intended to further optimize the pit slope angles as defined in their earlier study completed in April 2014. This program included the drilling of five oriented geotechnical core holes to identify any prevalent jointing throughout the Çöpler deposit. The results of the study have provided Alacer 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 modeling of alteration zones within the Çöpler deposit. Where slope angles were not able to be further refined, Golder recommended that Alacer follow the recommendations set forth in the 2014 geotechnical review study.

To account for haulage ramps and additional safety benches an additional 2° were subtracted from the inter-ramp angle when defining the slope angles for pit shell optimization. The resulting geotechnical design parameters for Whittle pit optimizations are shown in Table 16-4.

Table 16-4 Whittle Optimization Slope Parameters

Çöpler OPTIMIZATION and PIT DESIGN PARAMETERS				
*Slopes based on Golder site review March 2014				
	Altered - RQD<15		Un-Altered (Fresh) - RQD>15	
Rocktype	***Whittle Slope Code***	***Inter-Ramp Slope***	***Whittle Slope Code***	***Inter-Ramp Slope***
100 (Marble)	1	50.5	1	50.5
200 (Metasediments)	4	32	2	43
300, 400 (Gossan, Massive Sulfides)	3	40	3	40
500, 600 (Diorite)	6	23	5	38

16.1.4 Whittle Optimization Process

Using the economics and design parameters developed by Alacer, several Whittle pit optimizations were completed on the Çöpler deposit to determine the optimal mining extents. Various scenarios were examined, including:

- Unconstrained gold cutoff – Whittle was allowed to determine an optimal pit shell using cash flow analysis without regard to gold cutoff grade. In this type of scenario Whittle determines the maximum pit limits on a cash flow only basis. This scenario generates the largest economically mineable pits possible.
- Constrained gold cutoff grades – Whittle uses both cash flow and minimum gold cutoffs to determine an optimal pit shell. A number of scenarios were examined with varying heap leach and POX cutoff grades. A total of 19 various cutoff grade scenarios were evaluated. These scenarios tend to produce similarly sized ultimate pit shells with the various cutoff grades affecting classification of ore and waste in the pits.
- Pit shell sensitivity – For all scenarios the gold price was varied from US$500 to US$1,500 per troy ounce in US$50 increments. The process of varying gold price provides a holistic view of the resource's sensitivity to net revenue. Total discounted net revenue is the main basis for choosing the optimal pit shell.

For all of the Whittle scenarios examined, only material with a Measured or Indicated resource classification was considered as potential process feed. All Inferred material was considered as waste.

A large number of scenarios were examined during the pit optimization process. Due to the expected commissioning date of the sulfide ore processing facilities occurring around the same time that the oxide Mineral Reserve is expected to be depleted, an oxide-only pit shell was generated using Whittle. After a comparison of various cut-off grade scenarios pit shell number 16 ($1,250 Au), with a break-even oxide ore cut-off grade, was selected as the oxide design basis. This oxide pit shell was then used as a starting point for optimizing the sulfide resource below the oxide pit.

After a comparison of various cut-off grade scenarios and with the consideration that the TSF capacity is limited to 40.2 Mt tonnes of sulfide ore, pit shell number 8 ($850 Au) with a sulfide ore cut-off grade of 1.5 g/t Au was selected as the sulfide design basis. The total discounted cash flow for each pit shell in all scenarios was calculated and compared against one another. The maximum discounted cash flow for 'Best Case' and 'Worst Case' mining sequences were identified and the optimal pit was chosen by studying the 'Specified Case' mining sequence between those two points. The optimal pit and cut-off grade was then chosen where the maximum value within the above described capacity constraints was achieved.

The Whittle 'Best Case' mining sequence calculates pit value by processing each pit shell incrementally from the smallest revenue factor up to the largest revenue factor in order to maximize the number of phases in the pit and to provide the earliest possible cash flow delivery. The Whittle 'Worst Case' mining sequence calculates pit value by processing each pit shell as a whole pit with no phasing, maximizing the delay in potential cash flow. The Whittle 'Specified Case' mining sequence takes a more realistic approach to the processing of the nested pit shells. In this case two theoretical mining phases were applied at pit shell number 3 ($600 Au) and pit shell number 5 ($700 Au). This approach allows for two phases prior to the ultimate pit in order to bring cash flow forward and represents a more realistic representation of how the pit would be mined. The chart that is commonly used to assist in optimal pit selection is shown in Figure 16-2.

The ultimate sulfide pit shell for the chosen scenario is shown below in Figure 16-3.

Figure 16-2 Example Whittle Best/Selected/Worst Case Results

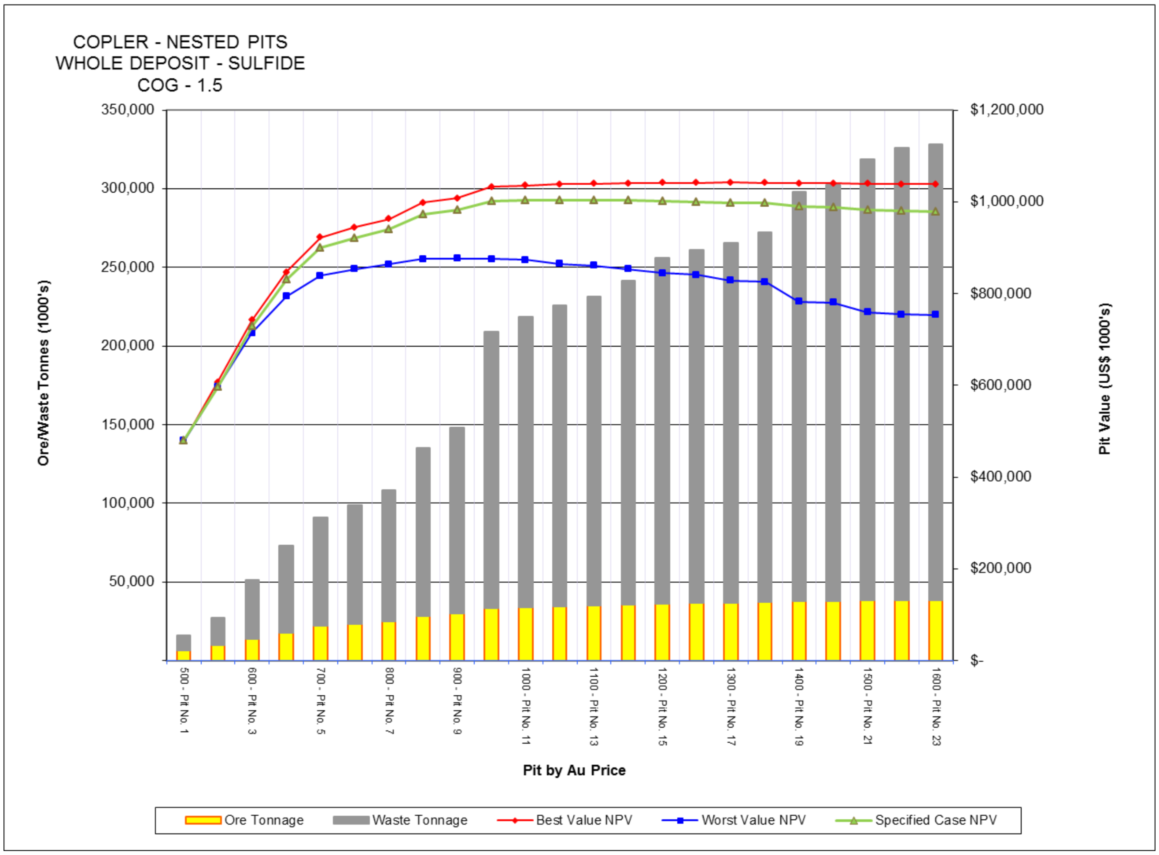
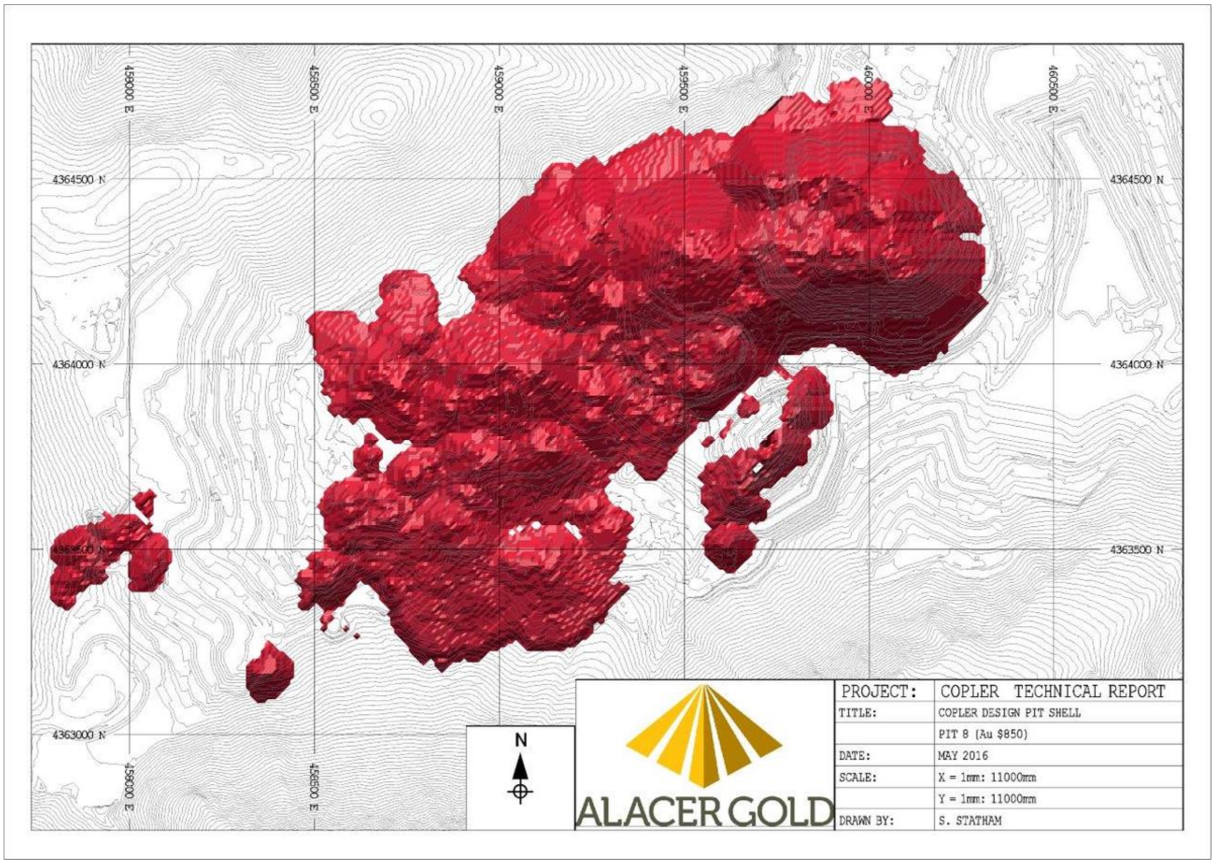


Figure prepared by Alacer, 2016.

Figure 16-3 Whittle Pit Shell for Design (Oxide Cut-off: Break-even, Sulfide Cut-off: 1.5, Pit 8-$850Au)



16.2 Pit Design

Once pit optimization was completed, as described above, the selected pit shell was used as the basis for detailed mine design. Pit designs are completed using MineSight's 3D pit expansion tool.

16.2.1 Geotechnical Design Parameters

Geotechnical design parameters are based on a comprehensive review by Golder of the pit slope stability conditions at Çöpler as described in Section 16.4. Geotechnical design parameters for pit design are shown in Table 16-5. Additionally, efforts are made to avoid designing potentially unstable wall configurations such as sharp noses and continuous sections of highwall greater than 100 m in height without additional catch bench relief.

Table 16-5 Alacer Pit Design Slope Parameters

Çöpler PIT DESIGN PARAMETERS								
*Slopes based on Golder 2014 pit slope review.								
	Altered - RQD<15				Un-Altered (Fresh) - RQD>15			
Rocktype	*Inter-Ramp Slope*	*Face Angle*	*Catch Bench*	*Total Bench Height (m)*	*Inter-Ramp Slope*	*Face Angle*	*Catch Bench*	*Total Bench Height (m)*
100 (Marble/Limestone)	52.5	75	7.49	15	52.5	75	7.49	15
200 (Metasediments)	34	50	6.43	10	45	70	6.36	10
300, 400 (Gossan, Massive Sulfides)	42	65	6.44	10	42	65	6.44	10
500, 600 (Diorite)	25	45	5.72	5	40	60	6.14	10

16.2.2 Bench Design

Mining production benches will be 5 m in height with a final wall height varying from 5 m to 15 m depending on the geotechnical requirements. The minimum mining width varies from 15 m to 30 m in width depending on the bench configuration.

16.2.3 Haul Roads

Haul road widths are calculated with the expectation that the current Mercedes Axor 36 t haul trucks will continue to be used throughout the mine life. The design width of 15 m for two-way traffic allows for 3.5 truck widths plus an internal drainage ditch and a safety berm that is 0.75 times the height of the largest wheel diameter of all mine equipment that is expected to be traveling that route. Single-lane haulage traffic is allowed in the lower benches of the mine and is set at 10 m wide. All haulage ramps are designed at a maximum gradient of 10%.

16.2.4 Phase Design

Phases are designed within the ultimate pit boundary in order to maximize mining and processing flexibility and cash flow. All phases are designed with consideration for haulage access and minimum mining width. For the optimal sequencing of oxide and sulfide ore extraction, 16 pit phases were designed. Ten pit phases target the oxide ore, and six subsequent pit phases target the sulfide ore. Once the ultimate pits and internal phases are designed, reserves are generated for each individual phase, and the phases are each ranked and sorted based on value and preferred mining sequence. Phase design tonnage and grade values are shown in Table 16-6. Figure 15-12 illustrates the ultimate pit design surface.

Table 16-6 2016 Çöpler Phase Design Tonnage and Grade

2016 Çöpler Phase Design Tonnage and Grade							
Pit Area	Phase	Oxide Ore		Sulfide Ore		Waste	Total
		Tonnes (x1000)	Au Grade	Tonnes (x1000)	Au Grade	Tonnes (x1000)	Tonnes (x1000)
Marble	Phase 03	737	1.73	250	3.14	1,003	1,990
	Phase 05	2,313	1.26	547	2.75	14,059	16,919
	Phase 06	426	1.27	955	2.78	14,740	16,121
	Phase 07	1,065	1.14	814	2.23	15,404	17,282
	Phase 08	1,584	1.29	6,736	2.38	45,215	53,535
	Phase 09	179	1.11	1,103	2.26	3,030	4,312
Manganese	Phase 02	6,878	1.00	2,917	2.78	30,372	40,166
	Phase 03	3,388	0.95	3,835	2.75	35,838	43,061
	Phase 04	249	1.38	19	3.63	384	652
Main Zone	Phase 02	27	0.78	-	-	115	142
	Phase 06	126	1.33	-	-	1,123	1,249
	Phase 07	76	1.29	50	1.97	574	700
	Phase 10	44	1.05	5,792	2.75	6,861	12,697
	Phase 12	92	1.18	251	2.63	1,556	1,899
	Phase 13	227	1.06	11,533	2.66	53,151	64,911
West Zone	Phase 02	426	2.06	78	2.69	1,332	1,836
Total	**Total**	**17,836**	**1.13**	**34,879**	**2.63**	**224,757**	**277,472**

16.3 Hydrology and Pit Dewatering

16.3.1 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 (EIE-2156) in the upper Euphrates Basin, is about 145 m^3/sec, draining an area of 15,562 km^2. The maximum flow rate recorded at this station was 1,320 m^3/sec in May of 1969 and the minimum was 43.8 m^3/sec in January 1974. Peak flow rates are observed in April and May following snow melt and precipitation (Ekmekci and Tezcan, 2007). At Gauging Station No. 2156, the average flow rate was approximately 275 m^3/sec. The maximum flow rate recorded at this station was 338 m^3/sec in May of 1969 and the minimum was 55.9 m^3/s in September 1986.

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 dam crest elevation is 918 m amsl. The operational flow rate is planned to be 330 m^3/sec, however, if inflow is less than this amount the dam will release 15 m^3/sec until they reach capacity to operate at 330 m^3/sec.

The Çöpler and Sabırlı streambeds in the study area do not flow perennially. They both discharge into the Karasu River. The drainage area of the Sabırlı Creek is about 35 km^2 and that of the Çöpler Creek is about 10 km^2. In 2005, streamflow measurements in Çöpler Creek indicated that the stream flow varied along the length of the stream bed. Discharges measured in the western part of Old Çöpler Village were approximately 10 L/sec. Measurements made in March 2007 indicated discharges of 1 L/sec before the Old Çöpler Village and 15 L/sec in the western part of Çöpler. Measurements in the Sabırlı stream were carried out in March and April 2005, and the discharge was measured as 20 L/sec (SRK, 2008).

16.3.2 Rainfall

Rainfall data from three weather stations in and around the Project area were reviewed as follows:

- Divriği station (approximately 37 km from the site), 41 years of record,
- Erzincan (approximately 90 km from the site), 45 years of record; and,
- WS2 (located on-site), 9 years of record.

Station WS2 had an insufficient length of record to be considered for this analysis. Monthly average and maximum daily rainfall for each month were reported along with IDF curves for storms with return frequencies of 2- to 100-years for a period of record of 1970-2010 for the Divriği station.

The Divriği station data were used to perform a detailed analysis using a dataset relatively close to the site. Calculations of 2- to 100-year storm event depths as well as a probable maximum precipitation (PMP) estimate were performed for this station.

Table 16-7 summarizes the results of the data review and indicates those gauges where sufficient information was available to derive design storm depth estimates.

Table 16-7 Summary of Gauge Records with Sufficient Data for Analysis

Gauge	Record Period	Monthly Average	2- to 100-Year Frequency Analysis	PMP
Erzincan (monthly)	1975-2010	X		
Erzincan (daily)	1963-1968, 1973-2012	X	X	X
Divriği (monthly)	1970-2010	X		
Divriği (daily)	1970-2011	X	X	X
İliç	n/a			
Sivas	1929-2004	X	X	X
WS2	2004-2012			

A graphical frequency analysis was performed using the daily rainfall data for the Divriği, Sivas, and Erzincan rainfall gauges to develop estimates of storm depths for 2- to 100-year storm events. The maximum daily rainfall for each year of record was analyzed using standard distribution methods (i.e. Gumbel, Generalized Extreme Value, Weibull, Log-Pearson Type III, Lognormal, and Exponential), to determine the best fit trend line.

A Weiss factor of 1.13 was applied to the calculated storm depth values to account for rainfall measurements that are only recorded once per day. The resulting peak rainfall data determined using the Divriği station data is similar to the values used in previous studies. The 24-hour storm depths determined for the Çöpler site are shown below in Table 16-8. Monthly average rainfall values are shown in

Table 16-9. The average annual rainfall for the site is 384.3 mm.

Table 16-8 24-hour Storm Depths for Çöpler Project

Frequency (yrs)	2	5	10	25	50	100
Divriği Gauge (mm)	29.6	38.2	44.2	52.1	58.2	64.6

Table 16-9 Average Rainfall for Çöpler Project

Month	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Average Rainfall (mm)	34	33.1	44.6	57.4	55.6	26.5	7.7	4.5	11.4	36.5	36.4	36.6	384.3

For the PMP analysis, the World Meteorological Organization's (WMO) Statistical Method was used to determine PMP estimates using the daily data from the Divriği, Erzincan, and Sivas gauges.

Calculated PMP estimates for the Divriği, Erzincan, and Sivas stations are 110.9 mm, 779.6 mm, and 212.8 mm, respectively. Based on the proximity of the Divriği and Erzincan gauges to the Çöpler Mine, linear interpolation of these PMP estimates based on distance from the Project site was performed. The Sivas station was excluded from the interpolation due to the station being a significant distance from the mine. This interpolation results in a 24-hour PMP estimate of 302.0 mm for the Çöpler Mine.

16.3.3 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 Sulfide Expansion Project included an evaluation of the current surface drainage conditions in order to develop the surface water diversion concept. Diversion facilities will consist of a network of diversion channels and retention structures to minimize storm water run-on to the proposed mine site facilities and to prevent mine-impacted storm water run-off from exiting the site and discharging to the Karasu River.

An update to the Surface Water Management Plan (SWMP) is currently being developed by Golder for submittal to regulatory agencies in the second quarter of 2016. The SWMP update includes an evaluation of the existing surface drainage conditions and planned conditions related to the planned Sulfide Expansion Project. This included development of an existing conditions surface water hydrology model and a fully developed conditions model. The updated August 2015 1 m site topography was used to delineate the contributing sub-basin areas upstream and downstream of the site. The surface run-off conditions for each sub-basin were developed from evaluation of previous soil and vegetation characterizations for the site, a review of recent aerial photography, and from experience gained from observations of actual site conditions by Anagold and Golder staff. Typical Soil Conservation Service (SCS) curve numbers were applied to the analysis.

The sub-basin areas, characterization of the surface run-off conditions, and design rainfall data were used to construct the existing conditions hydrology model. The hydrology analysis utilized HEC-HMS software to develop estimates of the peak flow rates and volumes generated by the existing watersheds.

16.3.4 Surface Water Management Structures

The results of the existing surface water hydrology provided the framework for the site-wide surface water management and the proposed diversions required to accommodate the changes planned for the Sulfide Expansion Project. Engineered surface water management structures are proposed to minimize effects of storm water run-on 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 utilized 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 required to be lined based on the EIA commitments.

Interim diversion channels are designed to convey the 25-year storm event with 0.5 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. Channel lining material is dependent on design flow velocity. Earth lined channels are allowed for velocities less than 1.5 m/s. Riprap is used for channel velocities between 1.5 m/sec and 3.0 m/sec. Riprap is used on channel banks when the design flow is subcritical and the entire channel perimeter is riprap lined when design flow is supercritical. Other channels are constructed in durable bedrock and have no specified revetment. HEC-RAS hydraulic modeling software is used to model permanent diversion channels to ensure capacity and velocity requirements are met. Manning's equation assuming normal depth is used to model temporary channels for flow depth and velocity.

Lined sediment ponds are planned for construction downgradient of the expanded 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. An emergency spillway or other upstream surface water diversions for this facility will be provided as part of the mine closure design.

In addition to the TSF, the proposed site-wide surface water control structures are as follows:

- Permanent South Diversion Channel located upstream of the southern ultimate pit limit and extending beyond the Phase IV leach pad expansion to divert water away from the pits, HLF, and WRSA that discharge into Sabirli Creek
- A diversion channel to be constructed adjacent to the Sabirli Road realignment up-gradient of the TSF as part of the Phase 3 expansion of the TSF
- A temporary diversion berm/channel above the Lower Çöpler WRSA
- Construction of internal flood control structures designed to carry water away from the pit rims and toward internal infiltration basins or the existing French drain below the Lower Çöpler West Waste Dump
- Construction of seven new sediment ponds below the WRSAs and upstream of discharge points to the environment
- Construction of an infiltration pond within the drainage basin adjacent to the planned sulfide plant area that allows for infiltration of surface water into the existing French drain below the Lower Çöpler WRSA.

16.3.5 Pit Dewatering

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 the predominant pathways for infiltration. The main hydrogeologic units and features considered in the groundwater model were:

- Munzur Limestone (modeled hydraulic conductivity = 0.6 m/day)
- Intrusive Diorite (modeled hydraulic conductivity = 0.0002 m/day)
- Metasediments (modeled hydraulic conductivity = 0.0002 m/day)
- Alluvium (modeled hydraulic conductivity = 10 m/day)
- Various fault systems (Sabirli, Çöpler, and Other) (modeled 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 (±20 m) elevation. Modeling results indicate that water from beneath Lower Çöpler West WRSA will take more than 1,000 years to flow to the Karasu River. Groundwater located beneath the Lower Çöpler East WRSA 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 m) will be higher than the minimum pit elevation simulated in the model (875 m). Additionally, the area on the north side of the pit and portions of the southern and southeastern 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.

16.4 Geotechnical Pit Slope Stability

The Çöpler Mine maintains an on-site geotechnical monitoring program 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 depressurization. It is expected that pit slope depressurization will be used extensively throughout the Main pit as the sulfide pit phases are progressed.

Golder completed the 2015 pit slope optimization 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 Mine 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. However, Golder was able to set a basis for improved pit slope angles at Çöpler once a detailed alteration model can be created. Anagold is currently reviewing the extent to which an alteration model can be reliably built and the potential benefit to the mine operations. It is likely that Anagold may choose to continue using the more conservative slope angle recommendations recommended by Golder in 2014 until further definition of all alterations and faults at the Çöpler Mine can be understood.

16.4.1 RQD Model

RQD was estimated in the resource model using the inverse distance cubed (ID3) method on 2 m 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 method was used to capture available samples. Block estimates were limited to the search distances used with no attempt to assign unestimated blocks.

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 Mine, it has been determined that RQD is a generally reliable indicator of alteration. Therefore, areas with RQD modeled 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 about 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.

16.4.2 Pit Slope Design Parameters

The pit slope design parameters remain unchanged and are shown below in Table 16-5.

16.4.3 Mine Operations Monitoring and Management

Pit slopes at Çöpler are monitored on a daily basis 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.

The Çöpler Mine operation utilizes perimeter pre-split blasting techniques in areas where competent rock is encountered (typically, limestone/marble, unaltered metasediments, and unaltered diorite). The pre-split holes are drilled according to the bench face angle recommendations as shown in Table 16-4. Blasting is conducted in a manner to minimize back-break though timing and adequate relief. A typical pre-split highwall at Çöpler is shown in Figure 16-4.

Figure 16-4 Typical 15 m pre-split at Çöpler Mine

Photo Courtesy Alacer Gold 2016

Where pre-splitting is not practical, highwalls are sloped by excavator to the recommended bench face angle as shown in Table 16-5. A typical bench face without pre-splitting is shown in Figure 16-5 and Figure 16-6.

Figure 16-5 Typical Bench Face without pre-split

Photo Courtesy Alacer Gold 2016

Figure 16-6 Typical Bench Face without pre-split

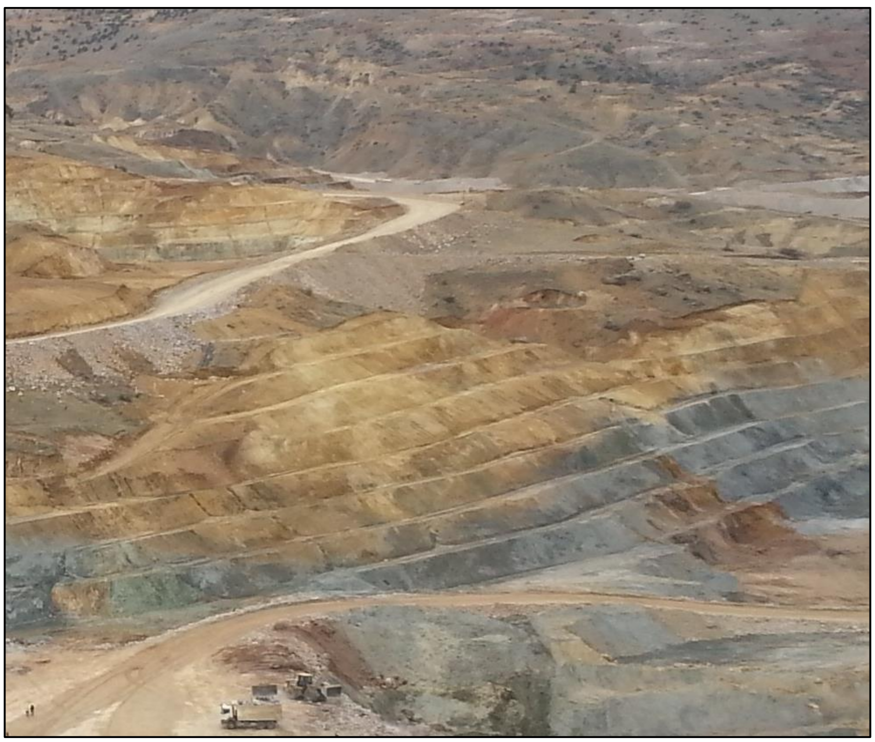


Photo Courtesy Alacer Gold 2016

16.4.4 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 – characterized by competent rocks and marbleized near the Çöpler intrusion.
- Fresh diorite – characterized 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 metasediments – highly weathered, extremely weak rock and soil that occurs in the oxidized zone (depth typically to 30 m).
- Fresh metasediments – fresh to slightly weathered, weak to moderately strong rock consisting of a turbidite sequence that may also be structurally complex near faults.
- Hydrothermally altered metasediments – alteration sufficient to significantly reduce strength relative to fresh metasediments, but without the shearing and intense clay alteration of Contact and Fault zones.
- Fault gouge including intrusive contact and intense sulfide alteration – slickensided 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 geologic model.

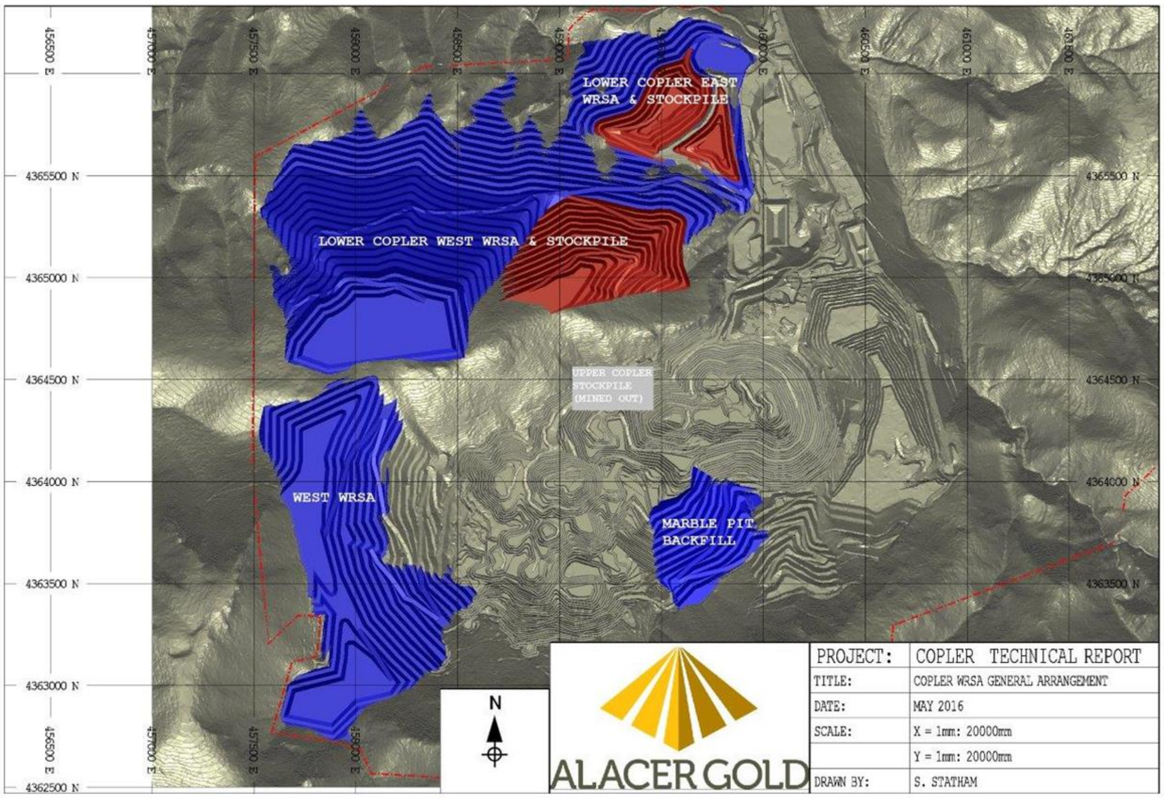
The above listed geotechnical domains are mostly well known and modeled in a geologic model. The alteration zones, however, vary significantly and have not been modeled to an extent to where variations by alteration type are well defined. It has been recommended by Golder that the best way to identify alteration zones is by modeling 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".

16.5 Waste Rock Storage Areas and Stockpile Storage

The Çöpler mine plan allows for the use of five WRSAs to safely store the resulting waste rock and sulfide ore that is extracted due to the mining operations. These five areas are defined as the Lower Çöpler East, Lower Çöpler West, Upper Çöpler, West, and Marble Backfill WRSAs. Current oxide operations utilize three of these WRSAs with the exception of the Lower Çöpler West and Marble Backfill WRSAs. The Lower Çöpler East and Upper Çöpler WRSAs will primarily be utilized as sulfide ore stockpile areas, with the Upper Çöpler WRSA being mined out starting in 2019 to allow for future pushback extension of the Marble pit towards the north.

Figure 16-7 details the location of the above described WRSAs in relation to the final pit surface.

Figure 16-7 Çöpler WRSA General Arrangement



The Lower Çöpler East WRSA has a capacity of 14.9 Mm^3 or 26.8 Mt of mine waste and 5.5Mm^3 or 9.9Mt of sulfide ore. The total surface area impacted by the Lower Çöpler East WRSA is 51.5 ha. The Lower Çöpler West WRSA has a capacity of 94.6 Mm^3 or 170.3 Mt or mine waste and 12.4Mm^3 or 22.3 Mt of sulfide ore. The total surface area impacted by the Lower Çöpler West WRSA is 206.5 ha. The Upper Çöpler WRSA has a capacity of 7.6 Mm^3 or 13.6 Mt of sulfide ore. The total surface area impacted by the Upper Çöpler WRSA is 26.1 ha. The West WRSA complex has a capacity of 34.4 Mm^3 or 61.9 Mt of mine waste. The total surface area impacted by the West WRSA is 108.9 ha.

An estimated 69.8 Mt of waste rock will be consumed in the construction of the tailings storage area, haul road, and tailings pipeline corridor as well. Total constructed waste rock storage capacity is 155.0 Mm^3 or 279.1 Mt. The total surface area impacted by all WRSAs 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.

16.5.1 Waste Rock Geotechnical Design

The WRSAs 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 WRSAs. The WRSAs 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 WRSA 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 optimization study and the updated waste dump designs and layouts developed by Alacer. Six of the most critical cross sections were evaluated to determine the minimum 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 WRSAs. 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 M7.0 operating basis earthquake (OBE). Therefore, a horizontal pseudo-static acceleration of 0.15 g was applied to the WRSA sections in the seismic stability analyses.

The results of the stability analysis are summarized in Table 16-10.

Table 16-10 Lower Çöpler East and West WRSA Design Factor of Safety

Waste Dump	Section	Loading Condition	Failure Surface Location	Minimum Computed FOS	Figure
Lower Çöpler East Dump	A	Static	Shallow	1.4	6
		Pseudo-Static		1.1	
		Static	Deep	1.9	
		Pseudo-Static		1.3	
	B	Static	Shallow	1.7	7
		Pseudo-Static		1.3	
		Static	Deep	1.9	
		Pseudo-Static		1.3	
Lower Çöpler West Dump	C	Static	Shallow	1.7	8
		Pseudo-Static		1.3	
		Static	Deep	1.9	
		Pseudo-Static		1.3	
	D	Static	Shallow	1.6	9
		Pseudo-Static		1.2	
		Static	Deep	1.8	
		Pseudo-Static		1.3	
West Çöpler Dump	E	Static	Shallow	1.6	10
		Pseudo-Static		1.1	
		Static	Deep	1.9	
		Pseudo-Static		1.3	
	F	Static	Shallow	1.6	11
		Pseudo-Static		1.2	
		Static	Deep	2.0	
		Pseudo-Static		1.4	

The Lower Çöpler East WRSA facility will be constructed over a portion of the existing Northeast WRSA. Foundation conditions underlying the existing Northeast WRSA and the proposed Lower Çöpler East (LCE) facility consist of Munzur Limestone. Minimum computed factors of safety for the LCE facility are 1.4 and 1.1 for static and seismic loading conditions, respectively.

The Lower Çöpler West WRSA 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 WRSA is to be constructed adjacent to the Çöpler open pit and will be founded on Munzur Limestone and metasediments with sporadic diorite intrusions. Minimum computed FOS are 1.9 and 1.3 for static and seismic loading conditions, respectively.

16.5.2 Waste Rock Geochemical Review

SRK established a criteria for identifying potentially acid forming (PAF) and non-acid forming (NAF) material as shown in Table 16-11. The Çöpler Mine has adopted a waste rock management plan to ensure proper disposal of PAF material as it is encountered during the ore control process.

Table 16-11 Waste Rock Geochemical Classification (SRK, 2012c)

Lithology	Sulfide Sulfur (%) Cut-off grade	Waste Rock Groups	Descriptions
Diorite	0.8	PAF / High Sulfide Diorite	Diorite rock with Sulfide S ≥ 0.8%
		NAF / Low Sulfide Diorite	Diorite rock Sulfide S < 0.8%
Metasediment	0.8	PAF / High Sulfide MTS	Metasediment rock with Sulfide S ≥ 0.8%
		NAF / Low Sulfide MTS	Metasediment rock with Sulfide S < 0.8%
Limestone / Marble	2	High Sulfide LMS	Limestone with Sulfide S ≥ 2%
		Low Sulfide LMS	Limestone with Sulfide S < 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 WRSA facilities. The key findings from the SRK report suggests that all WRSA facilities, except one, at Çöpler have a neutralizing potential (NP) to acid potential (AP) ratio of greater than 20:1; indicating that the Çöpler Mine has excellent neutralization capacity for ARD. The one exception to this was the West WRSA which was estimated to have a NP:AP ratio between 1 and 3. It was recommended that Alacer optimize the WRSA construction sequencing in order to take advantage of the neutralization potential of the other WRSA facilities by blending higher quantities of NAF material into the West WRSA. Alacer 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 which resulted in all samples generating near neutral and slightly alkaline paste pH as shown in Figure 16-8.

Figure 16-8 SRK Paste pH Test Results

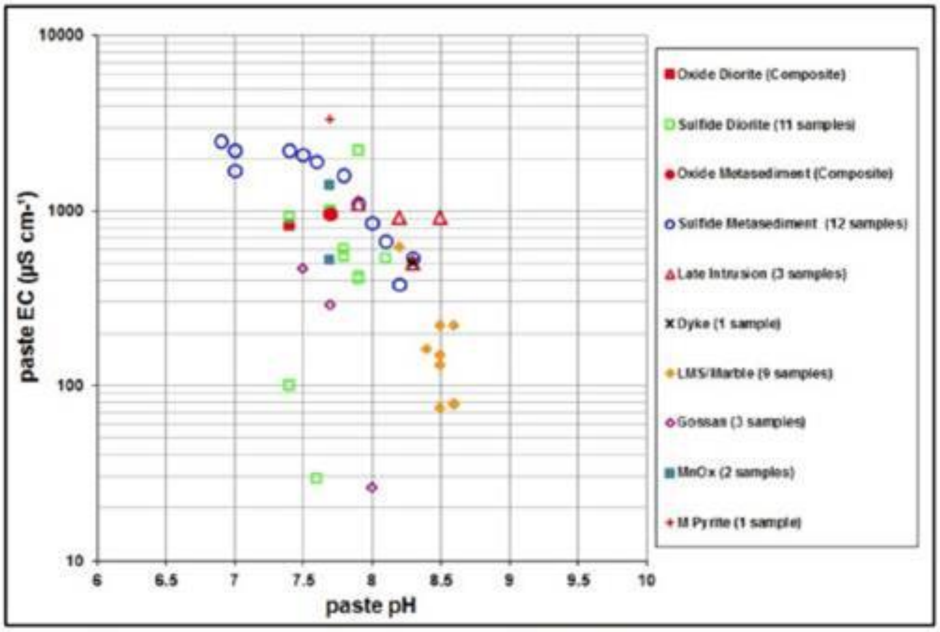
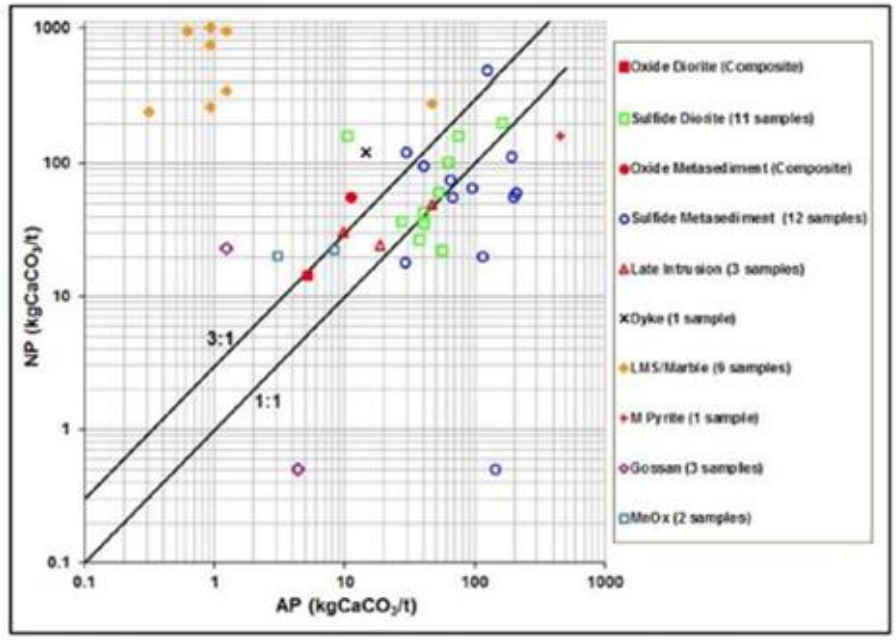


Figure prepared by SRK, 2016.

In regard to future acid potential,

Figure 16-9 shows the relation of NP to AP. A large majority of all samples taken reside above the NP:AP 1:1 boundary. The remainder of 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. Only two of the samples register well below the 1:1 NP:AP ratio.

Figure 16-9 SRK NP:AP Waste Rock Analysis



In regard to long-term acid potential, Figure 16-10 shows the results of a 45 week kinetic test with resulting pH values in the range of 5.5 to 8.5.

Figure 16-10 SRK 45 Week Kinetic Test

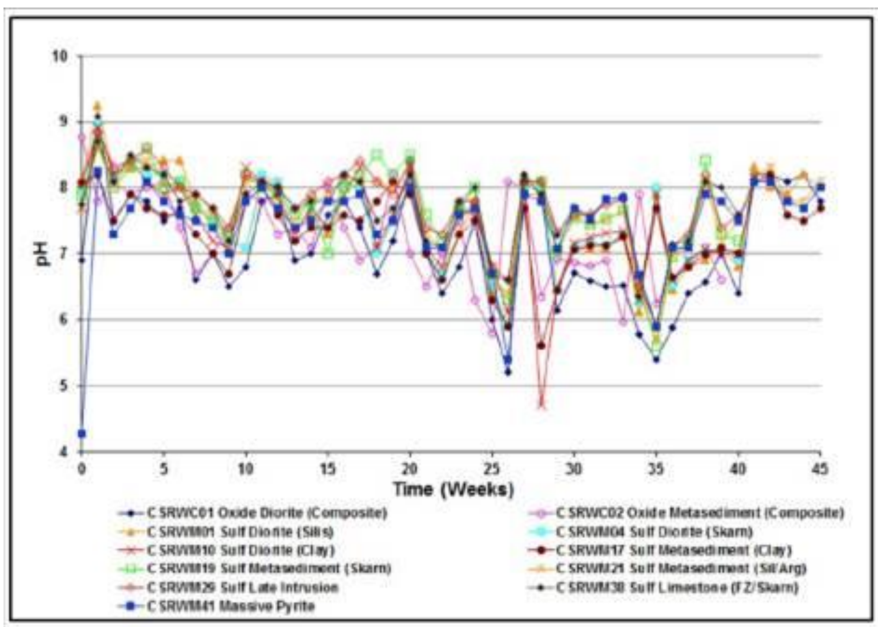


Figure prepared by SRK, 2016.

16.6 Mining Operations

16.6.1 Safety

In addition to the Anagold Health and Safety Department, which is responsible for conducting safety training and procedure implementation at the mine site, the mining contractor employs their own safety team, which is responsible for ensuring that all Anagold safety procedures are followed by the contractor's employees. Regular safety training and safety meetings are conducted with all employees at the Çöpler Mine. All employees are provided with appropriate Personal Protective Equipment (PPE) and task training for the job to which they are assigned.

16.6.2 Drilling and Blasting

Drilling operations are carried out by the mining contractor with the use of eight Atlas Copco Rock Drills for both production and presplit blast hole drilling. Blast holes are loaded with bulk ANFO delivered by the explosive supplier in 25 kg bags. The explosive supplier supplies bulk ANFO, non-electric detonators, boosters, and other blasting accessories to the contract miner's pit blasting crew. Explosives are stored on site at the underground explosive storage facility located directly south of the Manganese pit.

Production blast holes are drilled and loaded based on the following criteria:

- 102 mm hole diameter
- 5.5 m drill depth (0.5 m sub drill), except where a catch bench exists below.
- Staggered pattern
- 3.75 m spacing
- 3.25 m burden
- 18 kg ANFO per hole
- (1) 0.5 kg booster
- 2.75 m stemming
- Timing varies based on pattern configuration and amount of burden relief.

Pre-split blast holes will be drilled and loaded based on the following criteria:

- 89 mm hole diameter
- Inclined at 75°
- 6 – 16 m drill depth depending on design bench height (1 m sub drill)
- 0.8 m spacing along design crest
- (8) 0.5 kg boosters evenly spaced in the hole column
- No stemming
- Fired simultaneously in advance of production blast.

16.6.3 Grade Control and Ore Control

All grade control and 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 oxide or sulfide ore” or “potential oxide or sulfide waste” based on grade control data from the bench above and the reserve model prediction. A 10 m outside buffer is then applied to the potential ore areas to ensure proper sample density. All potential ore blast holes are sampled for AuFA (fire assay for gold). Approximately 50% of potential ore blast holes are sampled for AuCN (cyanide soluble gold), total carbon, and total sulfur. Additionally, all potential sulfide ore blast holes are sampled for sulfide sulfur. 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 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 drill hole identified (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. Figure 16-11 details the blast hole sampling procedure.

Figure 16-11 Çöpler Blast Hole Sampling Procedure

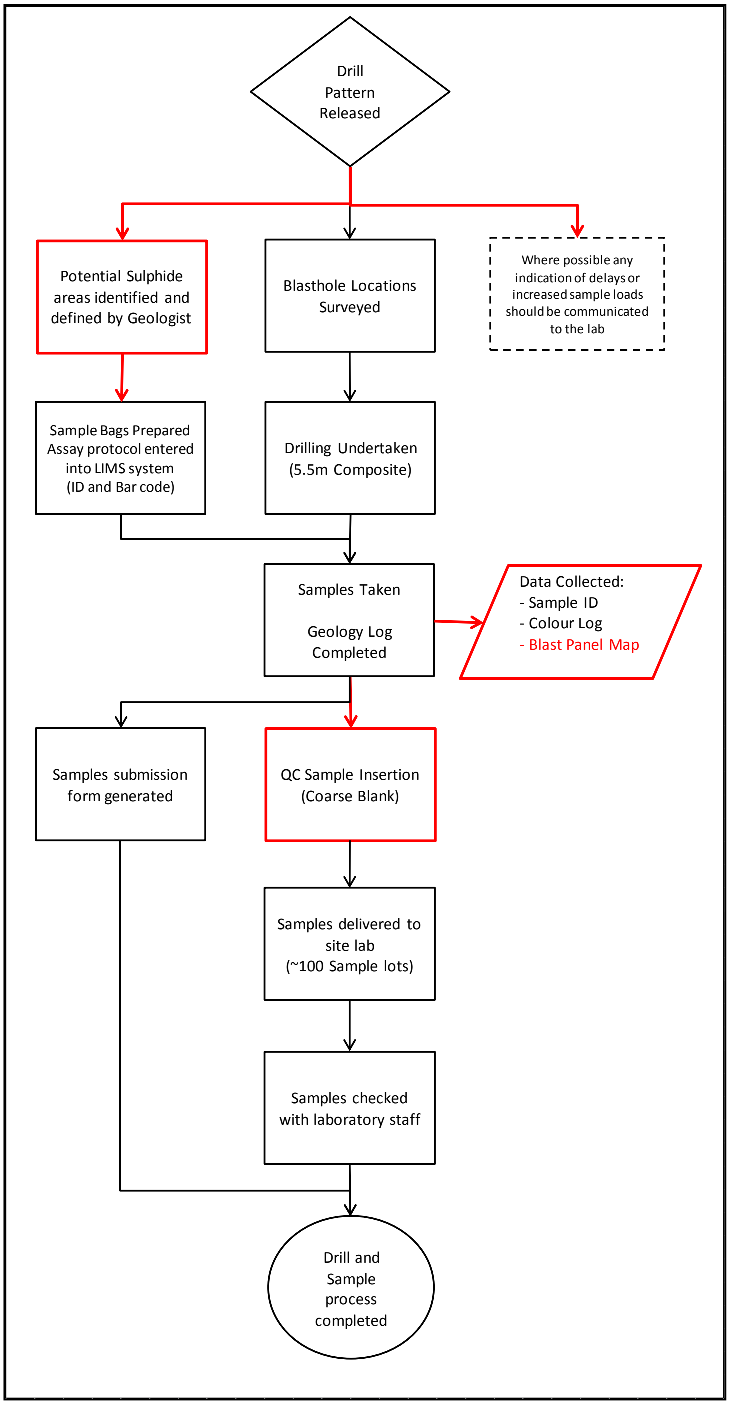


Figure prepared by Anagold, 2016.

Assay results are uploaded to the ore control database with reference to each specific drill hole ID. The assay results are then estimated in a Vulcan block model (block size 3 m x 3 m x 5 m) using ordinary kriging to estimate ore grade and type. The ore control geologist will then use Vulcan to digitize mining shapes with a minimum SMU of 3 m width and minimum tonnage of 500 t. These mining shapes are then sent to the survey group for layout in the mine using color coded flagging under the supervision of the ore control geologist.

16.6.4 Loading and Hauling

Loading and hauling of ore and waste is performed by the mining contractor. Primary production loading operations utilize ten Caterpillar 374D excavator-back hoes with 4.6 m^3 buckets. Over 100 Mercedes Axor 36 t haul trucks are used as the primary haulage equipment. Ore and waste material is routed by color coded flagging set out by ore control technicians and then delivered to the appropriate WRSA, stockpile, or crusher.

16.6.5 Ancillary Mine Equipment

Ancillary mine equipment is required to support mine operations. This equipment is primarily made up of two vibratory compactors, four Caterpillar 14H graders, four Caterpillar D8 and one Caterpillar D9 dozers, four water trucks, four Caterpillar 980 wheel loaders, and two Volvo 35 tonne articulated haul trucks. Additional equipment includes light plants, fuel trucks, and maintenance vehicles.

16.6.6 Ore Stockpile Rehandle

Oxide ore that is unable to be directly dumped into the crushing circuit is placed in the appropriate stockpile for processing at a later time. Oxide ore is typically segregated dependent on clay content and average 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. Upon plant commissioning, sulfide ore will be 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 will be rehandled by a Caterpillar 980 loader from the crushing pad into the crushing circuit. Oxide and sulfide ore will use separate crushing circuits for the processing of each ore type.

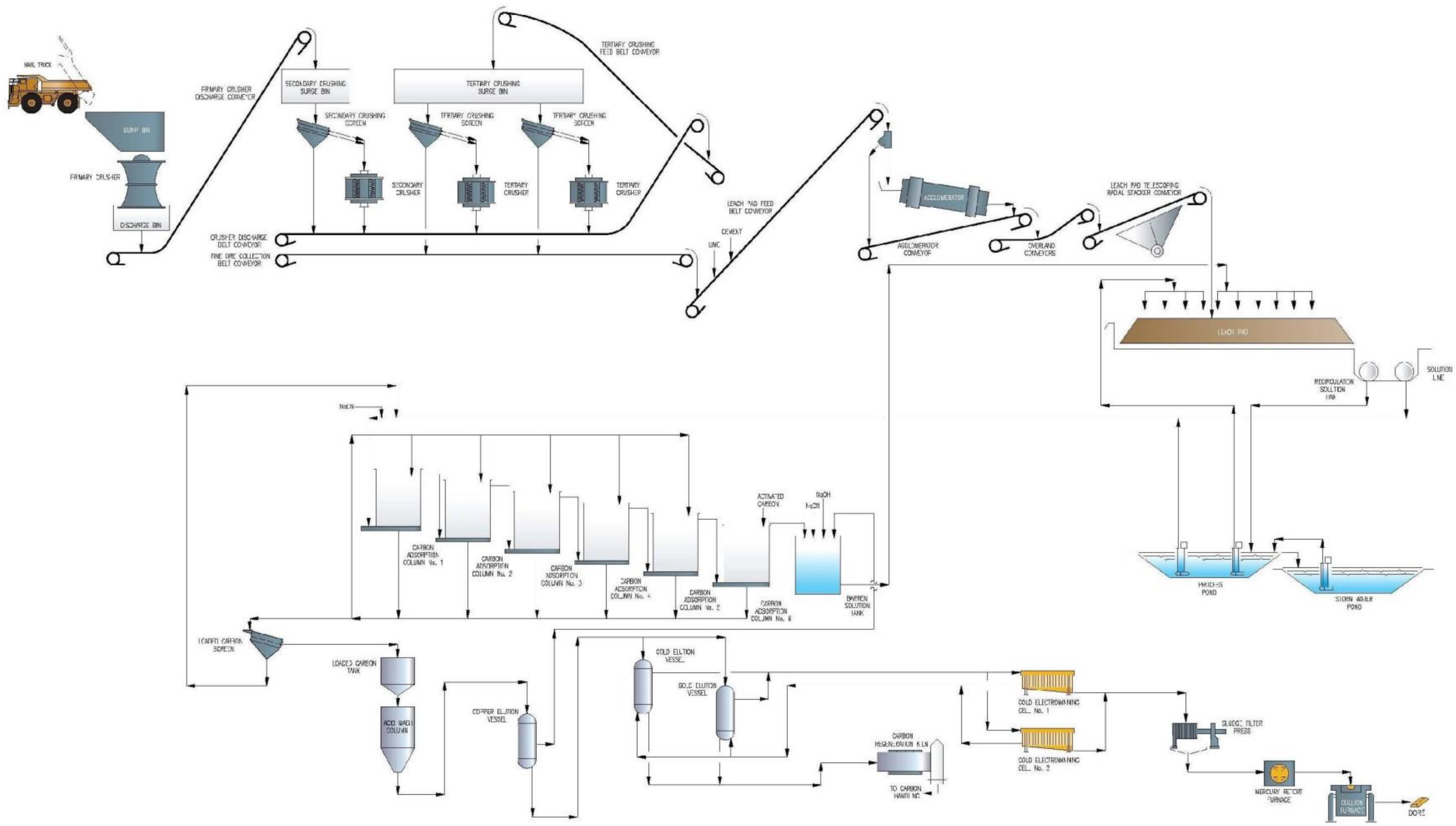
17.0 RECOVERY METHODS

17.1 Oxide Ore Heap Leach Processing

An oxide heap leaching process was constructed at the site between 2008 and 2010. The heap leaching and associated facilities were commissioned in the second half of 2010 and initial gold production was achieved in the fourth quarter of 2010. The process was designed to treat approximately 6.0 Mtpa of ore by three-stage crushing (primary, secondary and tertiary) to 80% passing 12.5 mm, agglomeration (with lime and water) 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 using high temperature, pressure elution process, and electrowinning, retorting and melting of the resulting product to yield a doré (containing gold and silver) suitable for sale. Carbon is regenerated using acid washing and reactivation in a rotary kiln, and the carbon is recycled back to the CIC system. Subsequent to commissioning of the plant, a SART plant has been constructed and commissioned to remove copper from the leaching solution and to regenerate cyanide. The process operates intermittently, on an as-needed basis. The process flowsheet is summarized in Figure 17-1.

Since commissioning through the end of December 2015, an estimated 1,734 koz of gold were placed on the heap contained within approximately 35.2 Mt of ore at an average grade of 1.52 g/t Au (0.049 oz/t). At the end of December 2015, a total of approximately 1,078 koz had been produced as bullion. It is noted that approximately 25% of the material placed onto the leach pad between 2010 and end of 2014 was placed as run-of-mine ore (no crushing or agglomeration).

Figure 17-1 Heap Leach Process Flowsheet



17.2 Proposed Sulfide Ore POX Processing

The basic flowsheet is shown in Figure 17-2 and comprises:

- Crushing and ore handling
- Grinding
- Acidulation
- Pressure oxidation
- Iron/arsenic precipitation
- CCD
- Gold leach, carbon adsorption and detoxification
- Carbon desorption and refining
- Neutralisation and tailings
- TSF

Figure 17-2 Çöpler Block Flow Diagram

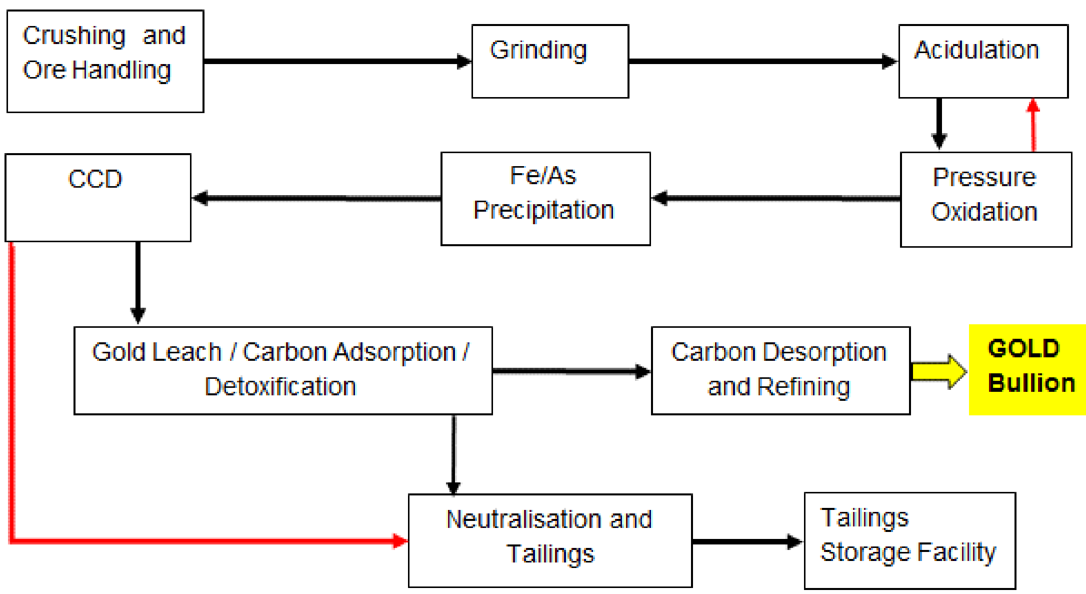


Figure prepared by Amec Foster Wheeler, 2016.

The following sections describe the process as it will be built and operated.

17.2.1 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 (aperture size 500 mm), and a mobile rock breaker is used to break apart large lumps of ore retained on the grizzly.

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. The product topsize is based on conveying, transfer chute and SAG (semi-autogenous grinding) mill feed chute requirements.

Discharge from the sizer drops down a chute onto the sizer discharge conveyor. When the sizer is off-line for maintenance, the 500 mm grizzly is replaced by a 300 mm aperture fine static grizzly. The sizer is rolled away on rails and a bypass chute is installed to transfer fine ore directly onto the sizer discharge conveyor from the apron feeder discharge.

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.

Grinding media (steel balls) is continuously added to the SAG mill feed conveyor by the SAG mill automatic ball charger to make up for consumption within the SAG mill.

17.2.2 Grinding

The SAG milling stage consists of a high aspect SAG mill with water cannon pebble recycle. The SAG mill grinds the crushed ore (by tumbling large ore particles and steel grinding media) to produce a discharge particle size distribution P80 of approximately 1400 µm.

The mill has an installed power of 2500 kW and dimensions of 6.71 m inside shell diameter and 3.35 m effective grinding length (EGL). Large ore particles are retained in the 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 (typically +10 mm -30 mm) 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 P80 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.

As its name suggests, the ball mill grinds the relatively fine cyclone underflow solids by tumbling steel media (grinding balls) alone. Media is replenished at regular intervals with a crane and kibble arrangement to maintain the power draw of the mill to close to the maximum level. The ball mill discharges through a trommel screen, designed to capture and discard small spent grinding media (steel scats) and the trommel undersize falls into the grinding cyclone feed pump box. The ball mill is a rubber lined overflow discharge unit fitted with a 3600 kW drive. The mill operates at a pulp density of 72% solids w/w.

The slurry product from the grinding circuit, trash screen undersize, is thickened in a high rate thickener and excess water reports to the thickener overflow for immediate reuse within the grinding circuit. The thickened slurry discharging from the thickener underflow is pumped to the grinding thickener underflow storage tanks.

17.2.3 Acidulation

The grinding thickener underflow storage tanks provide process surge and effectively decouple the upstream crushing and grinding circuits from the downstream hydrometallurgical circuit. If the acidulation feed tanks reach their high level limit then ore feed to the grinding circuit will be stopped. If the tanks are approaching their low level limit then the grinding 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 grinding circuit 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 if 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 having a total residence time of two hours when treating the whole grinding thickener underflow. 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 favors the formation of an iron mineral reaction product which exhibits better settling behavior in downstream thickeners (hematite favored over jarosite), while also reducing the potential for excessive CO2 gas evolution and gypsum scaling in the POX autoclaves.

The recycling of acidic decant thickener overflow solution is limited to a maximum flowrate of 1250 m^3/h. 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. In the thickener the slurry is thickened to a target underflow density of 40% solids. 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.

17.2.4 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 minimize 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 1, 2 or 3. These pumps operate in a 2 duty/1 standby configuration. 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 95ºC 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 150ºC 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: POX T1 feed booster pump 1 feeds POX T1 feed pump 1, and POX T1 feed booster pump 2 feeds POX T1 feed pump 2. Both pumping trains operate in series. When a pumping train is taken of-line to be maintained the other train has the capacity to ramp up throughput to achieve the target feed rate. The feed booster pumps are centrifugal slurry pumps designed to provide adequate NPSH to the high pressure POX feed piston pumps. The feed pumps discharge at a higher pressure than the autoclave to ensure positive flow into the autoclave.

If one full autoclave train is off line the remaining autoclave train can operate at 150% of normal capacity, provided both its feed pumping trains are operating.

A horizontal multi-compartment autoclave is used to oxidize 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 CO2.

Key autoclave design parameters include:

- Operating temperature 220°C
- Operating Pressure 3150 kPa.g
- Oxygen over pressure 350 kPa
- Residence time 60 minutes

Slurry discharges from the autoclave through a severe service let down valve to the POX T1 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, approximately 155ºC and 440 kPa.g respectively.

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 POX T1 venturi scrubber for treatment prior to discharge.

Slurry discharges from the HT flash vessel through a severe service let down valve to the POX T1 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 POX T1 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 POX T1 Venturi scrubber to remove entrained acidic slurry droplets.

Demineralized water is used in the POX circuit for steam production and for seal water. Demineralized water supply and steam production are vendor packages.

Flashed slurry is pumped from the LT flash vessel by POX T1 decant thickener feed pumps 1 and 2. The decant thickener was described previously and the decant thickener underflow is feed to iron/arsenic precipitation.

17.2.5 Fe/As Precipitation

Iron/arsenic precipitation uses limestone slurry addition to the decant thickener underflow slurry to neutralize the free acid and raise the pH to about 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 oxidize 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. Each tank has nominal retention time of 90 minutes. The treated slurry overflows from the second iron/arsenic precipitation tank to the CCD 1 Mix Tank.

The low pressure air and CO2 generated during the limestone neutralization 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 & 2 are vented via the iron/arsenic precipitation tank fans 1/2 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.

17.2.6 CCD

Counter current decantation 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 CCD1 overflow. The slurry discharging from CCD2 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 neutralization circuit. The CCD thickener 1 underflow pump transfers the thickener underflow slurry to CCD2 mix tank. Process water is added in the CCD2 mix tank as wash solution to wash the solids. Diluted flocculant solution is added in the CCD1 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.

17.2.7 Copper Recovery Allowance

The copper recovery circuit which had been included in the scope of the project in the FS has been removed from the flowsheet. Space for copper recovery equipment has been reserved in the plant layout for future inclusion if required. The CCD1 overflow solution containing a majority of the soluble metals leached in the POX circuit is now sent to the neutralization circuit for precipitation by lime neutralization.

17.2.8 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. The pre-leach tank has a residence time of 10 minutes and is used to raise the pH of the slurry to between pH 10 and 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 6 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 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 12 hours. Each tank contains activated carbon at a concentration of 5 g/L to adsorb the leached gold contained in solution. Slurry flows by gravity from tanks 1 to 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 recessed impeller pumps to protect the carbon. Each tank has an inter-stage screen 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 1 hour. Air and sodium metabisulfite are added to the circuit to destroy the residual cyanide down to a concentration of less than 5 ppm CN_{WAD}. Residual copper in the slurry catalyzes the cyanide destruction process.

17.2.9 Carbon Desorption and Refining

The carbon desorption method selected is a split AARL with cold cyanide strip for copper. 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 a 3% w/w nitric acid solution to remove loaded impurities such as calcium.

After acid washing, the carbon is rinsed with water before being subjected to a cold cyanide wash using a 3% w/w cyanide solution to remove loaded copper from the carbon. The carbon is then rinsed with water prior to commencement of gold elution. The resulting cold cyanide eluate solution is transferred to the existing oxide plant SART circuit for recovery of the contained copper and cyanide.

A pre-soak solution containing 3% w/w cyanide and 3% w/w caustic is added to the elution column prior to commencement of the eluent recycle for stripping of gold from the carbon. The elution is conducted at 110°C using indirect heating from a diesel fired oil heater and heat exchanger.

Pregnant eluate is collected in the pregnant eluate tank and pumped through three electrowinning cells with gold metal plated out onto stainless steel cathodes. Smelting of gold recovered from the stainless steel cathodes is conducted in the existing ADR circuit.

Desorbed carbon from the elution column is regenerated through a horizontal diesel fired rotary kiln to remove organic material loaded onto the carbon. Carbon discharging the kiln enters the carbon transfer vessel from where plant air is used to pressure transfer the regenerated carbon to tank 6 of the carbon adsorption circuit via the barren carbon dewatering screen.

17.2.10 Neutralization and Tailings

Slurry from cyanide destruction and the CCD 1 thickener overflow solution are neutralized with lime to precipitate residual metals in solution. Air is added for the oxidation and removal of ferrous iron and manganese.

Normally the two neutralization tanks operate in series. Discharge from the neutralization feed box gravity flows into neutralization tank 1 prior to overflowing into neutralization tank 2. Discharge from neutralization tank 2 gravitates into the tailings thickener mix tank. Low pressure air, which is required in the neutralization tanks in the neutralization process, is supplied by the low pressure air compressors.

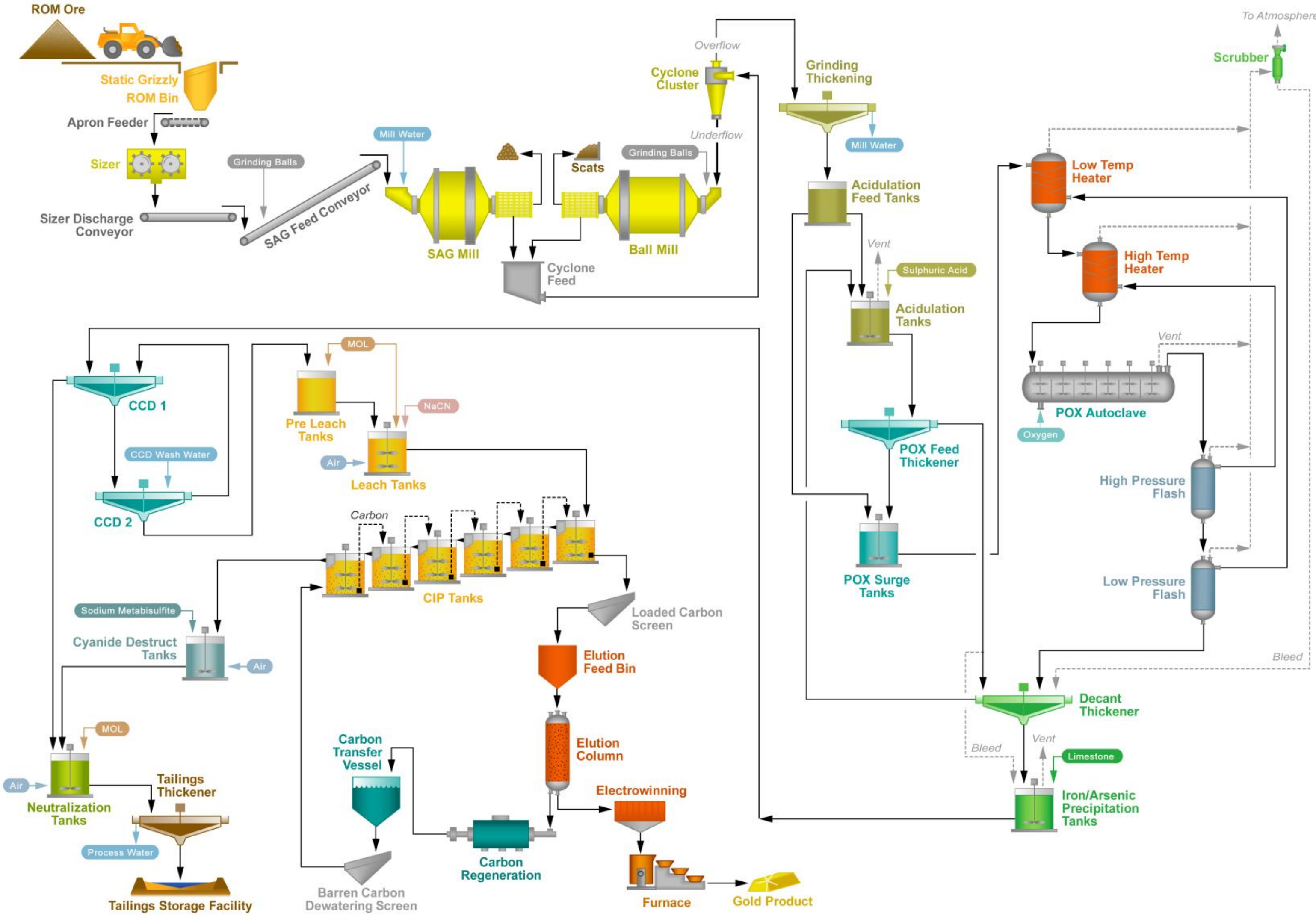
The first neutralization tank is equipped with a sodium metabisulfite addition system and this allows it to be used for the detoxification step when the normal detoxification tank is bypassed for maintenance or descaling. Both neutralization tanks can also be bypassed as required to allow for maintenance.

The discharge slurry from neutralization 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. A two stage cross-cut sampler is provided to take representative samples from the tailings thickener underflow stream. The discharge slurry from the tailings tank is pumped to a TSF on a continuous basis via the 4.3 km long tailings pipeline.

In the event of emergency power outage, tailings slurry from the pipeline can be dumped into the tailings dump pond to prevent sanding of the pipeline. Following recovery of power supply, slurry and water in the tailings dump pond is recovered via a sump pump and sent to the tailings thickener mix tank by the tailings dump recovery pump.

A schematic flowsheet of the process is shown in Figure 17-3.

Figure 17-3 Çöpler Sulfide Process Flowsheet



17.2.11 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 maximize 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 returned back to the process water system tank via a tailings water reclaim pumps.

17.2.12 Reagents

There are nine major reagents used in the process plant, listed as follows:

- Sulfuric acid
- Limestone
- Sodium hydroxide
- Flocculant
- Sodium metabisulfite
- Milk of lime
- Sodium cyanide
- Nitric acid
- Antiscalant

These are delivered in bulk tankers, containers or bags and there is storage on site ranging between seven days and 60 days depending on the reagent. Any reagents that require dilution or mixing prior to use are prepared onsite on a batch wise basis as required. Spillage containment systems are in place, with sump pumps returning spillage to mixing tanks or to appropriate parts of the operating plant.

17.2.13 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
- Oxygen – supplied from Air Liquide owned and operated oxygen plant under a gas supply agreement.
- Raw water
- Fire water
- Potable water
- Process water
- Diesel fuel

These utilities are reticulated throughout the process plant to their end user.

18.0 PROJECT INFRASTRUCTURE

18.1 Introduction

The design development of new facilities and/or changes to existing infrastructure in the feasibility phase of the project is discussed in detail in the following sections. The facility infrastructure has been divided into several distinct areas: buildings, water and sewage, bulk fuel storage, power supply, communications, site roads, plant fire protection system, and the plant lighting system.

Structure placement for the process plant facility is based on the best available topographic and geotechnical information provided by Alacer and Golder, respectively. While consideration has been made for the existing geotechnical situation, the discovery of other unknown conditions, such as during further geotechnical exploration or construction, could impact the extent of earthwork required in the affected area.

An infrastructure layout plan is included as Figure 18-1.

Figure 18-1 Infrastructure Layout Schematic

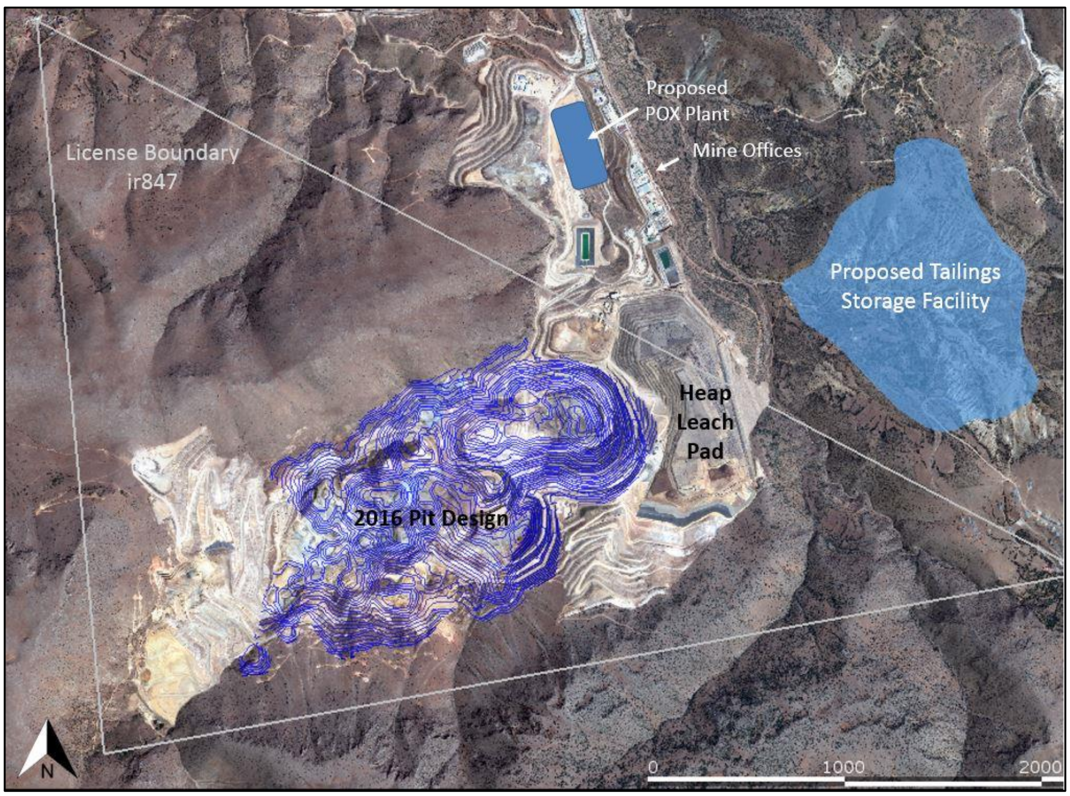


Figure prepared by Alacer, 2016.

18.2 Plant Site Geotechnical Considerations

A geotechnical investigation exploration program was completed to support the feasibility level design of the various sulfide plant buildings and structures by Golder (Golder, 2014b) and was further updated based on the Amec Foster Wheeler revised plant layouts (Golder, 2015c). Proposed facility foundation systems will be constructed to bear on either native conglomerate or limestone bedrock materials, or engineered structural fill. The proposed sulfide plant and related crushing and grinding facilities are anticipated to be constructed primarily on shallow foundation systems. Native subsurface conditions within the proposed sulfide plant site generally consist of clayey gravel, silty clay and clayey silt overburden with a thickness ranging from 0 to approximately 10 m below existing grades. The overburden material, where encountered, generally overlies conglomerate and limestone bedrock. Existing uncontrolled waste rock fill overlying limestone bedrock is generally encountered towards the south and western portions of the sulfide plant site. The existing uncontrolled waste rock fill was placed during previous construction of the Northeast WRSA. The Northeast WRSA has been extended north through Çöpler Creek valley as a valley fill and currently covers the original Çöpler Creek. Thicknesses of existing uncontrolled waste rock fill vary up to about 47 m. Structural rockfill placed during construction of the existing HLF storm pond overlies limestone bedrock and uncontrolled waste rock fill in the areas immediately surrounding the pond. Golder has continued to work with Anagold and Amec Foster Wheeler to provide specific recommendations and advice on geotechnical engineering.

Construction of the sulfide plant pad for the grinding structures will require the upper portion of the existing laydown area to be excavated to reach proposed grades for the grinding structures. Pads for the oxygen plant, the leach tanks and tailings thickeners will be constructed by cutting and filling to reach finished grade elevations. The primary crusher location is sited within the limits of structural rockfill placed during the construction of the existing HLF storm pond. Several foundations such as conveyor bents, bend frames, drive stations, gravity take up towers, transfer towers, HV switchyard, maintenance building, primary crusher, crusher electrical building, and reagents storage building may be located in areas where significant depths of existing undocumented waste rock fill were placed during construction of the existing Northeast WRSA. In addition, the western portion of the proposed sulfide plant abuts the eastern portion of the Northeast WRSA where several future proposed structures are sited over areas where significant depths of undocumented waste rock fill have been placed. The Northeast WRSA has been extended north through Çöpler Creek valley as a valley fill and currently covers the original Çöpler Creek. Mitigation for potential settlement in the undocumented waste rock fill in the western portion of the site is being mitigated by the application of preloads, pre-wetting, or a combination of both over several of the key structures including the primary crusher and the HV switchyard area.

For structures less sensitive to settlement that are located on deep deposits of uncontrolled waste rock fill, the structures should bear on a minimum 3 m of structural fill with a settlement monitoring system and contingencies for structural leveling or repair.

18.3 Buildings

Below is a summary of the existing infrastructure and additional infrastructure to support the Çöpler Sulfide Expansion Project.

18.3.1 Existing Infrastructure

The existing site infrastructure supporting the existing oxide heap leach operation that may also be used or supplemented to support the Sulfide Expansion Project includes the following:

- Site security gate and guard station
- Site administration building

- Site warehouse
- 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, & offices
- Site raw water wells, pumping system and storage tanks
- Site potable water treatment and distribution system
- Two sanitary waste water collection and treatment systems

18.3.2 Administration Building

No provisions have been made in the FS for space or for construction of an administration building. However, there are plans to leave the construction office for an administration building after POX plant project is complete. Details on planned office space to support the Sulfide Expansion Project are included in Section 18.3.11.

18.3.3 Maintenance Building

Consideration was given to equipment spacing and maintenance laydowns within the grinding and POX buildings to facilitate equipment maintenance.

The maintenance building will be an insulated pre-engineered structure 47.5 m long x 26 m wide x 13.4 m high with 20 t and 40 t bridge cranes. Adjacent the maintenance building is a two-story building which includes ablutions, showers, ground floor locker room, office, meeting room, cloak area and kitchenette. Common to all areas will be electrical power, lighting (standard and emergency), potable water, plumbing, fire and gas (F&G) detection and fire protection. All areas will have separate HVAC systems. In the maintenance area natural ventilation and heating will be used to keep the temperature to a minimum of 15ºC. Cooling and heating will be achieved with air conditioners in the other areas. The entire maintenance area will be fenced.

18.3.4 Warehouse

The warehouse will be an insulated pre-engineered structure. Adjacent to the building there is a one story building which includes four offices, archive room, ablutions and tea room. Common to all areas will be electrical power, lighting (standard and emergency), potable water, plumbing, F&G detection, fire protection and security. All areas will have separate HVAC systems. Cooling and heating will be achieved with air conditioners in the other areas. The entire warehouse will be fenced including the outside storage area.

18.3.5 Laboratory Facilities

Alacer has recently upgraded the existing assay laboratory on site. The laboratory is designed with space for the additional laboratory equipment required by the Sulfide Expansion Project. Most major metallurgical samples and composites taken at the sulfide plant will be analyzed or tested in this assay laboratory.

A small laboratory will be placed on top of the leach tanks. This facility will be used to process samples for monitoring and control of the downstream processes and carbon management.

18.3.6 Primary Crushing Control Room

The central control room will house distributed control system (DCS) input/output (I/O) cabinets, DCS servers, DCS engineering station and operator stations. A small remote control room will be located at the primary crusher. The primary crushing control room will be an insulated pre-fabricated building with a size of 2.3 m long x 2 3 m wide x 3 m high. The control rooms will include the following utilities; HVAC, electrical power, lighting (standard and emergency), and a communication system. A DCS operator's station will be located within the primary crushing control room.

18.3.7 Grinding Building

The grinding building will be an insulated engineered building. The primary grinding circuit control will be performed from the central control room. A secondary operator station for local control will be located within the grinding building. The grinding building will include the following utilities: electrical power, lighting (standard and emergency), potable water, plumbing, fire protection and a communication system. DCS I/O cabinets will be located in the electrical room.

18.3.8 POX Building

The POX building will be an insulated engineered building. The POX building will include the following utilities: electrical power, lighting (standard and emergency) and fire protection. Adjacent to the building there is an enclosure which houses the POX steam boiler utilities. A code check will also be performed on the building design during engineering. Ventilation will be provided in the POX building. Toilet facilities will also be provided in the maintenance shop. The maintenance shop and the metallurgical laboratory will be located on the first floor at the north end of the building. The electrical room will be located on the second floor at the north end of the building. The central control room will be located on the third floor at the north end of the building.

18.3.9 Carbon Desorption Building

Acid washing and elution (or desorption) of loaded carbon and regeneration of the resulting barren carbon will be conducted within a new carbon elution building to be erected adjacent to the CIP circuit. This will be a clad, but uninsulated building. Lines and equipment that may be subject to freezing will be appropriately heat traced and insulated. Within this circuit, the gold recovery process will proceed to the stage of production of pregnant electrolyte and cold-elution eluate if copper removal from carbon proves necessary.

18.3.10 Refinery

Two new electrolyte storage tanks will be installed adjacent to the existing oxide ARD building. These will receive pregnant electrolyte from elution batches alternately while one is being filled the other is being drained by pumping the electrolyte to the electrowinning cells.

A new gold electrowinning circuit will be installed adjacent to the existing secure oxide gold room. This new electrowinning circuit will include three electrowinning cells and an off gas collection and scrubbing system for those cells. It is anticipated that the vast majority of construction of this new electrowinning circuit can be conducted outside of the existing gold room security access zone and protocols. As a last step prior to commissioning, doorways into the new electrowinning circuit will then be cut through the building wall and the new circuit integrated into the existing security zone.

18.3.11 Office Space

Office spaces will be provided in the central control room and other areas of the plant as required. There will also be facilities provided for permit issuing offices and other work areas to support the efficient management of maintenance shutdowns. The office spaces will include the following utilities: HVAC, electrical power, lighting (standard and emergency), potable water, plumbing, toilets, an available communication system and furniture.

18.3.12 Electrical Buildings/Power Distribution Centers (PDC)

The electrical buildings will be distributed throughout the process plant and will be insulated buildings with a size of approximately 30 m long x 10 m wide x 6 m high. The buildings will include the following utilities: HVAC, electrical power (standard and emergency), fire protection, and a communication system. DCS I/O cabinets will be installed in the buildings.

18.3.13 Other Buildings

Other buildings that are required to support the sulfide process plant facility are listed as follows:

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
Raw Water Pump Building
Limeslaking (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
Elution Building
CIP CCD Ablutions Block
Pump Shelters with Monorails
Carbon Elution Building - Electrical Room
Raw Water Bores P/P House & Electrical Building
Gatehouse
Fire Water Pump House
Community Relations Center
Raw Water Wells

18.4 Water and Sewage

18.4.1 Fresh Water Supply

Fresh water is being supplied by existing wells to the site at a rate of 66 L/sec supporting the existing oxide heap leach operation. Additional wells will be provided to support the requirements of the combined existing oxide heap leach and new sulfide processing facility. The site is currently serviced by three fresh water wells. Two of these wells were installed in 2013 to replace a well inundated by the new lake created by the new dam south of the Çöpler Mine.

A new raw water storage tank will be constructed and tied into the existing raw water system. The changes to the raw water system will support the demands of the new sulfide process equipment and the fire water requirements. The system will continue to support the existing oxide heap leach operation while protecting the new sulfide processing facilities.

18.4.2 Potable Water Treatment

The site is currently 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 existing site potable water distribution system with no intermediate storage tank.

The existing potable water system will be integrated with a new potable water treatment system, similar to the existing system. The new system will support the demands of the new sulfide processing facilities and have an additional potable water storage tank. This tank will receive the combined treated water flow from the new and existing treatment systems. The new water tank will deliver treated water to the existing potable water distribution system and to the new distribution system for the Sulfide Project facilities.

18.4.3 Waste Management

Waste will be generated from multiple sources such as human waste, food spoilage, and process and maintenance wastes.

New holding tanks will be provided to support the new sulfide processing facilities. Septic trucks will deliver the waste to the existing sanitary waste water system.

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. Any offsite disposal of waste materials will be provided by the Owner.

18.5 Bulk Fuel Storage

No provisions have been made in the FS for space or the construction of an additional bulk fuel storage area to support light vehicle fueling. The existing light vehicle fueling station will be used to support the sulfide plant operations in addition to the existing heap leach operation.

New bulk fuel storage will be provided to supply fuel for the steam generators associated with the new sulfide processing plant, for the emergency diesel generator and for the elution heater. Dedicated daily fuel storage tanks will be provided in the POX steam generators area, and in the elution building area. Alternative fuels for these applications are being investigated by the Owner.

18.6 Power to Site

The existing 154 kV line will provide power to the mine and plant extensions.

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

18.7 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 will be applied to feed critical electrical users automatically in the event of a utility outage.

Generators are diesel fueled with a minimum of eight hours of diesel storage based on generators operating under full load.

18.8 Communications

The Project requires 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. The Project is providing all networking hardware and cabling for these networks within the sulfide expansion works.

Single mode (SM) fiber is dropped at the listed locations as well as Copper cabling in Table 18-1.

Table 18-1 Fiber Drop-off Locations

Primary Crushing	Limestone Preparation	Oxygen Plant
Grinding/Thickening	Warehouse	CCD Thickeners
Grinding Building	Compressor Building	Tailings Thickeners
POX building	Leach tank Lab	Elution
POX thickener	Reverse Osmosis Unit	Electrical Buildings
Decant Thickener	ADR Building	

SM fiber is provided between the tailing ponds and the main plant area. Drops are provided at the electrical building/power distribution center (PDC), at the tailings pipeline drain valves location and at the tailings pond electrical room.

18.9 Site Roads

The Sulfide Expansion Project will have access provided via the existing main access road and newly constructed sulfide plant roads. A site Road Plan will be developed during detailed design and will require detailed information and cooperation from site personnel.

Newly constructed roads for the Sulfide Plant Expansion Project will be integrated into the existing road infrastructure where practical.

Generally, site roads will have an overall width of 6 m and will 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 to be compacted hardstand surfaced with 100 mm wearing course and cross-sloped to provide positive drainage.

18.10 Plant Fire Protection System

A separate plant fire protection system will be provided for the sulfide facility. The existing raw water tank will be re-purposed to become a dedicated fire water storage tank. The raw water storage tank is currently 14 m diameter x 12.7 m tall. A new diesel driven fire pump will be installed in the existing fire pump house to deliver the increased demand of new facility.

A combined sprinkler, hose reel and hydrant underground piping system will be provided for the active fire protection of the facility.

A gas based fire suppression system will be used in the main control and electrical building.

18.11 Plant Lighting System

Lighting levels will be designed to meet IES standards. Estimates included exterior lighting, building lighting, and interior building lighting. Fixtures selected will be of the energy-efficient type.

18.12 Heap Leach Facility

The current HLF includes the leach pad and collection ponds that consist of process ponds and a storm pond. The current leach pad consists of three phases and was designed to accommodate approximately 34 Mt of ore heap with a nominal maximum heap height of 100 m above the pad liner. The Phase 4 pad expansion to the south and southwest is scheduled for completion at the end of 2016 and will increase the pad capacity to approximately 58 Mt of ore heap.

The existing natural grades in most of the Phase 4 pad area are relatively steep and range from approximately 20% to 55%, with the steeper grades in the southern portion of the pad. The grades are less steep in the vicinity of the existing mining shop in the pad's northwest corner.

The ore heap configuration on the ultimate Phases 1-4 leach pad has an approximate capacity of 58 Mt using a stacked ore heap density of 1.8 t/m^3, as provided by Anagold. The heap will rise to Lift 32 with a top surface elevation of 1382 m, and the maximum heap height will be 100 m above the pad liner.

The heap is planned to be 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.

18.13 Tailings Storage Facility

The TSF for the Sulfide Expansion Project has been designed to provide capacity for the disposal of 45.9 Mt of mill tailings in a fully lined tailings impoundment over an approximate 20-year mine life from the commissioning of the TSF from third quarter of 2018 through 2037. Approximately 6,293 tpd of tailings will be pumped at a slurry density of 28% by weight from the tailings thickener to the TSF.

The Sulfide Expansion Project has planned nominal ore feed to the mill of 1.9 to 2.2 Mtpa. The current refining process and tailing deposition methods are expected to yield average end-of-filling tailings density of approximately 0.93 t/m3.

The TSF design includes a zoned earth and rockfill embankment with downstream raise construction, an impoundment underdrain system, a composite liner system, and an overdrain system. A discussion on the site selection process and descriptions of the design components are included in the following sections.

18.13.1 Site Selection

A TSF siting study was conducted to determine the optimal location for the TSF. The site selection consisted of a multiple criteria decision evaluation process conducted on 12 sites identified as potentially viable for development the TSF for the Sulfide Expansion Project. The siting study (Golder, 2013c) was completed by Golder with input from Jacobs and Alacer staff in Denver and Ankara. Potentially viable sites for TSF development were identified, evaluated, and ranked for a number of environmental, social, technical, and economic considerations.

The TSF siting study resulted in the selection of TSF Site 1, the site considered in this Report, as the preferred site for TSF development.

18.13.2 TSF Geology

The TSF site geology was mapped by Sial in 2005 and 2007 and by Fugro-Sial in 2012. Golder visited the site in 2012 and 2014, reviewed the geologic mapping and site conditions and verified and/or modified the geologic contacts identified by previous mapping efforts.

In general, the bedrock units in the Çöpler district range in age from Permian to late Cretaceous (250 to 60 million years ago) and include limestone, basic and ultrabasic rocks associated with abducted ocean floor (ophiolite suite) and other metasedimentary units. During the Late Cretaceous and early Tertiary periods these basement rocks were intruded by granodiorites and associated volcanic units. The hydrothermal mineralization throughout the district (Marek et al, 2008) probably occurred during this phase of intrusion. Key units within the footprint of the proposed TSF are:

- Munzur Limestone - Gray to blue-gray, very strong and unweathered. Much of the limestone unit shows karstic development. Bedding within the unit is indistinct to massive.
- Ophiolitic mélange - Ophiolitic mélange consists of diabase and serpentinite units.

Diabase (dolerite) – Located within the upper zone of the ophiolitic mélange sequence. The rock mass consists of green to greenish black, fresh to slightly weathered and strong to very strong rock strength properties. It usually includes very close to close joint spacing. In general, joint surfaces are covered with calcite and iron oxide infill. In places, the rock mass shows a blocky-texture embedded in a fine matrix.

Serpentinite - This unit is characterized by a bluish green and light green color, is weak to medium strong and is usually argillic. The rock mass consists of very close to closely spaced clay-filled discontinuities.

- Granodiorite - The granodiorite intrusion appears to have followed the thrust zone developed between Munzur limestone and Ophiolitic mélange. The rock mass, in its fresh state, is light brown, orange-brown and gray to pale green granodiorite and consists of closely to widely spaced discontinuities and medium strong to strong rock strength properties. In general, joints display short persistence (1-3 m) and smooth planar surfaces as well as undulating surfaces. Most joint surfaces are filled with calcite and iron oxide sealing (Fugro-Sial, 2012). Much of the unit within the proposed TSF area is moderately to completely weathered (depth of weathering varies), appearing and behaving more like a soil.

- Skarn – The skarn zone is developed along the granodiorite contact with the limestone and ophiolitic mélange. This zone was probably developed under high pressure and temperature conditions during the intrusion of the granodiorite at depth. Skarn rocks are black to dark brown, silicified, very strong, and moderately weathered and locally include solution cavities.
- Based on the field mapping, the embankment footprints for the proposed TSF will be founded predominantly on excavations within the following materials: Primarily Munzur Limestone, granodiorite, and diabase with limited areas of skarn and serpentinite.

The extent, orientation and strength of the serpentinite unit is of special interest, because serpentinite often displays low strength properties that may have a significant impact on embankment stability and deformation under static and earthquake loading conditions.

The character of the foundation conditions outside of the limits of the area initially investigated by Tetra Tech in 2007 was evaluated by Golder using field reconnaissance, geophysical surveys, and geologic mapping performed by Fugro Sial. Additional geotechnical site investigation (i.e., drilling and test pit excavation) was also performed in 2014 to address the expanded footprint of the TSF and to add additional data in areas of potential concern (i.e., within the ophiolite mélange).

18.13.3 Ziyaret Tepe Fault Hazard Evaluation

Golder completed office and field-based investigations along about 20 km of the Ziyaret Tepe fault (ZTF) to the north and south of the proposed TSF site in November 2012 and documented the investigations as part of the TSF Feasibility Design Report (Golder, 2014d). The ZTF is a north-northwest striking fault with a trace located less than 1 km from the proposed TSF site. Golder's investigations of the ZTF build upon the field studies of the ZTF reported by Sial in June 2005.

Interpretations of the office and field data provide evidence for the existence of faulted and folded bedrock and basin infill units exposed within the ZTF zone. Extension of the mapped ZTF trace north-northwest across the Karasu River, however, shows that surface traces are absent on the surface of a major Late Pleistocene aggradation terrace surface that is well preserved about 30 m above the active Karasu River channel. Furthermore, the down-valley profile of this Late Pleistocene terrace surface is not deflected where it intersects the northward extension of the ZTF. These observations indicate that there is no evidence for surface rupture along this part of the ZTF during the Holocene Epoch (last 10,000 years) and at least during latest Pleistocene time (last ca 20,000 years). Thus, the ZTF is either seismically inactive or has a very low average slip rate during the Late Quaternary Epoch (last 130,000 years). Accordingly, Golder considers that the ZTF is unlikely to generate large earthquakes and/or surface fault rupture along the 20 km of surface trace examined. Seismic analysis and design of the TSF site need not consider earthquake and related hazards associated with the ZTF.

18.14 Tailings Storage Facility Design

18.14.1 TSF Description and Design Criteria

Site-specific design criteria for the Çöpler TSF are summarized and were developed based on the following agency publications:

- World Bank Standard Guidelines
- International Committee on Large Dams (ICOLD)
- Canadian Dam Association – Dam Safety Guidelines, 2007
- The Mining Association of Canada – A Guide to the Management of Tailings Facilities, September 1998.
- Turkish General Directorate of State Hydraulic Works (DSI), Dam Construction and Technical Specification Guidelines, December 2011

The engineering design intent was to assure that the facility design adheres to the design criteria set forth at the onset of the work. General design criteria for the Çöpler TSF are summarized as follows:

- The tailings embankment shall be physically and chemically stable, and shall not impose an unacceptable risk to public health and safety or the environment
- The facility is classified in accordance with ICOLD guidance as Large-High (size and hazard classification) for the operational phase and post-closure phases
- The tailings impoundment shall provide sufficient containment of contaminants to result in compliance with environmental standards
- All impoundment effluents will be restricted to the drainage basin in which the TSF resides
- Post closure ancillary facilities shall be decommissioned so as to not pose an unacceptable risk to public health and safety or the environment.

Specific engineering design criteria were developed. Some revisions were made to accommodate requests from the Ministry of Environment and Urbanization during the permit review process and to accommodate the changes in tailings properties with a revised version of the design criteria developed in early 2016 (Golder, 2016a) to support the detailed design. The changes to accommodate the additional tailings capacity up to 45.9 Mt are discussed in a Technical Memorandum (Golder, 2016b) and summarized in this Report.

SRK previously performed geochemical testing in order to determine the waste classification of the expected tailings stream. The SRK report includes the measured concentrations of various metals and non-metals, as well as other parameters such as pH, total dissolved solids (TDS), total organic carbon, and others. SRK determined that most of the parameters tested were within the limits of "inert" (Class-III) waste, the lowest risk category according to Turkish regulations. However, sulfate and TDS concentrations were higher, resulting in a classification of "non-hazardous" (Class-II). Subsequently, Anagold submitted an EIA (SRK, 2014) for the project which committed to meeting standards for Class-I waste. The liner system design for the TSF has therefore been developed based on the requirements for Class-I waste.

The planned TSF consists of a fully lined impoundment constructed in phases with a compacted earth and rock fill embankment. Phased development will include a starter facility plus six subsequent phases in the valley defining the TSF area. The TSF design includes the following primary components:

- Phased compacted earth and rock fill tailings embankment
- Composite geomembrane-geosynthetic clay liner (GCL) system
- Two-layer granular filter protection system for embankment
- Impoundment gravity flow underdrain system and collection pond
- Impoundment gravity flow overdrain system, collection pond, and seepage return system
- Liner protection components, including a geocomposite overlying the geomembrane, gravel filled geoweb (select locations), sediment basins, and temporary storm water management systems
- Construction access roads
- Perimeter roads and benches within and around the impoundment area for access, anchor trenching and tailings distribution/reclaim water pipes
- A sidehill rail reclaim system for tailings water recovery during Phases 1 and 2, with plans to utilize a barge system in later phases

The earth and rockfill embankment includes a residual freeboard allowance of 1.0 m minimum in addition to storage above each phased tailings impoundment level to contain a PMP event; plus additional storage for the maximum operational pool volume predicted by the probabilistic TSF water balance, assuming no upstream diversion. During Phase 3 development, a storm water diversion channel designed to convey the 100-year, 24-hour event will be constructed up-gradient of the TSF. The channel will collect surface water and reduce run-on to the facility.

The fully-lined tailings impoundment includes an underdrain system, composite geomembrane and low-permeability soil liner, overdrain system, and a water pool access road for reclaim water operations.

18.14.2 Embankment Type Selection and Design

The tailings facility is designed to contain the deposited tailings within a fully lined impoundment located behind an engineered compacted earth and rockfill embankment.

The type of embankment chosen for the TSF was based on considerations related to anticipated performance of the system in various areas, including earthquake resistance, environmental performance, ease of closure, ease of construction given the site conditions, and relative cost. For the Çöpler TSF, the earthquake resistance was deemed the most critical factor because the facility is located in a high-seismicity region and failure of the embankment resulting from earthquake loading would likely result in either temporary or permanent mine shutdown.

Accordingly, the design of the TSF includes a rockfill embankment with downstream raises to provide protection from the high seismicity in the region. The TSF design included an assessment of the slope stability under earthquake, or dynamic loading conditions, and assessed the TSF performance under the OBE and the maximum design earthquake (MDE). The OBE considered an earthquake with magnitude M7.0 or a 10% probability of exceedance in 50 years (e.g., the 475-yr event) and resulted in peak ground acceleration (PGA) of 0.30 g. The MDE considered an earthquake with magnitude M7.5 or a 2% probability of exceedance in 50 years (e.g., the 2,475-yr event) and resulted in a PGA of 0.53 g.

The embankment is designed as an earthfill/rockfill structure with a composite geomembrane-GCL-soil lined upstream embankment face, and appropriate filter and transition zones to provide containment integrity. The embankment will be constructed in phases, in the downstream direction, using high strength compacted rock fill materials in the structural zone for embankment slope stability. The geomembrane-GCL-low permeability soil liner will be placed in the upstream section for seepage control with two filter zones to provide transition from the upstream low permeability soil fill to the downstream rock fill section. The Phase 1 (starter facility) and ultimate tailings embankment sections and embankment configuration for the TSF, and fill descriptions are provided in the Golder TSF Feasibility Design Report (2014d) and in the subsequent Technical Memorandum (Golder, 2016b) to address the changes to facilitate the additional capacity to 45.9Mt. The Starter (TSF Phase 1) and Ultimate Embankment and TSF Impoundment Grading Plans are shown in Figure 18-2 and Figure 18-3, respectively. An embankment profile and selected cross sections through the embankment and impoundment are shown in Figure 18-4.

Figure 18-2 TSF Starter Embankment and Impoundment Grading Plan

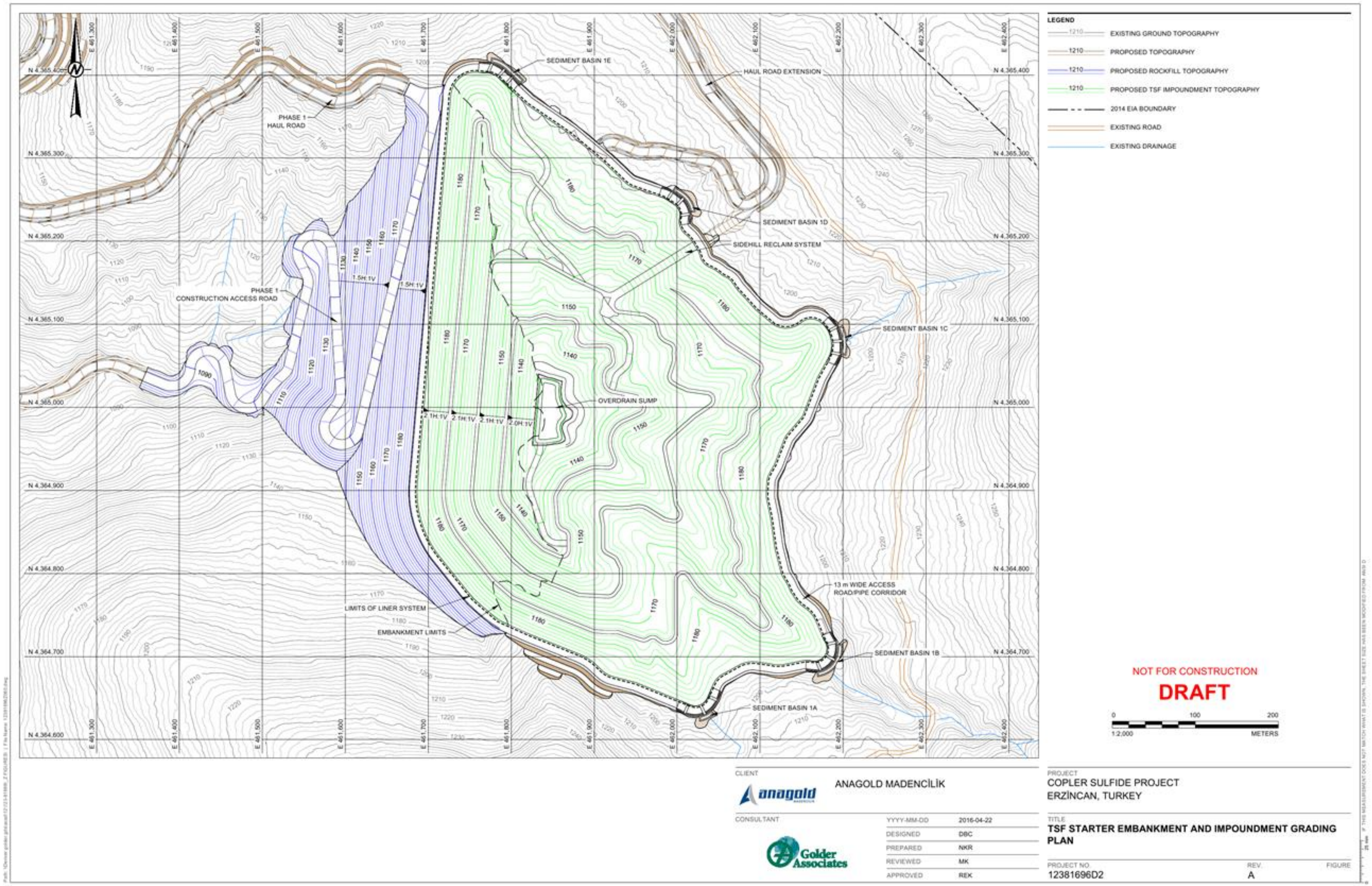


Figure 18-3 Ultimate TSF Grading Plan

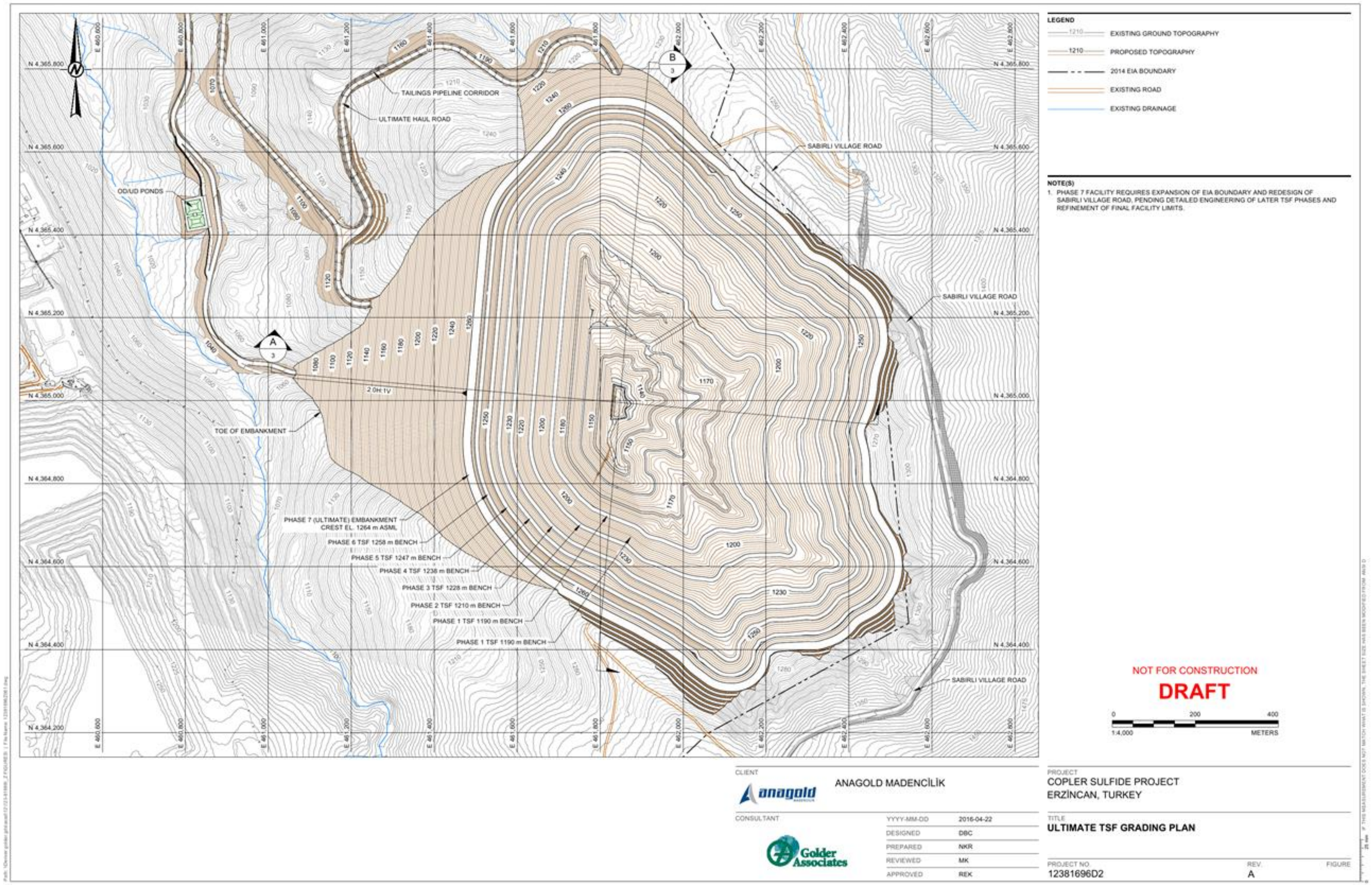
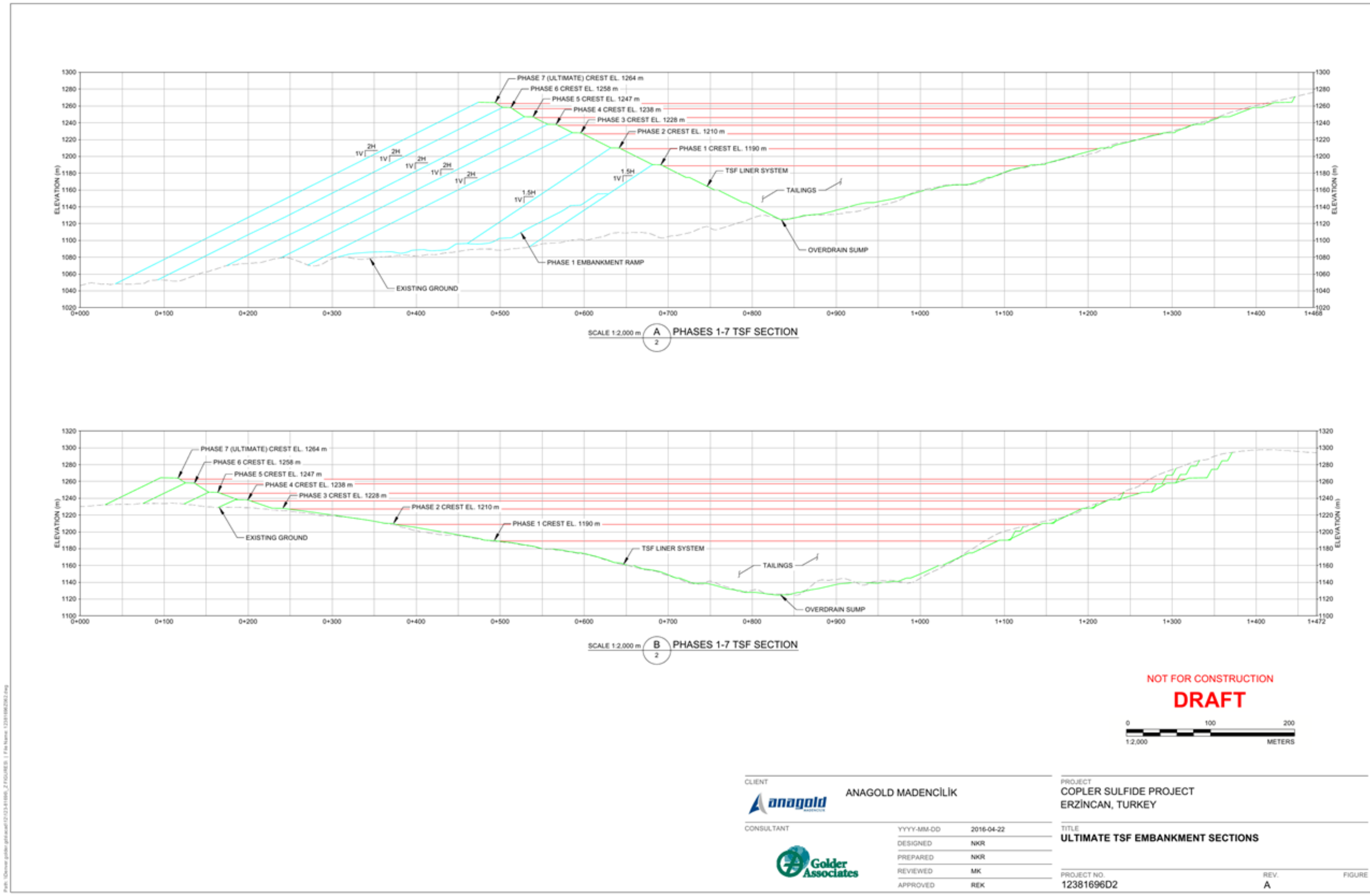


Figure 18-4 TSF Ultimate Embankment Profile



18.14.3 Impoundment Underdrain System

The tailings impoundment underdrains will be constructed in phases and allow natural drainage beneath the lined facility to be routed downstream of the tailings embankment and impoundment limits to a collection pond. The system allows the underdrain flows to be monitored, and then flow by gravity to the downstream natural drainage assuming water quality is acceptable. Once flows reach the seepage collection pond, discharge can be directed downstream to natural drainages, or water may be reclaimed and used within the process circuit, depending on operational requirements and results of water quality monitoring.

18.14.4 Composite Liner System

On slopes shallower than 3H:1V the selected composite liner system (clay/GCL/geomembrane) consists of a minimum 50 cm of low-permeability (1 x 10^{-9} m/sec) compacted soil liner, GCL and a 2.0 mm high density polyethylene (HDPE) geomembrane liner barrier for containment of seepage from the tailings. On slopes steeper than 3H:1V composite liner system consists of a minimum 50 cm compacted soil liner or General Clay Liner Fill (soil classify as a CL, SC or CH), GCL and a 2.0 mm high density polyethylene (HDPE) geomembrane liner. The selected composite liner system provides better protection than the regulatory prescribed liner system of at least 5 m of low-permeability (1 x 10^{-9} m/sec) soil liner based on a comparison of seepage rates through a composite liner compared to the prescriptive compacted clay liner system.

The soil materials for the low-permeability soil liner component will be obtained from the existing clay borrow source located north of the TSF and from overburden within the sulfide pit boundaries. The total amount of clay required for construction of the ultimate TSF is 832,420 m^3. Approximately 478,500 m^3 had been determined to be available from the existing permitted clay borrow source based on site investigations performed to date. Suitable low-permeability soils have also been observed within portions of the existing open pit overburden and have been used for construction of the heap leach pad composite liner system. Golder has recommended that Anagold quantify the availability of suitable materials from the pit. Additional evaluation including interim slope stability and compatibility testing would be required and is recommended as part of future engineering studies.

18.14.5 Impoundment Overdrain System

The overdrain dewatering system, which is to be operated during active deposition, minimizes hydraulic head on the impoundment liner system and accelerates consolidation of the overlying tailings mass. Accelerating consolidation allows for higher tailings density and storage within the TSF, while lowering hydraulic head will reduce the seepage rate through any potential defects in the liner system. The water collected by the overdrain system is generated by consolidation of the overlying tailings. Initially, water collected by the overdrain system will be primarily from pooled water within the impoundment, then liquids from tailings slurry during startup, followed by a combination of tailings consolidation water from settled tailings and inflow to the drains from precipitation over the liner system.

The overdrain system above the geomembrane consists of a nearly continuous layer of geocomposite drainage material, which will collect and convey water from the consolidating tailings to perforated pipes located in the base of drainage inverts. No geocomposite will be placed against the northeast side of the impoundment, adjacent to the sidehill reclaim system where the operational pool will be maintained. The pipes will convey fluids to a collection sump located at the upstream toe of the embankment which then drains by gravity beneath the embankment via a reinforced concrete encased steel pipe to a sampling port located downgradient of the TSF. Flows then drain by gravity to a seepage collection pond located on a relatively flat area approximately 500 m north of the TSF toe. The seepage collection pond was sized to contain a minimum of 24-hours of drain-down fluids based on the maximum flow rate of 1500 m^3/day. Collected fluids will be pumped back to the TSF for reuse or to the Plant in the sulfide process plant.

In addition to the protection provided by the geocomposite layer, an aggregate filled geoweb system will be constructed over the liner system along the perimeter road at designated drainage areas to provide protection from run-on flows and potential debris. Sediment basins will be constructed within these designated drainage areas immediately upgradient of the TSF perimeter road. The basins will serve to capture sediment and debris, in addition to providing some energy dissipation of storm water run-on during larger storm events. Riprap or other bank protection will be placed within the sediment basins, which have been designed to manage the 10-year, 24-hour storm event.

18.14.6 Seepage Analyses

A steady state seepage analysis was performed to estimate, through finite element computer modeling, the location of the phreatic surface and the magnitude of pore pressures developed through the embankment and foundation materials during Phase 1 and after construction of the ultimate TSF configuration. Results of the model indicate that the composite liner system, granular filters, and drainage systems will effectively reduce seepage through the liner system and provide effective drainage for any seepage through the liner system. The composite liner, filter, and drain systems will effectively limit phreatic water in the embankment's structural shell, thereby decreasing potential for environmental impacts and increasing the geotechnical stability of the TSF embankments.

18.14.7 Stability Analyses

Results of the seismic hazard analyses and seepage analyses were used to develop input to the slope stability analyses performed for the proposed embankment configuration. Limit equilibrium slope stability analyses were performed to ensure that that the embankment will remain stable under both static and dynamic (earthquake) loading conditions during both normal operations and after closure. Static slope stability analyses of the Phase 1 (starter) and ultimate embankments for the TSF indicate adequate factors of safety against slope instability under static conditions.

Pseudo-static analyses were performed using the method proposed by Hynes and Franklin (1984) for evaluating embankment response under seismic loading. The analyses indicate that the embankments will experience only minor displacements when subjected to the operational base earthquake (OBE – 1 in 475-yr event). However, the analyses indicate that some movement or deformation of the embankment may occur under the maximum design earthquake (MDE – 1 in 2,475-yr event). Therefore, in order to more accurately assess the deformation potential, dynamic deformation analyses were performed using a variety of simplified methods (i.e., Makdisi and Seed, Swaisgood, and SHAKE2000/Newmark and others) and state-of-the-art finite difference models (i.e., FLAC 7.0).

The results from the simplified Makdisi and Seed analysis indicate that the ultimate embankments may experience permanent crest displacements of approximately 1.4 m during the MDE. Of the various methods used to calculate TSF embankment crest deflections, the FLAC results are considered most accurate due to the increased number of site-specific variables considered and the increased rigor of the methods used. The average FLAC crest displacement, 0.22 m, is considered acceptable, as there will still be 0.78 m of residual freeboard remaining, meaning that no tailings will be released during the MDE seismic event. Table 18-2 shows the displacement results for different dynamic stability methods.

Table 18-2 Displacement results from different methods

Method	TSF Ultimate Design	
	Crest Displacement (m)	Base Displacement (m)
Makdisi and Seed (1978)	1.41	0.14
SHAKE2000/Newmark	0.17	-
Swaisgood (2003)	0.73	-
FLAC 7.0	0.22	0.18

Stability analysis for both static and dynamic conditions were performed for cases where the TSF ultimate embankment was constructed to elevation 1240 m, essentially through Phase 4 of the TSF development. Additional evaluation of stability for later phases of development (i.e. Phase 5-7) should be performed once additional foundation geotechnical parameters are obtained during the Phase 1 construction phase and in conjunction with optimization and final design for later phases of TSF development.

18.14.8 Embankment and Foundation Settlement

Settlements of the earth and rockfill embankments are expected to be minimal and to occur predominantly during construction. Based on geologic mapping and results of the site investigation and laboratory test results, the TSF embankment is expected to be founded on primarily hard rock (Munzur limestone, granodiorite, or serpentinite). Due to the lack of significant clay zones, there should be little time-dependent settlement (i.e., consolidation or creep). Therefore, because settlements will occur during construction, there should be no significant impact on embankment crest height.

18.14.9 Tailings Impoundment and Consolidation Settlement

Settlement of the tailings within the TSF impoundment was estimated using one-dimensional large-strain consolidation modeling. This analysis was conducted using a series of increasing areas of one-dimensional columns to simulate the average tailings production rate of 6,293 tpd. Results of column settling and flume deposition testing indicate that the whole tailings behave as a nonsegregating material; therefore, the tailings consolidation properties remain constant across the entire impoundment.

The results of the consolidation model indicate that the average tailings dry density will increase over time, reaching approximately 0.93 t/m^3 at the end of filling. Using the densities calculated from the consolidation model and the top of tailing surfaces developed in the deposition model, the starter TSF (Phase 1) with an embankment crest elevation of 1190 m will have a storage capacity of 3.0 Mt. Phase 2 with a raise to 1210 m will have a storage capacity 7.7 Mt. Phase 3, with a crest elevation of 1228 m will have a storage capacity of 16.1 Mt. The Phase 4 TSF will have a 23.0 Mt storage capacity with a crest elevation of 1238 m. The Phase 5 TSF will have a 30.0 Mt storage capacity with a crest elevation of 1247 m. The Phase 6 TSF will have a crest elevation of 1258 m and a 39.9 Mt storage capacity. The ultimate Phase 7 TSF will have a crest elevation of 1264 m and a 46.2 Mt storage capacity. Additionally, the model indicates that complete settlement of the tailings will be a slow process, requiring up to 30 years to reach steady state and settlements on the order of 27 m realized within the deepest portions of the TSF.

The sulfide tailings are expected to have a P_{80} ranging from 35 to 88 µm and classified as high plasticity clay (CH) under the Unified Soil Classification System (USCS). In addition, at the planned initial solids content, the POX treated sulfide tailings appear to have a nearly paste consistency rather than the segregating slurry behavior expected from typical hard rock tailings. As a result, the sulfide tailings settle more slowly, release less water and require more time to consolidate than the oxide tailings.

Consideration will need to be given with respect to closure and the time required to construct the closure cover for the TSF considering the length of time required for complete consolidation of the tailings.

18.15 Project Water Balance and Management

18.15.1 TSF Water Balance

The operational approach to storm run-off management is to maintain adequate storage within the impoundment to store the run-off from the PMP event plus maximum operational water volumes as predicted by the probabilistic TSF water balance, and gradually incorporate the accumulated volume into the facility water balance through recycle to the plant. The water balance and design criteria for the TSF have considered the PMP rather than the 100-year, 24-hour storm event due to the higher potential risk of the TSF and the long term permanent nature of tailings storage within the TSF.

The project design criteria require the tailings embankment to be constructed such that a residual freeboard allowance of 1.0 m minimum is maintained at all times. The design criteria also require additional freeboard be maintained as necessary in order to contain the PMP event plus the 95th percentile of the annual maximum operational water volume as predicted by the probabilistic TSF water balance, assuming no upstream diversion without encroaching on the minimum design freeboard. The PMP event was determined to be 302 mm of precipitation within 24 hours. The volume of water reporting to the TSF varies over the life of the facility, ranging from 535,145 m^3 to 761,249 m^3. The 95th percentile of the maximum annual operational pool volume predicted by the TSF water balance is 255,740 m^3, resulting in a total required water storage volume of 535,145 m3 to 761,249 m3. The highest storage requirement occurs during Phase 1. During Phase

3 development, a storm water diversion channel designed to convey the 100-year, 24-hour event will be constructed upgradient of the TSF. The channel will collect surface water and reduce runon to the facility for subsequent phases.

For the first six years, due to the higher rate of rise and tailings properties, Golder estimates the beach slope will be less than 1%. For the remaining mine life, Golder estimates the average beach slope of the tailings will be approximately 1%. Given this slope, the operational pool plus the entire volume of storm water produced by the PMP can be stored within the pool area (i.e., the conical depression in the center or rear of the TSF created by the 1% beach slope) for Phases 2 through 7. For the starter facility, a freeboard of 2 m (i.e. maximum tailings elevation of 1188 m) should be maintained during operations in order to allow sufficient contingency storage for the PMP event. A flatter beach slope is also possible, depending on deposition energy and slope re-adjustments due to static liquefaction induced by the rapid rate-of-rise (ROR) of the tailings surface. In any event, phased development and operational management of the TSF requires that the operator maintain a design freeboard of 1.0 m plus additional volume as required to contain the design event at all times during the life of the TSF.

The tailings supernatant water pool will be maintained away from the crest of the tailings embankment during normal operating conditions. The tailings impoundment water pool, formed on the settled tailings slurry surface away from the peripheral discharge points, will be directed toward the sidehill reclaim system located on the northeastern impoundment slope.

The sidehill reclaim system is anticipated to be utilized through at least Phase 2. During Phases 3 through 7, a barge system may be substituted. If a barge system is used, access roads will be constructed on the liner to accommodate maintenance and periodic moving of the mobile pump and floating barge intake pumping operations. All roads constructed over lined areas will utilize sufficient thickness of road base material such that traffic loads will not cause damage to the geocomposite drainage layer or liner systems.

18.15.2 Tailings Slurry Delivery and Reclaim Water

Tailings will be deposited in the facility via a slurry delivery pipeline system. Slurry deposition will take place throughout the year from multiple points along the embankment crest and around the impoundment perimeter. A rotational deposition plan is required to maintain the supernatant pool in the planned area of the impoundment and will result in thin lift tailings deposition. Tailings solids from the slurry will settle along the beach and free water will drain to the supernatant pool. Additionally, water seeping upwards from the consolidating tailings will flow by gravity to the process water pool. A reclaim water system will consist of a pump mounted to a cart which will move up the northeast impoundment slope along a sidehill rail as the impoundment fills. Reclaimed water will flow through a return water pipeline discharging at the mill. Slurry discharge from the spigots will be used to create an above water sub-aerial beach from which tailings will drain and consolidate. The discharge points may vary and include additional discharge points, as needed, to establish peripheral deposition and tailings beach development to the water pool in the eastern impoundment areas.

18.16 Construction Schedule

A preliminary construction schedule has been developed assuming start of construction in the third quarter of 2016. The initial construction effort will include clearing of vegetation, foundation preparation, installation of underdrains, and construction of perimeter access roads and the haul road from plant area to the TSF. Construction of the initial haul road to the TSF is anticipated to take 4-6 months to complete. Construction of the embankment for the starter TSF embankment will be initiated in the first quarter of 2017 and is estimated to take approximately 12-16 months to complete including placement and compaction of the embankment rockfill and coarse and fine filter materials. Construction of the composite liner system and overdrain system may occur in two construction seasons in 2017 and 2018. A portion of the geosynthetic liner system required within the overdrain sump area will be required to be completed by the fourth quarter 2016 in order to facilitate installation of the overdrain conveyance piping beneath the TSF embankment fills. Contingency plans for winter construction or other accelerated schedules may need to be developed depending on timing of the start of construction in 2016.

The geosynthetic liner system will likely be constructed in 2017 and 2018. Similarly, construction of the tailings pipeline corridor is expected to occur in 2017 and early 2018. Planned start-up and commissioning of the Sulfide Expansion Project is planned for the third quarter of 2018. Optimization of the construction schedule should be further evaluated during final detailed design and in discussions with qualified contractors initiated as part of the bid process and final development of construction execution plans.

Construction of the Sabırlı Village road realignment may be deferred until Phase 3 in 2020-2021.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Markets

The markets for gold and silver doré are international and generally robust but variable, depending on supply and demand marketing aspects. Due to low copper prices, copper precipitate production is not included in the current POX plant design but provisions have been made in the site layout for later inclusion.

Generally, 50% of the gold and silver from the Çöpler oxide heap leach operation is delivered to METALOR Technologies S.A. in Switzerland and the other 50% is delivered to the Istanbul Gold refinery. Following refining, the gold and silver bullion is priced and settled as a commodity on designated markets. Sales of gold recovered from the sulfide process plant will likely be similar to the current arrangement for gold doré.

19.2 Contracts

Anagold contracts the mining operations to a Turkish mining contractor. The contract term expires on February 1, 2017. 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.

Anagold has entered into a contract with Amec Foster Wheeler for engineering, procurement and construction management for the Sulfide Project. The Company has or will enter into a number of additional contracts for earthworks, oxygen supply and construction services in connection with the construction of the Sulfide Project.

19.3 Commodity Prices

Commodity pricing is set by Alacer Corporate; pricing is informed by a review of price forecasts from analysts, major banks, and industry peers.

20.0 ENVIRONMENTAL STUDIES, PERMITTING & SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

An Environmental Impact Assessment (EIA) study was completed in 2008 for the oxide ores of the Çöpler Gold Mine operating 15,500 tpd heap leach facility. The EIA permit was obtained from the Turkish Ministry of Environment and Urban Planning (MEUP) on April 16, 2008. The project description for the 2008 EIA included three main open pits (manganese, marble contact, and main zones), five waste rock dumps, 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 studies conducted and environmental permits received for oxide resources of the Çöpler Mine since the start of the gold mine operations are as follows:

- EIA permit dated April 10, 2012 for the operation of mobile crushing plant,
- EIA permit dated May 17, 2012 for capacity expansion involving (i) increasing the operation rate to 23,500 tpd; (ii) increasing the Çöpler waste rock dump footprint area; (iii) adding a SART plant to the process in order to decrease the cyanide consumption due to high copper content in some ores.

The EIA studies were conducted according to the format stipulated by the Turkish EIA Regulation. The scope of the Turkish EIA studies differ 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. In the period following the receipt of the 2008 EIA permit, Alacer conducted further studies to supplement the Turkish EIA study and subsequently meet the IFC requirements. These studies involved a Resettlement Action Plan (RAP) for the Çöpler village, a socio-economic baseline study for Çöpler Village, a human rights assessment study, an Environmental Management Plan (EMP), and a biodiversity study.

SRK Danışmanlık ve Mühendislik A.Ş. (SRK) was retained by Alacer to undertake the Çöpler Sulfide Expansion Project Environmental and Social Impact Assessment (ESIA) study for possible financing purposes. In addition, SRK was commissioned to prepare the EIA for Turkish permitting requirements for the Project, including water resources management studies, geochemical studies for mine waste management, and environmental and social studies.

The EIA permitting process for the Çöpler Sulfide Expansion Project started on April 07, 2014 and ended with the receipt of the "EIA Positive Statement" on December 24, 2014.

Much of the content in this section originated from the Çöpler Mine Sulfide Expansion FS – ESIA Report on the Çöpler Gold Mine Sulfide Expansion Project prepared by SRK Turkey (SRK, 2015).

20.2 Comparison of Turkish EIA and International ESIA Studies

The Turkish EIA regulation was promulgated in 1993. Since then it has been revised and amended several times, the most recent amendment being in November 2014. The Sulfide Expansion Project is subject to the previous regulation which was amended on October 3, 2013 (Official Gazette no. 28784). The Turkish EIA regulation was transposed from the European Union EIA Directive. Therefore, in many regards the Turkish EIA regulation is similar to those in various European countries. However, the infusion of various Turkish sub laws, conventions, and governmental practices into the EIA process makes the Turkish EIA unique in certain respects.

The Turkish EIA regulation classifies projects into categories through a project screening list which is based on project capacity, size, and activity/process criteria. The projects can be classified into three categories:

- Small projects that are exempt from the requirements of the EIA Regulation,
- Minor projects that are subject to EIA Regulation Annex-2 requirements,
- Major projects that are subject to EIA Regulation Annex-1 requirements.

Most mining projects, due to their nature, size, and areal coverage are subject to Annex-1 requirements. The Sulfide Expansion Project is subject to Annex-1 requirements. Annex-1 projects are required to conduct a comprehensive EIA study.

An EIA permit is the first step in Turkish environmental permitting system. An EIA permit is needed before construction activities can commence. Furthermore, any major modifications done on the project design after an EIA permit is obtained, may require that a new EIA permit is obtained for the modifications before they can be constructed.

The EIA permitting process starts with the project owner submitting a Project Description Report (PDR) to the MEUP. A PDR is a brief document that describes the project area and environs, a description of the project elements, and a preliminary impact assessment. It is a public document that is used for informing relevant stakeholders. Following the submittal of the PDR, an official public hearing is arranged, whereby the project is introduced to the stakeholders and comments are officially noted. Following the public hearing, an Investigation and Assessment Commission (IAC) consisting of various departments of the MEUP and several other governmental agencies is convened where Terms of Reference (ToR) for the project EIA studies are established. The ToR is mandatory and cannot be changed by the project owner.

The EIA regulation allows for a maximum of an 18-month period to complete and submit the draft EIA report for the project. Following the filing of the draft EIA report, the IAC reviews the draft report and convenes to reach a decision on the completeness and adequacy of the EIA study. The project owner and the EIA report preparers are invited to present and defend their projects. Depending on the project complexity, the IAC may convene more than once to reach a decision. Once a decision is reached and amendments (if any) to the EIA are completed, the final EIA report is posted at relevant official locations for final public written comment. With the inclusion of public comments, the EIA permitting process is finalized.

There are certain important differences between the Turkish EIA and international ESIA studies, as listed below:

- Turkish EIAs have social components integrated into the process. However, these are significantly less rigorous than IFC's requirements. The social baseline data in the Turkish EIA is usually based on published secondary data, whereas, in the international ESIA, primary data collected in the field is utilized. Primary data provides further details and up-to-date information about the socio-economic status of the project area.

- A minimum of one public hearing for stakeholder engagement is mandatory in the Turkish EIA regulation. This is done at the beginning of the EIA permitting process, after the submittal of the PDR to the MEUP. Public opinion is sought again at the end of the EIA permitting process after the submittal of the final EIA report. The final EIA report is posted at the offices of the nearest government administrative unit and no public hearing is held. Public opinion is only accepted in written form. While some stakeholder engagement is conducted during the EIA process, these are very limited in scope and the results from these engagements are generally not adequately integrated into the project designs and/or permit decision making process. IFC Environmental and Social Performance Standards (ESPS) require a more comprehensive and well documented Stakeholder Engagement process.
- Social and Environmental Management Systems are requirements of the ESIAs prepared under the IFC ESPSs, but not under the Turkish EIA regulations.
- Turkish EIAs are heavily structured according to numerous environmental regulations and circulars in effect. Land use and cadastral restrictions play a major role in a project's permits and progress.
- Turkish environmental regulations involve a prescriptive approach rather than a risk-based approach.

Certain IFC ESIA performance standards do not have a counterpart in the Turkish EIA. These are:

- Performance Standard 2: Labor and Working Conditions
- Performance Standard 4: Community Health, Safety and Security
- Performance Standard 5: Land Acquisition and Involuntary Resettlement
- Performance Standard 6: Biodiversity Conservation and Sustainable Natural Resource Management
- Performance Standard 8: Cultural Heritage

These standards are generally handled outside of the EIA system, but under other Turkish environmental permitting and regulatory processes.

The Turkish EIA regulation requires that EIA documents be prepared in a mandatory format. International ESIA documents are prepared in a flexible format as long as the guidelines and risks are identified and the risks are addressed adequately.

The Turkish EIA regulation requires that the following separate reports be prepared according to a mandatory format and attached to the EIA report:

- Mine Reclamation and Closure Plan,
- Acoustical Report, and

- Agricultural Soil Conservation Report.

20.3 Status of Permitting

The EIA permitting for the Çöpler gold mine for the oxide ore was completed in April 2008 with the issuance of an EIA positive certificate. Additionally, the Erzincan Provincial Directorate of Environment and Urban Planning has issued a certificate that “No EIA is required” for the Project’s clay borrow pits in July and August 2009. The construction of the Çöpler Mine commenced in December 2008. As a requirement of the Turkish EIA permit, the construction activities are audited bi-annually by an independent third-party with respect to the environmental monitoring and mitigation commitments provided by Alacer in the EIA report. Fourteen audits were completed from 2008 to 2015 for the oxide zone activities (October 2008, April 2009, October 2009, May 2010, September 2010, May 2011, November 2011, May 2012, November 2012, April 2013, May 2013, November 2013, December 2013, May 2014, and December 2014). EIA positive certificate for the Sulfide Expansion Project was received in December 2014. The audit periods were re-organized quarterly and four audits were completed for 2015 and in 2016 to date, two audits have been completed.

The EIA permit serves as a construction permit. The forestry land use permits for the construction of the Çöpler Sulfide Expansion Project were obtained on 20 April, 2016. Operational environmental permits are obtained within two years of the start of mine operation. All of the operational permits are already obtained for the existing oxide operation. These include: temporary and permanent explosive storage permits, groundwater use permit, EIA positive certificate for construction of a power transmission line, land use approvals for forest and pasturelands. The permanent explosive magazine storage permit was obtained on February 13, 2014 and the building use permit for permanent explosive magazine storage was obtained on April 7, 2014.

The list of the major environmental permits obtained for the Çöpler Mine (oxide) at the Report effective date is given in Table 20-1. The operational permits such as wastewater discharge, air emissions, hazardous waste etc. to support the project have been obtained. As stated in the previous section, the Sulfide Expansion Project has been subject to EIA process in accordance with EIA Regulation Annex-1. The EIA application was submitted to the MEUP on April 7, 2014 with the submission of the PDR. An EIA positive decision (permit) was obtained from the MEUP on December 24, 2014 for the project description as proposed in 2014.

Table 20-1 Environmental Permits Obtained for Çöpler Gold Mine

Permit/License	Regulation	Project Item	Date
Decision for "EIA is not Required"	EIA Regulation	Çöpler Column Test Operation	April 28th, 2006
EIA Positive Decision	EIA Regulation	154 kV Energy Transmission Line	November 19th, 2007
EIA Positive Decision	EIA Regulation	Çöpler Gold Mine	April 16th, 2008
Reclamation Plan Approval (obtained with EIA)	Regulation on Reclamation of Lands Degraded Due to Mining Activities	Çöpler Gold Mine	April 16th, 2008
EIA Positive Decision	Environmental Impact Assessement (EIA) Regulation	Mobile Crushing Plant	April 10th, 2008
EIA Positive Decision	Environmental Impact Assessement (EIA) Regulation	Expansion on waste rock dump, inclusion of SART process, increasing production rate	May 17th, 2012
Environmental Permit	Regulation on the Permits Required by the Environment Law	Çöpler Gold Mine	August 1st, 2012: renewed February 27th, 2014
EIA Positive Decision	Environmental Impact Assessement (EIA) Regulation	Expansion for Sulfide Resources, including the POX process, new WRDs, and enlarged open pit	December 24th, 2014

Some permits remain to be sourced for exploration activities, and in furtherance of operations (Table 20-2). None of these permits are required for the construction of the Sulfide Expansion Project.

Table 20-2 Çöpler Outstanding Permit Applications

License	Type	Permit Name	Application Date	Area (m^2)
Erzincan IR.847 (Çöpler)	Pasture	Waste Dump, Diversion Channels	29.12.2014	2.588.018,10
Erzincan IR.49729 (Çöpler Foot Print)	Pasture	Waste Dump, Diversion Channels, Road	29.12.2014	775.883,86
Erzincan IR.49729 (Çöpler Foot Print)	Forestry	Lay Down Areas	22.01.2016	92.322,11
Erzincan IR.20067313 (Çöpler Foot Print/Yakuplu)	Pasture	Operation, Waste Dump Top Soil Area and Road	Not confirmed	36.9
Erzincan IR.50237 (Çöpler Foot Print)	Forestry	Drillings and Roads	30.11.2015	18.373,03
Erzincan IR.76818 (Çöpler Foot Print)	Forestry	Clay License Operation, Waste Dump, Top Soil Area	12.01.2016	172.678,96
Erzincan IR.20067313 (Çöpler Foot Print)	Forestry	Operation, Waste Dump Top Soil	24.03.2015	4.928,65
Erzincan IR.1054 (Yakuplu)	Pasture	Operation Waste Dump Top Soil Area	27.10.2014	31.02
Erzincan IR.7083 (Bayramdere)	Forestry	Operation, Waste Dump, Top Soil	24.03.2016	32.030,43
Erzincan IR.1054 (Yakuplu)	Forestry	Operation, Waste Dump, Road	29.03.2016	625.243,52
Erzincan IR.1054 (Yakuplu)	Forestry	Drilling and Roads	29.03.2016	10.520,22
Erzincan IR.1054 (Yakuplu)	Pasture	Operation, Waste Dump	05.04.2016	78.300 104.500
Erzincan IR.7083 (Bayramdere)	Pasture	Operation, Waste Dump	05.04.2016	593.000 779.000
Erzincan IR.1054 (Yakuplu)	Forestry	Yakuplu North Waste Dump	N/A	Not confirmed

20.4 Public Consultation

Several social studies have been conducted since the early feasibility stages of Çöpler Mine development. Public consultation meetings were organized within the framework of SIA studies of the Sulfide Expansion Project. The Stakeholder Engagement Plan (SEP) is attached as an Appendix of the ESIA report of the Sulfide Expansion Project. Official Public Participation Meeting (PPM) was organized in the residential area nearest to the Project in order to inform the public and stakeholders of the Project and obtain their comments in connection with the Project. The essential point herein was to assess and incorporate into the Project comments, suggestions and concerns of the public and other relevant stakeholders.

At the PPM, that was held at the early stages of the official EIA process, the Project was presented to the stakeholders. Information disclosed included spatial coverage, information on the mining and processing operations, baseline studies that were conducted, potential impacts and the grievance mechanisms available to the public. Questions of the stakeholders were answered by experts and the remarks of the participants were noted.

The questions raised by the local people at the PPM related to community relations as well as potential environmental impacts of the Project. In addition, questions were also raised by the local people on the employment policies to be followed under the Project.

Some of the questions were addressed by Anagold; others were addressed by SRK, for and on behalf of Anagold.

20.5 Baseline Observations

Environmental baseline studies have the objective of characterizing the existing physical, biological, chemical, and socio-economic resources that may be impacted by the development of the project. Regarding the Çöpler Mine, the existing baseline information available from the 2008 and 2014 EIA reports, construction and operation phase monitoring, and results of the first round of soil and surface water sampling carried out by SRK Turkey staff for the expansion areas have been reviewed and summarized in the following sections.

20.6 Soil Types

20.6.1 Main Soil Groups

The main soil groups present within the Project area were determined from the soil maps produced by the General Directorate of Rural Services of the Ministry of Food, Agriculture and Livestock. The distribution of the main soil groups at the prospect and environs is shown in Figure 20-1. The common soil type in the study area and its vicinity is the brown soil (B) and rocky areas (ÇK). Brown soil group exhibit the entire characteristics of A, B and C horizons, with a calcification effect. Due to this calcification effect, a high amount of calcium is observed in the soil profile and the base saturation is high. In the brown soils, Horizon A1 is 10 to 15 cm thick, distinct, porous with medium organic matter content, neutral, or basic in pH, color gray-brown or brown. Horizon B has a color ranging from light brown to dark brown. It exhibits a coarse sub-angular blocky character. The lower soil gradually passes to pale brown or grayish highly calcareous parent material. In these soils the profile is totally calcareous leading to a caliche zone below Horizon B. Since this zone forms in places where annual precipitation is around 250 mm to 400 mm, caliche formation is encountered at very deep levels. The clay minerals observed in the profile are generally illite and smectite. The natural vegetation observed developing in these soils is composed of low or medium height meadow herbs. Parent material is clayey schist, calcite or clay stones intercalated with schist. In addition, in some places the parent materials can be loose alluvial material made up of calcite, clay stones or crystalline rocks. The major problems and constraints related to both of these soils are shallow soil cover, steep slopes, high erosion pressure, and the presence of gravel and cobbles. Depending on topography these problems limit the use of these soil types for grazing.

20.6.2 Soil Characterization

Soil sampling was conducted to determine the baseline physical and chemical characteristics that are prevalent within the expansion area. The soil samples were collected from the top 30 cm of soils at seven representative locations within the alternative footprints of the mine units proposed by Alacer. The site selection for the soil sampling locations was restricted by poor site accessibility. The sampling locations are shown on Figure 20-1. The soil sampling locations are labeled with the "CPSO" prefix.

The sampling was conducted in August 2012. All sample collection, preservation, handling, shipping and laboratory testing were performed according to SRK QA/QC protocols and SOP. Laboratory analyses were performed at ALS Laboratories in Canada, an accredited international laboratory. The results of the laboratory analyses are summarized in Table 20-3. The measured heavy metal levels are compared with the typical abundance values of heavy metals in the earth crust. The concentrations of silver, arsenic, bismuth, calcium, copper, lead, selenium, manganese, molybdenum, nickel, antimony, and zinc in the soil samples are generally found to be elevated with regard to the typical earth crustal abundances. All soil samples have slightly alkaline pH levels.

Figure 20-1 Main Soil Groups and Soil Sampling Locations

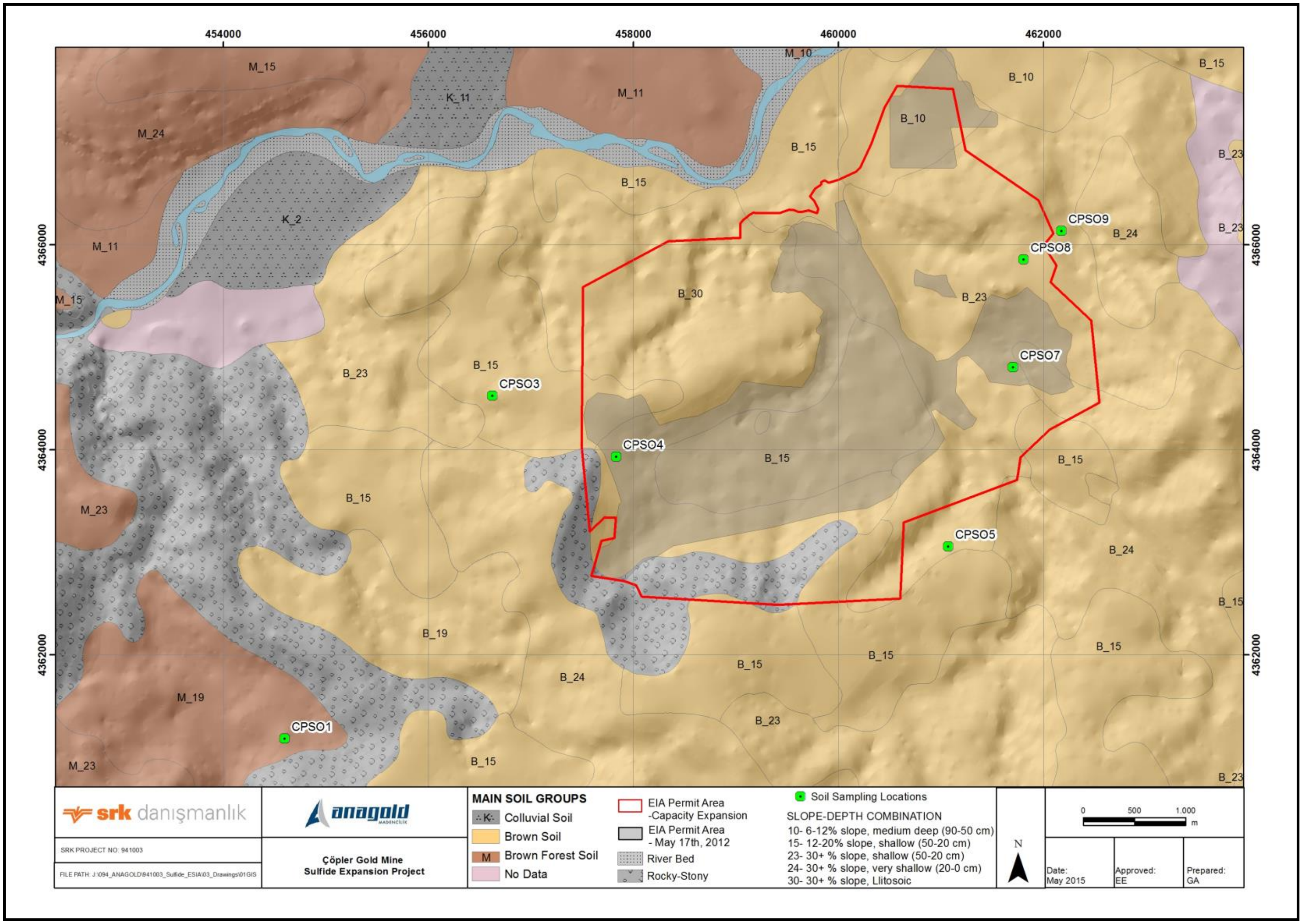


Table 20-3 Results of Soil Sample Analysis

Parameters	Units	Average Abundance in Earth Crust	Sampling Location						
			CPSO1	CPSO3	CPSO4	CPSO5	CPSO7	CPSO8	CPSO9
Conductivity	dS/m		0.047	0.094	0.064	0.081	0.047	0.038	0.052
pH	pH		8.31	8.12	8.42	8.27	8.51	8.6	8.59
Aluminum, Al	mg/kg	82,300	6,730	32,000	14,800	47,100	13,600	10,500	10,600
Antimony, Sb	mg/kg	0.2	0.42	1.3	1.28	7.4	0.52	0.15	0.67
Arsenic, As	mg/kg	1.8	6.49	21.9	107	36.4	14.6	5.09	26.7
Barium, Ba	mg/kg	425	24.3	158	60.4	201	59	99.5	69
Beryllium, Be	mg/kg	3	0.23	1.18	0.44	1.94	0.3	<0.20	0.49
Bismuth, Bi	mg/kg	0.2	0.25	0.78	2.7	0.46	0.23	<0.20	0.27
Cadmium, Cd	mg/kg	3	0.608	0.977	6.2	3.04	0.143	0.054	0.586
Calcium, Ca	mg/kg	41,500	284,000	106,000	175,000	21,800	4,220	29,100	205,000
Chromium, Cr	mg/kg	102	35.2	86.7	12.1	147	19.4	9.83	20.9
Cobalt, Co	mg/kg	25	7.76	22.3	9.47	32.4	8.68	9.16	12.6
Copper, Cu	mg/kg	60	9.13	76.4	374	53.2	28.3	10.9	67.5
Iron, Fe	mg/kg	56,300	8,970	34,700	24,800	52,500	23,500	28,700	12,800
Lead, Pb	mg/kg	14	8.86	42	458	159	10.1	1.51	14.9
Lithium, Li	mg/kg	20	<5.0	19.1	5.1	25.2	<5.0	<5.0	<5.0
Magnesium, Mg	mg/kg	23,300	9,580	9,770	3,690	8,780	9,410	6,750	6,330
Manganese, Mn	mg/kg	950	193	1,210	1,840	2,230	184	485	600
Mercury, Hg	mg/kg	0.085	0.0101	0.0254	0.0203	0.0868	<0.0050	<0.0050	0.0089
Molybdenum, Mo	mg/kg	1.2	<0.50	1.47	3.46	2.15	3.2	<0.50	<0.50
Nickel, Ni	mg/kg	84	97.9	152	23.9	208	18.6	4.73	57.5
Phosphorous, P	mg/kg	1,050	291	705	443	643	444	389	499
Potassium, K	mg/kg	20,900	1,000	4,750	610	3,100	640	1,870	800
Selenium, Se	mg/kg	0.1	<0.20	0.47	2.88	0.9	<0.20	<0.20	<0.20
Silver, Ag	mg/kg	0.07	<0.10	0.11	0.55	0.72	<0.10	<0.10	0.13
Sodium, Na	mg/kg	23,600	<100	<100	<100	<100	120	220	<100
Strontium, Sr	mg/kg	370	79.6	43.2	35.3	26.5	20.1	26	1,270
Thallium, Tl	mg/kg	0.85	0.065	0.601	3.41	0.526	<0.050	0.112	0.067
Tin, Sn	mg/kg	2.3	<2.0	<2.0	2	<2.0	<2.0	<2.0	<2.0
Titanium, Ti	mg/kg	5,650	127	290	28.9	174	519	1,070	183
Vanadium, V	mg/kg	120	18.6	75.5	62.5	132	80.9	79.7	23.7
Zinc, Zn	mg/kg	70	30.7	117	1,010	191	46.2	29.3	197

20.7 Physical Features

20.7.1 Climate

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 neighboring provinces.

Annual average temperature is 11.4°C. The hottest month is July with an average temperature of 24.3°C and the coldest month is February with an average of -0.5°C. An extreme maximum temperature of 41.0°C is observed in July and an extreme minimum temperature of -30.0°C is observed in January.

The long-term annual average precipitation for the project site is 383.9 mm. The intensity for 24-hour 100-year storm event is 2.75 mm/hr (66 mm). The annual average snowfall depth is 51 cm, which is approximately equivalent to 75 mm of water. Annual average evaporation is 1,121.5 mm. The highest monthly average evaporation is 241.4 mm and occurs in the months of July and August. The net annual water deficit in the region is 880.1 mm (1,121.5 mm – 241.4 mm). December through March are the months with water surplus.

The annual average wind speed is 2.6 m/s. Maximum wind speeds are observed in spring. The prevailing wind direction is south.

20.7.2 Air Quality and Noise

The Project site is located in a rural area with no significant commercial or industrial air pollution sources. The closest industrial facilities are stone/marble quarries and iron/copper mines that lie at distances of 50 km or more. Scattered slag piles and ore extraction sites remaining from the former manganese mining operations are the only possible fugitive dust sources within the Sulfide Expansion Project impact area.

Emission from residential heating in settlement areas of the region (Sabırlı, Çöpler and other nearby villages, and the town of İliç) is the only gaseous air pollution source in the vicinity of the Project.

The ambient air quality monitoring program 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. According to the X-ray fluorescence (XRF) analysis of the PM10 samples for heavy metals most of the metal concentrations including arsenic, cadmium and other metals were below the method detection limits. All of the concentrations were well below the limit values defined by European Commission (EC), World Health Organization (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 noise level measured in Sabırlı village is 41.8 dBA during day time and 37.2 dBA during the night time. The noise levels at the proposed resettlement area for Çöpler village were measured as 59.4 dBA during daytime and 36.2 dBA during night time. The noise levels observed were due to community activities of the villagers.

20.7.3 Surface Water Resources

The Euphrates-Karasu River is the largest surface water body near the Project; it borders the study area from the north (Figure 20-2). Based on 32-years of records, the average annual flow rate and the maximum flow rate of Euphrates River are 145 m^3/s and 1,320 m^3/s, respectively. Peak flow rates are observed in April and May following the snow melt and rainfalls (SRK, 2008). All other streams in the vicinity of the Project area are intermittent, flowing between March and June. Streams within the Project area are the Çöpler stream with a 10 km^2 catchment area and the Sabırlı stream with a 35 km^2 catchment area.

There is ongoing construction of the Bağıştaş I hydroelectric power plant (HEPP) and Dam and Bağıştaş II Regulator Dam on the Euphrates-Karasu River. The Bağıştaş I Dam's reservoir will be at a distance of 35 m to 50 m to the new Çöpler village settlement.

The surface water quality within the site was investigated through;

- Data gathered through the Environmental Baseline Study (EBS) for the 2008-EIA of Çöpler Mine (2005-2007),
- Data gathered through the Çöpler Mine Environmental Monitoring Program (EMP) conducted by Alacer (2008-2011),
- Data gathered through the first campaign of EBS for the Sulfide Expansion Project (August, 2012).

The surface water quality sampling locations for each study is presented in Figure 20-2. Summaries of the analyses results for EBS (2005 – 2007), EMP (2008 – 2011) and EBS (2013 – 2014) are presented in Table 20-4, Table 20-5 and Table 20-6, respectively.

The analysis of the water samples from surface waters indicates calcium-sodium cation and bicarbonate-chloride sulfate anion facies types. None of the surface water is deemed suitable for drinking or irrigation purposes. Significant seasonal fluctuations in water quality are not observed for the surface water monitoring points at the site. The comparison of the average analysis results with the Turkish Water Pollution Control Regulation (WPCR) Inland Water Quality Criteria (IWQC) 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. The Çaltı stream which connects to the Euphrates River downstream and outside the Project's impact area was determined to have Class III water quality. The water quality in the Euphrates is observed to improve after the confluence with the Çaltı stream and at that point it is classified as Class III water resource. For all streams, aluminum levels are observed to be high. Iron concentration is also high especially in the drainage from Sabırlı and Çöpler creek catchments. Elevated Al and Fe concentrations in these catchments are attributed to natural metallic enrichment from the surrounding geology. Copper, manganese, nickel, and lead are other parameters that are observed in relatively elevated concentrations in the drainage from Sabırlı and Çöpler catchments with respect to IWQC.

A quarterly water quality sampling program has been developed in order to determine the baseline conditions for the Sulfide Expansion Project that included 24 water quality monitoring locations. The first sampling campaign was conducted by SRK in August 2012, representing the summer conditions, however most of the sampling locations were dry due to the season and the measurements could be conducted at only two locations. The results of the measurements supported the previous results indicating calcium-chloride sulfate anion facies types. The water quality in the downstream of Karasu River is found to be Class III.

Figure 20-2 Water Quality Sampling Locations for Çöpler Project

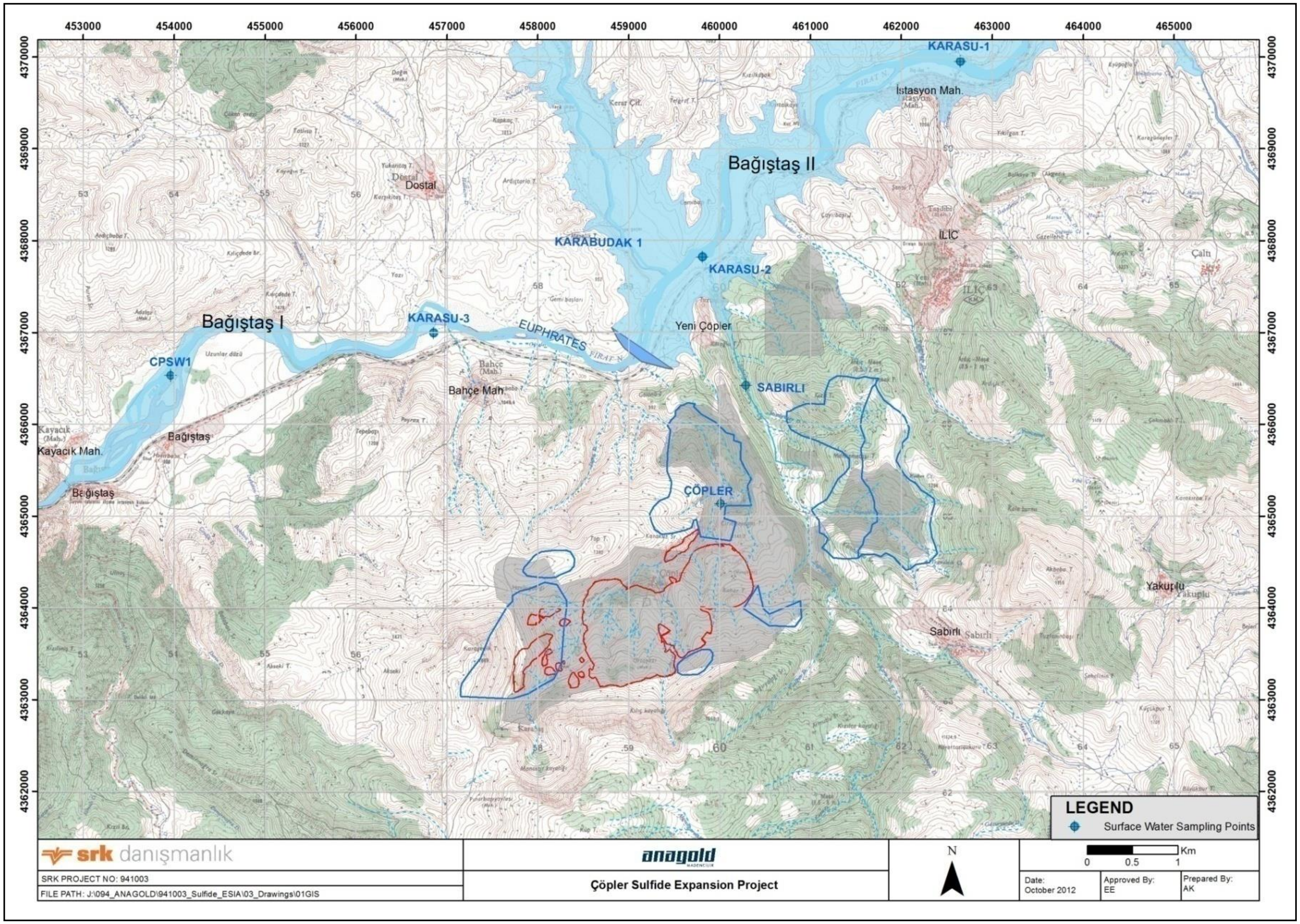


Table 20-4 EBS (2005-2006) Analyses Results

	WPCR (2004)			Drinking Water Criteria (MH, 2005)
	Class II	Class III	Class IV	
Çöpler	Ni, Hg, Cr, TP_2, TDS, COD	SO_4, Mn, Pb, N-NO_2	Al, As, Cu, Fe	SO_4, As, Mn, Ni, Pb, Sb
Sabırlı	Cu, COD	Mn, Ni, , TP	Al, Fe, N-NO_2	Al, Fe, Mn, Ni
Karasu (1)	TP, Cl, Fe, BOD,	TKN, N-NO_2	Al	Al, Ni, Sb
Karasu (2)	TP, BOD, Fe	TKN, N-NO_2	Al	Al, Fe, Mn, Ni, Sb
Karasu (3)	TP, Fe	TKN, N-NO_2	Al	Sb

Table 20-5 EMP (2008-2011) Analyses Results

	WPCR (2004)			Drinking Water Criteria (MH, 2005)
	Class II	Class III	Class IV	
Karasu-2	COD, Co, Cu, Mn	Cr, Ni	Al, Fe	Al, Cr, Fe, Mn, Ni
Karasu-3	COD, Co, Cu	Cr, Mn	Al, Fe, Ni	Al, Cr, Fe, Mn, Ni
Çöpler	TCN, TOC	Co	Al, As, Ba, COD, Cd, Cr, Cu, Fe, Mn, Ni, Pb, SO_4, Se, Zn	Al, As, Cd, Cr, Cu, Fe, Mn, Ni, Pb, SO_4, Se
Sabırlı	COD, Cr	As, Cu, Mn, Ni, Pb	Al, Fe, Se	Al, As, Fe, Mn, Ni, Se

Table 20-6 EBS (2012-2015) Analyses Results

	WPCR (2004)			Drinking Water Criteria (MH, 2005)
	Class II	Class III	Class IV	
Karasu-2	Al, Fe, P, N-NO_2	Al, N-NO_2		Al, Fe
CPSW-1	P, N-NO_2	N-NO_2		

20.8 Land Use

The prevalent land use and cadastral information in the Sulfide Expansion Project and its environs is presented in Figure 20-3. The land use patterns are based on maps produced by the General Directorate of Rural Services. As observed in Figure 20-3, most of the Project area consists of pasture land, forest and rocky areas.

The Land Use Capability Classes (LUCC) for the Project area and environs is given in Figure 20-4. 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 utilization 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 20-4, 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

The ownerships covering the greatest area of the project site are the Çöpler village pasture land with 29%, the Sabırlı village forest land with 24% and the lands owned by the Treasury within the borders of the Çöpler village with 19%.

Figure 20-3 Current Land Use Types and Cadastral Map for the Çöpler Sulfide Expansion Project

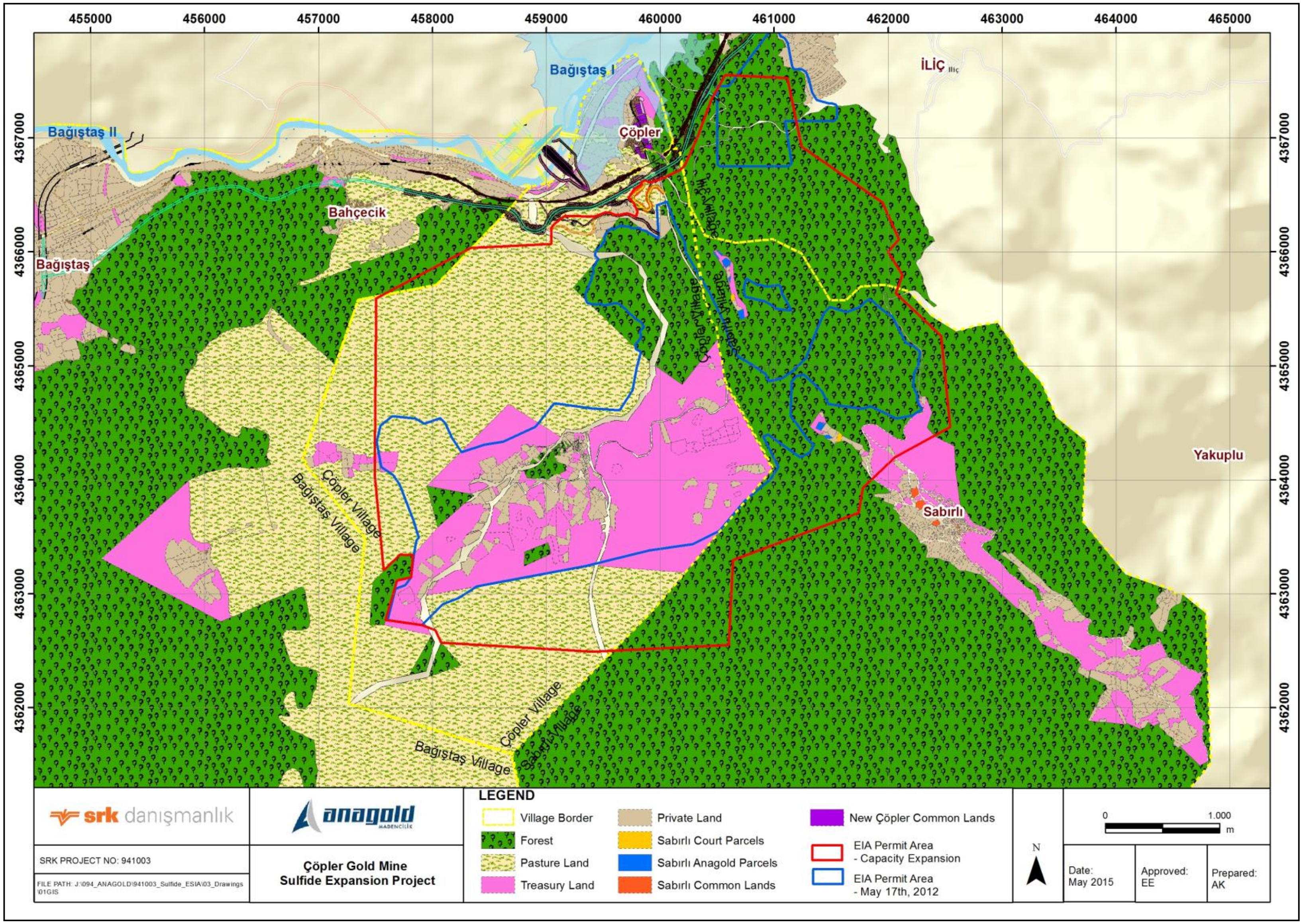
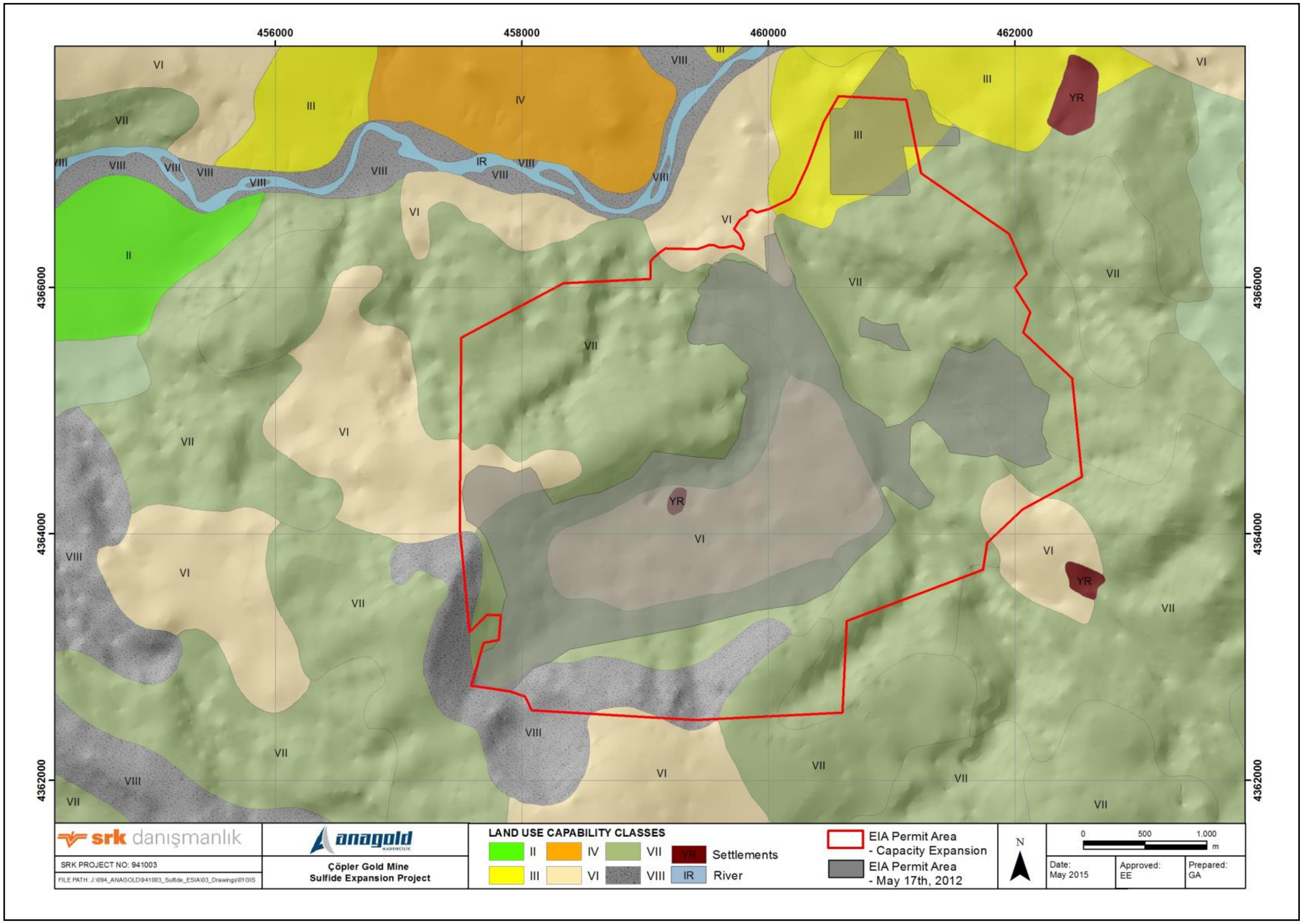


Figure 20-4 Land Use Capability Classes (LUCC)



The Project area and surroundings that are covered by B soil group, and rocky areas (ÇK) are generally of low land use capability and not suitable for agricultural activities. Typical site views from the Project area are shown on Figure 20-5 and Figure 20-6. 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. Much of the forestry lands are highly damaged and have lost their growth and budding capabilities, leading to mass flows and erosion.

Within the permitted mine area, forests cover 130 ha. The expansion of TSF will be on the forest area which are qualified as low density forest region and will totally cover 194.61 ha. The major tree types in the expansion area are Juniper and Oak as seen in Figure 20-7.

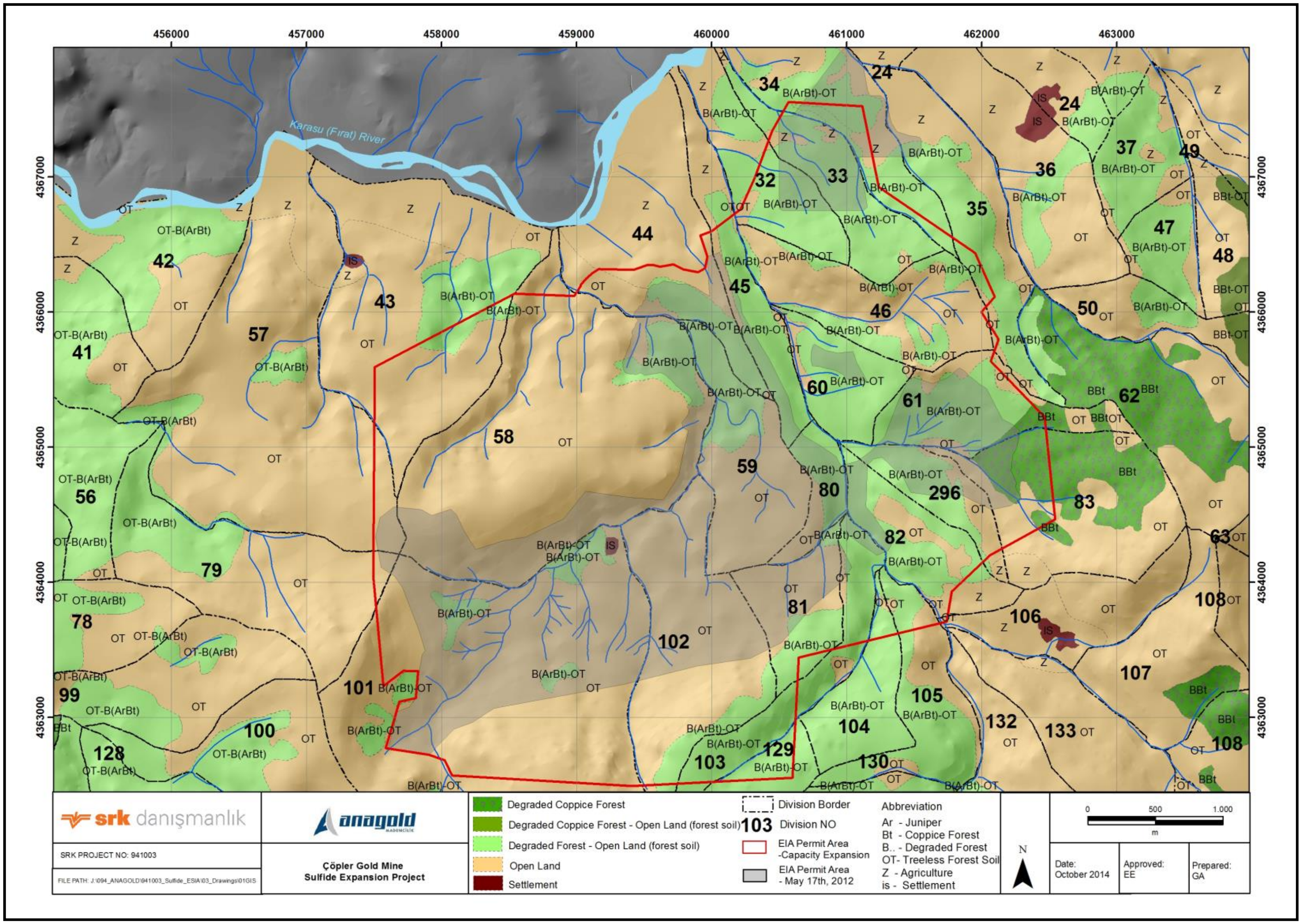
Figure 20-5 Site View from the Project Area - 1

Photograph by SRK, 2015.

Figure 20-6 Site View from the Project Area – 2

Photograph by SRK, 2015.

Figure 20-7 Forest Area Types

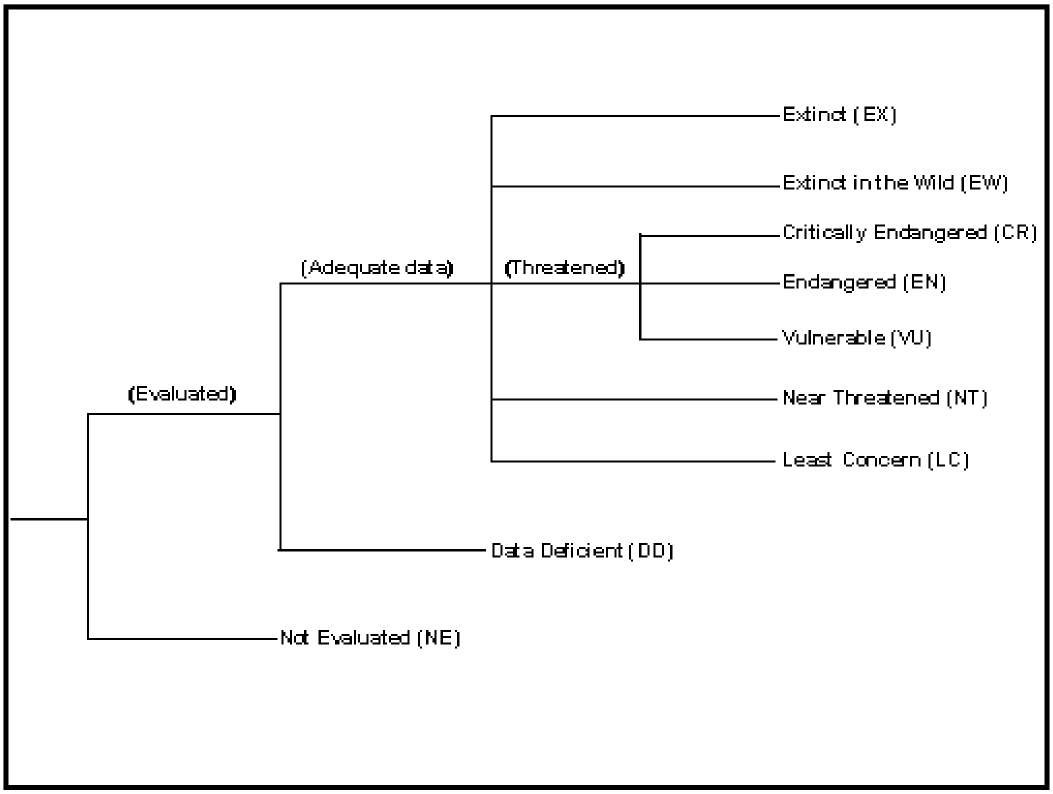


20.9 Biological Features

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 (Figure 20-8).

Figure 20-8 Structure of the IUCN Risk Categories



Note: Figure from IUCN, 1994

As a result of field surveys carried out within the Çöpler biodiversity study area a total of 322 taxa were identified. Approximately 47 of these identified species are endemic and rare, and 14 out of 47 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, wooded steppes and rock habitat, while the rest of the site is designated for main mining activities. There are no suitable places for reproduction, nursing or feeding of habitats as the area has a destroyed habitat structure. Thus, the faunal composition of the site is considered weak.

20.10 Socio-Economic Features

This section presents the general socio-demographic and economic characteristics of the settlements to be affected by the Çöpler Sulfide Expansion Project. The project will primarily affect Çöpler, Sabırlı, Bağıştaş, Dostal and Yakuplu villages within the İliç district of Erzincan Province. The villages are acknowledged as the locally affected communities.

According to 2013 results of the Address-based Population Registration System (APRS) of the Turkish Statistical Institute (TSI), the population of the Province of Erzincan with an area of 11,903 km2 is 799,724. There has been a small population increase when compared with the 2012 population of 789,750. As of 2013, the population growth rate for the Erzincan Province is 9.6‰ (9,974 persons). The 2013 APRS data published by TSI shows that the provinces receiving most of the migration from Erzincan and the provinces contributing to the majority of migrants to Erzincan are Istanbul, Ankara and Erzurum, and the net migration rate of Erzincan is 1.01%. According to the 2013 APRS results, the population of İliç is 7,367.

In general, in the high and mountainous regions of Erzincan, the main economic activity is animal husbandry. Orchards and gardens are widespread on the west side of the province. Dry land agriculture becomes a common agricultural activity in the mountain areas. Erzincan is also suitable for animal production; however, it is not able to realize its potential due to marketing problems. Wheat and barley production fulfills the needs of the province while other products constitute an important share of domestic production. These include sugar beet, dry bean, potato and feed crop. Apricots, plums, cherries, berries, quince, walnuts, almonds and apples are being grown in Erzincan. Milk production shows an increase in spring and summer seasons. Erzincan is convenient both for ovine and ovine breeding as the amount of meadow and pasture areas are higher than Turkey’s average amounts. Ovine breeding pastures are especially important sources of animal feed. “Caucasus Hybrid” is the most common bee species found in Erzincan. A total of 9-17 kg of honey yield is being obtained per beehive from both migratory and stationary apiculture.

The economy of İliç depends on agriculture, farming, dairy products and mining. Additionally, apiculture and honey production are also among the important sources of living in İliç and its vicinity.

Unemployment rates are higher outlying in districts compared to the city center. The unemployed population in Erzincan is primarily comprised of K-8 education graduates and secondary school graduates. The unemployment rates are higher for the age group of 20-34 and the number of registered unemployed males exceeds the number of registered unemployed females.

Table 20-7 presents the populations and number of households in the villages located within the project impact area. From the table it can be seen that the six villages have a combined population of 923 individuals in 211 households. This equates to an average household size of 4.4 persons. Çöpler and Sabırlı are the only villages of significant size and account for 82% of the combined population. When the number of households of Çöpler in 2012 and 2014 is compared, it is seen that there is huge increase in the number of households (from 37 to 61). This is mainly because of the mining activities. As the socio-demographic and economic assessment of the KORA in 2013 suggests, those who have been living in the village less than ten years is the highest in Çöpler with 30 %. It can be claimed that the same trend can also be traced in Sabırlı. The remaining four villages have a very small resident population and may be considered hamlets rather than fully established villages. The mine has also had an impact on the household structure. From the focus group discussions and the HH survey it is clear that, whereas in the past, in Çöpler and Sabırlı villages, extended families lived together in the rural villages. Currently these extended families have broken up into nuclear families, with those employed by the mine migrating to İliç center, where the standard of living is slightly higher. In Bağıştaş, Bahçecik, Dostal and Yakuplu villages, the majority of the households are also of a nuclear nature.

Table 20-7 Population and Number of Households of the Project Affected Villages

Village	Total Households	Total Population	Avg. Household Size
Bağıştaş	26	62	2.4
Bahçecik	10	31	3.1
Çöpler	61	255	4.2
Dostal	23	57	2.5
Sabırlı	78	495	6.3
Yakuplu	13	23	1.8
TOTAL	**211**	**923**	**4.4**

Source: UDA Consulting, HHS – December 2014

People of the Çöpler and Sabırlı villages are originally from the Kurdish Şavak tribe and they were resettled to their recent villages from Elazığ Province following the construction of the Keban Dam in 1973. These two villages are Sunni. The villages of Bağıştaş, Dostal and Yakuplu are Alevi Turkish villages.

In the İliç District the percentage of illiterate people is more than double the national rate at 9.6%; the representation of females among this group is similar to the national proportion at 82.3% (TurkStat, 2013). Table 20-8 shows the number of illiterate people (disaggregated according to sex) for the villages in the study area.

Table 20-8 Illiteracy Rates in the Study Area

Village	Sex	Number of Illiterate People	Percent in the Total Population
Sabrili	Female	34	7.8
	Male	4	0.9
Çöpler	Female	14	5.5
	Male	2	0.7
Bağıştaş	Female	6	9.6
	Male	2	3.2
Dostal	Female	6	10.5
	Male	4	7.0
Yakuplu	Female	1	4.3
	Male	0	0.0
Bahçecik	Female	3	9.6
	Male	2	6.4
TOTAL		**78**	

Source: UDA Consulting, HHS – December 2014

The main economic activities in Çöpler and Sabırlı are based on animal husbandry and apiculture. However, animal husbandry has ended in Çöpler due to the village resettlement resulting from mining activities. The villagers prefer to work at the mine and they have sold all their animals. The situation is almost the same in Bağıştaş village. In Bağıştaş, agriculture especially wheat and barley production, is a very common activity. There are only 3-5 households which are engaged in apiculture with total number of 150 beehives. The people in Yakuplu are mostly retired and/or working as artisans (carpentry, glassmaking, etc.).

The housing structure is similar within the region. New buildings constructed in the villages are primarily built of concrete and older houses are built from adobe. All of the villages mentioned above have their own elementary schools; however, for high school education they are subject to mobile education. All of the villagers benefit from the services of a state run hospital in İliç. For urgent treatments, patients are being referred to other hospitals in the Erzincan city center.

The Çöpler, Sabırlı, Bağıştaş and Yakuplu villages have sewerage systems; however, in Dostal village no sewer system exists. Water supply networks do exist in Çöpler, Sabırlı, Dostal, Bağıştaş and Yakuplu. One of the common infrastructure problems of the settlements is the waste management. The villagers burn or throw their wastes into uncontrolled disposal areas away from their settlements.

All villages have access to electricity for lightning and electronic devices and to land line communications. However, it is important to mention that land line subscriptions of the households are frequently cancelled due to high costs and people prefer to use their mobile phones for communication.

A comprehensive Social Impact Assessment (SIA) report and a Stakeholder Engagement Plan (SEP) have been prepared and issued under the ESIA reporting in September 2015.

20.11 Risks and Opportunities

The EIA permit for the Sulfide Expansion Project was obtained in December 2014 providing assurance for the permittability of the Çöpler Mine impacts. SIA and SEP studies have been conducted for the Project and were provided in the annex of ESIA report (September, 2015). According to the SEP, the main stakeholder concerns can be summarized as follows:

- Tensions in the relationship between Anagold and its stakeholders. This relates to concerns about lack of employment, lack of sufficient communication and information provision and lack of support for local development. These issues are being addressed through the overall stakeholder engagement and community development programs.
- Changes in local lifestyle as a result of changes in economic activity (from farming and remittance-based livelihoods to mine employment) and resettlement (Çöpler) to a more urbanized environment. This is currently also addressed in the overall stakeholder engagement and community development programs (i.e. women's projects).
- Loss of grazing land. This will need to be addressed in a land acquisition plan. The SEP report further provides a plan for the ESIA feedback consultations to be conducted in June 2016 and outlines stakeholder activities planned for the post ESIA period. The SEP report also discusses Anagold's grievance mechanism.

The SEP will be updated after the ESIA feedback activities to provide a full record of all ESIA stakeholder activities, and will then be incorporate into Anagold's overall SEP.

20.12 Conclusions and Recommendations

At the current stage of the Project, the ESIA and related technical studies have been completed. The results of these studies determined the potential impacts of the Sulfides Expansion Project. Further conclusions and recommendations can only be made following the achievement of these results.

It is recommended that during the next stage of the Sulfides Expansion Project, the environmental and social impacts of the Project be reviewed utilizing the information provided in these studies.

20.13 Mine Closure and Sustainability

This section presents a conceptual closure plan for the proposed Project. The closure activities outlined in the following sections are largely based on the requirements set out in the 2014 EIA (SRK, 2014) and a closure plan prepared by Anatolia in 2009 (Anatolia, 2009). These requirements are applied to the proposed facility arrangements and represent closure of the proposed land disturbances. Completion of the EIA for the Sulfide Expansion Project included additional commitments and obligations which are provided in the commitments register prepared by SRK. The objectives of this closure plan are based on compliance with current regulations and steps outlined in the approved EIA.

20.13.1 Legal Requirements

The Turkish "Regulation on Reclamation of Lands Disturbed by Mining Activities," published in December 2007 and amended in January 2010, requires that the operator abandon the site in a state that is physically, chemically, and biologically stable and which allows beneficial use by the public. The regulation does not prescribe mandatory closure methods, but rather lays the legal foundation for reclamation. At present, the regulation requires reclamation plans be submitted to the agencies in parallel with an EIA and permitting process. There are no regulatory requirements to submit closure cost estimates, post financial assurance, or conduct community consultation with respect to post-mining land use expectations. The Mining Waste Management Regulation, passed in July 2015 and to be implemented one year after publishing, mentions “financial guarantees” for Category A facilities, which Çöpler will not be operating; therefore, such requirements do not apply to the Çöpler operation at this time. There are regulatory studies that are aimed to transpose other EU directives, such as the Environmental Liability Directive, into Turkish environmental legislation. The timing for the transposition of these EU directives is currently unknown. Alacer currently has a reclamation plan in the appendices to the approved EIA report (2014) prepared in accordance with the Regulation on Environmental Impact Assessment for the Sulfide Expansion Project, which includes a reclamation plan in the format required by the Turkish Ministry of Environment and Urban Planning.

20.13.2 Rehabilitation Objectives/Sustainability

The proposed sulfide expansion is an expansion of the existing mine facilities. The pits, facility areas, heap leach and waste rock dumps will get larger. The proposed TSF will be an expansion of the previously permitted (but not constructed) facility.

The goals of reclamation and closure at the Project consider the objectives of sustainable development. These objectives include mitigation of the effects of land disturbances by minimizing or eliminating public safety hazards, providing long-term stable landform configurations, and reclaiming surface disturbances for ongoing beneficial use. The reclamation and closure process will be consistent with local land use objectives, the Project’s EIA, World Bank/IFC guidelines pertaining to reclamation and closure, and the Turkish Regulation for Reclamation of Mined Land (Official Gazette No. 26730, 2007).

The criteria listed below will be used to assess the compatibility of the rehabilitation works with the purpose of the Çöpler Mine.

20.13.3 Physical Stability

- Long-term stability of engineered structures such as the tailings impoundment
- Removal and proper disposal of all access roads, structures and equipment not required following the cessation of mining activities
- Long-term stabilization of all exposed erodible materials

20.13.4 Chemical Stability

- ARD prevention, control, and treatment
- Long-term preservation of water quality within and downstream of decommissioned operations
- Human health and safety

Specific objectives for each of the proposed facilities are discussed in the following sections.

20.13.5 Open Pits

- Ensuring the safety of local residents and livestock
- Development of stable post-closure pit wall conditions
- Protecting surface water and groundwater resources

20.13.6 Waste Rock Dumps

- Development of stable slopes capable of withstanding seismic events
- Placement of the cover and the development of a self-sustaining vegetative cover that supports the identified post-closure land use and controls erosion and sedimentation
- Limiting infiltration of surface water into the waste rock
- Limiting the quantity of drainage from the waste rock and its potential impact on surface water quality
- Limiting seepage to groundwater

20.13.7 HLF

- Stabilize the HLF to prevent wind and water erosion
- Establish a cover to limit infiltration into the HLF
- Establish surface drainage to limit run-on that is capable of passing the run-off from the design storm event
- Establish self-sustaining vegetation consistent with the proposed post-closure land use
- Maintain physical stability of the fill embankments under static conditions and dynamic loading conditions corresponding to the design earthquake loading
- Maintain chemical stability of the leached ore
- Minimize impacts to local water resources as a result of seepage from HLF
- Manage drain-down within the HLF area

20.13.8 Tailings Storage Facility

- Establish a surface configuration that will not impound water or define activities to remove impounded water from final reclaimed surface
- Waters from a probable maximum flood (PMF) storm can be passed safely off the impoundment surface either through positive drainage or activities designed to remove water from final reclaimed surface
- Establish a cover to limit infiltration into the tailings and migration of metals upwards into the vegetative cover
- Establish surface drainage to limit run-on that is capable of passing the run-off from the design storm event

- Establish self-sustaining vegetation consistent with the proposed post-closure land use
- Maintain physical stability of the fill embankments under static conditions and dynamic loading conditions corresponding to the design earthquake
- Maintain chemical stability of the tailings
- Minimize impacts to local water resources as a result of seepage from the TSF
- Manage surface water within the TSF area

20.13.9 Infrastructure

Buildings:

- Elimination of physical and chemical hazards;
- Demolition and removal of structures where a post-mining land use has not been identified
- Development of a self-sustaining vegetative cover that supports the identified post-closure land use and controls erosion and sedimentation
- Testing for and removal of any areas of contaminated soils

Yards:

- Development of a self-sustaining vegetative cover that supports the identified post-closure land use and controls erosion and sedimentation;

Roads:

- Removal and reconstruction of the approximate pre-mining contours where a post-mining land use has not been identified
- Development of a self-sustaining vegetative cover that supports the identified post-closure land use and controls erosion and sedimentation

20.14 Long Term Water Management

In order to meet the closure objectives to protect surface and groundwater from environmental degradation various strategies for managing water will be employed as discussed below.

The 2014 EIA discusses rapid filling of the pit lake over the course of three years. For the purposes of the closure cost estimate SRK has assumed pit lake treatment will be required in conjunction with the rapid refill.

20.14.1 Open Pits

SRK has assumed a one-time treatment of the pit lake will be required. Based on experience at other sites, SRK has assumed that each 1,000 L of pit lake water will be treated with ½ tonne of CaO. The lake level will equilibrate at 1050 m amsl in approximately 48 years.

20.14.2 Heap Leach

Closure of the heap leach pad will include the following activities.

- Recirculation of process solutions to recover residual gold and reduce the inventory will continue for up to 18 months.
- Active management of the solution drainage from the heap leach through forced evaporation for up to one year.
- Converting the process ponds to an evapotranspiration (ET) cell and passively manage the solution drainage from the heap.
- Re-grading the heap leach pad to a final slope of 2H:1V and construct new liner where required (assumed to be 50% of the perimeter).
- Placing one meter of fine-grained cover material on the surface of the heap leach and (if available) this will include any topsoil salvaged from the footprint of the facility. If topsoil is available, it will be placed at the top of the cover to facilitate re-vegetation. The one-meter cover will act as a store and release cover to minimize infiltration of surface water into the heap leach pad.

20.14.3 Tailings

The tailings impoundment will be constructed with an over-drain system and an under-drain system. Collected overdrain fluids will report to a sump constructed within the tailings impoundment at the upstream facing toe of the TSF embankment. Overdrain fluids will be transferred to a pipe, which will drain by gravity to a sampling port at the downstream facing toe of the TSF embankment and then to a holding pond located north of the TSF facility. Underdrain flows will flow through granular materials to a sampling port located at the downgradient toe of the TSF embankment. Flows will then be transferred to a pipe and flow via gravity to a holding pond adjacent to the overdrain pond, north of the TSF. During operations, seepage from the tailings will be pumped from the holding pond back to the TSF, and recycled back to the mill water requirements (Golder 2014d).

Following tailings operations, the over-drain seepage will be pumped from the holding pond and actively evaporated through misters or snowmakers with any balance applied to the tailings surface via sprinklers. The snowmakers will be located on the dam crest. A tailings closure water balance has not been completed at this time.

Given the settlement estimates for the TSF following the end of deposition (Golder 2014d), and that a strategy to prevent water from ponding on the TSF has not been developed at this time, SRK assumes that precipitation falling on the tailings surface will create a pond during the wet months and require pumping to a nearby drainage for removal. The duration for this requirement is assumed to be in perpetuity.

20.14.4 Waste Rock Storage Areas

Seepage from the WRSAs will be directed into the pits and treated as described for pit lake water. Diversions will be constructed to direct run-on surface water away from the waste rock facilities. In most cases the water will be directed to the open pits, however, the south waste rock diversion will empty into a settling pond and be directed to the Çöpler Creek.

Because the toe of the North WRSA is topographically below the pit crest, the seepage will need to be captured in a pond and pumped into the pit. For closure costs a small lined pond is assumed with a 2 km length of HDPE piping.

20.14.5 Surface Water

Diversions will be constructed to direct surface water away from the mine facilities. No treatment is contemplated for surface waters.

20.15 Closure Approach and Basis for Closure Cost Estimate

20.15.1 Open Pit

As discussed in Section 20.14, SRK has assumed a one-time treatment of the pit lake will be required. Based on experience at other sites, SRK has assumed that each 1,000 m^3 of pit lake water will be treated with ½ tonne of CaO. The lake level will equilibrate at 1050 m amsl in approximately 48 years. Per the EIA (2014), water can be pumped into the pit for rapid fill over the course of three years.

20.15.2 Tailings

The closure of the tailings will include the following tasks:

- Active management (evaporation) of over-drain seepage for 20 years.
- The tailings will be allowed to consolidate and dry for two years prior to placement of a traffic layer of waste rock which will be one meter thick. This rock layer will allow the operation of small equipment on the surface and function as a capillary break to prevent tailings water migrating into the surface cover.
- Given the settlement depths on the tailings surface based on the Phase 7 TSF design as provided in the TSF Design Report (Golder, 2014d), placement of the cover layer will not be sufficient to provide positive draining of the surface and ponding is expected, requiring an assumption for long-term pumping of water around the wet season to remove the pond. A conceptual closure diversion upstream of the TSF was developed by Golder as part of the Flood Management Plan (Golder, 2013a). The design of this channel has been included in the TSF costs.
- One meter of fine-grained cover material will be placed on the surface of the waste rock cover and (if available) will include any topsoil salvaged from the footprint of the facility. If topsoil is available it will be placed at the top of the cover to facilitate re-vegetation. The fine-grained cover will act as a store and release cover to minimize infiltration of surface water into the heap leach pad.
- The surface will be re-vegetated with a seed mix including native grasses and bushes. Trees will not be planted because they might cause damage to the cover.

20.15.3 Heap Leach Pad

Closure of the heap leach pad will include the following activities.

- Recirculation of process solutions to recover residual gold and reduction of the inventory will continue for up to 18 months as discussed in Section

20.14.2 (this is assumed to be an operational cost and not applied to closure).

- Active management of the solution drainage from the heap leach through forced evaporation for up to one year.
- Converting the process ponds to an ET cell and passively manage the solution drainage from the heap.
- Re-grading the heap leach pad to a final slope of 2H:1V and construct new liner where required (assumed to be 50% of the perimeter).
- Placing one meter of fine-grained cover material on the surface of the heap leach and (if available) this will include any topsoil salvaged from the footprint of the facility. If topsoil is available it will be placed at the top of the cover to facilitate re-vegetation. The one-meter cover will act as a store and release cover to minimize infiltration of surface water into the heap leach pad.
- Revegetation of the surface with a seed mix including native grasses and bushes. Trees will not be planted because they might cause damage to the cover.

20.15.4 Waste Rock Storage Areas

Closure of the WRSAs will include the following activities.

- Grading the slopes to 2.5H:1V.
- Placing one meter of fine-grained cover material on the surface. If available this will include any topsoil salvaged from the footprint of the facility. If topsoil is available it will be placed on top of the cover to facilitate re-vegetation. The one-meter cover will act as a store and release cover to minimize infiltration of surface water into the WRSA.
- Diversions will be constructed to direct run-on surface water away from the waste rock facilities. In most cases the water will be directed to the open pits, however, the south waste rock diversion will empty into a settling pond and be directed to the Çöpler Creek.

20.15.5 Yards and Roads

Closure of other surface facilities like Yards and Roads will include the following activities;

- Roads which do not have an identified post-mining land use will be removed and the ground graded to the approximate pre-mining slopes. Road construction often uses native surface soils which when placed back in the road bed will be sufficient for re-vegetation. No additional cover will be planned for road rehabilitation.
- Flat areas like yards will be graded back to near natural pre-mining conditions. In places where cut and fills were substantial this may include placement of fine-grained cover materials or topsoil if available.

20.15.6 Processing, Administration and Other Infrastructure

Process buildings and other infrastructure without a defined post-mining use will be demolished. The mill buildings, tanks and related piping will be decontaminated by rinsing with fresh water prior to demolishment. Sampling and testing will be done for removal of any areas of contaminated soils.

Materials with scrap value will be separated and hauled offsite for recycling. Although recent experience proves there is value in the metal from mill buildings no offset of the closure costs will be assumed. All other materials without scrap value will be hauled to a nearby licensed landfill.

20.15.7 Other Facilities

Water Wells

Production and monitoring wells will be abandoned after they are not needed for water supply or monitoring purposes. Pumps (if any) will be removed. A drill rig will be used to pump grout into the well to seal it.

Power Lines

Power lines without a post-mining land use or which do not serve customers outside the mine area will be removed. Metals will be hauled offsite for recycling. Closure costs have not been offset by scrap value.

Fences

The project perimeter fence will be removed at the end of the rehabilitation. Sections of the fence around the tailings will remain in place to prevent access.

Solid Wastes

Non-hazardous construction debris will be hauled offsite to a nearby licensed landfill. All solid and hazardous wastes present at the end of mine life will be disposed offsite by licensed handling companies. The autoclave will generate a large amount of brick material which will likely be considered hazardous waste. Information is not available at this time to estimate the quantity of waste. A cost of $100,000 has been included in the cost estimate as a provision for this cost until more information is developed.

Quarries

There are two limestone quarries proposed. SRK has assumed they will be covered with one meter of fine-grained material and any topsoil stripped from the footprint will be applied on the surface of the cover. The area will be re-vegetated with native grasses and bushes.

20.15.8 Post Closure Monitoring and Maintenance

According to Turkish regulations post-closure monitoring will be conducted for 30 years following the cessation of mining. These activities include collecting surface water and groundwater samples and monitoring the site for erosion or other geotechnical issues. A provision for this program has been included in the closure cost estimate.

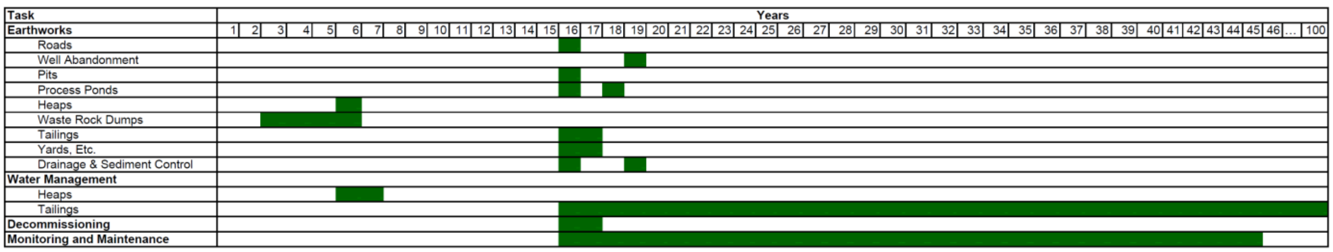
Because of the settlement in the tailings impoundment surface, ponding will occur. Pumping to remove the ponding water will be required for the long term to maintain the physical stability of the facility and this will require a long-term presence at the site to maintain the system. Provisions for this have been included for 100 years.

20.15.9 Closure Schedule

The currently approved oxide mining operations are expected to end in 2022. Sulfide mining operations are expected to end in 2023 with processing of stockpiled ore through 2037. Following the cessation of mining it is expected the active closure period (earthworks) will continue for up to five years, water management for up to 100 years and the post-closure monitoring will continue for a total of 30 years (concurrent with earthworks and water management).

A schedule for major activities is provided in Table 20-9.

Table 20-9 Closure Schedule



20.16 Closure Costs

20.16.1 Closure Cost Unit Rates

Equipment

Equipment costs were based on costs provided by the mining contractor (Ciftay) obtained in December 2015. The costs quoted include the following elements:

- Ownership costs
- Preventative maintenance
- Ground engaging tools
- Fuel
- Tire wear (where applicable)

The Standard Reclamation Cost Estimator (SRCE) model has been used reclamation and closure cost estimation. The SRCE cost model automatically calculates fuel consumption as part of the equipment rates. Therefore, fuel consumption costs were backed out of the provided equipment rates based on fuel burn rates provided by Ciftay. The current mining contactor is assumed to be available to conduct earthworks for closure.

For equipment not available in the mining fleet costs were assumed to be equivalent to US rental costs.

The contractor uses 36 t capacity over-the-road dump trucks. Because the cost model SRK uses is standardized on Caterpillar equipment we have assumed a Caterpillar 735 for calculations.

Because the closure will be performed by the mining contractor SRK has not included equipment mobilization fees.

Labor

Labor rates including benefits, taxes, and insurance were provided by Jacobs Engineering as used in the FS.

Other

Material costs for fuel and electricity were obtained from Alacer in 2015. For miscellaneous items which are used in small quantities in the cost calculation, rates were obtained from RS Means (2015) or other US sources.

20.16.2 Closure Cost Productivities

Productivity data and calculations were based on:

- Caterpillar Handbook, Edition 35 for productivity calculations as incorporated into the SRCE;
- Means Heavy Construction Handbook for productivities/crews was used for demolition
- Where first principles methods for calculating productivities were not applicable or adequate information was not available, best professional judgment of local engineers and data from similar projects was used.

20.16.3 Key Assumptions

Key assumptions used to complete the closure cost estimate are provided below.

- The reclamation slope for heaps and dumps is 2.5H:1V, (Anatolia 2009 and SRK 2014).
- 50% of the perimeter of the heap leach will require additional liner (4 m) to accommodate re-grading the heap leach slope to 2.5H:1V. Costs for liner materials and installation were provided by Golder (2014d) and were inflated by 50% to account for the increased work required for smaller installations.
- A total of 7.4 Mm^3 (minus existing stockpiled growth media) of cover soil would be obtained from a developed borrow area and temporarily stored near the foot of the tailings dams. All haul distance calculations are based on this assumption.
- Tailings long-term fluid management was calculated for 20 years based on the consolidation and draindown calculated for the Phase 5 TSF (Golder, 2014d). Further work is required to determine if this assumption is still valid for the Phase 6 and Phase 7 design.
- In addition to the one meter of waste rock another one-meter average thickness of waste rock will be applied to the tailings surface. The source of this material will be the haul road crossing the Sabırlı Creek.
- The placement of cover and growth media material on the tailings impoundment surface will not be sufficient to create a positive drainage surface as the settlement at the surface of the tailings is estimated to continue up to approximately 27 m around the center of the tailings impoundment surface (roughly above the location of the upstream toe of the embankment) by Year 100 after end of deposition. Long-term pumping of water will be required to maintain physical stability of the facility.

20.16.4 Final Closure Costs

The closure cost estimate includes both a closure and post-closure period. The closure period includes the physical reclamation activities and is scheduled to last four years. The post-closure monitoring period will last 30 years and long-term maintenance and fluid management for the TSF is scheduled for 100 years.

The total closure costs estimated for the Sulfide Expansion Project is US $88.8 million. This includes a contingency cost of 20% and contractor profit of 15%. The closure costs through December 31, 2121 are provided in Table 20-10.

Table 20-10 Closure Costs

Breakdown of Total Closure Cost	Total Cost
Infrastructure Demolition	$ 4,086,924
Waste Rock Earthworks	$ 10,854,188
Heap Leach Earthworks	$ 1,261,690
Tailings Earthworks	$ 12,132,134
Misc. Earthworks	$ 2,903,193
Revegetation/Drainge Control	$ 701,572
Tailings Draindown Management	$ 2,394,059
Tailings Surface Pond Management	$ 80,400
Heap Draindown Management	$ 95,725
Pit Lake and WRD Water Management	$ 3,820,934
Misc. Water and Waste Management	$ 2,654,194
Monitoring	$ 3,097,734
Construction Management	$ 775,908
Closure Planning, Camp, Property Holding	$ 2,608,487
Human Resources (incl. water mgmt crew)	$ 13,803,925
Contingency (20%)	$ 12,254,213
Other G&A	$ 6,127,107
Contractor Profit (15%)	$ 9,190,660
Total	**$ 88,843,047**

20.17 Risks and Opportunities

The SRK closure team has reviewed the available information and conducted a site visit to review the proposed locations of the project facilities.

Based on our work we have identified the following potential risks to a successful closure of the mine;

- There is a lack of available topsoil at the site. A study by the Istanbul Technical University identified a range of between 0 to 30 cm of poor quality topsoil was available within the footprint of the facilities at the site. However, the EIA and 2009 Closure Plan call for the placement of 1 m of topsoil on disturbed areas. It is likely the project will not be able to comply with this requirement because of this disconnect between available topsoil and the quantity required. For the closure cost estimate, SRK has assumed that material will be sourced from the borrow areas designated in the mine plan.
- Because of the steepness of the terrain at the site there are limited locations to store topsoil. Currently topsoil is stored adjacent to the administration area on the west side of the fill slope. This topsoil will need to be relocated prior to construction of the fill for the sulfide mill. Some topsoil is also stored

on the top of the south waste rock dump which might also need to be moved prior to building this dump to its capacity.

- Due to the design of the heap leach side slopes and the required reclamation slope of 2-2.5H:1V it is possible that either ore will need to be offloaded or the liner will need to be extended for closure. Unit rates for these activities were provided by Golder.
- There is a roughly 110 m high angle of repose fill slope adjacent to the heap leach on the east side. The crest is within 27 m of the toe of the heap leach and the toe is about 10 m from Sabırlı Creek. It will not be possible to re-grade this slope to meet the reclamation goals for slopes. It will also not be possible to place topsoil or re-vegetate this slope.
- The tailings impoundment operating design specifies the supernatant pond will be located against the hill on the east side with tailings deposition on beaches to the north, west and south. This design will require additional fill to create a positive draining surface that will not pond water. Since the fill will need to be placed in the supernatant pond area it will also be difficult to predict the amount of fill required to overcome differential settlement in that area.
- Based on discussions with the engineering team designing the process and tailings systems, it is likely the tailings at closure will not support equipment and may not do so for quite some time. This requires the placement of a traffic layer of waste rock and or placement of geosynthetic fabric in order to place the final cover.
- The consolidation of a draindown from the tailings for the Phase 7 design have not been calculated. The Phase 7 TSF crest elevation will be 1264 mamsl. The larger footprint and greater volume of tailings could increase the quantity of seepage and time required for the TSF to draindown, and increase the quantity of long-term seepage. Additional analysis will be required to determine if this would affect the costs estimated for managing draindown during and after closure. This should be evaluated as part of the preparation of a closure water balance.

Based on SRK's site visit and discussions with mine staff SRK has identified the following potential opportunities.

- It may be possible to change the required reclamation slope for the heap to 2H:1V to minimize the amount of offloading and/or liner extension. A geotechnical study should be done to establish whether this steeper slope would meet the reclamation objectives.
- Consideration should be made whether the supernatant pond could be moved across the dam surface to the west as the tailings elevation approaches final design. This would allow the placement of less fill to create a positive draining tailings surface.

20.18 Conclusions and Recommendations

Closure of the Çöpler Mine will include decommissioning all facilities without an identified post-mining land use, grading of all fill slopes to 2.5H:1V, covering facilities with a reclamation cover and re-vegetating the surface of all disturbed areas. It is anticipated the earthwork and decommissioning work will take four years and cost US$31.9 million. Treatment of pit lake water will be conducted on an ongoing basis and will cost an estimated US $3.8 million. Process fluid management of the seepage from heaps and tails will be conducted in a zero-discharge fashion and will cost US $14.0 million. Monitoring is estimated to cost US $3.0 million. General and administration costs including land holding, camp, overhead labor and other costs are estimated to be about $11.0 million or 18% of direct costs. These items comprise the majority of the closure costs calculated for the Çöpler Mine.

SRK recommends that Alacer review the tailings design to see if it can be modified to provide drainage surface on the tailings storage facility surface after closure and prevent ponding of water where settlement of the surface will be occurring. Alternately, a FS should be undertaken to evaluate closure design without positive drainage from the tailings surface.

A trade-off study should be undertaken to evaluate whether the application of geotextile on the tailing surface might allow a thinner waste rock cover.

SRK also recommends that Alacer conduct studies for growth media material sourcing.

Alacer recognizes its commitment to properly close the Çöpler Mine in accordance with the obligations derived from the EIA process and the identified post-mining land use. Alacer also recognizes its obligation to ongoing monitoring and maintenance until the land can be successfully relinquished.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost (Capex) Summary

All heap leach costs are considered as either sustaining capital or are included in the operating costs. Sustaining capital for the heap leach pad is for phased expansion of the lined area for stacking oxide ore.

Amec Foster Wheeler prepared the capital cost estimate for the Sulfide Expansion Project. The update reflects the decision to adopt two horizontal autoclaves over the vertical autoclave arrangement, updated material quantities, updated equipment pricing and revised construction direct and indirect costs.

The estimate was based on the scope of work as outlined in the facilities description and work breakdown structure (WBS), and was defined by the following preliminary designs and design parameters:

- Process design criteria
- Process flow diagrams with mass balance
- Piping and instrument diagrams
- Mechanical and electrical equipment lists
- 3D plant model
- Site/plot plans
- Purchase agreements with vendors and contractors for 100% of the total equipment value
- Final earthwork quantities

The estimate is considered to have an accuracy of +10% / -5%. The total estimated initial capital cost to design, procure, construct and start-up the facilities described in this section is US$743.7 million as of April 2015, including owner's costs. Table 21-1, summarizes the estimated initial capital costs.

Table 21-1 Overall Capital Cost Summary

PROJECT AREA	USD $M
1000 - Process Plant	269.9
2000 - Process Plant Utilities & Services	74.1
3000 - Tailings Storage Facility	30.7
4000 - Support Infrastructure & Temp Facilities	101.0
5000 - Engineering, Procurement & Construction Management	93.6
6000 - Start-up & Commissioning	10.3
8000 - Owner's Costs	87.1
9000 - Provisions (incl. Growth Allowance & Contingency)	76.9
TOTAL	**743.7**

The estimate is expressed in Q4 2015 United States dollars.

Items not included in the initial capital spending estimate are as follows:

- Sunk costs: costs prior to April 1, 2015 (i.e. exploration drilling, sample preparation, metallurgical testwork, PFS, FS, EIA, basic engineering, etc.)
- Oxygen plant (included as an operating cost)
- Owner's corporate costs
- Allowance for special incentives (schedule, safety, etc.)
- Value added tax (VAT) and withholding tax
- Foreign currency exchange rate fluctuations
- Working capital (spares and first fills are included)
- Sustaining capital (included in cash flow model)
- Interest and financing cost
- Risk due to political upheaval, government policy changes, and labor disputes, permitting delays, weather delays or any other force majeure occurrences.

Where source information was provided in other currencies, these amounts have been converted at the following rates:

- 1USD = 2.75 TRY (Turkish lira)
- 1USD = 0.90 EUR (Euros)
- 1USD = 1.35 AUD (Australian dollar)

Mining operations for the mine are currently contracted to an outside party and this arrangement is expected to continue during the foreseeable future. Therefore, 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.

21.2 Basis of Estimate

21.2.1 General

The complete basis of estimate (BOE) for the capital cost estimate for the Sulfide Expansion Project was prepared per Amec Foster Wheeler guidelines and standards for a capital cost estimate.

This BOE describes the guidelines by which the estimate was prepared, the areas of responsibility, the scope of the estimate, the estimate methodologies, and the significant assumptions/clarifications and exclusions.

21.2.2 Estimate Type / Accuracy

The estimate was prepared per Amec Foster Wheeler' guidelines and standards for a capital cost estimate per Amec Foster Wheeler SOPs. This estimate provides a basis for evaluating the economic viability of the project and for approving the project for advancement, as well as providing a basis for advance commitments. The capital cost estimate identifies the capital costs associated with the agreed upon scope of work.

The resultant accuracy range (+10%/-5%) of the estimate was determined using a Monte Carlo risk analysis, estimator and project personnel judgment and industry standards. The estimate basis documents represent an average definition of process facility design completion of approximately 30-35% of engineering.

21.3 Direct Cost Elements

21.3.1 Tailings Storage Facility, Including Sustaining Costs

The TSF is designed and estimated by Golder. The estimate by Golder includes costs developed for that portion of the TSF design, including the TSF haul road and other access roads from a demarcation boundary north of the plant area near the planned new gatehouse to the TSF.

Initial capital and sustaining capital costs for construction of the TSF were estimated from the detailed level design drawings prepared by Golder for the Phase 1 TSF and updated feasibility level design drawings prepared for Phases 2 through 7. Tailings deposition is expected to start during the third quarter of 2018, assuming the start of TSF construction in the third quarter of 2016.

The costs for construction of the TSF are presented in Table 21-2 on an annual basis and by phase of development. Costs for the construction of the rockfill embankment make up approximately 40% of the total costs and are the largest component of the TSF and have been shown separately. TSF initial capital costs are from 2016 to the first half of 2018 inclusive and are allocated in the project capital costs.

Table 21-2 TSF Initial and Sustaining Capital Costs

Year	Ongoing Raise Construction	Annual TSF Rockfill Cost ($)	Annual TSF Raise Construction Cost ($)	Total Cost For Ancillary Facilities ($)	Total Annual Construction Cost ($)
2014	--	0	--	--	**0**
2015		0			**0**
2016	Haul and Construction Roads & TSF Starter (Phase 1)	3,727,259	5,830,916	16,605,458	**26,163,634**
2017	TSF Starter (Phase 1)	10,969,482	11,661,833		**22,631,315**
2018		0	5,830,916		**5,830,916**
2018	TSF Raise 1 (Phase 2)	13,031,952	2,664,712		**15,696,664**
2019		12,397,328	11,667,530		**24,064,858**
2020	TSF Raise 2 (Phase 3), Sabırlı Village Road Reroute	15,185,007	2,812,871		**17,997,878**
2021		0	15,716,364	2,296,793	**18,013,156**
2022	--	0	--	--	**0**
2023		0			**0**
2024	TSF Raise 3 (Phase 4)	11,977,895	6,334,884		**18,312,779**
2025		4,124,992	6,929,451		**11,054,444**
2026	--	0	--		**0**
2027	TSF Raise 4 (Phase 5)	27,269,563	8,704,558		**35,974,121**
2028		0	9,826,733		**9,826,733**
2029	--	16,372,755	--		**16,372,755**
2030	TSF Raise 5 (Phase 6)	16,372,755	10,803,901		**27,176,656**
2031		0	9,578,460		**9,578,460**
2032	--	0	--		**0**
2033		8,477,042			**8,477,042**
2034	TSF Raise 6 (Phase 7)	8,477,042	1,561,863		**10,038,906**
2035		0	14,450,415		**14,450,415**
2036	--	0	--		**0**
2037		0			**0**
2038		0			**0**
TOTAL		**148,383,075**	**124,375,406**	**18,902,251**	**291,660,732**

21.3.2 Mining

As current mining costs are associated with the existing oxide heap leach operations, there are no capital expenditures associated with the mine portion of the Sulfide Expansion Project.

Mining operating costs are discussed in the operating cost estimate in Section 21.8.

21.3.3 Power to Site

The estimate includes such electrical items as primary power distribution equipment, power distribution overhead cables and poles, motor control centers, interconnecting cables and raceways, and overload protection equipment.

Electrical equipment sizes and quantities have been determined by the discipline engineer and are based on power requirements derived from vendor data, preliminary electrical one-line sketches, and from the distribution routing shown on the preliminary electrical routing sketches.

21.4 Indirect Cost Elements

21.4.1 EPCM Cost

Typical professional services for the detailed design provided by Amec Foster Wheeler are as described in the Project Execution Plan included in Section 24.0 of this document. Engineering costs associated with the PFS and FS are considered sunk costs and have been excluded from this estimate.

21.4.2 Construction Management

Construction Management (CM) will be by a qualified third party designated by Owner.

CM costs include the following:

- Construction manager
- Construction site managers
- Construction personnel involved in leading and overseeing the construction process, including monitoring progress and ensuring the Alacer requirements are being met
- Translators
- Exempt operations representatives that function as part of the project team (PT) performing oversight of construction services
- Material control
- Overall site safety
- Overall site security
- Quality assurance/control
- Coordination of all onsite contractors
- Management of contractor equipment and material billings and payments
- Contract execution and administration
- Project controls including cost reporting and scheduling
- Pre-qualification of bidders
- Verification and certification costs
- Commissioning and startup (C&SU).

This account includes members of the Project C&SU Team through successful operation of the plant, along with associated expenses. The CM team will be responsible for a majority of the commissioning activities with some assistance from the Alacer project team. Start-up will be performed by the Alacer Operations team with some assistance from the CM team.

Pre-production costs are included in this account, including such pre-startup costs as pre-operations, pre-commissioning, C&SU. Related CM support services and facilities (office space, vehicles, administrative support, utilities, etc.) are included in the CM estimate.

21.4.3 Owner's Cost

The Owner's cost estimate was prepared by Alacer and is included in the project capital cost estimate. Owner's costs include all costs for maintaining a project management team throughout the engineering, procurement and construction term.

21.4.4 Temporary Buildings and Facilities

Temporary Facilities have been factored as an allowance.

21.4.5 Temporary Construction Utility Services

Temporary construction utilities have been factored as an allowance.

21.4.6 Construction Fuel

Construction fuel has been factored as an allowance on top of the detailed construction equipment list from construction that will be required for the project.

21.4.7 Spare Parts

Pre-commissioning spare parts has been estimated by process and mechanical engineering disciplines and the needs outlined by the various vendors and Alacer sparing philosophy. Operational Spare Parts have not been accounted for.

21.4.8 Initial Fills

Initial fills include grinding media, flocculent, water treatment chemicals, lubricants and other reagents. These quantity requirements have been identified by process and mechanical engineering disciplines and the needs outlined by the various vendors and Alacer first fills requirements that are not considered operating expenses.

21.4.9 Freight

The scope includes ocean freight, special freight and ex-country transportation cost. Costs include freight and handling for all equipment, bulk materials and indirect materials that are purchased outside of Turkey and shipped to Samsun then trucked to site.

Ocean freight rates were assumed to be applicable to the portion of the mechanical equipment not sourced in Turkey based on historical factors related to ocean freight in the region. The remoteness of the site necessitated a transportation study; the results of that study have been incorporated into the estimate to handle inland logistics/obstacles and transportation. Logistic input by the transportation consultant indicated that during transport of the autoclave vessels two heavy load trailers will be required for corners that are too sharp for the primary trailer. Allowances for dock fees, equipment staging areas, dock rental, warehousing/storage rental, demurrage, as well as the fees for a Duties broker are accounted for in this section.

It is assumed that 50% of the material (70% of equipment) will be shipped from overseas; the remainder of mechanical equipment will be provided from within Turkey as well as most of the bulk materials.

21.4.10 Vendor's Representatives – Construction/Commissioning

Vendor representatives and subcontract support costs were factored based on Amec Foster Wheeler historical and in-house data, or based on actual written vendor quotes where available. A separate request for these costs was issued on the material requisitions sent out for budgetary quotes. Vendor representatives will be present for receipt of major equipment. An itemized list of vendor representatives and expected durations has been developed by the project team to produce a cost for assistance with construction and commissioning. These costs include an assumed vendor daily rate and travel cost.

21.4.11 Commissioning and Start-up

Commissioning and start-up costs in the estimate were factored based on Amec Foster Wheeler historical and in-house data as a percentage of direct field man-hours.

21.5 Escalation

Estimated costs in the body of the estimate represent 4Q 2015 constant U.S. dollars.

Escalation from 4Q 2015 to project midpoint of expenditure has been excluded from the estimate. Consequently, the estimated costs represent 4Q 2015 constant US dollars.

21.6 Provisions

The estimate contingency was based on the following definition:

Contingency is a specific provision added to the base estimate to cover items that have historically been required but cannot be specifically identified in advance. It is expected to be spent in accomplishing the project scope as defined; it is not intended to cover scope changes. It is based on actual project experience and intended to cover:

- As yet undefined items needed to complete the current project scope.
- Variability in market conditions, material prices, wage rates, labor units, productivity assessments, allowances and project execution parameters.
- Estimating errors and omissions.

Contingency does not cover the cost of additional work or scope changes after the scope of the project has been defined for the estimate. It is also not intended to cover acts of God, unusual economic situations, potential currency fluctuations, strikes, and work stoppages for items like alternative location of tailings, community relocation issues, acute material shortages or catastrophes.

Note: Potential currency fluctuation impacts are not addressed in the Project contingency.

Contingency should not be confused with design development allowance or growth allowance, which is an allowance applied to the defined preliminary scope items, such as equipment, and materials that have been physically quantified to certain level of accuracy. In general contingency is utilized for those items that have not been quantified, and for design development that is beyond what is covered by the design development allowances.

21.6.1 Application of Contingency

As indicated in the definition above, contingency is a monetary allowance for items/considerations that were not defined at the time of estimate preparation, and that experience has demonstrated must be added to the base estimate to produce the total final cost. The level of contingency must be aligned with the desired probability of overrun / under run.

21.6.2 Contingency Calculation

Contingency for this Project estimate was calculated using a Monte Carlo risk analysis simulation program. The program quantifies the range of possible project cost outcomes and their relative likelihood of occurrence. This contingency analysis was developed to determine a recommended contingency and to provide the accuracy range of the estimate between the probabilities of P_{10} and P_{90}.

Contingency was then calculated by the program over the spectrum of various levels of probability of under run. A P_{80} was used to select the amount of recommended contingency.

21.7 Capital Cost Qualifications and Exclusions

See the Amec Foster Wheeler basis of estimate for the estimate assumptions and clarifications, as well as exclusions.

21.8 Sustaining Capital Costs (LOM)

The project sustaining capital costs for LOM includes all equipment that will be required by the Sulfide Expansion Project. This includes rebuilds of front end loaders, cranes and forklifts, as well as rental of operations pickup trucks. The sustaining capital also includes the purchase of new Bobcats for sulfide plant operations. Autoclave refractory (brick lining) replacement is included in sustaining capital.

Capital improvements of $14.2 million (included in sustaining capital) are included to add CCD clarifier and gold circuit laundering screen capacity, which will bring the maximum plant throughput to 2.2 Mtpa in 2021. The sustaining capital cost estimate also includes a $3.31 million allowance for modifications after startup of the sulfide plant to rectify minor design or equipment issues during the initial commissioning and production ramp up period.

21.9 Operating Costs (OPEX)

LOM operating costs were calculated for mining, processing and support costs. The summary of LOM unit operating costs per ore tonne are shown in Table 21-3. Due to rounding, some totals listed in the tables below may differ slightly from the sum of the numbers above. Where possible, operating costs are based on the mine 2016 budget. Budgeted costs are calculated each year by reviewing the previous year actual costs and making adjustments for consumption rates, supply pricing exchange rates and other parameters. In areas where budget or historical information is not available, costs are built up using first principles.

Table 21-3 Summary of Life-of-Mine Operating Costs

Activity	Unit	Life of Mine Average Unit Cost
Mining	per tonne mined	1.50
Rehandle	per tonne ore rehandled	1.12
Heap Leach Processing	per tonne HL ore processed	8.09
POX Processing	per tonne POX ore processed	31.80
Site Support and Offsite	per tonne ore processed	5.83

The LOM all in cash costs per gold ounce are represented in Table 21-4.

Table 21-4 Summary of All-In Cash Costs Net of By-Products

Costs per Ounce (Cash Basis)	Units	Amount
Cash Operating Costs (C1)	**US$/oz**	**563**
By-Product Credits (Ag, Cu)	US$/oz	(9)
Cash Operating Costs net of By Products (C1)	**US$/oz**	**554**
Royalties	US$/oz	17
Total Cash Costs (C2)	**US$/oz**	**570**
Sustaining Capital	US$/oz	74
All In Sustaining Costs (AISC)	**US$/oz**	**645**
Sulfide Preproduction Capital	US$/oz	183
Reclamation	US$/oz	17
All In Costs (AIC)	**US$/oz**	**844**

Reported as Unit Cost per Ounce. Negative costs indicated in this table reflect the positive revenue from the silver and copper by-product sales that are deducted from the operating cash costs. Totals may not sum due to rounding.

Mining costs are based on current contract mining costs. Heap Leach processing and support costs are based on 2016 budgeted operating costs. Support costs include community relations, security, safety, health and environmental and general and administrative activities. Support costs in Table 21-3 include gold doré refining charges.

21.9.1 Mine Operating Cost Estimate

A contractor is used to conduct mining operations and this is expected to continue over the project life. The cost for the mining contractor is US$1.38 per mined tonne and is based on 2016 budgeted mining costs. Mining costs include all mining operations, mining equipment, supplies, blasting materials, and manpower required to operate the mine. Additionally, mine management, technical staff, and ore control will be provided by Alacer. These functions are estimated at US$0.12 per mined tonne based on historical 2015 mine operating costs. The total mine operating cost is US$1.50 per mined tonne.

21.9.2 Heap Leaching Costs

Heap leach operating costs are based on 2016 budgeted operating costs. Table 21-5 shows the average heap leaching operating costs for the LOM.

Table 21-5 Alacer Heap Leach Processing Costs

Activity	$/t
Crushing Cost	$1.44
Conveying and Stacking	$1.28
Processing	$5.14
SART Plant	$0.23
Total Heap Leach Processing Costs	$8.09

21.9.3 Sulfide Processing Costs

Sulfide processing costs were developed from first principles. Processing labor is based on an estimate of personnel numbers developed by Alacer and 2016 budgeted labor rates. Sulfide processing personnel is summarized in Table 21-6.

Table 21-6 Sulfide Processing Labor Personnel

Salaried Positions	# of Personnel	Operating Labor	# of Personnel
Process Manager	1	Crusher Operator	4
Process Superintendent	1	Grinding Operator	4
Process General Foreman	1	Control Room Operators	12
Process Supervisor	7	Autoclave Operators	8
Chief Metallurgist	1	Leach/CIP Operators	4
Senior Metallurgist - Ore Blending Engineer	1	CCD/Iron-Arsenic/Copper Precip Operators	4
Senior Metallurgist	2	ADR	4
Junior Metallurgists	3	Refiners	2
Shift Supervisor	4	Neutralization and Tailings Operators	4
Chief DCS Programmer	1	Shift Helper	20
Junior DCS Programmer	1	General Laborers/Reagent Mixing/Helpers	5
Report Administrator	1	Tea Room Attendants	4
Maintenance Superintendent	1	**Total Estimated Operating Labor**	**75**
Maintenance General Foreman	1		
Mechanic Foreman	4	*Maintenance Labor*	
Electrical Foreman	1	Mechanic - Day Shift	12
Instrument Foreman	1	Mechanic- day shift helper	5
Chief Maintenance Planner	1	Mechanic - Rotating Shifts	8
Planner	2	Mechanic Helper - Rotating Shifts	16
Reliability Engineer	1	Electrician	12
Change Management Supervisor	1	Electrician Helper	8
Data Entry Clerk	3	Instrumentation Technicians	12
Salaried Labor Total	**40**	**Total Maintenance Labor**	**73**
Technicians and Assayers			
Metallurgical Technician	6		
Assayers - Main Site Lab	6	**Total Salaried Labor**	**40**
Assayers - POX Lab	6	**Total Non-Exempt Labor**	**172**
Samplers	6	**Total Sulfide Plant Labor**	**212**
Total Technicians and Assayers	**24**		

Labor for refining is in place at the mine in sufficient numbers to service the additional gold production of the sulfide plant.

Consumption rates for major reagents, such as cyanide, flocculent, and lime, are based on metallurgical test results. Consumption rates for minor reagents and consumables are based on industry norms. Fuel consumption for carbon elution/regeneration and pressure oxidation steam generation are based on vendor advised hourly consumption rates and estimates of annual operating hours. Unit pricing for fuel, reagents, consumables and power were confirmed with nominated vendors by Anagold's procurement department in the third quarter 2015. Estimated LOM average consumption rates and cost for reagents, fuel and power are summarized in Table 21-7.

Table 21-7 Sulfide Processing Reagents, Fuel and Power

Item	Average Consumption (units)	Average Consumption	Delivered Cost $/Unit	LOM Avg. Cost $/t sulfide
Flocculent	kg/t	0.235	3.11	0.79
Antiscalant	kg/t	0.02	2.78	0.06
SAG Mill Grinding Media	kg/t	0.38	1.02	0.39
Ball Mill Grinding Media	kg/t	0.51	0.97	0.49
Limestone Mill Grinding Media	kg/t	0.018	0.97	0.02
Burnt/Pebble Lime	kg/t	36.20	0.07	2.60
Glycerine	kg/t	0.04	0.83	0.03
Crushed Limestone	kg/t	16.10	0.012	0.05
Sodium Cyanide	kg/t	1.25	2.01	2.51
Activated Carbon	kg/t	0.03	2.69	0.08
Sodium Metabisulfate	kg/t	2.00	0.39	0.78
Sulfuric Acid	kg/t	10.28	0.13	1.36
Nitric Acid	kg/t	0.26	0.23	0.06
Sodium Hydroxide	kg/t	0.075	0.28	0.02
Total Reagents and Grinding Media	**kg/t**			**9.23**
Power (non-O_2 plant)	kWh/t	79.8	0.06	4.52
Fuel	l/t	0.9	1.00	0.92

Estimated costs for oxygen supply to the pressure oxidation circuit are based on currently negotiated terms for a Gas Supply Agreement between Anagold and Air Liquide. These costs include a fixed monthly base facility charge plus a per unit of oxygen consumption charge. Estimated oxygen consumption rates are based on the tonnage and sulfur grade of ore treated and vary through the life of the operation, averaging 1750 Nm^3 of oxygen per tonne of sulfide sulfur processed.

Maintenance materials for the sulfide operations are factored as 4% of the estimated capital cost for equipment, bulk materials and delivery of same for the sulfide processing plant. Large mobile equipment used for run of mine ore and limestone for the sulfide plant will be supplied by the mining contractor and costs are based on current contract rates.

Commissioning costs include a $6.75 million allowance for temporary support personnel and supplies during startup of the sulfide plant and the first year of operation.

The LOM average process operating costs for the Sulfide Expansion Project are provided in

Table 21-8.

Costs are shown on a $/tonne of sulfide feed processed, $/oz of gold recovered, and the average total operating cost in million $/year.

Table 21-8 Life-of-Mine Total Sulfide Processing Costs by Cost Component

Item	$/t Sulfide	$/oz Sulfide	Annual Cost, $M
POX Processing - Labour	4.50	53	9.5
POX Processing - O_2 Plant Fixed	4.05	48	8.5
POX Processing - O_2 Plant Variable	3.41	40	7.2
POX Processing - Reagents	9.23	108	19.4
POX Processing - Fuel Oil	0.92	11	1.9
POX Processing - Electrical	4.52	53	9.5
POX Processing - Maintenance Materials	3.89	46	8.2
POX Processing - Large Mobile Equipment	0.22	3	0.5
POX Processing - Laboratory	0.89	10	1.9
POX Processing - Commissioning	0.17	2	0.4
Total Sulfide Processing Costs	**31.80**	**373**	**66.9**

21.9.4 Tailing Management Facility

Golder reviewed the projected labor and consumables prepared by Amec Foster Wheeler for the sulfide process plant and believe that the operating cost estimate developed is sufficient to provide necessary resources for operations and maintenance within the TSF. Operations and maintenance for the TSF is anticipated to consist of monitoring and relocation of tailings pipelines, control and maintenance of tailings spigots, monitoring of underdrain and overdrain flows and maintenance of the pumps, and for general maintenance and inspection of the TSF.

22.0 ECONOMIC ANALYSIS

The economic analysis is based on detailed engineering progress and updated pricing for consumables, labor and other economic inputs. Capital, operating and mine reclamation costs have been updated from previous reports. In as much as possible, all costs are expressed in fourth quarter 2015 US dollars.

22.1 Introduction

A financial analysis for the Sulfide Expansion Project was carried out using an incremental or differential cash flow approach. Cash flow models were developed for the Sulfide Expansion Project with the oxide heap leach as well as for the oxide heap leach alone without the Sulfide Expansion Project. A differential cash flow was calculated between the two sets of cash flows to determine the financial benefit of the Sulfide Expansion Project. The IRR and NPV using a discount rate of 5% NPV were calculated using this differential cash flow.

The results of the economic analysis summarized below represent forward–looking information as defined under Canadian securities law. Actual results may differ materially from those expressed or implied by forward-looking information. The reader should refer to Section 2.2 for more information regarding forward-looking statements, including material assumptions (in addition to those discussed in this section and elsewhere in the Report) and risks, uncertainties and other factors that could cause actual results to differ material from those expressed or implied in this section (and elsewhere in the Report).

22.2 Methods, Assumptions, and Basis

The financial analysis was performed using the following basis and assumptions:

- The assumptions as to Mineral Reserves in Section 15, the mine plan in Section 16, the recovery plan in Section 17, the infrastructure as described in Section 18, and the social, permitting and environmental considerations as described in Section 20.
- The base case gold, silver and copper prices are USD $1,250/oz., $18.25/oz. and $2.75/lb. respectively.
- Cash flow analysis starts on January 1, 2016 and ends on December 31, 2046 and includes closing costs.
- Capital costs are derived from Section 21.1; remaining initial capital as of Jan 1, 2016 was estimated to be $721.1 million.
- Operating costs are derived from Section 21.9
- The cash flows take into account depreciation, taxes, working capital, and tax credits. The economic analysis is an after tax analysis.
- Commercial production will begin in third quarter 2018.
- The US dollar to Turkish lira exchange rate used is 3.00.
- The US dollar to Euro exchange rate used is 0.90.
- The US dollar to Australian dollar exchange rate used is 1.35.

- All cost and sales estimates are in constant Q4 2015 US dollars with no escalation factors taken into account.
- Royalties are as described in Section 22.3
- Metal recoveries are derived from Section 16.1
- Sustaining capital is outlined in Section 21.3.1 and 21.8
- All gold and silver is sold in the same year of production.
- All Project related payments and disbursements incurred prior to the effective date of this Report are considered as sunk costs.
- Analysis is unleveraged on a 100% equity basis (no Project financing or debt)
- Analysis is on a stand-alone project basis
- Taxes follow the Turkish tax regime
- Annual cash flows are discounted on an end of year basis
- All dollars are in US dollar, unless specifically noted.

The general assumptions used for this financial model are summarized in Table 22-1.

Table 22-1 Financial Model Inputs

Metal Prices		
Gold	$/oz	$1,250
Silver	$/oz	$18.25
Copper	$/lb	$2.75
Metal Sales Costs		
Dore Refining & Shipping	$/oz Au	$8.54
Gold Royalties	% of NSR	2.00%

Cash flow models were developed for the Sulfide Expansion Project with the oxide heap leach as well as for the oxide heap leach alone without the Sulfide Expansion Project. A differential was calculated between the two sets of cash flows to determine the financial benefit of the of the Sulfide Expansion Project.

Table 22-2 summarizes the key values for both the sulfide plant operating together and oxide heap leach operating together and the oxide heap leach only operation.

Table 22-2 Respective Life of Mine Key Values used for Differential Cash Flow Calculation

Description	Units	Heap Leach Only	Heap Leach and Sulfide
Waste Tonnes Mined	Mt (LOM)	61.6	224.8
Heap Leach Tonnes Mined	Mt (LOM)	12	18
Sulfide Feed Stock Mined	Mt (LOM)	-	34.9
Total Tonnes Mined	Mt (LOM)	73.6	277.6
Total Heap Leach Rehandle	Mt (LOM)	7.2	10.8
Total POX Rehandle	Mt (LOM)	-	35.3
Total Mine Rehandle	Mt (LOM)	7.2	46.1
Heap Leach Feed Processed	Mt (LOM)	12.0	18.0
Heap Leach Feed Gold Grade	g/t (LOM)	1.16	1.13
Heap Leach Gold Absorbed	koz (LOM)	383	540
Heap Leach Gold Recovery	% (LOM)	75.9%	76.0%
Heap Leach Feed Silver Grade	g/t (LOM)	3.16	3.5
Heap Leach Silver Absorbed	koz (LOM)	389	632
Heap Leach Silver Recovery	% (LOM)	30.1%	29.9%
Heap Leach Feed Copper Grade	% (LOM)	0.14%	0.13%
Heap Leach Payable Copper Product	Mlbs (LOM)	1.5	2.1
Heap Leach Copper Recovery	% (LOM)	6.1%	6.2%
Sulfide Feed Stock Processed	Mt (LOM)	-	40
Sulfide Feed Gold Grade	g/t (LOM)	-	2.76
Sulfide Gold Recovered to Dore	koz (LOM)	-	3,408
Sulfide Gold Recovery	% (LOM)	-	96.10%
Sulfide Feed Silver Grade	g/t (LOM)	-	6.3
Sulfide Silver Recovered to Dore	koz (LOM)	-	1,093
Sulfide Silver Recovery	% (LOM)	-	13.50%
Sulfide Feed Sulfide Sulfur	% (LOM)	-	4.30%
C1 Cash Operating Costs	M$ (LOM)	$301	$2,223
Royalties	M$ (LOM)	$8	$67
C2 Total Cash Costs	M$ (LOM)	$308	$2,290
Sustaining Capital Expenditures	M$ (LOM)	$17	$294
AISC All-In Sustaining Costs	M$ (LOM)	$325	$2,583
Total Capital Expenditures	M$ (LOM)	$17	$1,015
Reclamation Expenditures	M$ (LOM)	$44	$67
Total Revenue	**M$ (LOM)**	**$490**	**$4,973**
Operating Profit	**M$ (LOM)**	**$182**	**$2,683**

22.3 Royalties

The royalty rate for precious metals under Turkish Mining Law is variable and tied to metal prices. The applicable rate is then subject to a 50% discount as Çöpler ores are processed onsite. Calculation of royalties payable is based on sales less certain qualifying operating costs. The royalty rate for gold based on the metal price assumptions used in the financial model is 2%.

22.4 Salvage Value

Salvage values are not credited in the financial model.

22.5 Taxation

The corporate tax rate in Turkey is 20%.

For tax purposes, 20% accelerated depreciation is applicable for both oxide and sulfide capital.

Anagold has received an incentive certificate for the Sulfide Project. Investment incentive certificates are available for investments made in Turkey to promote economic development. An investment incentive certificate generates investment credits that can be used to offset corporate income taxes generated by the Project. The amount of investment credits generated from the investment incentive certificate is based on eligible capital expenditures relating to the Project. The investment credits generated by the investment incentive certificate can reduce the corporate tax rate for the Project to a minimum floor of 2% in a given tax period. The unused portion of incentive tax credits can be carried forward to future tax periods indefinitely until exhaustion.

VAT for this Project are levied at 18%. The Project complies with Turkey's mineral exemptions for mining projects and mining equipment will not be subject to VAT. Some effort and cost will go into obtaining reimbursements (cost for employing government certified tax accountants). Maintaining the investment certificate which occurs after the project is complete will get greater attention. Investment incentives are available for imported process equipment and materials which are listed as exempt items from VAT as well as other duties. For specifically designated provinces of Turkey (e.g. Erzincan), a reduced corporate tax practice covers earnings derived from investment under an incentive certificate issued by the Under Secretary of the Treasury. There are no restrictions on equipment sourcing location with regards to the incentive. Location does determine how the incentive is credited. If sourced in Turkey, VAT is paid then recovered through application to the government. If sourced, outside of Turkey, no VAT is paid.

No import duties are included in the capital cost estimate for mining related imported equipment because they are exempted in the incentive certificate. The nature of equipment and materials determines whether these taxes are applicable. Customs and duties fees are not included in the price of equipment in this estimate for the same reason. An allowance has been included, however, for a customs broker.

22.6 Financial Analysis Summary

A discount rate of 5% was applied to the differential cash flow to derive the Project's NPV. This is summarized in

Table 22-3.

IRR and NPV calculations are after taxes, royalties, and depreciation.

Table 22-3 Financial NPV, IRR, and Payback Period

Description	Unit	Amount
METAL PRICES		
Gold Price LOM	US$/oz	1,250
Silver Price LOM	US$/oz	18.25
Copper Price LOM	US$/lb	2.75
PROJECT CASH FLOWS		
Sulfide and Oxide Projects	US$M	1,577
Oxide Project	US$M	94
Project Differential Cash Flow	US$M	1,483
PROJECT FINANCIALS		
NPV at 5% of Differential Cash Flows	US$M	728
IRR of Differential Cash Flows	%	19.2
Payback on Sulfide Project Cash Flow (from Start of Sulfide Production)	years	3.0

The project payback period based on the cash flow for the combined sulfide processing and heap leach operation is 3.0 years following the startup of the sulfide processing plant.

Table 22-4 shows the cash flow for the Sulfide Expansion Project with oxide heap leach, oxide heap leach only and the differential cash flows between the two.

Table 22-4 Sulfide Project with Oxide Heap Leach Cash Flow

Year	Gold Price ($/oz)	Copler Sulfide Project and Oxide Heap Leach ($M)	Oxide Heap Leach Only ($M)	Cash Flow Differential ($M)
2016	1,250	(184)	44	(228)
2017	1,250	(328)	14	(342)
2018	1,250	(20)	81	(101)
2019	1,250	229	(5)	234
2020	1,250	213	(4)	217
2021	1,250	185	(10)	195
2022	1,250	210	(8)	218
2023	1,250	225	(4)	229
2024	1,250	89	(5)	94
2025	1,250	93	(1)	93
2026	1,250	107	(1)	108
2027	1,250	59	(1)	60
2028	1,250	89	(1)	90
2029	1,250	77	(0)	77
2030	1,250	64	(0)	64
2031	1,250	60	(0)	60
2032	1,250	87	(0)	88
2033	1,250	69	(0)	70
2034	1,250	83	(0)	83
2035	1,250	80	(0)	80
2036	1,250	100	(0)	101
2037	1,250	15	(0)	16
2038	1,250	(20)	(0)	(20)
2039	1,250	1	(0)	1
2040	1,250	(3)	(0)	(2)
2041	1,250	(1)	(0)	(1)
2042	1,250	(1)	(0)	(1)
2043	1,250	(1)	(0)	(1)
2044	1,250	(1)	(0)	(1)
2045	1,250	(0)	(0)	(0)
2046	1,250	(0)	(0)	(0)

LOM cumulative cash flows are shown in Figure 22-1. The graph includes cumulative cash flow projections for the Sulfide Expansion Project with the oxide heap leach and for the oxide heap leach only.

Figure 22-1 Cumulative Cash Flows for Sulfide Project with Oxide Heap Leach and for the Oxide Heap Leach Only

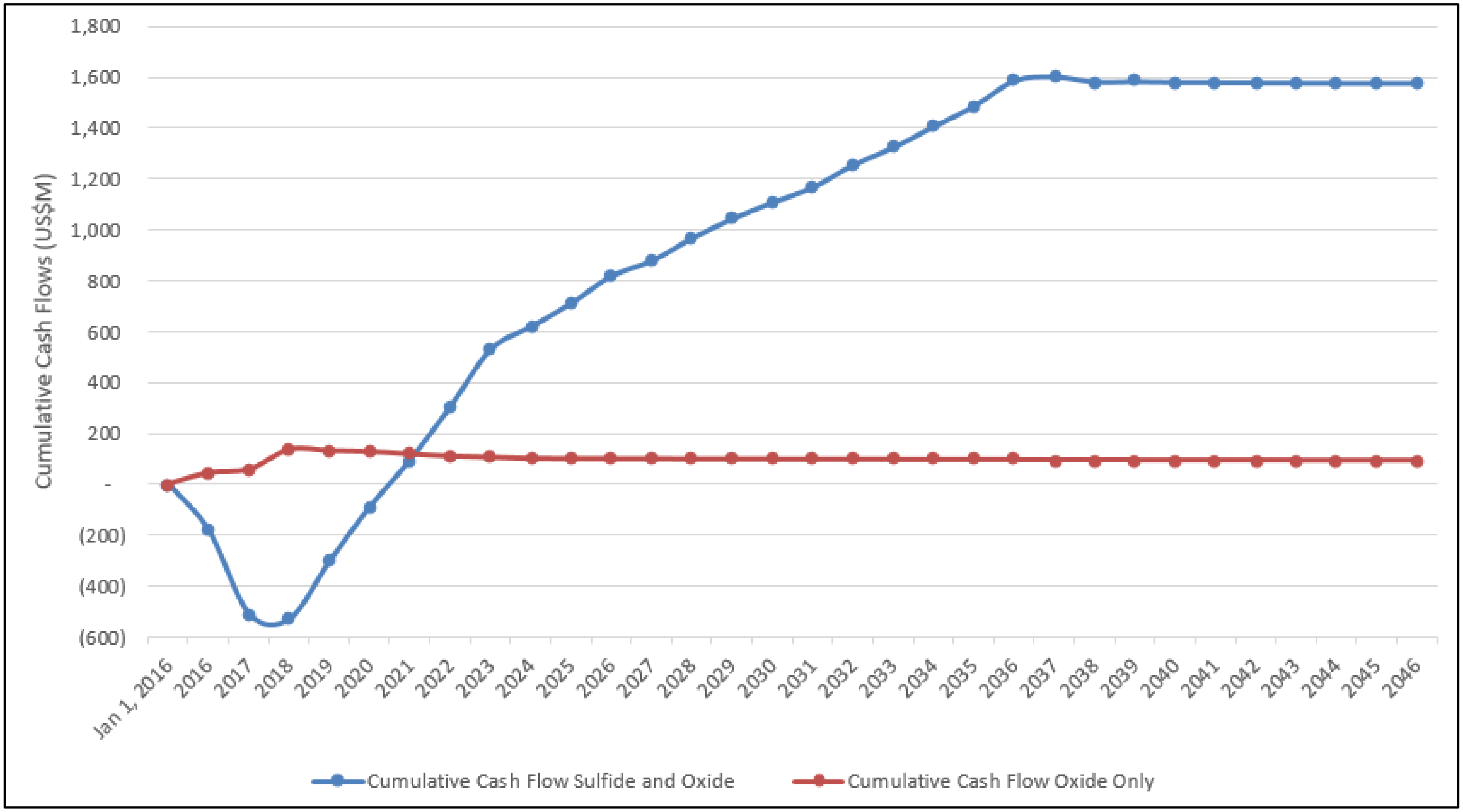


Figure prepared by Alacer, 2016.

22.7 Sensitivity Analysis

Figure 22-2 and Figure 22-3 contain incremental NPV and IRR sensitivities around gold price, operating costs, initial capital costs, sulfide gold grade and Turkish lira (TL) exchange rate.

The NPV and IRR is most sensitive to the gold grade and gold price followed by initial capital costs and operating costs.

Figure 22-2 Incremental NPV Sensitivity

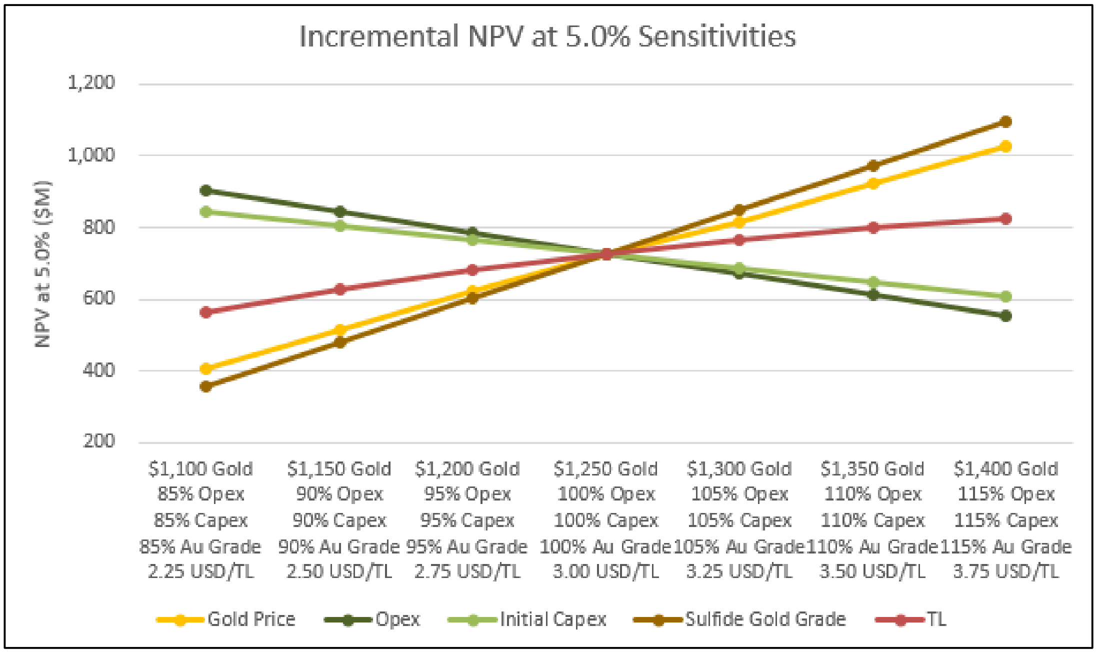


Figure prepared by Alacer, 2016.

Figure 22-3 Incremental IRR Sensitivity

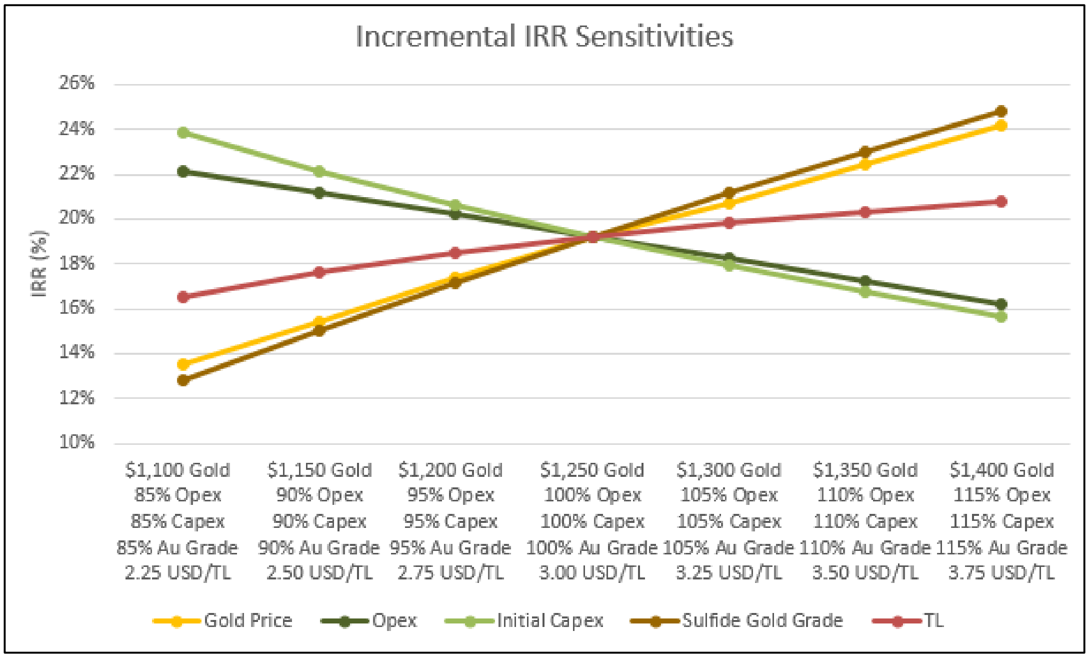


Figure prepared by Alacer, 2016.

23.0 ADJACENT PROPERTIES

There are no adjacent properties outside of Anagold interest that are relevant to the development of the Çöpler Expansion Project or the ongoing activities at the Çöpler Mine.

24.0 OTHER RELEVENT DATA AND INFORMATION

24.1 Project Execution and Schedule

24.1.1 Introduction

The Project Execution Plan (PEP) has been updated during the detailed engineering phase of the Sulfide Expansion Project.

The PEP is the principal planning and management document developed and maintained by the individual disciplines. When assembled, the discipline sections will collectively form the comprehensive PEP.

Detailed discipline execution procedures are contained in individual sections of the comprehensive PEP. The PEP contains discipline plans for execution functions during the engineering and construction phases of the project.

Key components of the PEP are as follows:

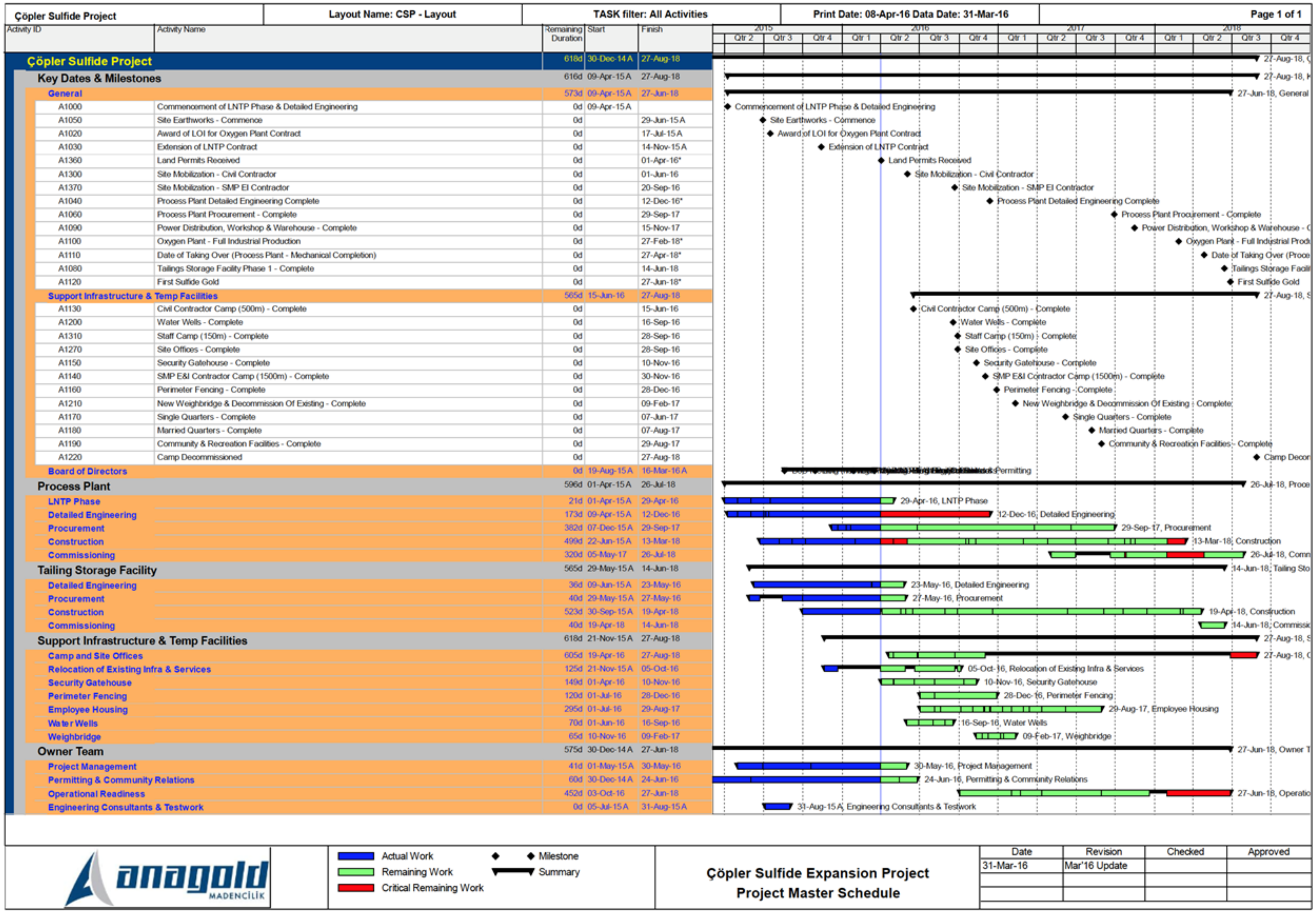
- The safety goal is "Zero Incidents." A joint safety manual will be prepared for the project incorporating the requirements of the Çöpler Mine and corporate safety programs. The construction management contractor will lead the safety effort for the project, with all parties responsible for working toward a zero incident rate. Contractors shall submit a site specific safety plan for approval by Alacer and the EPCM contractor.
- The Project WBS is the primary mechanism for management of costs, schedule, project data and scope of work.
- The EPCM contractors project controls team is responsible for accurate and timely reporting of the cost and schedule status of the project and management of change to scopes of work, schedule and cost.
- Quality assurance is an ongoing process during all stages of the project, from engineering through purchase and delivery of equipment, supplies and materials, contract administration, construction and erection, start-up and commissioning. QA/QC procedures are established for each phase of the project and contractors and vendors shall be required to adhere to project standards.
- Reviews and audits are to be conducted during the various stages of the project. A calendar of reviews will be developed; these will include but not be limited to the following:
    - Design reviews
    - Estimate reviews
    - Schedule reviews
    - Constructability reviews
    - Risk reviews
    - Health, safety and environment (HSE) reviews
    - Quality audit reviews
    - Process safety management, hazard and operability analysis (HAZOP) and Hazard evaluation and risk assessment (HERA) reviews

- Contract performance reviews
- Compliance audits

24.1.2 Schedule

A Project schedule has been maintained during the detailed engineering phase and updated as start dates and activity durations have been updated The Sulfide Expansion Project schedule as of March 31, 2016 is included in Figure 24-1.

Figure 24-1 Preliminary Milestone Schedule



25.0 INTERPRETATION AND CONCLUSIONS

Interpretation and conclusions for the Project were identified by the various report contributors and are included below. The QPs have also made specific interpretative and conclusion comments in Sections 14, 15 and Section 20 of the Report.

25.1 Mineral Resources

The Çöpler open-pit Mineral Resource estimation method was designed to address the variable nature of the epithermal structural and disseminated styles of gold mineralization while honoring the bi-modal distribution of the sulfur mineralization that is critical for mine planning. The modeling method was designed so that a) the Mineral Resources could be updated with additional drilling, and b) changes in cut-off grades could be recalibrated using up-to-date production data.

Since no obvious correlations were observed between gold and sulfur, they were domained and estimated separately. Gold showed little correlation with lithology, and was domained by mining areas (Manganese, Main, Marble and West) to reflect the different trends of the mineralization that commonly follow structures and lithological contacts. Due to the strong correlation between sulfur and lithology, sulfur was first domained by lithology. However, since each lithology may contain both < 2% S and, ≥ 2% S material, each lithology was additionally separated into < 2% S and ≥ 2% S sub-domains.

PACK was selected as the best method to estimate the gold mineralization. Probabilistic envelopes were first generated to define the limits of the economic mineralization, and then used in the Mineral Resource estimation to prevent the economic assays from being smeared into non-economic zones, and conversely, to restrict waste assays from diluting the economic mineralization. Two gold PACK models were constructed. The first low-grade gold model used a 0.3 Au g/t indicator estimate threshold to reflect the gold cut-off grade indicator estimation for the < 2% S material, and the second, high-grade, gold model used a 1.5 Au g/t threshold to reflect the approximate gold cut-off grade in ≥ 2% S material.

The oxide and sulfide gold models were reconciled to past production to calibrate the model to historic production data. Geology, EDA, composite / model grade comparisons, and other checks were performed to adjust the parameters used to construct the model. In the final Mineral Resource model, the low-grade gold model values were applied to the < 2% S material, and the high-grade gold model gold grades were applied to the ≥ 2% S material. The Mineral Resource Model has an implicit SMU of 5 x 10 m x 5 m.

The Mineral Resource model was validated by comparison with a NN model on a local and global basis. The selectivity implicit in the Mineral Resource model was compared to ore control models by pit and time period.

Mineral Resource categories of Indicated and Inferred classification were applied to each block based on drill hole density and data quality. No blocks in the model were classified as Measured Mineral Resources.

Additional sampling and assay analysis are needed to obtain stockpile grades for sulfur, sulfide sulfur, copper, silver, and manganese. This program was being carried out at the middle of 2016.

With additional drilling in selected areas, and completion of reconciliation studies involving blast hole assay database and on-site laboratory audits, there is potential to convert some of the Indicated Mineral Resources to Measured Mineral Resources.

It is Amec Foster Wheeler's opinion that the resource model has been constructed according to industry best practices and conforms to the requirements of the 2014 CIM Definition Standards. Factors that may affect the Mineral Resources are described in Section 14.22 "Risks and Opportunities".

25.2 Mining and Mineral Reserves

The Çöpler Mineral Reserve is amenable to mining through conventional open-pit mining techniques. The Mineral Reserve estimate has been calculated using industry best practices and conforms to the requirements of the NI 43-101.

Based on the parameters defined in Section 16.0, the Çöpler pit limits have been calculated based on the objective of maximizing NPV, while adhering to the limitations set forth by mining practicality and processing throughputs. Detailed pit designs have been generated within these limits which allow for the safe and efficient extraction of ore. The detailed pit designs allow for roadway access, geotechnical pit slope, and safety considerations to ensure the successful extraction of the ore.

These designs have been scheduled using industry accepted scheduling methods to a level of detail that ensures the mine plan is readily achievable. Two processing methods for ore at the Çöpler mine will be in use. Oxide ore will be processed through conventional heap leaching. Sulfide ore will be processed through a mill and POX processing system. The mine will deliver the oxide and sulfide ores to the appropriate crushing systems and stockpiles based on the selected processing method. The processing of sulfide ore at the Çöpler mine will continue through the year 2037.

The mining method selected for the extraction of the Çöpler Mineral Reserve is appropriate for the deposit style and location. Factors which may affect the Mineral Reserves estimate are described in Section 15.2 "Risks and Opportunities".

25.3 Waste Rock Storage Areas

Waste rock storage areas have been designed in a manner that ensures safety, stability, environmental compliance, and operational efficiency. The geochemical composition of the waste rock is well understood. While unlikely, there exists some risk for acid rock drainage from the Çöpler WRSA facilities. This risk is managed on a daily basis through the mine operations.

25.4 Metallurgy and Mineral Processing

25.4.1 Heap Leach Operation

For the existing heap leaching operation:

- The extraction and recovery of gold and silver from oxide and transition ore types to be mined and placed on the heap leach pad in the future could vary from the amounts stated in Section 13.2 due to variations in the gold and silver mineralogy with location in the deposit, and/or due to variation in particle size distribution, agglomeration quality and other operational factors. There is a risk that the extraction of gold and silver could decrease below the projected values as the oxidation boundary within the deposit is approached.

25.4.2 Sulfide Expansion Project

For the Sulfide Expansion Project ores:

- Metallurgical testing indicates that the gold in the sulfide ores is refractory to direct cyanidation and that an oxidation pre-treatment is required to liberate gold and make it amenable to recovery by cyanidation.
- POX will be used to break down the sulfides and the effectiveness of POX been confirmed by a number of test programs using continuous pilot autoclaves.
- Pressure oxidation feed must be ground fine to liberate sulfide minerals and to minimize wear on the autoclave feed pumps, the slurry heating system, the autoclave components and the depressurization system. A crushing and grinding circuit has been designed to produce whole ore slurry with a P_{80} of 100 µm.
- The mineralization is relatively soft and contains a high proportion of clay mineralization that is expected to present problems in handling and processing the material. To minimize the potential for handling problems the ore is primary crushed in a toothed roll sizer and directly fed to a SAG mill. The final grind to a P_{80} of 100 µm is performed in a ball mill.
- Çöpler sulfide mineralization also contains high carbonate levels. To better control the POX outcomes and reduce operating costs some of the carbonate is eliminated in an acidulation circuit ahead of POX. There is some uncertainty over the acidulation circuit operation remaining after the testing programs, and flexibility has been incorporated in the design to accommodate this. Acidulation remains a moderate risk area in the flowsheet.
- A number of feed properties are required to be within design limits to ensure the POX system can operate effectively. Two of the important control parameters are sulfur grade (which determines oxygen demand and heat generation in the autoclave) and carbonate grade (which determines acid demand and gas generation in the autoclave). Ore blending will be performed ahead of primary crushing to maintain feed properties within acceptable operating limits. The final ore blending configuration is being developed by the Owner.
- After POX the gold is available for cyanidation recovery but soluble copper, iron and arsenic are present in the liquor. Limestone slurry is added to precipitate the iron and arsenic and increase the pH to about 2.8. A separate circuit to precipitate and recover copper has been allowed for in the layout but does not form part of the PFS design. Some of the soluble copper will be consumed in the cyanide detoxification step, and the remainder will precipitate in the high pH cyanidation tailings. Little or no solution is expected to be reclaimed or recycled from the tailings, and therefore there is a low risk that copper will be returned to the processing circuit. It will not be possible to reclaim or recycle solution to the process from the tailings unless the copper is removed.
- CCD (two thickeners in series) is used to separate the remaining soluble metals (mainly copper) from the solids, which carry the gold.

- The solids are then leached with cyanide in a CIL circuit and the loaded activated carbon is sent to gold recovery and carbon regeneration.
- The CIL tailings slurry is sent to cyanide destruction, tailings thickening and then to the TSF.
- Gold recovery from the oxidized solids has been consistently very high in testing, typically >95%. However, there is some risk that gold extractions and recoveries could be lower than expected due to ore type variation and/or variation in operating conditions and operational upsets.

25.5 Infrastructure

The infrastructure design developed during the FS is adequate to support the Project facility.

Proposed facility foundation systems will be constructed to bear on either native conglomerate or limestone bedrock materials, or engineered structural fill. The proposed Sulfide Plant and related crushing and grinding facilities are anticipated to be constructed primarily on shallow foundation systems. Deep foundation alternatives may be considered on a case by case basis with feasibility of a deep foundation system depending on the subsurface conditions beneath the proposed structure(s).

Existing site infrastructure supports the heap leach facility, and some of this will be used, modified or unmodified, to support the new sulfide operations. A large amount of new infrastructure will be added in the sulfide expansion.

Major existing facilities that will support the sulfide expansion include:

- Site security gate and guard station
- Site administration building
- Warehouse
- New 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, & offices
- Site potable water treatment and distribution system
- Two sanitary waste water collection and treatment systems

Major new infrastructure to be constructed as part of the Sulfide Expansion Project includes:

- Maintenance building
- Warehouse
- Primary crushing control room

- Grinding building
- POX building and POX utilities building
- Carbon elution building
- Office space

Additional infrastructure includes:

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
Raw water pump building
Limeslaking (MOL) building
Fe/As air compressor building
Emergency diesel generators building
TSF reclaimeElectrical building
TSF drainage tank electrical building
TSF OD-UD pond electrical building
Elution building
CIP CCD ablutions block
Pump shelters with monorails
Carbon elution building - electrical room
Raw water bores P/P house & electrical building
Gatehouse
Fire water pump house
Community relations center
Raw water wells

25.6 Tailings Storage Facility

The TSF as planned for the Sulfide Expansion Project is feasible and can be constructed in accordance with the current mine plan and project schedule. The TSF provides for 45.9 Mt of tailings capacity in a fully lined tailings impoundment over an approximate 20-year mine life assuming an average ore feed to the mill of 1.9 to 2.2 Mtpa and average tailings delivery of 6,293 tpd at a slurry density of 28% by weight.

Golder developed the following conclusions for the TSF:

- Some risk exists relative to potentially varying geologic and geotechnical conditions within the TSF foundation. Specifically, understanding the extent, orientation and strength of the serpentinite and other bedrock and the extended TSF footprint must be determined to confirm the feasibility of and/or optimize the design of latter TSF raises as currently planned. Embankment stability and deformation evaluations should be updated to reflect planned ultimate embankment height and understanding of geologic conditions, following excavation and evaluation of actual bedrock conditions during initial Phase 1 construction. In addition, the deposition model should be updated using the updated design and with the new geotechnical data from tests completed on new tailings samples during the pilot program in the third quarter 2015.
- Some risk also exists with respect to schedule and ability of local contractors to coordinate construction of the embankment with installation of the liner system.

25.7 Marketing

Sale of the gold and silver doré will continue to be through existing contracts. Copper from the SART plant is sold into local Turkish markets and is expected to continue through the life of the heap leach operations.

25.8 Environmental/Permitting

The planned developments under the Sulfide Expansion Project are subject to EIA process in accordance with paragraph (ç) of Article 7 as well as Annex I, Article 29, paragraphs (a) and (c) of the 2013 EIA Regulation. The EIA permitting process had been started on April 7, 2014 and ended by receiving the “EIA Positive Statement” on December 24, 2014. The Stakeholder Engagement Plan (SEP) and the Social Impact Assessment (SIA) for the Expansion Project was prepared and reported in May 2015 and provided in the annex of ESIA report (September 2015). The following remarks can be made on the risks and opportunities in general where detailed information could be found in ESIA, SEP and SIA reports.

- The Çöpler Mine production from oxide ores was permitted in 2008 and is currently in operation. In terms of permitting, the existing mine provides an opportunity to demonstrate to the regulators the quality of environmental performance of the mine operations.
- The current Çöpler Mine is being operated in accordance with the sound management of the biophysical, health, social and economic factors, as well as in compliance with the relevant legislative requirements. The further development of Çöpler Mine with the Sulfide Expansion Project will have a range of positive and negative impacts on the biophysical and socioeconomic environments. Some of the positive impacts that have been identified as having the most potential to benefit the social conditions are direct local employment and local procurement starting from the construction phase of the Project.
- The impacts to the biophysical environments, without mitigation measures, results in the potential of some negative impacts, however when mitigation measures are implemented, these impacts can be reduced significantly, as addressed in EIA and ESIA reports.

- The Sulfide Expansion Project ESIA process did not identify any impacts which are considered to be fatal flaws in the project due to the limited nature of sensitive environmental and human receptors and the disturbed nature of the site.

The residual environmental and socio-economic impacts are identified in accordance with the information provided in ESIA report (September, 2015) and summarized in Table 25-1.

Table 25-1 Residual Environmental and Socio-Economic Impacts Identified for the Çöpler Sulfides Project in ESIA Report (September, 2015)

Impact	Residual Impact	Risks (Negative Impact)	Opportunities (Positive or No Impact)
Geology, Geophysics and Geomorphology			
Removal of mineable resource	Medium	Negative	
Changes to topography and significant site features due to mining; Visual Impacts	Medium	Negative	
Soils, and Land Use Capability			
Degradation and/or loss of soil resulting in reduce land capability	Low	Negative	
Air Quality and Climate Change			
Increase in nuisance and health risks to residents due to increase in ambient dust concentrations - PM_{10} & TSP	None		
Health risk to workers and local villages due to increases in ambient gas concentrations - SO_x, NO_x and Volatile Organic Compounds	None		
The influence of mine generated greenhouse gas emissions on atmospheric processes - Climate change	None		
Noise and Vibration			
Increase in background noise and vibrations levels causing disturbance to site workers and nearby village residents due to the Çöpler Project	None		
Surface Water			
Change in the natural hydrological regime of affected catchment areas resulting in potential changes to the distribution and availability of clean surface water (Biological and human)	Low		Neutral
TSF failure due to faulting or seismic activity and/or poor design/installation and/or operational management leading to contamination of surface water resources and human exposure (community health & safety).	Low	Negative	
HLF failure due to faulting or seismic activity and/or poor design/installation and/or operational management leading to contamination of surface water resources and human exposure (community health & safety).	Low	Negative	
WRD failure due to faulting or seismic activity and/or poor design/installation and/or operational management leading to contamination of surface water resources and human exposure (community health & safety).	Low	Negative	
Groundwater			
Reduction in available groundwater to surrounding users due to mine consumption and open pit dewatering	Low		Neutral
Contamination of groundwater aquifer due to seepage from HLF, TSF and WRDs (Cumulative Assessment of the Seepage from the Project Units)	Low	Negative	

Impact	Residual Impact	Risks (Negative Impact)	Opportunities (Positive or No Impact)
Pit Lake Water Quality			
Pit Lake formation and surface decant leading to water quality deterioration in natural receiving water bodies	Low	Negative	
Biodiversity (Flora and Fauna)			
Loss of biodiversity due to mining disturbances	None		
Cultural Heritage and Natural Assets			
Damage to archaeological sites and historical artefacts due to project related excavation and construction activities	None		
Solid and Liquid Wastes			
Uncontrolled collection, storage, treatment and disposal of solid and liquid wastes from mining activities	None		
Socio-economic Impacts of the Çöpler Project			
Employment generation	Medium		Positive
Increased economic activity, leading to inflation in local prices	High	Negative	
Opportunities for local suppliers and contractors	Medium		Positive
Loss of Land and Natural Resources			
Loss of private land and access to grazing and firewood	Medium	Negative	
Loss of traditional livelihoods and threat to long term sustainable livelihoods	Medium	Negative	
Sense of Place			
Change in rural landscape impacting on people's feeling of wellbeing	Medium	Negative	
Social and Cultural Practices			
Increase in social problems	Low	Negative	
Real or perceived lack of or unequal distribution of Project benefits potentially leading to social tension	Low	Negative	
Social Services			
Additional pressure on social services	Low	Negative	
Decommissioning and Closure			
Closure of mine leading to economic decline	Medium	Negative	
Cumulative Impacts of the Active Projects in the Region	Medium	Negative	

25.9 Mine Closure and Sustainability

Closure of the Çöpler Mine will include decommissioning all facilities without an identified post-mining land use, grading of all fill slopes to 2.5H:1V, covering facilities with a reclamation cover and re-vegetating the surface of all disturbed areas. The closure earthwork and decommissioning work will take approximately four years and cost US$31.9 million. Treatment of pit lake water will be conducted on an ongoing basis and will cost an estimated US$3.8 million. Process fluid management of the seepage from heaps and tails will be conducted in a zero-discharge fashion and will cost US$14.0 million. Monitoring is estimated to cost US$3.0 million. General and administration costs including land holding, camp, overhead labor and other costs are estimated to be about US$11.0 million or 18% of direct costs. These items comprise the majority of the closure costs calculated for the Çöpler Mine.

The SRK closure team has reviewed the available information and conducted a site visit to review the proposed locations of the project facilities. They have developed the following conclusions:

- It may be possible to change the required reclamation slope for the heap to 2H:1V to minimize the amount of offloading and/or liner extension. A geotechnical study should be done to establish whether this steeper slope would meet the reclamation objectives.
- Consideration should be made to if alternatives can be determined to provide positive surface drainage to overcome the ponding as a result of the settlement of the tailings.
- A study should be prepared to evaluate the feasibility and consequences of allowing long-term closure of the tailings impoundment with a depression that will capture run-on.
- There is a lack of available topsoil at the site. A study by the Istanbul Technical University identified a range of between 0 to 30 cm of poor quality topsoil was available within the footprint of the facilities at the site. However, the EIA and 2009 Closure Plan call for the placement of 1-meter of topsoil on disturbed areas. It is likely the project will not be able to comply with this requirement because of this disconnect between available topsoil and the quantity required.
- Because of the steepness of the terrain at the site there are limited locations to store topsoil. Currently topsoil is stored adjacent to the administration area on the west side of the fill slope. This topsoil will need to be relocated prior to construction of the fill for the sulfide mill. Some topsoil is also stored on the top of the south waste rock dump which might also need to be moved prior to building this dump to its capacity.
- Due to the design of the heap leach side slopes and the required reclamation slope of 2-2.5H:1V it is possible that either ore will need to be offloaded or the liner will need to be extended for closure. Unit rates for these activities were provided by Golder.

- There is a roughly 110 meter high angle of repose fill slope adjacent to the heap leach on the east side. The crest is within 27 meters of the toe of the heap leach and the toe is about 10 meters from Sabırlı Creek. It will not be possible to re-grade this slope to meet the reclamation goals for slopes. It will also not be possible to place topsoil or re-vegetate this slope.
- The tailings impoundment operating design specifies the supernatant pond will be located against the hill on the east side with tailings deposition on beaches to the north, west and south. This design will require additional fill to create a positive draining surface that will not pond water. Since the fill will need to be placed in the supernatant pond area it will also be difficult to predict the amount of fill required to overcome differential settlement in that area.
- Based on discussions with the engineering team designing the process and tailings systems, it is likely the tailings at closure will not support equipment and may not do so for quite some time. This requires the placement of a traffic layer of waste rock and or placement of geosynthetic fabric in order to place the final cover.

25.10 Capital Cost Estimate

The capital cost estimate prepared for Alacer is advanced with 72% of equipment costs secured through a purchase order or other form of firm commitment. The cost of the Project has increased over the feasibility study due to conversion to horizontal autoclaves, increased material quantities and increase in schedule.

The estimated cost to design, procure, construct and start-up the facilities from January 1, 2016 is $721.2 million.

The capital cost estimate was prepared in Q4 2015 US dollars according to Amec Foster Wheeler procedures. The capital cost reflects the reuse of existing infrastructure where appropriate, the addition of essential new infrastructure, the engineering, procurement, construction management and commissioning of a POX based gold recovery circuit. The estimate also includes construction of all required facilities, equipment first fills and the spares holdings necessary to operate this additional plant at the Çöpler site. Allowances are incorporated in the estimate for growth, contingency and escalation.

25.11 Operating Cost Estimate

LOM operating costs were developed by Alacer for mining, processing and support costs. During the development of the operating costs, the following conclusions were made:

- Delivered reagent cost estimates were updated in October 2015 by Anagold's procurement department based upon contracts in place for reagents in use in the current oxide heap leach operation (such as sodium cyanide) and on the basis of budgetary quotations from prospective local suppliers for those reagents that will be used in the sulfide plant operation.
- While Anagold has begun to develop more data for carbonate content in the ore body, additional definition is needed. Anagold is assaying carbonate content and the data will be available for ore blending planning once operations begin. The possible impact is on ore control and acid additions. The risk is increased reagent consumptions, resulting in higher costs.

- Unit supply costs for consumables from 2015 to end of mine life have not taken into account escalation in price increases.

25.12 Economic Analysis

Section 2.2 summarizes the forward-looking statement cautionary language applicable to this section.

A financial analysis for the Sulfide Expansion Project was carried out using an incremental or differential cash flow approach. As part of the financial analysis, the following interpretations and conclusions were made.

- The Sulfide Expansion Project is economically viable yielding an IRR of 19.2% and an NPV of $728 million at a discount rate of 5%. The project payback is 3 years.
- A decrease in the price of gold would cause a drop in IRR and NPV and lengthen the payback period. However, there is an opportunity that if the gold price increases, IRR and NPV will increase and the payback period shortened.
- There is a lack of ore carbonate content data in the mine model. Higher than assumed carbonate values could increase operating costs by requiring more sulfuric acid, thus decreasing the financial metrics. Lower than assumed carbonate costs will result in decreased operating costs and improved financial metrics.
- There is a risk that metal grades could be less than modeled, causing decreases in the cash flow and financial metrics. Conversely, grades could be higher than modeled, increasing cash flows and financial metrics.
- There is a risk that the capital cost can increase due to:
    - Commodity price volatility
    - Global inflation / deflation
    - Decrease in supplier production capacity
    - Shortage in skilled labor
    - Currency exchange changes

  Capital cost increases will cause a decrease in IRR and NPV and increase the payback term.

25.13 Project Execution Plan

A stand-alone preliminary PEP was developed during the detailed design phase of the Sulfide Expansion Project and it was concluded that there are no project execution issues identified at this time that could jeopardize the success of the Project.

25.14 Conclusions

It was concluded that the Sulfide Expansion Project is economically feasible and that the Project should move to construction. On May 12, 2016 Alacer released the announcement "Alacer Gold Announces Çöpler Sulfide Project Approval" stating that the Board of Directors had approved full construction of the Çöpler Sulfide Project.

26.0 RECOMMENDATIONS

It was concluded that the Sulfide Expansion Project is economically and technically feasible. The remaining cost as of January 1, 2016 is estimated at $721.2 million.

Additional recommendations for the project were compiled by the various report contributors and are included below. These recommendations should be evaluated and action taken where necessary to support a successful Project.

26.1 Database, QA/QC and Mineral Resource Estimation

The following recommendations were made by Amec Foster Wheeler:

- Anagold should initiate a procedure to review collar and down-hole information by the responsible geologist, then signed, dated and added to the drill hole folder.
- Anagold should attempt to obtain as many historical logs as possible and implement procedures to ensure current data are collected and stored in a series of folders. Ideally each drill hole would be stored in an individual folder. For current and future holes, the Anagold Senior Geologist should review, sign and date the final log.
- Differences noted in the ALS and SGS assays should be corrected in the Datashed master database, and the updated database should be verified prior to future resource estimation.
- Based on the sample number of the blank samples, it appears only 1 in 60 samples is submitted as a blank. Amec Foster Wheeler recommends that Alacer commence submitting 1 in 20 samples as a blank.
- Anagold should follow QA/QC protocol by sending 5% of the samples to a secondary laboratory for check analysis. Samples should be sent on a regular basis, and not at the end of the drilling program.
- QA/QC results should be monitored on a regular basis during a drilling program and the laboratory asked to follow up on samples that are outside the acceptable range.
- Anagold should add additional CRMs to monitor copper and silver assays. In addition, Anagold should add additional CRMs to monitor sulfur assays near the current oxide/sulfide threshold of 2% sulfur.
- The current detection limit for SGS procedure (ICP40B) for silver is 2 g/t. Amec Foster Wheeler recommends employing an analytical method such as GE ICM40B which would provide a detection limit of 0.02 g/t.
- Drill samples from when the property was managed by Rio Tinto were sent to the OMAC. Anagold should compare original assay values to those stored in the database.
- Since the mineralization locally follows the lithological contacts, using a search ellipse that follows these contacts (dynamic anisotropy) should be evaluated in future models.

- The distinctively different sulfur populations for each lithology (although each lithology hosts both low- and high-sulfur mineralization) suggests that sulfur should be domained by lithology for estimation which was done in the current model. This relationship, however, may be more complex and should be studied again for future models.
- Çöpler is a geologically complex deposit with multiple metals that must be tracked along with oxidation type and lithologies. Further work should be done to verify and adjust resource model domains and parameters used. This would allow for an improved resource estimate and a greater understanding of the deposit.
- The quantity and distribution of the total carbon and sulfate sulfur should be studied to better quantify any potential impacts of these elements in the POX circuit, and for implications for acid rock drainage.

Depending on the requirement to utilize third-party consultants in the above work, Amec Foster Wheeler considers all of the work can be completed concurrently in a single work phase and is budgeted between $100,000 and $200,000.

26.2 Development Drilling

A total of $0.56 million was approved for 3,600 m of RC drilling in the Çöpler Main pit to define additional leachable oxide ore for near term mining. The program started in January 2016. Additional funding will be considered based on drill results from this program.

A total of $0.73 million was approved for approximately 4,000 m of DD drilling to confirm sulfide stockpiled ore grade, grade distribution and mineralogy. The program was in progress during Q1 2016.

26.3 Mining and Mineral Reserves

A credible Mineral Reserve exists within the confines of the designed open pit presented within this report. The design is well suited for open-pit mining operations by conventional mining equipment. The production schedule is readily achievable and the mining operation will continue in the same manner as the existing oxide operation at the Çöpler Mine.

The following items are recommended as part of the next phase of engineering and design associated with the project. These recommendations are:

- 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.
- Further refinement of the modeled carbonate and sulfide sulfur grades in the resource model should be completed.
- Further mapping and definition of alteration types and zones should be completed so that improved pit slope angles can be realized and geotechnical risk can be reduced.
- A detailed pit dewatering and depressurization plan should be designed and implemented to account for the increased depths of mining activities through the sulfide phases of the pit design.

- Further mapping and definition of the local and regional fault structures should be completed to reduce or realize geotechnical risk in the areas where these structures intersect the pit.
- Pit designs should be further optimized for haulage requirements, blend scheduling, and backfill potential. Conduct limit equilibrium analysis for both static and pseudo-static cases using the FS pit design. The purpose of the analysis is to determine a FOS to quantify the risk of open pit wall failure in various areas of the project.

26.4 Waste Rock Storage Areas

It is recommended that further optimization of the sequencing of the WRSA facilities be conducted through mine planning in order to improve haulage efficiency and to ensure an adequate blend of NAF and PAF material.

26.5 Metallurgy and Mineral Processing

The following are recommendations for metallurgy and mineral processing identified during the FS engineering. These should be examined during the next phase of the project.

26.5.1 Heap Leach Operation

- It is recommended that an effective heap leach production model be maintained and that the model be calibrated at least annually against actual gold production from the heap leaching facilities. The calibration should be used to check the long term gold recovery assumptions for the heap leaching operation on any annual basis, and the assumptions should be modified and updated as required for mine planning and production planning.
- Sulfide sulfur content in heap leach feed materials, as well as column and IBRT feed materials should be measured routinely and correlated against gold extraction.

26.5.2 POX Processing

- Alacer should begin development of all aspects of the sulfide process feeding blending program including:
  - Perform assaying on drill-hole samples for any parameters not in the database but required for the blending program.
  - Perform complete assays and comminution characterization tests on material fed or currently residing in the sulfide stockpile per the list of blending parameters developed.
  - Develop a plan to incorporate all blending parameters in the sulfide resource model.
  - Develop plans for coordination of the mine and process personnel who will be in charge of the sulfide process feed blending program.
  - Develop the basics and details for the blending program defining personnel, responsibilities, and methodologies.

- It is critical that assaying of carbonate be initiated and completed on drill hole samples supporting the resource model and that the resource model be modified to include carbonate data to support the sulfide plant feed blending program.
- The manganese diorite rock type was demonstrated to have some negative impacts on the metallurgical performance of the POX system if present in the process feed at a proportion greater than 40% w/w. It is imperative that the manganese diorite be differentiated from the Main diorite in the sulfide resource model and be part of the sulfide plant feed blending program.
- It is recommended that comminution variability testing be performed on an additional 50 to 100 drill hole interval samples to develop enough data to allow incorporation into the resource model to predict sulfide plant feed hardness over the life of the mine.
- It is recommended that a review of the plant materials of construction be conducted early in the next phase of the project to ensure that the proper materials have been selected for the respective applications and to optimize the materials of construction with the goal of capital cost savings without sacrificing the integrity of the plant or causing an increase in operating costs.
- Perform a study of tailings disposal, optimizing slurry disposal and examining slurry disposal versus dry tailings to meet project closure and reclamation requirements.
- Perform a scoping evaluation during second phase of variability testing comparing SO_2/air process to an alternative cyanide detoxification process such as Caro's acid.
- Review and optimize the design of the neutralization system (i.e. elimination of one of the two neutralization tanks) in the next phase of the project.
- Review and optimize the plant layout to take full advantage of fluid flow by gravity.
- Review the CIP plant and ADR plant design parameters in the next project phase to ensure adequate carbon adsorption, carbon handling equipment, and ADR plant capacities are provided to address any concerns with CIP operation at temperatures ranging from 55°C to 65°C.

26.6 Infrastructure

Discussion with Alacer of lessons learned from the construction of the heap leach facility and SART plant should be planned during early basic engineering. This information will greatly benefit the infrastructure engineering.

These reviews may include the following:

- Scheduling construction time of year (winter versus summer) and to optimize transportation of heavy and large equipment to site and critical lifts during favorable weather.
- Review size and requirements of maintenance and warehouse facilities for the new facility.

26.7 Tailings Storage Facility

Embankment stability and deformation evaluations should be updated to reflect planned ultimate embankment height and understanding of geologic conditions, following excavation and evaluation of actual bedrock conditions during initial Phase 1 construction. In addition, the deposition model should be updated using the updated design and with the new geotechnical data from tests completed on new tailings samples during the pilot program in the third quarter 2015.

26.8 Marketing

Alacer should continue to evaluate copper market conditions and re-examine Cu circuit costs and benefits when copper prices warrant.

26.9 Environmental/Permitting

Recommendations from the ESIA report are outlined below:

- An Integrated Water Management Plan will be developed for the Çöpler Mine. The management plan will enable the detailed assessment of process water use and water management during the operation phase as well as planning for the closure lake formation. Within the framework of this plan, the management of the project water use, inflows to and outflows from the project site and process and the project site interception channel management will be assessed in detail by specialized studies. Integrated water management report will be prepared every 5 years (changes in water quality and their comparison with modeling results, amount of water consumption and its impact, amount of waste water, its disposal and impact, effectiveness of the measures taken at the facility etc.), in the light of the estimations stated at the EIA report for the closure and the post-closure period, and will be submitted to the General Directorate of State Hydraulic Works.
- A monitoring program will be conducted in accordance with the commitments in the EIA report and reported to the Ministry of Environment and Urban Planning, to the General Directorate of State Hydraulic Works, and to any other public institutions and organizations upon request.
- Anagold will review the environmental and social assessment regularly throughout the life of the Project to:
    - Identify and evaluate aspects and impacts not covered by this initial ESIA
    - Address any changes in the project or new developments arising subsequent to the completion of the initial ESIA.
- Currently, SEP includes the record of consultations conducted till the release of this report and list of stakeholder issues; the plan for disclosure of information and consultation with project stakeholders during feasibility, construction, operation and closure phases of the project; and a grievance mechanism for receiving concerns about the Project’s environmental and social performance and for facilitating the resolution of the concerns (it applies to stakeholders, particularly affected communities, and workers). When the project enters the construction phase, and throughout the remaining life of the project, stakeholder engagement will also include:
    - Reporting on the Environmental and Social Management Plan (ESMP) and relevant supporting management plans; and

- Opportunities for stakeholders to respond to the information received.

26.10 Mine Closure and Sustainability

SRK recommends that Alacer review the tailings design to see if it can be modified to provide drainage surface on the tailings storage facility surface after closure and prevent ponding of water where settlement of the surface will be occurring. Alternately, a FS should be undertaken to evaluate closure design without positive drainage from the tailings surface.

A trade-off study should be undertaken to evaluate whether the application of geotextile on the tailing surface might allow a thinner waste rock cover.

SRK also recommends that Alacer conduct studies for growth media material sourcing.

26.11 Capital Cost Estimate

The capital cost estimate prepared for Anagold is advanced with 72% of equipment costs secured through a purchase order or other form of firm commitment. The cost of the project has increased over the feasibility study due to conversion to horizontal autoclaves, increased material quantities and increase in schedule.

The estimated cost to design, procure, construct and start-up the facilities from January 1, 2016 is $721.2 million.

26.12 Operating Cost Estimate

The following are recommendations pertaining to the operating costs. They should be clarified and resolved as the project progresses.

- As the project gets closer to start-up, Anagold should secure suppliers for reagents and consumables in order to lock in favorable pricing and to ensure adequate supplies can be obtained in a timely manner.
- Anagold should continue to compile carbonate data from the ore body to validate acid consumption rates.

26.13 Economic Analysis

It was concluded that the Çöpler Sulfide Expansion Project is economically feasible and that the project should move to construction.

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DATE AND SIGNATURE PAGE

The effective date of this report is June 9, 2016.

Signed this 9 of June, 2016

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