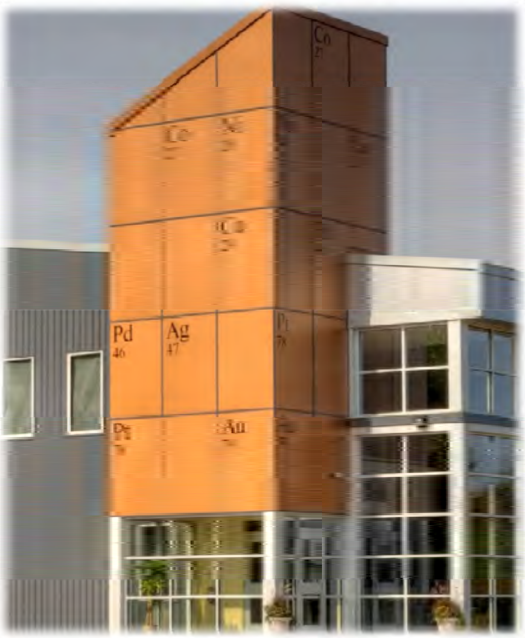
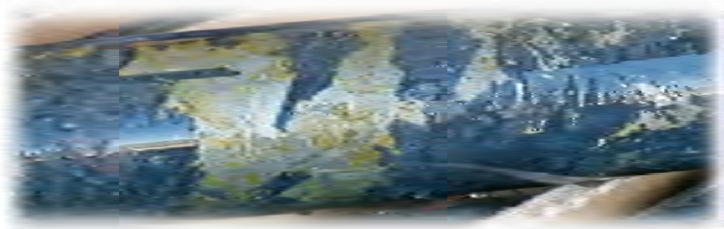
Twin Metals Minnesota Project
Ely, Minnesota, USA
NI 43-101 Technical Report on Pre-feasibility Study





Prepared for:
Duluth Metals Corp.

Prepared by:
Mr John Barber, P.E., AMEC
Dr Harry Parker, RM SME, AMEC
Mr David Frost, F.AusIMM, AMEC
Ms Janine Hartley, P.E., AMEC
Mr Trey White, P.E. AMEC
Mr Chris Martin, C.Eng., Blue Coast
Dr Robert Sterrett, P.G., Itasca
Ms Joanna Poeck, RM SME, SRK
Dr Ted Eggleston, RM SME, AMEC
Dr Lynton Gormely, P.Eng., AMEC
Mr Simon Allard, P.Eng., AMEC
Dr Srikant Annavarapu, P.E., AMEC
Mr Tom Radue, P.E., Barr
Mr Matthew Malgesini, P.E., Golder
Dr Matthew Pierce, P.E., Itasca

Effective Date:
20 August 2014
Amended:
6 October 2014

Project Number:
176916

CERTIFICATE OF QUALIFIED PERSON

I, John Barber, P.E., am employed as the Technical Director, Underground Mining with AMEC E&C Services Inc.

This certificate applies to the technical report titled “Twin Metals Minnesota Project, Ely, Minnesota, USA, NI 43-101 Technical Report on Pre-feasibility Study” that has an effective date of 20 August 2014 (the “technical report”).

I am a Registered Member of the Society for Mining, Metallurgy and Exploration (#149130). I am licensed as a Professional Engineer in the State of Minnesota (#51889). I graduated from Virginia Polytechnic Institute and State University with a B.SC. in Mining Engineering in 1979.

I have practiced my profession for 35 years. I have been involved in the engineering, planning, and operations of a variety of underground base metal mines. I have been involved in the management of studies of various levels for underground base metals mines, including Oyu Tolgoi (Mongolia), Resolution Copper (Arizona), and Voisey’s Bay (Labrador).

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 Twin Metals project from 23–24 July, 2013.

I am responsible for or co-responsible for Sections 1.2, 1.3, 1.19, 1.22, 1.23.5, 1.26 and 1.27; Section 2; Section 3; Section 5; Section 16.3.5; Section 16.3.10, Section 16.4.6; Section 16.9; Sections 18.3; 18.8.1; and 18.16.2; Sections 21.1, 21.3.1, 21.3.2.6 to 21.3.2.10, 21.3.5 to 21.3.7 and 21.4; Section 23; Sections 24.1.4, 24.1.9, 24.1.11, 24.2.5, 24.2.11, and 24.2.13, Sections 25.15, 25.16 and 25.19; Sections 26.1, 26.2.1, 26.2.2.2, and 26.3, and Section 27 of the technical report.

I am independent of Duluth Metals Limited as independence is described by Section 1.5 of NI 43–101.

I have served on the TMM Technical Committee representing Duluth Metals from May, 2011, to June, 2014.

I have been involved with the Project during the preparation of this technical report as the Project Manager.

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 October, 2014

“Signed and sealed”

John Barber, P.E.

AMEC E&C Services, Inc.
Mining & Metals
1640 S. Stapley Dr. Suite 241
Mesa, AZ. 85204
Tel: (480) 253-4930
Fax: (480) 253-4932

CERTIFICATE OF QUALIFIED PERSON

I, Dr. Ted Eggleston, RM SME, am employed as a Principal Geologist with AMEC E&C Services Inc.

This certificate applies to the technical report titled “Twin Metals Minnesota Project, Ely, Minnesota, USA, NI 43-101 Technical Report on Pre-feasibility Study” that has an effective date of 20 August 2014 (the “technical report”).

I am a Registered Member of the Society for Mining, Metallurgy and Exploration (#4115851RM) and licensed as a Professional Geologist in the States of Wyoming (PG-1830) and Georgia (PG002016). I graduated from Western State University of Colorado with a BA degree in 1976 and from the New Mexico Institute of Mining and Technology with MSc and PhD degrees in Geology in 1982 and 1987 respectively.

I have practiced my profession for 35 years during which time I have been involved in the exploration for, and estimation of, mineral resources and mineral reserves, for various mineral exploration projects and operating mines. I have explored for, provided technical assistance for, or audited Ni, Cu and PGE resources for a number of mineral deposits, including Munali (Zambia); Niquelândia and Fortaleza (Brazil); Stillwater (Montana); McCreedy East, McCreedy West, Levack, Thunder Bay North (Ontario); Bucko Lake (Manitoba); and Kabanga (Tanzania).

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).

Under the supervision of Dr. Parker, Dr. Eggleston visited site and/or Project offices on 26 to 30 April, 2011, 6–18 June 2011, 6–16 September 2011, 10–22 March 2012, 4–7 April 2012, 7–23 May 2012, 6–22 June 2012, 19–10 July 2012, 17–22 February 2013, 7–27 April 2013, 5–10 August 2013, and 14–15 October 2013.

I am responsible for or co-responsible for Sections 1.4 to 1.11, 1.13 to 1.14, 1.26 and 1.27; Sections 2.2 and 2.3; Section 3; Section 4; Section 6; Section 7; Section 8; Section 9; Section 10; Section 11; Section 12; Section 14; Sections 24.1.1, 24.2.1, and 24.2.2; Sections 25.1, 25.2, 25.3, 25.5 and 25.19; Section 26.2.2.1; Section 27 and Appendix A of the technical report.

I am independent of Duluth Metals Limited as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Project since 2011 during which time I have prepared or supervised mineral resource estimates on the Project. I have been a co-author on the following technical reports on the Project:

Parker, H.M. and Eggleston, T.L., 2012a: Maturi, Birch Lake, and Spruce Road Cu-Ni-PGE Projects Ely, Minnesota USA; 27 July 2012, NI 43-101 Technical Report prepared by AMEC E&C Services Inc. for Duluth Metals Limited, 302 p.

Parker, H.M. and Eggleston, T.L., 2012b: Maturi, Birch Lake, and Spruce Road Cu-Ni-PGE Projects Ely, Minnesota USA; 15 September 2012, NI 43-101 Technical Report prepared by AMEC E&C Services Inc. for Duluth Metals Limited, 301 p.

Parker, H.M. and Eggleston, T.L., 2014: Maturi, Birch Lake, and Spruce Road Cu-Ni-PGE Projects Ely, Minnesota USA: NI 43-101 Technical Report prepared by AMEC E&C Services Inc. for Duluth Metals Limited, effective date 2 January 2014, 376 p.

AMEC E&C Services, Inc.
961 Matley Lane, Suite 110
Reno, NV, 89502
Tel: 775 331 2375
Fax: 775-331-4153

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 October, 2014

"Signed and stamped"

Dr. Ted Eggleston, RM SME

AMEC E&C Services, Inc.
961 Matley Lane, Suite 110
Reno, NV, 89502
Tel: 775 331 2375
Fax: 775-331-4153

CERTIFICATE OF QUALIFIED PERSON

I, Dr. Harry Parker, RM SME, am employed as a Consulting Geologist and Geostatistician with AMEC E&C Services Inc.

This certificate applies to the technical report titled "Twin Metals Minnesota Project, Ely, Minnesota, USA, NI 43-101 Technical Report on Pre-feasibility Study" that has an effective date of 20 August 2014 (the "technical report").

I am a Fellow of the Australian Institute of Mining and Metallurgy (#113051), and a Registered Member of the Society for Mining, Metallurgy and Exploration (#2460450). I am a registered geologist in the State of 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 46 years during which time I have been involved in the estimation of mineral resources and mineral reserves for various mineral exploration projects and operating mines. I have either estimated or audited Ni, Cu and PGE resources for a number of mineral deposits, including the Area 5 deposit (Maine), Stillwater (Montana); McCreedy East (Ontario), Voiseys Bay (Labrador) and the Platreef deposit (South Africa). From 1966 to 1969 I mapped surface outcrops, logged drill core and undertook preliminary resource estimates for The Hanna Mining Company on lands now contained within the Maturi and Spruce Road portions of the Project.

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 Twin Metals project and/or Project offices from 26 to 30 April, 2011, 6–16 September 2011, 5–7 April 2012, 19–20 June 2012, 23–27 April 2013, 28 June 2013, 6 August 2013 and 28 August 2013.

I am responsible for or co-responsible for Sections 1.4 to 1.11, 1.13 to 1.14, 1.26 and 1.27; Sections 2.2 and 2.3; Section 3; Section 4; Section 6; Section 7; Section 8; Section 9; Section 10; Section 11; Section 12; Section 14; Sections 24.1.1, 24.2.1, and 24.2.2; Sections 25.1, 25.2, 25.3, 25.5 and 25.19; Section 26.2.2.1; Section 27 and Appendix A of the technical report.

I am independent of Duluth Metals Limited as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Project since 2011 during which time I have prepared or supervised mineral resource estimates on the Project. I have been a co-author on the following technical reports on the Project:

Parker, H.M. and Eggleston, T.L., 2012a: Maturi, Birch Lake, and Spruce Road Cu-Ni-PGE Projects Ely, Minnesota USA; 27 July 2012, NI 43-101 Technical Report prepared by AMEC E&C Services Inc. for Duluth Metals Limited, 302 p.

Parker, H.M. and Eggleston, T.L., 2012b: Maturi, Birch Lake, and Spruce Road Cu-Ni-PGE Projects Ely, Minnesota USA; 15 September 2012, NI 43-101 Technical Report prepared by AMEC E&C Services Inc. for Duluth Metals Limited, 301 p.

Parker, H.M. and Eggleston, T.L., 2014: Maturi, Birch Lake, and Spruce Road Cu-Ni-PGE Projects Ely, Minnesota USA: NI 43-101 Technical Report prepared by AMEC E&C Services Inc. for Duluth Metals Limited, effective date 2 January 2014, 376 p.

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 October, 2014

"Signed and stamped"

Dr. Harry Parker, RM SME

CERTIFICATE OF QUALIFIED PERSON

I, Lynton Gormely, Ph.D., P.Eng., at the effective date of the report was employed as a Principal Process Engineer with AMEC Americas Ltd.

This certificate applies to the technical report titled "Twin Metals Minnesota Project, Ely, Minnesota, USA, NI 43-101 Technical Report on Pre-feasibility Study" that has an effective date of 20 August 2014 (the "Technical Report").

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia, registration number 10005. I graduated from the University of British Columbia with a Bachelor of Applied Science degree in 1968 and from the University of British Columbia with a Ph.D. in Chemical Engineering in 1973.

I have practiced my profession for 39 years. I have been directly involved in process engineering design and construction projects for the mining industry for the recovery of base and precious metals. I have experience with the principles of the design of the metallurgical testwork, the design of the process flow sheets, and the selection of the mineral processing equipment.

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 Twin Metals project.

I am responsible for or co-responsible for Sections 13.2 and 13.3 of the technical report.

I am independent of Duluth Metals Limited as independence is described by Section 1.5 of NI 43–101.

I have reviewed some aspects of the completed metallurgical testwork on the Twin Metals project since 2012 in support of considerations of reasonable prospects of eventual economic extraction for mineral resource estimates.

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 October, 2014

"Signed and sealed"

Lynton Gormely, Ph.D., P.Eng.

AMEC Americas Limited
111 Dunsmuir Street, Suite 400
Vancouver, B.C. V6B 5W3
Tel (604) 664-3030
Fax (604) 664-3057

CERTIFICATE OF QUALIFIED PERSON

I, David Frost, FAusIMM am employed as Technical Director, Process, with AMEC International Ingeniería y Construcción Lida.

This certificate applies to the technical report titled "Twin Metals Minnesota Project, Ely, Minnesota, USA, NI 43-101 Technical Report on Pre-feasibility Study" that has an effective date of 20 August 2014 (the "technical report").

I am a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM; #11089). I graduated with a Bachelor of Metallurgical Engineering (B. Met Eng) from the Royal Melbourne Institute of Technology in 1991.

I have worked as a metallurgist and process engineer for over 21 years since my graduation from university. I have been involved in process operations and process plant design in various commodities and in various capacities during that time.

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 Project site.

I am responsible for or co-responsible for Sections 1.18, 1.23.3, 1.26 and 1.27; Section 2.2; Section 3; Section 13.1; Section 17; Sections 21.3.4, 21.3.5; Sections 24.1.8, and 24.2.9; Sections 25.11 and 25.19 and Section 27 of the technical report.

I am independent of Duluth Metals Limited as independence is described by Section 1.5 of NI 43–101.

I have no previous involvement with the project.

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 October, 2014

"Signed and sealed"

David Frost, F.AusIMM.

AMEC International Ingeniería y Construcción Limitada
Av. Apoquindo 3846, piso 15, Las Condes
Santiago, Chile
Tel (56) 2 210 9500
Fax (56) 2 210 9510

CERTIFICATE OF QUALIFIED PERSON

I, Simon Allard, P.Eng., am employed as a Principal Consultant and Study Manager with AMEC Americas Ltd.

This certificate applies to the technical report titled “Twin Metals Minnesota Project, Ely, Minnesota, USA, NI 43-101 Technical Report on Pre-feasibility Study” that has an effective date of 20 August 2014 (the “technical report”).

I am a registered Professional Engineer in the Province of British Columbia. I graduated from Université Laval in 2004 with a Baccalauréat coopératif en génie des mines et de la minéralurgie Degree.

I have practiced my profession for 10 years. I have been directly involved in cash-flow modelling, risk evaluation, real-options valuation, financial analysis, marketing studies and financial review of mines located in Africa, Mongolia and North 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 Twin Metals project.

I am responsible for or co-responsible for Sections 1.1, 1.20, 1.24 to 1.27, Section 2.2, Section 3, Section 19, Section 22, Section 25.13, 25.17, 25.18, 25.19, Section 27 of the technical report.

I am independent of Duluth Metals Limited as independence is described by Section 1.5 of NI 43–101.

I have no previous involvement with the Twin Metals project.

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 October, 2014

“Signed and sealed”

Simon Allard, P.Eng.

AMEC Americas Limited
111 Dunsmuir Street, Suite 400
Vancouver, B.C. V6B 5W3
Tel (604) 664-3030
Fax (604) 664-3057

CERTIFICATE OF QUALIFIED PERSON

I, Janine Hartley, P.E., am employed as a Senior Engineer with AMEC E&I Services, Inc.

This certificate applies to the technical report titled "Twin Metals Minnesota Project, Ely, Minnesota, USA, NI 43-101 Technical Report on Pre-feasibility Study" that has an effective date of 20 August 2014 (the "technical report").

I am licensed as a Professional Engineer in the State of Nevada. I graduated from Michigan Technological University with a Bachelor of Science degree in Metallurgical Engineering in 1980. I graduated from Louisiana State University with a Master of Engineering degree in 1989.

I have practiced my profession for thirteen years. I have been directly involved in regulatory permit writing, and environmental permit application and Environmental Impact Statement preparation.

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 Project site.

I am responsible for or co-responsible for Sections 1.21, 1.26 and 1.27; Section 2.2; Section 3; Section 20; Sections 24.1.10, and 24.2.12; Section 25.14, Section 26.2.2.3, and Section 27 of the technical report.

I am independent of Duluth Metals Limited as independence is described by Section 1.5 of NI 43–101.

I have no previous involvement with the project.

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 October, 2014

"Signed and sealed"

____

Janine Hartley, P.E.

CERTIFICATE OF QUALIFIED PERSON

I, Dr. Srikant Annavarapu, P.E. am employed as a Principal Mining Engineer with AMEC E&C Services Inc.

This certificate applies to the technical report titled "Twin Metals Minnesota Project, Ely, Minnesota, USA, NI 43-101 Technical Report on Pre-feasibility Study" that has an effective date of 20 August 2014 (the "technical report").

I am a Registered Member of the Society for Mining, Metallurgy and Exploration (#4037554). I obtained my Bachelor of Technology degree in Mining Engineering from the Indian Institute of Technology in Kharagpur, India in 1980. I graduated from the University of Arizona in Tucson, Arizona with a M.S. and Ph.D. in Mining and Geological Engineering in 1998 and 2013 respectively.

I have practiced my profession for a total of 33 years since my graduation from university during which time I have been involved in geotechnical studies and design for various mining projects and operating mines. I have either been involved in the geotechnical design or reviewed designs for a number of mining projects, including the Cortez Hills Underground (Nevada), Arenal Deeps (Uruguay); Bokan Mountain (Alaska), Voiseys Bay (Labrador), and the Platreef deposit (South Africa).

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 Twin Metals project.

I am responsible for for Section 16.1.11 of the technical report.

I am independent of Duluth Metals Limited as independence is described by Section 1.5 of NI 43–101.

I have no previous involvement with the Twin Metals project.

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 October 2014

"Signed and sealed"

Dr. Srikant Annavarapu, P.E.

AMEC E&C Services, Inc.
Mining & Metals
1640 S. Stapley Dr. Suite 241
Mesa, AZ. 85204
Tel: (480) 253-4930
Fax: (480) 253-4932

CERTIFICATE OF QUALIFIED PERSON

I, John (Trey) White, P.E., am employed as a Principal Mining Engineer with AMEC E&C Services Inc.

This certificate applies to the technical report titled "Twin Metals Minnesota Project, Ely, Minnesota, USA, NI 43-101 Technical Report on Pre-feasibility Study" that has an effective date of 20 August 2014 (the "technical report").

I am licensed as a Professional Mining and Mineral Processing Engineer in the State of Colorado (license # 47237). I graduated from the Colorado School of Mines with a Bachelor of Science in Mining Engineering degree in 1991. I graduated from the University of Washington with a Master of Business Administration degree in 2009.

I have practiced my profession for 23 years since graduation. I have been directly involved in the feasibility, permitting, start-up, and/or operation of several underground metal mines in the States of Nevada, Washington, California, and Colorado over my career.

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 visited the Twin Metals project from 9 to 10 July 2013, under the supervision of Mr. Barber.

I am responsible for Sections 2.2, 2.3, Section 3, 16.3.7, 18.8.2, 18.8.3, 18.8.4, 18.8.5, 18.8.6, 18.8.7, and 18.9.4. Section 27

I am independent of Duluth Metals Limited as independence is described by Section 1.5 of NI 43–101.

I have no previous involvement with the Twin Metals project.

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 October, 2014

"Signed"

John (Trey) White

John (Trey) White, P.E.

AMEC E&C Services, Inc.
Colorado Center
Tower 2, 2000 S Colorado Blvd,
10th Floor
Tel: 303 935 6505
Fax: 303 935 6575

BARR

CONSENT OF QUALIFIED PERSON

I, Tom Radue, P.E. consent to the public filing of the technical report titled "Twin Metals Minnesota Project, Ely, Minnesota, USA, NI 43-101 Technical Report on Pre-feasibility Study" that has an effective date of 20 August 2014 (the "Technical Report") by Duluth Metals Limited (Duluth).

I also consent to any extracts from, or a summary of, the Technical Report in the Duluth press release entitled "Duluth Metals Highlights Low Copper (C1) Cash Costs and Strong Operating Margins in its Pre-feasibility Study for Twin Metals Minnesota Project" and dated 20 August 2014 (the "press release").

I certify that I have read the press release being filed by Duluth and that it fairly and accurately represents the information in the sections of the Technical Report for which I am responsible.

Dated: 6 October, 2014

"Signed"

____

Tom Radue, P.E.

Minnesota P.E. License No. 20951
Exp. Date 06/30/2016

CONSENT OF QUALIFIED PERSON

I, Christopher John Martin, C.Eng. consent to the public filing of the technical report titled “Twin Metals Minnesota Project, Ely, Minnesota, USA, NI 43-101 Technical Report on Pre-feasibility Study” that has an effective date of 20 August 2014 (the “Technical Report”) by Duluth Metals Limited (Duluth).

I also consent to any extracts from, or a summary of, the Technical Report in the Duluth press release entitled “Duluth Metals Highlights Low Copper (C1) Cash Costs and Strong Operating Margins in its Pre-feasibility Study for Twin Metals Minnesota Project” and dated 20 August 2014 (the “press release”).

I certify that I have read the press release being filed by Duluth and that it fairly and accurately represents the information in the sections of the Technical Report for which I am responsible.

Dated: 6 October, 2014

“Signed and stamped”

Chris Martin, C.Eng.

Blue Coast Metallurgy Ltd.
Unit #2-1020 Herring Gull Way
Parksville, British Columbia
V9P 1R2
Canada

Tel: +1 250 586 0600
Fax: +1 250 586 0445

www.bluecoastgroup.ca

CERTIFICATE OF QUALIFIED PERSON

CONSENT OF QUALIFIED PERSON

I, Matthew Malgesini, P.E., consent to the public filing of the technical report titled “Twin Metals Minnesota Project, Ely, Minnesota, USA, NI 43-101 Technical Report on Pre-feasibility Study” that has an effective date of 20 August 2014 (the “Technical Report”) by Duluth Metals Limited (Duluth).

I also consent to any extracts from, or a summary of, the Technical Report in the Duluth press release entitled “Duluth Metals Highlights Low Copper (C1) Cash Costs and Strong Operating Margins in its Pre-feasibility Study for Twin Metals Minnesota Project” and dated 20 August 2014 (the “press release”).

I certify that I have read the press release being filed by Duluth and that it fairly and accurately represents the information in the sections of the Technical Report for which I am responsible.

Dated: 6 October 2014

“Signed and sealed”

Matthew Malgesini, P.E.

CERTIFICATE OF QUALIFIED PERSON

I, Dr. Robert Sterrett, P.G., am employed as a Principal Hydrogeologist with Itasca Denver, Inc.

This certificate applies to the technical report titled "Twin Metals Minnesota Project, Ely, Minnesota, USA, NI 43-101 Technical Report on Pre-feasibility Study" that has an effective date of 20 August 2014 (the "Technical Report").

I am a member and trustee of the Geological Society of America, a member of the Society of Mining Engineering, and a member of the National Ground Water Association (I am on the Editorial Review Board for the ***Water Well Journal***). I graduated from Indiana University, Bloomington, IN, with a BS degree in Geology (with honors) in 1972. I graduated from the University of Wisconsin, Madison, WI, with a MS in Water Resources Management in 1974. I graduated from the University of Wisconsin, Madison, WI, with a MS in Geology and Geophysics in 1975. I graduated from the University of Wisconsin, Madison, WI, with a PhD in Geology and Geophysics in 1980.

I have practiced my profession for 36 years. I have been directly involved in the analysis of hydrogeologic data from the project site, supervised the groundwater flow modeling that Itasca Denver performed to estimate groundwater inflows to the mine, and I participated in the production of a report regarding the estimates of groundwater inflows. Information from Itasca's work are summarized in the current 43-101 document.

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 Twin Metals Minnesota Project (the "Project") between August 14 and 15, 2012.

I am responsible for or co-responsible for Sections 1.17.2, 1.26, and 1.27; Sections 2.2 and 2.3; Section 3; Section 15.3; Section 16.2; Sections 24.2.7; Section 25.8; Section 26.2.6.1 and Section 27 of the Technical Report.

I am independent of Duluth Metals Limited as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Project during the preparation of the pre-feasibility study and this Technical Report.

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 October 2014

"Signed and sealed"

Dr. Robert Sterrett, P.G.

Itasca Denver, Inc.
143 Union Blvd., Ste. 525
Lakewood, CO 80228
T: 303-969-8033 F: 303-969-8357 www.itascadenver.com

ITASCA
Consulting Group, Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Dr. Matthew Pierce, P.E., am employed as a Principal Engineer with Itasca Consulting Group, Inc.

This certificate applies to the technical report titled "Twin Metals Minnesota Project, Ely, Minnesota, USA, NI 43-101 Technical Report on Pre-feasibility Study" that has an effective date of 20 August 2014 (the "Technical Report").

I am registered professional engineer in the provinces of Ontario and Alberta in Canada. I graduated from Queen's University, Canada with a B.Sc. in Geotechnical Engineering in 1995 and with a M.Sc. in Mining Engineering in 1997. In 2010 I completed my Ph.D. in Mining Engineering from the University of Queensland, Brisbane, Australia.

I have practiced my profession for 16 years and have been directly involved in the application of numerical models to design, sequencing and support of underground excavations, assessment of backfill strength requirements and liquefaction potential, analysis of pit slope stability, and prediction of mining-induced seismicity and surface subsidence.

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 Twin Metals Minnesota Project (the "Project") 21-22 Dec 2011, 16-20 Jan 2012, 13-19 Feb 2012 and 20-21 Mar 2012.

I am responsible for or co-responsible for Sections 1.17.1, 1.26, and 1.27; Sections 2.2 and 2.3; Section 3; Section 15.2; Sections 16.1.1 to 16.1.9, and 16.1.12; Sections 24.1.5, and 24.2.6; Section 25.7; Section 26.2.6.2 and Section 27 of the Technical Report.

I am independent of Duluth Metals Limited as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Project during the preparation of the pre-feasibility study and this Technical Report.

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 October 2014

"Signed and sealed"

Dr. Matthew Pierce, P.E.

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CERTIFICATE OF QUALIFIED PERSON

I, Joanna Poeck, B.Eng., SME-RM, MMSA-QP, do hereby certify that:

1. I am a Senior Mining Engineer of SRK Consulting (U.S.), Inc., 7175 W. Jefferson Ave, Suite 3000, Denver, CO, USA, 80235.
2. This certificate applies to the technical report titled "Twin Metals Minnesota Project, Ely, Minnesota, USA, NI 43-101 Technical Report on Pre-feasibility Study" that has an effective date of 20 August 2014 (the "Technical Report").
3. I graduated with a degree in Mining Engineering from Colorado School of Mines in 2003. I am a QP member of the Mining & Metallurgical Society of America. I have worked as a Mining Engineer for a total of 11 years since my graduation from university. My relevant experience includes open pit and underground design, mine scheduling, pit optimization and truck productivity analysis.
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 visited the property.
6. I am responsible for or co-responsible for 1.15, 1.16, 1.17.3, 1.17.4, 1.17.5, 1.23.1, 1.23.2, 1.26 and 1.27; Section 2.2; Section 3; Sections 15.1, and 15.4 to 15.11; Sections 16.1.12.2, 16.3.1 to 16.3.4; 16.3.6; 16.3.8; 16.3.9; 16.4, 16.5, 16.7, and 16.8; Sections 21.3.2.1 to 21.3.2.5; Sections 24.1.3, and 24.2.4, Sections 25.6, 25.9, 25.10, and 25.19; Section 26.2.7 and Section 27 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 been involved with the Project during the preparation of the pre-feasibility study and this Technical Report.
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 6th Day of October, 2014.

"Signed" *"Stamped"*

____

Joanna Poeck, B.Eng., SME-RM, MMSA-QP

IMPORTANT NOTICE

This report was prepared as National Instrument 43-101 Technical Report for Duluth Metals Limited (Duluth) by AMEC E&C Services Inc, Barr Engineering Co.., Blue Coast Metallurgy Ltd., Golder Associates Inc., Itasca Consulting Group, Inc., Itasca Denver, Inc., and SRK Consulting (US) Inc. (SRK), collectively the “Report Authors”. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors’ 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 Duluth subject to terms and conditions of its respective contracts with each of the Report Authors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other use of, or reliance on, this report by any third party is at that party’s sole risk.

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1.0 SUMMARY

1.1 Principal Outcomes

Table 1-1: Study Outcomes

Production Statistics						
Metal Price	**Units**	**Year 1**	**Year 2**	**Year 3**	**Avg. Y1–10**	**LOM**
Copper	US$/lb	3.50	3.50	3.50	3.50	3.50
Nickel	US$/lb	9.50	9.50	9.50	9.50	9.50
Gold	US$/oz	1,300	1,300	1,300	1,300	1,300
Palladium	US$/oz	815	815	815	815	815
Platinum	US$/oz	1,680	1,680	1,680	1,680	1,680
Silver	US$/oz	21.50	21.50	21.50	21.50	21.50
Copper	klbs	208,046	241,910	248,490	230,315	5,826,868
Cu Concentrate	klbs	188,870	220,885	226,893	210,188	5,332,942
Ni Concentrate	klbs	19,176	21,025	21,597	20,127	493,926
Nickel	klbs	39,669	53,333	55,692	50,771	1,235,014
Cu Concentrate	klbs	4,899	5,643	5,789	5,404	133,670
Ni Concentrate	klbs	34,770	47,690	49,903	45,367	1,101,345
Gold	koz	29.1	33.1	34.7	36.4	1,011
Cu Concentrate	koz	23.9	27.5	28.8	30.2	841
Ni Concentrate	koz	5.2	5.6	5.9	6.2	171
Palladium	koz	111.5	125.4	127.6	138.4	4,022
Cu Concentrate	koz	56.5	65.2	66.4	71.8	2,099
Ni Concentrate	koz	54.9	60.2	61.3	66.6	1,923
Platinum	koz	39.6	44.5	46.1	51.2	1,493
Cu Concentrate	koz	14.6	16.9	17.5	19.4	571
Ni Concentrate	koz	25.1	27.6	28.6	31.8	922
Silver	koz	890	1,023	1,047	994	25,230
Cu Concentrate	koz	740	857	877	833	21,218
Ni Concentrate	koz	150	165	169	161	4,012
Cash Flow Statistics						
Metal Revenue	**Units**	**Year 1**	**Year 2**	**Year 3**	**Avg. Y1–10**	**LOM**
Total Revenue	000 US$	1,031,373	1,253,084	1,295,059	1,211,109	30,698,594
Operating Costs						
On Site Costs	000 US$	332,645	369,105	352,908	351,007	11,450,323
Off Site Costs	000 US$	157,851	195,385	200,905	187,697	4,658,849
Royalties	000 US$	36,550	54,051	63,177	53,507	1,265,699
Operating profit	000 US$	504,328	634,542	678,069	618,898	13,323,723
Taxes, Capex and Working Capital						
Taxes	000 US$	18,094	29,280	34,080	72,307	1,910,283
Capex	000 US$	207,322	139,409	125,387	137,744	5,410,489
Changes in Working Capital	000 US$	(183,174)	(33,588)	(7,785)	(20,938)	(0)
Cash Flow						
Cash Flow Pre Tax	000 US$	113,832	461,546	544,897	460,216	7,913,233
Cash Flow After Tax	000 US$	95,738	432,266	510,817	387,909	6,002,950
Operation Statistics						
	Units	**Year 1**	**Year 2**	**Year 3**	**Avg. Y1–10**	**LOM**
Metal Equivalent						
Copper payable (Cu revenue)	klbs	189,966	221,252	227,271	210,616	5,331,701
Nickel payable (Ni revenue)	klbs	29,013	39,794	41,640	37,855	918,993
Copper equivalent (Cu + Ni revenue)	klbs	268,717	329,264	340,293	313,366	7,826,110
Copper equivalent (all metals revenues) *****	klbs	294,678	358,024	370,017	346,031	8,771,027
Operating Costs and Profit margins per lbs of Cu						
Copper price	US$/lb	3.50	3.50	3.50	3.50	3.50
C1 costs / lb Cu **	US$/lb	0.65	0.39	0.24	0.31	0.76
Operating Margin / lb Cu	US$/lb	2.85	3.11	3.26	3.19	2.74
Operating Costs & Profit Margins per lbs of CuEq						
Copper price	US$/lb	3.50	3.50	3.50	3.50	3.50
C1 costs / lb CuEq ***	US$/lb	1.49	1.41	1.32	1.36	1.64
Operating Margin / lb CuEq	US$/lb	2.01	2.09	2.18	2.14	1.86

	Units	Year 1	Year 2	Year 3	Avg. Y1–10	LOM
Operating costs and Profit margins per dst milled						
Revenue / dst milled	US$/dst	62.79	68.66	70.96	66.99	58.27
Operating cost / dst milled	US$/dst	29.86	30.93	30.35	29.82	30.58
Operating Margin / dst milled	US$/dst	32.93	37.73	40.62	37.17	27.69

Notes: ***** metal revenues do not include any payments for nickel and PGMs contained in the copper concentrate; please see Section 19 of the report for the discussion on payabilities for the concentrates; ** C1 Cu cost = (onsite costs + offsite cost – royalties – revenue from (Ni, Au, Ag, Pt, Pd))/ (Cu revenue/Cu price), where the units are US$/lbs of Cu; *** C1 CuEq cost = (onsite costs + offsite cost – royalties – revenue from (Au, Ag, Pt, Pd))/ ((Cu revenue/Cu Price)+(Ni revenue/Cu price)) where the units are US$/lbs of CuEq; dst = dry short ton; Avg = average; LOM = life-of-mine..

1.2 Introduction

AMEC E & C Services Inc. (AMEC) was commissioned by Duluth Metals Limited (Duluth) to compile an independent NI 43-101 Technical Report (the Report) for the Twin Metals Minnesota Project (the Project) located near Ely Minnesota, USA.

The firms and consultants who are responsible for the content of this Report, which is based on a prefeasibility study completed in 2014 (the PFS) and supporting documents prepared for the PFS, are, in alphabetical order, AMEC, Barr Engineering Co. (Barr), Blue Coast Metallurgy Ltd. (Blue Coast), Golder Associates Inc. (Golder), Itasca Consulting Group, Inc., Itasca Denver, Inc. (collectively Itasca), and SRK Consulting (US) Inc. (SRK).

Some of the work preparation for the PFS was completed by two third-party consulting firms, which are unable to be identified due to the terms of their respective contract agreements with TMM, and are referred to in the Report as “TMM’s Independent Engineer” and “TMM’s Environmental Consultant”, respectively.

The Report will be used in support of Duluth’s press release dated 20 August 2014, that is entitled “Duluth Metals Highlights Low Copper (C1) Cash Costs and Strong Operating Margins in its Pre-feasibility Study for Twin Metals Minnesota Project”. The report was amended 6 October 2014 as Table 22-4 had not been reproduced properly during the conversion to pdf.

Currency is expressed in US dollars unless stated otherwise; units presented are typically US standard units, such as short tons, unless otherwise noted.

1.3 Cautionary Notes

1.3.1 Caution Regarding Forward-Looking Information

Section 1.3 of this Report applies to forward-looking statements throughout the Report.

Certain information and statements contained in this Report are “forward looking” in nature. Forward-looking statements include, but are not limited to, statements with respect to the economic and prefeasibility parameters of the Project; Mineral Resource and Mineral Reserve estimates; the cost and timing of the development of the Project; the proposed mine plan and mining methods; dilution and mining recoveries;

processing method and rates and production rates; projected metallurgical recovery rates; infrastructure requirements; capital, operating and sustaining cost estimates; the projected life of mine and other expected attributes of the Project; the net present value (NPV) and internal rate of return (IRR) and payback period of capital; capital; future metal prices; the Project location in proximity to the Boundary Waters Canoe Area Wilderness; the timing of the environmental assessment process; changes to the Project configuration that may be requested as a result of stakeholder or government input to the environmental assessment process; government regulations and permitting timelines; estimates of reclamation obligations; requirements for additional capital; environmental risks; and general business and economic conditions.

All forward-looking statements in this Report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled or predicted. Material assumptions regarding forward-looking statements are discussed in this Report, where applicable. In addition to, and subject to, such specific assumptions discussed in more detail elsewhere in this Report, the forward-looking statements in this Report are subject to the following assumptions:

- There being no signification disruptions affecting the development and operation of the Project
- Exchange rate assumptions being approximately consistent with the assumptions in the Report
- The availability of certain consumables and services and the prices for power and other key supplies being approximately consistent with assumptions in the Report
- Labor and materials costs being approximately consistent with assumptions in the Report
- Permitting and arrangements with stakeholders being consistent with current expectations as outlined in the Report
- All environmental approvals, required permits, licenses and authorizations will be obtained from the relevant governments and other relevant stakeholders within the expected timelines indicated in the Report
- Certain tax rates, including the allocation of certain tax attributes, being applicable to the Project
- The availability of financing for Duluth and TMM's development activities
- The timelines for exploration and development activities on the Project
- Assumptions made in Mineral Resource and Mineral Reserve estimates, including, but not limited to, geological interpretation, grades, metal price assumptions,

metallurgical and mining recovery rates, geomechanical and hydrogeological assumptions, operating cost estimates, the premises made in regards to the modifying factors considered when converting Mineral Resources to Mineral Reserves, and general marketing, political, business and economic conditions.

Forward-looking statements involve known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements to be materially different from any of the future results, performance or achievements expressed or implied by forward-looking statements. These risks, uncertainties and other factors include, but are not limited to: the assumptions underlying the PFS and this Report and economic parameters discussed in this Report not being realized; decrease of future metal prices; cost of labor, supplies, power, fuel and equipment rising; actual results of current exploration; adverse changes in Project assumptions and parameters; discrepancies between actual and estimated production, statements related to “reserves” and “resources” involve the implied assessment, based on realistically assumed and justifiable technical and economic conditions, that an inventory of mineralization will become economically extractable; metallurgical and mining recoveries; exchange rate fluctuations; delays and costs inherent in consulting with, and accommodating, stakeholder inputs and rights; title risks; regulatory risks and political or economic developments in Minnesota and the U.S. generally; changes to taxation and royalty rates; risks and uncertainties with respect to obtaining necessary surface rights and permits or delays in obtaining same; risks associated with maintaining and renewing permits and complying with permitting requirements; failure of plant, equipment or processes to operate as anticipated; accidents, labor disputes and other risks in the base metals and precious metals exploration and development industry; as well as those risk factors discussed elsewhere in this Report.

1.3.2 Cautionary Note to U.S. Readers Concerning Estimates of Mineral Reserves and Mineral Resources

Information concerning the Project has been prepared in accordance with Canadian standards under applicable Canadian securities laws, and may not be comparable to similar information for United States companies. The terms “Mineral Resource”, “Measured Mineral Resource”, “Indicated Mineral Resource” and “Inferred Mineral Resource” used in this Report are Canadian mining terms as defined in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves adopted by CIM Council on May 10, 2014 and incorporated by reference in National Instrument 43-101 (NI 43-101). While the terms “Mineral Resource”, “Measured Mineral Resource”, “Indicated Mineral Resource” and “Inferred Mineral Resource” are recognized and required by Canadian securities regulations, they are not defined terms under standards of the United States Securities and Exchange Commission. As such, certain information contained in this Report concerning descriptions of mineralization and resources under Canadian standards is not comparable to similar information made public by United States companies subject to the reporting and disclosure requirements of the United States Securities and Exchange Commission.

An "Inferred Mineral Resource" refers to a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Interred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. Readers are cautioned not to assume that all or any part of an "Inferred Mineral Resource" exists or is economically or legally mineable. Under Canadian securities legislation, estimates of an "Inferred Mineral Resource" may not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed prefeasibility or feasibility studies, or in the life of mine plans and cash flow models of developed mines.

Under United States standards, mineralization may not be classified as a "Reserve" unless the determination has been made that the mineralization could be economically and legally produced or extracted at the time the Reserve estimation is made. Readers are cautioned not to assume that all or any part of the Measured or Indicated Mineral Resources that are not Mineral Reserves will ever be converted into Mineral Reserves. In addition, the definitions of "Proven Mineral Reserves" and "Probable Mineral Reserves" under CIM standards differ in certain respects from the standards of the United States Securities and Exchange Commission.

1.3.3 Cautionary Note Regarding Production Dates

The production schedules and financial analysis annualized cashflow table are presented with calendar dates shown. Calendar years shown in these tables are for illustrative purposes only. Additional mining, technical, and engineering studies are planned which may alter the Project assumptions as discussed in the PFS and this Report, and may result in changes to the calendar timelines presented. No development approval has been forthcoming from the Duluth or TMM Boards, and statutory permits are required to be granted prior to mine commencement.

1.4 Ownership

Twin Metals Minnesota LLC (TMM) is a limited liability company that, since 2010, has been operated as a joint venture between Antofagasta PLC (Antofagasta) and Duluth, under a Participation and Limited Liability Company Agreement (the Participation Agreement). TMM is 35% owned by Duluth Metals Holdings (USA) Inc. (which is indirectly held by Duluth), 25% owned by Twin Metals (USA) Inc (which is indirectly owned by Duluth) and 40% owned by Northern Minerals Holding Co. (which is indirectly owned by Antofagasta). Accordingly, Duluth holds, directly or indirectly, a 60% controlling interest in TMM and is currently the Project operator. For the purposes of this Report TMM and Duluth are used interchangeably.

1.5 Project Setting

From the city of Duluth, the Project can be accessed by US Highway 53 north for 64 miles (mi) to its junction with State Highway 169 north of the town of Virginia, thence 42 mi northeast on State Highway 169 to the town of Ely. From the town of Ely, the Project can be reached by taking State Highway 1 south, which crosses the South Kawishiwi River just north of the Project, a distance of 12 mi.

The northern Minnesota climate is mid-continental. Exploration operations continue year-around with much of the drilling completed in the winter months to minimize surface disturbances. Future mining activities could be conducted on a year-round basis.

The Project is located at the eastern end of the Mesabi Iron Range, a major center for iron ore mining for over 100 years. The region currently has eight large operating taconite mines and associated process plants, and two future operations are in development. As a result of mining activity, an extensive network of railroads and paved roads has developed throughout the region that today provides excellent transport communications. Duluth has access to a large pool of skilled and unskilled labor in the region, and the engineering and technical resources supporting the iron ore mining operations.

Elevations on the Project range from 1,425 ft to 1,550 ft above mean sea level (amsl). Topographic relief is generally low and controlled by bedrock exposures.

1.6 Mineral Tenure, Surface Rights, and Royalties

For mineral tenure and surface land positions, TMM holds permits, leases, options to purchase, fee title, and fee title to limited mineral rights, to about 27,000 acres of mineral rights across a patchwork of federal, state, and private mineral interests. The federal and state mineral rights are administered by the Bureau of Land Management (BLM) and the Minnesota Department of Natural Resources (DNR), respectively, and the US Forest Service (USFS) administers federal surface that overlies both federal and state minerals (i.e., split estates). AMEC was provided with legal opinion that supports Duluth's interpretation that the mining tenure held is valid and is sufficient to support declaration of Mineral Resources and Mineral Reserves. Additional surface lands will need to be acquired to allow the Project infrastructure as envisaged in this Report to be constructed.

Duluth has currently identified 11 unique royalty combination schemes within the proposed mine plan area boundaries that govern the royalties that will be payable to federal, state, and private parties.

1.7 History

Exploration and development activities on the Project prior to Duluth's interest have been conducted by a number of mining and exploration companies, including Inco, American Copper and Nickel Company Inc (ACNC), Hanna Mining Company, BHP Utah, Duval Corp., Bear Creek Mining Company, Beaver Bay Joint Venture, Franconia Minerals Corporation Inc. (Franconia), Lehmann Exploration Management, Wallbridge Mining Company Limited, and International Platinum Company Inc.

Work completed during the period 1954 to date has included geophysical surveys, core drilling, shaft sinking, bulk sampling, metallurgical testwork, technical and engineering studies, and environmental baseline studies.

1.8 Geology and Mineralization

Mineralization at the Maturi, Maturi Southwest, Birch Lake, and Spruce Road deposits is hosted by the Duluth Complex, a composite intrusion comprising 12 sub-intrusions emplaced over a period of 10 to 12 million years starting about 1,108 Ma. The basal portion of the South Kawishiwi intrusion (SKI) hosts all four deposits in what is locally known as the basal mineralized zone (BMZ) that is locally more than 1,000 ft thick.

The BMZ at Maturi and Maturi Southwest has been subdivided into four stratigraphic units based on geochemical and geological similarities. The apparent order of intrusion, from oldest to youngest, is: the Upper Heterogeneous (UH), Stage 1 (S1), Stage 2 (S2), and finally, Stage 3 (S3).

Mineralization primarily comprises chalcopyrite, cubanite, pentlandite, pyrrhotite, and talnakhite with numerous base and precious metals-bearing trace minerals. These minerals are disseminated within the BMZ.

The geological setting is sufficiently well known to support Mineral Resource and Mineral Reserve estimation and preliminary mine planning.

1.9 Drilling

Drilling has primarily consisted of core methods, due to the depth to mineralization. To 4 February 2014, a total of 1,339 pilot and wedge holes have been completed on the Project for a total of approximately 2,083,027 ft. Of this total, 765 holes (1,523,181 ft) were drilled at Maturi, and 71 holes (69,918 ft) at Maturi Southwest. Birch Lake includes 243 current and 26 legacy holes (347,631 ft) and Spruce Road has two current and 232 legacy holes (141,482.7 ft). Drilling is considered legacy if it was completed before 2000 at Maturi, Maturi Southwest, and Birch Lake, and prior to 1999 at Spruce Road.

Core logging at all four deposits is considered to be adequate to support resource estimation and preliminary mine planning.

Current collar surveying at Maturi, Maturi Southwest and Birch Lake utilizes industry-standard instrumentation and procedures and is adequate to support resource estimation and preliminary mine planning. Collar surveying at Spruce Road is believed to have been performed with theodolites and chains, which was industry-standard practice at the time the holes were drilled, but that has not been confirmed.

Legacy (pre-TMM) downhole surveying was done primarily with acid-tubes which do not provide adequate control on the azimuth of drill holes; AMEC has restricted blocks which are informed predominantly by legacy data to the Inferred category. AMEC determined the influence of legacy holes on each block. Where the influence was more than 25%, Measured blocks were downgraded to Indicated. Where the influence was more than 50%, Indicated blocks were downgraded to Inferred. Current practice is to use gyroscopic tools that are unaffected by magnetic minerals in the rocks. These tools are widely used in the industry and provide orientation data that are adequate to support resource estimation and preliminary mine planning at all confidence levels.

1.10 Sampling and Analysis

Current core sampling conforms to industry-standard practices and is adequate to support resource estimation and preliminary mine planning.

Density determinations at Maturi, Maturi Southwest, and Birch Lake were performed using standard procedures and are adequate to support resource estimation and preliminary mine planning. No density determinations have been performed at Spruce Road.

Recent sample preparation and assaying was performed at accredited commercial laboratories. Legacy samples were prepared and analyzed at a number of commercial and at least one company laboratory.

Legacy sample preparation by ACNC is not documented; however, AMEC considers that it is reasonable to consider sample preparation procedures as adequate, largely because ACNC's parent company, Inco, was an industry leader in Cu–Ni mining at the time the samples were collected and analyzed. Sample preparation for recent exploration programs completed by Franconia, Duluth, and TMM has been performed using standard procedures and is adequate to support resource estimation and preliminary mine planning.

Analytical procedures used for legacy ACNC samples is not documented, but is believed to be adequate to support resource estimation. Analytical procedures employed by Franconia, Duluth, and TMM are industry-standard procedures and are adequate to support resource estimation and preliminary mine planning.

Quality assurance and quality control (QA/QC) for legacy samples is not documented. QA/QC for current samples is considered by AMEC to be adequate to support resource estimation and preliminary mine planning.

Sample security for legacy samples is not documented. Sample security for modern samples is considered to be sufficient to support Mineral Resource and Mineral Reserve estimation and preliminary mine planning.

1.11 Data Verification

The combined Maturi, Maturi Southwest, and Birch Lake database is adequate to support estimation of Mineral Resources without restriction. AMEC considers that the Spruce Road database is adequate to support estimation of only Inferred Mineral Resources because the data are largely unverifiable.

1.12 Metallurgical Testwork

The mineral processing and metallurgical information for the PFS has been derived from testwork conducted on a variety of samples acquired during drilling campaigns conducted between 2010 and 2012. The majority of the mineral processing testwork to develop the final processing flowsheet and conditions for the PFS was performed between 2012 and 2014 at ALS Metallurgy (ALS), and Blue Coast, both in Canada. Significant supporting testwork and analysis was also conducted by a number of third parties.

Sampling of the deposit created:

- Variability samples. Designed to provide broad spatial coverage of the major geological units S2 and S3 from throughout the Maturi deposit. A variability sample was a single 10–15 ft continuous intercept from a single drill hole
- End-member samples. Designed to represent an enrichment of individual domains, these were used for grindability testing. Each 300 kg composite was from 100 ft intervals from either multiple holes or multiple wedges from a single drill hole
- Pilot plant composites: Three composites were created from multiple holes mostly located on the west side of the Maturi deposit, and likely representing mineralization that would be mined early in the mine life. PP-3, the largest composite, was used for piloting for the PFS.

In addition, sub-domain composites (SDCs) were prepared using material from the variability and end-member samples. These were based on the most up-to-date information available on the mine plan in December 2013, and were designed to represent the two main S2 and S3 geological units from the Shallow, Deep and Deep East mineralogical zones, with variable ratios of pyrrhotite to pentlandite.

Finally, life-of-mine composites were prepared to, where possible, represent the source location, S2/S3 mix and type of mineralogy expected over the life of mine.

Each of these samples/composites were subjected to a different permutation of mineralogy, grindability, batch flotation, locked-cycle testing and pilot plant testing as indicated in Table 1-2.

Copper mineralization is dominated by chalcopyrite and cubanite, an iron-rich, copper-poor sulfide, and the ratio of abundance between the two minerals in any given sample drives the ultimate copper concentrate grade. Nickel mineralogy is substantially more complex: while pentlandite is the primary host of nickel (78% of the nickel in the 212 samples analyzed to date), variable amounts are contained in olivine (average 15%), pyroxenes (>2%) chalcopyrite and mica (>1%). Pyrrhotite hosts very little nickel (average 0.5%). Both the copper sulfides and pentlandite are relatively coarse-grained and are well liberated (typically ~80%) at a grind k80 of 120 µm.

The PGM and gold mineralogy is dominated by discrete minerals with arsenides, bismuthinides and gold/silver alloys being the main hosts of platinum, palladium and gold respectively.

The mean grindability characteristics of up to 130 samples tested using a variability of tests, are shown in Table 1-3. Maturi Southwest tends to be slightly harder than Maturi, the two samples tested to date averaging (Bond ball mill work index) 13.9 kWh/t, (rod mill work index) 12.6 kWh/t, and A x b at 47.9.

Flotation testwork to develop a process capable of producing saleable copper and nickel concentrates was effectively started at ALS in late 2012. This early bench-scale work developed a crude procedure that, in batch tests, yielded copper concentrates assaying roughly 23% copper and 0.4–0.6% nickel, with copper recoveries to the copper concentrate in the 70–80% range. These relatively poor copper recoveries limited the quality of the concentrates produced in the nickel circuit, with the copper/nickel ratio being close to 1:1.

Table 1-2: Characterization and Testwork Conducted on the Different Metallurgical Composites

Composite type	Pilot Plant (PP-3) Composite	End Member Samples	Life of Mine Composites	Sub-Domain Composites	Variability
Number of samples	**1**	**19**	**4**	**16**	**115**
QEMSCAN mineralogy	yes	yes	yes**	yes**	yes
Grindability testing		yes			yes
Flowsheet development	yes				
Batch Rougher Tests	yes	yes	yes	yes	yes
Locked-Cycle Tests	yes	yes*	yes	yes	
Pilot Plant	yes				

Notes: *: using ALS flowsheet; **: calculated from other analyses

Table 1-3: Average Grindability Characteristics of Maturi Mineralization

Test	BWI (kWh/t)	AI (g)	CWI (kWh/t)	Relative Density	RWI (kWh/t)	JK Parameters DWi(kWh/m^3)	A x b	CEET Ci	SPI (Min)
Maturi mean	12.9	0.149	16.4	3.1	10.8	5.1	64.4	8.3	57.3

Note: tonnage units are metric tonnes.

The process involved a primary P80 grind of 140–150 µm, with a combination of 100 g/t sodium sulfite and 25 g/t triethylenetetramine (TETA) added as nickel depressants to the primary mill. The copper was floated with starvation doses of the phosphine-based Cytec Aerofloat 3418A to a rougher concentrate, which was reground with more nickel depressant and cleaned again using starvation doses of 3418A. The copper rougher and cleaner tails were combined into the nickel circuit, where all other sulfides were floated using larger doses of potassium amyl xanthate. The best of three locked-cycle tests on the PP-3 composite yielded a 23% copper concentrate (with 0.63% nickel) at 87% copper recovery, and a 9.0% Ni concentrate (with 3.3% Cu) at 56% recovery.

This process was then tested in locked-cycle mode using the end member composites, but with less success. The locked-cycle tests mostly succeeded in generating saleable copper concentrates from most of the samples, with the nickel content in the copper concentrates being consistently well under control (ranging from 0.4–0.9%), but copper grades ranged from 17–27% with several of the samples yielding grades below 20% copper. Nickel flotation was poor, with only two of the 17 nickel concentrates assaying over 9% nickel, and the assay ratio of nickel to copper averaged about 1.5:1.

It was assumed at the time that these products would be unattractive to smelters therefore this led to a follow-up program at Blue Coast, which started in October 2013. This program followed a more structured approach to optimizing the existing flowsheet, starting with the copper circuit then moving to the nickel circuit. Most of the focus was on the copper circuit, time constraints limiting the opportunity for extensive optimization of the nickel circuit where opportunities for potential further enhancements exist.

Overall, the direction was towards the use of smaller reagent doses (less depressants and 3418A in the copper circuit and less of a shorter chain xanthate in the nickel circuit), at a pH optimized at 10.8 in the copper circuit and 10.0 in the nickel circuit. The primary grind was established at 80% passing 120 µm. Other changes included the adoption of inert regrinding media and the redirection of the copper cleaner tails to the nickel regrind.

The only major change in flotation chemistry was applied to pyrrhotite-rich samples, which, due to the high levels of pyrrhotite in the concentrate, were creating a poor quality product. The use of sodium sulfite and TETA, this time added in a way to depress pyrrhotite in the nickel cleaner stages, allowed for the creation of high-grade nickel concentrates, even from the most pyrrhotite-rich samples.

This flowsheet was tested in locked-cycle mode using the PP-3 composite and on the Sub-domain and life-of-mine composites (which had been designed to reflect the latest resource model thinking and the mine plan in effect in December 2013). Results from 19 locked-cycle tests using the basecase flowsheet, and six tests using the pyrrhotite rejection flowsheet are illustrated graphically in Figure 1-1.

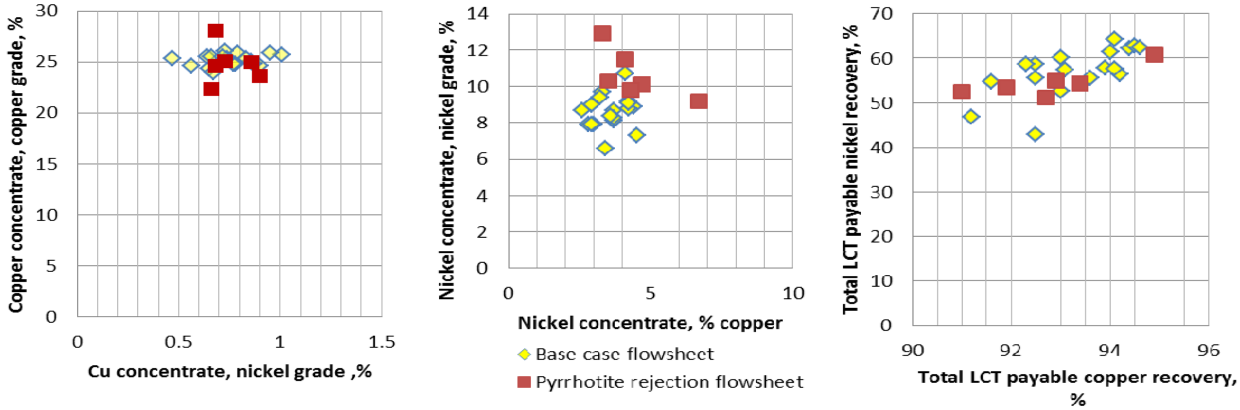
The mean copper recovery from tests using the basecase flowsheet was 85%, to a copper concentrate assaying on average 25% copper and 0.75% nickel. All tests yielded copper concentrates assaying above 24% copper, and only one test yielded a nickel grade in the copper concentrate above 1%—at 1.01%. The nickel metallurgy was also consistently better. On average the nickel circuit yielded a nickel product assaying 8.6% nickel, and 3.8% copper, at 57% nickel recovery. The mean overall copper recovery was 93.3%.

Combined precious metal recoveries to the combined concentrates averaged 78%, 61% and 74% for gold, platinum and palladium respectively.

The pyrrhotite rejection flowsheet also tended to perform well. The copper circuit, unchanged from the basecase circuit, again yielded a 25% copper concentrate at 85% copper recovery from the seven tests, but now the mean nickel concentrate grade was 10.6%, achieved at, on average, a slightly lower 54.5% recovery. Precious metal recoveries were somewhat lower at 74%, 44% and 64%.

Scale-up to continuous mode appears to favor good metallurgy. Pilot plant testwork conducted at the time of the ALS laboratory study, using a developmental ALS procedure, yielded better metallurgy than prevailed in the laboratory at the time.

Figure 1-1: Key Locked-cycle Metallurgical Testwork Results



The seven pilot plant runs, operated under what would be considered optimal conditions at the time, yielded a mean copper recovery of 83% and a mean nickel recovery of 5% to the copper concentrate, with the copper concentrate assaying 25.5% copper and 0.6% nickel. The gold, platinum and palladium recoveries to the copper concentrate were 68%, 22% and 43% to grade 2.4 g/t, 1.3 g/t and 6.2 g/t respectively. The nickel circuit performed substantially better, yielding concentrates assaying 11.1% nickel and 4.4% copper, at nickel and copper recoveries of 60% and 10% respectively. The gold, platinum and palladium recoveries were 12%, 36% and 33% to grade 0.7 g/t, 3.4 g/t and 7.5 g/t respectively.

Rougher variability flotation testwork was conducted on some 94 variability samples in the Maturi deposit. The results describe a picture of very consistent copper rougher recoveries. The recoveries to the combined concentrates averaged 96.5% from the S3 samples and 96.0% from the S2 samples, with a standard deviation in both cases of roughly 1%. Nickel recoveries varied more widely, however, mainly driven by the content of nickel in non-sulfide form in the sample. The nickel rougher recoveries, to both concentrates averaged 73% for S3 and 70% for S2, both with a standard deviation of 9%.

Metallurgical forecasting is based on geometallurgical algorithms using the rougher database to link copper and nickel rougher recoveries to parameters in the resource model. The locked-cycle data are then used to predict how the recovered metal is distributed to the two final concentrates and the cleaner tails.

As the pilot plant tended to yield cleaner copper and nickel concentrates than the respective locked-cycle tests, often at equal or better recoveries, the forecast also assumes some degree of cleaner performance enhancement in continuous mode over the results achieved in the Blue Coast locked-cycle program.

1.13 Mineral Resource Estimation

Mineral Resources have been estimated using ordinary kriging (OK) for the Maturi, Maturi Southwest, Birch Lake, and Spruce Road Cu–Ni–PGE deposits. These resources are estimated assuming underground mining as the preferred option. The Mineral Resource estimate for Spruce Road is a re-tabulation of a 2007 resource estimate produced by Scott Wilson Roscoe Postle Associates Inc. (SWRPA).

The Mineral Resource estimates for Maturi and Maturi Southwest are based on an approximate $22/st net smelter return (NSR) that in turn assumes a mining cost of $13.00/st, a process cost of $6.00/st and general and administrative charges of $3.00/st; global metallurgical recoveries of 93.4% (Cu), 63.9% (Ni), 78.2% (Au), 76.2% (Pd), 61.3% (Pt) and 66.9% (Ag); and long-term consensus metal prices of $3.30/lb Cu, $10.0/lb Ni, $1,350/troy oz Au, $850/troy oz Pd, $2,000/troy oz Pt, and $21.00/troy oz Ag. The $22/st NSR equates to an approximate 0.3% Cu cutoff grade. Maturi tabulations assume a 400 ft thick safety pillar above the Mineral Resource; tabulations at Maturi Southwest are based on a 15 ft allowance for overburden and no safety pillar.

The Mineral Resource estimates for Birch Lake are based on a US$30/st NSR that in turn assumes a mining cost of $16/st, a process cost of $12/st and general and administrative charges of $2/st; global metallurgical recoveries of 90.8% (Cu), 57.4% (Ni), 63.3% (Au), 63.6% (Pd) and 55.2% (Pt); and long-term consensus metal prices of $3.00/lb Cu, $9.38/lb Ni, $1,050/troy oz Au, $805/troy oz Pd and $1,840/troy oz Pt. The NSR equates to an approximate 0.3% Cu cutoff grade. At Birch Lake, the mineral resources are located at least 600 ft below the surface. AMEC considers that depth sufficient to not require additional allowances for a safety pillar.

The Mineral Resource estimates for Spruce Road are based on a US$30/st NSR that in turn assumes a mining cost of $16/st, a process cost of $12/st and general and administrative charges of $2/st; global metallurgical recoveries of 90.8% (Cu), 68.8% (Ni); and long-term consensus metal prices of $3.00/lb Cu, and $9.38/lb Ni. The NSR equates to an approximate 0.3% Cu cutoff grade. AMEC assumed a 164 ft safety pillar.

1.14 Mineral Resource Statement

The Maturi, Maturi Southwest, and Birch Lake Mineral Resource estimates were prepared under the supervision of Dr. Harry Parker, AMEC Consulting Geologist and Geostatistician. All three estimates used Vulcan software and OK interpolation. SWRPA produced a resource estimate for the Spruce Road deposit in 2007 for Franconia. AMEC reviewed and accepted the SWRPA model and recast the resource estimate based on underground mining assumptions.

Mineral Resources have been classified using the 2014 CIM Definition Standards as incorporated by reference in NI 43-101. Mineral Resources are reported in million short tons (Mst) in Table 1-4. Mineral Resources are reported inclusive of Mineral Reserves and are reported on a 100% basis. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

1.15 Mineral Reserve Estimation

The PFS assumes that the Maturi and Maturi Southwest deposits will be mined. The PFS does not consider mining the Spruce Road and Birch Lake deposits.

Measured and Indicated Mineral Resources for the Maturi and Maturi Southwest deposits were converted to Mineral Reserves by applying appropriate mining dilution and recovery factors to the triangulations that were created during the mine design stage. While some triangulations consist entirely of Measured and Indicated Mineral Resources, other triangulations may include small amounts of Inferred Mineral Resources and unclassified material. Where Inferred and unclassified material has been included in a triangulation, such material has been assigned a grade of zero. Where appropriate, a "development allowance" was applied to certain types of triangulations to account for re-muck bays, fan cut-outs, etc. In some cases this development allowance was in ore.

An NSR calculation was used that takes into account revenue for five elements (Cu, Ni, Au, Pd, and Pt). Plant recoveries assumed in the NSR equation were based on current testwork for concentrate production. The NSR was evaluated for each block in the block model. A US$25.00 NSR value was selected based on an estimated average LOM operating cost for the Project of US$23.53/st.

The Maturi and Maturi Southwest deposits are planned to be mined using a combination of two underground mining methods:

- Post-pillar cut-and-fill
- Long-hole stoping.

These mining methods were selected because they were able to produce at a high throughput rate and had the ability to be adjusted to the specific geometries (dip and thickness) of the deposits.

The deposits were subdivided into mining areas or tiers, based on depth below surface to address geometric characteristics and productivity opportunities. Ore dilution and mining recovery were calculated based on detailed designs of the mining areas and recommendations in regards to paste and hanging wall dilution.

Table 1-4: Mineral Resource Statement

Deposit	Category	Tons (Mst)	CuEq (%)	Cu (%)	Ni (%)	Pt (ppm)	Pd (ppm)	Au (ppm)	Ag (ppm)
Maturi	Measured	308	1.02	0.63	0.20	0.146	0.339	0.083	2.26
	Indicated	822	0.96	0.58	0.19	0.155	0.350	0.083	2.10
	Inferred	531	0.81	0.49	0.16	0.138	0.314	0.070	1.81
Maturi Southwest	Indicated	103	0.77	0.48	0.17	0.080	0.185	0.048	1.58
	Inferred	32	0.70	0.43	0.15	0.065	0.157	0.041	1.43
Subtotal Maturi and Maturi Southwest	Measured	308	1.02	0.63	0.20	0.146	0.339	0.083	2.26
	Indicated	924	0.94	0.57	0.18	0.147	0.332	0.079	2.04
	Measured + Indicated	1,233	0.96	0.58	0.19	0.147	0.334	0.080	2.10
	Inferred	563	0.81	0.49	0.16	0.134	0.305	0.068	1.79
Birch Lake	Indicated	100	1.02	0.52	0.16	0.235	0.515	0.115	—
	Inferred	239	0.88	0.46	0.15	0.180	0.370	0.087	—
Spruce Road	Inferred	480	0.66	0.43	0.16	—	—	—	—

Deposit	Category	Contained Cu (B lb)	Contained Ni (B lb)	Contained Pt (M oz)	Contained Pd (M oz)	Contained Au (Moz)	Contained Ag (M oz)
Maturi	Measured	3.9	1.2	1.3	3.0	0.7	20.3
	Indicated	9.5	3.0	3.7	8.4	2.0	50.3
	Inferred	5.2	1.7	2.1	4.9	1.1	28.0
Maturi Southwest	Indicated	1.0	0.3	0.2	0.6	0.1	4.7
	Inferred	0.3	0.1	0.1	0.1	0.0	1.3
Subtotal Maturi and Maturi Southwest	Measured	3.9	1.2	1.3	3.0	0.7	20.3
	Indicated	10.5	3.4	4.0	8.9	2.1	55.1
	Measured + Indicated	14.3	4.6	5.3	12.0	2.9	75.4
	Inferred	5.5	1.8	2.2	5.0	1.1	29.4
Birch Lake	Indicated	1.0	0.3	0.7	1.5	0.3	—
	Inferred	2.2	0.7	1.3	2.6	0.6	—
Spruce Road	Inferred	4.1	1.5	—	—	—	—

Notes to Accompany Mineral Resource Table:

1. The Mineral Resource estimates have different effective dates as follows: Maturi: 4 February 2014; Maturi Southwest: 15 June 2013; Birch Lake: 15 September 2012; Spruce Road: 15 September 2012.
2. The Qualified Person for the estimates is Dr. Harry Parker, RM SME, AMEC Consulting Geologist and Geostatistician, who is a Professional Geologist licensed in Minnesota.
3. Mineral Resources are reported inclusive of Mineral Reserves and on a 100% basis. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
4. Mineral Resources were estimated assuming underground bulk mining methods and are reported at an approximate cutoff grade of 0.3% Cu.
5. Maturi and Maturi Southwest copper equivalent (CuEq) grades are based on the following assumptions: CuEq = Cu + 1.459*Ni + 0.265*Au + 0.101*Pd + 0.228*Pt + 0.004*Ag; where global metallurgical recoveries are 93.4% (Cu), 61.4% (Ni), 78.5% (Au), 74.9% (Pd), 63.2% (Pt), and 76.5% (Ag); smelter returns are 94.3% (Cu), 77.1% (Ni), 54.9% (Au), 35.0% (Pd), and 45.2% (Pt) and 47.6% (Ag), and long-term consensus metal prices are $3.50/lb Cu, $9.50/lb Ni, $1,300/troy oz Au, $815/troy oz Pd and $1,680/troy oz Pt and $21.50/troy oz Ag. The Birch Lake CuEq formula is based on November 2012 parameters: CuEq = Cu + 1.58*Ni + 0.285*Au + 0.219*Pd + 0.435*Pt, where concentrate metallurgical recoveries are 94.3% (Cu), 60.0% (Ni), 85.0% (Au), 90.0% (Pd), and 93.0% (Pt); CESL metallurgical recoveries are 96.3% (Cu), 95.6% (Ni), 74.5% (Au), 70.7% (Pd), and 59.4% (Pt); smelter returns are 100% (Cu), 80% (Ni), 80% (Au), 80% (Pd), and 80% (Pt); long-term consensus metal prices of $3.00/lb Cu, $9.38/lb Ni, $1,050/troy oz Au, $805/troy oz Pd and $1,840/troy oz Pt. The Spruce Road CuEq formula is based on the Maturi parameters, and restricted to Cu and Ni: CuEq = Cu + 1.459*Ni; where global metallurgical recoveries are 93.4% (Cu), 61.4% (Ni); smelter returns of 94.3% (Cu), 77.1% (Ni); long-term consensus metal prices of $3.50/lb Cu, and $9.50/lb Ni.
6. Silver was not included in the 2012 resource estimate for Birch Lake as QA/QC results had not been reviewed at the time of the estimate. Silver is not a contributor to either the NSR calculation or the CuEq equation for Birch Lake. Gold, platinum,

palladium and silver assays were not available to support estimation in the 2012 Spruce Road resource model. Gold, Ag, Pt and Pd do not contribute to either the NSR calculation or the CuEq equation for Spruce Road.

7. No allowances for mining recovery and external dilution have been applied. Mineral Resources for Maturi assume a 400 ft thick safety pillar above the Mineral Resource. Mineral Resources for Maturi Southwest are tabulated based on a 15 ft allowance for overburden and no safety pillar. Mineral Resources for Birch Lake do not have a safety pillar allowance since the mineralization is located 600 ft below ground surface. Mineral Resources at Spruce road assume a 164 ft thick safety pillar.
8. Tonnage figures are reported as million US short tons (st); grade figures as parts per million (ppm) or percent (%); contained copper and nickel are reported in billion pounds (B lb), contained platinum, palladium, gold and silver are reported in million troy ounces (M oz). Contained metal is reported as in situ metal content and does not include any adjustments for recoverability.
9. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content

1.16 Mineral Reserves Statement

Mineral Reserves have been classified using the 2014 CIM Definition Standards as incorporated by reference in NI 43-101. The QP for the estimate is Ms. Joanna Poeck, RM SME, of SRK. Mineral Reserves are as summarized in Table 1-5, and are reported on a 100% basis.

1.17 Proposed Mine Plan

1.17.1 Geomechanical Considerations

Geomechanical interpretations are based on:

- Intact rock strength (from point load and laboratory testing)
- Joint characterization (from geotechnical core logging)
- Joint orientation (from ATV logging)
- Fracture frequency (from exploration drill hole logging)
- Intact rock material constant m_i (derived from laboratory test results).

Horizontal in situ stresses are approximately two to 2.5 times the vertical stress. This stress regime is expected to lead to fairly significant shear stresses in the plane of the orebody.

Uniaxial compressive strength (UCSi) measurements have been conducted in the laboratory on 134 samples from Maturi. Typical UCSi values based on the 30th percentiles of large scale domains range from 124 to 181 MPa, and the 30th percentile geologic strength index (GSI) values range from 73 to 98. Point load testing in the Maturi Southwest deposit indicates that typical 30th percentile UCSi values range from 123 to 156 MPa, and 30th percentile GSI values range from 65 to 76 on a large scale domain basis.

Table 1-5: Mineral Reserves Statement

Deposit	Classification	Tons (Mst)	Cu (%)	Ni (%)	Pt ppm	Pd ppm	Au ppm	Ag ppm
Maturi	Proven	130	0.65	0.21	0.155	0.344	0.092	2.31
	Probable	354	0.59	0.19	0.158	0.371	0.096	2.16
	Combined Proven and Probable	*484*	*0.60*	*0.19*	*0.159*	*0.373*	*0.090*	*2.20*
Maturi Southwest	Proven	0	0.00	0.00	0.000	0.000	0.000	0.00
	Probable	43	0.48	0.17	0.069	0.206	0.034	1.61
	Combined Proven and Probable	*43*	*0.48*	*0.17*	*0.069*	*0.206*	*0.034*	*1.61*
Total Maturi and Maturi Southwest	Proven	130	0.65	0.21	0.155	0.344	0.092	2.31
	Probable	397	0.58	0.19	0.148	0.353	0.089	2.10
	Total Combined Proven and Probable	*527*	*0.59*	*0.19*	*0.154*	*0.350*	*0.090*	*2.15*

Area	Classification	Tons (Mst)	Contained Cu Blbs	Contained Ni Blbs	Contained Pt Moz	Contained Pd Moz	Contained Au Moz	Contained Ag Moz
Maturi	Proven	130	1.7	0.5	0.6	1.3	0.3	8.8
	Probable	354	4.2	1.3	1.6	3.8	1.0	22.3
	Combined Proven and Probable	*484*	*5.8*	*1.9*	*2.2*	*5.1*	*1.3*	*31.1*
Maturi Southwest	Proven	0	0.0	0.0	0.0	0.0	0.0	0.0
	Probable	43	0.4	0.1	0.1	0.3	0.0	2.0
	Combined Proven and Probable	*43*	*0.4*	*0.1*	*0.1*	*0.3*	*0.0*	*2.0*
Total Maturi and Maturi Southwest	Proven	130	1.7	0.5	0.6	1.3	0.3	8.8
	Probable	397	4.6	1.5	1.7	4.1	1.0	24.3
	Total Combined Proven and Probable	*527*	*6.2*	*2.0*	*2.4*	*5.4*	*1.3*	*33.1*

Notes to Accompany Mineral Reserves Table:

1. The Qualified Person for the Mineral Reserve estimate is Ms. Joanna Poeck, an employee of SRK Consulting (U.S.), Inc. Mineral Reserves have an effective date of 1 July, 2014 and are reported on a 100% basis.
2. Mineral Reserves are contained within mine designs based on Measured and Indicated Mineral Resources, and assume a mining rate of 50,000 st/d of ore over a 30 year mine life. Underground mining will utilize conventional post-pillar cut-and-fill and long-hole open stoping methods. Paste backfill will be employed. The mine plan includes the mining of remnant ore, which is ore that is above the marginal cutoff grade, but is left behind during the first pass mining of higher-grade material.
3. Mineral Reserves are contained within Measured and Indicated mine designs using the following net smelter return (NSR) calculation inputs. Recovery assumptions used in the calculations were 94.0% for Cu, 60.8% for Ni, 82.3% for Au, 36.1% for Pd and 42.5% for Pt. Payability assumptions were 76.4% for Cu, 70.8% for Ni, 45% for Au, 68.6% for Pd and 69.3% for Pt. Metal price assumptions were US$3.00/lb for Cu, US$9.50/lb for Ni, US$1,200/oz for Au, US$700/oz for Pd and US$1,650/oz for Pt. Operating cost assumptions used in the NSR equations total $23.53/st mined and include mining costs of $13.80/st, process costs of $5.02/st, paste backfill costs of $1.28/st, water management costs of $0.21/st, tailings costs of $0.06/st, general and administrative costs of $2.44/st; technical services costs of $0.45/st and financial assurance costs of $0.27/st.
4. Mineral Reserves are reported using an NSR cutoff of $US25.00/st.
5. Mineralization that was either not classified or assigned to the Inferred Mineral Resource category was set to waste within the above NSR cutoff mining shapes. Mine design incorporates geotechnical and hydrogeological considerations that take into account paste and hanging wall dilution. Dilution is allocated in the mine design based on the mining method, and ranges from 3–5%. Recovery of the planned mine design is assumed at 95%.
6. Tonnage figures are reported as million US short tons (st); grade figures as parts per million (ppm) or percent (%); contained copper and nickel are reported in billion pounds (B lb), contained platinum, palladium, gold and silver are reported in million troy ounces (M oz). Contained metal is reported as in situ metal content and does not include any adjustments for recoverability.
7. Rounding as required by reporting guidelines may result in apparent summation differences between US short tons, grade and contained metal content

The deposits will be divided into panel areas with regional pillars in-between. Pillar spacing will be 1,700 ft along strike and approximately every 1,700 ft along dip for a maximum hydraulic radius of 425 ft. Regional pillar sizes are 200 ft wide in the Tier 1 area and 250 ft in all other tiers.

Modeling results suggest a minimum crown pillar thickness of 400 ft for Maturi and 300 ft for Maturi Southwest.

Overall extraction ratios within the mining panels range from 67–81% depending on mining method, depth, and geomechanical considerations. Stope sizes for the post-pillar cut-and-fill method will range from 26 ft wide x 20 ft high to 46 ft wide x 40 ft high, and long-hole stopes will range in strike length from 100 ft to 150 ft.

1.17.2 Hydrogeological Considerations

Four hydrogeological investigations have been conducted for mine planning purposes since 2008 at the Maturi site. These hydrogeological models are not suitable for use for environmental planning purposes as they are restricted to the immediate mine footprint. Although a preliminary groundwater flow model for mining purposes was undertaken for Maturi Southwest, the assumption in that model was that mining would be by open pit methods, rather than the current underground mine plan. The model remains to be updated for underground mining.

Potential groundwater inflows to the underground mines and infrastructure will be governed by the permeability of the discontinuities within the rock (e.g. fractures and faults) along with the connections of these discontinuities to sources of recharge. Current information indicates that the intact rock is of very low permeability and thus, does not transmit much water. In addition, available data suggest that the discontinuities within the rock are not well connected, and therefore groundwater inflows into the mine are likely to be low. Additional field investigations and groundwater modeling are necessary to provide more reliable estimates of groundwater inflows for both the Maturi and Maturi Southwest deposits.

The preliminary maximum inflow rate to the Maturi mine assumed as a basecase scenario is approximately in the range of 550 gal/min. This estimate was for the mine workings (panels) and does not include groundwater inflows to declines or other underground excavations or infrastructure.

1.17.3 Mine Design Assumptions and Design Criteria

1.17.3.1 NSR Cutoff Strategy

An NSR cutoff strategy was employed to maximize the net present value (NPV) for the deposits. The cutoff grade strategy prioritizes a higher NSR cutoff in the early years of the mine plan and uses a lower NSR cutoff in later years.

Material above marginal cutoff grade, located in the footwall behind high cutoff grade panels, is referred to as remnant material. Mining of the remnant material is included in the mine design and production schedule once targeted cutoff grade material is depleted.

1.17.3.2 Mining Methods

Post-pillar cut-and-fill is a man-entry mining method. It recovers the ore in horizontal slices, starting from a bottom level and advancing upwards. A level will be extracted by developing a horizontal slot[1] or room, from footwall to hanging wall, followed by cross-cuts that are perpendicular in both directions from the slot, which are mined on retreat. Unmined pillars will remain between the slots to provide local geomechanical stability. After the slot and cross-cuts have been extracted, a bulkhead will be installed and the mined-out area will be backfilled. Mining will continue with a new level mined immediately above the backfilled level. Pillars typically extend vertically through several levels. The pillars have been designed to yield underneath working levels where they are confined by backfill.

Long-hole stoping is a traditional blast hole stoping method where extraction and drilling drifts will be developed within the orebody. A slot raise will be mined between the drilling and extraction drifts to create a void for blasted material. Ore will be drilled from either the drilling (upper) drift or extraction (lower) drift, then loaded with explosives, and blasted towards the slot raise. Broken ore will be mucked both manually and remotely from the extraction drift. After a stope has been mined out, it will be backfilled with low-strength paste backfill. Stope walls will not be vertical but rather will be angled at 45° to create a diamond-shaped stope. This will allow for the use of lower-strength fill material, which will be engineered to stand at a 45° angle, and will conform the stope shape to the dip of the deposit. Stopes in each panel will be mined from the bottom up.

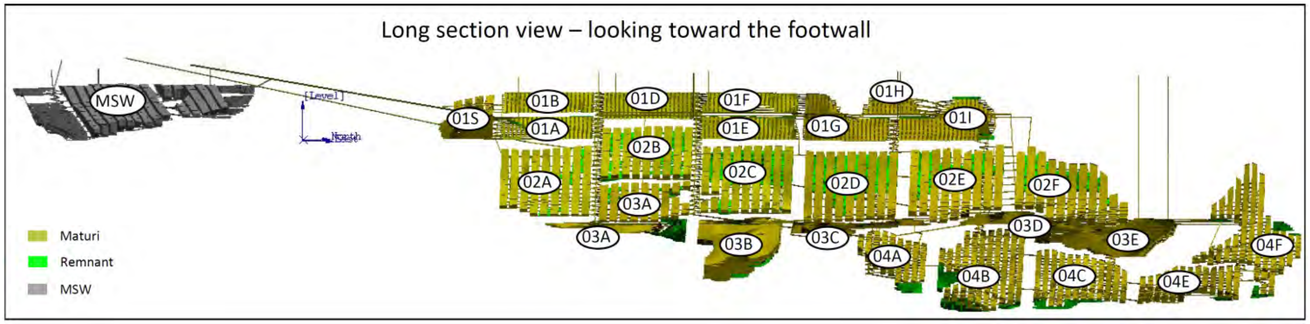
1.17.3.3 Mine Design

A 3D mine design was generated for the LOM, including development ramps and ventilation, and is shown in Figure 1-2.

The maximum mining depth will be approximately 4,300 ft below the surface elevation. The underground operation will be accessed via four declines from surface, three to Maturi and one to Maturi Southwest. Maturi Southwest will also be accessed underground from Maturi. All mining access will use ramp systems from the declines. Ventilation raises will connect levels, and tie into ventilation plenums connected by raises to the surface.

[1] A horizontal opening driven in ore and perpendicular to strike in a post-pillar cut-and-fill stope

Figure 1-2: Completed Mine Design



Note: figure prepared SRK, 2014. In the top figure, numbers starting with 01 = Tier 1; 02 = Tier 2; 03 = Tier 3 and 04 is Tier 4. Figures are schematic and not to scale.

1.17.4 Ventilation

Including the crusher and infrastructure areas, a total airflow of 3.25 million cubic feet per minute (cfm) will be required, which is approximately equivalent to 65 cfm/st based on a 50,000 st/d ore production rate.

1.17.5 Production Schedule

Scheduling was undertaken with the goal of providing 18.25 Mst/a of run-of-mine (ROM) ore to the process plant (50,000 st/d). To ramp-up as quickly as possible, three years of pre-production mining will be required to develop ramp systems, footwall drifts, stope accesses, ventilation raises, and other mine infrastructure. Because multiple working areas will be developed and numerous production faces will be exposed during the pre-production phase, it is expected that the mine will be able to achieve full ore production (i.e., 50,000 st/d) in Q2 of Year 1.

Productivity estimates for mining long-hole and post-pillar cut-and-fill stopes and associated development were generated using a first-principles methodology. The following parameters were used when creating the mine schedule:

- Quarterly ramp-up of the mine production rate (30% in January, 60% in February and 90% in March , i.e. beginning in Q1 of Year 1 and reaching capacity in Q2 of Year 1)
- Surface-stockpiled ore will be fed into the mill when required
- Long-hole stoping areas in Maturi Tier 2 and Maturi Southwest will be mined using a primary/secondary methodology. Mined-out stopes must be filled and cured prior to mining adjacent stopes
- Due to higher stresses in the Maturi Tier 4 area, a chevron-shaped mining front without secondaries was recommended to help transfer stresses up the panel and towards the regional pillars
- A 28 day backfill delay was used for all long-hole stoping areas. This constraint applies to mining adjacent stopes as well as to mining stopes that are up dip of a backfilled stope
- A 20 day backfill delay was used for all post-pillar cut-and-fill areas. This allowed for a two-stage pour and a 14 day cure time after completion of pouring.

Table 1-6 shows a summarized annual schedule for underground ore production and waste mining.

Table 1-6: Yearly Production Schedule

Year	Ore Tons (k st)	Cu (%)	Ni (%)	Pt (oz/st)	Pd (oz/st)	Au (oz/st)	Ag (oz/st)	Waste Tons [(1)] (k st)
-3	-				-	-	-	576
-2	540	0.526	0.165	0.003	0.006	0.002	0.055	1,659
-1	1,215	0.564	0.176	0.003	0.006	0.002	0.058	1,825
1	16,471	0.712	0.233	0.004	0.009	0.002	0.073	1,250
2	18,494	0.700	0.230	0.004	0.009	0.002	0.072	1,752
3	18,481	0.719	0.238	0.004	0.009	0.002	0.074	1,235
4	18,327	0.705	0.237	0.004	0.010	0.002	0.073	1,164
5	18,250	0.674	0.231	0.004	0.010	0.003	0.071	1,566
6	18,250	0.647	0.220	0.005	0.011	0.003	0.068	1,037
7	18,247	0.668	0.223	0.005	0.011	0.003	0.071	944
8	18,253	0.666	0.219	0.005	0.010	0.003	0.072	1,191
9	18,250	0.654	0.215	0.005	0.011	0.003	0.072	1,380
10	18,250	0.649	0.202	0.005	0.012	0.003	0.071	1,185
11	18,250	0.610	0.183	0.006	0.013	0.003	0.068	1,492
12	18,231	0.584	0.182	0.006	0.013	0.003	0.062	862
13	18,245	0.611	0.199	0.006	0.013	0.003	0.063	1,860
14	18,253	0.609	0.184	0.006	0.014	0.003	0.064	1,357
15	18,272	0.605	0.181	0.007	0.016	0.004	0.065	666
16	18,250	0.625	0.187	0.007	0.015	0.003	0.069	544
17	18,251	0.634	0.190	0.006	0.015	0.003	0.069	939
18	18,251	0.594	0.185	0.005	0.012	0.003	0.064	524
19	18,251	0.565	0.177	0.004	0.010	0.002	0.061	604
20	18,250	0.547	0.173	0.004	0.009	0.002	0.059	667
21	18,250	0.527	0.175	0.004	0.008	0.002	0.056	609
22	18,250	0.513	0.169	0.003	0.008	0.002	0.054	823
23	18,250	0.509	0.168	0.004	0.008	0.002	0.053	492
24	18,250	0.506	0.167	0.004	0.008	0.002	0.052	484
25	18,250	0.497	0.164	0.003	0.007	0.002	0.050	401
26	18,250	0.483	0.158	0.003	0.007	0.002	0.049	307
27	15,660	0.457	0.149	0.002	0.006	0.001	0.046	330
28	15,073	0.442	0.144	0.002	0.006	0.001	0.045	306
29	10,906	0.449	0.148	0.002	0.006	0.001	0.046	156
30	10,174	0.451	0.153	0.002	0.006	0.003	0.047	181
Total	*526,843*	*0.593*	*0.191*	*0.004*	*0.010*	*0.002*	*0.063*	*30,368*

Note: Ore mined in Years -3 through -1 and select low-grade material in Years 1 to 4 is stockpiled underground or on surface and fed into the mill in later years. (1) Includes waste tons mined by the contractor in Years -3 and -2.

The equipment provisions include all primary and secondary equipment needed to meet the LOM production schedule requirements.

1.18 Proposed Recovery Plan

The concentrator facilities proposed for the Project comprise a process plant with an ore capacity of 50,000 st/d, a single process line using semi-autogenous grind (SAG) and ball milling with sequential copper and nickel flotation, high-rate tailings thickening, concentrate receiving system, filter plant, concentrate storage, and rail load-out.

Metallurgical projections for the PFS financial model were created through the sequential use of rougher kinetics testing, locked cycle testing and pilot plant testing. Models provide predictions of rougher flotation recoveries, and the performance of the cleaner circuit in the processing of the rougher concentrate which are derived from metallurgical testing and are based on input parameters available in the resource model.

Ramp-ups have been assumed with respect to throughput, recoveries for both copper and nickel and also copper and nickel concentrate grades based upon industry experience for similar polymetallic operations producing separate concentrate products. After throughput ramp-up is achieved at the end of the first quarter after commissioning (six months from start up) maximum throughput rates are maintained through for 26 years through until Year 27. Copper and nickel head grades fed to the plant are highest at the beginning of operations and reduce through until the end of operations in Year 30. The feed grades are matched by the copper and nickel concentrate production figures which are highest after the second year of operation and reduce gradually through until the end of the LOM, thereby maximizing the Project value.

Excess material mined over 18.25 Mst will stockpiled and fed back to the plant when excess capacity becomes available. Ore becomes available two years before process plant production begins, due to the mine development work, and is stockpiled on the surface. A three-month period of commissioning commences with this surface stockpile material treated through the plant. A minor stockpile of around 2.3 Mst is generated in Years 1 to 4 of mining and is fed into the plant when shortfalls in mine production occur. Maturi Southwest material is introduced to the plant in the 19th year of operation and is fed through until the 28th year of operation. This material represents 8.2% of the feed to the plant over the LOM. A pyrrhotite rejection circuit is used during the treatment of Maturi Southwest material to maintain nickel concentrate grades despite the lower nickel feed grades and higher pyrrhotite to pentlandite ratios in the Maturi Southwest material.

1.19 Proposed Infrastructure

The Project would be subdivided into three non-contiguous primary sites consisting of the mine site, the concentrator site, and the tailings storage facility (TSF) site. The mine site would include the Maturi and Maturi Southwest deposits, located on either side of Birch Lake. The concentrator site would be 1–2 mi west of the Maturi Southwest and Maturi deposits. The TSF site would be approximately 13 mi southwest of the concentrator site, southwest of the town of Babbitt. Fresh water would be locally sourced. The mine site and concentrator site will be located in the Rainy River headwaters watershed of the Rainy River water basin, which drains north to Hudson Bay. The TSF site would be located in the St. Louis River watershed of the Great Lakes water basin, which drains south to Lake Superior.

At the mine site, primary and secondary portals for mine access would be constructed southwest of the proposed Maturi mine, and near the process plant, on the west side of Birch Lake. The primary portal would be a single decline, and the secondary portal would include two declines. Ore derived from underground mining and crushing would be conveyed to the surface for processing at the concentrator. Paste plants would process tailings (pumped via pipeline from the concentrator to the mine site), cement, and flyash for paste backfill into the mine. Three paste plants would be located in the Maturi area, to the east of Birch Lake, and one would be located in the Maturi Southwest area. Multiple ventilation facilities (total of 13) would provide the required air intake and exhaust for mine ventilation. Utilities, consisting of electric, water, sewer, and natural gas for mine heating (derived from liquefied natural gas) would service the mine site.

The concentrator site, which overlaps with the mine site in the mine portal area, would include the concentrator, stockpiles, laydown areas, concentrator process water pond, administration and operational support buildings, and the craft support service installation. Utilities would include electric, water, and sewer. Fresh water would be pumped from a water source to the concentrator and mine sites via pipelines. Tailings would be pumped from the concentrator site to the TSF and the paste plants. Copper concentrate and nickel concentrate would be pumped via pipeline from the concentrator to the TSF site for processing at the concentrate filter plant. Concentrate would be transported to market via rail.

The TSF site would include a conventional, lined TSF, a filter plant to process concentrate, facilities for loading concentrate to rail cars, and facilities for receiving, storing and distributing supplies. Return water would be piped from the TSF site to the concentrator site. Utilities would include electric, water, and sewer.

Labor, materials, and concentrates would be transported to and from the Project sites by roads (state, county, and local) and via railroad. Supplies arriving via rail would be transferred to trucks and transported to the point of use.

1.20 Markets

Wood Mackenzie assessed the proposed products from the Project and confirmed their suitability for sale into the custom nickel and copper concentrate markets. The quality of the copper concentrate is suitable for the custom concentrate market and therefore would attract standard commercial terms, including benchmark copper treatment and refining charges for contained silver and gold, and payable metal percentages. No penalties for deleterious elements such as As, Hg, Pb, or Bi are expected. The customers for the nickel concentrate will likely be nickel smelters in North America, Europe, Russia and China. China will be a potential market for the copper concentrate, along with other custom smelters in Europe and Asia.

No contracts are currently in place for any production from the Project.

1.21 Permitting, Environmental, and Social License

The Project is located within the area that was ceded to the United States by the Chippewa of Lake Superior in the 1854 Treaty of LaPointe. Current land use in the region includes mining, forestry, and recreation on a mixture of private and public lands. The Boundary Waters Canoe Area Wilderness is in close proximity to the proposed Project area.

1.21.1 Permitting

In Minnesota, mine permitting, operation and reclamation are regulated by the Minnesota Department of Natural Resources (DNR). Also, the proposed Project is located, at least in part, on federally-administered public land, and includes federal minerals, which makes various elements of the project subject to permitting by the U.S. Forest Service and the Bureau of Land Management. Regulatory oversight for the Project would be conducted by both state and federal agencies. There would be at least eight federal and state agencies involved in reviewing the Project. In particular, because of the patchwork of federal, state, and private mineral and surface properties involved, multiple agencies may have jurisdiction over the same lands and/or closely related regulatory issues.

The Project will be subject to review under both the National Environmental Policy Act (NEPA) and the Minnesota Environmental Policy Act (MEPA), and under these frameworks, the project is subject to review by multiple state and federal agencies.

Certain permits, such as the State of Minnesota permit to mine, are not issued until the environmental review and Environmental Impact Statement (EIS) have been finalized in accordance with the requirements of MEPA and NEPA. Other permits and authorizations, including but not limited to such as authorization to access federal and state lands and minerals for drilling, or decisions on lease issuance or renewal, may be sought during early stages of the Project. During TMM's preparation and filing of the mine plan of operations (MPO), TMM would undertake a number of additional Project-related activities requiring federal and state agencies to make decisions under various statutes and regulations. Only after the agencies have completed the required environmental review process (es) would the agencies issue decisions on the proposed activity. Accordingly, the environmental review process would be a critical preliminary regulatory step for agency approval of almost any activity proposed by TMM.

1.21.1.1 Environmental Review and Environmental Impact Statement

The first step in the environmental review process would be collection of data to characterize the project area and identify issues of special concern. Baseline environmental data collected to date and proposed for collection during future planned work would be used to develop the MPO, and would be considered in the draft scoping environmental assessment worksheet (EAW) for the Project. Both documents would be prepared by Project personnel and submitted to state and federal agencies for review.

Pursuant to Minnesota regulations governing non-ferrous mining, an EIS is required. After finalizing the MPO, TMM would be required to file the MPO with the Bureau of Land Management (BLM) and equivalent documentation with the DNR. The MPO filing would trigger a joint federal–state EIS environmental review in which the BLM and DNR, as the agencies responsible for federal and state minerals, respectively, (and likely the United States Forest Service (USFS), as the federal surface manager), would act as the lead agencies in developing an extensive Project EIS, which would take several years to complete.

The Project EIS and permitting processes would require co-ordination between TMM and the relevant federal and state agencies, as well as tribal bands and local governments in the vicinity of the Project (e.g., Lake and St. Louis Counties and the cities of Ely and Babbitt).

The lead agency (or agencies) has the discretion to determine whether studies conducted during environmental review would be performed by Project personnel, or the lead agency. Data collected or used to describe baseline environmental conditions would have to meet data quality objectives (DQOs) that are based on scientific and engineering principles for technically defensible results. DQOs include data of sufficient quality and quantity to allow application, evaluation, and/or comparison to existing statutory regulations.

Once the environmental review is complete, the draft EIS (DEIS) will be published, a period of public comment will be held, and any received comments must be fully addressed. Public hearings will likely be held. It may be necessary to issue a supplemental EIS (SEIS) if the comments require significant changes to the proposed actions, or if the comments engender additional studies. Once the public comments are fully addressed, and/or the SEIS is issued, a determination of adequacy by the state of Minnesota agencies will be made, and a record of decision (ROD) by the federal agencies will be published. Once these conditions are satisfied, the final EIS (FEIS) will be published. Permits to operate the Project can only be issued after the determination of adequacy and the ROD have been issued and the FEIS is published. Appeals of the final FEIS and final permit issuance may be made by TMM or other affected individuals or groups. The appeals process can add additional time to the Project timeline.

1.21.1.2 Permitting

Before constructing the Project, TMM would have to obtain a number of federal, state, and local permits. The permitting process will be strongly influenced by the information obtained, the multiple "alternatives" considered and selected, and the related mitigation options identified and selected by the agencies during the Project EIS. For the most part, with respect to the Project EIS outcome, the federal and state agencies are required by law to identify and select the least environmentally damaging practicable alternative (LEDPA).

As stated previously, prior to and concurrently with the EIS process, TMM intends to file draft applications for a wide variety of permits with federal and state agencies, which would initiate other regulatory review procedures. There is some risk associated with this strategy, as the EIS process may result in a significantly altered Project that would require modifications to the draft permit applications. This could cause delays in the issuance of necessary permits.

TMM has indicated that the Project will be operated as a zero-discharge facility, and process water will not be discharged. However, AMEC has not been able to verify the water balance prepared by TMM's Environmental Consultant is zero-discharge for all cases and all years during the LOM. Further study would be required to verify the zero-discharge condition for all cases and all years. If it is determined that discharge is likely for any point during the LOM, TMM would likely be required to apply for and maintain an NPDES permit for the discharge of process water. Treatment of the water prior to discharge would likely be required, and would require the inclusion of a water treatment facility in the infrastructure.

1.21.2 Environmental Considerations

The environmental study area would encompass currently proposed Project facilities including: surface lands above the underground mining areas and surface facilities (the mine site), the concentrator site, TSF and ancillary facilities, and utility corridors. The utility corridors would include roads, rail lines, power transmission lines, natural gas pipelines, tailing and concentrate pipelines, and water pipelines. For many resources, the environmental study area would extend substantially beyond those facilities to include an adequate geographic area for baseline characterization and modeling needs.

The Boundary Waters Canoe Area Wilderness is the largest designated wilderness in the eastern U.S. and is the only lake–land wilderness of its kind and size in the National Wilderness Preservation System. The Boundary Waters Canoe Area Wilderness is located near the proposed Project area, and some parts of the Boundary Waters Canoe Area Wilderness will likely be included as part of the environmental study area for the EIS assessments. The Boundary Waters Canoe Area Wilderness is the most prominent environmental issue in the public debate over the future of copper–nickel mining in Minnesota.

A number of desktop reviews of publicly-available data, together with Project-specific field studies, have been initiated in support of preliminary preparations for the Project environmental review and EIS. These include reviews of available climate data; hydrologic surface water and groundwater data; surface water quality conditions; baseline stream morphology data; aquatic, vegetation and wildlife biota; wetland data; sediment sampling; fish sampling and fishery management; cultural and paleontological resources; air quality levels; noise levels; and land use and recreation. TMM has performed some limited baseline data collection in some of these areas, though specific work plans for these efforts have not been reviewed by the regulatory agencies.

As part of the work program to complete environmental review and permitting for the mine site, TSF site, and associated infrastructure, studies and reports including, but not limited to, the following are likely to be required:

- Hydrology, hydrogeology, and water quality field studies
- Soil and sediment analysis
- Stream morphology analysis
- Aquatic biota studies
- Wetlands and waters of the U.S. – field surveying, delineation, and mapping
- Vegetation mapping and field reconnaissance
- Rare plant survey and report
- Wildlife species field survey, habitat assessment study, and report
- Canada lynx report
- Fisheries and aquatic resources field sampling
- Fisheries and aquatic resources report
- Cultural resource identification
- Air quality and meteorological data collection and modeling
- Greenhouse gas evaluation
- Noise monitoring, and possibly vibration monitoring
- Land use report
- Socioeconomic studies
- Visual resources studies and modeling.

The detailed scopes of these studies, with the exception of the hydrogeology study, have not yet been developed.

Water resources within the Project study area consist of lakes, reservoirs, larger rivers, medium-sized perennial streams, smaller perennial to intermittent tributaries, and variously-sized wetlands. Water features are contained within two major drainages: Lake Superior and Hudson's Bay, which are separated by the Laurentian Continental Divide that traverses the study area. Most of the study area including the mine site, the concentrator site, and a portion of the utility corridor are located north of the Laurentian Divide within the Hudson's Bay drainage (also referred to in this Report as the Rainy River water basin). The TSF site and portions of the utility corridor near the TSF site are south of the Laurentian Divide and lie within the Lake Superior drainage (also referred to in this Report as the Great Lakes water basin). Depending on the source of the water to be used in the processing of ore, some permits for the transfer of water from one basin to another (inter-basin transfer) may be required. Stormwater management will also be important in meeting surface and groundwater quality standards, and possibly inter-basin transfer requirements.

1.21.3 Current Environmental Liabilities

Liabilities associated with the mineral exploration program would be related to abandonment of boreholes and drill pad and road reclamation. Reclamation bonds have been posted with the BLM and the Minnesota Department of Transportation (MnDOT).

Historical mine features on the Project site include two former bulk sample sites; an underground shaft and workings developed in 1968 and a surface excavation developed in 1974. TMM has reclamation responsibilities under applicable leases, and may be responsible for additional reclamation of the bulk samples sites if required; however, no specific reclamation has been requested by any agencies to TMM's knowledge and no reclamation plans have been developed by TMM at the Report effective date.

Ongoing liabilities at the adjacent Cliffs-Erie Dunka property, which are part of the TMM holdings, include permitted discharges from a sulfide-bearing rock stockpile and wetland treatment system, and permitted discharges of untreated mine pit water.

1.21.4 Environmental Risks

The environmental risks of highest consequence would be related to:

- Contamination of surface water and soils due to a containment failure at stockpiles, ponds, pipelines, TSF, or other facilities
- The possibility of discharge of process water during years of high precipitation, which would likely require the installation of a water treatment facility

- Refusal of permits for backfill with additives such as fly ash or slag due to the potential for unacceptable environmental impacts
- Unanticipated fugitive dust emissions from stockpiles, roads, and TSF
- Unanticipated impacts to surface waters due to mine dewatering
- Unanticipated impacts to sensitive receptors, including, but not limited to, the Boundary Waters Canoe Area Wilderness, and federally-listed endangered species.

These risks would be investigated during the MPO and as part of more detailed studies additional engineering, and environmental testing, and mitigations would be developed.

1.21.5 Closure

As the Project is in very early stages of development, closure costs are at a conceptual level of detail. Definition of closure requirements is expected to begin during MPO development. At that time, TMM would develop a closure strategy plan for discussion with state and federal agencies and local communities. Facility closure plans would be defined in permits and required to be annually updated. The development of the Project's closure plan would also be subject to public input during environmental review.

As required by applicable laws it is expected that TMM's closure responsibilities for the Project would include all surface and underground facilities including buildings and structures, roads, utilities, and services. Presently, TMM has no plans to release properties post-closure, and it is TMM's intent to maintain properties after closure.

Site maintenance and monitoring would occur for a period of time beyond closure completion (post-closure) as defined in the facility permits and plans. Specific activities have not yet been identified by TMM for post-closure periods. It is expected that TMM will identify these actions during the MPO and during more detailed Project studies.

No closure costs were included in the PFS. AMEC has included a conceptual closure cost allocation for closure of the entire Project site in this financial model of $210 M, based on benchmarking with similar projects. The closure cost estimate does not include any allocations for post-closure monitoring or permit maintenance. AMEC notes that the final closure cost estimate will depend on the MPO phase, when the Project design is optimized, and will also depend on the conditions that may be imposed on TMM during permitting.

1.21.6 Social License

Project stakeholders are likely to include local, state, or federal government elected bodies or regulatory agencies, state and local business interests, educational institutions, local community interests, tribal bands, and non-government organizations (NGOs). While some informal discussions have been undertaken, to date no formal stakeholder consultations have been initiated.

A thorough socio-economic baseline analysis, analysis of projected and potential socio-economic impacts of the proposed Project, analysis of potential project alternatives (including a "no build" alternative), and a "cumulative impacts" analysis that will include identification and assessment of any known "regional development plans" or economically significant projects, will be required as part of the Project's draft EIS. Local or regional trends/transformations that may affect the Project would be identified in that analysis.

In order to determine the adequacy of the investment climate, Duluth and TMM will continuously monitor workforce availability for the Project; trends in public opinion of the Project; and regional economic development projects, proposals, and/or trends that may impact the Project or the perception of the economic value of copper–nickel mining projects.

1.22 Capital Cost Estimates

The capital cost estimate for the Project was developed by TMM's Independent Engineer, with input from consultants for specific areas. The capital cost estimates are based on a combination of quotes, vendor pricing, and experiences with similar-sized operations. The costs were reported by TMM's Independent Engineer at a prefeasibility level of accuracy where the estimate accuracy range is defined as +25%/-20% including contingency and are consistent with an AACE International (formerly Association for the Advancement of Cost Engineering) Class 4 Estimate.

Costs in the PFS were reflective of Q3 2013 market conditions. TMM's Independent Engineer and its consultants assessed overall construction personnel requirements, material availability and logistics, work methods, and risks. Escalation was excluded from all estimates.

AMEC performed a detailed estimate review of the PFS capital cost estimate. AMEC considered that the earthworks, excavation costs, concrete works, and contingencies were underestimated, and made an upward adjustment of approximately $156 M to cover these areas. This increased the initial capital cost estimate to $2,774.86 M.

A similar review was performed on the PFS sustaining capital estimate, and AMEC noted that the earthworks were underestimated, and made an upward adjustment of approximately $98 M.

When sustaining capital ($2,635.63 M) costs, including closure costs of $210 M, are incorporated, the total Project capital cost estimate as restated by AMEC is $5,410.49 M. The capital costs, as endorsed by AMEC, are summarized in Table 1-7. Sustaining capital costs are included as Table 1-8.

1.23 Operating Cost Estimates

The operating cost estimate for the Project is provided in Table 1-9.

1.23.1 Mining Costs

The mine operating costs for the Project were developed using a bottom-up first-principles method. All direct and indirect mining costs were calculated from this method using mine activity performance and a cost modeling process. Unit costs were generated from budgetary quotations from industry suppliers. Labor and utility costs were provided by TMM. All costs were benchmarked.

Operating costs over the LOM are $6,615.4 M, and average $12.56/st mined. Mining costs do not include operation of the paste backfill system; costs for the tailings and paste system are their own line item.

1.23.2 Underground Infrastructure Costs

The LOM infrastructure operating cost of $1.69/st was based on the LOM production schedule. Costs over the LOM total $890 M.

1.23.3 Process Operating Costs

The process plant operating cost estimate has a targeted accuracy of ±25%. The operating costs for all surface facilities have been based on similar, currently-operating facilities.

Total power consumption of surface facilities including filtration is estimated to be 19,852 kWh/st and a unit power cost of $48.90 per MWH, this equates to $17.72 M per year. The annual reagent cost is estimated to be $16.5 M or $0.90/st. Grinding media and liner requirements are estimated to total $22.02 M/a or $1.21/st. Maintenance materials have been estimated by factoring at $0.25/st or $4.563 M/a, which would cover replacement wear items such as the hydrocyclones, screens, wear plates. An allowance has been made for heating of $10/st or $1.825 M/a. An allowance of $0.01/st or $0.216 M/a has been estimated for the replacement of filter cloths.

Total annual operating costs for the concentrator and filter section are US$72.87 M or US$3.99/st.

Table 1-7: Initial Capital Cost Estimate (restated)

Description	US$ (millions)
Mine	794.0
Process	955.6
Tailings and paste	546.6
Surface infrastructure	378.7
Owners costs	100.0
Total initial capital	*2,774.9*

Table 1-8: Sustaining Capital Cost Estimate (restated)

Description	US$ (millions)
Mine	1,800.4
Tailings and paste	835.2
Total sustaining capital	*2,635.6*

Note: Reclamation costs are included in the mine area; surface infrastructure sustaining is included under the tailings area.

Table 1-9: Operating Cost Estimate Summary (restated)

Area	Costs (US$ x 1,000)	Unit Cost	Units
Mining	6,615.4	$12.56	US$/st ROM
Processing	2,103.0	$3.99	US$/st milled
G&A	1,421.0	$2.70	US$/st milled
Surface Infrastructure	1,311.0	$2.49	US$/st milled
Total	11,450.3	$21.73	US$/st milled

1.23.4 Infrastructure Operating Costs

Costs include operating and maintenance labor and materials, reagents (including cement and fly ash), equipment operating costs, and power costs. Included in the operating cost estimate in this area are tailings transport (slurry lines) to the TSF and paste plants, TSF operation, paste plant operations, underground paste distribution system, surface water management, and general site operations costs. Over the LOM, infrastructure operating costs total $45.44 M/a or $2.49/st.

1.23.5 General and Administrative Costs

General and administrative costs include management, site services, administrative support functions, safety department, and the technical services group. Over the LOM, these costs are estimated at $49.27 M/a, or $2.70/st.

1.24 Economic Analysis

The cautionary statements in Section 1.3 should be read in conjunction with this sub-section.

The Project has been evaluated using a discounted cash flow (DCF) analysis. Cash inflows consist of annual revenue projections for the mine. Cash outflows such as capital, including the three years of preproduction costs, operating costs, taxes, and royalties are subtracted from the inflows to arrive at the annual cash flow projections.

To reflect the time value of money, annual net cash flow (NCF) projections are discounted back to the project valuation date using several discount rates. The discount rate appropriate to a specific project depends on many factors, including the type of commodity; and the level of project risks, such as market risk, technical risk and political risk. The discounted, present values of the cash flows are summed to arrive at the project's net present value (NPV).

In addition to NPV, internal rate of return (IRR) and payback period are also calculated. The IRR is defined as the discount rate that results in an NPV equal to zero. Cash flows are taken to occur at the end of each period. Capital cost estimates have been prepared for initial development and construction of the project, and ongoing operations (sustaining capital).

The resulting net annual cash flows are discounted back to the date of valuation end-of-year 2014 dollars, and totaled to determine NPVs at the selected discount rates. The payback period is calculated as the time needed after the start up of operations to recover the initial capital spent.

Table 1-10 presents the base case metal prices assumptions used to value the Project.

The after-tax NPV at an 8% discount rate over the estimated mine life is $753 million. The after-tax IRR is 11.4%. Payback of the initial capital investment is estimated to occur in 7.2 years after the start of production. A summary of the financial analysis in US$ is presented as Table 1-11.

1.25 Sensitivity Analysis

Sensitivity analysis was performed on the base case net cash flow and examined sensitivity to copper price, nickel price, operating costs and capital costs. For the purposes of the analysis, changes in nickel and copper grades were found to be reasonably represented by the changes in metal prices, and are not shown.

Figure 1-3 summarizes the sensitivities in the after-tax scenario. The Project is most sensitive to changes in copper prices, less sensitive to changes in operating costs, less sensitive to changes in capital costs and least sensitive to changes in nickel price. Table 1-12 summarizes the sensitivity to these factors on after-tax NPV using 8% discount rate.

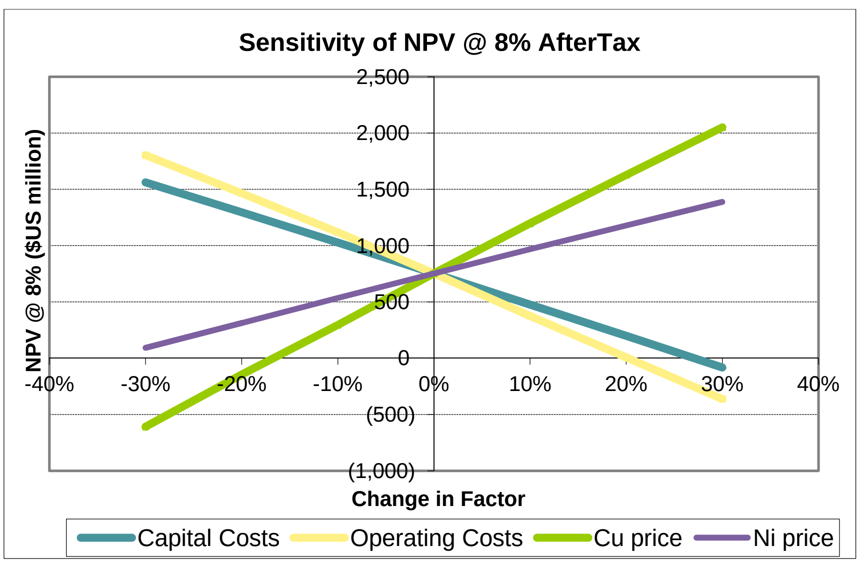
Table 1-10: Metal Price Assumptions

Metals prices	Units	LOM
Copper	US$/lb	3.50
Nickel	US$/lb	9.50
Gold	US$/oz	1,300
Palladium	US$/oz	815
Platinum	US$/oz	1,680
Silver	US$/oz	21.50

Table 1-11: Cashflow Summary Table

Pre Tax	Units	LOM
Cumulative Cash flow Pre Tax	US$M	7,913
NPV 6%	US$M	2,231
NPV 8%	US$M	1,358
NPV 10%	US$M	732
Payback period	Years	6.4
IRR before tax	%	13.6%
After Tax	**UNITS**	**LOM**
Cumulative Cash flow After Tax	US$M	6,003
NPV 6%	US$M	1,449
NPV 8%	US$M	753
NPV 10%	US$M	257
Payback period	Years	7.2
IRR after tax	%	11.4%

Figure 1-3: Sensitivity of After-Tax NPV at 8% Discount Rate



Note: Figure prepared by AMEC, 2014.

Table 1-12: Sensitivity of the Financial Analysis to Changes in Metal Prices, Operating Costs and Capital Costs (basecase is highlighted)

Sensitivity Of NPV @ 8% After Tax		Change in Factor -30%	-20%	-10%	0%	10%	20%	30%
Factor	Capital Costs	1,562	1,296	1,027	753	477	199	(82)
	Operating Costs	1,803	1,465	1,118	753	375	7	(362)
	Cu price	(607)	(145)	296	753	1,197	1,628	2,051
	Ni price	93	312	536	753	968	1,179	1,388

1.26 Conclusions

Based on the assumptions detailed in this Report, the Project shows a positive financial return and supports the declaration of Mineral Reserves.

Should the Duluth and TMM Boards make such a decision, there is sufficient support from the Report results for progression to a feasibility study.

1.27 Recommendations

A two-phase work program is recommended to complete a MPO, feasibility study, and EIS, and to prepare associated permit applications.

Phase 1 will provide data support to allow TMM to complete the necessary testwork, engineering, and documentation to support the application for a mine plan of operation (MPO). The application for the MPO describes the configuration of the Project, so must be supported by sufficient engineering to adequately define all major variables, facility locations, and production rates. The submission of the MPO will conclude the Phase 1 work program, and will trigger the EIS.

It is likely that the technical component of the MPO will cost between $70 and $100 million to complete, with the approximate budget estimate allocation by key area being as follows for Phase 1:

- Engineering: $7–10 million
- Bulk sample and pilot plant program: $20–25 million
- Drilling: $8–13 million
- Environmental: $36–49 million.

Phase 2 will build on Phase 1, and can be conducted in part concurrently with Phase 1.

Phase 2 will provide engineering and data support to allow completion of the feasibility study and the required EIS. The EIS and feasibility study will need to be undertaken concurrently, as the Project design as contemplated in the feasibility study must accommodate the recommendations arising out of the EIS; and the EIS must correctly reflect the proposed Project design. It is likely that the technical component of the EIS and feasibility studies in Phase 2 will cost between $57 and $74 million to complete, with the approximate budget estimate allocation by key area being as follows by Phase 2:

- Engineering: $6–8 million
- Ongoing pilot plant program: $5–10 million
- Drilling: $11–16 million
- Environmental: $35–40 million.

AMEC notes that the estimate for the environmental portion in Phase 2 is likely to be the upper end of the potential expenditure. The estimate allocation assumes that third-party data verification for the EIS of the MPO work phase will be required by the regulatory authorities.

The budget estimates are restricted to technical work, and no provision has been made in the estimates for items such as corporate overheads, land acquisition, legal and other consulting fees, additional work or program changes that may be required as a result of interactions with regulatory agencies, community and stakeholder consultations, or permit applications and acquisition.

2.0 INTRODUCTION

AMEC E & C Services Inc. (AMEC) was commissioned by Duluth Metals Limited (Duluth) to compile an independent NI 43-101 Technical Report (the Report) for the Twin Metals Minnesota Project (the Project) located near Ely Minnesota, USA. The Project location is shown in Figure 2-1.

The firms and consultants who are responsible for the content of this Report, which is based on a prefeasibility study completed in 2014 (the PFS) and supporting documents prepared for the PFS, are, in alphabetical order, AMEC, Barr Engineering Co. (Barr), Blue Coast Metallurgy Ltd. (Blue Coast), Golder Associates Inc. (Golder), Itasca Consulting Group, Inc., Itasca Denver, Inc. (collectively Itasca) and SRK Consulting (US) Inc. (SRK).

Some of the work preparation for the PFS was completed by two third-party consulting firms, which are unable to be identified due to the terms of their respective contract agreements with TMM, and are referred to in the Report as “TMM’s Independent Engineer” and “TMM’s Environmental Consultant”, respectively.

2.1 Terms of Reference

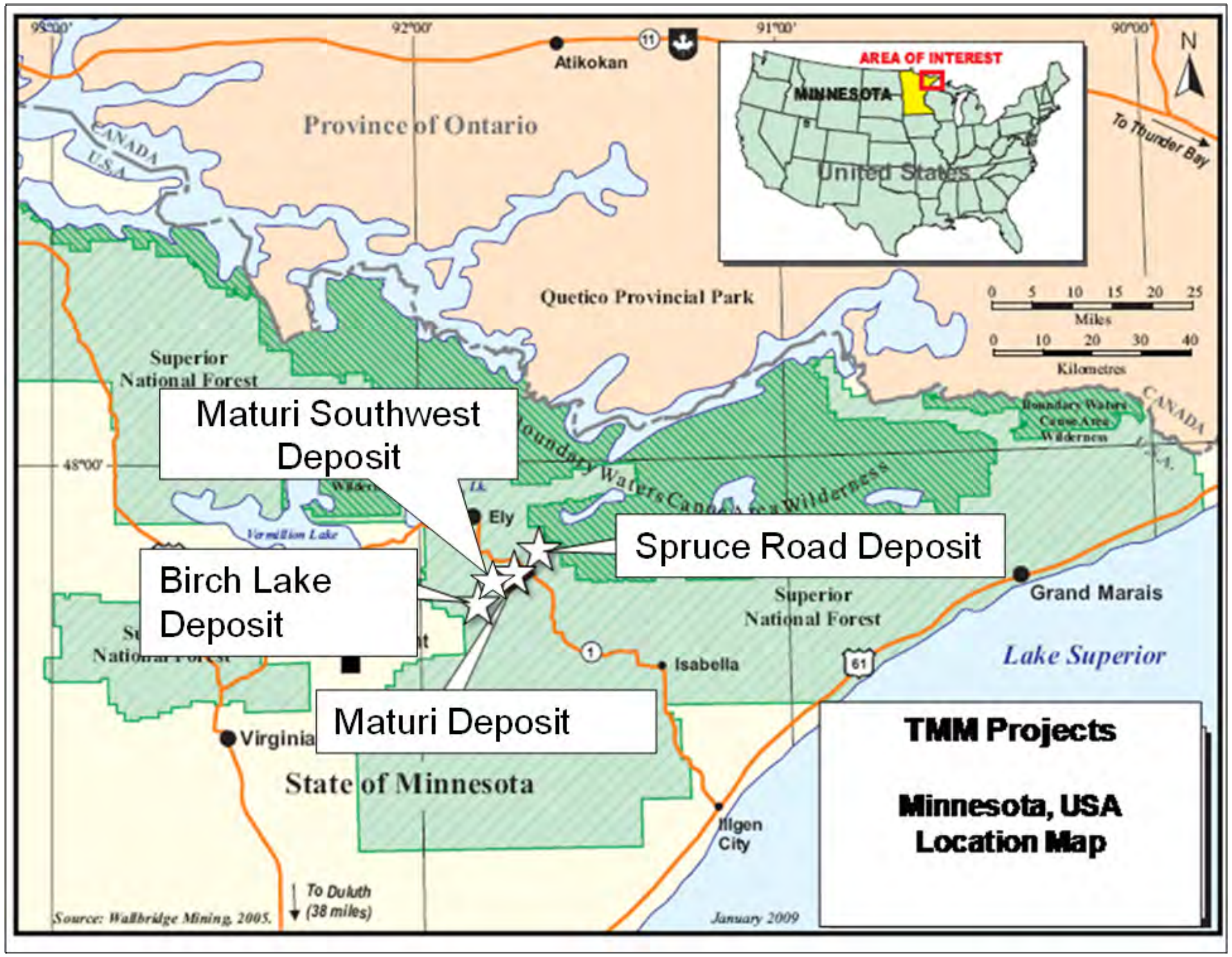
The Report will be used in support of Duluth’s press release dated 20 August 2014 that is entitled “Duluth Metals Highlights Low Copper (C1) Cash Costs and Strong Operating Margins in its Pre-feasibility Study for Twin Metals Minnesota Project”. The report was amended 6 October 2014 as Table 22-4 had not been reproduced properly during the conversion to pdf.

TMM is a limited liability company that, since 2010, has been operated as a joint venture between Antofagasta PLC (Antofagasta) and Duluth, under a Participation and Limited Liability Company Agreement (the Participation Agreement). TMM is 35% owned by Duluth Metals Holdings (USA) Inc. (which is indirectly held by Duluth), 25% owned by Twin Metals (USA) Inc. (which is indirectly owned by Duluth) and 40% owned by Northern Minerals Holding Co. (which is indirectly owned by Antofagasta). Accordingly, Duluth holds, directly or indirectly, a 60% controlling interest in TMM. For the purposes of this Report TMM and Duluth are used interchangeably.

All measurement units used in this Report are US units, and currency is expressed in US dollars unless stated otherwise. The Report uses US English.

A number of abbreviations are used throughout the report to refer to current and former corporations involved in Project development, regulatory bodies, and regulatory requirements (Table 2-1).

Figure 2-1: Project Location Plan



Note: Figure courtesy Duluth, 2014.

Table 2-1: Abbreviations

Abbreviation	Explanation	Abbreviation	Explanation
ACNC	American Copper and Nickel Company Inc.	Altoro	Altoro Gold Corporation
ARPA	National Historic protection Act, Archaeological Resources Protection Act	BBJV	Beaver Bay Joint Venture
BLM	U.S. Bureau of Land Management	BWSR	Minnesota Board of Water and Soil Resources
CERCLA	Comprehensive Environmental Response, Compensation and Liability Act	DNR	Minnesota Department of Natural Resources
EIS	Environmental Impact Statement	EMP	Environmental Management Plan
FRA	Franconia Minerals Corporation Inc.	IPCO	International Platinum Company Inc.
LEM	Lehmann Exploration Management	MAAQS	Minnesota Ambient Air Quality Standards
MBS	Minnesota Biological Survey	MDA	Minnesota Department of Agriculture
MEPA	Minnesota Environmental Policy Act	MnDOT	Minnesota Department of Transportation
MPCA	Minnesota Pollution Control Agency	MPO	Management Plan for Operations
NAAQS	National Ambient Air Quality Standards	NAGPRA	Native American Graves Protection and Repatriation Act
NEPA	National Environmental Policy Act	NHP	Minnesota Natural Heritage Program
NPDES	National Pollution Discharge Elimination System	NNRI	Minnesota Natural Resources Research Institute
NWI	National Wetlands Inventory	PMA	primitive management area
RFSS	Regional forester sensitive species	ROD	Record of Decision
SDEIS	Supplemental Draft Environmental Impact Statement	SDS	State Disposal System
SGCN	Species of Greatest Conservation Need	SNF	Superior National Forest
USDA	U.S. Department of Agriculture	USEPA	U.S. Environmental Protection Agency
USFS	U.S. Forest Service	USFWS	U.S. Fish and Wildlife Service
USGS	U.S. Geological Survey	Wallbridge	Wallbridge Mining Company Limited

2.2 Qualified Persons

The following serve as the qualified persons for this Technical Report as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101F1:

- Mr. John Barber, P.E., Technical Director, Underground Mining, AMEC
- Dr. Ted Eggleston, RM SME, Principal Geologist, AMEC
- Dr. Harry Parker, RM SME, Consulting Geologist and Geostatistician, AMEC
- Dr. Lynton Gormely, P.Eng., Principal Process Consultant, AMEC
- Mr. David Frost, F.AusIMM, Technical Director, Process, AMEC
- Mr. Simon Allard, P.Eng., Principal Consultant and Study Manager, AMEC

- Ms. Janine Hartley, P.E., Senior Engineer, AMEC
- Mr. Trey White, P.E., Principal Mining Engineer, AMEC
- Dr. Srikant Annavarapu, P.E. Principal Mining Engineer, AMEC
- Mr. Tom Radue, P.E., Vice President and Senior Geotechnical Engineer, Barr
- Mr. Chris Martin, C.Eng., President and Principal Metallurgist, Blue Coast
- Mr. Matthew Malgesini, P.E., Senior Consultant, Golder
- Dr. Matthew Pierce, P.E., Principal Engineer, Itasca
- Dr. Robert Sterrett, P.G., Principal Hydrogeologist, Itasca
- Ms. Joanna Poeck, RM SME, Senior Consultant (Mining), SRK.

2.3 Site Visits and Scope of Personal Inspection

Mr. Barber visited the site from 23–24 July, 2013. During this visit, he inspected the area proposed as the process plant site, the tailings storage facility (TSF) site, and the land surface at the Inco shaft site and Maturi area, and visited the Ely core storage facility where he reviewed selected drill core samples.

Dr. Parker visited the Project site and/or Project offices from 26 to 30 April, 2011, 6–16 September 2011, 5–7 April 2012, 19–20 June 2012, 23–27 April 2013, 28 June 2013, 6 August 2013 and 28 August 2013.

Under the supervision of Dr. Parker, Dr. Eggleston visited site and/or Project offices on 26 to 30 April, 2011, 6–18 June 2011, 6–16 September 2011, 10–22 March 2012, 4–7 April 2012, 7–23 May 2012, 6–22 June 2012, 19–10 July 2012, 17–22 February 2013, 7–27 April 2013, 5–10 August 2013, and 14–15 October 2013.

During these site visits, Dr. Parker and Dr. Eggleston reviewed the current and historical drill hole database, core handling, logging and cutting procedures, density measurements, preparation procedures, assaying quality assurance and quality control (QA/QC), collar surveys and down hole surveys. Discussions on geology and mineralization were held with Duluth and TMM personnel, and field site inspections were performed.

Mr. White visited the site from 9 to 10 July 2013, under the supervision of Mr. Barber. The visit consisted of personal inspection/reconnaissance of the proposed mine site including possible sites for the mine portals. The visit included inspection of the types of terrain that are present within the Maturi Project and the types of surface accesses such as roads, which are currently available. Mr. Radue visited the Project site on July 9 and July 10, 2013. The visit consisted of personal inspection/reconnaissance of the proposed mine site, concentrator site, TSF site, planned utility corridors and primary river crossings, and primary project-related roadways. This visit included site viewing from state and county highways and gravel surfaced forest roads, and various hikes and off-road vehicle travel to explore Project site areas not otherwise accessible.

Mr. Martin visited the Project site between 1 and 2 October 2013. During this visit Mr. Martin discussed the Project with Project geologists. This discussion included visiting some of the drill sites, and visiting the core shed and studying some of the core with the geologists from a mineralogical perspective. He also worked with Duluth and TMM personnel on the design of further metallurgical test programs.

Mr. Malgesini visited the Project site on July 9 and July 10, 2013. The visit consisted of reconnaissance of the TSF site, the mine site and proposed paste plant locations, and the concentrator site. This visit included site viewing of surface conditions from state and county highways and gravel surfaced forest roads. Various hikes and off-road vehicle travel was required to explore the Project site areas not otherwise accessible. Where accessible, the sites for planned utility corridors, primary lake crossings, and primary Project-related roadways were viewed.

Dr. Pierce visited the Project site from 21–22 December 2011, 16–20 January 2012, 13–19 February 2012 and most recently from 20–21 March, 2012. During these site visits, Dr Pierce had discussions on general geology, structural geology, orebody geometry, drilling and logging and available geotechnical data with Duluth and TMM personnel, examined core from a number of boreholes and directed geotechnical logging of core from Maturi and Birch Lake.

Dr. Sterrett was on site from August 14 and 15, 2012. The site visit consisted of personal inspection/reconnaissance of the proposed mine site, observation of core hole drilling and observations of hydrogeological borehole testing.

2.4 Effective Dates

The Report has a number of effective dates as follows:

- Date of supply of database used in resource estimation: 4 February 2014
- Date of Mineral Resource estimate for Maturi: 5 February 2014
- Date of Mineral Resource estimate for Maturi Southwest: 5 February 2014
- Date of Mineral Resource estimate for Birch Lake and Spruce Road: 15 September 2012
- Date of Mineral Reserve estimate: 1 July 2014
- Date of letter regarding taxation considerations that supports the financial analysis: 10 September, 2014
- Date of financial analysis: 20 August, 2014
- Date of supply of latest information on mineral tenure, surface rights and Project ownership: 25 September, 2014

- Date of supply of latest information on Project ownership: 26 September, 2014.

The overall effective date of the Report is taken to be the date of the financial analysis, and is 20 August, 2014.

2.5 Information Sources and References

The key information sources for the Report include:

- AMEC, 2014: Maturi Underground Mine Prefeasibility Study Report: report prepared by AMEC for Twin Metals, AMEC Project No. 173843, 27 May 2014, 324 p.
- Golder Associates Inc., 2014: Tailings and Mine Paste Backfill Pre-feasibility Study, Twin Metals Minnesota Project: report prepared for Twin Metals, March 2014, 7 vols.
- Itasca Consulting Group, 2014a: Geomechanical Analysis of Prefeasibility Mine Design for Twin Metals Minnesota, Maturi Orebody: report 4-2717-10-18 prepared by Itasca for Twin Metals, 14 May 2014, 442 p.
- Itasca Consulting Group, 2014b: Conceptual Level Geomechanical Analysis of Mine Design for Maturi Southwest Orebody: technical memorandum ICG14-2717-26-20TM prepared by Itasca for Twin Metals, April, 2014.
- Itasca, Denver, Inc., 2014: Predictions of Groundwater Inflows into the Maturi Underground Mine: report #1973 prepared by Itasca for Twin Metals, V4, 19 September 2014, 43 p.
- SRK Consulting (U.S.), Inc., 2014: Prefeasibility Mining Study, Twin Metals Project, Minnesota: SRK Project No. 349400.070, June 10, 2014, 249 p.
- Twin Metals Minnesota, 2014: Twin Metals Minnesota Project, Pre-feasibility Study: Internal report prepared by Twin Metals Minnesota, June 2014, 25 vols.

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 TMM and Duluth personnel where required.

2.5.1 Golder

Mr. Matthew Malgesini, the Golder QP, has relied upon input from Golder discipline specialists for use in the sections of the report for which he is responsible. Mr. Malgesini relied upon Mr. Isaac Ahmed, P.Eng of Golder Associates Ltd., for information relating to the paste backfill plant designs and underground distribution systems. He also relied upon Mr. Rens Verburg, PhD, Professional Geochemist, Province of British Columbia, Canada, of Golder Associates Inc., for information on geochemical characterization of tailings and waste. Mr. Malgesini further relied upon Mr. Don Roberts, P.Eng., of Golder Associates Ltd., to provide information relating to ground support recommendations presented in Section 16.1.10.

2.6 Previous Technical Reports

The following technical reports have been filed on the Project by Duluth or Duluth's predecessor companies. Due to the Project history, not all reports will cover the same tenure holdings as this Report.

- Caracle Creek International Consulting Inc, 2004: Independent Technical Report: San Francisco Zinc (Utah), Mahoney Zinc (New Mexico), and Birch Lake PGE (Duluth Complex, Minnesota) Properties, United States of America; 16 April 2004, NI 43-101 Report Prepared by Caracle Creek International Consulting Inc. for Franconia Minerals Corp., 360 p.
- Carghill, D.G., 2005: Technical Report on the Maturi Extension Property, Minnesota, U.S.A.; 30 December 2005, NI 43-101 Technical Report Prepared by Roscoe Postle Associated, Inc. for Wallbridge Mining Company Limited, 90 p.
- Clow, G.G., Cox, J.J., Routledge, R.E., and Hayden, A.S., 2006: Technical Report on the Preliminary Assessment of the Birch Lake and Maturi Deposits, Minnesota, U.S.A.; 20 October 2006, NI 43-101 Technical Report by Scott Wilson Roscoe Postle Associates Inc. for Franconia Minerals Corporation, 175 p.
- Clow, G.G., Hwozdyk, L.R., Routledge, R.E., McCombe D.A. and Scott, K.C., 2008: Technical Report on the Preliminary Assessment on the Nokomis Project, Minnesota, U.S.A.; NI 43-101 Technical Report prepared by Scott Wilson Roscoe Postle Associates Inc. for Duluth Metals Limited, 184 p.
- Clow, G., and Routledge, R.E., 2005: Preliminary Assessment of Mineral Resources of the Birch Lake Property, Minnesota, U.S.A.; 19 November 2005, NI 43-101 Technical Report prepared by Roscoe Postle Associates Inc. for Franconia Minerals Corporation, 93 p.
- Cox, J.J., Routledge, R.E., and Krutzlemann, H., 2009: Preliminary Assessment of the Nokomis Project, Minnesota, U.S.A.; 8 January, 2009, NI 43-101 Technical Report prepared by Scott Wilson Roscoe Postle Associates Inc. for Duluth Metals Limited, 182 p.
- Moreton, C., and Routledge, R.E., 2009: Technical Report on the Mineral Resource Estimate for the Nokomis Deposit on the Nokomis Property, Minnesota, U.S.A.; 10 December 2009, NI 43-101 Technical Report Prepared by Scott Wilson Roscoe Postle Associates Inc. for Duluth Metals Limited, 115 p.

- Parker, H.M. and Eggleston, T.L., 2012a: Maturi, Birch Lake, and Spruce Road Cu-Ni-PGE Projects Ely, Minnesota USA; 27 July 2012, NI 43-101 Technical Report prepared by AMEC E&C Services Inc. for Duluth Metals Limited, 302 p.
- Parker, H.M. and Eggleston, T.L., 2012b: Maturi, Birch Lake, and Spruce Road Cu-Ni-PGE Projects Ely, Minnesota USA; 15 September 2012, NI 43-101 Technical Report prepared by AMEC E&C Services Inc. for Duluth Metals Limited, 301 p.
- Parker, H.M. and Eggleston, T.L., 2014: Maturi, Birch Lake, and Spruce Road Cu-Ni-PGE Projects Ely, Minnesota USA: NI 43-101 Technical Report prepared by AMEC E&C Services Inc. for Duluth Metals Limited, effective date 2 January 2014, 376 p.
- Routledge, R.E., 2004: Review of the Mineral Resources of the Birch Lake Property, Minnesota, U.S.A.; 22 January 2004, NI 43-101 Technical Report prepared by Roscoe Postle Associates Inc. for Franconia Minerals Corporation, 92 p.
- Routledge, R.E., 2006: Technical Report on the Maturi Extension Property, Minnesota, U.S.A.; 31 May 2006, NI 43-101 Technical Report Prepared by Roscoe Postle Associated, Inc. for Duluth Metals Limited, 68 p.
- Routledge, R.E., 2007: Technical Report on the Resource Estimate for the Nokomis Deposit on the Maturi Extension Properties, Minnesota, U.S.A.; 8 August 2007, NI 43-101 Technical Report Prepared by Scott Wilson Roscoe Postle Associates Inc. for Duluth Metals Limited, 112 p.
- Routledge, R.E., 2008a: Technical Report on the Resource Estimate for the Nokomis Deposit on the Maturi Extension Properties, Minnesota, U.S.A.; 18 July 2008, NI 43-101 Technical Report Prepared by Scott Wilson Roscoe Postle Associates Inc. for Duluth Metals Limited, 107 p.
- Routledge, R.E., 2008b: Technical Report on the Resource Estimate for the Birch Lake Property, Minnesota, U.S.A.; 22 August 2008, NI 43-101 Technical Report by Scott Wilson Roscoe Postle Associates Inc. for Franconia Minerals Corporation, 139 p.
- Routledge, R.E., 2009: Technical Report on the Resource Estimate for the Birch Lake Property, Minnesota, U.S.A.; 18 September 2009, NI 43-101 Technical Report by Scott Wilson Roscoe Postle Associates Inc. for Franconia Minerals Corporation, 164 p.
- Routledge, R.E. and Cox, J.J., 2007: Technical Report on the Resource Estimate for the Spruce Road Deposit, Minnesota, U.S.A.; 15 November 2007, NI 43-101

Technical Report by Scott Wilson Roscoe Postle Associates Inc. for Franconia Minerals Corporation, 130 p.

- Routledge, R.E. and Galyen, R., 2010: Technical Report on the Resource Estimate Update for the Birch Lake Property, Minnesota, U.S.A.; NI 43-101 Technical Report by Scott Wilson Roscoe Postle Associates Inc. for Franconia Minerals Corporation, 151 p.
- Routledge, R.E. and Greenough, G.F., 2006: Technical Report on the Mineral Resource Estimate for the Maturi Property, Minnesota, U.S.A.; 30 June 2006, NI 43-101 Technical Report prepared by Roscoe Postle Associates Inc. for Franconia Mineral Corporation, 96 p.

3.0 RELIANCE ON OTHER EXPERTS

The QPs have relied upon the following other expert reports, which provided information regarding mineral rights, surface rights, property agreements, royalties, taxation and marketing sections of this Report as noted below.

3.1 Mineral Tenure, Surface Rights, and Royalties

The QPs have not independently reviewed ownership of the Project area and the underlying property agreements. The QPs have fully relied upon, and disclaim responsibility for, information derived from TMM and legal experts retained by Duluth for this information through the following documents:

- Duluth Metals Limited, 2014: Duluth Metals Limited 43-101 Report: Letter prepared by Duluth Metals Limited, addressed to Mr. John Barber of AMEC, 26 September, 2014
- Baker, V., 2014: Opinion letter: letter from Mr. Vern Baker, President, Duluth Metals Ltd., addressed to Dr. Ted Eggleston of AMEC, dated 31 March, 2014.

This information is used in Section 4.2 of the Report.

The QPs have not independently reviewed the Project mineral tenure and the overlying surface rights. The QPs have fully relied upon, and disclaim responsibility for, information derived from legal experts retained by Duluth for this information through the following documents:

- Fontaine, G.A., 2014: Twin Metals Minnesota LLC, 7 April 2014 (effective March 28, 2014): letter report to Dr. Ted Eggleston from Stinson Leonard Street LLP, 7 April 2014, 6 p.

This information is used in Sections 4.3.1, 4.5 and 4.8 of the Report.

- Fryberger, Buchanan, Smith and Frederic, 2014a: Twin Metals Minnesota, LLC Mineral and Surface Interest Holdings: Letter opinion prepared by Fryberger, Buchanan, Smith and Frederic, P.A., addressed to Dr. Ted Eggleston of AMEC, 2 January 2014, 6 p.
- Fryberger, Buchanan, Smith and Frederic, 2014b: Duluth Metals Limited 43-101 Report: Letter opinion prepared by Fryberger, Buchanan, Smith and Frederic, P.A., addressed to Mr. John Barber of AMEC, 25 September, 2014, 10 p.

This information is used in Sections 4.3.2, 4.4, and 4.7 of the Report.

The QPs have not independently reviewed the Project royalty burden. The QPs have fully relied upon, and disclaim responsibility for, information derived from legal experts retained by Duluth for this information through the following document:

- HollandHart, 2014: Letter Report - Review of Royalty Agreements and Draft 43-101 for Duluth Metals, Inc: letter report prepared by HollandHart, addressed to Mr. John Barber of AMEC, dated 25 September 2014, 3 p. with attachments.

This information is used in Section 4.4, 4.9, and 4.10 of the Report.

The information as indicated in this sub-section 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.

3.2 Markets

The QPs have not independently reviewed the marketing information. The QPs have fully relied upon, and disclaim responsibility for, information derived from experts retained by Duluth for this information through the following document:

- Wood Mackenzie, 2014: Market Analysis for NI 43-101 Final Report for Duluth Metals Limited: report prepared by Wood Mackenzie for Duluth, May 2014, 14 p.

This information is used in Section 19 and in support of the financial analysis in Section 22 and the Mineral Reserves estimate in Section 15.

Metals marketing is a specialized business requiring knowledge of supply and demand, economic activity and other factors that are highly specialized and requires an extensive database that is outside of the purview of a QP.

The QPs consider it reasonable to rely upon Wood Mackenzie for marketing information as the company is a global leader in commercial intelligence for the energy, metals and mining industries, and provides independent analysis and advice on assets, companies and markets to these industries.

3.3 Taxation

The QPs have fully relied upon, and disclaim responsibility for, information supplied by Duluth staff and experts retained by Duluth for information related to taxation as applied to the financial model as follows:

- PriceWaterhouseCoopers, 2014: Tax Narrative for Twin Metals NI43-101: note prepared by PriceWaterhouseCoopers for Duluth Metals Limited, 18 August, 2014, 3 p.

This information is used in support of the financial analysis in Section 22, and the Mineral Reserve estimation in Section 15.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Maturi, Maturi Southwest, Birch Lake, and Spruce Road deposits are located east to southeast of Ely, Minnesota (refer to Figure 2-1).

The Maturi deposit is located in Lake County, Townships 61N and 62N, Range 11W in the Kangas Bay and Bogberry Lake 7.5' quadrangles. The deposit is centered at approximately:

- North latitude 47° 47′ 0″; west longitude 91° 42′ 30″
- UTM coordinates Zone 15, 595,516E, 5,295,082N (NAD 27 CONUS)
- UTM coordinates Zone 15, 595,500E, 5,295,300N (NAD 83).

The Maturi Southwest deposit is located in Lake County, Township 61 N, Range 11W, S 6 in the Kangas Bay 7.5' quadrangle. The deposit is centered at approximately:

- North latitude 47° 47' 13"; west longitude 91° 47' 08"
- UTM coordinates Zone 15, 591,072E, 5,292967N (NAD 27 CONUS)
- UTM coordinates Zone 15, 591,070E, 5,293,200N (NAD 83).

The Birch Lake deposit is in Lake and St. Louis counties, approximately 125 km north–northeast of Duluth, Minnesota, in the Kangas Bay and Babbitt NE 7.5' quadrangles. The deposit is centered approximately at:

- North latitude 47° 41' 49"; west longitude 91° 47' 30"
- UTM coordinates Zone 15, 589,700E, 5,285,200N (NAD 27 CONUS)
- UTM coordinates Zone 15, 589,684E, 5,285,418N (NAD 83).

The Spruce Road deposit lies for the most part on Federal Lease US ES01353 located in northern Minnesota, Lake County, Townships 62N and Range 10W and 11W in the Bogberry Lake 7.5' quadrangle. The deposit is centered approximately at:

- North latitude 47° 50' 09"; west longitude 91° 40' 00"
- UTM coordinates Zone 15, 599,800E, 5,298,700N (NAD 27 CONUS)
- UTM coordinates Zone 15, 599,784E, 5,298,918N (NAD 83).

4.2 Ownership and Company Structure

4.2.1 Duluth

AMEC was provided with a copy of the latest Duluth corporate ownership structure (Figure 4-1), which outlines the holdings and cross holdings of the various Duluth interests in the Project and the relationships of the subsidiaries to the Canadian-listed parent entity, Duluth Metals Limited.

4.2.2 TMM

4.2.2.1 Ownership

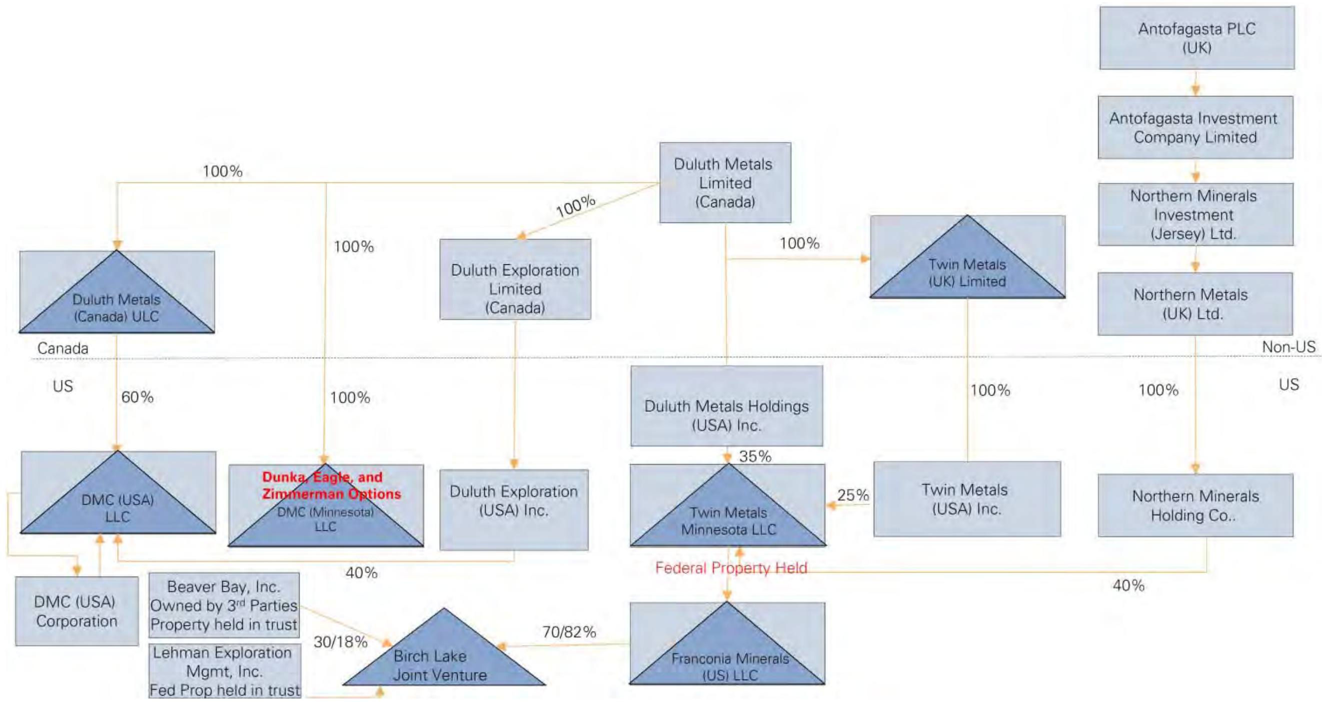
AMEC was provided with documentation that supports that Twin Metals Minnesota LLC (TMM) is a limited liability company duly formed, validly existing, and in good standing under the laws of the State of Delaware. Since 2010, TMM has been operated as a joint venture between Antofagasta and Duluth (Figure 4-2, and see also Section 2.1).

TMM's operations and activities are governed by a Participation Agreement (signed 21 July, 2010 and amended 20 December 2010, 20 April 2011, 14 July, 2011, 26 August 2011, 31 July 2012, and 20 June 2013), and by a board of directors (the TMM Board) consisting of individuals appointed by Duluth and Antofagasta. The Board may also appoint committees, in particular, a TMM Technical Committee. The Participation Agreement contains a number of key terms that govern the relationships among the TMM members, Project funding, ownership, and other significant matters.

On 3 July, 2014, a formal 25% Option Termination Notice (the notice) was provided to Duluth by Antofagasta. By delivering the notice, Antofagasta no longer has the right to acquire 25% of TMM from Duluth after the delivery of a bankable feasibility study and the permitting of the Project. In addition, Duluth becomes the operator of the Project and controls the TMM Board and the TMM Technical Committee by having three members and Antofagasta having two members on each. The notice results in the disproportionate funding of the joint venture ceasing, and Duluth is now responsible for its proportionate share, or 60%, of future joint venture expenditures.

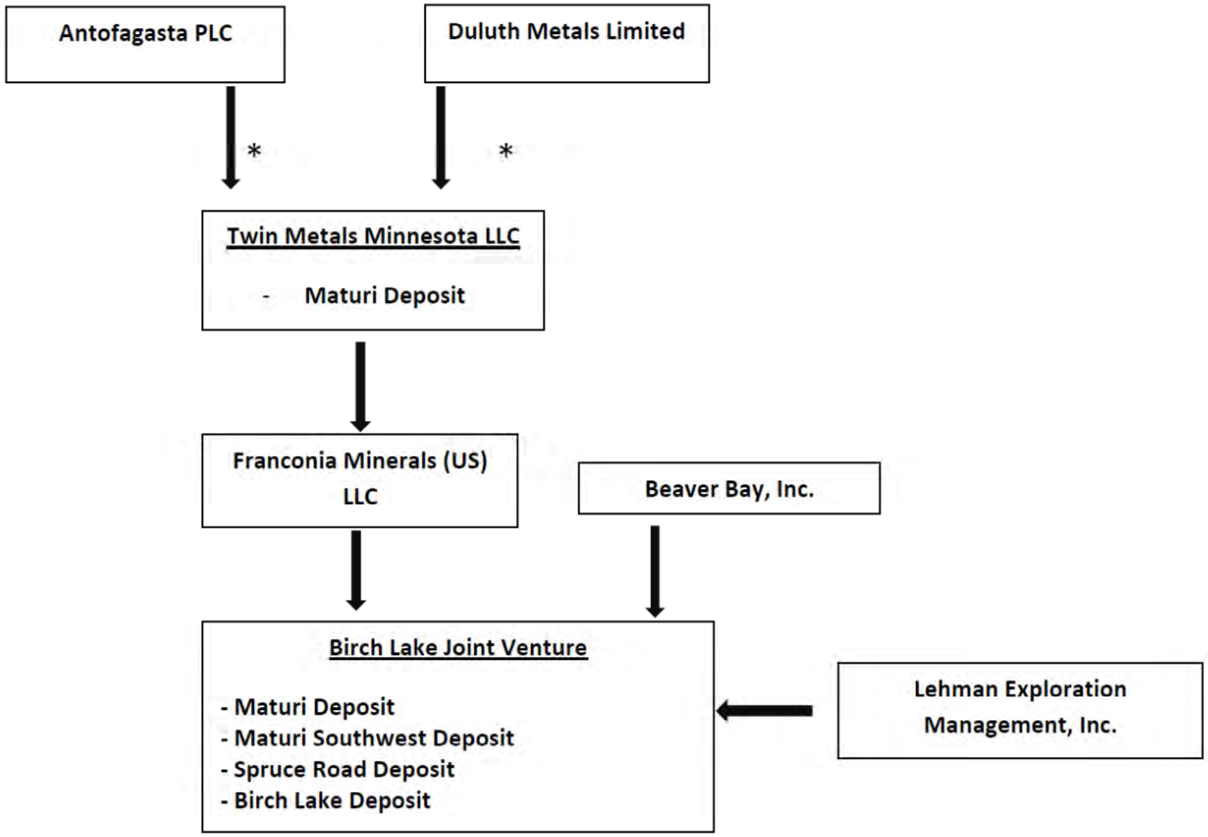
On 28 September, 2012, Duluth and Antofagasta had entered into a Secured Bridge Loan Agreement (the Loan Agreement) whereby an Antofagasta subsidiary company made a secured revolving bridge loan (the Bridge Loan) available to Duluth. The Loan Agreement contains some non-financial terms and conditions, including a list of events of default. The full $10 million proceeds of the Bridge Loan were invested in the joint venture on the same date, to fund Duluth's share of a cash call. The Bridge Loan is subject to interest accrual.

Figure 4-1: Duluth Corporate Structure (current as at 14 July, 2014)



Note: Figure courtesy Duluth 2014.

Figure 4-2: Project Ownership Flowsheet



Note: Figure courtesy Duluth, 2014. Note Figure 4-2 is a simplification of the holdings shown in Figure 4-1. * indicates this portion has been abbreviated and does not show the full ownership chain.

Duluth has a right on, or before December 30, 2014, to purchase Antofagasta's 40% equity position at a price equal to Antofagasta's previously-invested costs, which are approximately $219.8 million. If this right is exercised, Duluth is required to repay the outstanding principal amount of the Bridge Loan, including all accrued and unpaid interest, in cash. If Duluth does not exercise the purchase right, Antofagasta will continue to own 40% of TMM, and Duluth will be required to repay the Bridge Loan, plus all accrued and unpaid interest, at its option, in cash or Duluth shares, before Antofagasta's ownership interest is recalculated and diluted as provided in the Participation Agreement.

4.2.3 Franconia Minerals (US) LLC

Franconia Minerals (US) LLC is a wholly owned subsidiary of TMM (refer to Figure 4-2). Franconia holds a 70% participating interest in the Birch Lake Joint Venture. The other 30% participating interest is held by Beaver Bay, Inc., an independent third-party entity. By virtue of its 70% ownership share, Franconia controls the decision-making of the Birch Lake Joint Venture and the management committee appointed to oversee the Birch Lake Joint Venture's operations.

The operations and activities of Franconia are governed by the Combined Member Control and Operating Agreement, as amended from time to time (Franconia Operating Agreement). Under the Franconia Operating Agreement, TMM is the sole member, selects Franconia's officers who are responsible for the company's day-to-day operations, and appoints Franconia's Board of Governors that oversees the officers and the company operations. Currently, Franconia's officers and governors are all members of TMM's management.

Under the terms of the Birch Lake Joint Venture agreement, Franconia may exercise an option to acquire a further 12% participating interest in the Birch Lake Joint Venture, which would result in Franconia holding an 82% participating interest and Beaver Bay, Inc. holding an 18% participating interest.

The Birch Lake Joint Venture agreement contains provisions relating to the relationship between Franconia and Beaver Bay, their respective rights and obligations, and other matters relating to mineral rights and operations.

As described in Section 4.4, various mineral interests are held by Franconia and through the Birch Lake Joint Venture.

4.2.4 DMC (Minnesota) LLC

Duluth's wholly owned subsidiary, DMC (Minnesota) LLC, a Delaware limited liability company (Dunka Holdco), holds interests in an option agreement for property rights related to the Dunka open pit (Dunka Properties). The Participation Agreement requires Dunka Holdco to transfer to TMM all of the Dunka Properties upon the satisfaction of certain conditions.

4.2.5 Additional Duluth Property Interests

Duluth originally retained approximately 31,000 acres of mineral interests on exploration properties adjacent to and near-by the joint venture holdings. This acreage changes from time to time as new properties are acquired and explored and in some cases abandoned when the circumstances warrant.

In addition to actively participating in the joint venture on Nokomis (now Maturi); Duluth undertakes exploration programs on its independently-retained exploration properties.

4.3 Mineral Title in Minnesota

4.3.1 Introduction

Land in Minnesota is held by a combination of private, state and federal ownership, and land is subject to typical United States split-estate holdings, where the surface owner(s) may be different from the sub-surface owner(s).

Locations for mineral leases and other property locations are normally described in the United States Public Land Survey System of township, range, section, and section subdivisions. There are some minor exceptions that do not relate to the Public Land Survey System descriptions, such as land under water and islands.

4.3.2 Relevant Federal Legislation

Originally established from public domain lands, the Superior National Forest was designated and approved by Presidential Proclamation No. 848 in 1909 by President Theodore Roosevelt. It encompasses more than three million acres of land in northeast Minnesota. Subject to applicable laws and regulations, certain areas within the Superior National Forest are open to commercial development, including mining. The proclamation establishing the Superior National Forest “reserved” the public domain lands from the General Mining Law of 1872. While the General Mining Law of 1872 provides for a claim system for federal mineral tenure acquisition, the Superior National Forest is regulated under different federal laws providing for a permitting and leasing system. Hardrock mineral leasing is available on both public domain and acquired lands in the Superior National Forest. The Bureau of Land Management (BLM) is the agency primarily responsible for overseeing this permitting and leasing system and promulgating regulations to establish its regulatory guidelines.

Under the BLM regulations, a mining company may apply for prospecting permits, which have an initial two-year term and may be renewable for up to an additional four years. These prospecting permits can be converted to preference right leases, a type of federal mineral lease, upon satisfying all regulatory requirements. Under the BLM regulations, the initial term for preference right leases may not exceed 20 years, with the possibility of successive 10-year renewals. A preference right lease includes the right to develop and construct a mine under the terms thereof, but additional permits are required before work can commence. Subject to applicable laws and regulations, BLM has discretion as to whether to issue or renew any prospecting permit and any preference right lease, as well as discretion with respect to the terms and conditions to be included in any such prospecting permits and preference right leases. Issuance and renewal of prospecting permits and preference right leases also are subject to review by the United Sates Forest Service (USFS) under applicable federal law. Additionally, before prospecting permits and preference right leases may be issued or renewed, federal agencies must complete requirements for environmental review under the National Environmental Policy Act (NEPA), the National Historic Preservation Act (NHPA), and the Endangered Species Act (ESA). In some cases, consultation with tribal governments may be requested.

Proposed uses of lands subject to prospecting permits or preference rights leases have been issued are also subject to:

- Requirements for agency approvals of such uses
- The terms and conditions established in the prospecting permits or preference right leases, as applicable
- Rental fees and royalties
- The requirements of the above-referenced federal statutes and regulations authorizing such permits and leases or requiring environmental review and consultations
- Additional requirements as described in Section 4.5 of this Report.

4.3.3 Relevant State Legislation

State leases for nonferrous metallic mining are issued by the Minnesota Department of Natural Resources (DNR) and may be held for up to 50 years. These leases allow a mining company to engage in mineral exploration and mineral development located on the state-owned property, subject to compliance with all laws and issued permits. An operating mining company must pay a production royalty in addition to lease payments.

At the mineral development stage, a “permit to mine” is required for any new nonferrous metallic mineral mine in addition to the mining lease. This is required for mining of all nonferrous metallic mineral interests, irrespective of whether the ownership is state, federal, or private. A permit to mine may be issued for whatever term the DNR deems necessary for the completion of the proposed mining operation, including reclamation or restoration.

4.4 Mineral Tenure

4.4.1 Introduction

The land tenure package held by TMM under fee, lease, permit, application and option agreement pertinent to the infrastructure proposed in the PFS is shown in Figure 4-3.

TMM has the benefit of various mineral interests including fee lands, state leases, federal leases, federal prospecting permits, federal prospecting permit applications, preference right lease applications and private leases, as summarized in Table 4-1. The mineral interests that specifically pertain to the planned mining operation are highlighted in Table 4-2.

The mineral interests on a Project-wide basis are shown in Figure 4-4 and on a proposed mining operational basis in Figure 4-5. Information included in Table 4-2 and shown on Figure 4-5 is a subset of the information shown in Table 4-1 and Figure 4-4.

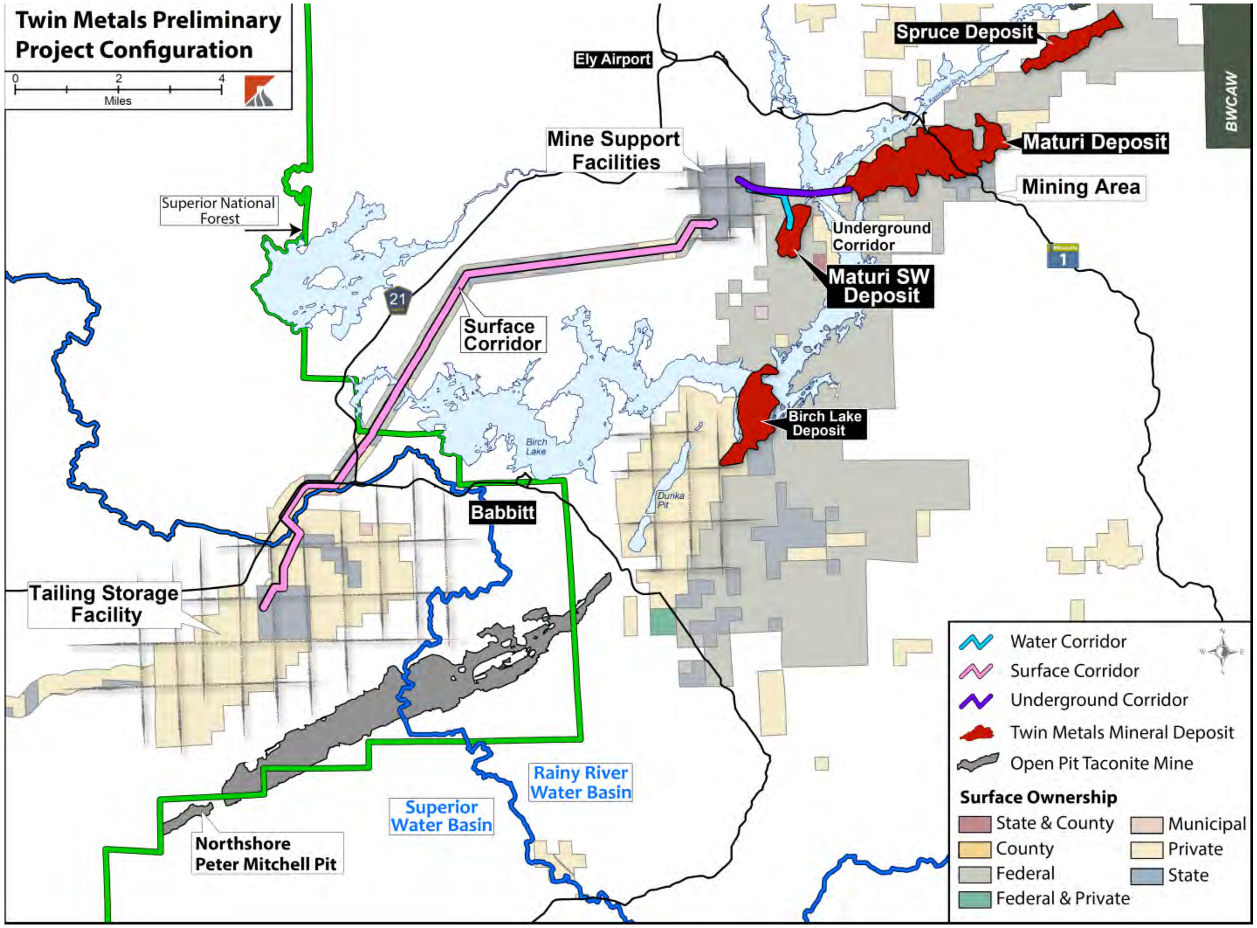
When not held in TMM's own name, TMM's mineral interests are held by Franconia or through the Birch Lake Joint Venture (Figure 4-6).

A description of TMM's company structure is included in Section 4.2.

Subject to certain exceptions, TMM's mineral interests under private mineral leases, state and federal leases, and federal prospecting permits are insured pursuant to title insurance policies issued by First American Title Insurance Company on August 4, 2010 and August 31, 2011 as policy nos.: NCS-428640 (Nokomis; now Maturi) and NCS-471210 (Franconia) (note that TMM's mineral interests lying beneath the beds of reservoirs or other bodies of water and all federal prospecting permits issued after August 31, 2011 are not insured by the above-referenced title polices).

Of the total of about 25,000 acres of federal, state, and private minerals controlled together by the TMM corporate group, TMM and Franconia individually hold about 9,590 acres (38%) and 15,725 acres (62%), respectively. With respect to the 1,574 acres of mineral permits and leases that give the TMM corporate group rights to the minerals that would be mined within the Project mine design, TMM and Franconia individually hold about 1,064 acres (68%) and 510 acres (32%), respectively.

Figure 4-3: Project Ground Holdings in Relation to Infrastructure Proposed in PFS



Note: Figure courtesy Duluth, 2014. Infrastructure shown in the figure is proposed and not constructed. Spruce deposit noted on the figure is the Spruce Road deposit.

Table 4-1: Summary of TMM Mineral Interests*

Type	Number	Net Acres	Hectares
Federal Mineral Leases**	2	4,698.83	1,901.51
Federal Prospecting Permits	10	7,755.11	3,138.45
Federal Prospecting Permit with Preference Rights Lease Application	3	1,058.03	428.18
Federal Prospecting Permit Applications	4	699.30	288
State Mineral Leases	27	5,612.24	2,271.16
Private Mineral Leases	18	4,770.70	1,930.60
Fee Minerals	N/A	521.92	211.21
Total	64	25,116.13	10,169.11

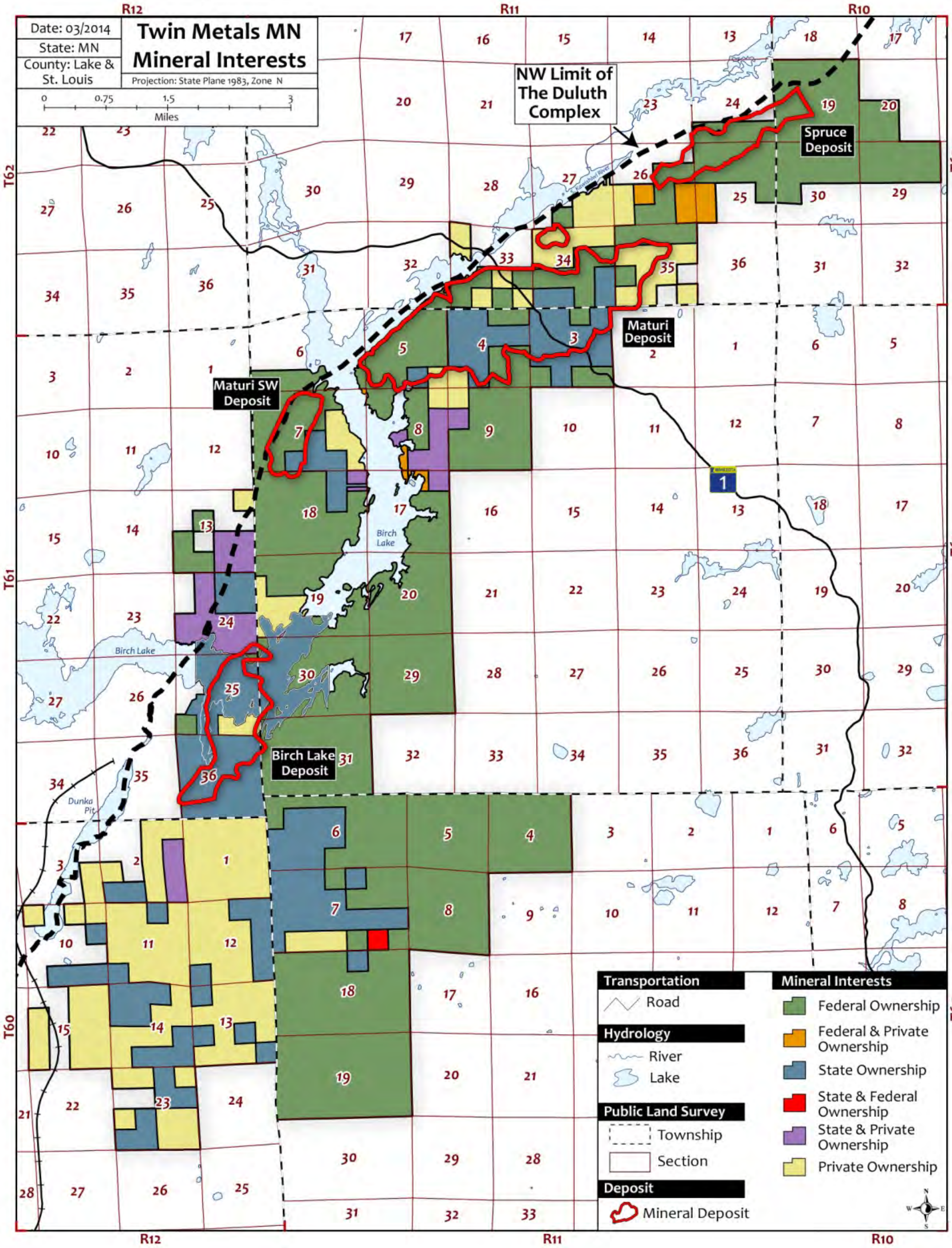
*In some instances, TMM holds undivided fractional mineral interests.
**Federal Mineral Leases 1352 and 1353 have been submitted to the BLM for third renewal.

Table 4-2: Subset of TMM Mineral Interests in Proposed Mining Area

	Project Mineral Right	TMM Company	Lease/Permit Number	Approx. Total Acres	Approx. Acres in Proposed Operational Areas
Federal	BLM Preference Right Lease	Franconia	MNES 1352	2,610	489
	BLM Preference Right Lease Application	TMM	MNES 50652 MNES 50846	1,000	245
State	DNR Nonferrous Metallic Mineral Lease	TMM	MM-9755	460	201
	DNR Nonferrous Metallic Mineral Lease	TMM	MM-9756	160	153
	DNR Nonferrous Metallic Mineral Lease	TMM	MM-9764	350	316
	DNR Nonferrous Metallic Mineral Lease	TMM	MM-9828	40	22
	DNR Nonferrous Metallic Mineral Lease	Franconia	MM-10206-N	160	20
Private	RGGS Mineral Lease	TMM	N/A	560	122
	Maki, Foster, & Adolfson Mineral Leases *	TMM	N/A	160	6

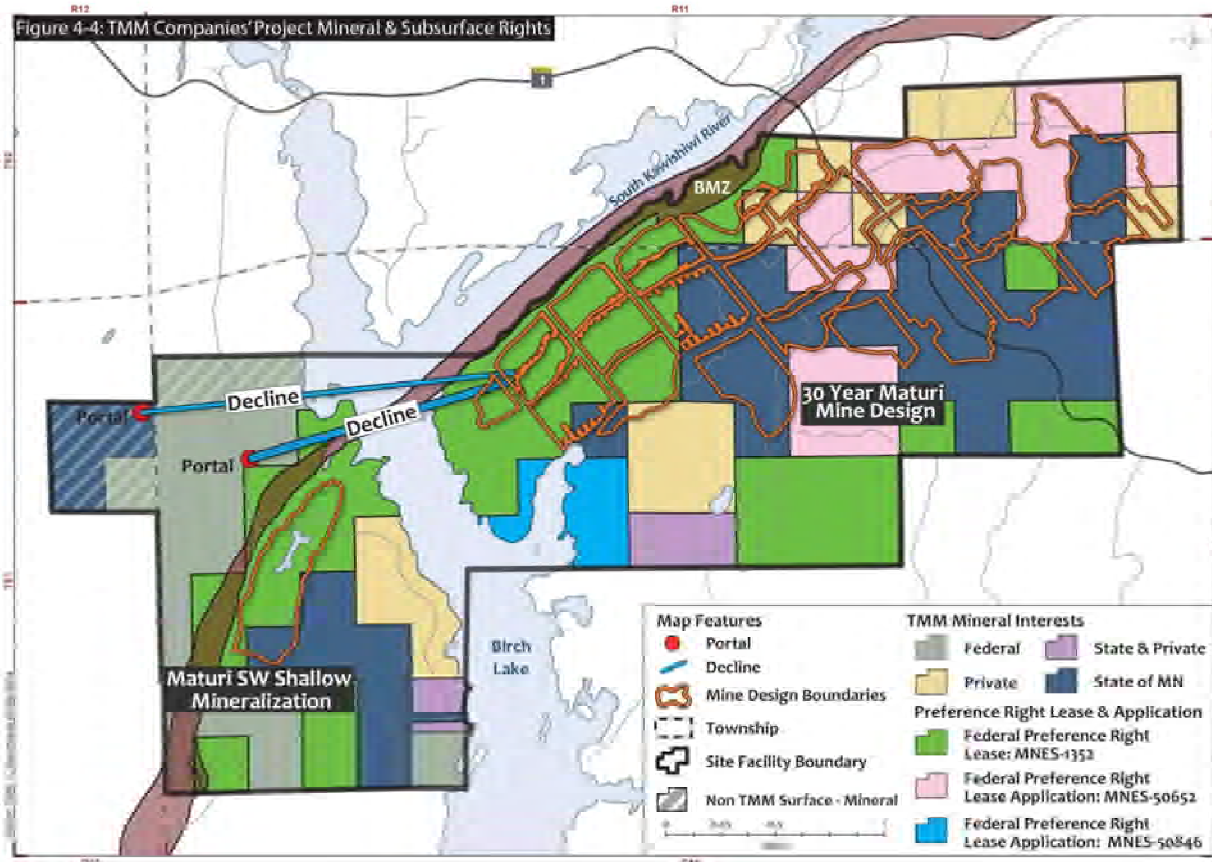
Note: * Minerals subject to these leases are fractionalized. TMM holds a majority right in the form of undivided fractionalized interest.

Figure 4-4: TMM Mineral Interest Map showing Mineral Ownership



Note: Figure courtesy Duluth, 2013. North is to top of plan. Spruce deposit noted on the figure is the Spruce Road deposit.

Figure 4-5: Subset TMM Mineral Interest Map showing Mineral Ownership within Proposed Mining Area



Note: Figure courtesy Duluth, 2014. Map north is to top of plan. Infrastructure shown on plan is proposed.

Figure 4-6: TMM and Franconia Mineral Interests within the Birch Lake Joint Venture Area of Interest

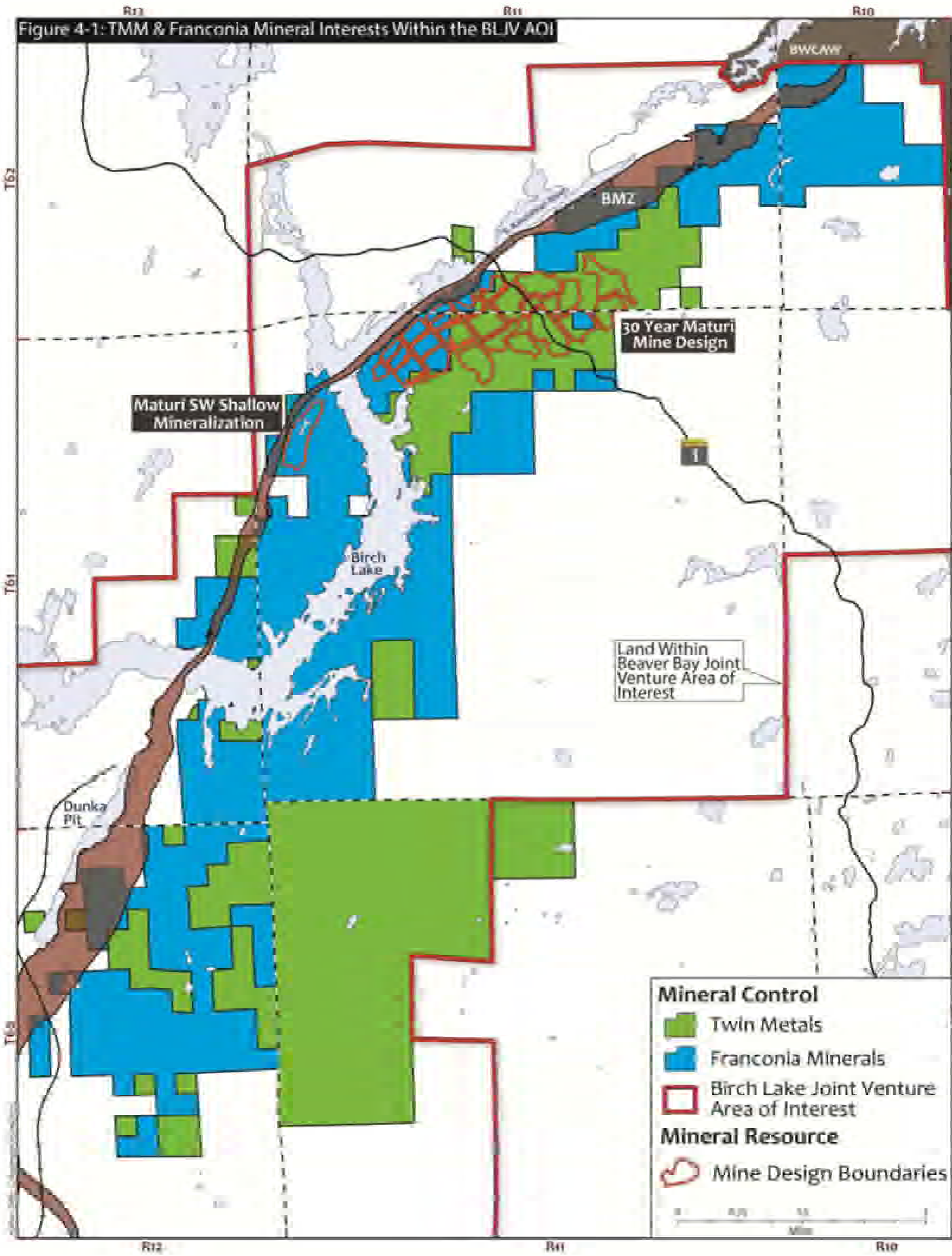


Figure courtesy Duluth, 2014. Map north is to top of plan. Mine layout is that proposed in this Report.

4.4.2 Current Mineral Interests Status—Federal Mineral Leases

TMM holds rights to federal lease nos. MNES-01352 and MNES-01353, dated June 1, 1966, as part of the Birch Lake Joint Venture Agreement dated June 18, 2008. Figure 4-7 shows the location of these leases, which total approximately 4,698.83 acres. About 489 acres of federal lease no. MNES-01352 will be included in the future mining operations.

Royalties and carrying costs vary by lease. The lease-by-lease details for the federal leases are included as Appendix A. No annual work requirements exist, but monthly periodic reporting of results to the BLM is required.

Annual rentals of $1 per acre are required until production is achieved. Thereafter, annual minimum royalty of $10.00 per acre is required during each renewal period of the leases. The minimum royalty may be waived, reduced, or suspended at the discretion of the BLM.

The base royalty for the federal mineral leases is 4.5% of the “gross value” of the minerals mined and shipped to the concentrating mill. The base royalty is subject to adjustment by the BLM during renewal periods under the terms of the leases. “Gross value” is defined as one-third of the market prices of a quantity of fully-refined copper and of a quantity of fully-refined nickel equal to the respective quantities of unrefined copper and unrefined nickel contained in said minerals shipped to the concentrating mill.

To compensate the lessor for associated products[2], there is an additional royalty of 0.3% of the gross value of a quantity of fully-refined copper and of a quantity of fully-refined nickel equal to the respective quantities of unrefined copper and unrefined nickel contained in said minerals shipped to the concentrating mill. The leases require the payment of this additional royalty irrespective of whether any associated products are produced.

There is a further additional royalty of 1% of the gross value of "associated products" if the value of such products exceeds 20% of the aggregate market price as fully-refined metals of the quantity of copper and nickel contained in the minerals mined under the leases and shipped to the concentrating mill. Following a lease year in which the 1% additional royalty has been paid, if the value of such products exceeds 30% of the

[2] “associated products” shall mean (i) the fully-refined chemical elements (other than copper and nickel) not further processed, and (ii) end products containing such elements produced by the Lessee (prior to full refining) for their value as such (other than products valuable chiefly by reason of their copper and nickel content), which are, in either case, recovered by the Lessee from minerals mined under this lease and sold or used by the Lessee during the lease year for which additional royalty, if any is due, and the fross value of such products shall be taken to be the aggregate of the market prices of the respective quantities of associated products so sold or used by the Lessee.

aggregate market price, the additional royalty will be subject to renegotiation by TMM and the BLM.

Advance minimum royalty payments have been and continue to be made annually in the amount of $14,180.00 pursuant to an agreement with Fredrick S. Childers, Roger V. Whiteside and other individuals dated June 30, 1952 as amended by a supplemental agreement dated August 9, 1954 (Childers–Whiteside Agreement). Additional minimum royalty payments have been and continue to be made quarterly in the amount of $1,622.50 pursuant to an agreement with E.J. Longyear Company dated June 25, 1953 (Longyear Agreement).

The Project also includes about 15 acres of Federal subsurface necessary for construction and operation of the proposed primary and secondary declines. TMM does not currently hold sufficient Project subsurface rights to construct and operate the declines, but does have exclusive rights to these federal minerals and subsurface through a Prospecting Permit application.

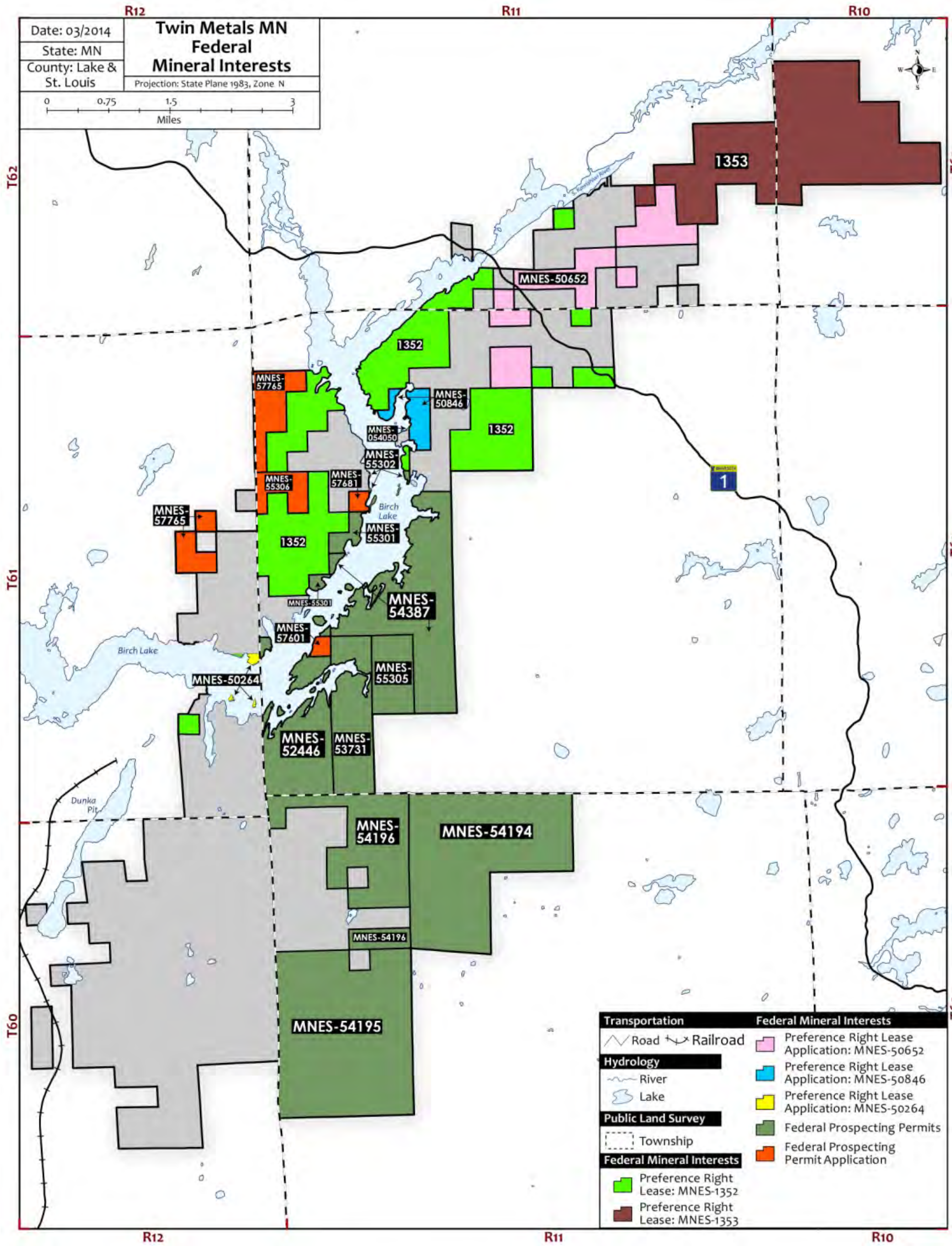
TMM will eventually need to negotiate a separate lease with the BLM in relation to acquiring the necessary subsurface non-mineral construction rights.

4.4.3 Current Mineral Interests Status—Federal Prospecting Permit Applications, Permits, and Preference Right Lease Applications

TMM has the benefit of 10 federal prospecting permits and four federal prospecting permit applications as well as three preference right lease applications for a total of approximately 9,512.44 net acres. Figure 4-4 and Figure 4-7 included the location of the federal prospecting permits, prospecting permit applications and preference rights lease applications. Details of the terms of federal prospecting permit applications, prospecting permits, and preference rights lease applications are included in Appendix A.

Under the standard property advancement procedures for federal prospecting permits, TMM is required to convert its federal prospecting permits to a preference rights lease in order to retain and further explore and develop the properties. According to federal regulations, in order to obtain a preference rights lease, the applicant must hold a federal prospecting permit for the area it wants to lease, apply for a preference rights lease, submit the first year annual lease payment, provide information required as stated in the U.S. Code of Federal Regulations, including maps, a proposed mining and processing approach, a description of salable products and markets, utilities, and infrastructure in the area, and the applicant must demonstrate that it has discovered a valuable deposit covered by its prospecting permit.

Figure 4-7: Federal Mineral Interest Map



Note: Figure courtesy Duluth, 2014.

A valuable deposit is principally determined by the geological assessment of the mineral deposit, detailing the type and extent of the work programs exploration (including drill logs and other exploration results) that have occurred on the lands covered by the federal prospecting permit as well as the exploration on adjacent lands both before and during the prospecting permit term.

TMM holds three federal prospecting permits, MNES-50264, MNES-50652, and MNES-50846, for which a federal preference right lease application has been submitted. Of the total preference right lease application area, about 250 acres are directly required for mine design.

As appropriate, TMM will continue to submit its applications for preference rights leases on its federal prospecting permits in accordance with federal regulations and specific application dates.

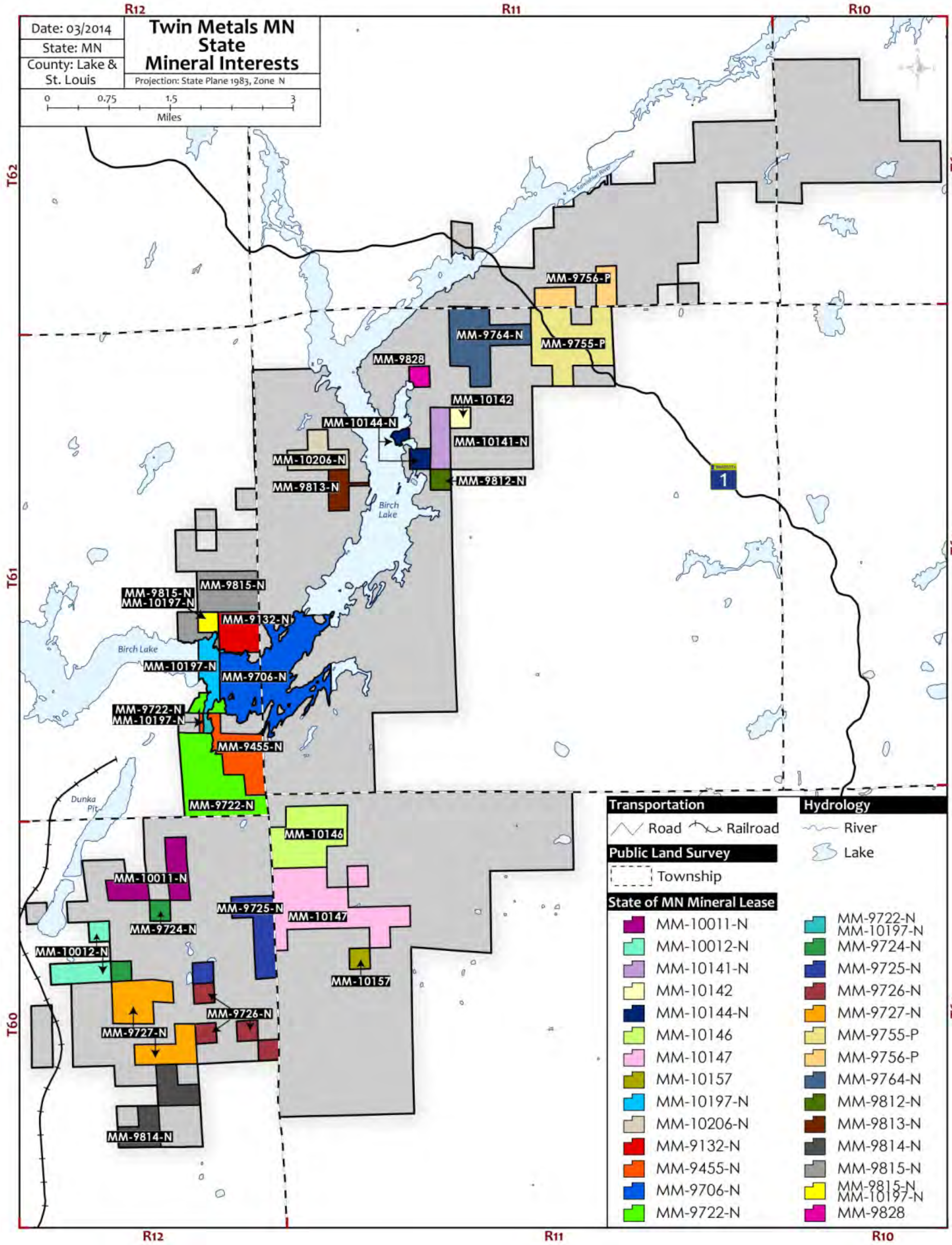
Royalties on preference rights leases will be negotiated at the time the federal prospecting permits are advanced to preference rights leases.

4.4.4 Current Mineral Interests Status—State Leases

State leases to explore for, mine and remove metallic minerals are held for a period of 50 years. Rights conveyed in these leases exclude the extraction of iron ore, taconite ores, coal, oil, gas, and other liquid or gaseous hydrocarbons, which are either reserved by the State of Minnesota or are covered under separate state leases involving third parties. TMM has the benefit of 27 state leases (state leases) for a total of approximately 5,612.24 net acres (Figure 4-8). Details of the State leases are included in Appendix A.

Five of these state mineral leases, which are administered by the DNR, govern the approximately 820 acres of state minerals within the mine design. The Project also includes about 10 acres of state subsurface underlying Birch Lake necessary for construction and operation of the two proposed declines. TMM does not currently hold sufficient state subsurface rights for the Project to construct and operate the declines. However, it is the practice of the state to not grant mineral leases for land under water unless the applicant holds the mineral interests on the contiguous land and that they have demonstrated the mineralization for the land portion continues under the subject body of water. In this case, TMM holds mineral interests on both sides of the land abutting the underwater portion, covering the 10 acres.

Figure 4-8: TMM State of MN Mineral Lease Map



Note: Figure courtesy Duluth, 2014.

The State has a standard lease for non-ferrous minerals such that in n Minnesota, an operating mining company pays a production royalty in addition to lease payments and applicable taxes. The royalty consists of a base rate, and in some cases, an additional bid rate. State leases also contain a royalty escalation clause that increases the base royalty as the net return value per ton of raw ore increases.

The State of Minnesota has an option to cancel a mineral lease after the end of the 20th year if, by that time, a lessee is not actively engaged in mining ore under the lease from the mining unit, a mine within the same government township as the mining unit or an adjacent government township and has not paid at least $100,000 to the state in earned royalty under a metallic mineral lease in any one calendar year. The state must exercise that option within the 21st year of the lease. If the state does not cancel within the 21st year, the lessee has until the end of the 35th calendar year to meet the conditions. If the lessee has not met the conditions by the end of the 35th year, the state has another window to cancel the lease during the 36th calendar year of the lease. Two state leases are beyond their 21st calendar year, but the State of Minnesota did not exercise its right to cancel, and TMM now has until the end of the 35th calendar year to commence production and pay royalties.

4.4.5 Current Mineral Interests Status—Private Leased Lands

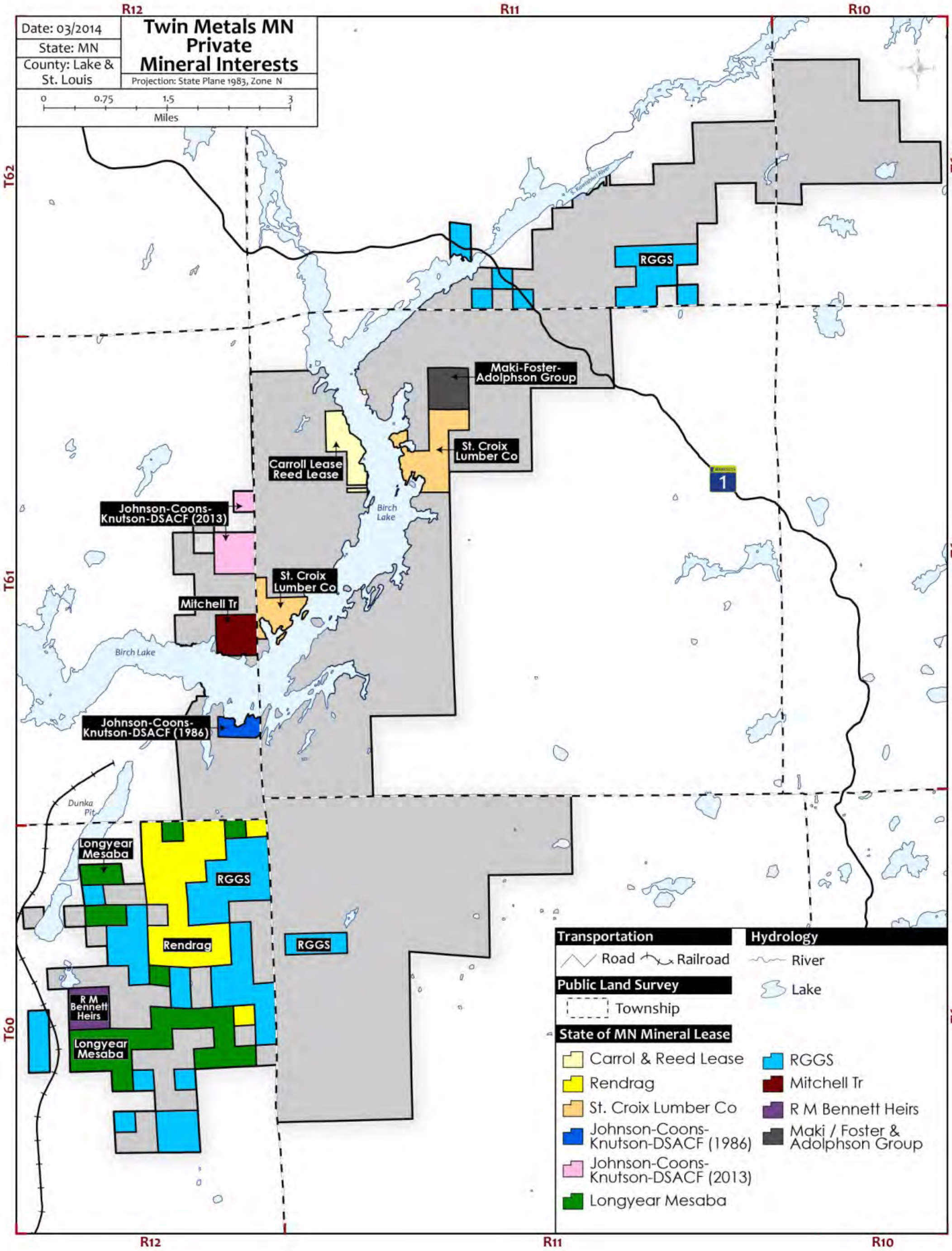
TMM currently has benefit of 18 mineral leases with private parties that cover approximately 4,770.70 net acres (Figure 4-9). Four of these leases are located within the mine design area.

The provisions and terms of each lease are specific to the individual leases. The terms, including initial and renewal terms, range from 40 to 50 years. The surface rights are owned either by TMM, its affiliates, the state or federal government, or private parties. The private leased lands are leased in an "as is" condition to TMM for the purposes of exploring, prospecting, drilling and test pitting the properties and grant TMM the sole and exclusive right to mine and extract and to carry on mining, milling and refining operations with respect to all mineral substances of a metalliferous nature. In most leases, hydrocarbons and taconite deposits are reserved to the lessor.

Royalties are variable by lease. Some of the properties contain a royalty escalator that increases royalties as the net return value per ton of raw ore increases.

Details of the private party leases are included in Appendix A.

Figure 4-9: TMM Private Mineral Lease Map



Note: Figure courtesy Duluth, 2014.

4.4.6 Current Mineral Interests Status—Fee Mineral Interests and Fee Surface Interests

TMM has the benefit of fee mineral ownership of approximately 521.92 net acres. Additionally, TMM has the benefit of fee ownership of, or an option to acquire, 19,361.61 net acres of surface lands that do not include mineral rights, though some surface lands overlie TMM-held mineral interests. Locations of fee mineral interests are tabulated in Table 4-3 and are shown in Figure 4-10.

4.4.7 Current Mineral Interests Status—Minnesota Power Purchase

TMM has purchased fee ownership of 1,420.6 acres from Minnesota Power. These lands are included in Appendix A. Figure 4-11 shows the locations of these lands.

4.5 Surface Rights, Surface and Subsurface Use, and Access

All surface rights for Maturi and Maturi Southwest are either included with the state mineral leases, federal prospecting permits/leases or are privately owned by TMM, with the exception of certain lands bordering the shoreline of Birch Lake.

For lands subject to the regulatory framework described in Section 4.3.2, use of the surface on federal land is provided for in the federal prospecting permits or preference right leases where the surface and mineral estates are held by the federal government. Surface rights on state land are provided for in the state mineral leases where the surface and mineral estates are both held by the State of Minnesota. Issuance of such state mineral leases is governed by statutes and regulations enacted by the State of Minnesota.

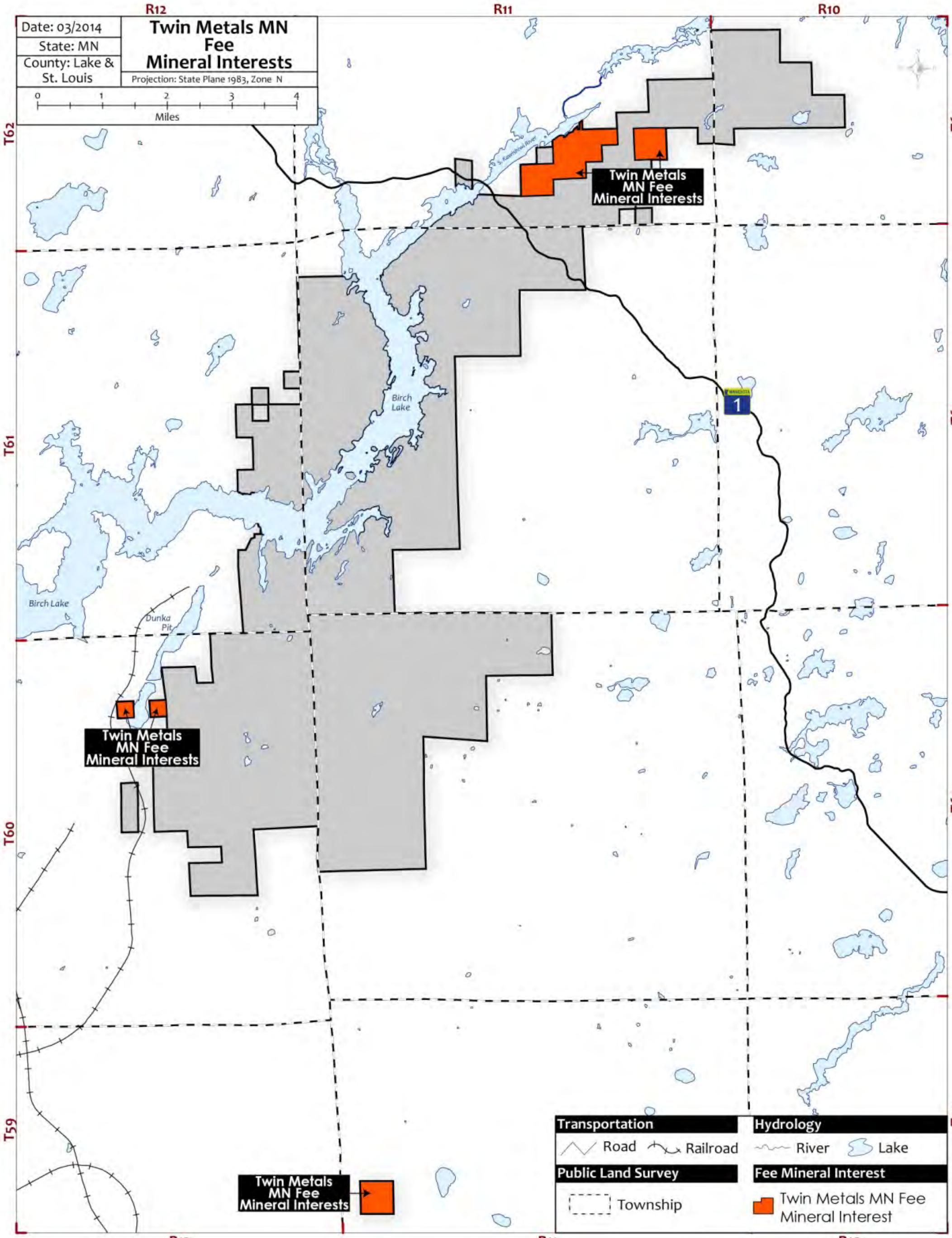
In instances where the surface estate is held by a private party and the State of Minnesota owns the mineral estate, the state government may issue a mineral lease for certain uses as authorized by state statutes and regulations, but may also require notice to the surface owner. In those areas of state-owned minerals within the proposed mine development area, all surface rights are held by the state and included with the mineral lease or are fee lands owned by TMM.

Use of the surface of federal or state lands is subject to approval by the applicable regulatory agencies. Such use is also subject to the terms and conditions provided in the federal prospecting permits, federal preference right leases, or the state leases, as applicable, and the applicable federal and state mineral statutes and regulations.

Table 4-3: Summary of TMM Fee Mineral Interests

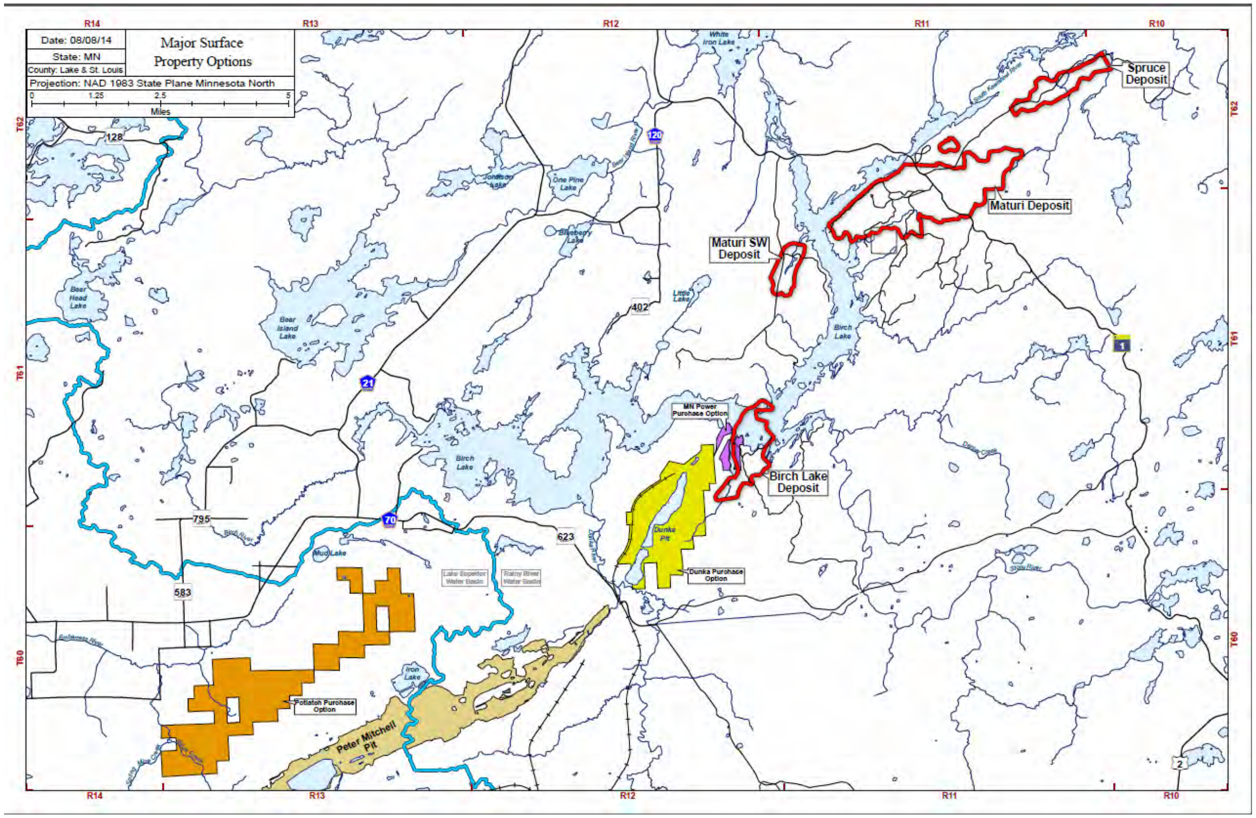
County	Section	Twp	Range	Surface Owner	Net Acres	Comments
Lake	18	59	11	Franconia Minerals (US) LLC	160	
	19	59	11	Franconia Minerals (US) LLC		
Lake	25	62	11	USA	80	Fee mineral interest indicated is an undivided ½ interest
	26	62	11	USA		
Lake	34	62	11	USA	80	Fee mineral interest indicated is an undivided ½ interest
Lake	26	62	11	USA	161.92	Fee mineral interest indicated is an undivided ½ interest
	27	62	11	USA/Private		
	34	62	11	USA		
St. Louis	10	60	12	Twin Metals Minnesota, LLC	40	Mineral rights.

Figure 4-10: TMM Fee Mineral Interests



Note: Figure courtesy Duluth, 2014.

Figure 4-11: Surface Property Options



Note: Figure courtesy Duluth, 2014. Red outlines mark the deposit outlines as projected to surface. Spruce deposit as indicated is the Spruce Road deposit.

In the case of private mineral leases where there has been no severance of the surface and mineral estates, surface use is generally provided as part of the mineral lease. Where the mineral and surface estates are severed and where surface rights are held privately, surface access is typically negotiated with the surface owner. The surface rights overlying all private leased mineral lands within the proposed mine development area are held by the federal government, the state, or are privately-owned by TMM.

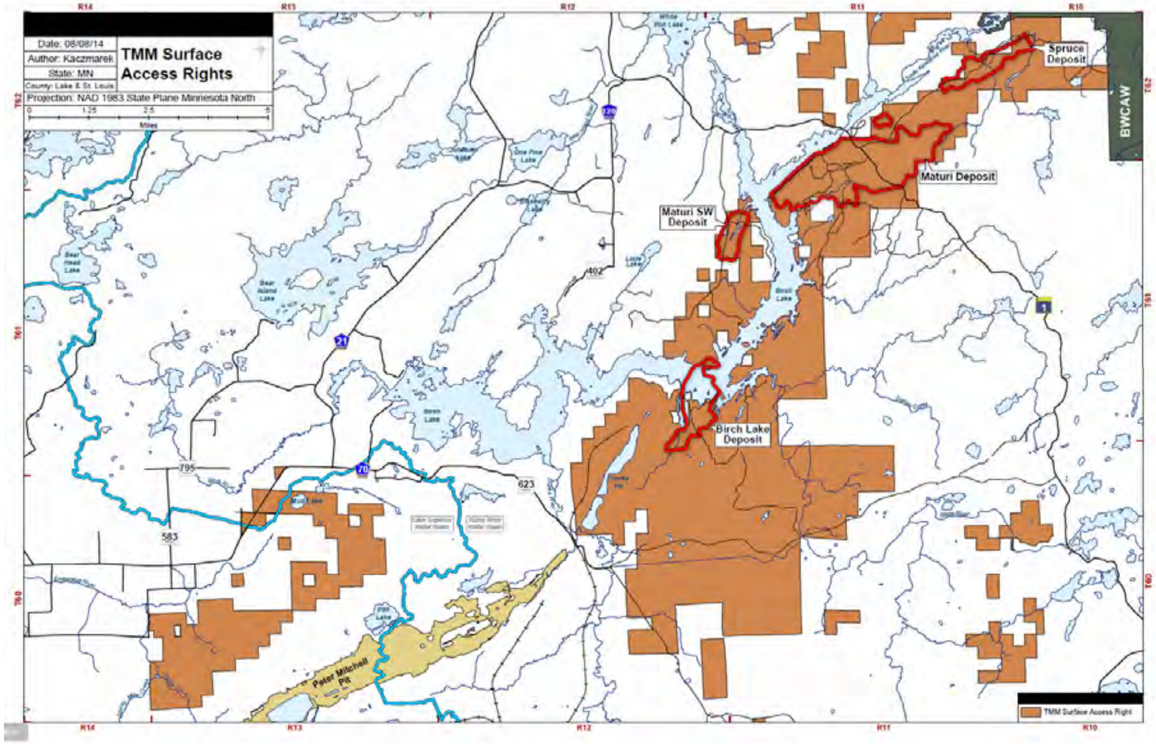
Proposed uses of federal, state or private surface lands will be subject to additional federal, state, and/or local laws and regulations governing such uses regardless of ownership of the lands on which the proposed uses may occur. These laws and regulations may require the proponent to obtain permits and other regulatory approvals from federal, state, and/or local agencies and other governmental authorities. There may also be circumstances where tribal authorities must be included in consultations.

Access to federal, state, and private lands may require additional agreements with other land owners if those lands are not accessible except by crossing other lands.

Figure 4-12 shows the locations of the surface access rights in the area of the proposed operation. These surface rights and others are possible locations for surface infrastructure subject to engineering/environmental/political assessments and reviews to confirm it is the most suitable location(s). Information provided by activities such as geological testing, environmental and engineering reviews may determine that alternative sites are more suitable for the surface infrastructure. While Duluth through TMM is of the opinion that it owns or controls more than sufficient surface rights, including additional distal surface rights suitable for “swapping” for government lands to serve all of its surface rights needs, this does not preclude consideration and or acquisition of alternative locations if they are determined to be more suitable.

The surface rights held in the various leases and permits contain approximately 1,600 acres of federal, state, and private surface overlying the Maturi deposit and the Maturi Southwest deposit, including about 740 acres of federal surface, 690 acres of state surface, and 150 acres of private surface. Of the total, about 1,300 acres are unified estates, meaning that the same owner has the rights to both the surface and the minerals. The remaining approximately 280 acres, however, are split estates, meaning that the minerals were severed from the surface interests and the surface and minerals are owned by different parties. These split estate sites have private minerals under federal surface rights.

Figure 4-12: Surface Access Rights



Note: Figure courtesy Duluth, 2014. Map north is to top of plan. Spruce deposit noted on the figure is the Spruce Road deposit.

According to the terms of the Project mineral rights for unified estates, TMM may be able to site on these lands the ventilation shafts, paste backfill plants, high voltage (HV) transmission lines, and service and contact roads necessary for the mine. Because all of the unified estates involve federal and state agencies, TMM would need to comply with applicable federal and state laws and secure approvals from the USFS and BLM (for federal lands) and from the DNR (for state lands) before proceeding with any construction or operation.

For the severed estates, the Project mineral rights grant TMM the right to reasonable use of the surface, but the right is more susceptible to dispute when a different party owns the surface. For the area proposed for underground ore extraction, the necessity for agreements with third-party surface owners is limited, because in most instances the state or federal mineral title includes the surface, or TMM already owns the overlying private surface rights, or leased private minerals are under federal surface rights.

TMM has state and private Project mineral rights underlying federal surface administered by the USFS. For these split estates, federal case law, federal regulations, and USFS policy authorizes TMM, as the mineral lessee, to reasonably access the federal surface for development of the underlying minerals. For those state surface lands and minimal federal surface lands on which TMM anticipates siting facilities or infrastructure, TMM may consider pursuing a land exchange policy.

Additional discussion on the Project permitting requirements and environmental reviews that may be required is included in Section 20.

4.5.1 Concentrator Site

The concentrator site would be located within the boundaries of the Superior National Forest, and would cover about 1,000 acres. A preferred location was identified for the purposes of the PFS; however, the location may change during future more detailed studies. Depending on the final site location, TMM will need to acquire surface rights to private and/or public lands. . The land package would host the concentrator, primary portal, temporary stockpiles, and a process water pond.

4.5.2 TSF Site

TMM has identified about 7,000 acres of surface rights deemed suitable for hosting the TSF and associated facilities. About 4,800 acres of private lands within the 7,000 acre area are controlled by TMM through the Potlatch option. As development progresses, TMM will review the suitability of acquiring additional lands within the 7,000 acre footprint. The proposed site would host the TSF, a concentrate filtration plant, an intermediate pond, a substation, and rail load-out facilities.

4.5.3 Corridors

TMM plans to use existing power corridors for power and pipeline infrastructure where practical, and develop new corridors where necessary. The location of these corridors is flexible, subject to engineering and environmental considerations. Several tentative corridors have been identified. A preferred corridor location was identified for the purposes of the PFS; however, the location may change during future more detailed studies. The utility corridors will be used for a concentrate slurry pipeline from the concentrator to the TSF, tailings slurry pipelines from the concentrator to the TSF (tailings) and to the mine (paste), a makeup water pipeline from the water source, a return process water pipeline from the TSF to the concentrator, high voltage transmission lines between the TSF and the concentrator, an electrical distribution line between the concentrator and the mine, service and contact roads between the mine, concentrator, TSF, and the water source, and a rail extension from the TSF to an existing railroad.

TMM has not yet committed to purchasing or otherwise acquiring any rights to specific locations necessary for the corridors.

4.6 Water Rights

In Minnesota, water is a public resource held in the public trust regardless of whether it is located on federal, state, or private land. Federal and state law heavily regulates the appropriation, use, management, and discharge of water as well as the water quality of any receiving surface water and ground water. TMM's acquisition and management of water for the Project would require withdrawal of makeup water from a suitable water source. It would also require multiple transfers, including transfers between natural water drainage basins, of water between the various facilities. The water management system would require a variety of federal and state permits.

Additional information on the permitting relating to water rights is included in Section 20.

4.7 Surface Option Agreements

A number of option agreements have been signed for lands in the Project vicinity. Figure 4-11 also included the locations of the key optioned parcels.

4.7.1 Potlatch

Twin Metals Minnesota LLC and Potlatch Minnesota Timberlands LLC (Potlatch) entered into an option agreement to purchase real property as to the Potlatch property on 23 October 2012. Lands covered under the Potlatch option total 3,084.11 acres and are summarized in Appendix A.

The Potlatch property is covered by title commitment no. T-61635, issued by Arrowhead Abstract and Title Company, and dated 13 January 2013. This title commitment confirms that Potlatch is the owner of the Potlatch property and is able to grant an option to purchase the property to TMM, subject to the various conditions, restrictions, exceptions, easements and encumbrances referred to in the title commitment.

The property was optioned by TMM as an area suitable for location of Project-related infrastructure. TMM has four years to exercise the option to purchase these lands, and is required to make annual option payments that increase over time.

4.7.2 Dunka

An option agreement to purchase real property as to the Dunka property was entered into by Cliff's Erie, LLC (Cliffs) and Duluth Metals Corp on 15 February 2008. Lands covered under the Dunka option total approximately 1,845 acres and are summarized in Appendix A.

The property is covered by title commitment no NCS-302909-2, issued by First American Title Insurance Company, and dated 1 May 2007, that confirms that Cliffs, LTV Steel Mining Company and Erie Mining Company were the formal property owners. The title commitment supports that Cliffs can grant an option to sell the Dunka property to Duluth, subject to the various conditions, restrictions, exceptions, easements and encumbrances referred to in the title commitment.

The agreement envisages that on completion of the option, TMM will either directly or indirectly acquire the Cliffs assets, and will also become directly or indirectly liable for liabilities on the property when title is transferred. Transfer of the lands will be subject to certain state of Minnesota consent/approvals. The assets include conventional surface rights, and surface leases. The location hosts a closed taconite open pit. The site has been rehabilitated and is the subject of on-going maintenance and monitoring.

The property was optioned by Duluth because the former Dunka open pit was considered to have potential as a future water source for Project development, and, as a brownfields site, was evaluated as a potential location for a process plant and other Project infrastructure. Duluth gave notice of its intent to exercise the option on 14 February 2011; however title remains to be formally transferred.

4.7.3 Minnesota Power

An option agreement at future appraised value was entered into between Allete Inc. and RendField Land Company (collectively Minnesota Power) and Lehmann Exploration Management Inc. Lands covered under the Minnesota Power option total 141.3 acres and are summarized in Appendix A.

The property is covered by title insurance policy no. NCS-471210 (OP), issued by First American Title Insurance Company, and dated 31 August 2011, that confirms that Minnesota Power is the formal property owner and can grant an option to purchase the Minnesota Power property to Lehmann, subject to the various conditions, restrictions, exceptions, easements and encumbrances referred to in the title policy.

These lands are complimentary to a much larger land package that was purchased and is now included in fee lands held by TMM (listed in Appendix A). They lie on the periphery of the larger Minnesota–Dunka land package. The rights to the land now are effectively a first right of refusal.

The property was optioned with others as part of a larger package by Lehmann as it was considered to be a potential candidate for infrastructure locations. Most of the optioned lands were purchased and are now fee lands held by TMM.

4.7.4 Additional Options and Agreements

While preferred locations have been selected during the PFS for siting of infrastructure, water sources and other key elements of the Project, until all relevant permits have been obtained, the locations in the PFS may not be the final permitted sites. Duluth has evaluated, and continues to evaluate alternative locations for some Project aspects. In support of these evaluations, Duluth has and will continue to enter into options to acquire selected land packages for these potential alternative locations as and when opportunities allow.

4.8 Exploration Permits and Approval

In addition to the mineral interests and regulatory requirements, prospecting and exploration programs may require permits and approvals from federal, state and/or local government agencies. Additionally, prospecting and exploration programs requiring federal agency approvals may be subject to stipulations and/or restrictions imposed by federal agencies through the environmental review and consultation processes under NEPA, NHPA and ESA. For example, surface access to lands subject to federal prospecting permits or preference right leases allowing drilling in certain areas may be subject to seasonal restrictions (such as restricting drilling to the period from November 1 to April 30 and/or periods of limiting drilling only to frozen ground conditions). Similarly, state approvals of exploration programs may be subject to stipulations and/or restrictions imposed by state agencies such as notification requirements under various laws or through the environmental review and consultation processes under MEPA.

4.9 Royalties

A general discussion of the royalties applicable on a Project-wide basis is provided in Section 4.4.

Duluth has currently identified 11 unique royalty combination schemes within the proposed mine plan area boundaries that will be payable to Federal, State, and private parties. These royalties, where applicable, are included in the economic analysis in Section 22 and are calculated based on the terms and certain assumptions within and for the respective leases. The royalty payments by scheme are summarized in Figure 4-13.

4.9.1 US Federal Royalty

Federal mineral leases MNES-1352, MNES-1353, and prospecting permits MNES-50652, MNES-50846, and MNES-57765 apply two different royalties payable to the federal government for the removal of:

- Copper and nickel
- Any associated products from the lease areas.

Duluth has assumed the leases be issued on its prospecting permits will have the same terms as federal mineral leases MNES-1352 and MNES-1353; therefore, all US federal leases have identical terms for the purpose of calculating royalties.

The copper and nickel royalty rate is defined as 4.5% of the "gross value" of the mineral mined and shipped to the concentrator. The royalty rate is subject to adjustment by the BLM during renewal periods under the terms of the leases. "Gross value" is as defined in Section 4.4.2. Thus in order to determine the royalty payable, TMM must also know the recovery percentage and payable metals of fully refined copper and nickel achieved from the unrefined ore sent to the concentrator. The market prices for both copper and nickel are assumed to be the TMM Board-approved long range prices of $3.50 and $9.50 per pound, respectively.

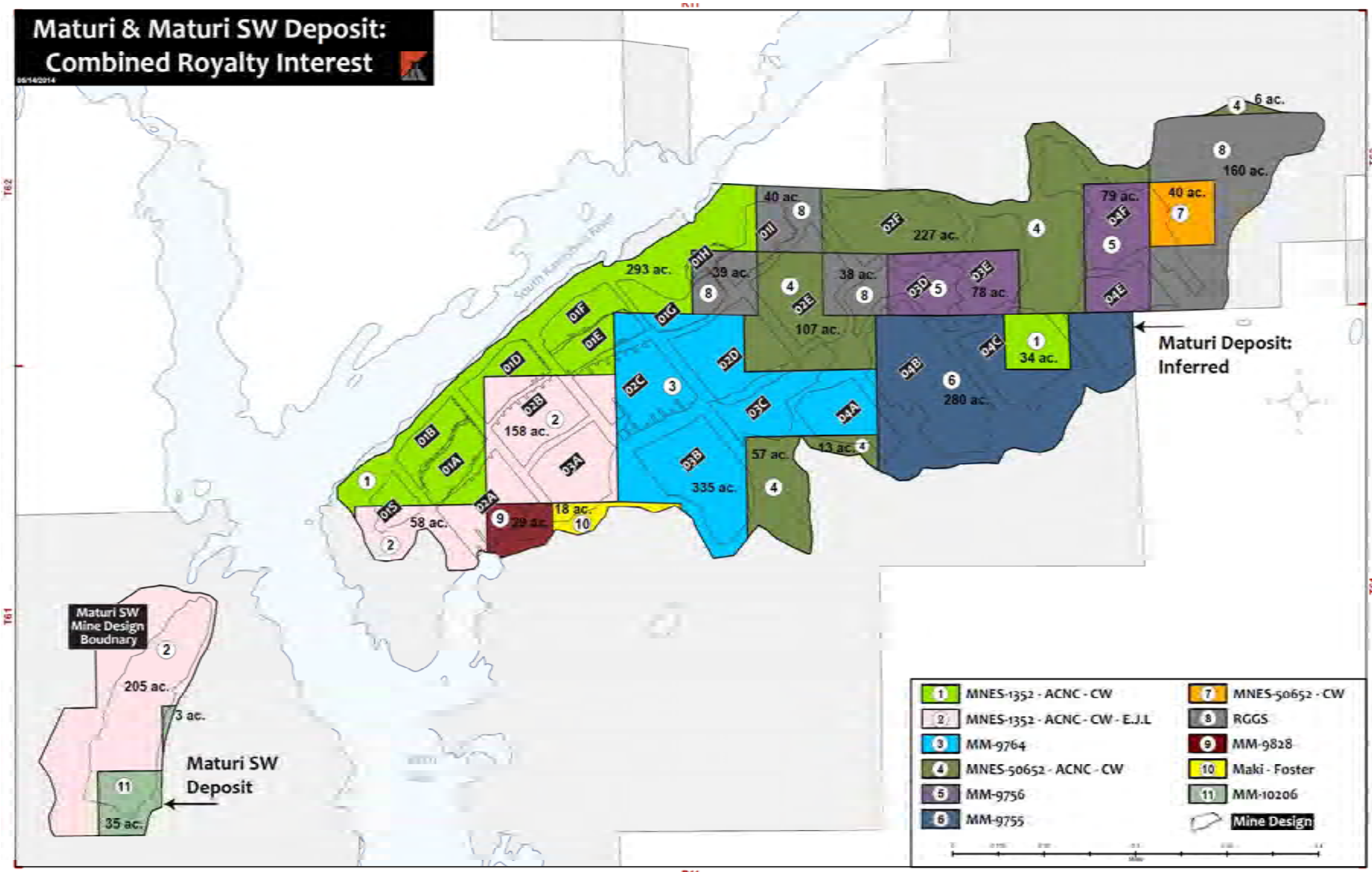
The associated products royalty rate is defined as 0.3% of the gross value of the minerals mined and shipped to the concentrator. The obligation directed by the leases in which the payment of an additional royalty is 0.3% of the gross value of minerals, as defined above, is for any associated products (e.g. gold, silver, platinum, palladium) that are recovered and either sold or used by TMM. Since the gross value is based on the value of copper and nickel, this royalty is generic in its applicability irrespective of the value of any associated product. The leases require the payment of this additional royalty irrespective of whether any associated products are produced.

Annual rentals of $1 per acre are required until production is achieved. Thereafter, annual minimum royalty of $10.00 per acre is required during each renewal period of the leases. The minimum royalty may be waived, reduced, or suspended at the discretion of the BLM.

The final royalty equation for the federal leases is

*[(Payable Cu) * $3.50 + (Payable Ni) * $9.50] * 1/3 * 4.80%.*

Figure 4-13: Royalty Interests



Note: Figure courtesy Duluth, 2014.

4.9.2 Minnesota State Royalty

State mineral leases MM-9755, MM-9756, MM-9764, MM-9828, and MM-10206 require the payment of annual rentals and each have an identical royalty calculation. Even though a separate calculation is required for each mineral produced, the royalty methodology is the same for each calculation. According to the lease agreements, definitions for:

- "Associated mineral products" means those intermingled or associated materials and substances recovered from each ton of crude ore mined from the mining unit that are excluded from the definition of metallic minerals
- "Metallic minerals" means any mineral substances of a metalliferous nature, except iron ores and taconite ores.

The royalty to be paid to the state by TMM for the metallic minerals and associated mineral products recovered from each ton of ore mined from the mining unit is the sum of the base rate ranging from 3.95% to 20% (determined according to a Base Royalty Rate Table attached to the lease) and additional bid rate (if any) stated in the agreement multiplied by the net return value of the metallic minerals and associated mineral products recovered from the each ton of dried crude ore. The net return value in the royalty calculation is the net smelter return prior to the payment of freight charges as calculated according to the detailed and fact-specific provisions in the lease.

The final royalty equation for the Minnesota state leases is:

*Net Return Value * [3.95% + Additional Bid Rate (varies by lease; ranging from 0% to 0.5%)].*

4.9.3 Private Royalty: RGGS, Saint Croix Lumber, Maki, Foster, and Adolfson

4.9.3.1 Saint Croix Lumber, Maki, Foster, and Adolfson

Private mineral leases Saint Croix Lumber and the Maki, Foster, and Adolfson leases have identical royalty calculations. Recoverable minimum royalty payments are required annually, escalating from $1,000 to $15,000 per year for the Maki, Foster, and Adolfson leases, and from $1,500 to $20,000 per year for the Saint Croix Lumber lease. The royalty on production is calculated as 3% of the net return value on production obtained from the premises. The payable royalty is calculated by multiplying the base rate specified in the agreement times the net return values on products obtained from the premises for both private agreements. The net return value in the royalty calculation is the net smelter return prior to the payment of freight charges as calculated according to the detailed and fact-specific provisions in the leases.

The final royalty equation for the private holder leases is:

*Net Return Value * Royalty Rate (varies by lease).*

4.9.3.2 RGGS

The RGGS mineral lease, which covers part of the Maturi deposit, requires annual rental payments escalating over time from $10.00 per acre or $7,500 (whichever is greater) to $50.00 per acre or $50,000 (whichever is greater). Rental payments cease upon commencement of royalty payments. Royalty payments commence after achieving commercial production at a rate of 5% of the net return value (as defined in the lease), subject to a minimum royalty of $200,000 per year, payable quarterly, which is recoverable against future production royalties in excess of the annual minimum for any particular year. The lease contains a work commitment of at least $25,000 during the first two years, and $25,000 each year thereafter, provided that the lessee may credit expenditures in excess of $25,000 against obligations for work expenditures in any future year.

The final royalty equation for this lease is also:

*Net Return Value * Royalty Rate.*

4.9.3.3 Childers–Whiteside Royalty

The Childers–Whiteside royalty is considered an overriding royalty to the federal mineral lease royalty. Annual minimum royalty payments are required (see Appendix A), and the payable royalty is in respect to the royalty payable to the United States Government in any leases or mining permits granted by the United States Government pursuant to the certain prospecting permits, except that the rate of such royalty shall be the lesser of:

- One-half of the rate of royalty payable to the United States Government provided in such leases or mining permits, or
- 1%.

With regards to the royalty analysis, 1% is the lower of the copper and nickel rates (4.5%) and one-half of the associated products royalty rate (0.3%) payable to the United States Government would be the lower rate. Therefore, 1.15% is multiplied by the gross value to calculate the royalty payable to the Childers–Whiteside heirs.

The final royalty equation for the Childers–Whiteside leases is:

*[(Payable Cu) * $3.50 + (Payable Ni) * $9.50] * 1/3 * 1.15%.*

4.9.4 E.J. Longyear Royalty

The E.J. Longyear royalty is also considered an overriding royalty to the federal mineral lease royalty. Annual minimum royalty payments are required (see Appendix A), and the payable royalty is in respect to the royalty payable to the United States Government in any lease granted by the United States Government pursuant a certain prospecting permit, except that the rate of such royalty shall be one-half of the rate of royalty payable to the United States Government.

With regards to the royalty analysis, one-half of the royalty payable to the United States Government is 2.4%. The 2.4% is multiplied by the gross value to calculate the royalty payable to the E.J. Longyear company. The E.J. Longyear royalty percentage (2.4%) may fluctuate in the future if the royalty percentage payable to the United States Government fluctuates.

The final royalty equation for the E.J. Longyear royalty is:

*[(Payable Cu) * $3.50 + (Payable Ni) * $9.50] * 1/3 * 2.4%.*

4.9.5 American Copper and Nickel Company Royalty

The ACNC agreement is considered an overriding royalty to the federal mineral lease royalty but is calculated in a different manner from the E.J. Longyear and Childers–Whiteside royalties. The payable royalty to ACNC is 7.5% of the “net distributable earnings” on all mineral products produced from the mineral properties upon commencement of commercial production, as defined in the ACNC agreement. “Net distributable earnings” is defined in the ACNC agreement as the aggregate of the revenues received during such quarter from or in connection with carrying on the business relating to the mining, milling and/or other treatment of any ores or concentrates and/or marketing of any product resulting from operations upon the mineral properties, less certain deductions.

Schedule “C” of the agreement details the deductions applicable when calculating the "net distributable earnings," which include, but are not limited to, pre-development depreciation and amortization allowable by the US Internal Revenue Service, and post-commencement of commercial production costs for construction, capital, operating, administrative, and financing, as well as royalties or similar payments made to any third party. With regards to the royalty analysis, the applicable deductions are subtracted from revenues received to yield net distributable earnings. The net distributable earnings are then multiplied by 7.5% to calculate the royalty payable to ACNC.

The final royalty equation for the ACNC royalty is:

*[(Revenues received) – (Deductions Under Schedule “C”)] * 7.5%.*

4.9.6 Royalty Buy-back

In the case of the Maki, and Foster private leases, TMM may buy up to 50% of each royalty for US$1.5 M each, or a prorated portion thereof if less than the full 1.5% royalty is purchased.

In the case of the Saint Croix Lumber private lease, TMM may buy up to 50% of the royalty for US$2 M, or a prorated portion thereof if less than the full 1.5% royalty is purchased.

4.10 Patriot Provision

Certain provisions in the Project mineral rights may be interpreted to impose restrictions relating to the transfer of ownership of the mineral products. Specifically, MNES-1352 and MNES-1353 each contain a Patriot Provision, which provides that if minerals from the leased lands are “shipped outside the United States for treatment,” Franconia may be required to return to the US an equal quantity of any copper shipped outside the US. This Patriot Provision, which is similar to those used for oil and gas leases, appears to have been originally included in MNES-1352 and MNES-1353 in 1966 for trade or national security reasons.

The BLM has discretion in determining whether the Patriot Provision is triggered. Duluth is of the opinion that, because the Patriot Provision is only triggered by shipping minerals for treatment, TMM has a strong basis for asserting that its treatment of the copper at the concentrator and subsequent shipment of the recovered copper concentrate would not trigger this provision.

4.11 Comments on Section 4

The AMEC QPs note:

- AMEC was provided with legal opinion that supports Duluth’s interpretation that the mining tenure held is valid and is sufficient to support declaration of Mineral Resources and Mineral Reserves. Tenure arises from a combination of private mineral leases, fee mineral lands, federal mineral leases and prospecting permits and state mineral leases
- A number of different royalties are associated with the tenure holdings
- The current financial model assumes a total federal royalty rate of 4.8% under the federal leases. The federal leases contain language allowing the Secretary of the Interior for the BLM, at his discretion, to increase the royalty rates at the time the federal leases are renewed. Currently the third renewal for these leases is in process and the BLM has expressed an interest in renegotiating the terms and conditions of the royalty. While the lease allows a maximum royalty of 6% to be applied for the third renewal period, the exact royalty rate will be subject to

negotiations with BLM. In the event the BLM adjusts the total rate for royalties payable under the federal leases, the financial model will need to be adjusted to reflect any changes that occur

- Additional surface lands will need to be acquired to allow the Project infrastructure as envisaged in this Report to be constructed
- Duluth advised AMEC that the company is not aware of any other significant environmental, social or permitting issues other than those presented in this Report that would prevent future exploitation of the Project deposits.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

The Project is located at the eastern end of the Mesabi Iron Range, a major center for iron ore mining for over 100 years. The region currently has eight large operating taconite mines and associated process plants, and two future operations are in development. As a result of the iron ore mining activity, an extensive network of railroads and paved roads has developed throughout the region that today provides excellent transport communications.

From the city of Duluth, the Project can be accessed by US Highway 53 north for 64 mi to its juncture with State Highway 169 north of the town of Virginia, thence 42 mi northeast on State Highway 169 to the town of Ely.

The Project is readily accessible by road from the town of Babbitt, a planned mining community of 1,200 inhabitants located approximately 10 mi to the southwest. From Babbitt, take County Road (CR) 70 west for 3 mi to the Ely–Babbitt CR (Highway) 21, north 7.5 mi to CR 120, north and east on CR 120 for 5 mi to State Highway 1, thence south and east to cross the South Kawishiwi River just north of the Project, a distance of 7 mi.

From the town of Ely, the Project can be reached by taking State Highway 1 south, which crosses the South Kawishiwi River just north of the Property, a distance of 12 mi. Forest service roads provide access within the individual properties.

5.2 Climate

The northern Minnesota climate is mid-continental. The average annual temperature is 38°F, with local temperatures averaging 4°F in January and 66°F in July. Annual rainfall averages approximately 28 in., with 30% occurring from November to April and 70% from May to October. Annual snowfall averages 60 in., with accumulation on the ground of 24 in. to 35 in.

Exploration operations continue year-around with much of the drilling completed in the winter months to minimize surface disturbances. Future mining activities could be conducted on a year-round basis.

5.3 Local Resources and Infrastructure

A major asset of the area is the engineering and technical resources supporting the iron ore mining operations that are accessible to TMM. Similarly, there is a large pool of skilled and unskilled labor in the region that is available to TMM.

The local infrastructure related to mining is excellent. Low-cost electric power, railroad networks, paved state highways, mine equipment suppliers, mining professionals, and relatively low-cost labor are available locally to service the eight operating Mesabi Range iron ore mines to the west.

The region has an extensive and reliable power supply network with two coal-fired thermal power stations located eight and 85 mi from the Project. Power is supplied to the area by 138 kV overhead transmission line linked to the regional power grid.

The nearest rail access for the Project is at Hoyt Lakes, approximately 9 mi to the southwest, and connects to the port of Duluth. The port of Duluth on Lake Superior is linked to the rail system and provides worldwide shipping access via the Great Lakes and St. Lawrence Seaway.

Proposed Project infrastructure is described in Section 18 of this Report.

5.4 Physiography

Elevations on the Project range from 1,425 ft to 1,550 ft. Topographic relief is generally low and controlled by bedrock exposures.

Wisconsin-age (110,000 to 10,000 years ago) continental glaciation scoured bedrock, leaving low hills thinly mantled by glacial drift. Bedrock exposures are generally less than 5% of surface area. Glacial deposits are as thick as 65 ft in low areas occupied by swamps, which are prominent in the north–central and northeast portions of the main block of the Maturi area.

The upland areas of the Project leases are forested by second-growth mixed conifers and deciduous trees including white, red and jack pines, spruce, balsam, poplar and birch. Treed swamps and open marshes support reeds, sedges, and sphagnum mosses.

5.5 Comments on Section 5

In the opinion of the AMEC QPs:

- There is sufficient suitable land available within the general area for the planned tailings disposal, mine waste disposal, and mining-related infrastructure such as underground mines, process plant, workshops and offices.
- While a significant portion of the land requirements for Project development are held under option or are owned by TMM, no surface rights for targeted infrastructure locations are currently held (refer to Section 4). However, the process for obtaining such rights is well understood.
- Mining activities can be conducted year-round.

6.0 HISTORY

6.1 General

The region was opened to prospecting following the 1854 Treaty of LaPointe. Initial efforts focused on copper, and an 1865 gold rush led to discovery of iron ore. Iron ore mines opened in the Archaean Soudan Iron Formation of the Vermilion Range in 1884. Mining of direct shipping iron ore from the Archaean Vermilion iron range commenced in 1892, and large scale production of iron ore pellets (taconite) from magnetite iron-formation began in 1955. Copper and nickel sulfides were discovered in the Duluth Complex in the 1890s; however, large-scale exploration began in the 1950s. The DNR reports more than 1,900 diamond drill holes and 310 mi of core have been drilled to explore the base of the Duluth Complex for copper and nickel (Cargill, 2005b). Starting in 1985, the DNR re-analyzed core from the copper–nickel exploration and found significant platinum group elements (PGEs), which has prompted the re-evaluation of a number of known deposits in the western portion of the Duluth Complex.

Each of the four deposits that comprise the Project has a somewhat different history, and is discussed separately in the following sub-sections.

6.2 Maturi

Until combined into the Maturi project by TMM, the Nokomis (aka Maturi Extension) and Maturi projects were separate and had somewhat different exploration histories. The combined histories are summarized in Table 6-1. What was known as the Nokomis deposit was not well explored prior to about 2006 when Duluth began drilling the area. Maturi, on the other hand, was well explored by Inco with a combination of drilling and underground exploration.

6.3 Maturi Southwest

The history of the Maturi Southwest deposit is closely tied to the Maturi deposit, but only a small number of legacy holes were drilled in the 1950s through 1970s by Inco (1957 and 1969; seven holes; 5,830 ft), Duval (1976; two holes; 9,394 ft), and Bear Creek Mining company (1968–1970; five holes; 8,625 ft). In 1990, Lehmann drilled two holes in the extreme south of what is now considered Maturi Southwest (2,707 ft). In 2012–2013, TMM drilled 53 holes and four wedges (46,068.5 ft). No additional work in known to have been completed over the area.

Table 6-1: Summary of Maturi History (modified from Routledge and Greenough, 2006)

Year	Exploration
1954–1957	Surface exploration only. Exploration suspended when Federal Department of the Interior would not issue mining permits pending Congress enacting proposed wilderness legislation.
1966	ACNC granted two federal leases at Maturi and Spruce Road deposits.
1967	153 diamond drill holes for 81,699.23 ft drilled at Maturi and Spruce Road. Exploration shaft sinking started at Maturi, and Bechtel completed a scoping study for Maturi.
1968	Shaft sunk to 1,090 ft, a 634.9 ton bulk sample taken and underground exploration carried out on the 1,000 ft level at Maturi. 2,689.6 tons from the drift were stockpiled on surface and the shaft capped. Fifteen holes were drilled from underground (21,400.85 ft).
1969	ACNC drilled 16 holes for 17,473.7 ft.
1973	Maturi buildings and head frame removed and site restored; exploration focused on Spruce Road.
1975–1979	All ACNC work suspended because of State moratorium on copper-nickel exploration and mining.
1985	DNR samples 1970-1975 Duval core from Birch Lake area and discovers 2 m of PGE mineralization associated with chromite-rich oxides.
1986	Since earlier drilling had not assayed for PGEs and Au, ACNC investigates Maturi drill core and assays for PGE and gold; only anomalous values found.
1988	ACNC Joint venture with Lehmann and BHP Utah to explore for PGE mineralization; one hole diamond drilled.
1989	Joint venture dissolved and a new ACNC joint venture with LEM was formed.
1990	LEM drilled one hole on ACNC property; seven others in the area (14,150.22 ft total).
1992	LEM unable to obtain financing, LEM joint venture with Inco (ACNC) dissolved.
2000	1,400 coarse reject samples from 26,247 ft in 26 holes were assayed by Wallbridge for Cu, Ni, Co, Pt, Pd, Au and S. Wallbridge prepares a resource estimate under JORC code and assessment (scoping study) of the potential for economic mineralization. Hole 11526R (1166.7 ft) drilled to twin Inco hole 11526.
2005	In May, Franconia acquired from ACNC, through its Beaver Bay Joint Venture partner, an interest in 5,201 ac (2,105 ha) covering the Spruce Road and Maturi deposits.
2006	Preliminary assessment of the Birch Lake and Maturi projects completed.
2006–2014	Duluth Metals and TMM drilled 500 exploration core holes with 190 wedge holes (1,413,292 ft).
2011	Franconia acquired by TMM.

6.4 Birch Lake

Birch Lake was explored by a series of operators between 1955 and now. Duval did a significant amount of work in the 1970–1975 period (Table 6-2). In 2000, the Beaver Bay Joint Venture began serious drill exploration of the area. In 2002, Franconia optioned the property and continued exploration drilling of the known mineralization. In 2010, TMM was formed and acquired the property with the acquisition of Franconia in 2011. TMM drilled 30 holes in 2011 and 2012 which were used (in part) in preparation of the resource estimate described in Section 14 of this Report.

6.5 Spruce Road

Disseminated sulfide mineralization was discovered at Spruce Road in 1951 (Table 6-3). Between 1954 and 1974 Inco performed intensive exploration and applied for a mining license in 1975 which was put on hold because of the moratorium on copper and nickel exploration. Since then, only two holes have been drilled.

Table 6-2: Summary of Birch Lake History

Year	Exploration
1970–1975	Duval Corporation diamond drilled a fence of wide-spaced holes to the base of the Duluth Complex
1985	BBJV undertakes prospecting in vicinity of property and acquires initial mineral permits for property. Under an earn-in agreement with Utah International Inc. to explore for PGEs in the Duluth Complex, BBJV carries out data compilation and stream sediment survey extending to the shore of Lake Superior. DNR samples Duval core and discovers 2 m of PGE mineralization in DU-15 associated with chromite-rich oxides. LEM leased ground for Cascade Joint Venture and drilled wedged offset hole to confirm PGEs.
1985–1987	Mapping and geophysical surveys; BBJV joint venture with Utah and ACNC (Inco) on land under option from ACNC (Inco) north of Birch Lake.
1988	Hole C88-1 drilled west of Duval hole DU-15 intersected copper mineralization but no PGEs. Utah and ACNC (Inco) terminated their earn-in agreements with the BBJV. Joint venture earn-in agreement signed with International Platinum Company Inc. (IPCO).
1989	Holes 89-1 and 2 drilled under IPCO agreement.
1990	BL90-1 and 2 drilled south of Birch Lake and 90-3 to north for assessment on lands sub-leased from ACNC (Inco). IPCO earn-in agreement terminated.
1995	BBJV reorganized with new partners; BL-95-1and BL95-1W drilled to test magnetic anomaly at the north edge of Birch Lake, no encouragement.
1997	MN Natural Resources Research Institute (NRRI) work suggests PGE's associated with Birch Lake fault zone; BBJV acquires State Lease for lake bottom, obtains funding from State and Amplats.
1998	BL98-1 and 1W from south shore west into Birch Lake and intersects PGE values. Preliminary metallurgical tests by Amplats. Earn-in joint venture agreement signed with Altoro Gold Corporation.
1999	BL99-1 and 2 and wedges drilled and property land package expanded. Altoro abandoned agreement with BBJV.
2000	Earn-in joint venture agreement signed with Impala Platinum Holding Ltd. Eleven holes and 25 wedges of BL00 series drilled (33,796 ft) to delineate Cu–Ni–PGE mineralization.
2001	22,821 ft in seven drill holes and 19 wedges drilled, five holes collared from barge in Birch Lake as step outs to further delineate mineralization. Drilled wedge hole off old Exxon hole D-5 south west of Birch Lake deposit.
2002	A wild cat hole drilled on boundary of property 600 ft. southwest of old Exxon hole. Resource estimates by Snowden and LEM. Impala drops option late in year; property optioned to Franconia.
2004	Flotation and pilot plant Platsol™ hydrometallurgical testwork on Birch Lake core composites performed at SGS Lakefield Research.
2005	Four holes and four wedge offset holes (13,022 ft) diamond drilled on the Birch Lake property. Agreement to use Platsol™ technology arranged by BBJV on behalf of Franconia.
2006	Preliminary assessment of the Birch Lake.
2010	Franconia drilled 11 exploration core holes.
2011	Franconia acquired by TMM.
2011–2012	TMM drilled 30 exploration core holes (82,945 ft) and began metallurgical testwork, environmental baseline studies, and preliminary engineering studies.

Table 6-3: Summary of Spruce Road History

Year	Exploration
1951	Discovery of disseminated sulfide mineralization by Fred Childers and Roger Whiteside. Drilled one hole (188 ft).
1951–1954	ACNC (Inco) acquired the property and performed ground magnetometer and vertical loop electromagnetic surveys (VLEM), geological mapping and sampling.
1954–1957	ACNC (Inco) drilled a total of 17,930.2 ft of AX core in 17 holes.
1957	Exploration suspended pending passage of wilderness legislation.
1966–1968	ACNC (Inco) granted Federal Leases ES-01352 and ES-01353. 100,714 ft of drilling completed in 166 holes and a 1,300 ton bulk sample was collected. Geological mapping and geophysical surveys including Ronka EM 16 (VLF), ground magnetometer and induced polarization (IP) were completed.
1969	Additional private and Federal leases obtained by ACNC (Inco).
1972	Horizontal Loop electromagnetic surveys, IP surveys, and magnetometer surveys completed.
1973	ACNC (Inco) drilled a total of 769 ft in 26 holes to test a bulk sample area.
1974	A 10,000 ton (9,072 tonne) bulk sample was collected and processed at the Inco's Creighton Mill in Sudbury, Ontario.
1975	ACNC (Inco) submitted a formal mining proposal.
1975	Minnesota declared a moratorium on copper–nickel exploration until 1979. The project was put on hold.
1988	ACNC (Inco) entered joint venture with Lehman Exploration Management (LEM) and BHP Utah to explore for PGE mineralization (the Beaver Bay Joint Venture - BBJV). The JV dissolved in 1989 after one hole was drilled.
1989	JV between ACNC (Inco) and LEM reformulated the BBJV.
1990	Eight holes drilled.
1992	ACNC (Inco) and LEM JV dissolved.
1997	Downhole Crone PEM Survey completed.
1999	Wallbridge optioned the properties from Inco.
1999–2000	Wallbridge drilled 4,054.04 ft in two holes.
2002	Franconia enters into agreement with Beaver Bay Joint Venture to acquire the Spruce Road deposit and other properties.
2010	Antofagasta and Duluth form TMM, a joint venture company.
2011	Duluth and Antofagasta acquire all of the common shares of Franconia.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Introduction

The geology of the Maturi, Maturi Southwest, Birch Lake, and Spruce Road deposits is summarized from Parker and Eggleston (2014), and more detail can be found in that technical report. Copper–nickel–PGE mineralization within the Duluth Complex is hosted by mafic to ultramafic intrusive rocks that are part of the overall complex. Table 7-1 summarizes the lithologies that occur within the Duluth Complex. Minerals (and their formulae) reported from the various deposits are summarized in Table 7-2.

7.2 Regional and District Geology

The Maturi, Maturi Southwest, Birch Lake, and Spruce Road properties lie within the Mesoproterozoic Midcontinent Rift System which is exposed in central and northeastern Minnesota and extends north into Ontario (Routledge, 2004; Figure 7-1). To the north and west of the project area, rocks of the Superior Province of the Canadian Shield include Archaean (>2,600 Ma) mafic to felsic metavolcanic rocks, metasedimentary rocks, ortho- and paragneisses, and granitic intrusions; and Paleoproterozoic (ca. 1,850 Ma) iron-formation, clastic, and carbonate metasedimentary rocks of the Animikie Basin. Archaean and/or Paleoproterozoic rocks form the footwall to the four deposits.

In eastern Minnesota, the Midcontinent Rift System developed as crustal scale extension during the Mesoproterozoic. The rift system is traceable, as exposures of mantle-derived tholeiitic to subalkaline mafic lava flows, intrusive rocks, and rift-filling fluvial sedimentary rocks, and in the subsurface as a gravity anomaly (high), from the eastern end of Lake Superior, arcing west across the lake basin, and extending south-southwest to northeastern Kansas. Intrusion of the main stage of the Duluth Complex which hosts the Cu–Ni–PGE mineralization (ca. 1,099 Ma) was related to rifting and is co-genetic with the North Shore Volcanic Group volcanic rocks, forming its hanging wall to the southeast.

The Duluth Complex is defined as the more or less continuous mass of mafic to felsic plutonic rocks that extends for more than 170 mi in an arcuate fashion from Duluth nearly to Grand Portage in Minnesota. It is bounded by a footwall of Paleoproterozoic sedimentary rocks and Archaean granite–greenstone terranes (Peterson and Severson, 2002), and a hanging wall largely of co-magmatic anorthosite, rift related flood basalts, and hypabyssal intrusions of the Beaver Bay Complex.

Table 7-1: Lithologies Discussed in this Report

Lithology	Description
Anorthosite	An essentially monomineralic intrusive rock composed almost entirely of plagioclase feldspar, which is usually labradorite but may be as calcic as bytownite or as sodic as andesine or oligoclase. Accessory mafic minerals include olivine, augite, and oxide.
Dunite	An ultramafic intrusive rock consisting almost entirely olivine with accessory magnetite and/or ilmenite. Chromite is an important accessory at Birch Lake but rare at Maturi and Spruce Road.
Gabbro	A group of dark-colored, mafic intrusive rocks composed principally of basic plagioclase (commonly labradorite or bytownite) and clinopyroxene (augite), with or without olivine and orthopyroxene. It is the approximate intrusive equivalent of basalt.
Gabbronorite	A gabbroic rock containing both clinopyroxene and orthopyroxene as the mafic minerals. Generally occurs at the base of mafic intrusions as a result of contamination by footwall rocks.
Norite	A coarse-grained mafic intrusive rock containing basic plagioclase (labradorite) as the chief constituent and differing from gabbro by the presence of orthopyroxene (hypersthene) as the dominant mafic mineral.
Troctolite	A mafic intrusive rock composed of 50% to 80% calcic plagioclase (e.g. labradorite) and mafic minerals dominated by olivine.
Melatroctolite	A mafic troctolite with 50% to 80% olivine and 20% to 50% plagioclase.
Anorthositic Troctolite	A mafic intrusive rock composed of 70–80% plagioclase with 20–30% olivine and pyroxene. ol>px.

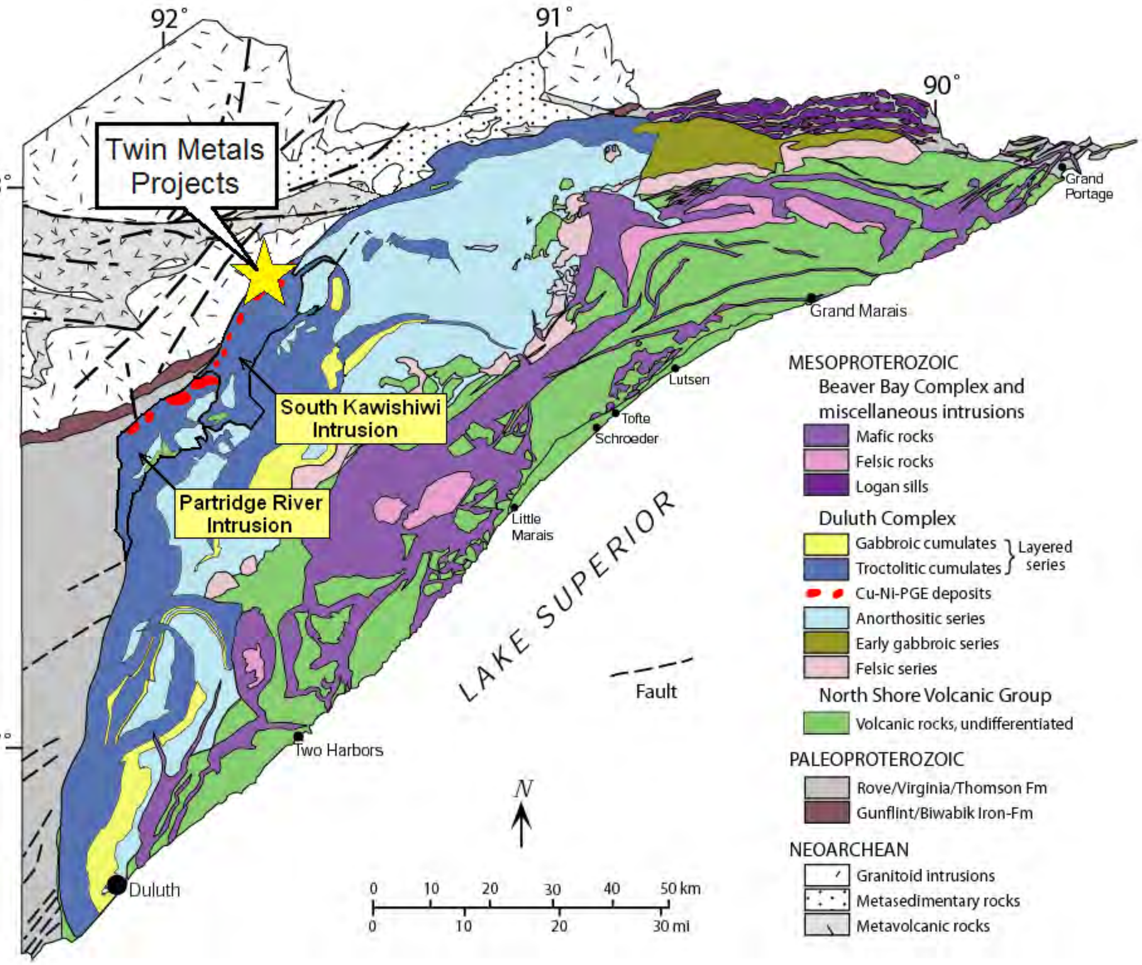
Note: ol – olivine; px – pyroxene

Table 7-2: Minerals Identified at Maturi, Maturi Southwest, Birch Lake, or Spruce Road

Cu Minerals	Formula
native copper	Cu
bornite	Cu_5FeS_4
chalcocite	Cu_2S
chalcopyrite	$CuFeS_2$
covellite	CuS
cubanite	$CuFe_2S_3$
cuprite	Cu_2O
digenite	Cu_9S_5
haycockite	$Cu_4Fe_5S_8$
mooihoekite	$Cu_9Fe_9S_{16}$
neodigenite	Cu_9S_5
putoranite	$Cu_9Fe_9S_{16}$
talnakhite	$Cu_9(Fe,Ni)_8S_{16}$
tenorite	CuO
Ni Minerals	**Formula**
heazlewoodite	Ni_3S_2
mackinawite	$(Fe, Ni)_9S_8$
millerite	NiS
pentlandite	$(Fe,Ni)_9S_8$
violarite	Ni_2FeS_4

Precious Metals Minerals	Formula
native silver	Ag
electrum	$Au(Ag)$
froodite	$PdBi_2$
hessite	Ag_2Te
insizwaite	$Pt(Bi,Sb)_2$
irarsenite	$(Ir,Ru)As_2$
michenerite	$PdBiTe$
moncheite	$(Pt,Pd)(Te,Bi)_2$
paolovite	Pd_2Sn
polarite	$Pd(Bi,Pb)$
silver telluride	$AgTe$?
sobolevskite	$PdBi$
sperrylite	$PtAs_2$
Gangue Minerals	**Formula**
altaite	$PbTe$
frobergite	$FeTe$
galena	PbS
pyrite	FeS_2
pyrrhotite	$Fe_{1-x}S$ (x= 0 to 0.2)
sphalerite	$(Zn,Fe)S$
troilite	FeS
chromian spinel	$Mg(Al,Cr)_2O_4$
chromite	$(Fe, Mg)(Cr, Al)_2O_4$
ilmenite	$FeTiO_3$
magnetite	Fe_3O_4

Figure 7-1 Regional Geological Map



Note: Figure after Soever, 2002.

In genetic terms, the Duluth Complex is composed of multiple discrete intrusions of mafic to felsic tholeiitic magmas that were episodically emplaced into the base of a co-magmatic volcanic edifice between 1,108 and 1,099 Ma. Within the nearly continuous mass of intrusive igneous rock forming the Duluth Complex, four general rock series are distinguished on the basis of age, dominant lithology, internal structure, structural position, and geochronology within the complex:

- Felsic series: Massive granophyric granite and smaller amounts of intermediate rock that occur as a semi-continuous mass of intrusions strung along the eastern and central roof zone of the complex, emplaced during an early-stage magmatism (~1,108 Ma)
- Early gabbro series: Layered sequences of dominantly gabbroic rocks that occur along the northeastern contact of the Duluth Complex, emplaced during early-stage magmatism (~1,108 Ma)
- Anorthositic series: A structurally complex suite of foliated, but rarely layered, plagioclase-rich anorthositic rocks emplaced throughout the complex during main-stage magmatism (~1,099 Ma)
- Layered series: A suite of stratiform troctolitic intrusions that comprises at least 12 variably differentiated mafic layered intrusions that occur mostly along the base of the Duluth Complex and host Cu–Ni–PGE mineralization. These intrusions were emplaced shortly after the Anorthositic series (~1,099 Ma).

In the Project area, mineralization is hosted by troctolitic rocks at the base of the Layered series. In the Project area, most of the known mineralization is hosted in what is known as the Basal Mineralized Zone (BMZ) at the base of the South Kawishiwi Intrusive (SKI) within a few hundred ft of the footwall contact of the Duluth Complex.

The Duluth Complex has not been significantly deformed since magma consolidation, but it has been subjected to displacements along reactivated basement faults as well as cross faults. These faults have been active pre-, syn- and post-emplacement of the SKI. Where exposed in parts of the SKI and footwall rocks, movement on these faults ranges from 10 to 400 ft.

7.3 Basal Mineralized Zone (BMZ)

Mineralization in the Project area is all hosted by the BMZ of the SKI which consists of four regionally extensive sub-units (Severson, 1994).

These sub-units are, from top to bottom, the PEG, U3, BH, and BAN:

- Pegmatitic Unit (PEG): Medium to very coarse-grained, locally sulfide-bearing, troctolitic to gabbroic rocks that grade into pegmatoidal (0.4–0.8 in) and pegmatite

(>0.4 in) zones. The unit occurs immediately above the U3 unit and separates the sulfide-bearing lower units from the sulfide-free upper units of the South Kawishiwi intrusion

- Ultramafic Three (U3): Layered ultramafic (melatroctolite–peridotite) and troctolite horizons with lenses and pods of Fe–Ti oxide-bearing (>5%) ultramafic rocks and/or massive oxide. Disseminated sulfide occurs from trace amounts to 5%, and typically includes pyrrhotite, chalcopyrite, cubanite and pentlandite
- Basal Heterogeneous Zone (BH): The main sulfide-bearing unit characterized by variably textured troctolite, augite troctolite, anorthositic troctolite, and olivine gabbro with 0.5–5% disseminated pyrrhotite, chalcopyrite, cubanite and pentlandite
- Bottom Augite Troctolite/Norite (BAN): Variably textured, sulfide-bearing gabbronorite, norite, and augite troctolite. The unit grades upward into the BH Unit: both are heterogeneous and are sulfide-bearing. In all likelihood the BAN Unit represents a footwall contamination zone of the BH Unit along the basal contact (Severson, 1994).

The base of the BMZ is invariably the unconformity between the Archaean or Paleoproterozoic rocks that comprise the footwall rocks to the Mesoproterozoic SKI (1.1 Ga). The majority of footwall to the SKI is composed of the Giants Range Batholith (GRB), a 2.68 Ga granitoid batholith composed of silica-poor rocks ranging from diorite to quartz monzonite in composition. Locally, in the Birch Lake area, footwall is composed of Paleoproterozoic metasedimentary rocks (~1.85 Ga.) of the Biwabik Iron Formation (banded iron formation) and/or the Virginia Formation (shales to greywacke).

Hanging wall rocks to the BMZ fall into one of three main geologic units: PEG, Main AGT, or the An-Series. PEG is the most prevalent hanging wall unit consisting generally of a weakly to well-developed very coarse-grained to pegmatoidal anorthositic troctolite to anorthositic gabbro and is mostly barren of sulfides. Directly above PEG are the hanging wall rocks of the Main AGT. If PEG is not present, Main AGT directly overlies the top of the BMZ. Comprised of a homogenous augite troctolite to anorthositic troctolite, this unit is commonly very thick, sometimes exceeding 1,000 ft. Within the central and eastern portions of the Maturi Deposit the AN-Series locally forms the hanging wall. This unit is an extremely large block of an earlier phase of the Duluth Complex measuring thousands of ft laterally and locally, a few thousand ft thick. Dominantly anorthosite to anorthositic gabbro in composition, the lower portions of the block that comprise the immediate hanging wall to the BMZ are usually a coarse to very coarse-grained anorthosite.

The BMZ is quite variable in its overall thickness, ranging from tens of feet to hundreds of feet, but always occurs at the base of the SKI as a continuous sheet-like body dipping southeasterly. The base of the BMZ is defined as the base of the SKI, and the top is defined as the uppermost U3, BH, or BAN occurrence coincident with the first occurrence of sulfide mineralization. Although pervasively containing sulfide mineralization, the BMZ has low-grade to barren portions that may occur at the top, middle, or bottom of the unit. Despite the lack of sulfides in these instances, it is apparent by the distinctive rock types, textures, and stratigraphic position that these rocks are part of the BMZ even though the mechanism resulting in the dearth of sulfides is not understood. Thus, although definition of the BMZ is generally based on the presence of sulfides, it is in fact the hosting package of rocks that defines this unit.

Recent work by TMM and AMEC, based on current drilling, has redefined the internal stratigraphy of the BMZ in the deposit areas. Although the redefined stratigraphy generally correlates with the regionally defined stratigraphy, TMM chose to use different nomenclature to identify the local stratigraphic units to avoid confusion and possible miscorrelation.

7.4 Local Geology

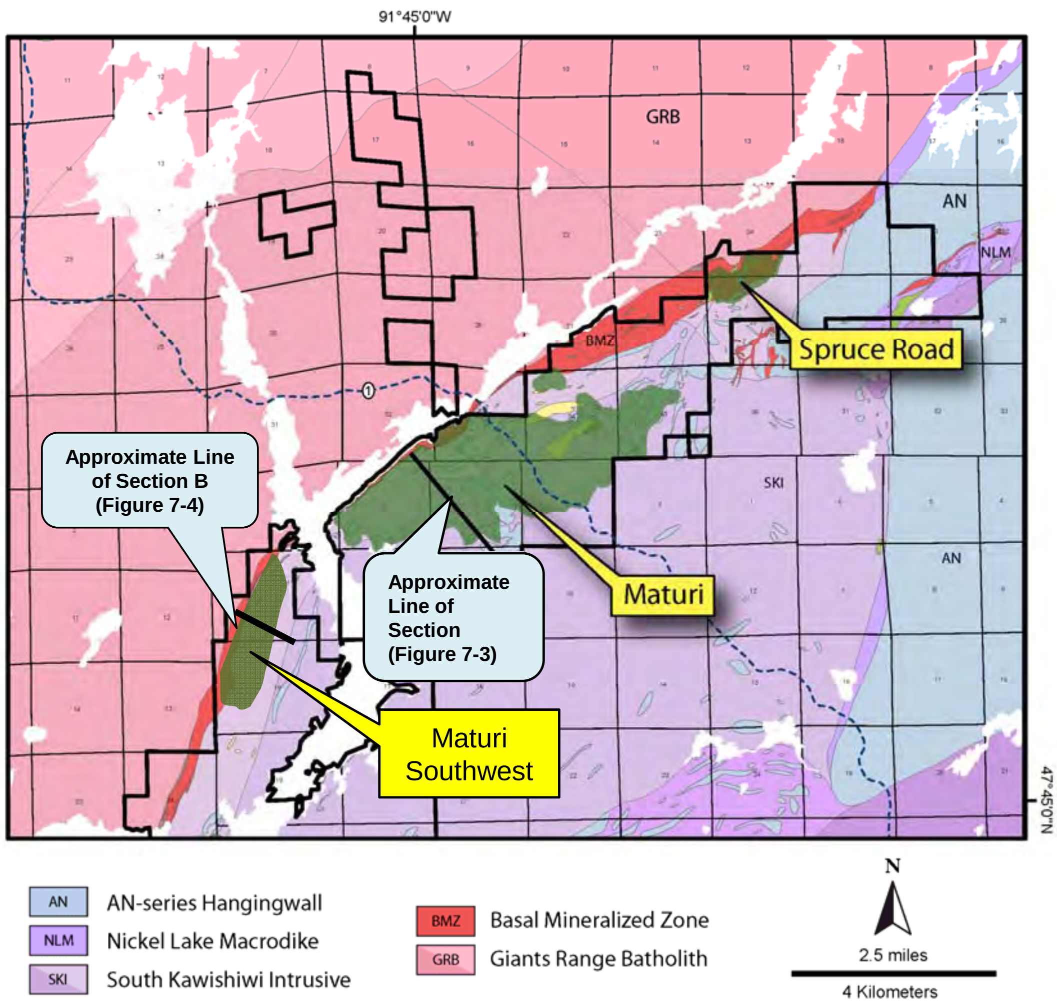
7.4.1 Lithology

7.4.1.1 Maturi and Maturi Southwest

The Maturi and Maturi Southwest deposits consist of a tabular sheet of disseminated copper-nickel-iron sulfide mineralization 5 ft to 865 ft thick (average 215 ft) in the BMZ which rests on or close to the SKI–granite contact. The exposed basal SKI–granite contact trends northeasterly (60^{o} azimuth at Maturi; 30^{o} azimuth at Maturi Southwest) and generally dips about 20° southeast, but dips at depth range from 20^{o} to 55^{o} based on interpretation of the drill data. Figure 7-2 shows the property geology and Figure 7-3 shows a typical cross section across the Maturi deposit. Figure 7-4 is a typical cross section from the Maturi Southwest deposit.

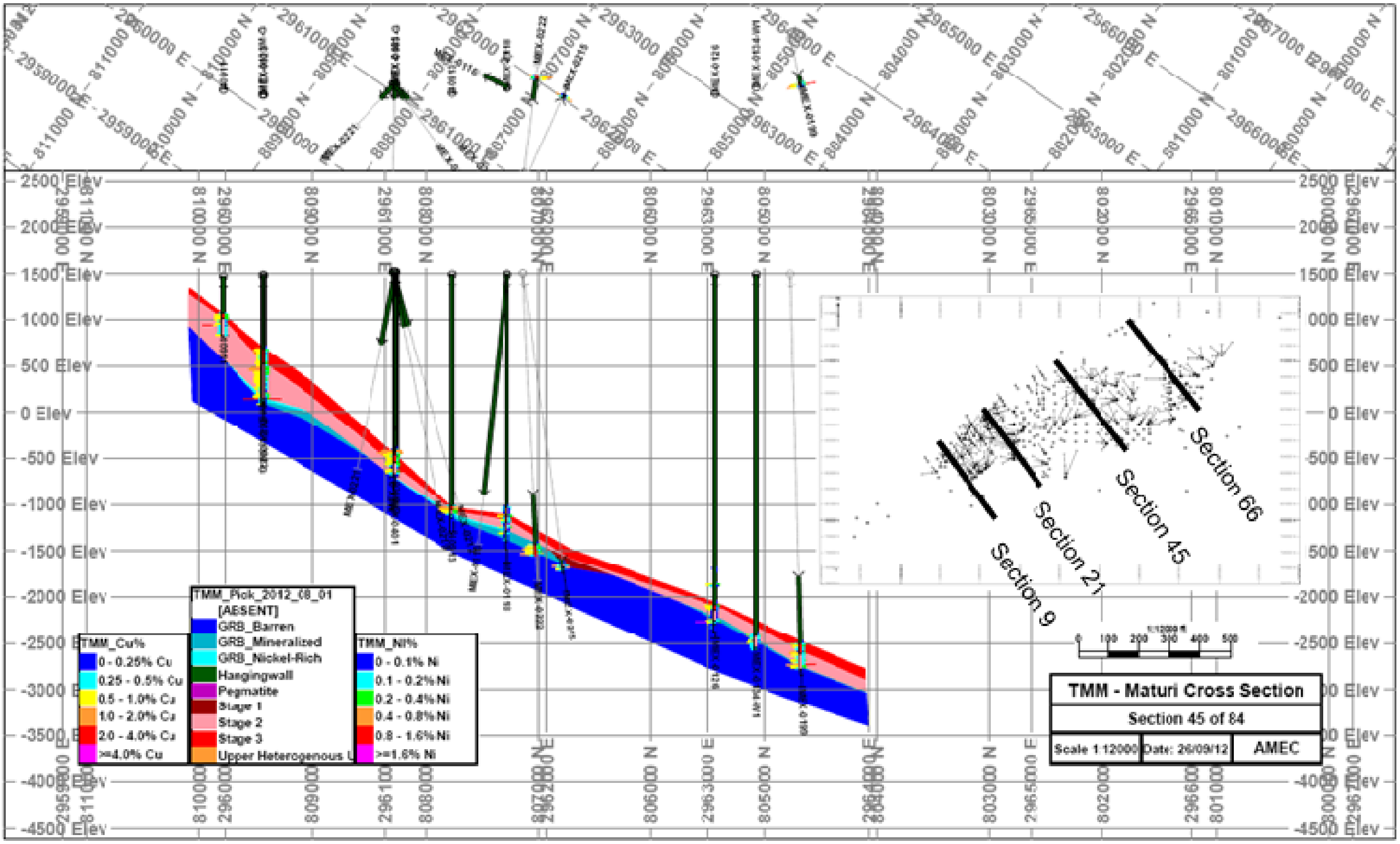
The BMZ at Maturi and Maturi Southwest has been subdivided into four stratigraphic units based on geochemical and geological similarities. The apparent order of intrusion, from oldest to youngest, is: the Upper Heterogeneous (UH), Stage 1 (S1), Stage 2 (S2), and finally, Stage 3 (S3). These units appear to be stratigraphically “layered”; however, each stratigraphic layer is interpreted to comprise multiple magma pulses of similar composition that may crosscut previous stages. These subunits are similar to the regional subunits, but TMM has not correlated them with the regional subunits.

Figure 7-2: Local Geological Map of the Maturi, Maturi Southwest, and Spruce Road Deposits



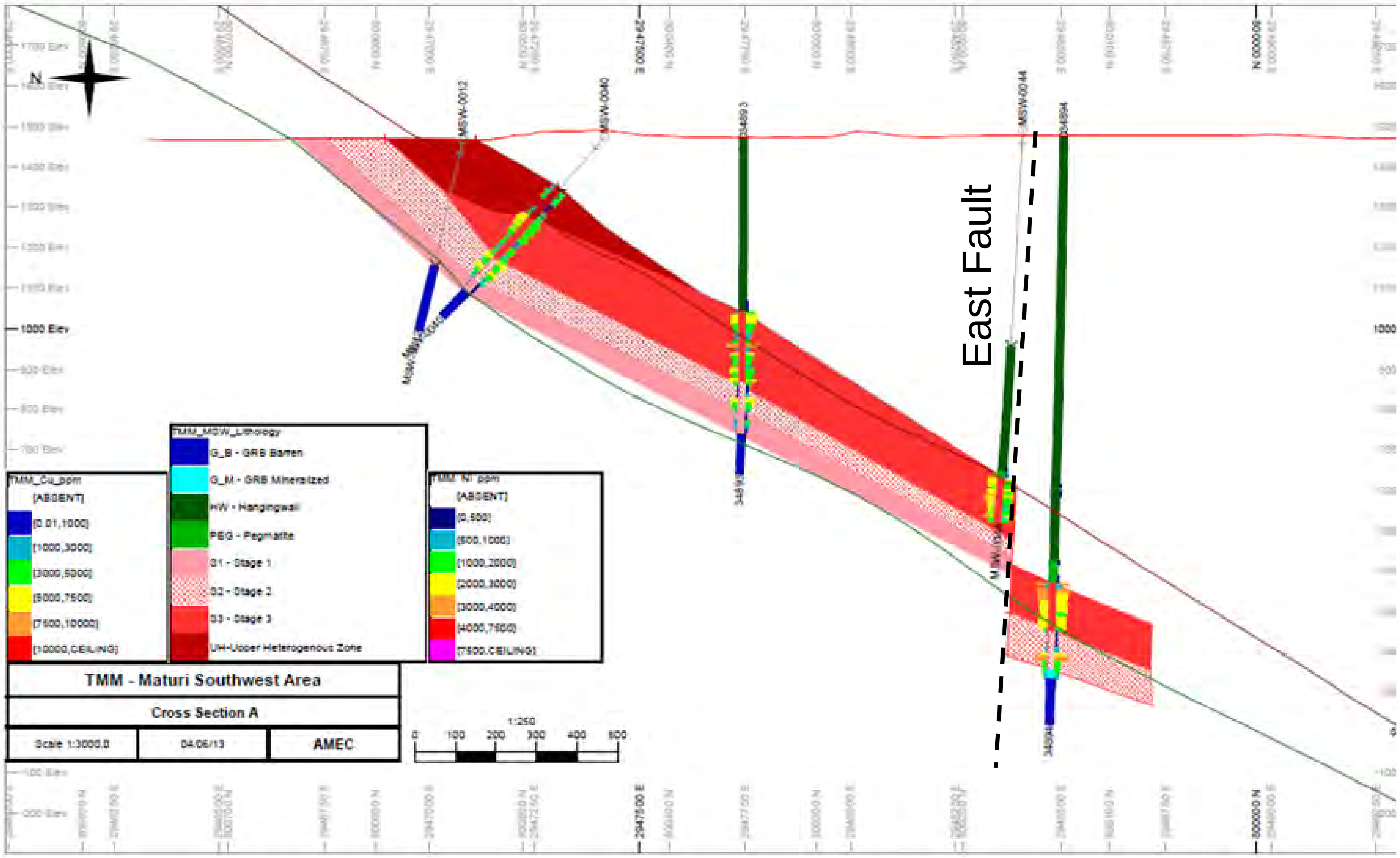
Note: Figure adapted by AMEC, 2014, from Routledge and Greenough, 2006

Figure 7-3: Maturi Section 45 Lithology (see Figure 7-2 for location)



Note: Figure prepared by AMEC, 2014. Columns are, from left to right, Cu, Lithology, Ni as indicated in the legends. The interpreted lithology is indicated on the section and corresponds to the central legend.

Figure 7-4: Maturi Southwest Section B (see Figure 7-2 for location)



Note: Figure prepared by AMEC, 2014. S1 – Stage 1 anorthositic rocks; S2 – Stage 2 troctolitic rocks; S3 – Stage 3 melatroctolitic rocks

In addition to the stratigraphy within the BMZ, the footwall was subdivided into three stratigraphic units for resource estimation purposes. The individual BMZ and footwall units are discussed below.

7.4.1.1.1 Upper Heterogeneous Unit (UH)

The UH intrusive subunit occurs at the top of the BMZ and appears to be discontinuous remnants of an early intrusive along the base of the SKI. UH is generally troctolitic to melatroctolitic in composition, and chemically and texturally quite similar to S3 rocks, but generally lacks sulfide minerals. Base and precious metals grades in these early melatroctolitic rocks are uniformly low with few samples returning >0.2% Cu or any significant precious metals. UH may be the earliest intrusion in the BMZ, but its relationship with S1 is not clear.

7.4.1.1.2 Stage 1 (S1)

The majority of S1 is augite troctolite and olivine gabbro. S1 or UH are considered to be the oldest intrusive phases in the BMZ, but it is not possible to determine which is, in fact, the oldest. S1 is much thinner under S2 and S3 rocks to the northwest, where it has been thermally and/or mechanically eroded from the sequence. Under the main part of the deposit, only scattered remnants of S1 remain.

Base and precious metals grades are uniformly low with rare Cu grades above 0.1%. Total precious metals grades average about 0.1 ppm.

7.4.1.1.3 Stage 2 (S2)

S2 is composed of augite troctolite, troctolite, and anorthositic troctolite and is thus somewhat less mafic in composition than the S3. S2 is extensive throughout the deposit; however, it is somewhat less continuous than S3 (refer to Figure 7-3). The distribution of S2 is notably patchy in the central portion of Maturi, where it may have been thermally and mechanically eroded by the later S3 intrusive subunit.

Metal grades are significantly lower in S2 than in S3, especially Pd, Pt, and Au.

7.4.1.1.4 Stage 3 (S3)

S3 consists of a heterogeneous mix of melatroctolite and troctolite with minor augite troctolite and anorthositic troctolite that is extensive throughout the deposit. The transition from UH to S3 is generally marked by a sharp increase in sulfide minerals.

Within S3, a number of troctolitic intercepts occur. Most troctolite intercepts are well mineralized, but some contain low grades of base and precious metals. These low-grade intercepts are interpreted to be either inclusions of pre-existing units (S1 and S2), or phenocryst-poor selvages to magmatic lobes within S3.

S3 hosts the highest and most consistent base and precious metals grades.

7.4.1.1.5 Giants Range Batholith (GRB)

The GRB forms the footwall to the Duluth Complex at Maturi and is locally mineralized. Mineralization was divided into three domains at Maturi: nickel-rich sulfide mineralized (G_N), disseminated sulfide mineralized (G_M), and barren (G_B). At Maturi Southwest, no nickel-rich material was identified. The nickel-rich domain is taken to be all of the material below the BMZ and above the last appearance of approximately equal Cu and Ni concentrations. Grades may locally be very high, but typically run in the range of 0.2 to 0.5% Cu + Ni. Below the G_N is the disseminated mineralized GRB (G_M) which is typically more copper-rich with respect to nickel but lower grade. The bottom of this domain was placed at the base of significant disseminated mineralization, usually at a cutoff of 0.1 to 0.2% Cu. Any GRB intervals below significant disseminated or Ni-rich mineralized GRB were considered to be barren (G_B).

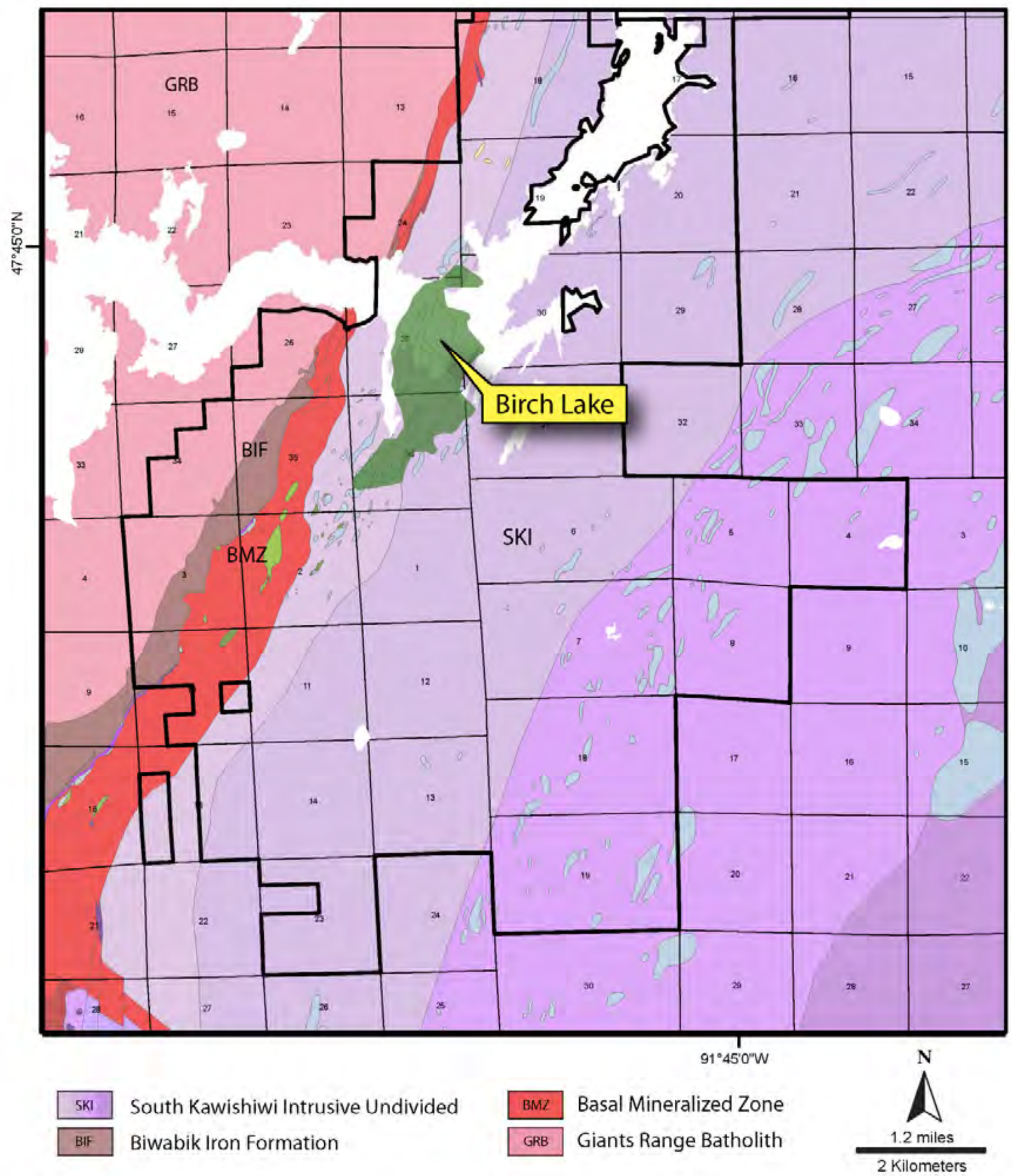
7.4.1.2 Birch Lake

The geology at Birch Lake is similar to Maturi but distinct because the BMZ of the SKI in this area includes numerous ultramafic and oxide (magnetite/ilmenite/chromite) layers that are not present at Maturi or Maturi Southwest (Routledge, 2004) (Figure 7-5; Figure 7-6). Relative to Maturi and Maturi Southwest, Birch Lake contains significantly more ultramafic intrusive rocks.

TMM and AMEC geologists delineated three intrusive subunits at Birch Lake: an upper melatroctolitic intrusive similar to S3 at Maturi (BL_MT), a lower troctolitic intrusive similar to S1 at Maturi (BL_T), and a basal hybrid rock sequence unique to Birch Lake (BL_HX; Figure 7-6). The thickness of all three units is quite variable, but the stratigraphic succession does not vary across the deposit (Figure 7-6). Any of the three units can be missing from a specific drill hole.

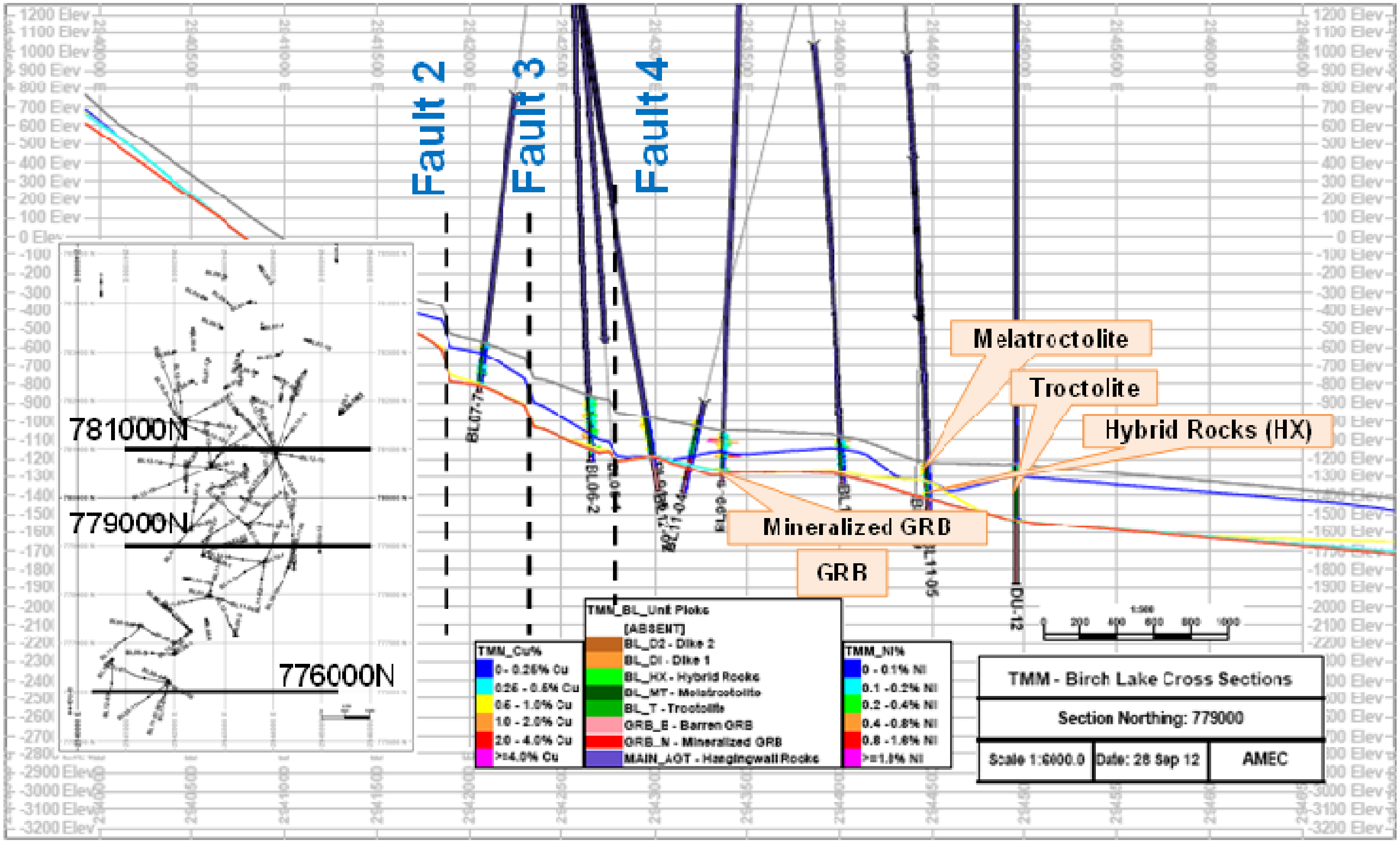
The upper melatroctolitic sequence (BL_MT) hosts the highest grade mineralization and is correlative across the deposit. BL_MT is similar to, but somewhat more mafic than S3 at Maturi. Immediately below BL_MT is a lower-grade troctolitic sequence (BL_T). BL_T is correlative over much of the deposit and is somewhat similar to S1 at Maturi. The lithology at the base of the BMZ is locally a hybrid rock (BL_HX) that shows similarities to both BL_T and the underlying GRB. Much of the BL_HX unit may indeed be metasomatized GRB, but local magmatic oxide layers consisting of magnetite and ilmenite in variable proportions indicate that some of the unit is a troctolitic intrusive that has assimilated footwall rocks including Biwabik Iron Formation, Virginia Formation and Giants Range Batholith.

Figure 7-5: Birch Lake Property Geology



Note: Figure adapted from Routledge and Greenough, 2006.

Figure 7-6: Section 779000N at Birch Lake



Note: Figure prepared by AMEC, 2014.

7.4.1.2.1 Melatroctolite (BL_MT)

The top of BL_MT is easily picked by the first appearance of 6% Mg in down-hole plots. The top of BL_MT generally correlates with the top of the mineralized interval, but Cu–Ni mineralization may begin somewhat above or below this contact. Almost all of the significant Cu–Ni and precious metals mineralization in the Birch Lake deposit is hosted by the BL_MT.

7.4.1.2.2 Troctolite (BL_T)

BL_T is a heterogeneous troctolite that ranges from very fine-grained to pegmatoidal, but medium-grained troctolites are most common. Base and precious metals grades in BL_T are uniformly lower than in BL_MT. Locally, the top of BL_T is mineralized, and there are small mineralized zones near the base of BL_T.

7.4.1.2.3 Basal Hybrid Zone (BL_HX)

BL_HX is a hybrid rock. It is marked by an abrupt increase in P in the lower portions of the BMZ and erratic Sr, Ba, Mg, Mn, and V concentrations, possibly because of assimilation of footwall rock, or metasomatism of GRB and other footwall rocks. Fe ranges from 2 to 45% largely because of assimilation of Biwabik Iron Formation, but there are local magmatic magnetite intervals. These rocks are present to some extent in most holes.

7.4.1.2.4 Other Mafic Intrusive Rocks

A large troctolite sill (BL_D2) in the footwall that appears to intrude into the base of the BMZ occurs in the extreme southern part of the deposit and largely truncates the mineralization. This sill is largely devoid of mineralization.

A second isolated troctolite intrusion into the footwall, possibly a dike (BL_DI), is known only from two widely-separated drill holes (BL11-08 and BL12-01). The orientation of this body is in question because of the limited number of intercepts. Copper and PGE grades are significant; however, Ni grades are very low.

7.4.1.2.5 GRB

The footwall at Birch Lake consists of locally mineralized GRB, Biwabik Iron Formation, and Virginia Formation. That mineralization has been identified as GRB_M and is modeled separately. Mineralization generally consists of disseminated Cu-Ni sulfides. Local massive sulfide veins and bodies contribute significantly to the Cu-Ni grade. Non-mineralized material is identified as GRB_B.

7.4.1.3 Spruce Road

Troctolitic rocks comprise much of the SKI at Spruce Road and carry abundant rafted basement inclusions of sedimentary hornfels, basaltic hornfels, anorthosite and iron formation (Routledge and Cox, 2007) (Figure 7-2; Figure 7-7). Inclusions are mostly barren. In contrast to the Maturi and Birch Lake deposits, there does not appear to be any specific correlation of mineralization to lithology and there is no key unit or hanging wall marker horizon, such as the PEG that overlies the mineralized unit at Maturi.

Typical features of the rocks are discontinuous layering, variable textures and common inclusions and erratic disseminated copper–nickel mineralization. The basal contact with the GRB locally contains orbicular blocks of troctolite, referred to by ACNC geologists as Spruce breccia.

There is some uncertainty as to the attitude and geometry of the mineralization, but, for the purpose of resource estimation, mineralization trends are assumed to parallel intrusive layering and conform to the overall geometry of the SKI, which, based on review of cross sections, is a reasonable assumption.

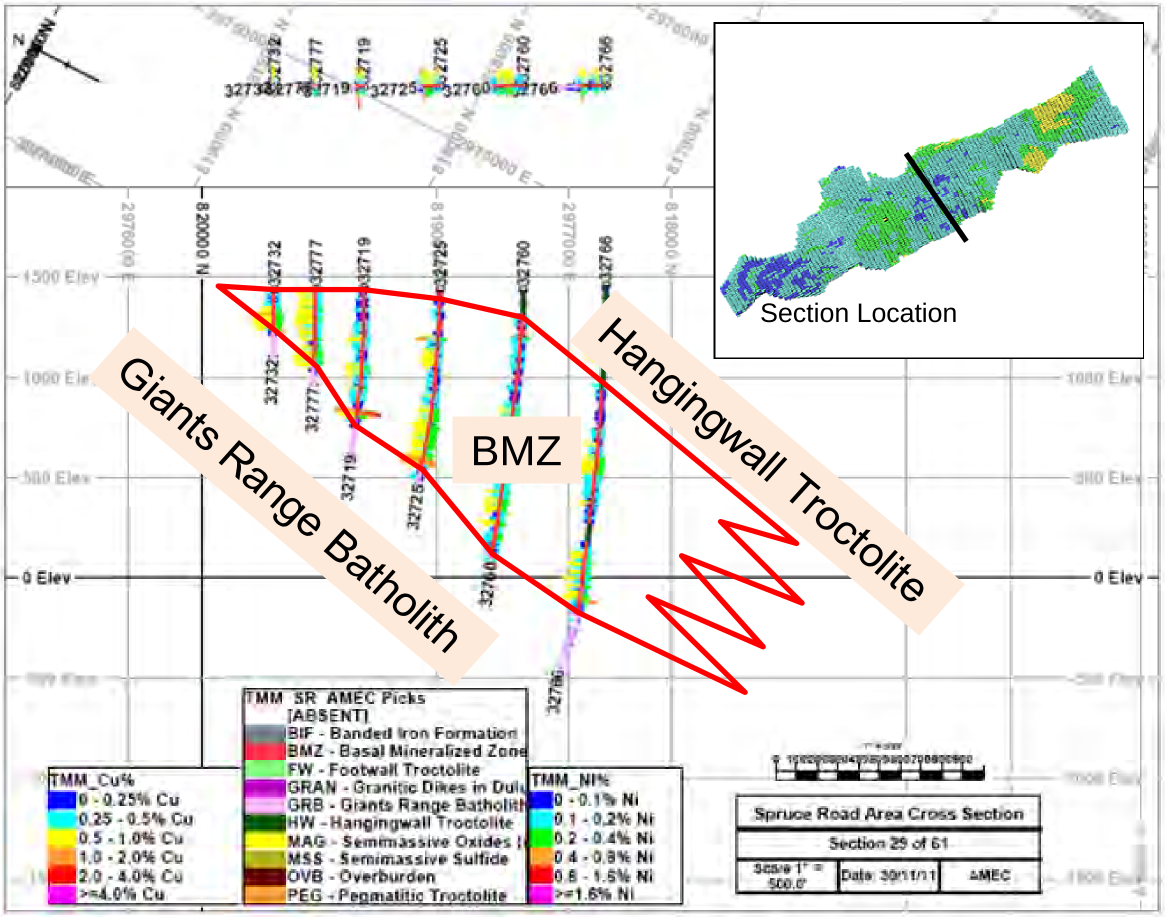
TMM did not reinterpret the geology at Spruce Road.

7.4.2 Alteration

At all four deposits, three main types of alteration were noted. Alterations exhibit no obvious relationship to mineralization but are logged so that such relationships can be identified in the future if they are present:

- Saussuritization (replacement of plagioclase by fined-grained aggregates of zoisite, epidote, albite, calcite, sericite, and zeolites) is found throughout all deposits
- Serpentinization (replacement of olivine by serpentine) is common in ultramafic packages and is noted where present
- Uralitization (the alteration of an igneous rock in which pyroxene is changed to amphibole) is commonly encountered and is also logged.

Figure 7-7: Spruce Road Cross Section



Note: Figure prepared by AMEC, 2014

7.4.3 Mineralization

In all four deposits, mineralization consists of 1% to 5% disseminated chalcopyrite, cubanite, talnakhite, pyrrhotite, and pentlandite in a tabular zone, parallel to the contact. Except at Spruce Road, better grades of copper, nickel and PGEs are associated with more mafic units located near the top of the BMZ, and there is excellent continuity of widths and values from hole to hole and section to section.

Magmatic sulfide mineralization in the South Kawishiwi Intrusion is restricted to the BMZ but rarely can be found in the overlying PEG (including ultramafic) units and in the footwall granitoids as well (Gál et al., 2010). Sulfides are usually disseminated-patchy and interstitial to the host silicates.

7.4.3.1 Maturi

At Maturi, the most common sulfide minerals are as follows:

- Pentlandite is also abundant as flame-like exsolution lamellae in pyrrhotite (Gál et al., 2010)
- Chalcopyrite occurs as interstitial patches between silicates, replacing pyrrhotite, pentlandite and silicates (pyroxene and plagioclase; Gál et al., 2010). Chalcopyrite also forms rounded primary inclusions in plagioclase and very rarely in clinopyroxene. Oriented star-shaped exsolution lamellae of sphalerite in chalcopyrite were noted
- Cubanite is always present in the form of exsolution lamellae in chalcopyrite (Gál et al, 2010)
- Talnakhite occurs as irregular patches associated with chalcopyrite
- Pyrrhotite forms anhedral grains often showing oriented lamellae of different Fe:S ratios (Gál et al., 2010). Pyrrhotite is usually intergrown with rounded pentlandite grains.

Sulfide minerals often occur in micro-scale (1–5 µm thick) veinlets crosscutting all silicate phases and interconnecting interstitial sulfide patches (Gál et al., 2010). These veinlets are primarily filled with chalcopyrite and to lesser extent, cubanite as exsolution lamellae; however, some minor amount of pentlandite and pyrrhotite can also be found in such textural positions. The vein-filling occurrences of sulfides imply that, after the solidification of the silicate host rock, the immiscible sulfide melt was still in liquid state and could migrate through micro-cracks in the rock.

Bornite, covellite, talnakhite, and millerite occur in subordinate amounts and are products of late-stage differentiation of the crystallizing sulfide melt. These minerals occur as replacing phases of chalcopyrite and along grain boundaries in sulfide patches and silicates (Gál et al., 2010).

Significant Cu–Ni mineralization occurs at the top of the footwall Giants Range Batholith, is hosted within the contact thermal aureole to the Duluth Complex, and is interpreted to be directly derived from the Duluth Complex. Mineralization in the footwall occurs in approximately 85% of the holes drilled to date, but many of the holes without footwall mineralization did not penetrate to sufficient depth to encounter the footwall. The sulfide mineral assemblage in the footwall is the same as in the BMZ, being dominated by chalcopyrite and pyrrhotite with lesser cubanite, pentlandite, bornite, and talnakhite. Pentlandite is the principal nickel mineral, although small amounts of nickel also occur in talnakhite and pyrrhotite. Chalcopyrite, cubanite, talnakhite, and bornite are the principal copper-bearing minerals. A number of localized, Ni-rich massive sulfide bodies have been encountered by drilling at and below the footwall contact. These bodies are as thick as 18.5 ft (5.64 m) and tend to have much higher Ni:Cu ratios than the disseminated mineralization in the BMZ.

Platinum group minerals (PGMs) have been found in various textural positions (Gál et al, 2010) but most commonly occur as finely disseminated grains within sulfide patches. Pyrrhotite and pentlandite are the preferred hosts; however, chalcopyrite can host PGMs. Work by Gál et al. (2010) indicates that these grains are mostly Pt–Pd–bismuth–tellurides (michenerite, moncheite) or Pd–Sn-bearing phases (paolovite) in composition. Rare grains of Ir–arsenides (irarsenite) are enclosed in pyrrhotite.

The largest concentrations of PGMs occur along the grain boundaries of plagioclase and massive sulfide patches or in thin sulfide veinlets. In such places, Ca-alteration of plagioclase is almost always present with some amount of chlorite or serpentine. Grain boundaries of sulfides and biotite or apatite also host PGMs. Most of the Pt–Pd–bismuth–tellurides (michenerite, moncheite, polarite/sobolevskite) and sperrylite are located in such positions.

PGMs not associated with sulfides are less abundant. Some of the grains were found along the boundary of K-feldspar and quartz in a granophyric segregation near to the footwall contact, others have been identified in sericitized plagioclase or in K-feldspar in a felsic mass close to abundant apatite inclusions.

PGMs also occur in semi-massive, net-textured sulfide patches associated with felsic inclusions or quartz pegmatite, clearly showing evidence that during mixing of felsic material originating from the footwall and the intruding troctolitic magma, these metals are mobile and may be concentrated in the felsic material.

7.4.3.2 Birch Lake

At Birch Lake, mineralogical investigations consisted of:

- Reflected light microscope and scanning electron microscope study of polished thin sections prepared from heavy minerals concentrated by heavy liquid separation of crushed and ground core (Cabri, 2002). These samples had relatively high PGE grades
- Detailed petrography and electron microprobe work on drill core from four holes has also been done by the University of Minnesota and the University of Minnesota Natural Resource Research Institute (NRRI); (Marma et al., 2002).

Microscopy on Birch Lake samples by Cabri (2002) identified the major sulfide minerals as chalcopyrite and undefined members of the chalcopyrite family, possibly talnakhite, mooihoekite, putoranite, and/or haycockite. Oxide minerals include chromian spinel, ilmenite, magnetite, and chromite. Native copper and troilite occur locally. Other identified minerals include bornite, chalcocite, and cubanite as well as nickel sulfide minerals heazlewoodite and pentlandite. Trace amounts of altaite, digenite, frobergite, galena, mackinawite, millerite, sphalerite and unidentified PGE-bearing minerals, native silver, silver telluride and alloys of silver and gold were identified. Pentlandite contains as much as 2.12% Co. Iron sulfide gangue is pyrrhotite and troilite. PGMs occur as various fine-grained Pd tellurides with other Pt, Os, Ru, Au, Ag, Te, and Bi bearing minerals. Ninety percent of the PGMs are associated with copper sulfides as discrete grains attached to sulfides, as sulfide inclusions, and at the margins between sulfides and gangue silicates (Cabri, 2002). The PGMs locally form halos around, or are included in, interstitial copper sulfides, pyroxenes, secondary amphiboles and biotite. PGMs are also remobilized in chlorite, serpentine, or secondary magnetite.

7.4.3.3 Spruce Road

Work by the University of Minnesota (Inco, 1966) on concentrates from Spruce Road on behalf of ACNC found that chalcopyrite, cubanite, pyrrhotite and pentlandite were the primary sulfide minerals. About 70% of the chalcopyrite was present as individual grains or as compound grains with pyrrhotite. Compound grain size was 100 µm to 1,800 µm and averaged about 500 µm. The balance of the chalcopyrite occurs as minute inclusions in olivine corona structures or in pyrrhotite, magnetite, and olivine. Cubanite is not common but, when present, is always associated with chalcopyrite. Pyrrhotite has a similar mode of occurrence to chalcopyrite. Pentlandite occurs as compound grains with pyrrhotite and chalcopyrite or included in pyrrhotite.

SGS Lakefield Research Limited (SGS Lakefield) performed a mineralogical study of core samples from Wallbridge's drill hole WM-001 (Soever, 2000). SGS Lakefield identified pentlandite, chalcopyrite, cubanite, bornite, mackinawite, violarite, pyrrhotite, pyrite, magnetite, and ilmenite. Soever (2007) notes that chalcopyrite and cubanite were identified as the main copper minerals with particle size ranges of 5 µm to 250 µm and 20 µm to 500 µm, respectively. Nickel was mainly present as pentlandite with grain sizes 2 µm to 250 µm and occurring as exsolution flames in pyrrhotite.

7.4.3.4 QEMSCAN Studies

In 2007, Duluth commissioned a QEMSCAN mineralogical study of samples from Maturi by SGS Lakefield Research Limited (SGS Lakefield, 2007) in conjunction with the bench-scale flotation test work on a composite sample of drill core from the Maturi deposit to determine the deportment of copper and nickel mineralization. An important observation was that a significant portion of the nickel (about 22%) is non-sulfide and potentially non-recoverable.

TMM performed QEMSCAN analysis of 118 samples from 15 drill holes at Maturi, 66 samples from eight drill holes at Maturi Southwest, and 58 samples from eight drill holes from Birch Lake. Those samples were also analyzed for major and trace elements by X-ray fluorescence spectrometry (XRF) and inductively-coupled plasma spectrometry (ICP) methods. Sulfur was determined by Leco. Precious metals were analyzed by fire assay. These methods were used to determine quantitative mineralogy and geochemistry of each sample in order to facilitate geometallurgical modeling.

7.4.4 Structural Geology

The Maturi deposit appears to occupy a gentle flexure in the contact that has formed a broad, easterly plunging embayment in the base of the SKI (Soever, 2000). In the Maturi area, evidence suggests that the Duluth Complex has not been significantly deformed since magma consolidation.

The Maturi Southwest area is more structurally complex than the Maturi area. The strike of the footwall contact of the Duluth Complex changes by almost 30^0 immediately north of Maturi Southwest, and the northern part of the deposit is located near the axis of the bend. Numerous faults, small to large, occur in the area. Many of these faults form breccia zones that appeared as broken core and effectively stopped drilling in a small number of holes. Based on geological logging of core and surface mapping (Peterson, 2008) two faults, called East and West, were recognized and included in the block model (refer to Figure 7-4).

The Birch Lake area has not been significantly deformed since magma consolidation, but it has been subjected to displacements along reactivated basement faults as well as cross faults. Mapped structures are mostly sub-vertical north–northeasterly striking faults and fault zones that are evident as linear features on air photos and topographic maps. Rowell (2002) believes that these faults have been active pre, syn, and post emplacement of the SKI and offset the mineralized U3 unit. Where exposed in parts of the SKI and footwall rocks, movement on these faults ranges from 10 ft to 400 ft.

West–northwest faults cut the northeasterly faults and show left lateral displacements in the south portion of the property and right lateral offsets under Birch Lake (Rowell, 2001; Pratt, 2010). These late faults have vertical displacements in the order of 30 ft to 400 ft and may be akin to transform faults that accompany rifting elsewhere. Interpreted faults were included in the Birch Lake geological model (refer to Figure 7-6).

7.5 Comments on Geology

The geology of the deposits, including the location of the mineralization and the associated rock types, is well known. Details of the relationship between mineralization and host rocks are well understood, and are adequate to support Mineral Resource and Mineral Reserve estimation and preliminary mine planning. AMEC considers the current geological models at Maturi, Maturi Southwest, and Birch Lake to be adequate to support resource estimation and preliminary mine planning. Additional drilling will be required to precisely locate the faults at Birch Lake.

The geological model for Spruce Road was not updated by TMM, and while adequate for Inferred Mineral Resources, it is not adequate to support higher resource classifications.

Continuing exploration is providing insight into the relationships between host rocks and mineralization and will support significantly more detailed interpretations for future resource estimates.

8.0 DEPOSIT TYPES

The Maturi, Maturi Southwest, Birch Lake, and Spruce Road deposits are classified as contact-type magmatic nickel–copper–platinum group element deposits which are a broad group of deposits containing nickel, copper, and PGEs occurring as sulfide concentrations associated with a variety of mafic and ultramafic magmatic rocks (Zientek, 2012; Eckstrand and Hulbert 2007). The magmas originate in the upper mantle and contain small amounts of nickel, copper, PGE, and variable but minor amounts of sulfur. The magmas ascend through the crust and cool as they encounter cooler crustal rocks. If the original sulfur content of the magma is sufficient, or if sulfur is added by assimilation of crustal wall rocks, a separate sulfide liquid forms as droplets dispersed throughout the magma. Because the partition coefficients of nickel, copper, and PGEs, as well as iron, favor sulfide liquid over silicate liquid, these elements preferentially concentrate in sulfide liquid droplets within the surrounding magma. The sulfide liquid droplets tend to sink toward the base of the magma because of their greater density, and can form massive sulfide concentrations. Alternately, as in the BMZ, sulfide liquid droplets adhere to phenocrysts in the magma by surface tension and are transported by moving magma. The sulfide liquid droplets remain disseminated in the magma. On further cooling, the sulfide liquid droplets crystallize to form disseminated Cu–Ni–PGE sulfide minerals. When crystallized, these sulfides are a rock-forming component of the intrusive body.

The mafic and ultramafic magmatic bodies that host the Ni–Cu sulfide ores are diverse in form and composition, and are subdivided by Eckstrand and Hulbert (2007) into the following four subtypes:

- A meteorite-impact mafic melt sheet that contains basal sulfide ores (Sudbury, Ontario is the only confirmed example)
- Rift and continental flood basalt-associated mafic sills and dike-like bodies (Noril'sk-Talnakh, Russia; Duluth Complex, Minnesota; Muskox, Nunavut)
- Komatiitic (magnesium-rich) volcanic flows and related sill-like intrusions (Thompson, Manitoba; Raglan and Marbridge, Quebec)
- Other mafic /ultramafic intrusions (Voisey's Bay, Labrador; Lynn Lake, Manitoba; Giant Mascot, British Columbia; Kotalahti, Finland; Råna, Norway; and Selebi-Phikwe, Botswana).

The Duluth Complex is associated with the Midcontinent Rift and continental flood basalt-associated mafic sills and dike-like bodies (Eckstrand and Hulbert, 2007). Ni–Cu deposits of the rift and continental flood basalt associated subtype are the products of the magmatism that accompanies intracrustal rifting events. These deposits are associated with large magma systems, and within these systems the Ni–Cu sulfide ores tend to be associated with conduits or feeders to the larger igneous masses (in this last respect, the Duluth Complex is an exception in which the low-grade Ni–Cu sulfides may not be associated with conduits or feeders but rather lobes of sulfide-enriched magmas). Much of the sulfur in the sulfide has been derived by contamination of the magma by incorporation of sulfur from adjoining wall rocks (Zientek, 2012).

Magmatic PGE deposits and Ni–Cu sulfide deposits are the source of essentially all of the world's PGEs (Eckstrand and Hulbert 2007).

9.0 EXPLORATION

9.1 Exploration

Exploration from 1951 to about 1998 is not well documented. Records indicate that geological mapping and surface geochemical sampling were used to trace the exposed extent of mineralization. Surface geological mapping helped establish the geological framework of the area and guided exploration. This work culminated in works such as the Phinney et al. (1969) map of the Gabbro Lake Quad and Peterson's (2008) mapping of the South Kawishiwi Intrusion.

Various geophysical surveys were performed with the objective of defining down-dip extensions of known mineralization and discovery of new mineralization. Existing records indicate that horizontal and vertical loop electromagnetic (EM) surveys, magnetometer surveys, induced polarization (IP), self-potential (SP), gravity surveys, magnetotelluric surveys, and various downhole methods have been used with limited success for near surface exploration. SP was successfully used to locate oxidized sulfides (gossan) below thin veneers of glacial till at the South Filson Creek Occurrence.

Controlled source audio frequency magnetotellurics (CSAMT) was employed by Franconia at Birch Lake in 2008 to map major faults, depth and thickness of mineralization, and map subsurface structures (Routledge, 2008). The survey was conducted on the ice of Birch Lake in February 2008 with lines crossing the north portion of the deposit. The method reasonably well mapped the basement granites and low resistivity above the contact (mineralization) as well as discontinuities as probable faults and low resistivity areas as possible intrusive bodies.

For the most part; however, the disseminated mineralization is too deep to be successfully explored using geophysical methods. For that reason, drilling has been the preferred method of exploration in all areas since about 1955 and is summarized in Section 10.

9.2 Comments on Section 9

Exploration of the TMM deposits has been done largely by drilling of the deposits. That work is consistent with industry-standard practices and is adequate to support resource estimation and preliminary mine planning.

10.0 DRILLING

10.1 Drilling Summary

Table 10-1 summarizes the drilling on the TMM properties. All drilling was completed with diamond tipped core tools. Drilling is considered “legacy” if it was completed before 2000 at Maturi, Maturi Southwest, and Birch Lake, and prior to 1999 at Spruce Road. Much of the legacy drilling reportedly utilized A-sized (1.067 in) core tools. Current drilling utilized P-, H-, and N-sized core tools. Most current holes were collared with H-sized tools (2.4 in) and reduced to N (2.155 in), and locally B (1.655 in) where drilling conditions did not permit larger core. P-sized tools (3.245 in) were used to collect metallurgical samples during the current exploration programs. The core diameters noted above are nominal. In some cases small variations in core diameter occurred because of the use of different tooling with the same hole diameter. An example is N-sized core—NTW (2.44 in), NQ (1.875 in), NX (2.155 in), and NQ2 (1.99 in) were all used during the course of the drill programs. Much of the current drilling at Maturi and Maturi Southwest used NQ2 (1.99 in) tools. At Birch Lake, most of the current holes were collared with PQ diameter tools.

During many of the site visits, drills were active on the Maturi and Maturi Southwest properties where AMEC observed drilling procedures. All procedures observed were consistent with industry-leading practices.

Recent drilling at Maturi, Maturi Southwest, and Birch Lake was completed by IDEA Drilling LLC, Virginia, MN, using a variety of different truck and skid mounted drill machines. Foraco (26 Plage de l'Estaque, 13016, Marseille, France) drilled a number of holes at Maturi in 2007–2008 using a variety of equipment. E.J. Longyear is reported to have drilled some of the Duval holes in the 1960s as well as some of the later ACNC holes. E.J. Longyear equipment is not documented. There is no record of the contractor for most of the legacy holes, but the record indicates that E.J. Longyear was active in the area as early as 1955 and may have drilled most of the holes for ACNC and the other contractors. Dr. H. Parker, who worked in the area in the 1960s, noted a prevalence of Longyear 38 and 44 drills used by E.J. Longyear for exploration in 1967-1969. Hole K-8 was drilled by Odgers using Boyles Brothers equipment.

At both Birch Lake and Maturi, pilot holes and wedges from those holes were utilized to complete holes through the BMZ and/or to obtain sample for metallurgical testing.

Table 10-1: Summary of Drilling on the TMM Properties (as of 4 February 2014)

	Number of Holes*	Feet Drilled
Maturi		
Current	690	1,413,292.0
Legacy	75	110,950.4
Total	765	1,524,242.4
Maturi Southwest	**Number of Holes***	**Feet Drilled**
Current	52	43,113.5
Legacy	16	26,556.0
Total	68	69,669.5
Birch Lake	**Number of Holes***	**Feet Drilled**
Current	243	297,299.1
Legacy	26	50,331.8
Total	269	347,630.9
Spruce Road	**Number of Holes**	**Feet Drilled**
Current	2	4,054.1
Legacy	232	137,429.6
Total	234	141,482.7
Total	**1,339**	**2,083,025.5**

* includes wedge holes
Note that not all holes in this table were ultimately used for resource estimation

10.2 Collar Surveying

10.2.1 Maturi and Maturi Southwest

ACNC collar surveys were originally recorded using ACNC's New Minnesota grid system and were converted to UTM NAD27 coordinates, which required a 66.9 ft shift west and 137.8 ft shift south, and a grid origin correction to 539,980.4E, 5,199,956.8N. Collars of legacy drill holes have been relocated in the field by the NRRI using global positioning system (GPS) instruments (Cox et al, 2009). Modern hole collars were initially located by hand-held GPS units and recorded in UTM coordinates (Cox et al, 2009). Completed drill hole collars were then surveyed using survey-grade GPS (Trimble Instruments) and reported in Minnesota State Plane coordinates (ft).

10.2.2 Birch Lake

Duval early reconnaissance drilling along fences of widely-spaced holes relied on pace-and-compass collar locations (Routledge and Galyen, 2010). Later drilling to delineate Cu–Ni–PGE mineralization employed hand-held GPS 12-channel instrumentation for collar locations, accurate to ±16.4 ft (Routledge and Galyen, 2010). In 2007, all pre-2006 drill collars that could be located in the field were surveyed by a registered land surveyor. All subsequent drill collars have been surveyed by a registered land surveyor (Routledge and Galyen, 2010).

10.2.3 Spruce Road

ACNC collar surveys were originally in ACNC's New Minnesota grid system and were converted to UTM NAD27 coordinates, which required a 66.9 ft shift west and 137.8 ft shift south, and a grid origin correction to 539,980.4E, 5,199,956.8N. Instruments used for these surveys are not recorded but were likely theodolites. TMM converted the UTM coordinates to Minnesota State Plane coordinates (ft).

10.3 Downhole Surveying

10.3.1 Maturi

ACNC performed downhole surveys using acid-tube inclination tests (Cox et al, 2009; Routledge and Greenough, 2006). The acid-tube test only determines the inclination and not the azimuth of the hole. No documentation exists as to whether or not an appropriate meniscus correction was applied to inclinations for angle holes (19 holes). This correction is required because the angle indicated by the surface of the acid in the tube is not the angle of the hole and must be corrected using experimental data. The position of the toe and mineralized intervals is somewhat uncertain because of the lack of azimuth deviation data. Inclination tests were taken at intervals ranging from 50 ft to 200 ft. According to Dr. H. Parker, a Tropari (magnetic) survey was attempted in legacy hole K8. There was sufficient magnetite in the rock that wide (45º) swings in azimuth were noted between close-spaced readings, making the instrument useless.

Duval, Newmont, Kennecott, and US Steel holes were all vertical with no downhole azimuths recorded. US Steel holes have acid-tube inclination tests and, of the others, only DU-03 and NM-03 have acid-tube inclination tests. The other historic holes have no downhole surveys.

Downhole surveys for the modern holes through MEX-045 were done using Flexit Smart Tool Instrumentation by Flexit AB. Readings were generally taken at approximately 20 ft intervals, coinciding with pulling drill rods in 20 ft lengths at the end of drilling the hole when the survey is run. The tool is a magnetic field-based instrument that can be affected by magnetic minerals. After MEX-045, downhole surveys were performed by gyroscopic tools using the magnetic tools as quality control measures. Most of the pre-MEX-045 holes were re-entered and resurveyed with a gyroscope. Gyroscope-based downhole surveys are unaffected by magnetic minerals in the surrounding rocks.

Review of the downhole survey data for the MEX series holes indicated that dips were generally reasonable, but some obvious spurious readings were rejected and adjustments made to the survey data (Cox et al, 2009).

For some inclined holes, the Flexit readings did not agree with the collar orientation recorded when the drill was set up. The orientation of those collars was checked and confirmed in the field. Discrepancies were corrected in the database. Consequently, the downhole deviation for five inclined holes was modeled based on the collar azimuth and dip and the average deviation of other MEX series holes. The positioning of the hole toes and mineralized intervals in three dimensions is now reasonable for drill spacings greater than 200 ft.

In summary, all of the angle holes and most of the vertical holes have been surveyed using a Flexit gyroscopic tool. A few of the vertical holes drilled before MEX-45 were not surveyed due to blockages in the holes or bad hole conditions.

10.3.2 Maturi Southwest

Downhole surveys for legacy holes were performed using acid-tubes. Acid-tube surveys provide accurate inclination data but do not provide azimuth information. In all cases, the holes were -90±10º; thus meniscus corrections were not necessary. Most holes were reasonably straight to 700 or more ft; thus the surveys are sufficiently accurate to support resource estimation at an Inferred Mineral Resource level of confidence.

2012–2013 downhole surveys were done by Minex, a local downhole survey contractor, using a Reflex gyroscopic instrument. Each hole was surveyed on 20 ft increments down the hole.

10.3.3 Birch Lake

Duval's early reconnaissance drilling relied on acid-tube inclination tests for downhole deviation except where wedging was performed (Routledge and Galyen, 2010).

Later drilling to delineate Cu–Ni–PGE mineralization employed Flexit magnetic tools for surveys to track azimuth and inclination deviations downhole (Routledge and Galyen, 2010). Beginning in 2007 with drill hole BL07-5, all downhole surveys were performed with gyroscopic instruments.

Most of the pilot and wedge holes were surveyed from the bottom of the hole to the collar. In 2011, AMEC noted a number of discrepancies with survey azimuths while verifying data. The azimuths were as much as 180^{o} different between the various pilot holes and wedges in a single location. In 2012, the last wedge of all accessible holes was resurveyed as deeply as possible. In some cases, caving prevented complete resurveys. These new surveys were used to correct the problematical azimuth data.

10.3.4 Spruce Road

Downhole surveys for ACNC holes were acid-tube inclination tests (Routledge and Cox, 2007). Acid-tube inclination tests were taken at intervals ranging from 50 ft to 200 ft but generally at 100 ft. No documentation exists as to whether or not an appropriate meniscus correction was applied to inclinations for angle holes (22 holes).

The position of the hole toe and mineralized intervals is somewhat uncertain because of the lack of azimuth deviation data. In general acid-tube inclination deviation was plotted in the grid north direction (approximately 331.6° azimuth) for the purpose of geologic interpretation and wireframing.

10.4 Drilling Data and Results

Drilling results are individually summarized in the following sub-sections for each of the three properties.

10.4.1 Maturi

Table 10-2 summarizes drilling at Maturi by year and operator. A total of 765 holes (1,524,232.4 ft; including wedge holes) were drilled in the area from 1951 to 2014. This included 75 legacy holes (110,950.40 ft) and 690 TMM holes (1,413,282.00 ft). In addition, ACNC excavated a 1,095 ft deep shaft in 1968. Collar locations are shown on Figure 10-1. Figure 7-3 in Section 7 shows a cross section across the main part of Maturi.

10.4.2 Maturi Southwest

Table 10-3 summarizes drilling at Maturi Southwest by year and operator and includes some holes that were not used in the resource estimate. A total of 68 holes (69,669.5 ft) were drilled in the area. Table 10-4 summarizes drilling at Maturi Southwest used for resource estimation. Collar locations are shown on Figure 10-2. Figure 7-4 in Section 7 is a cross section of Maturi Southwest.

10.4.3 Birch Lake

At Birch Lake a total of 269 holes (347,631 ft) were completed, including wedge holes, to explore the property (see Table 10-5, Figure 10-3, and Figure 7-6 in Section 7). A total of 154 wedge holes (198, 360 ft) were drilled to confirm the location and tenor of mineralization and to obtain sample for additional assays and metallurgical testing.

10.4.4 Spruce Road

The Spruce Road database contains 234 holes (141,482.7 ft) (Table 10-6). The area included in the resource estimate contains 210 holes (118,303 ft; Figure 10-4). Figure 7-7 in Section 7 is a representative cross section at Spruce Road.

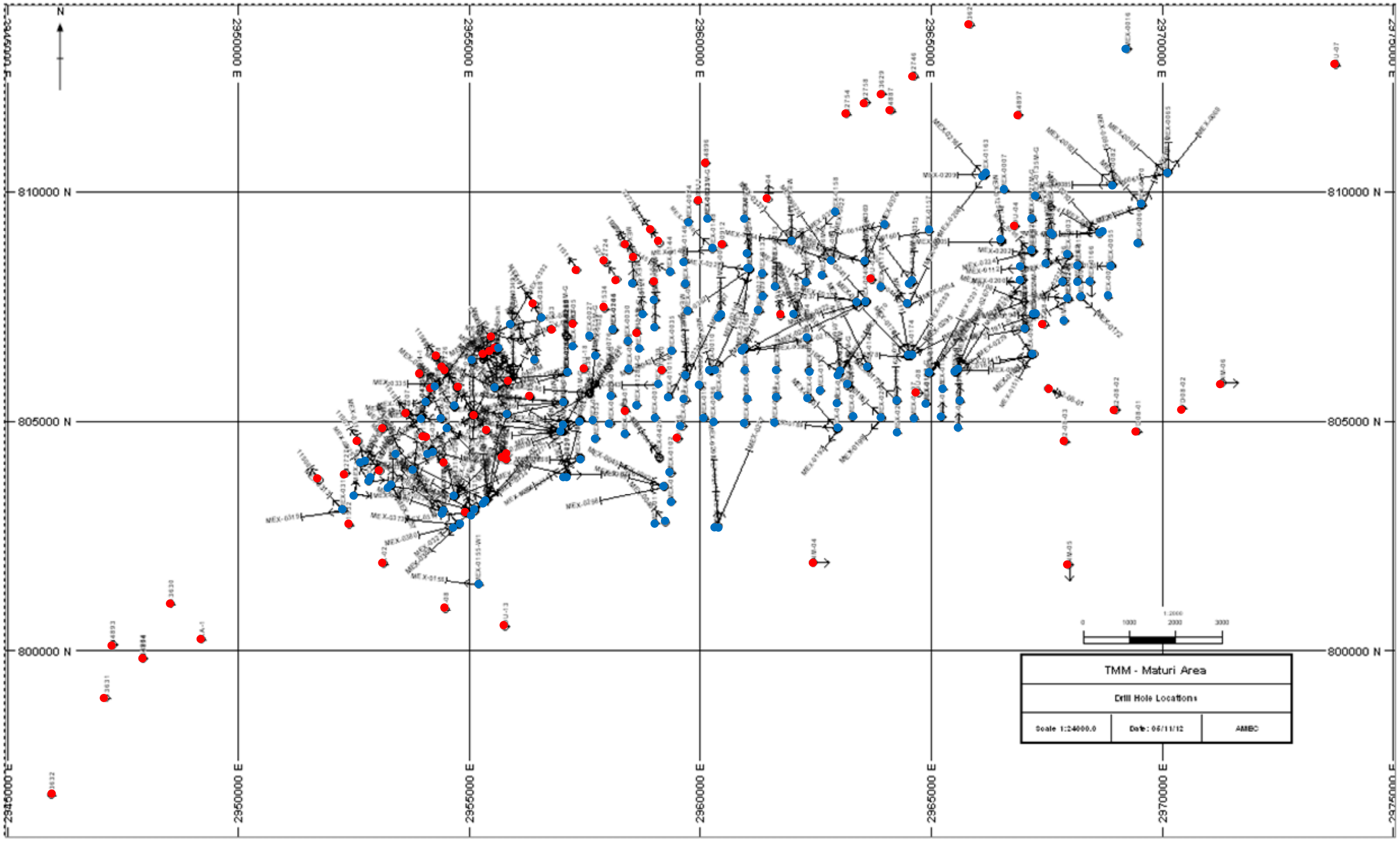
Drill logs indicate that most of the core was AX diameter and that drilling was done by E. J. Longyear/Longyear Canada (Routledge and Cox, 2007). Two short PQ holes were drilled in 1971.

Table 10-2: Summary of Maturi Drilling by Year Completed (as of 4 February 2014)

Year	Operator	Type	Number of Holes	Feet Drilled
1951*	Childers–Whiteside	Core	1	188.0
1954*	ACNC	Core	14	12,603.0
1955*	ACNC	Core	7	11,419.0
1966*	ACNC	Core	8	6,817.0
1967*	Duval	Core	2	5,675.0
	Hanna	Core	1	2,240.0
	ACNC	Core	8	9,700.5
	Newmont	Core	3	11,089.0
1968*	Duval	Core	2	6,480.0
	Hanna	Core	2	5,117.0
	ACNC	Core	13	3,075.9
1969*	Duval	Core	3	8,755.0
	ACNC	Core	1	1,095.0
1970*	Duval	Core	1	3,806.0
	ACNC	Core	3	5,196.0
	Newmont	Core	1	5,225.0
1971*	Newmont	Core	1	4,818.0
1977*	Duval	Core	3	6,485.0
2000*	Wallbridge	Core	1	1,166.0
2006	Duluth	Core	10	30,922.0
2007	Duluth	Core	78	224,024.4
2008	Duluth	Core	132	243,229.1
2009	Duluth	Core	2	4,725.0
2010	Duluth	Core	26	94,693.5
	TMM	Core	25	104,230.5
2011	TMM	Core	154	379,726.5
2012	TMM	Core	206	267,432.5
2013	TMM	Core	48	53,888.5
2014	TMM	Core	9	10,410.0
Total			*765*	*1,524,232.4*

Note: * indicates legacy drilling

Figure 10-1: Maturi Drill Hole Locations (legacy holes in red; TMM holes in blue)



Note: Figure prepared by AMEC 2014

Table 10-3: Drill Summary at Maturi Southwest (all holes*)

Year	Operator	Type	Number of Holes	Feet Drilled
1957**	ACNC	Core	5	3,530
1968**	Bear Creek	Core	2	2,796
1969**	Bear Creek	Core	1	3,179
1969**	ACNC	Core	2	2,300
1970**	Bear Creek	Core	2	2,650
1976**	Duval	Core	2	9,394
1990**	Lehmann	Core	2	2,707
2013	TMM	Core	52	43,113.5
Total			*68*	*69,669.5*

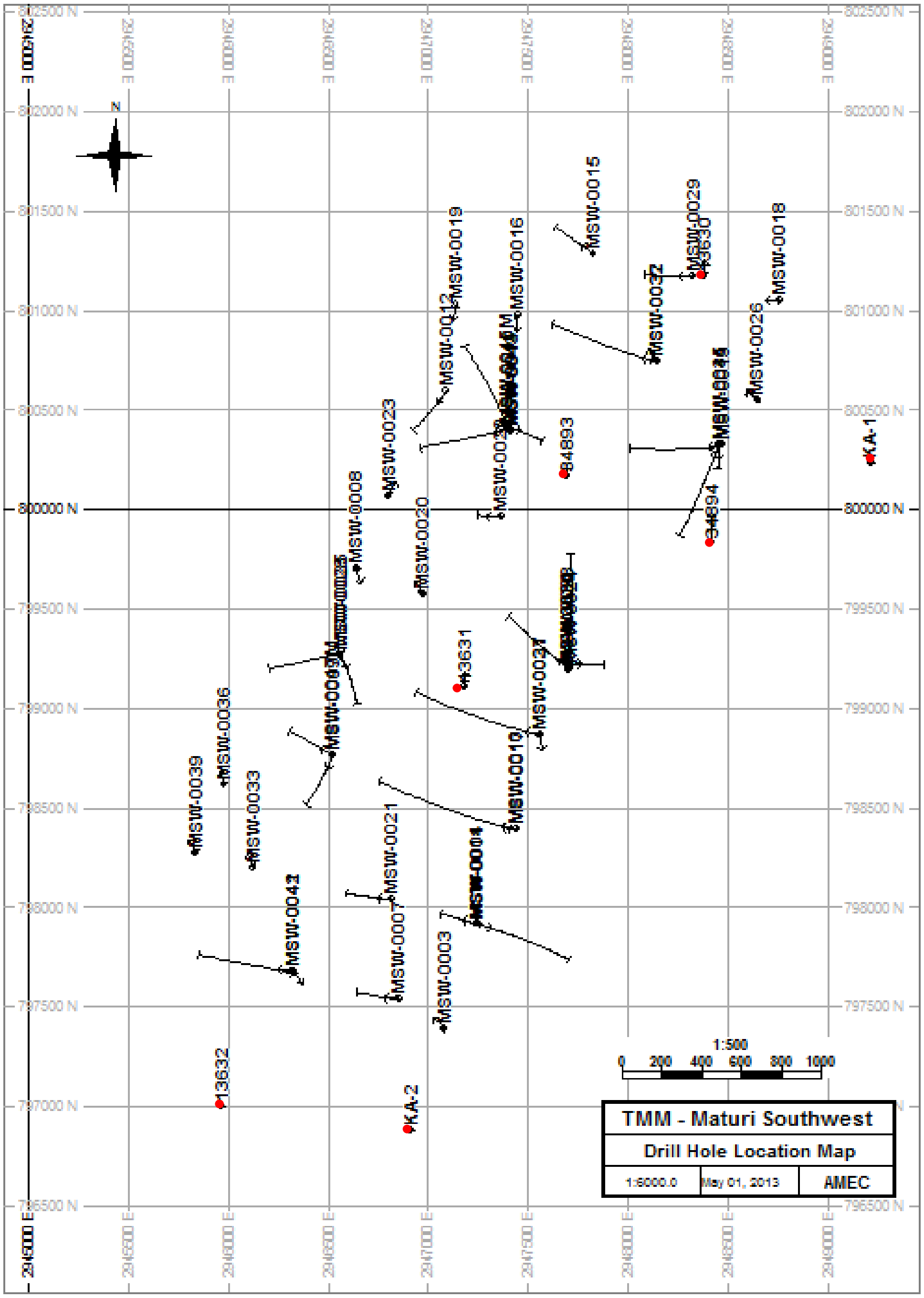
* Not all of these holes were used for resource estimation
** Considered to be legacy drilling

Table 10-4: Maturi Southwest Drill Campaigns (Drill holes used in grade estimate)

Year	Company	Type	Number of Holes	Feet Drilled
1960's*	ACNC	Core	5	4,447.0
1960's*	Bear Creek	Core	2	2,696.0
2013	TMM	Core	42	36,033.5
Total			*49*	*43,276.5*

***** Legacy data

Figure 10-2: Maturi Southwest Drill Hole Location Map (legacy holes in red)



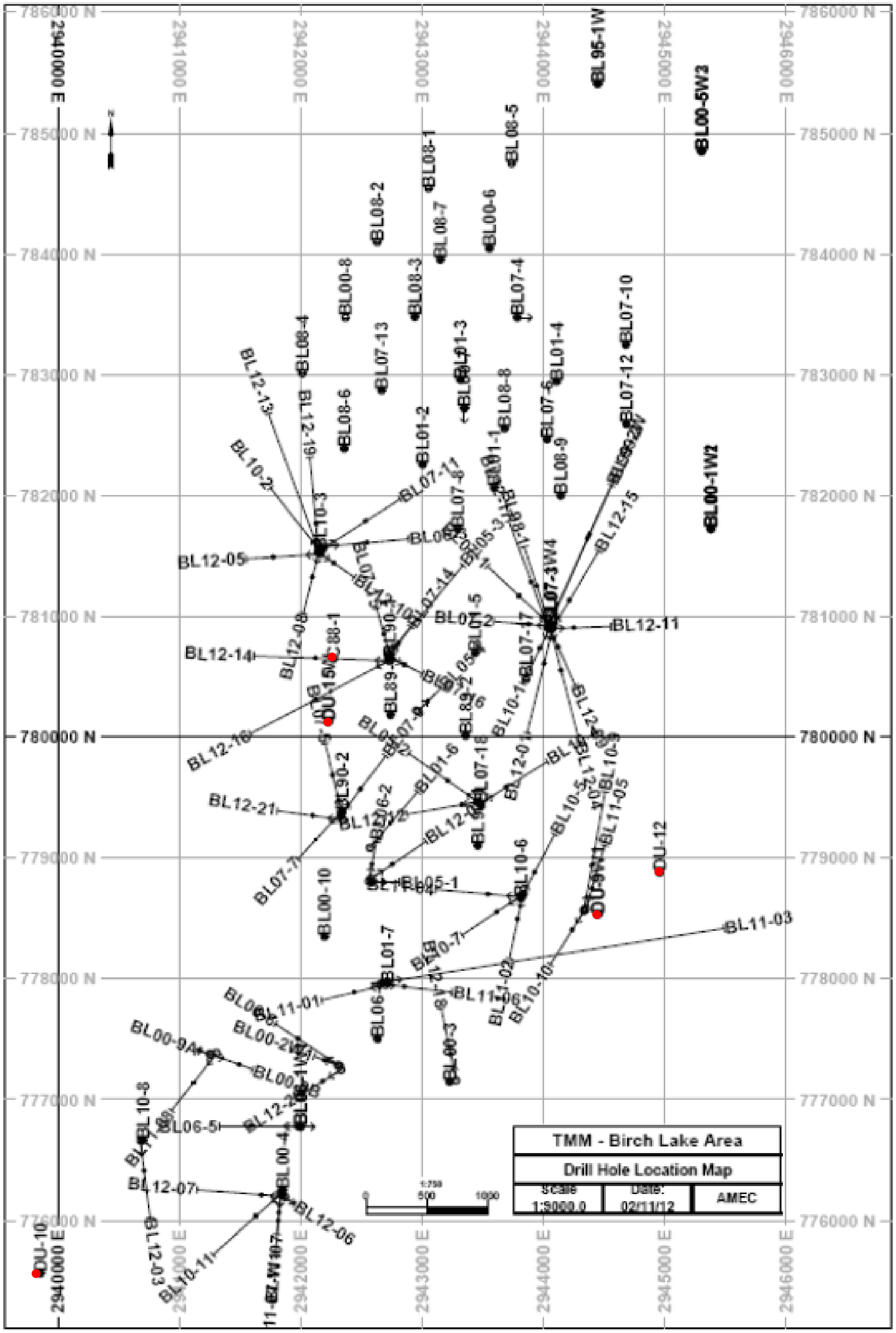
Note: Figure prepared by AMEC 2014

Table 10-5: Summary of Drilling at Birch Lake by Year (excluding wedge holes)

Year	Operator	Type	Number of Holes	Feet Drilled
1969-1976*	Duval	Core	6	20,469
1967*	Kennecott	Core	1	1,883
1988*	Lehmann	Core	1	2,402
1989*	Lehmann	Core	1	2,569
1990*	Lehmann	Core	4	9,180
1995*	Lehmann	Core	1	1,961
1998*	Lehmann	Core	1	2,559
1999*	Lehmann	Core	2	5,368
2000*	Lehmann	Core	12	26,975
2001*	Lehmann	Core	7	16,766
2005	Franconia	Core	4	10,973
2006	Franconia	Core	6	14,705
2007	Franconia	Core	16	39,115
2008	Franconia	Core	11	20,052
2010	Franconia	Core	11	29,238
2011	TMM	Core	9	25,248
2012	TMM	Core	21	57,697
Total			*114*	*287,160*

Note: * indicates legacy data

Figure 10-3: Birch Lake Drill Hole Locations (legacy holes in red)



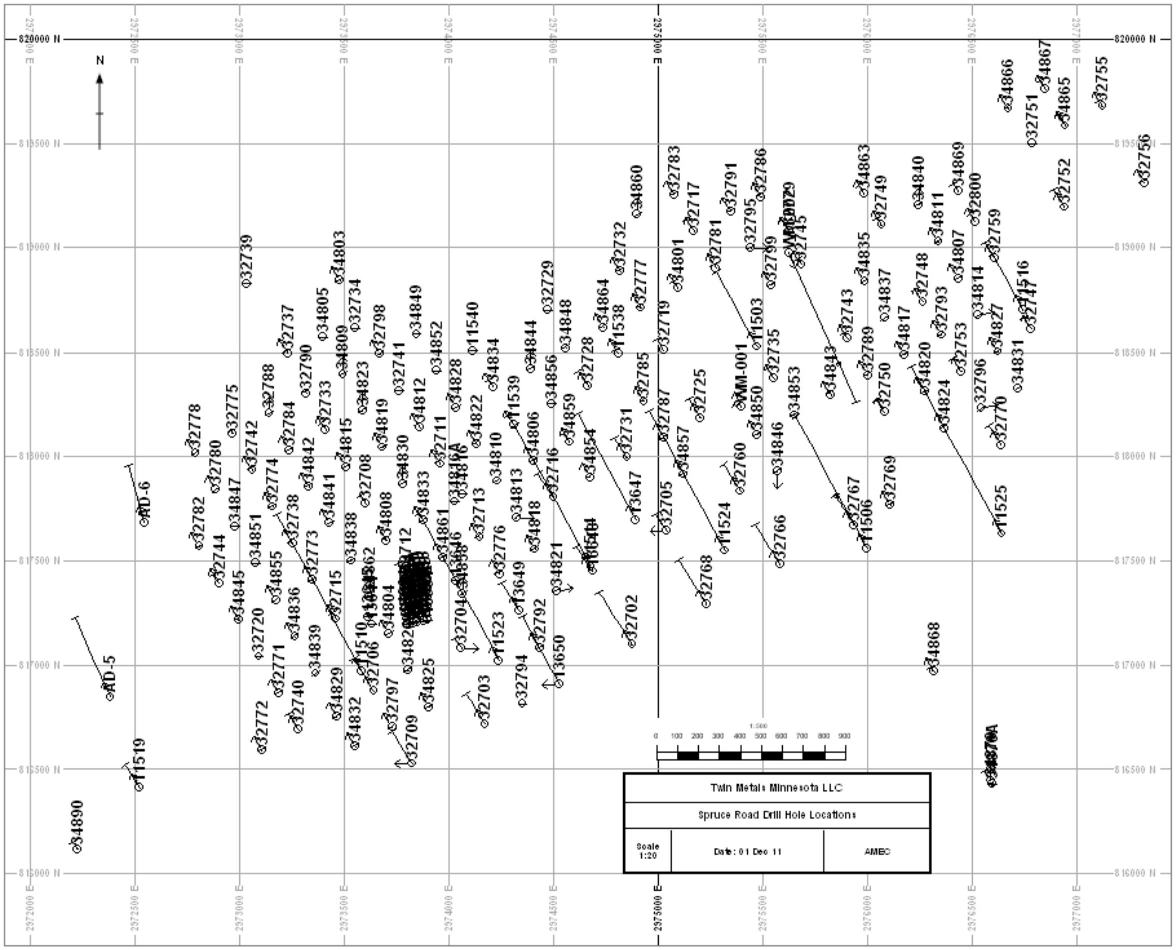
Note: Figure prepared by AMEC 2014

Table 10-6: Summary of Drilling at Spruce Road by Year

Year	Operator	Type	Number of Holes	Feet Drilled
1953*	ACNC	Core	3	685.7
1954*	ACNC	Core	14	16,261.0
1955*	ACNC	Core	4	2,204.0
1957*	ACNC	Core	9	6,334.0
1966*	ACNC	Core	24	24,099.0
1967*	ACNC	Core	135	68,292.0
1968*	ACNC	Core	13	16,004.0
1969*	ACNC	Core	2	2,780.0
1971*	ACNC	Core	2	100.0
1973*	ACNC	Core	24	669.0
1999	Wallbridge	Core	1	1,754.0
2000	Wallbridge	Core	1	2,300.0
Total			*232*	*141,482.7*

Note: * indicates legacy data

Figure 10-4: Spruce Road Drill Hole Locations



Note: Figure prepared by AMEC 2014

10.5 Core Logging

10.5.1 Maturi

Legacy logs from ACNC, Duval, and others include a description of the lithology and mineralization. No descriptions of logging procedures were discovered for these projects, but the companies were major mining companies at the time and they generally followed industry best practices for that period.

Core for the 2006–2010 Duluth programs was placed in 10 ft. capacity waxed cardboard core boxes at the drill and moved to Duluth's Ely logging and sampling facility by Duluth geologists. The core was logged for lithology, texture, structure, alteration, ore mineralogy, and mineralization.

Core for the 2011–2014 TMM programs was placed in 10 ft. capacity waxed cardboard core boxes at the drill and moved to TMM's Ely logging and sampling facility by TMM geologists. The core was logged for lithology, texture, structure, alteration, ore mineralogy, and mineralization. Geotechnical logging included recovery, rock quality designation, and fracture density. Core was digitally photographed. Samples were marked at the time the core was logged.

10.5.2 Maturi Southwest

Core for the 2012–2013 TMM program was placed in 10 ft. capacity waxed cardboard core boxes at the drill and moved to TMM's Ely logging and sampling facility by TMM geologists. The core was logged for lithology, texture, structure, alteration, ore mineralogy, and mineralization. Geotechnical logging included recovery, rock quality designation, and fracture density. Core was digitally photographed. Samples were marked at the time the core was logged.

10.5.3 Birch Lake

Geological logging recorded major and minor lithology descriptions as well as RQD and recovery percentages. Sample intervals were marked by geologists at the time of logging. Owing to the number of drilling campaigns on the property since the 1970s, core logging has been done by various geologists. Available older core has been re-logged to standardized geologic descriptions.

10.5.4 Spruce Road

Over the course of 20 years of drilling (1954–1973), several ACNC geologists logged core. Two geologists accounted for 90% of the holes, suggesting that the logging should be reasonably consistent. The logs include a description of the lithology and mineralization. The logs are consistent with best practices for the time period. The logs have been re-interpreted in order to standardize nomenclature for the current resource estimate.

10.6 Core Recovery

Core recovery is about 100% at Maturi and Maturi Southwest and 99.8% at Birch Lake. Recovery data were not recorded at Spruce Road, but rock conditions were similar, and it is likely that core recovery was about 100% (H. Parker, personal recollection).

10.7 Core Sampling

10.7.1 Maturi

TMM samples were routinely taken (74%) at 5 ft. lengths but some are as long as 10 ft. Samples are marked by geologists logging the core and split with a diamond saw. One-half was bagged for transport to the sample preparation laboratory; the other half was returned to the box and archived.

Legacy sample lengths range from 0.3 ft to 25 ft. Core was split with a manual core splitter and one-half was sent to the sample preparation laboratory and one-half was returned to the box and archived.

10.7.2 Maturi Southwest

ACNC sampled on nominal 10 ft. intervals that honored lithological boundaries. Approximately 70% of the samples are 10 ft. long. Core was split with a manual splitter, and approximately one-half was collected for analysis.

Bear Creek Mining Co. sampled on 5 ft and 10 ft intervals generally honoring lithological boundaries. The core was split with a manual core splitter, and approximately one-half was collected for analysis.

TMM sampled core on nominal five ft intervals but honored lithological boundaries. Approximately 95% of the samples are 5 ft long. Core was marked by geologists with intervals and a cut line. The core was then sawed using diamond saws. One-half of the core was placed in sample bags, the other half was archived.

10.7.3 Birch Lake

Modern samples were routinely 2–3 ft. lengths but are as short as 0.5 ft or as long as 22 ft. Samples are marked by geologists logging the core and split with a diamond saw. One-half was bagged for transport to the sample preparation laboratory; the other half was returned to the box and archived. Legacy sample lengths range from 0.5 ft to 10 ft.

A small number of intervals in the hanging wall in holes C88-1 and D3 have intervals as long as 101 ft. Core was split with a manual core splitter and one-half was sent to the sample preparation laboratory, and one-half was returned to the box and archived.

10.7.4 Spruce Road

All of the data at Spruce Road with the exception of the Wallbridge holes WM-001 and WM-002 are considered to be legacy data. Sample lengths of 5, 10, and 15 ft are the most common sample lengths. The minimum sample length is 0.20 ft and the maximum sample length is 180 ft. Samples longer than 20 ft (105 samples) were excluded from resource estimation. Core was split with a manual core splitter and one-half was sent to the sample preparation laboratory; one-half was returned to the box and archived.

10.8 Comment on Drilling

10.8.1 Maturi and Maturi Southwest

Legacy collar surveying is not documented, but is believed to have been done with theodolites and chains which were the standard tools at the time. Re-surveys of legacy collars have discovered some minor discrepancies that were corrected in the project database. Legacy collar surveys are believed to be sufficiently accurate to be used for resource estimation at an Inferred Mineral Resource level of confidence but may not be adequate for preliminary mine planning.

Current collar surveying at Maturi and Maturi Southwest utilizes industry-standard instrumentation (DGPS) and procedures and is adequate to support resource estimation and preliminary mine planning at all confidence levels.

Legacy downhole surveying was done primarily with acid-tubes which do not provide adequate control on the azimuth of drill holes, but in this case, drill hole azimuths are expected to vary very little because of the homogeneity of the rock. Those surveys are thus believed to be sufficient for estimation of Inferred Mineral Resource level of confidence. Current practice is to use gyroscopic tools that are unaffected by magnetic minerals in the rocks. These tools are widely used in the industry and provide orientation data that are adequate to support resource estimation and preliminary mine planning at all confidence levels.

Drill results indicate the presence of mineralization with economically interesting grades over significant widths.

Core logging is adequate to support resource estimation and preliminary mine planning. Core sampling conforms to industry-standard practices and is adequate to support resource estimation and preliminary mine planning.

10.8.2 Birch Lake

Current collar surveying at Birch Lake utilizes industry-standard instrumentation and procedures and is adequate to support resource estimation and preliminary mine planning.

Legacy collar surveying is not documented, but is believed to have been done with theodolites and chains which were the standard tools at the time. Re-surveys of legacy collars uncovered some minor discrepancies that were corrected in the project database. Legacy collar surveys for the few legacy holes at Birch Lake are considered to be sufficiently accurate to be used for resource estimation but may not be adequate for preliminary mine planning.

Legacy down-hole surveys were largely acid-tube surveys that provide only inclination information. The locations of the ends of legacy holes are thus somewhat uncertain. Down-hole deflections in legacy holes deserve additional study. Legacy holes were surveyed with acid tubes. AMEC concludes that the legacy downhole surveys are sufficiently accurate to support Inferred Mineral Resources.

Modern holes have been downhole surveyed with gyroscopic and magnetic instruments that are widely used within the industry. TMM resurveyed the accessible holes at Birch Lake to eliminate problems with downhole surveys noted previously. The last wedge of all accessible holes was resurveyed and those surveys were used to properly orient all other wedges. AMEC concludes that the modern downhole survey data at Birch Lake are sufficiently accurate to support resource estimation at all resource classifications as well as preliminary mine planning.

Drill results indicate the presence of mineralization with economically interesting grades over significant widths.

Core logging is adequate to support resource estimation and preliminary mine planning. Core sampling conforms to industry-standard practices and is adequate to support resource estimation and preliminary mine planning.

10.8.3 Spruce Road

Collar surveying is believed to have been performed with theodolites and chains, which was industry-standard practice at the time the holes were drilled, but that has not been confirmed.

Legacy downhole surveying was done with acid tubes. That method does not generally provide adequate control on the trajectory of drill holes, but in this case, drill hole azimuths vary very little because of the homogeneity of the rock and most holes are less than 1,000 ft deep. Those surveys are thus believed to be adequate for estimation of Inferred Mineral Resources but may not be adequate to support preliminary mine planning. Modern drilling should confirm the trajectories of the legacy holes.

Drill results indicate the presence of mineralization with economically interesting grades over significant widths.

Core logging is adequate to support resource estimation. It is adequate to support resource estimation at an Inferred Mineral Resource level, but the knowledge of the rock hosting the mineralization has progressed significantly since the original logging, and many of the codes and descriptions, and interpretations, may no longer be appropriate. A program of twin hole drilling is needed to validate the location and tenor of the mineralization and to allow more direct comparison of current lithological nomenclature with the legacy nomenclature.

Core sampling conforms to industry-standard practices at the time the core was drilled and is considered to be adequate to support resource estimation.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Introduction

Recent sample preparation and assaying was performed at accredited commercial laboratories (Table 11-1). Legacy samples were prepared and analyzed at a number of commercial and at least one company laboratory.

ALS Chemex in Vancouver (now ALS Global) performed most of the recent analyses at Maturi, Maturi Southwest, and Birch Lake and is ISO/IEC 17025:2005 accredited by the Standards Council of Canada (SCC) for the following methods:

- Fire assay Au by atomic absorption spectroscopy (AAS)
- Fire assay Au and Ag by gravimetric finish
- Fire assay Au, Pt, and Pd by inductively coupled plasma atomic emission spectroscopy (ICP-AES)
- Aqua regia Ag, Cu, Pb, Zn and Mo by AAS
- Four acid Ag, Cu, Pb, Zn, Ni and Co by AAS
- Aqua regia multi-element by ICP-AES and inductively-coupled plasma mass spectrometry (ICP-MS)
- Four acid multi-element by ICP-AES and ICP-MS
- Peroxide fusion multi-element by ICP-AES.

11.2 Sample Preparation

11.2.1 Maturi and Maturi Southwest

The Maturi and Maturi Southwest areas contain significant legacy data from ACNC. AMEC believes that ACNC followed standard industry practices (at that time) for sample preparation, but those procedures are not documented. At the time that this exploration was being performed, ACNC's parent company, Inco, was an industry-leading Cu–Ni miner in Canada and had significant internal expertise in Cu and Ni sample preparation and assaying.

For the Duluth and subsequent TMM drilling campaigns of 2006-2014, core samples were crushed, split, and pulverized in ALS Chemex's sample preparation facility in Thunder Bay, Ontario. The standard procedure employed by ALS Chemex is to weigh and dry the sample followed by crushing of the entire sample to better than 70% passing 2 mm (procedure CRU-31). The sample was then split to 250 g (procedure SPL-21) and pulverized to better than 85% passing 75 µm (-200 mesh) in a ring-and-puck pulverizer (procedure PUL-31). Barren material is used to clean the mill between sample batches to prevent cross contamination.

Table 11-1: Laboratories and Accreditations

Laboratory	Location	Function	Accreditation	Independence
Inco Process Technology Laboratory	Sudbury, ON	Sample preparation, assaying	Unknown	Not independent
Lakefield	Lakefield, ON	Sample preparation, assaying	ISO/IEC Guide 25 accreditation	Independent
Bondar Clegg (now ALS Global)	Vancouver, BC	Sample preparation, assaying	ISO 9000/9002	Independent
Bondar Clegg (now ALS Global)	Ottawa, ON	Assaying	Unknown	Independent
Bondar Clegg (now ALS Global	Sparks, NV	Sample preparation, assaying	ISO 9000/9002	Independent
ALS Chemex (now ALS Global)	Thunder Bay, ON	Sample Preparation	Unknown	Independent
ALS Chemex (now ALS Global	Vancouver, BC	Assaying	ISO 900/9002; ISO/IEC 17025:2005	Independent
Genalysis	Maddington, WA	Assaying	Unknown	Independent
ACME Analytical Laboratories Ltd.	Vancouver, BC	Assaying	ISO 9001:2000	Independent

11.2.2 Birch Lake

Prior to 2006, sample preparation was done at ALS Chemex and Bondar Clegg in Vancouver and consisted of crushing and splitting, with a split of the crushed material ring pulverized to -150 mesh (-106 µm).

For the 2006–2008 drilling campaigns, core samples were prepared by ALS Chemex in Thunder Bay, Ontario. Samples were crushed to 70% passing 6 mm and ring-and-puck pulverized to 75% passing 75 µm (-200 mesh).

For the 2010–2012 drilling campaign, core samples were prepared by ALS Chemex in Thunder Bay, Ontario. The samples were crushed to 70% passing 2 mm (procedure CRU-31), riffle split to 250 g (procedure SPL-21), and pulverized to 85% passing 75 µm (-200 mesh) in a ring-and-puck pulverizer (procedure PUL-31).

11.2.3 Spruce Road

AMEC assumes that ACNC followed standard industry practices for sample preparation, but the sample preparation procedures are not documented. At the time that this exploration was being performed, ACNC's parent company, Inco, was an industry-leading Cu–Ni miner in Canada and had significant internal expertise in Cu and Ni sample preparation and assaying.

11.3 Assaying

11.3.1 Maturi and Maturi Southwest

Assay data from Maturi and Maturi Southwest are a mix of modern data with legacy data generated by ACNC and a few other companies. Detailed information about analytical methods employed by ACNC is not available, but prior to 1965, Inco's Process Technology Laboratory in Sudbury typically determined copper and nickel colorimetrically (Routledge and Cox, 2007). By 1966 copper and nickel AAS was adopted. The cost of analysis for precious metals was significant prior to 1970 and assaying for those analytes was generally done on core composites (Routledge and Cox, 2007). Precious metals analysis was done by lead-collector fire assay followed by wet chemical extraction of the individual precious metals. AMEC is not aware of any certifications for the Inco laboratory.

For the Duluth and TMM samples, analysis of copper, nickel, cobalt, silver, sulfur and 30 other elements was done by the ALS Chemex ME-ICP61 procedure which calls for a four acid digestion of a 0.5 g pulp with an ICP-AES finish. Table 11-2 summarizes the lower and upper detection limits for the elements analyzed by procedure ME-ICP61.

Precious metal analysis calls for a one assay ton (±30 g) aliquot of pulp that was fire assayed using a lead collector with an ICP-AES finish (procedure PGM-ICP23). Table 11-3 summarizes the lower and upper detection limits for the elements analyzed by procedure PGM-ICP23.

When Cu and Ni exceeded 10,000 ppm (1%) in a sample, the sample was re-assayed using a four acid digestion finished by AAS (procedure AA62). Detection limits are given in Table 11-4.

11.3.2 Birch Lake

Sample assaying has been performed at the Ottawa, Ontario; Vancouver, British Columbia; and Sparks, Nevada laboratories of Bondar Clegg & Company (now ALS Global), Acme Analytical Laboratories Ltd. of Vancouver and ALS Chemex Labs Inc. (now ALS Global) in Vancouver. These companies are ISO 9000/9002 accredited mineral laboratories.

Multi-element analyses were performed for core samples in 1999 and onward. Prior to this, only selected elements, generally Cu and Ni, were analyzed.

A 32 element ICP-AES geochemical package was used at ALS Chemex for 1998 and 1999 samples. Pt, Pd, and Au were analyzed by lead collector fire assay finished by ICP fluorescence spectroscopy (FAICP-AFS).

Table 11-2: Lower and Upper Detection Limits for 2006-2011 Analyses at Maturi (ALS procedure ME-ICP61, values in ppm unless otherwise indicated)

Analyte	Range	Analyte	Range	Analyte	Range	Analyte	Range
Ag	0.5–100	Cr	1–10,000	Na	0.01%–10%	Ti	0.01%–10%
Al	0.01%–50%	Cu	1–10,000	Ni	1–10,000	Tl	10–10,000
As	5–10,000	Fe	0.01%–50%	P	10–10,000	U	10–10,000
Ba	10–10,000	Ga	10–10,000	Pb	2–10,000	V	1–10,000
Be	0.5–1,000	K	0.01%–10%	S	0.01%–10%	W	10–10,000
Bi	2–10,000	La	10–10,000	Sb	5–10,000	Zn	2–10,000
Ca	0.01%–50%	Mg	0.01%–50%	Sc	1–10,000		
Cd	0.5–1,000	Mn	5–100,000	Sr	1–10,000		
Co	1–10,000	Mo	1–10,000	Th	20–10,000		

Table 11-3: Lower and Upper Detection Limits for 2006-2011 Analyses at Maturi (ALS procedure PGM-ICP23)

Analyte	Range (ppm)
Pt	0.005–10
Pd	0.001–10
Au	0.001–10

Table 11-4: Lower and Upper Detection Limits for 2006-2011 Analyses at Maturi (ALS procedure AA62)

Analyte	Range (%)
Cu	0.001–40
Ni	0.001–30

Bondar Clegg analyzed 35 elements by HCL:HNO_3 (aqua regia) acid digestion and ICP including sulfur for 2000–2001 samples. Elevated copper was also analyzed by AAS. For some 1989 holes, only Au, Pt, Pd, Cu, Ni, and Cr were analyzed; in this case the precious metals were analyzed by the FA-DCP (direct coupled plasma emission method), Cu and Ni by hot acid extraction–AAS, and Cr by XRF. Table 11-5 summarizes lower detection limits for pre-2006 analyses at Birch Lake.

For the 2006–2012 drilling campaigns, core samples were analyzed by ALS Chemex in Vancouver, British Columbia. The ICP41 procedure was used for Cu, Ni, and Co and the ICP24 procedure was used for Pt, Pd, and Au. Table 11-6 summarizes detection limits for procedure ICP41, and Table 11-7 summarizes detection limits for procedure ICP24.

Samples exceeding the 10,000 ppm limit for Cu or Ni were analyzed using the OG46 procedure. Upper and lower detection limits are summarized in Table 11-8.

Table 11-5: Lower Detection Limits for Pre-2006 Analyses at Birch Lake

Element	Detection Limit
Cu	1 ppm
Cu	0.01%
Ni	1 ppm
Pt	5 and 15 ppb
Pd	1 and 2 ppb
Au	1 and 2 ppb
Ag	0.2 and 0.5 ppm
Co	1 ppm

Table 11-6: Lower and Upper Detection Limits for ALS Procedure ICP41 (ppm unless otherwise indicated)

Analyte	Range	Analyte	Range	Analyte	Range	Analyte	Range
Ag	0.2–100	Co	1–10,000	Mn	5–50,000	Sr	1–10,000
Al	0.01%–25%	Cr	1–10,000	Mo	1–10,000	Th	20–10,000
As	2–10,000	Cu	1–10,000	Na	0.01%–10%	Ti	0.01%–10%
B	10–10,000	Fe	0.01%–50%	Ni	1–10,000	Tl	10–10,000
Ba	10–10,000	Ga	10–10,000	P	10–10,000	U	10–10,000
Be	0.5–1,000	Hg	1–10,000	Pb	2–10,000	V	1–10,000
Bi	2–10,000	K	0.01%–10%	S	0.01%–10%	W	10–10,000
Ca	0.01%–25%	La	10–10,000	Sb	2–10,000	Zn	2–10,000
Cd	0.5–1,000	Mg	0.01%–25%	Sc	1–10,000		

Table 11-7: Lower and Upper Detection Limits for ALS Procedure ICP24

Analyte	Range (ppm)
Pt	0.005–10
Pd	0.001–10
Au	0.001–10

Table 11-8: Lower and Upper Detection Limits for ALS Procedure OG46 (units are % except Ag which is ppm)

Analyte	Range	Analyte	Range	Analyte	Range	Analyte	Range
Ag	1–1,500	Co	0.001–20	Mn	0.01–50	Pb	0.001–20
As	0.01–60	Cu	0.001–40	Mo	0.001–10	S	0.01–10
Cd	0.0005–10	Fe	0.01–100	Ni	0.001–10	Zn	0.001–30

For the 2010 drilling campaign, core samples were prepared by ALS Chemex in Thunder Bay, Ontario, with assays at ALS Chemex in Vancouver, British Columbia. ALS Chemex protocols for 2010 included ALS procedure ICP41 for Cu, Ni, and 33 other elements (Table 11-6) and procedure ICP24 for Pd, Pt, and Au (refer to Table 11-7). Overlimit base metals were reanalyzed by procedure OG46 (refer to Table 11-8). Overlimit Pt, Pd, and Au were reanalyzed by procedure ICP27 (Table 11-9).

Table 11-9: Precious Metals Analytical Ranges

Analyte	Range (ppm)
Pt	0.03-100
Pd	0.03-100
Au	0.03-100

Acme Analytical Laboratories (Vancouver) Ltd. protocols for the outside laboratory checks on pulps were:

- Fire assay fusion with lead collector for Au, Pt, and Pd finished by ICP-ES (procedures G606 (30 g sample) or G610 (50 g charge))
- Aqua regia digestion ICP-ES analysis for Cu, Ni, Co (procedure 7AR).

11.3.3 Spruce Road

With the exception of two Wallbridge holes, data from Spruce Road are all legacy data generated by ACNC. Detailed information about analytical methods is not available. Prior to 1965, Inco's Process Technology Laboratory in Sudbury determined copper and nickel colorimetrically (Routledge and Cox, 2007). By 1966, copper and nickel determination by atomic absorption spectrometry was adopted. The cost of analysis for precious metals and sulfur was significant prior to 1970, and assaying for those analytes was generally done on core composites (Routledge and Cox, 2007).

Samples from the 11500 series holes and most of the 13600 and 32700 series holes were analyzed for copper and nickel only. Gold, silver, platinum and palladium in samples from other holes were determined by fire assay or spectrographically.

Analyses of Cu, Ni, Co, Pt, Pd, and Au for core samples from Wallbridge's drill hole WM-001 in 1999 and WM-002 in 2001 were performed by Lakefield as part of Wallbridge's metallurgical testing. Those procedures are not documented, but Lakefield generally employs pyrosulfate fusion for digestion and XRF for Ni, Cu, Co and Fe analysis. Au, Pt and Pd are analyzed by fire assay (lead collector) with ICP finish. Lakefield is an independent, commercial laboratory and has ISO/IEC Guide 25 accreditation (Routledge and Cox, 2007).

11.4 Density Measurements

11.4.1 Maturi

A total of 24,644 density determinations were made by TMM for samples from the Maturi deposit. Early in the program, volume was determined by immersing the sample in a graduated cylinder and determining the sample volume by the volume of water displaced by the sample (19,829 samples). Mass was determined by balance in air. Later in the program, density was determined by weighing the dry core in air and again submerged in water (4,815 samples). The dense, nonporous core was not sealed prior to immersion. These are standard techniques for determination of density. Table 11-10 summarizes density by rock type where rock types were indicated (178 samples had no assigned rock type).

11.4.2 Maturi Southwest

The Maturi Southwest database contains 1,439 density data within the resource area (29 samples were not assigned a rock type). TMM determined density on each 10 ft run of core. Density was determined by weighing the dry core in air and again submerged in water. The core was not sealed prior to immersion. Table 11-11 summarizes the density data by lithology at Maturi Southwest. This procedure is adequate for these very nonporous rocks.

11.4.3 Birch Lake

The Birch Lake density database consists of 1,603 water immersion density determinations on drill core. The procedure involves weighing the dry sample and immersing it in water in a graduated flask and determining the sample volume by the volume of water displaced by the sample. The method is a standard method and was performed by Franconia and TMM personnel. Table 11-12 summarizes the data by lithology.

11.4.4 Spruce Road

No density data are available for the Spruce Road deposit. A bulk density of 3.02 g/cm^3 was used for conversion of volume to resource tonnage for the resource estimate. This bulk density is the same as the estimated average density for the Maturi mineralized material (dominantly S3 at 3.02 g/cm^3) deposit where density was determined by TMM. Waste bulk density at Spruce Road was assumed to be 3.00 g/cm^3.

11.5 Sample Security

11.5.1 Maturi and Maturi Southwest

Sample security for the legacy samples generated by ACNC is not documented. Most ACNC core was lost when a storage facility in Canada was burned during a labor action against Inco. Remaining core is stored at the DNR core library in Hibbing along with other legacy core.

Sample security for current samples consists of collecting core at the drill twice a day and storing it in a lockable core logging facility prior to sampling. Sampled core is stored in sturdy plastic bags in a locked room until it is shipped to the sample preparation laboratory by contract carrier.

Table 11-10: Summary Density Data by Lithology at Maturi

Strat	Number	Mean (g/cm^3)
HW	2,968	2.89
PEG	974	2.96
UH	815	3.02
S3	7,396	3.02
S2	5,471	3.05
S1	1,478	3.02
G_N	526	2.83
G_M	2,836	2.79
G_B	2,002	2.74
None	178	2.86
Total	24,644	2.96

Table 11-11: Summary Density Data by Lithology at Maturi Southwest

Unit	Number	Mean (g/cm^3)
HW	81	2.91
PEG	40	2.92
UH	117	2.99
S3	406	2.99
S2	371	3.02
S1	230	3.01
G_M	36	2.75
G_B	129	2.70
Total	*1,410*	*2.96*

Table 11-12: Summary Density Data by Lithology at Birch Lake

Unit	N	Density (g/cm^3)
MAIN_AGT	114	2.93
BL_MT	484	3.00
BL_T	327	3.06
BL_HX	164	3.00
BL_DI	16	3.09
BL_D2	71	3.04
GRB_M	188	2.76
GRB_B	239	2.79
Total	1,603	2.95

11.5.2 Birch Lake

Some legacy samples were shipped to analytical laboratories in Nevada by courier services including UPS and FedEx. Pre-2006 samples sent to Bondar Clegg and ALS Chemex in Canada were driven to the U.S. border for direct pick-up by laboratory personnel. The 2006–2012 core samples were placed on pallets in Franconia's secure facility at Babbitt and trucked via commercial freight to ALS Chemex sample preparation facility in Thunder Bay, Ontario.

11.5.3 Spruce Road

Sample security for the legacy samples generated by ACNC is not documented. Most ACNC core was lost when a storage facility in Canada was burned during a labor action against Inco. Remaining core is stored at the DNR core library in Hibbing along with other legacy core.

11.6 Quality Assurance and Quality Control (QA/QC)

This section summarizes the QA/QC measures employed for these projects. Additional details can be found in Parker and Eggleston (2014).

11.6.1 2011–2012 Maturi Quality Check Programs

11.6.1.1 2011 Test Work

TMM completed five assay evaluation programs in 2011, designed to test for potential issues with sample preparation and assaying. These included:

- Determining whether dust losses during sample preparation could be biasing metal grades. Dust losses are sufficiently low that a bias exceeding 5% is unlikely
- Testing whether more aggressive grinding during sample preparation could lead to higher grades of platinum group elements (PGEs). The results showed no improvement by more aggressive grinding. In AMEC's opinion, the current grind protocol is sufficient
- Determining whether cuttings lost during core sawing are biasing metal grades. Grade biases between the core and the cuttings were shown to be less than 5% and are not considered significant; therefore, no significant bias occurs due to the cutting of the core. AMEC recommends no changes be made to the core cutting method in present use
- Determining the proportion of unrecoverable nickel sequestered in silicate minerals using ammonium citrate peroxide leach assays (ACPL) which does not recover Ni from silicate minerals. Results of the ACPL assays show that recoverable nickel ranges between 68 and 81% and varies by rock type (Table 11-13). AMEC found that the amount of nickel sequestered in silicates to be sufficiently high and variable between BMZ subunits that it should be taken into consideration when evaluating metallurgical test results and possibly in establishing domains. Ni in silicates appears to be proportional to the total olivine content in the rocks

Table 11-13: Summary of Ammonium Citrate Peroxide Leach Assays by Domain and BMZ Unit

Domain Code	Unit Code	Number	Mean Total Ni (ppm)	Mean ACPL Ni (ppm)	Recoverable Ni (%)
PEG	PEG	24	1,589.6	1,126.7	71
BMZ		72	1,686.1	1,237.8	73
	U3	23	1,741.4	1,180.4	68
	BH	24	1,824.2	1,318.3	72
	BAN	25	1,502.8	1,213.2	81

- Testing whether higher grades for PGEs can be expected using nickel sulfide fusion assays. These results indicate it is possible that more platinum and palladium may be recovered from the BH unit than indicated in the resource estimate, and more platinum may be recovered from the PEG unit than indicated in the resource estimate; however, AMEC concluded the results of this limited test were inconclusive and that additional NiS assays should be conducted to better define possible biases.

11.6.1.2 2012 Ni in Silicate Study

Eggleston (2012) demonstrated a strong correlation between Mg and Ni in the Maturi and Birch Lake Project drill core samples that were low in Cu and S. This suggested the possibility that the amount of Ni sequestered in silicate minerals such as olivine may be quantitatively estimated from the Mg concentration in rocks containing nickel sulfides (Long, 2012c). As part of the check assay program on Maturi drill core samples conducted by AMEC (2012a; 2012b), Acme Laboratories (Vancouver) assayed 105 samples for Ni by the ammonium citrate-peroxide leach method.

Franconia metallurgical test work on Birch Lake drill core samples included ACPL assays and metallurgical recovery results on 30 samples. These two data sets allow an initial assessment of the use of Mg for estimating sulfide Ni in rocks with economically important concentrations of Ni.

Long (2012c) concluded that Mg and total nickel can be used to predict sulfide nickel, but more robust ties between the ACPL results and metallurgical recovery data are required. This will necessitate additional ACPL results on metallurgical samples where actual flotation recoveries are determined. Accuracy of Mg results must become a focus of all analytical work. Mg has not been a focus of quality control previously and Mg coverage is not complete for all samples. At Birch Lake, demonstrated biases (Long, 2012b) limit the predictive ability of this method. Areas where Mg is absent or significantly biased and where economically important levels of Ni occur should be identified so that consideration can be given to remedial assaying for Mg.

11.6.1.3 2012 Re-assay Program

Due to a significant negative bias in Ni assays from the pre-Mex-112 drill holes at Maturi discovered during the QA/QC review in 2011, 8,748 samples were selected for re-assay to provide more confidence in the data and to remove the bias. Those samples were reassayed at ALS Laboratories using procedure ME-ICP61. When compared to the original assays, the overall improvement in Ni grade was 4.1% relative to the average of the original data. TMM replaced the prior data with the new data in the database and the new data were used for resource estimation.

11.6.2 Assay QA/QC

11.6.2.1 Maturi

11.6.2.2 Legacy Drilling

Legacy drilling campaigns at Maturi likely did not employ a modern QA/QC protocol and no QA/QC data are known to exist for the legacy drilling campaigns.

11.6.2.3 Duluth/TMM Drilling

Duluth and TMM have consistently applied an assay QA/QC protocol consisting of the following control samples inserted into batches of mineralized samples:

- Five CRMs randomly placed in the sample stream
- Two blanks placed typically after strong visual mineralization
- Two ¼ core duplicates selected from average visual mineralization.

A drill hole submittal typically consists of the BMZ plus 30 ft (9.1 m) of hanging wall and 30 ft (9.1 m) of footwall, ranges in total length between 150 and 650 ft (45.7 and 198.1 m), and includes 30 to 130 samples. Control sample insertion rates therefore range between 20% (6/30) to 5% (6/130).

A number of CRMs have been used through the Project history, including CANMET standard WMG-1 (2006 through March 2008), CANMET standard WPR-1 (March 2008 through May 2011), and more recently AMIS0073, AMIS0093, AMIS0170, AMIS0319, AMIS0320, CFRM-101, GBM311-3, GBM910-4, and OREAS13b.

The Project's initial monitoring of assay accuracy used a single CANMET standard: WMG-1. This CRM has certified values for 15 elements including gold, platinum, and palladium. The values used for copper and nickel are "provisional", not certified values. Results for an element are classified as provisional if the laboratories participating in the round robin do not agree adequately.

AMEC documented variation in copper results over time from TMM's external CRM WMG-1 assayed by ALS Chemex (Figure 11-1). The middle horizontal red line represents the recommended value for the CRM, and the upper and lower red lines represent 10% above and below the recommended value.

Copper assays from CRM WMG-1 indicate that the original copper assays for drill holes MEX-0001 to MEX-0112 are biased high 10% relative to the best value and 8% high relative to the average SRM results for drill holes MEX-0113 to MEX-0149. An additional check assay program conducted by Antofagasta Minerals, and duplicate assays conducted by TMM, suggest that the bias is real, with the high bias ranging between 4 and 6%.

Based on the evidence, AMEC recommended that copper assays for drill holes MEX-0001 to MEX-0112 be reduced by 6%, the value consistent with the preponderance of evidence. To account for this bias TMM and AMEC agreed to reduce the copper grade for these holes by 6% (Wakefield, 2011b). This reduction was applied only to the ICP results (ALS Chemex method ME-ICP61); the copper results re-assayed with method AA62 were not reduced.

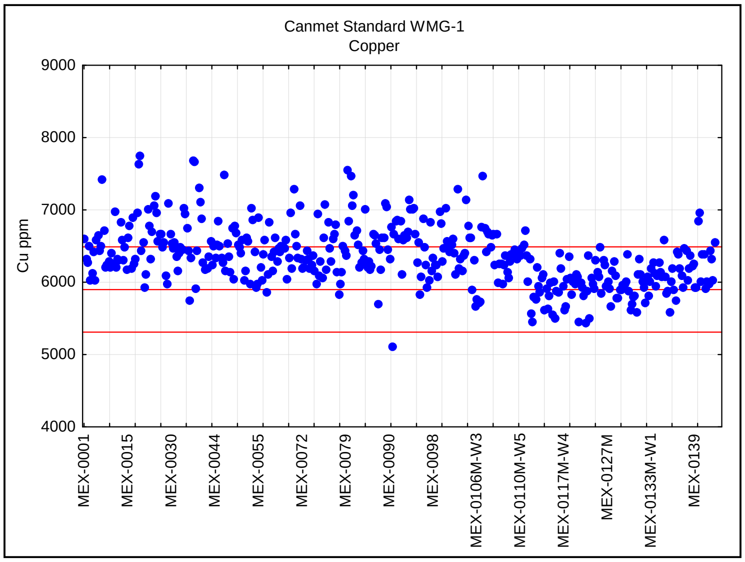
When the supply of CRM WMG-1 was exhausted, TMM acquired CRM WPR-1 material from CANMET to control assay accuracy. This CRM was used from March 2008 through early 2011. This CRM has certified values for 11 elements including gold, copper, platinum, and palladium. No significant biases are shown for any of the elements over the period that the WPR-1 was used. Results for platinum and palladium are generally tightly clustered within the acceptable limits. Results for the CRM samples indicate the accuracy of gold for these batches to be good, but the spread of the results indicates poor precision at these low concentration levels.

In early 2011 TMM began using CRM AMIS-0093 to monitor assay accuracy. This CRM is provided by African Mineral Standards (AMIS) in South Africa, and has certified values for copper, nickel, platinum, and palladium and a more uncertain 'provisional' value for gold. In mid-2011, TMM purchased additional CRMs for use in the assay test programs that are currently being used in TMM routine submittals, AMIS CRM AMIS-0073 and the Ore Research CRM OREAS 13b.

Copper and nickel results for AMIS-0093 indicate that assay accuracy for these elements in the batches tested is acceptable. Platinum and palladium results show no significant bias. Gold results are biased low, on average, and the high degree of scatter in the results indicates poor precision at these very low grade levels.

An insufficient number of results for AMIS-0073 and OREAS 13b have been received to date to provide meaningful analysis.

Figure 11-1: WMG-1 Results for Copper



Note: Figure prepared by AMEC, 2014.

Blank samples have been inserted into TMM/Duluth batches consistently throughout the project history. In AMEC's opinion, there is no significant carryover contamination in the sample preparation process at ALS Chemex for copper, nickel, platinum, palladium, and gold.

TMM has consistently employed a program of ¼ core duplicates, consisting of two intervals selected in visual mineralization. Results for copper and nickel show acceptable precision, with 90% of the duplicate pairs yielding absolute relative difference (ARD) values of less than 30%. Platinum, palladium, and gold show poor precision, yielding ARD values of 62, 41, and 79% respectively at the 90% population level.

11.6.2.4 2012 QA/QC Results

For the 2011–2012 Maturi drill program, TMM inserted crusher duplicate, pulp duplicate, blank, and standard samples. TMM and AMEC monitored QC results throughout the drill program and concluded that:

- Accuracy for Cu, Pd, Pt, and Au analyses are adequate to support resource estimation and preliminary mine planning
- Ni accuracy below 2,000 ppm is adequate to support resource estimation, but above 2,000 ppm, there appears to be a small negative bias relative to standard samples. The bias is on the order of 4–6%
- Blank samples indicate no significant contamination for Pt, Pd, and Au

- Precision estimates for all elements in coarse duplicate samples (crusher samples at -2 mm) are in the range anticipated by AMEC and are adequate to support resource estimation and preliminary mine planning
- Precision estimates for all elements except Pt and Au in pulp duplicate samples are in the range anticipated by AMEC and are adequate to support resource estimation and preliminary mine planning. Au precision (±49%) is outside the anticipated range (±25%), but the overall low Au grade makes improvements to precision very difficult without significant changes to the sample preparation protocol. Similarly, Pt precision (±28%) is somewhat outside the anticipated range but is not considered by AMEC to be a significant concern.

11.6.2.4.1 Comments on Maturi QA/QC Program

AMEC recommends that resource blocks strongly influenced by Maturi legacy drill holes be classified as Inferred Mineral Resources, and the influence of Maturi legacy drill holes was taken into account in resource classification (see Section 14.8.1).

CRM, check assay, and duplicate results indicate that the original copper assays for drill holes MEX-0001 to MEX-0112 are biased high between 4 and 10%. Copper assays for drill holes MEX-0001 to MEX-0112 were reduced by 6% to account for a high bias noted in those data

Maturi assay accuracy is acceptable for nickel, platinum, palladium, and gold.

In AMEC's opinion, there is no significant carryover contamination in the sample preparation process at ALS Chemex for copper, nickel, platinum, palladium, and gold.

Duplicate results for copper and nickel show acceptable precision. Platinum, palladium, and gold show poor precision. AMEC recommends that TMM conduct a heterogeneity study to determine whether the sample preparation scheme can practically bring the platinum, palladium, and gold assays into acceptable precision levels.

Legacy drilling campaigns at Maturi likely did not employ a modern QA/QC protocol, and no QA/QC data are known to exist for the legacy drilling campaigns. However, one legacy ACNC drill hole was twinned by Lehmann Exploration in 1989, and TMM drilled a series of twin drill holes at Maturi in 2011 to validate the legacy assays. The tenor and location of the legacy data were generally validated.

11.6.2.5 Maturi Southwest

11.6.2.5.1 Legacy Data QA/QC

Legacy data at Maturi Southwest was mostly from ACNC. QA/QC measures employed for those samples are not known to TMM or AMEC. ACNC's parent company, Inco, was a major Ni-Cu mining house with considerable experience with Ni and Cu assaying; therefore, AMEC has no real concerns about the quality of the data, but the quality of the data has not be verified.

11.6.2.5.2 2013 QA/QC Results

For the 2011–2012 Maturi drill program, TMM inserted quarter core, -2 mm crusher duplicate, blank, and standard samples. ALS Chemex inserted pulp duplicate, standard, and blank samples. TMM and AMEC monitored QC results throughout the drill program and concluded that:

- Accuracy for Cu, Ni, Pd, Pt, and Au analyses are adequate to support resource estimation
- Ni biases are less than 5% except for one standard, GBM910-4 (30 ppm; 7.8%). That standard has a best value equivalent to 30 x lower detection limit and is thus not a concern. Note that all of the biased data, except for the very low-grade standard, are biased slightly low. This suggests to AMEC that although significant time and energy have been spent reducing the Ni bias noted in the Maturi data, a small negative bias remains at ALS Chemex. That bias is on the order of 1-2% and is not a particular concern
- Blank samples indicate no significant contamination for Pd, Pd, and Au
- Precision estimates for all elements in coarse duplicate samples (crusher samples at -2 mm) are in the range anticipated by AMEC and are adequate to support resource estimation and preliminary mine planning
- Precision estimates for all elements except Pt and Au in pulp duplicate samples are in the range anticipated by AMEC and are adequate to support resource estimation and preliminary mine planning. Au precision (±47%) is outside the anticipated range (±25%), but the overall low Au grade makes improvements to precision very difficult without significant changes to the sample preparation protocol. Similarly, Pt precision (±36%) is somewhat outside the anticipated range but is not a significant concern.

11.6.2.5.3 Maturi Southwest QA/QC Comment

Accuracy and precision of the Maturi Southwest data are acceptable and adequate to support resource estimation and preliminary mine planning.

11.6.2.6 Birch Lake

The drilling campaigns at Birch Lake can be logically divided into four major phases: pre-Franconia (Duval and Lehmann), early Franconia (1989 to 2005), recent Franconia (2006 to 2010), and TMM.

11.6.2.6.1 Legacy (pre-Franconia) Drilling

Legacy campaigns at Birch Lake are not known to have included assay QA/QC programs, and no QA/QC data are available to TMM for any of these drill holes. Legacy drill holes consist of six Duval drill holes and one Lehman Exploration drill hole. Together, these account for 3.0% of the drill holes and 3.4% of the drill footage at Birch Lake.

11.6.2.6.2 Early Franconia Drilling (1989 to 2005)

Control samples (standards, blanks, and duplicates) were not inserted into the project sample batches for any of the early Franconia drilling campaigns. However, in 2001, Franconia performed a check assay program on drill holes completed from 1989 to 2000 to confirm mineralized intercepts. Original mineralized assay intervals were selected and composited into lengths ranging from 9 to 14 ft to approximate minimum mining widths. Composite samples were compiled from rejects retrieved at the Minnesota Department of Natural Resources (DNR) core facility in Hibbing, Minnesota or from pulps from the Franconia core storage facility in Babbitt, Minnesota. Insufficient checks were performed for campaigns 1990, 1995, 2001, and 2005 to make conclusions regarding the assay accuracy of the original results.

A total of 692 individual samples were combined into 102 composite samples for check assay. Samples were assayed at Bondar Clegg in Vancouver, Canada for Cu and Ni by aqua regia acid digestion and ICP finish and Pt, Pd, Rh, Ir, Os, Ru, and Au primarily by neutron activation and less frequently by fire assay and AA finish. Splits of the composites were also sent to Genalysis (Maddington, Western Australia), where Cu and Ni were determined by three-acid digestion and AA finish, and Pt, Pd, Rh, Ir, Os, Ru, and Au were assayed by NiS fire assay and ICP-MS finish. There is no evidence that external quality control samples were inserted in the check assay batches.

In addition, 33 individual samples from drill hole BL00-7 were sent to Genalysis to directly compare with the original Bondar Clegg individual assays.

Composite check assays for copper and nickel, when all campaigns are plotted together, agree reasonably well with the original assays, and no significant bias is evident. Certain campaigns show a bias for either copper or nickel, but the number of check assays for all campaigns except for 2000 is too few to draw any firm conclusions. No significant bias was noted in the 2000 data. The individual check assays show a potentially significant constant low bias in copper and a very significant low bias in nickel.

Composite check assays for platinum and palladium show significant scatter and possible significant low bias in the original platinum assays above 1,000 ppb. Certain campaigns show a bias for either platinum or palladium, but the number of check assays for all campaigns except for 2000 is too few to draw any conclusions. The individual assays support the conclusions from the composite assays, where platinum assays are biased about 16% low, and palladium assays are biased low, but not significantly so.

Composite check assays for gold show some evidence of high bias in the original results for values below 400 ppb, and some evidence of low bias above 400 ppb. The individual gold check assays show that gold is biased high, but marginally so.

11.6.2.6.3 Recent Franconia Drilling (2006 to 2010)

Beginning in 2006, Franconia instructed their primary laboratory, ALS Chemex, to generate a second pulp of every 10th sample and to periodically send these samples to ACME Laboratories in Vancouver, Canada for check assays.

Copper check assays agree with the 2006 to 2010 Birch Lake original assays from ALS Chemex, with between-laboratory biases acceptably small, between 1 and 6%. Nickel check assays are consistently high, between 4 and 13%, relative to the original ALS nickel assays. The lack of inserted reference materials in the check assay submissions makes it impossible to determine the sources of the observed relative biases.

Platinum and palladium check assays agree with the original assays for years 2006, 2007, and 2010. Platinum and palladium check assays for the 2008 drilling campaign are significantly higher than the original assays, between 15 and 18%, on average. Gold check assays agree with the original assays for years 2007, 2008, and 2010, but are significantly higher for year 2006, 30% on average. It should be noted that there is a high degree of scatter, or imprecision, in some of the platinum, palladium, and gold check assays.

In 2007, Franconia initiated the insertion of blank samples consisting of cement core-shaped intervals in every batch of 20 samples submitted to ALS Chemex. Several very high concentrations are observed in the blank assays for all elements, likely indicating sample switches, where the blank and an adjacent sample were switched. Ignoring the likely sample switches, there appears to be carryover of about 50 to 200 ppm copper, 30 to 60 ppm nickel, and 0.005 to 0.010 ppm palladium in the sample preparation process. Platinum and gold consistently report at or below five times the lower detection limit for the assay. These levels of copper, nickel, and palladium are relatively small, and carryover contamination at these levels is not likely to significantly bias grades in the resource estimate.

11.6.2.6.4 2012 Check Assay Program

In 2012, TMM performed two check assay programs to investigate the accuracy of the analyses performed by Franconia from 1998 through 2010. The first program covered

1998–2001 and 2005 (Long 2012a). The second program covered 2006–2008 and 2010 (Long 2012b).

Samples for the reassay program were submitted to Acme (Vancouver) in late April 2012 (Long, 2012a; 2012b). These submissions include sample pulps and coarse rejects from drill holes and included approximately 1,912 samples. Coarse reject samples were submitted instead of pulps if sample pulps could not be located in storage. The submission included blind CRMs.

All samples were fire assayed (30 g, Acme procedure 3B) for Au, Pt and Pd and underwent a four-acid digestion with multi-element ICP determinations of base metals and many major elements (Acme procedure 7TD).

TMM had Acme re-assay selected samples using an aqua regia digestion (Acme procedure 7AR) for comparison purposes and a sub-set of these underwent ammonium citrate peroxide leach for nickel (Acme procedure 8NiS) that selectively dissolves nickel in sulfide minerals, leaving silicate minerals largely intact.

A few samples in the coarse reject submissions were screen tested to determine the quality of the original sample preparation by ALS Thunder Bay, prior to being pulverized by Acme. Similarly, a few pulp samples were checked for grind quality prior to Acme re-blending samples in a pulverizer in order to check the original grind quality produced by ALS Thunder Bay. The original preparation met specification with a few minor exceptions.

The most important conclusions from the work are (Long, 2012a; 2012b):

- PGE results for the 1998–2001 and 2005 Franconia work are acceptably accurate
- Copper results for BL98 and BL99 should be adjusted downward by multiplying by 0.9 for use in the resource model. Copper results from BL05 time period average 9% high. There may be a time period within this year (2005) that warrants a downward adjustment. In order to do this, the job numbers of the original assay results would need to be compiled in order to get the results correctly ordered. Because of the small number of samples involved this recommendation was not implemented
- Nickel results have a probable low bias of nine to fourteen percent for drill holes with prefixes BL98, BL99, BL00, and BL01; these can probably best be dealt with in future resource models by developing a sulfide-Nickel model based upon Mg data and ammonium citrate peroxide Ni assays (ACPL.Ni), and by using different linear equations for the different time periods which will remove the biases from the sulfide-Ni model
- Mg results show marked biases in the 1998-2001 and 2005 Franconia work; hence both the original Mg and original Ni results from the same time period should be

used to fit with new ACPL.Ni assays for creating these linear equations for each assay time period

- There are insufficient checks on the Cu–AA method for Cu results greater than 1 % Cu to judge the accuracy of these results for the 1998-2001 and 2005 Franconia work
- PGE results for the 2006–2008 and 2010 Franconia work are acceptably accurate
- ALS ICP copper results have a high bias of approximately 5%, possibly closer to 10% for 2006. A 5% downward correction to the 2006 ICP Cu results may be warranted. Additional investigation of this possible bias was recommended prior to implementing the adjustment
- The ALS ICP nickel results have a low bias, which occurs in all years checked. Total nickel is likely to be underestimated between 6 and 10%. This could be compensated for in a sulfide nickel model, because an empirical estimate of sulfide nickel, based upon empirically derived formulas using Mg, ammonium citrate peroxide leach Ni (ACPL.Ni), and total Ni results, would, because it is an empirical fit of the existing data plus new ACPL.Ni data, compensate for any biases, provided that ACPL.Ni assays are consistently accurate and precise, and the obtained correlations are sufficiently robust. However, any total nickel model will suffer from this low nickel bias. AMEC recommended that all samples be reassayed for Ni in order to remove this bias. No adjustments were made to the data for the current resource estimate
- Mg shows a marked low bias in the Franconia data relative to the Acme data. Use of the existing Mg data may be problematical because the less expensive aqua regia digestion method was apparently used in 2006, and a three-acid digestion was used in other years. This provides a low bias in the Mg data which may make it much less effective for use in an empirical formula for estimating sulfide nickel. This must be determined by studies that determine how well the existing Mg results correlate with the difference between the existing Ni results and new ACPL.Ni results.

11.6.2.6.5 2012 QA/QC Results

For the 2011–2012 Birch Lake drill program, TMM inserted crusher duplicate, pulp duplicate, blank, and standard samples. TMM and AMEC monitored QC results throughout the drill program and concluded that:

- Accuracy for Cu, Pd, Pt, and Au analyses are adequate to support resource estimation and preliminary mine planning. Au bias for some standards is outside the ±5% window that AMEC normally uses, but is with ±10% which is adequate for resource estimation

- Ni accuracy below 2,000 ppm is adequate to support resource estimation, but above 2,000 ppm, there appears to be a small negative bias relative to standard samples. The bias is on the order of 4–6%. No adjustments were made to the data for the current resource estimate
- Blank samples indicate no significant contamination for Pd, Pd, and Au
- Precision estimates for all elements in coarse duplicate samples (crusher samples at -2 mm) are in the range anticipated by AMEC and are adequate to support resource estimation and preliminary mine planning
- Precision estimates for all elements except Pt and Au in pulp duplicate samples are in the range anticipated by AMEC and are adequate to support resource estimation. Au precision (±32%) is outside the anticipated range (±25%), but the overall low Au grade makes improvements to precision very difficult without significant changes to the sample preparation protocol. Similarly, Pt precision (±37%) is somewhat outside the anticipated range but is not a significant concern.

11.6.2.6.6 Comments on Birch Lake QA/QC Program

Based on evaluation of the QA/QC data, AMEC concludes that the TMM and Franconia base and precious metals data are sufficiently accurate and precise to support resource estimation at all levels of classification. Ni accuracy below 2,000 ppm is adequate to support resource estimation and preliminary mine planning, but above 2,000 ppm, there appears to be a small negative bias relative to standard samples. The bias is on the order of 4–6%. Assay data were not adjusted for this bias which represents an opportunity to modestly improve Ni grades.

Precision estimates for all base and precious metals except Pt and Au in pulp duplicate samples are within the range anticipated by AMEC and are adequate to support resource estimation. The low grades of Pt and Au make significant improvements in precision very difficult. AMEC has accepted the precision for those elements as adequate to support resource estimation, but cautions that the precision is somewhat outside the limits normally used by AMEC. The data appear to be unbiased which means the overall estimated Pt and Au grades will likely be accurate but that Pt and Au grades will be underestimated or overestimated locally.

TMM Mg data are sufficiently accurate and precise to support resource estimation. Franconia Mg data exhibit significant biases, largely because of the three-acid digestion used for much of those data. Those data should not be used for resource estimation. Much of the Franconia Mg data is now covered by Acme check assays that can be used for silicate Ni estimation. If a geometallurgical model is to be based on Mg data, many of the Franconia samples will need to be reassayed for Mg.

Pre-Franconia base metal data are adequate to support resource estimation but AMEC notes that approximately three holes drilled in 1998-1999 may have a 5–9% high Cu bias. This bias will not significantly affect the resource estimate and should be confirmed by additional sampling. Assay data were not adjusted for this bias.

11.6.2.7 Spruce Road

No QA/QC data are available for Spruce Road assays. AMEC has restricted the classification to Inferred Mineral Resources, in part, for this reason.

11.7 Comment on Section 11

11.7.1 Sample Preparation

Legacy sample preparation by ACNC is not documented, but it is the AMEC QPs' opinion that it is reasonable to consider sample preparation procedures as adequate, largely because ACNC's parent company, Inco, was an industry leader in Cu–Ni mining at the time the samples were collected and analyzed.

Sample preparation for recent exploration programs completed by Franconia, Duluth, and TMM has been performed using standard procedures and is adequate to support resource estimation and preliminary mine planning.

11.7.2 Sample Analysis

Analytical procedures used for legacy ACNC samples is not documented, but is believed to be adequate to support resource estimation.

Analytical procedures employed by Franconia, Duluth, and TMM are industry-standard procedures and are adequate to support resource estimation and preliminary mine planning.

11.7.3 Density Analysis

Density determinations at Maturi, Maturi Southwest, and Birch Lake were performed using standard procedures and are adequate to support resource estimation and preliminary mine planning.

No density determinations have been performed at Spruce Road.

11.7.4 Sample Security

Sample security for legacy samples is not documented. Sample security for modern samples is considered to be sufficient to support resource estimation and preliminary mine planning.

11.7.5 QA/QC

QA/QC for legacy samples is not documented.

QA/QC for current samples is considered by AMEC to be adequate to support resource estimation and preliminary mine planning. Problems noted by AMEC were remedied by TMM. Minor adjustments to the TMM procedures were implemented.

12.0 DATA VERIFICATION

12.1 Database Compilation and Validation

12.1.1 Introduction

TMM maintains the database in an acQuire database after migration from an Access-based database in 2011–2012. On 15 May 2012, AMEC received a database export from TMM to verify that the migration of the TMM database from Access to acQuire was successful. AMEC noted a number of discrepancies that were subsequently corrected prior to the final database audit in June and July of 2012.

In order to validate the data for Maturi and Birch Lake, AMEC performed two audits of the databases for those two properties (Wakefield, 2011; 2012) and a single audit of the Maturi Southwest data. AMEC's audits consisted of checking the database records against the original documentation for the data that are material to the resource estimation process. This includes the drill collar location information, the down-hole surveys, the core lithological logging data, and the assays.

AMEC also performed a number of database integrity checks which included:

- Checking that all drill holes have collar, assay, survey, and lithology records
- Checking ranges of collar location coordinates
- Checking ranges of assay fields
- Checking ranges of down-hole survey readings
- Check for unusually small or large intervals that have assays
- Check for gaps in sampling/assaying.

12.1.2 Maturi Database Audit

In 2011 and 2012, AMEC selected approximately 10% of the TMM, Duluth and legacy drill holes for the purposes of the database audit (Wakefield, 2011; 2012). Audit drill holes were selected to be spatially (equally spaced throughout the Maturi deposit), and temporally (equally spaced throughout the drilling campaign period) representative.

Results of the 2011–2012 Maturi audit are:

- Collar Locations
  - Collar locations for the MEX series holes drilled by Duluth and TMM are considered adequately accurate. No errors were noted in the database. AMEC located 16 collars in the field with a hand-held GPS unit and found that the coordinates agreed well with those in the database
  - Legacy collars were surveyed using a variety of coordinate systems. Several legacy collars were located by TMM staff and resurveyed.

- Downhole Surveys
  - AMEC checked the depth, azimuth, and inclination (dip) values for a total of 8,597 downhole surveys against the original paper survey files found in the drill hole folders in the TMM offices in Ely. No errors were found
  - Downhole surveys for legacy drill holes consist of acid-tube tests that provide only inclination (dip) information. AMEC checked the depth and inclination values for a total of 97 acid-tube surveys from nine drill holes that had been surveyed down hole.
- Lithology Logs
  - Lithology logs from the TMM and Duluth drill campaign have been logged in a consistent manner. AMEC checked the “From”, “To”, and “RockType” values for a total of 783 logged intervals from 36 drill holes. A total of 9 errors were found out of the 2,118 values checked for an error rate of 0.4% in 2011. In 2012, Maturi and Birch Lake were audited as a unit. A total of 1,122 records were audited and an error rate of 0.1% was discovered (one error)
  - Legacy lithology data in the database are a product of the original drill logs or re-logs conducted by the NRRI. AMEC did not audit original lithology logs, and instead relied upon the Unit code picks by TMM staff
  - AMEC compared lithological logs from Maturi to core from ten holes and found no significant discrepancies
- Assays
  - MEX drill core has been consistently submitted to ALS Chemex for assay, and assay methodology has also remained consistent through the years. ALS Chemex provided AMEC with digital copies of original assay certificates for the audit drill holes through secure login to their website. AMEC checked the From, To, Cu, Ni, Pt, Pd, and Au and found an error rate of 0.01%, which is acceptable
  - Assay data for the Maturi legacy drill holes typically consist of hand-written or typed Cu and Ni values entered into the margins of the lithology log for the drill hole. In most cases, the assay method, laboratory, and even units are not known for certain. AMEC checked From, To, Cu, and Ni values for 744 assay intervals from 13 legacy audit drill holes. AMEC also checked From, To, Au, Pt, Pd, Cu, Ni, and S values from 88 assay intervals from four legacy drill holes from the NRRI re-sampling/re-assaying program. AMEC found a total of five errors in 2011 for an acceptable error rate of 0.6%.

In 2014, AMEC audited data added to the database since the 2012 audit and found:

- Collar Surveys

  - AMEC located the collars of 12 holes drilled in the 2013-2014 drill program and two holes drilled earlier and determined the coordinates with a Garmin GPSmap 62sc hand-held instrument. AMEC compared the locations, in NAD 83, Zone 15 coordinates, to the data provided by Northern Lights and compared the AMEC coordinates converted to MN State Plane coordinates to the database. In both cases, the differences noted are within acceptable limits.
  - AMEC audited 100% of the collars added since the 2012 audit and found no discrepancies.

- Downhole Surveys

  - AMEC performed a 100% audit of the data comparing the database to original data and QC checks comparing multiple surveys with a single instrument and/or comparison of different instruments. No discrepancies were noted
  - AMEC also checked for excessive deviations for the entire data set using a proprietary program called KinkCheck. KinkCheck revealed 71 points where deviation exceeded 3º in 20 ft. 37 points were flagged "do not use". The remainder was found to be related to wedge deviations and considered reasonable by AMEC.

- Logging

  - AMEC reviewed core for three holes with TMM geologists responsible for logging those holes. The logs were found to accurately represent the lithology seen in core. No other audit was possible because lithology is logged directly into the acQuire database.

- Assaying

  - AMEC audited the 100% of the assay data for holes MEX-0436 through MEX-0495 including all QC data for a total of 50,986 assays. AMEC discovered two errors which were immediately corrected.

12.1.3 Maturi Southwest Database Audit

In February and March 2013, AMEC visited drill sites, reviewed quality control measures, and audited the project database. The quality control review and database audit process compared a minimum of 5% of the data in the database to original documents.

12.1.3.1 Collar Surveys

AMEC compared collar locations in the database to the original location documents provided by Northern Lights Surveying Co. and found no discrepancies. Two holes were not included in the check (MSW-009 and MSW-0020) because drill equipment was over the collars of those holes when the surveys were completed. AMEC noted four discrepancies in total depth data in the Vulcan database that were not present in the acQuire database. Those were corrected prior to resource estimation.

On 20 February 2013, AMEC located four pads representing seven drill holes with a hand-held GPS instrument. Multiple collars were located on a single pad, and the AMEC location reflected the “center of mass” of the multiple collars. In all cases AMEC considers the collar locations to have been verified by the hand-held GPS instrument.

12.1.3.2 Downhole Surveys

TMM used a Reflex gyroscopic instrument for downhole surveys at Maturi Southwest. AMEC obtained the original survey documents as digital files, compiled a new downhole survey database, and compared the compiled database to the acQuire database. No errors were discovered. AMEC also checked for excess deviations using a proprietary program, KinkCheck. No excess deviations were found.

12.1.3.3 Assay Data

Current assay data were audited by comparing the Vulcan and acQuire databases to an assay database compiled from digital versions of original assay certificates. Approximately 98% of the assay data were audited, and no errors were discovered.

During the process, AMEC noted that two samples in hole MSW-0018 had anomalous Ag and W values (733-738 ft – 276 ppm Ag, 1,180 ppm W; 748-753 ft – 14 ppm Ag, 70 ppm W). Investigation of those results indicate that the anomalous Ag and W were due to drilling through a stuck bit and reamer shell and represent silver solder and tungsten carbide from the stuck tools. Those values were removed from the database and replaced by the average of the adjacent intervals.

12.1.3.4 Density

Raw density data are entered directly into the acQuire database; thus there is no audit trail. AMEC reviewed the data and recalculated the density and found no obvious errors. Three samples with densities lower than 2.3 g/cm^3 are possibly errors, and AMEC did not use those data for resource estimation.

12.1.3.5 Lithology

TMM logs directly into the acQuire database; thus there is no audit trail. AMEC personnel visited the core logging facility numerous times to observe logging procedures. AMEC found that core was being logged properly. Lithology is not directly used for resource estimation.

12.1.3.6 Geotechnical Logging

While reviewing core recovery data in the geotechnical database, AMEC noted three intervals with exceptionally high core recoveries. The high core recoveries were due to misplacement of a decimal point and were corrected in the database (C. Totenhagen, 3 May 2013, pers. comm.).

12.1.4 Birch Lake Database Audit

In 2011, AMEC selected approximately 6% of the Franconia drill holes and 71% of the legacy drill holes for the purposes of the database audit. Collar locations, downhole surveys, lithology logs and assays were checked against the original documentation for all these drill holes. The audit drill holes were selected to be spatially (equally spaced throughout the Birch Lake deposit), and temporally (equally spaced throughout the drilling campaign period) representative.

Results of the Birch Lake audit are:

- Collar Locations
  - Collar locations for BL drill holes were surveyed by Livgard Surveying, Inc. of Superior, Wisconsin. In 2011 AMEC checked easting, northing, and elevation values for the 14 audit drill holes and found one discrepancy in the elevation values that is likely due to truncation of the original value. AMEC located 24 collars in the field with a hand-held GPS unit and found that the coordinates agreed well with those in the database
  - Legacy drill collars at Birch Lake were surveyed by Livgard Surveying, Inc. of Superior, Wisconsin where they could be located in the field. AMEC checked easting, northing, and elevation values for the five legacy audit drill holes and found one small discrepancy in the elevation values of the three drill holes.
  - In 2012, AMEC checked easting, northing, and elevation values for 31 drill holes in the master database and found one error.
- Downhole Surveys
  - In 2011, AMEC checked the depth, azimuth, and inclination (dip) values for a total of 772 downhole surveys against the original paper survey files. Significant issues were found with six of the 14 audited drill holes. AMEC also found significant issues with some pilot and wedge hole surveys. AMEC recommended that all BL holes that can be re-entered be resurveyed
  - In 2012, TMM re-entered 31 holes and resurveyed the last wedge hole to the bottom of the hole or as deep as possible given hole conditions. Some holes had caved preventing complete resurvey of the holes. Those data were used to adjust the other wedges in each hole set. The 2012 resurvey program

included a number of QC measures including down-the-hole and up-the-hole surveys on most holes and duplicate surveys on some holes

- Downhole surveys for legacy drill holes generally consist of acid-tube tests whose results are typed into the margin or at the end of the lithology log. AMEC checked the depth, azimuth, and inclination (dip) values for downhole surveys from the two legacy audit drill holes that have downhole surveys against the surveys recorded on the original drill logs and found them to accurately represent the original records
- AMEC compared downhole surveys to original documents and found no errors in 2012.

- Lithology Logs
  - In 2011, AMEC checked the From, To, and rock type values for a total of 428 logged intervals from 14 Franconia (BL) drill holes. A total of six errors were found out of the 1,284 values checked for an error rate of 0.5%
  - The lithology codes for the seven material legacy drill holes at Birch Lake were not audited
  - In 2012, Maturi and Birch Lake were audited as a unit. A total of 1,122 records were audited and one error was discovered for an acceptable error rate of 0.1%
  - Lithology logs were compared to core from four holes and no discrepancies were noted.
- Assays
  - In 2011, the four drill holes assayed at Bondar Clegg were audited against paper copies of the assay certificates. ALS Chemex provided AMEC with digital copies of original assay certificates for the remaining 10 audit drill holes through secure login to their website. AMEC checked the From, To, Cu, Ni, Pt, Pd, and Au for 412 assay intervals from the four audit drill holes assayed by Bondar Clegg and found nine errors, for an error rate of 0.3%. AMEC then checked the From, To, Cu, Ni, Pt, Pd, and Au for 1,065 assay intervals from the 10 audit drill holes assayed by ALS Chemex and found no errors
  - In 2012, AMEC checked the From, To, Cu, Ni, Pt, Pd, and Au for 2,381 assay intervals from 31 drill holes and found two errors, for an error rate of 0.01%. AMEC checked the sample interval database records against the sample sheets found in the TMM drill hole folders, and checked the assay values against digital assays downloaded from the ALS Webtrieve website.
  - AMEC checked the From, To, Cu, Ni, Pt, Pd, and Au for assay intervals from four legacy drill holes. Database values matched all original assays, but AMEC found that the Pt, Pd, and Au original assays for two drill holes were not in the database.

12.1.5 Spruce Road Data Checks

The Spruce Road database consists of legacy data with the exception of two holes. The legacy data are from ACNC who explored the area. The assay and lithology data have not been verified by twin holes or other methods. None of the core from that exploration remains. Comparison of a limited number of assay data for ACNC exploration during that time period at Maturi suggests that there are no significant biases at Spruce Road. AMEC believes that the data are adequate to support Inferred Mineral Resources, but additional verification by twin holes is required to support higher resource confidence classification.

12.2 Comment on Section 12

The combined Maturi, Maturi Southwest, and Birch Lake database is adequate to support estimation of mineral resources without restriction.

AMEC considers that the Spruce Road database is adequate to support estimation of only Inferred Mineral Resources because the data are largely unverifiable.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Metallurgical testwork has been completed on the Maturi, Maturi Southwest, Birch Lake and Spruce Road deposits in the Duluth Complex since 1973. Testwork has been based on crushing, grinding and flotation, either to make bulk copper–nickel concentrates, or differential copper and nickel concentrates. Various owners of the deposits have considered either making concentrates for sale to smelters, or for treatment in hydrometallurgical processes that could upgrade copper, nickel and PGEs into higher-value saleable products.

Table 13-1 summarizes the testwork completed on the Project since 1973. Testwork on all four deposits was used for support of the evaluation of reasonable prospects of eventual economic extraction in Section 14, and provided knowledge and understanding that has guided the flowsheet development and metallurgical predictions for Maturi and Maturi Southwest for the PFS. This 2014 PFS focused on treatment of the Maturi and Maturi Southwest deposits, using a crushing, grinding and sequential flotation flowsheet to produce separate copper and nickel concentrates.

13.2 Metallurgical Testwork, Spruce Road and Birch Lake

13.2.1 Spruce Road

In 1973 a 10,000 ton bulk sample from surface pits at Spruce Road was processed at INCO's Creighton mill in Sudbury, Ontario. INCO performed extensive testwork that defined and demonstrated a workable flotation process that gave an average recovery of 89% for copper and 63% for nickel for a bulk flotation concentrate grade of 13.4% Cu and 2.8% Ni.

Spruce Road metallurgical testing was based on a bulk sample taken from a small open pit on the deposit. That sample is not likely representative of the overall deposit, but represented the first few years of production based on the plans at the time.

In 2000, Wallbridge submitted 90 core samples for metallurgical test work. A single composite was prepared from the samples and a series of scoping froth flotation tests. Results indicated that a bulk concentrate with a combined Cu + Ni grade of 15% can be produced at recoveries of 90% for copper and 66% for nickel.

Table 13-1: Metallurgical Testwork Summary Table

Program Name	Laboratory	Date	Description	Flowsheet	Deposit	Composites	Number of Tests
Bulk sample - INCO Creighton Mill	INCO	1973	Flotation	Bulk concentrate	Spruce Road	10,000 t surface bulk sample	
Spruce Road composite testing - Wallbridge	unknown	2000	Flotation	Bulk concentrate	Spruce Road		unknown
Birch Lake composite testing - flotation	SGS Lakefield	2005	Flotation	Bulk concentrate	Birch Lake		unknown
Birch Lake composite testing - crushing	SGS Lakefield	2006	Comminution		Birch Lake		unknown
Birch Lake composite testing - grinding	SGS Lakefield	2008	Comminution		Birch Lake		unknown
Bench scale flowsheet development	SGS Lakefield	2007	Flotation	unknown	Maturi		
Mineralogy	SGS Lakefield	2007	Mineralogy	unknown	Maturi		
Bench scale flowsheet development	SGS Lakefield	2009	Flotation	Copper and bulk concentrates	Maturi		
Platsol™ testing on bulk flotation concentrate	SGS Lakefield?	2011	Flotation conc leaching		unknown		
CESL™ testing	CESL	2011?	Flotation conc leaching		unknown		
Bench scale flowsheet development	SGS Lakefield	Jan–Jul 2011	Flotation	Bulk concentrate	Maturi		
bond impact testing	Phillip Enterprises LLC	2012	Comminution	Grinding	Maturi, Birch Lake		
SMC test report	SGS Lakefield	Aug 2012	Comminution	Grinding	Maturi, Birch Lake		13
SAG comminution design	Dawson Metallurgical Laboratories	May–Aug 2012	Comminution	Grinding	Maturi, Birch Lake		23
Mineralogical characterization - variability samples	SGS Lakefield	2012	Mineralogy - ore		Maturi		60
Mineralogical characterization - composites	SGS Lakefield	2012	Mineralogy - ore		Maturi, Birch Lake		16
Mineralogical characterization - composites	SGS Lakefield	2012	Mineralogy - ore		Maturi, Birch Lake		10
Mineralogical characterization - composites	SGS Lakefield	2012	Mineralogy - ore		Maturi, Birch Lake		2
Mineralogical characterization - bench tests	SGS Lakefield	2012	Mineralogy - bench test products		Maturi, Birch Lake		27
Mineralogical characterization - pilot plant	SGS Lakefield	2012	Mineralogy - bench test and pilot plant products		Maturi, Birch Lake		27
Mineralogical characterization - bench tests and pilot plant	SGS Lakefield	2012	Mineralogy - bench test and pilot plant		Maturi, Birch Lake		27

Program Name	Laboratory	Date	Description	Flowsheet	Deposit	Composites	Number of Tests
			products				
Concentrate mineralogy	Cabri Consulting	2013	Mineralogy - concentrates	Bulk concentrate	Maturi	unknown	1
Sample characterization, flocculant screening, gravity sedimentation, pulp rheology, vacuum filtration and pressure filtration	Pocock Industrial	Jun 2013	Solid liquid separation		Maturi		
Bench scale flowsheet development	ALS Kamloops	Oct 2012–Feb 2013	Flotation		Maturi		72
Investigative pilot plant Testing	ALS Kamloops	Jan 2013–Apr 2013	Flotation		Maturi	PP-3	41
Bench scale testing to support pilot plant	ALS Kamloops	Feb 2013–Mar 2013	Flotation	Sequential flotation - Cu and Ni concentrates	Maturi		85
Locked cycle testing using PP-3 samples	ALS Kamloops	Mar 2013	Flotation		Maturi	PP-3	3
Locked cycle testing using end-member composites	ALS Kamloops		Flotation		Maturi, Maturi Southwest, Birch Lake	End members	17
Variability rougher testing	ALS Kamloops		Flotation		Maturi, Maturi Southwest, Birch Lake		98
ALS comminution testing on variability samples	ALS Kamloops	2013	Comminution		Maturi, Maturi Southwest, Birch Lake		
Copper circuit optimization on PP-3 composite	Blue Coast		Flotation		Maturi		131
Nickel circuit optimization on PP-3 composite	Blue Coast		Flotation		Maturi		31
Pyrrhotite rejection flowsheet development	Blue Coast		Flotation		Maturi		22
Locked cycle confirmation testing using PP-3 samples	Blue Coast		Flotation	Sequential flotation - Cu and Ni concentrates	Maturi		7
Flowsheet fine-tuning on sub-domain composites (SDC)	Blue Coast		Flotation		Maturi		29
Locked cycle confirmation testing on SDCs	Blue Coast		Flotation		Maturi		12
Locked cycle confirmation testing on LOM composites	Blue Coast		Flotation		Maturi		4
Locked cycle confirmation testing on pyrrhotite rejection	Blue Coast		Flotation		Maturi		18

13.2.2 Birch Lake

Metallurgical samples were collected from pilot and wedge holes at Birch Lake (Figure 13-1) and comprised composite drill core. The samples are considered to be representative of the overall deposit, but may not account for local metallurgical variability.

Bench-scale flotation testwork on a composite sample of drill core from the Birch Lake deposit was undertaken at SGS Lakefield Research (Lakefield) in 2005. Additional flotation testwork was initiated in late 2006, and crushing and milling work index and grindability testing was done in late 2008.

In 2012, a selection of variability samples was processed through laboratory-scale locked-cycle tests. The results obtained were:

- Cu recovery to bulk concentrate varies from 90 to 95% depending on grinding size and Cu feed grade
- Ni recovery to bulk concentrate varies from 60 to 76%, and it is limited by the amount of silicate-hosted Ni, which is greater than Maturi, comprising 25% to 30% of total nickel in the sample
- Au, Pt and Pd recovery varies from 82% to 93%, depending on head grade, grind size, and final concentrate grade.

Production of medium-grade bulk concentrate for hydrometallurgical testing with 14%–16% S content and 11%–12% Cu plus Ni was evaluated at pilot plant scale (450 kg/h) in a simple rougher and regrinding–cleaner circuit.

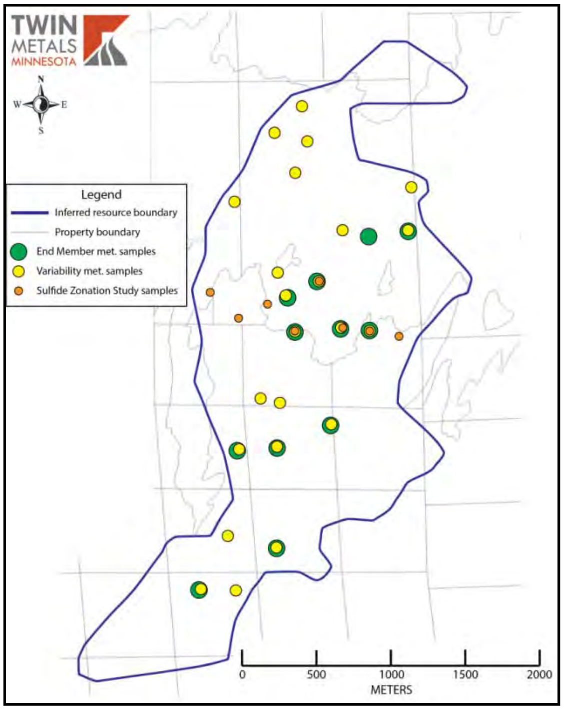
The flotation testwork allowed preliminary definition of a bulk flotation flowsheet (Figure 13-2).

13.3 Alternative Process Routes Tested Prior to Selection of PFS Configuration

13.3.1 Platsol™ Testwork

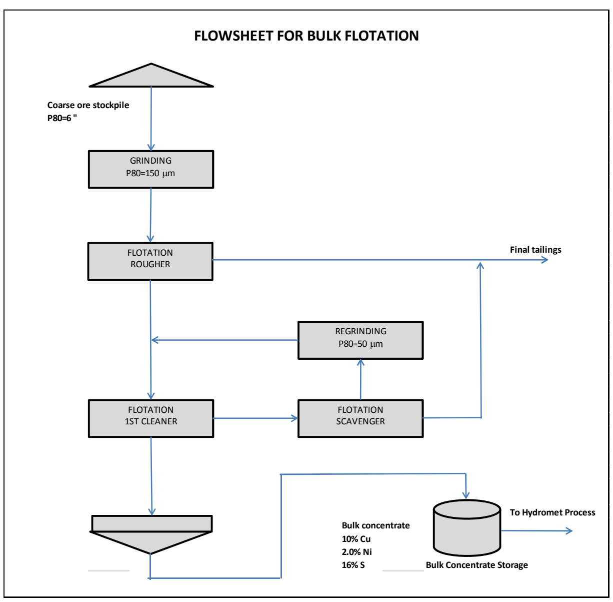
Early testwork on the Maturi and Birch Lake deposits reviewed the use of Platsol™ technology for treatment of bulk Cu/Ni flotation concentrates; however, this recovery method was not progressed through the PFS. For completeness, a summary of the results of the work has been included in this sub-section. Platsol™ is a high-temperature, chloride-assisted, pressure-leaching procedure.

Figure 13-1: Locations of Birch Lake Metallurgical Samples



Note: Figure prepared by TMM, 2014.

Figure 13-2: Preliminary Bulk Flotation Flowsheet



Note: Figure prepared by TMM, 2012.

By the end of 2011, almost 70 individual bench Platsol™ tests had been completed on several different flotation concentrates, with results indicating that Maturi and Birch Lake concentrates are amenable to Platsol™ technology extraction and recovery: typical extraction values from bulk concentrate are 99% for Cu, 99% for Ni, 90% for Pt, 90% for Pd, and 85% for Au.

13.3.2 CESL™ Testwork

Prior to the development of the process circuit in the PFS, several technologies, including Teck Resources Limited's (Teck) CESL™ Technology (CESL™) were considered to process the bulk Cu/Ni concentrate produced from Maturi, Birch Lake, and other deposits. CESL™ is a medium-temperature, chloride-assisted, pressure-leaching procedure.

Trade-off studies performed as part of the PFS indicated that with the current state of Project metallurgical testwork information, CESL™ was not the chosen process route, and the CESL™ process was not investigated further as part of the PFS.

For completeness, a summary of the results of the work has been included in this sub-section.

The testwork indicated that three major-value products could be produced from the Maturi concentrate in the CESL™ testwork:

- LME Grade A copper cathode
- Mixed hydroxide precipitate (MHP) cake
- PGM concentrate.

Typical extraction values are 98% for Cu, 97% for Ni, and recoveries of 75% Pt, Pd, and Au.

Based on the test results, the LME Grade A copper cathode will meet the specification of 99.99% copper. Table 13-2 summarizes the approximate composition of the mixed hydroxide precipitation cake produced from application of the CESL™ process.

The PGM concentrate produced during sulfur flotation will have the composition in Table 13-3.

Table 13-2: CESL™ Mixed Hydroxide Precipitate Composition

Ni (%)	Co (%)	S (%)	Mn (%)	Mg (%)	Cl (%)
46.2	1.4	4.7	0.2	0.9	0.1

Table 13-3: Sulfur Flotation PGM Concentrate

Pt (g/t)	Pd (g/t)	Au (g/t)	Ag (g/t)	S (%)
8.08	28.6	5.65	76	85

The sulfur flotation PGM concentrate would have an overall PGM grade of 42.3 g/t and will require additional downstream upgrading before being sold to the market. CESL™ proposed that upgrading of the PGM concentrate can be done in an acid plant where the sulfur is converted to acid, and the upgraded PGM containing dust can be collected and sold.

13.4 PFS Metallurgical Sampling

The mineral processing and metallurgical information for the PFS has been derived from extensive testwork conducted on a variety of samples acquired during drilling campaigns conducted between 2010 and 2012. The majority of the mineral processing testwork for this PFS was performed between 2012 and 2014 and was conducted primarily at ALS Metallurgy (previously G&T Metallurgical Services), Kamloops, BC, Canada, and at Blue Coast Research, Parksville, BC, Canada.

Significant supporting testwork and analysis was conducted by others, including Blue Coast Metallurgy, Parksville, BC, Canada; DJB Consultants Inc., Vancouver, BC, Canada; SGS Minerals, Lakefield, ON, Canada; Golder Associates Inc., Redmond, WA, USA and Mississauga, ON, Canada; Pocock Industrial Inc., Salt Lake City, UT, USA and FLSmidth Inc., Midvale, UT, USA. As part of the overall project evolution and evaluation, but ultimately not relevant to the PFS, many other facilities and organizations were involved in testwork programs and trade-off studies.

13.5 PFS Metallurgical Sampling, Sample Preparation and Characterization

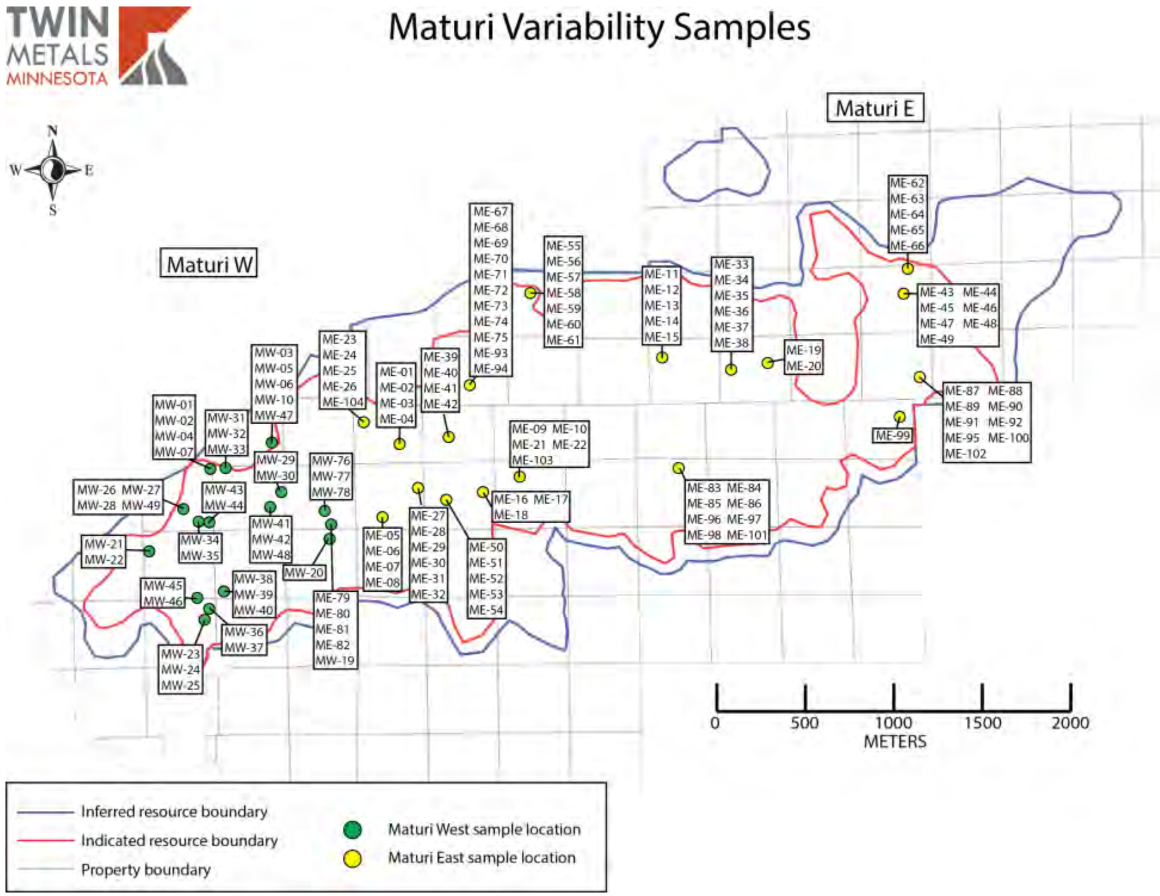
Metallurgical sampling for the PFS consisted of collection of drill core from various drill programs from 2010 through 2012. The drill core was available either in NQ or PQ sizes. From the set of drill core available, a variety of metallurgical samples were created described as follows:

- Variability samples: 10–15 ft. continuous intervals from a single drill hole and typically 25–30 kg sample weight. A total of 143 variability samples from the Maturi deposit and up to 50 variability samples from the Birch Lake deposit were acquired in several drilling programs. The source locations for the Maturi variability samples are shown in Figure 13-3. Of the 143 samples, 62 were sourced from S3 and 52 from S2 (refer to Section 7 for a description of these geological units).

- End-member (or domain) composite samples: approximately 100 ft intervals, sourced from multiple drill holes or multiple wedges from a single drill hole. A total of 17 end-member samples from the Maturi deposit and two end-member samples from the Maturi Southwest deposit were acquired in several drilling programs. Grindability testwork was the only work conducted on these composites that is relevant to the PFS. The source locations for the various Maturi end-member samples are shown in Figure 13-4.
- Sub-domain composite (SDC) samples; 48 kg composites blended from two to five variability samples, designed to cover a broad spectrum of pyrrhotite to pentlandite ratios, and represent one of the six Maturi geometallurgical sub-domains. The SDCs were named for their location (S (shallow), D (deep), DE (deep east)), their geological unit (S2 or S3), and the ratio of pentlandite to pyrrhotite (H (high), M (medium), L (low)) as determined by QEMSCAN. Sub-domain composite drill collar locations are included in Figure 13-5.
- Life-of-mine composite samples; four composites of 30–40 kg, blended from multiple sub domain composites or variability samples reflecting as well as practicable the four distinct phases in the mine life as outlined in the December 2013 proposed mine plan. The criteria used to design the composites were copper and nickel head grade, ratios of chalcopyrite to cubanite, and pentlandite to pyrrhotite, deposit metallurgical zone (S, D, DE) and geologic units (S2 vs S3).
- Pilot plant composite samples (PP composite 3); total of ~140 st collected during 2012; large composite samples, multiple drill holes. The location of the holes used for the pilot plant composites is shown in Figure 13-6.

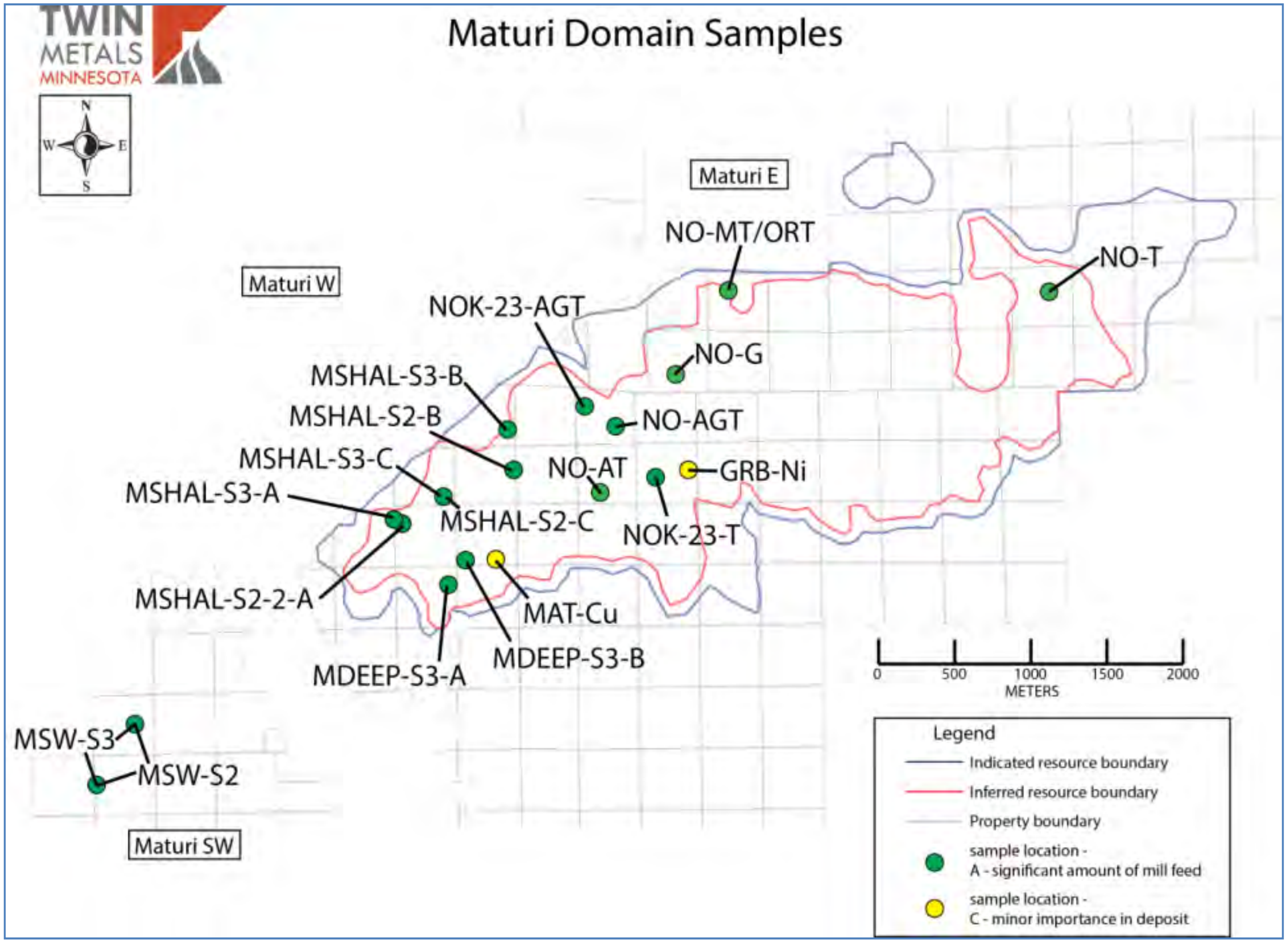
The outline of the geometallurgical domains are shown in Figure 13-7, superimposed on the mine plan that was current at the time the domains were devised. Head assays for the SDCs are shown in Table 13-4. Head assays, source material proportions and mineralogical ratios of the actual life-of-mine composite samples in comparison to design targets are shown in Table 13-5. Head assays for the pilot plant composite as determined by ALS, are shown in Table 13-6.

Figure 13-3: Maturi Variability Sample Location Plan



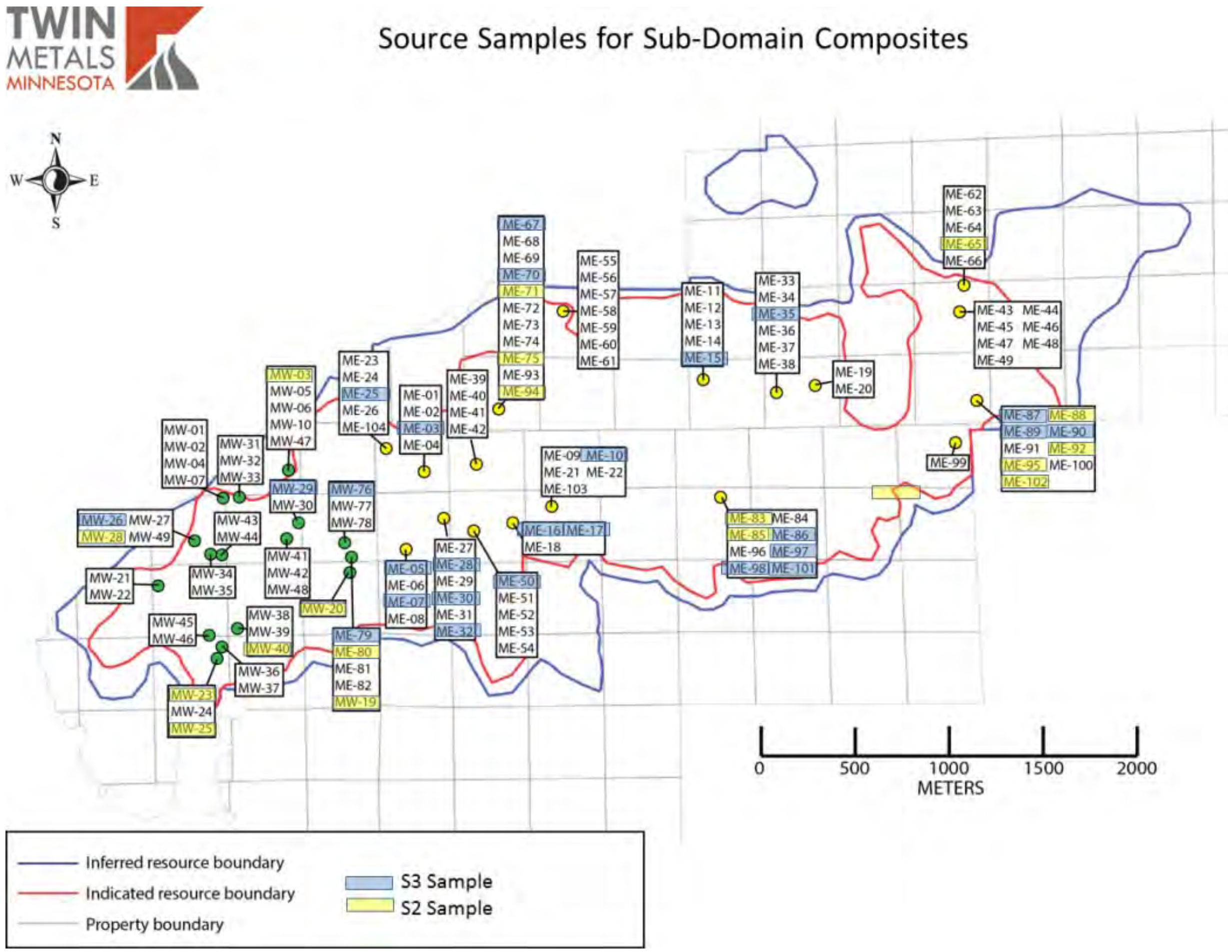
Note: Figure prepared by Blue Coast, 2014. Resource boundaries shown on the figure were current as of November 2012 and are not the current 2014 estimate boundaries.

Figure 13-4: Plan View of the Deposit Showing End Member Sample Locations



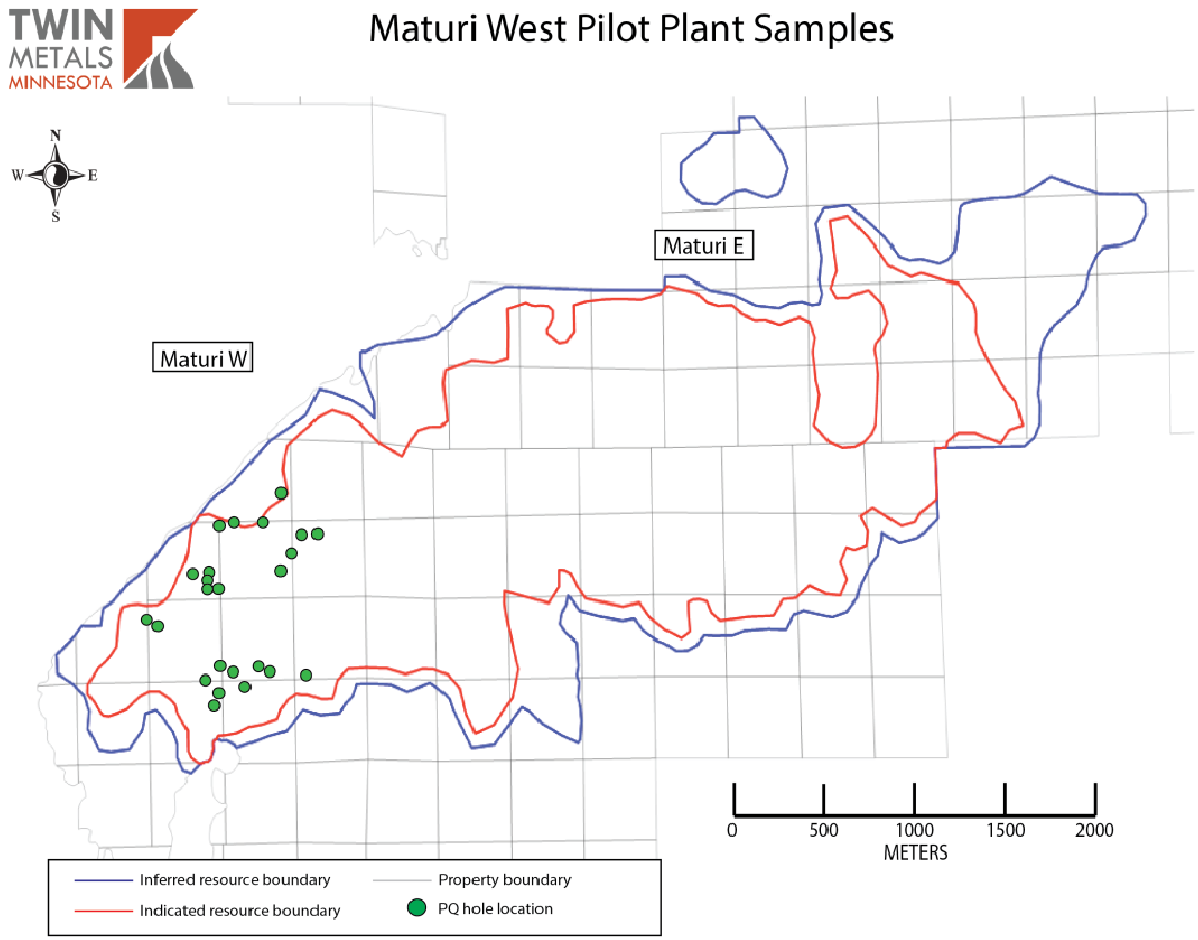
Note: Figure prepared by Blue Coast, 2014. Resource boundaries shown on the figure were current as of November 2012 and are not the current 2014 estimate boundaries.

Figure 13-5: Source Sample Locations for the Sub Domain Composites



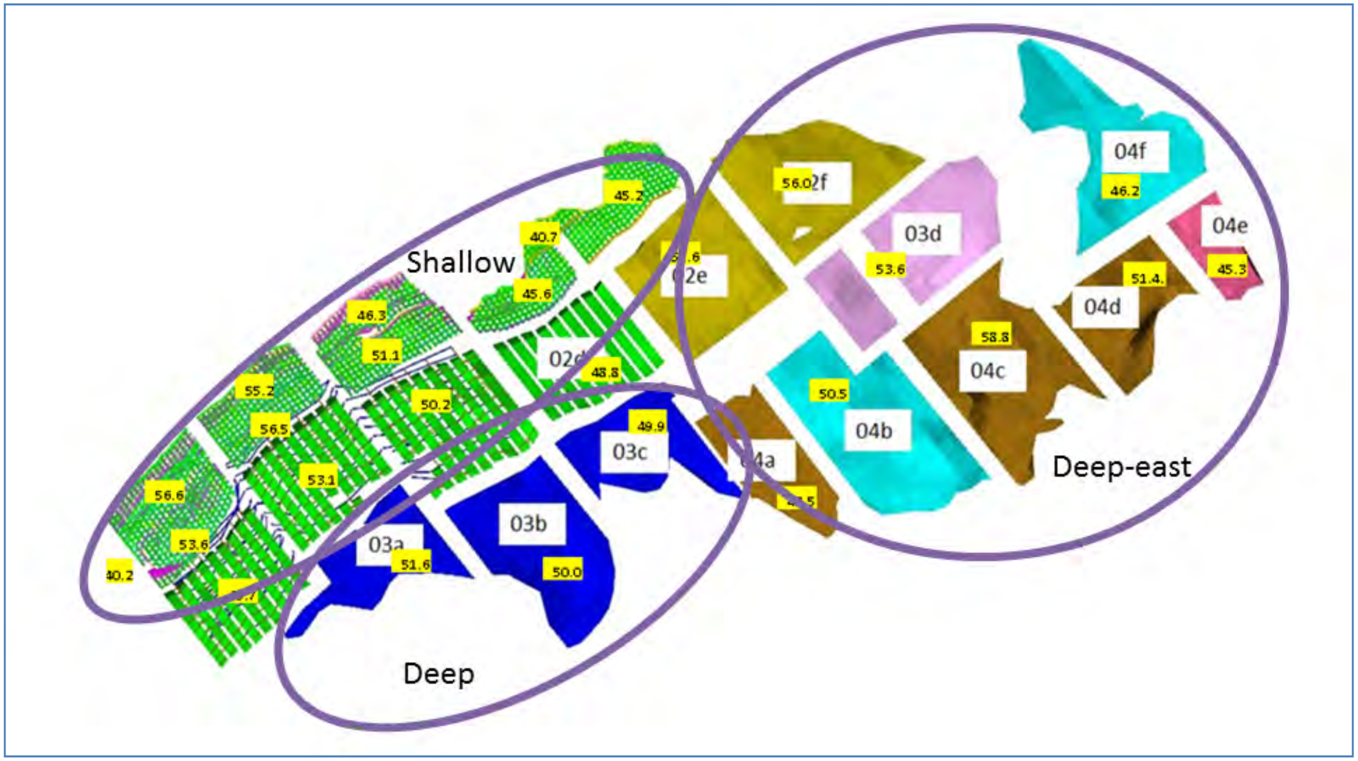
Note: Figure prepared by Blue Coast, 2014. Resource boundaries shown on the figure were current as of November 2012 and are not the current 2014 estimate boundaries.

Figure 13-6: Location of Parent Holes Used to Create the Pilot Plant Material



Note: Figure prepared by Blue Coast, 2014. Resource boundaries shown on the figure were current as of November 2012 and are not the current 2014 estimate boundaries.

Figure 13-7: Metallurgical Zones in Relation to Mining Panels, Maturi Deposit



Note: Figure prepared by Blue Coast, 2014. Mining panel boundaries shown on the figure are of a preliminary design that was current as of fall 2013 and are not the current 2014 panel configurations as discussed in Section 16 of this Report.

Table 13-4: Head Assays of the Sub Domain Composites (SDC)

	Cu (%)	Ni (%)	Ni(S) (%)	Fe (%)	S (%)	Au (g/t)	Pt (g/t)	Pd (g/t)
Shallow-S3-L	0.60	0.20	0.13	8.28	0.95	0.07	0.12	0.27
Shallow-S3-M	0.59	0.22	0.15	13.76	0.98	0.05	0.15	0.26
Shallow-S3-H	0.77	0.26	0.13	12.27	1.15	0.08	0.18	0.36
Shallow-S2-L/M	0.51	0.17	0.12	12.95	1.00	0.04	0.07	0.18
Shallow-S2-H	0.53	0.16	0.11	15.64	0.91	0.04	0.07	0.22
Deep-S3-L	0.64	0.21	0.17	9.29	1.09	0.08	0.27	0.35
Deep-S3-M	0.60	0.24	0.13	10.44	0.91	0.08	0.21	0.36
Deep-S3-H	0.64	0.22	0.14	11.15	1.06	0.06	0.15	0.41
Deep-S2-L/M	0.53	0.19	0.13	10.92	0.90	0.05	0.10	0.24
Deep-S2-H	0.40	0.14	0.08	9.93	0.70	0.05	0.07	0.15
Deep East-S3-L	0.73	0.22	0.13	10.84	1.21	0.12	0.23	0.53
Deep East-S3-M	0.72	0.23	0.16	9.66	1.26	0.09	0.26	0.49
Deep East-S3-H	0.80	0.26	0.18	10.31	1.35	0.11	0.26	0.65
Deep East-S2-L	0.47	0.13	0.09	10.63	0.76	0.08	0.13	0.28
Deep East-S2-M	0.44	0.14	0.08	11.42	0.65	0.05	0.10	0.25
Deep East-S2-H	0.69	0.19	0.13	11.51	1.11	0.07	0.22	0.46

Table 13-5: Target and "As Produced" Life-of-Mine Blends

Target Blend	Cu (%)	Ni (%)	Po:Pn (ratio)	Cb:Cp (ratio)	Percentage in Blend (%) Shallow S2	Shallow S3	Deep S2	Deep S3	Deep East S2	Deep East S3
Yr 1–3 Composite	0.70	0.23	1.15	0.60	22	78	—	—	—	—
Yr 4–8 Composite	0.64	0.22	1.0	0.55	8	73	—	1	—	18
Yr 9–19 Composite	0.63	0.19	0.8	0.50	1	7	4	18	6	63
Yr 20+ Composite	0.45	0.15	1.6	0.50	31	18	15	11	14	11
As Produced	**Cu (%)**	**Ni (%)**	**Po:Pn (ratio)**	**Cb:Cp (ratio)**	**Percentage in Blend (%) Shallow S2**	**Shallow S3**	**Deep S2**	**Deep S3**	**Deep East S2**	**Deep East S3**
Yr 1–3 Composite	0.70	0.23	1.15	0.46	24	76	—	—	—	—
Yr 4–8 Composite	0.64	0.22	0.94	0.60	19	50	—	—	—	31
Yr 9–19 Composite	0.61	0.20	0.80	0.50	6	6	12	6	24	47
Yr 20+ Composite	0.43	0.15	1.60	0.50	40	—	14	16	14	16

Note: Po = pyrrhotite, Pn = pentlandite, Cb = cubanite, Cp = chalcopyrite

Table 13-6: ALS Determined Head Assays in PP Composite 3

Composite	Assay (%) Cu	Fe	Ni (total)	Mg	S (total)	Assay (ppm) Au	Pt	Pd
PP Composite 3	0.67	9.3	0.28	6.37	1.29	0.06	0.11	0.36

13.5.1 Mineralogy

Mineralogy is also discussed in Section 7.4.3. Extensive QEMSCAN studies of the copper, nickel and host rock mineralogy have been conducted through the program. All mineralogy discussed in this Report section was conducted by SGS in Lakefield.

Copper mineralization is dominated by chalcopyrite and cubanite. Very minor bornite and secondary copper sulfides are also present. The biggest impact of the copper speciation is on the achievable copper concentrate grade as the mineral cubanite has a lower copper content (23% Cu) than chalcopyrite (34% Cu).

The resource-wide copper deportment has been determined by QEMSCAN from a total of 212 S2 and S3 samples. On average across the mineable zones the ratio of mineral abundance between cubanite and chalcopyrite is 0.52:1 and the copper grade in the copper sulfides was 30.6%. Although described in the QEMSCAN data for this project as chalcopyrite, some of the chalcopyrite is in fact talnakhite, where some of the iron has been replaced with nickel. The average probed concentration of nickel in TMM "chalcopyrite" is 0.22%.

Nickel mineralization has been studied extensively by SGS and is quite complex. Most of the nickel is present as pentlandite. However, nickel occurs in a wide variety of host minerals in the Maturi deposit, including the sulfides chalcopyrite (talnakhite) and pyrrhotite (a very minor host), and non-sulfide minerals such as olivine (the second-largest host), iron oxides, mica and chlorite. The proportion of nickel present as pentlandite is the primary driver behind the nickel recovery.

The overall mean resource-wide mineral modal abundance, based on the 212 samples analyzed to date, is shown in Table 13-7. This has been calculated by averaging the S2 and S3 analyses for each mineable zone within the deposit and weighting these by the mineable tonnage for each of the zones. The mineralogical limiting concentrate grade, driven by the ratio of chalcopyrite to cubanite, is shown in Figure 13-8.

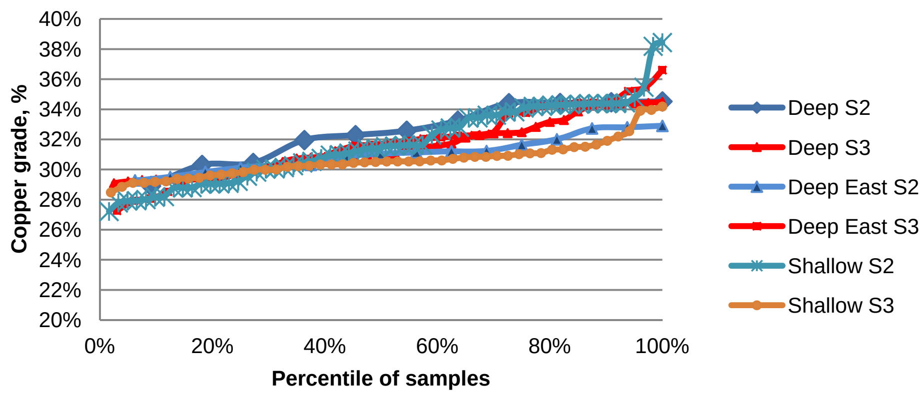
Aside from the copper and nickel sulfides, pyrrhotite is a key component in the deposit. Pyrrhotite typically has similar flotation characteristics to pentlandite, and would typically report to a nickel flotation concentrate. The ratio of pyrrhotite to pentlandite is therefore a key parameter in predicting nickel concentrate grade and averages close to 1:1 across the Maturi deposit. Such a ratio potentially allows for the production of salable nickel concentrates, even if the pyrrhotite is floated; however, the lower-grade more S2-rich feed materials prevalent in the latter years of mill feed contain an elevated pyrrhotite content that would likely reduce nickel concentrate grades to below saleable grade. Modifications to the reagent scheme to reject pyrrhotite would therefore need to be employed on this material, coming at the cost of lower nickel recoveries.

Maturi pyrrhotite is a mix of troilite and hexagonal pyrrhotite, is non-magnetic, and hence is not amenable to magnetic recovery to remove from the nickel concentrate. Figure 13-9 describes the *calculated* nickel grades assuming all the pentlandite and pyrrhotite are recovered to the nickel concentrate.

Table 13-7: Modal Mineral Abundance of Major Minerals by Zone and Geologic Unit (modal abundance in %)

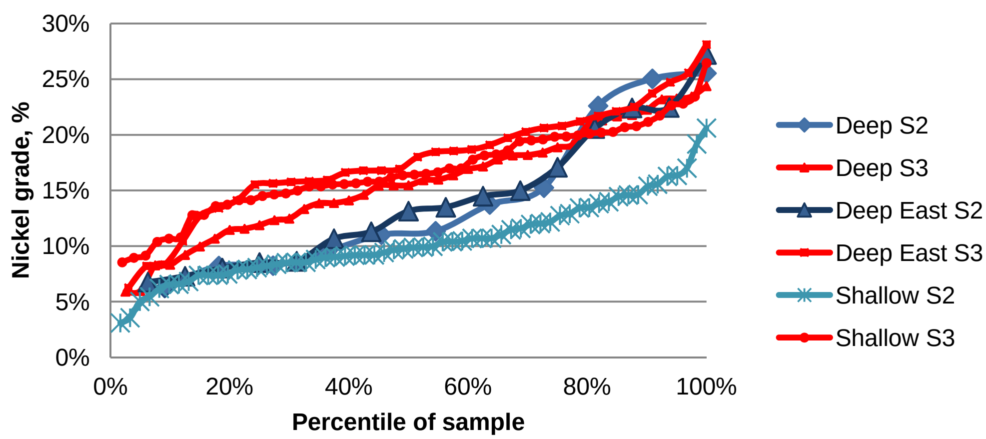
Location	Geologic Unit	# Samples	Chalcopyrite	Cubanite	Pentlandite	Pyrrhotite	Pyrite	Olivine	Ilmenite	Clinopyroxene
Deep	S2	11	1.14	0.39	0.43	0.80	0.04	17.5	2.62	9.2
	S3	40	1.49	0.84	0.68	0.78	0.01	22.6	1.73	4.8
Deep East	S2	16	1.42	0.47	0.38	0.52	0.09	13.1	3.90	8.3
	S3	33	1.78	0.97	0.65	0.51	0.04	20.1	1.98	4.1
Shallow	S2	61	1.22	0.76	0.44	0.97	0.02	17.6	4.42	9.5
	S3	51	1.56	0.93	0.69	0.59	0.02	26.7	1.66	4.3

Figure 13-8: Mineralogical Limiting Copper Grade by Zone and Lithology



Note: Figure prepared by Blue Coast, 2014.

Figure 13-9: Mean Nickel Grade in Combined Pentlandite/Pyrrhotite Minerals



Note: Figure prepared by Blue Coast, 2014.

The Maturi silicates are almost entirely primary in nature. Altered silicates are found in isolated shear zones, and these are relatively uncommon in the deposit. For example, the mean talc content in the deposit is just 0.1%, thus the resulting potential for low MgO and SiO_2 levels in the nickel concentrates will be favorably received by nickel smelters.

The mineralogical compositions of the SDC, LOM and pilot plant composites are shown in Table 13-8. Liberation of copper sulfides and pentlandite in the SDC and LOM samples is shown in Figure 13-10.

In all cases, the sulfides are sufficiently liberated for rougher flotation, and these data point to the need for a light regrind to enhance silicate rejection and achieve target (~90%) liberation for cleaner flotation.

13.5.1.1.1 Pilot Plant Composite 3

Chalcopyrite dominates the copper deportment, and hosts 68% of the copper, while 78% of the nickel is in pentlandite and 17% in olivine. The pyrrhotite:pentlandite ratio in PP Comp 3 is a relatively favorable 0.7:1. The gangue mineralization is mostly primary with very minor altered mineralization. PGEs mainly occur as discrete minerals. Only palladium and rhodium deport to a significant degree in pentlandite (averaging 11 ppm, and 3 ppm respectively). Rhodium also deports to pyrrhotite and pyrite but at very low levels (~0.01 to 1.85 ppm and ~0.01 to 1.16 ppm respectively), and pyrite also contains trace quantities of gold (~0.01 to 1.36 ppm). Gold was not measured in any other sulfide mineral.

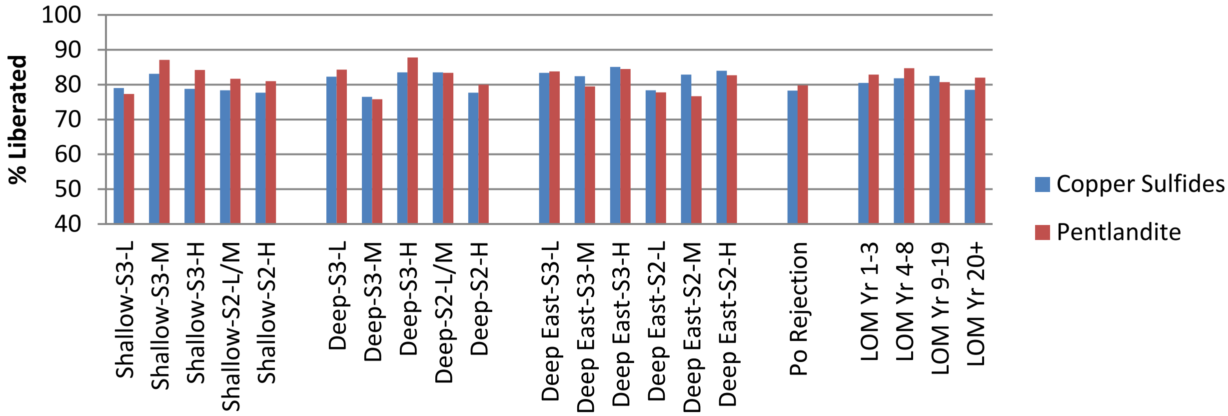
Platinum does not deport as a trace constituent in any of the sulfides, occurring only as discrete Pt minerals. Chalcopyrite and cubanite do not host any of the heavy PGEs (Os, Ir, and Pt) or Au, while preliminary data indicate that about 20% of the Pd deports within pentlandite.

The vast majority of the gold and PGE are therefore present as discrete minerals, with sperrylite, silver–gold alloys, sobolevskite, and froodite being the dominant minerals comprising 31%, 22%, 17% and 15% respectively of the gold and PGEs.

Table 13-8: Mineralogical Compositions of SDC, LOM and Pilot Plant Composites (figures in %)

Composite ID	Chalcopyrite	Cubanite	Pentlandite	Pyrrhotite	Fe-Oxides	Ilmenite	Quartz	K-Feldspar	Plagioclase	Pyroxene	Olivine	Talc	Other
Shallow-S3-L	1.3	0.7	0.5	0.6	0.7	1.6	0.8	0.7	60.9	9.0	18.7	0.1	4.4
Shallow-S3-M	1.6	0.8	0.6	0.4	3.4	5.9	2.6	1.4	41.3	9.4	26.4	0.2	5.7
Shallow-S3-H	2.1	1.3	0.9	0.5	1.0	1.5	0.5	0.7	45.4	11.3	29.5	0.1	5.1
Shallow-S2-L/M	1.4	0.9	0.5	0.9	1.7	4.8	0.0	0.6	45.4	17.0	22.8	0.1	3.9
Shallow-S2-H	1.4	0.8	0.4	0.5	6.2	8.0	0.4	0.4	42.8	16.9	17.4	0.1	4.6
Deep-S3-L	1.8	0.4	0.6	0.8	1.2	1.4	1.2	1.3	54.7	22.5	10.9	0.2	2.9
Deep-S3-M	1.8	0.7	0.5	0.6	0.7	1.7	0.1	0.4	51.5	17.6	18.8	0.2	5.7
Deep-S3-H	1.6	1.3	0.6	0.4	0.6	1.4	0.0	0.2	55.6	7.3	28.1	0.1	2.7
Deep-S2-L/M	1.3	0.9	0.4	0.8	1.4	3.1	0.1	0.6	51.7	16.6	20.8	0.1	2.4
Deep-S2-H	1.2	0.4	0.3	0.2	0.9	2.4	0.0	2.2	51.1	12.8	22.7	0.2	5.4
Deep East-S3-L	1.8	1.3	0.6	0.6	1.9	3.3	0.0	0.5	52.1	11.5	24.4	0.0	2.0
Deep East-S3-M	2.1	1.0	0.6	0.4	0.7	1.9	0.2	0.7	56.0	10.2	23.5	0.0	2.4
Deep East-S3-H	2.0	1.2	0.7	0.3	0.5	1.5	0.0	0.2	53.9	9.1	28.6	0.0	2.1
Deep East-S2-L	1.3	0.6	0.3	0.7	2.6	4.4	0.3	0.7	50.6	17.1	18.7	0.0	2.6
Deep East-S2-M	1.3	0.4	0.3	0.3	1.8	4.0	0.1	0.4	51.4	16.0	20.6	0.1	3.3
Deep East-S2-H	2.0	0.8	0.7	0.5	2.0	3.8	0.1	0.4	47.2	17.7	21.4	0.1	3.2
Po Rejection	1.5	0.5	0.5	1.1	3.5	3.5	1.7	2.4	42.1	27.3	12.1	0.1	3.6
LOM Yr 1–3	1.7	0.8	0.6	0.7	1.8	3.3	0.7	0.7	47.4	10.3	27.4	0.2	4.8
LOM Yr 4–8	1.7	1.0	0.6	0.6	2.3	4.3	1.1	0.9	45.9	11.7	25.5	0.1	4.2
LOM Yr 9–19	1.7	0.9	0.5	0.4	1.4	3.0	0.3	0.6	52.5	12.2	23.3	0.1	3.1
LOM Yr 20+	1.2	0.6	0.4	0.7	1.6	2.9	0.8	1.1	50.3	16.5	20.5	0.0	3.2

Figure 13-10:Liberation of Copper Sulfides and Pentlandite in the SDC and LOM Samples



Note: Figure prepared by Blue Coast, 2014.

13.6 PFS Comminution Studies

Comminution testing was conducted by SGS on 17 end-member composites, and 143 variability samples by ALS (of which up to 114 represent materials that would be processed). The mean, 20th, 50th and 80th percentile numbers are shown in Table 13-9. The distribution of selected work indices is shown in Figure 13-11.

The crusher work index data are bi-modally distributed, apparently driven by the laboratory that conducted the tests. Investigations into the causes of this, at the time of writing, have failed to reveal any satisfactory explanations for the difference, however industry-wide, crusher work index data tend to be lower than both the Bond rod and Bond ball mill work indices, potentially making the SGS data the more valid data.

Similarly, the abrasion index data are bi-modally distributed, again driven by the source laboratory of the data. However none of the data point to a highly abrasive material. The distribution of data on JK parameters are shown in Figure 3-12. The mean DWI is 5.1 and the mean A x b is 64.4, which is moderately soft from a SAG milling perspective.

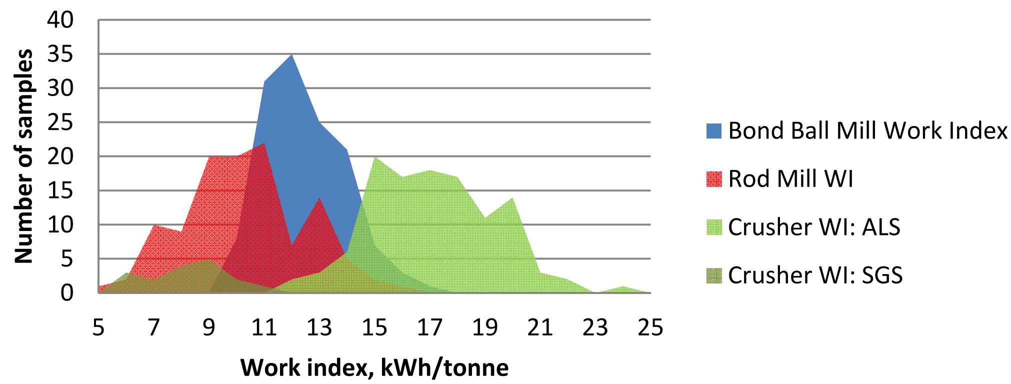
A total of 16 high pressure grinding roll tests were performed as part of this program. These tests are modified versions of the Polysius "Labwal" test and are designed for conceptual level analysis. The data distribution from these tests is shown in Table 3-10.

Table 13-9: Grindability Characteristics of Maturi Samples

Percentile /mean	BWI (kWh/t)	AI (g)	CWI (kWh/t)	Relative Density	RWI (kWh/t)	JK Parameters DWi(kWh/m^3)	A x b	CEET Ci	SPI (Min)
20th	11.7	0.102	14.6	3.0	9.0	4.1	49.7	6.3	40.7
50th	12.7	0.133	17.0	3.0	10.6	5.1	59.9	7.1	48.6
80th	14.1	0.173	19.3	3.2	12.7	6.1	73.9	10.4	80.4
Mean	12.9	0.149	16.4	3.1	10.8	5.1	64.4	8.3	57.3

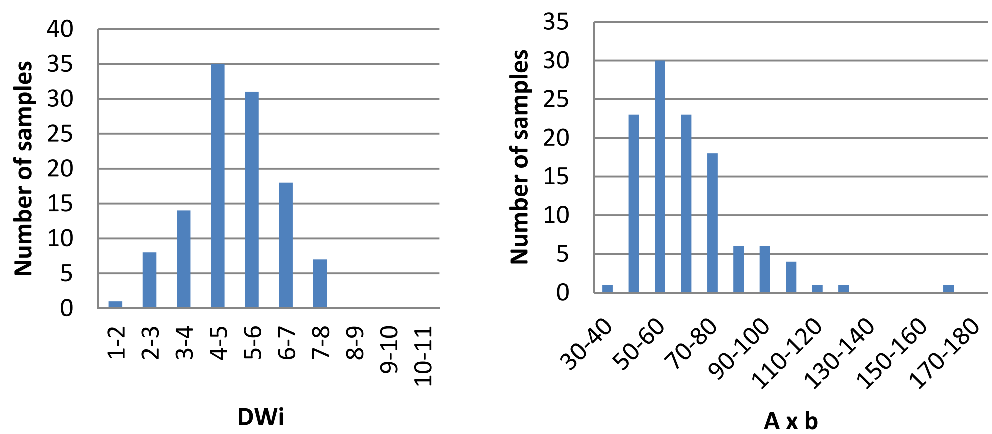
Test notes: Bond Ball Mill Work Index: 114 tests on variability samples, closing screen size of 212 µm and 17 tests on end members at a closing screen size of 150 µm. Difference in mean BWI between dataset was 0.1 kWh/tonne. Bond Rod Mill Work Index: 79 tests on variability samples and 17 tests on end members, closing screen size 1.2 mm for all tests. Tonnage units are metric tonnes.

Figure 13-11: Data Population Distributions of Work Indices



Note: Figure prepared by Blue Coast, 2014. Tonnage units are metric tonnes.

Figure 13-12: Distribution of JK Grindability Data



Note: Figure prepared by Blue Coast, 2014.

Table 13-10: Distribution of HPGR Test Data from Analyses of 16 End-Member Samples

Percentile/mean	t/h	Net (kWh/t)	N/mm^2	m_f	P_{80}
20^{th}	2.9	1.35	2.93	256	3773
50^{th}	3.0	1.43	3.01	270	3996
80^{th}	3.1	1.52	3.05	277	4130
Mean	3.0	1.46	3.01	267	3992

Note: Tonnage units are metric tonnes

13.6.1 Maturi Southwest Samples

In addition to the variability samples which ALS performed comminution testwork on, ALS also performed the bond rod mill grindability test, bond ball mill grindability test, bond abrasion index test and SMC tests on two Maturi Southwest samples, one from S2 and one from S3. The results are summarized in Table 13-11 and are slightly harder than the greater Maturi domain, i.e. Maturi Southwest has a slightly higher Bond mill BWi and lower A x b values. Increased sampling of this deposit will be required in the next phase of the Project.

13.7 Metallurgical Testing in Support of PFS Design

Metallurgical (flotation) testing relevant to this prefeasibility report was started at ALS in Kamloops in October 2012. Since then more than 550 flotation tests and pilot plant runs have been conducted in 14 different test programs:

- Bench scale preliminary flowsheet development: This initial program first developed the use of triethylenetetramine (TETA) and sodium sulfite for Ni rejection from the copper concentrate and created the potential for sequential copper/nickel flotation (Johnston, 2013)
- Pilot plant testing: Some 26 of the 41 pilot plant runs done at 200 kg/hr were devoted to demonstrating the production of saleable grade copper and nickel concentrates (Mehrfert, 2013; Crowie and Thorpe, 2014)
- Bench scale testing to support pilot plant: This program provided bench scale benchmarking of the PP-3 composite, and also examined ways of treating circulating water in the circuit (Mehrfert, 2013)
- Locked-cycle testing of PP-3 composites: Locked-cycle simulation of the pilot plant using the ALS flowsheet (Mehrfert, 2013)
- Locked-cycle testing of end-member composites: Variability program of locked-cycle tests using the ALS-derived flowsheet. This program exposed the flaws in the ALS flowsheet that prompted the work to follow at Blue Coast (Johnston, 2013).

Table 13-11: Summary of Maturi Southwest Comminution Tests

Sample Name	BWI (kWh/t)	AI (g)	RWI (kWh/t)	JK Parameters DWi kWh/m^3	A x b
MSW-S2	13.5	0.145	12.9	6.79	44.6
MSW-S3	14.3	0.117	12.3	5.92	51.1

Note: Tonnage units are metric tonnes

- Variability rougher testing: This formed a large part of the variability program and was mostly completed at ALS. A short program was also conducted at Blue Coast using the same flowsheet to complete the dataset (Johnston, 2013)
- Copper circuit optimization on PP-3 composite: This systematic optimization program was conducted to optimize the copper circuit (Middleditch, 2014)
- Nickel circuit optimization on PP-3 composite: A similar optimization program on nickel flotation (Middleditch, 2014)
- Pyrrhotite rejection flowsheet development: This program exploited the ability of TETA and especially sodium sulfite used with xanthate to effect a separation between pentlandite and pyrrhotite flotation. This flowsheet has been adopted for the last few years of the mine plan when pyrrhotite-rich materials are expected to be delivered to the mill (Hegarty, 2014)
- Locked-cycle confirmation testing on PP-3: This program evaluated the response of the PP-3 composite to the Blue Cost optimized flowsheet through seven locked-cycle tests (Middleditch, 2014)
- Flowsheet fine-tuning on SDCs: The variability “sub-domain composites” were tested in batch mode to fine-tune the flowsheet for locked-cycle testing on each composite (Colebrook, 2014)
- Locked-cycle confirmation testing on SDCs: This program created a picture of how the fine-tuned flowsheet would respond to different material types, sampled from different parts of the deposit (Colebrook, 2014)
- Locked-cycle confirmation testing on LOMs: This suite of tests was aimed at providing some insight into the locked-cycle response on sample loosely designed to represent different periods in the mine life (Colebrook, 2014)
- Locked-cycle confirmation of the pyrrhotite rejection flowsheet: This small program included just a few locked-cycle tests, and was designed to better understand the pyrrhotite rejection flowsheet and its associated metallurgy (Middleditch, 2014).

13.7.1 Bench Scale Flowsheet Development at ALS

Early testwork was conducted at ALS and was relatively limited in scope. It followed a broad procedure widely used in the nickel industry, that of producing a selective copper concentrate floated from the nickel and iron sulfides using small doses of a selective collector and/or nickel depressants, then nickel and iron sulfide bulk flotation using higher doses of xanthate collectors.

The test program led to the flowsheet described below, variants of which were used for all locked-cycle testing:

- The primary grind ranged from 140–150 µm, with a combination of 100 g/t sodium sulfite (Na_2SO_3) and 25 g/t TETA, together with lime added to the mill to achieve partial depression of the pentlandite in the ensuing copper roughers
- Cytec 3418A was used as a collector with doses in locked-cycle testing varying from 5 to 25 g/t, driven by an aim to maintain good copper recoveries to the rougher concentrate
- The rougher concentrate was reground to 23–39 µm, and cleaned three times to make the copper final concentrate. Up to 100 g/t sodium sulfite and 50 g/t TETA was used in the copper regrind, and a further 3 g/t 3418A used in copper cleaner flotation.

The copper cleaner tails were added to the nickel feed, and the nickel circuit then consisted of a rougher, concentrate regrind and three stages of cleaning:

- Nickel roughing was conducted at pH 9–10 using 100 g/t potassium amyl xanthate (PAX) to promote bulk sulfide flotation at the maximum possible recovery.
- The concentrate was subjected to a fine regrind, to between 24–40 µm and then cleaned at pH 8.5–9.2 using 210g/t PAX.

Two locked-cycle tests were completed at ALS using the sequential flotation flowsheet on the PP-3 composite. The results are provided in Table 13-12.

Test 69 achieved a clean copper concentrate (25% Cu) albeit at a relatively low recovery of 73%. The nickel circuit produced a bulk product assaying 8.7% nickel and 6.4% copper. The overall copper recovery was 88%, with 60% of the nickel reporting to the nickel concentrate.

Table 13-12: Locked-cycle Test Results from Testwork at ALS on PP-3 Composite

Test Number	Sample	Copper Concentrate				Nickel Concentrate			
		Grade		Recovery		Grade		Recovery	
		% Cu	% Ni	% Cu	% Ni	% Cu	% Ni	% Cu	% Ni
69	PP Composite 3	25	0.40	73	3	6.4	8.7	15	60
81	PP Composite 3	23	0.63	87	6	3.3	9.0	8	56

Test 81 employed modified conditions to force more copper to the copper concentrate. This yielded a copper concentrate assaying 23% copper and 0.63% nickel. The resulting nickel concentrate was better quality at 9% nickel and 3.3% copper. The reader should note that these tests were run entirely with fresh water, thus no issues were noted with respect to water circulation in the pilot plant.

The flowsheet as developed in the laboratory at ALS was tested on Maturi end-member samples in locked-cycle mode. Figure 13-13 shows the results of the locked-cycle tests using the ALS flowsheet, whereas Figure 13-14 indicates the nickel locked-cycle test results. The copper concentrate would likely be marketable but at, on average, the result of 22% copper left upside in grade, while the nickel concentrate was unlikely to be marketable. This prompted the follow-up studies at Blue Coast, discussed in the next subsection of this Report. Note that all the tests at ALS employed fresh water throughout, which the QP understands is standard protocol at the Kamloops laboratory.

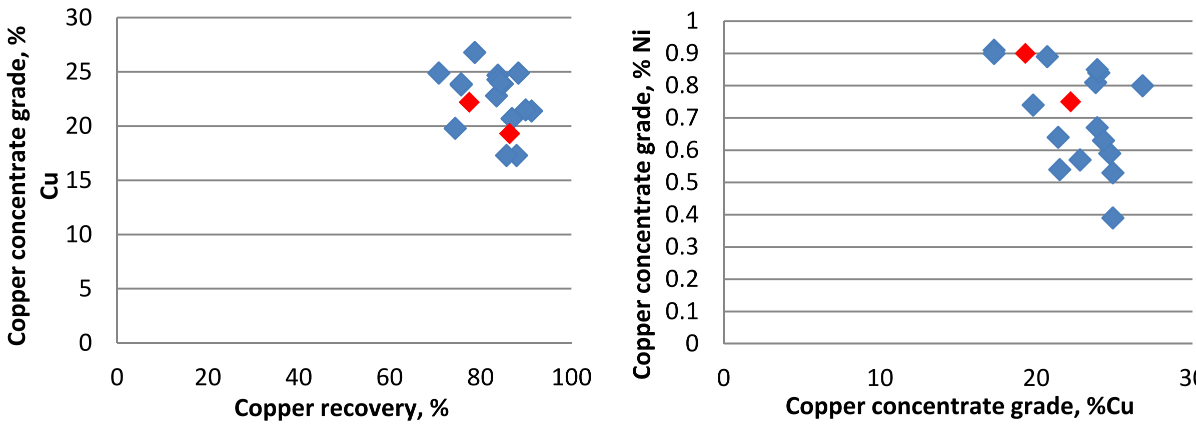
13.7.2 Flowsheet Development at Blue Coast

Subsequent testwork at Blue Coast aimed to build on the foundation of what was learned at ALS in order to arrive at an improved flowsheet by following a systematic approach to optimization of each stage of the process. The sequence of testing followed the optimization of the copper rougher and cleaners, then nickel roughing and finally nickel cleaning.

The direction was towards lower doses of shorter chain xanthates and lower doses of depressants, regrinding with inert grinding media and flotation at generally higher pH levels. The following conditions were established as optimal through this test program:

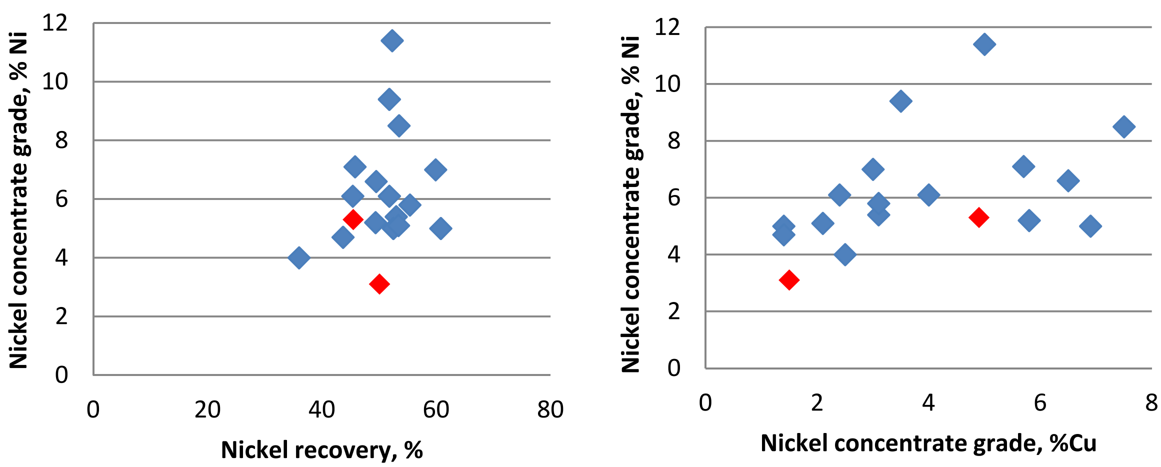
- Copper/nickel selectivity was found to be pH dependent, and the rougher pH was optimized and pegged at a pH of 10.8
- The depressant doses to the primary grind were dropped to typically 35 g/t each of TETA and Na_2SO_3, with 25 g/t of each of the depressants being added to the copper regrind mill. The primary grind size P_{80} was established at 120 µm. In reality the depressant dose varied with each sample tested and needed some degree of optimization in each case

Figure 13-13:Locked-cycle Copper Circuit Metallurgy using ALS Flowsheet



Note: Figure prepared by Blue Coast, 2014. Red diamonds indicate Maturi Southwest samples.

Figure 13-14: Locked-cycle Nickel Circuit Metallurgy using ALS Flowsheet



Note: Figure prepared by Blue Coast, 2014. Red diamonds indicate Maturi Southwest samples.

- The Cytec phosphine collector 3418A remained the collector of choice in copper flotation, its dose being pegged at 5 g/t. Overdosing led to excessive nickel flotation making Cu/Ni separation more challenging in cleaning
- Copper rougher flotation was completed in three minutes
- Copper concentrate regrinding was conducted to a target P_{80} size of 40 µm and cleaning was conducted in three stages at pH11, with the residence time of each cleaner kept at approximately two minutes. The 3418A collector dose was kept at close to starvation levels of 1 g/t of primary mill feed.

The careful use of reagents in the copper circuit had a major spinoff effect on the nickel float. Specifically, the use of less depressants in the copper circuit with less collector kept nickel flotation under control in the copper circuit while rendering the pentlandite more floatable in nickel flotation, with attendant higher overall nickel recoveries. The nickel circuit was not as completely optimized as the copper circuit in the Blue Coast study, and further upside remains:

- Nickel rougher flotation was conducted at pH 10 with small doses of lime to maintain this level
- The copper first cleaner tails was directed to the nickel first cleaner to avoid nickel regrinding
- The collector dosage was cut back substantially, from 300 g/t of amyl xanthate in the ALS flowsheet to 130 g/t of the shorter chain isopropyl xanthate in the Blue Coast flowsheet.

Brief test programs were executed to optimize the nickel cleaner circuit, and overall nickel grades and recoveries were both substantially better in the Blue Coast program than the ALS program, but excessive non-sulfide gangue was still recovered to nickel concentrates, and there remains potential to further increase nickel concentrate grade.

Only two locked cycle tests were completed on Maturi Southwest material. Due to this low number of tests, the level of certainty of Maturi Southwest metallurgical performance is lower than for the Maturi deposit. Copper concentrate grade and recovery for Maturi Southwest seemed to lie in a similar range to Maturi (refer to Figure 13-13). The Ni grade in the copper concentrate was in the upper range of values from the Maturi tests, suggesting that some efforts might be needed to improved nickel rejection from the copper concentrate. The nickel grade–recovery performances for Maturi Southwest samples were lower than the average than the average Maturi results, particularly for the Maturi Southwest S2 sample tested (refer to Figure 13-14). The S2 test produced a low nickel concentrate grade (3.2%), and the concentrate was also low in copper (1.5%). Test results showed elevated levels of non-sulfide gangue in this concentrate compared with others tested. The results suggest that more conservative nickel grade and recovery targets should be assumed for Maturi Southwest compared with Maturi. Further testwork is required to fully evaluate and optimize the nickel separation characteristics of Maturi Southwest.

13.7.3 Pyrrhotite Rejection Flowsheet Development:

The baseline flowsheet has been developed to float all the sulfides, with the majority of the copper sulfides floated to the copper concentrate, and the nickel and iron sulfides floated to the nickel concentrate. Especially in the treatment of S2 materials, this practice of floating the pyrrhotite to nickel concentrate can, however, reduce nickel concentrate grade.

A program of tests exploring the rejection of pyrrhotite was conducted. Key findings are summarized as follows:

- The sample used a composite specifically designed with a high content of pyrrhotite and termed the “Po rejection composite”, that assayed 0.55% Cu and 0.17% Ni. The ratio of pyrrhotite to pentlandite was 2.4:1—typical of much of the S2 material scheduled to be mined later in the mine life. The sample had a relatively high ratio of abundance of chalcopyrite to cubanite
- Magnetic separation as a means of removing the pyrrhotite was eliminated early in the TMM studies. Testwork at ALS and probe data provided by Cabri had shown that the pyrrhotite was non-magnetic
- Therefore, flotation was pursued as the primary means of rejecting pyrrhotite.

Elsewhere in the nickel industry, much has been done to develop a pentlandite/pyrrhotite separation process. Where magnetic separation is not effective, the industry has focused on the use of sodium sulfite and TETA to selectively reject pyrrhotite. As these reagents are already being used in the copper/nickel separation circuit, their use for pyrrhotite rejection is convenient. The following modification to the nickel cleaner flowsheet was developed:

- Sodium sulfite and TETA were added to the nickel regrind at a dose ratio optimized at 3:1—typically 45–75 g/t sodium sulfite and 15–25 g/t TETA.

Five locked-cycle tests were conducted using the flowsheet.

13.7.4 Variability Bench-Scale Studies

The variability studies used to characterize the base case metallurgical response for the prefeasibility study involved rougher kinetics testwork and locked-cycle testwork. The former were used to define how the rougher recovery varied across the deposit; the latter were used to establish how the metal, once recovered to a rougher concentrate, responded to cleaning in closed circuit mode.

13.7.5 Rougher Testwork

Some 94 rougher tests were conducted in all on S2 and S3 samples at ALS and Blue Coast using the same flowsheet.

13.7.5.1 S2

The average S2 copper rougher recovery was 96%, with a standard deviation of 1.1%, the average S2 nickel recovery was 70% with a standard deviation of 9%. On average 89% of the copper floated to the copper rougher concentrate, together with 41% of the nickel.

The distribution of copper and nickel recoveries, both spatially within the resource and profiled by depth (or distance from the top of S3) is illustrated in Figure 13-15. Where more than one sample has been tested per hole, the number of tests is illustrated in superscript. There is little spatial trend evident from the data, although there may be zones within the heart of both Maturi West and Maturi East that floated both copper and nickel particularly well.

For any distance from the top of S3 greater than 100 ft, both copper and nickel recoveries followed no clear trend; however, where the deposit pinches and S2 is less than 100 ft from the top of S3, both metal recoveries drop.

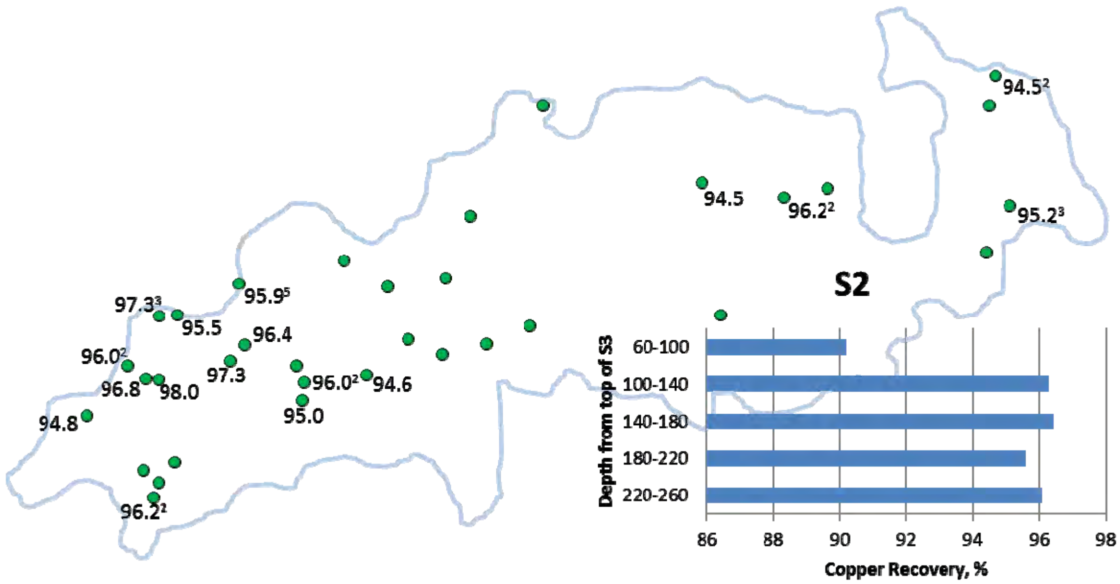
13.7.5.2 S3

The average copper recovery to the combined concentrates was 96.5%, with a standard deviation of 1.3%, and nickel recovery was 73.3% with a standard deviation of 9%. The distribution of copper and nickel recoveries, both spatially within the resource and profiled by depth (or distance from the top of S3) is illustrated in Figure 13-16. On average 92% of the copper floated to the S3 copper rougher concentrate, together with 55% of the nickel. Overall, copper rougher selectivity against nickel was somewhat poorer for S3, resulting in higher nickel levels in copper concentrate than for S2.

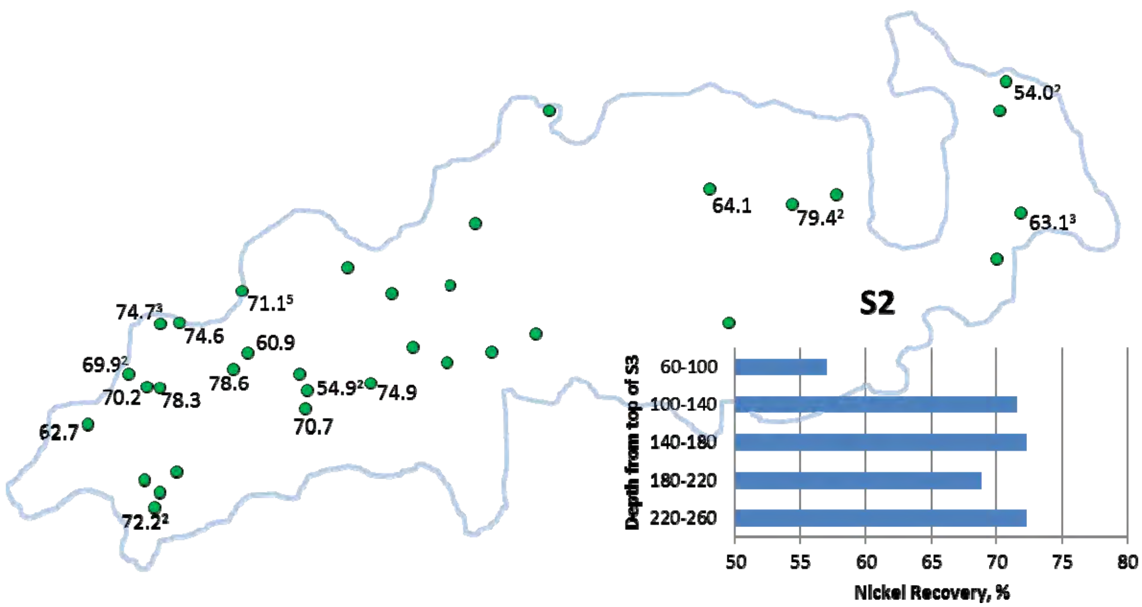
As with S2, there is a weak trend towards better copper and especially nickel recoveries in portion of Maturi termed Maturi West by the metallurgists. Copper recoveries show no trends with depth, while nickel recoveries tend to rise with depth from the top of S3.

Figure 13-15: Distribution in Copper and Nickel Rougher Recoveries

(a) Distribution in Copper Rougher Recoveries, Maturi S2 Variability Samples



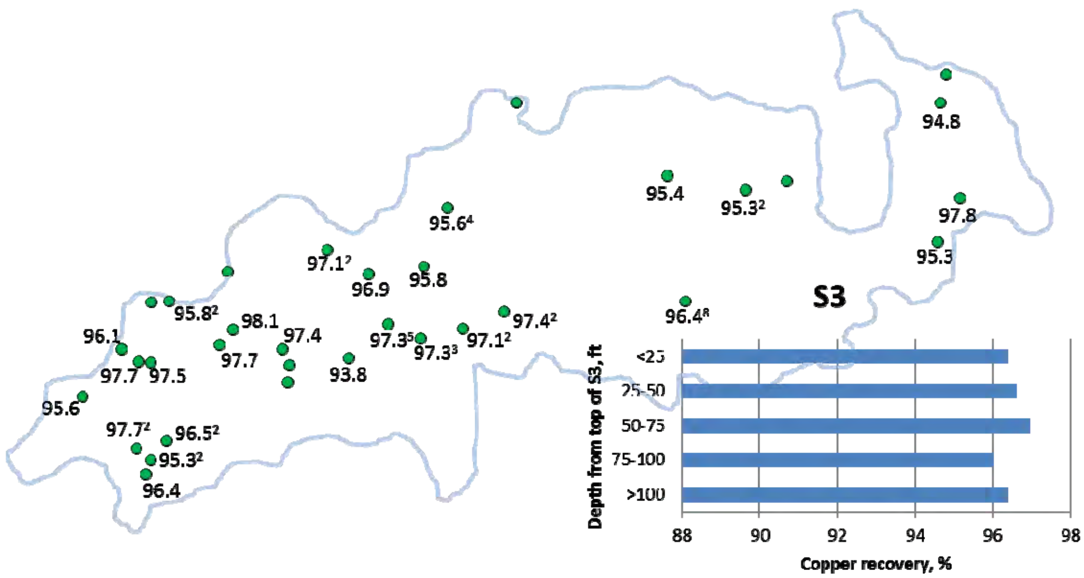
(b) Distribution of Nickel Rougher Recoveries, Maturi S2 Variability Samples



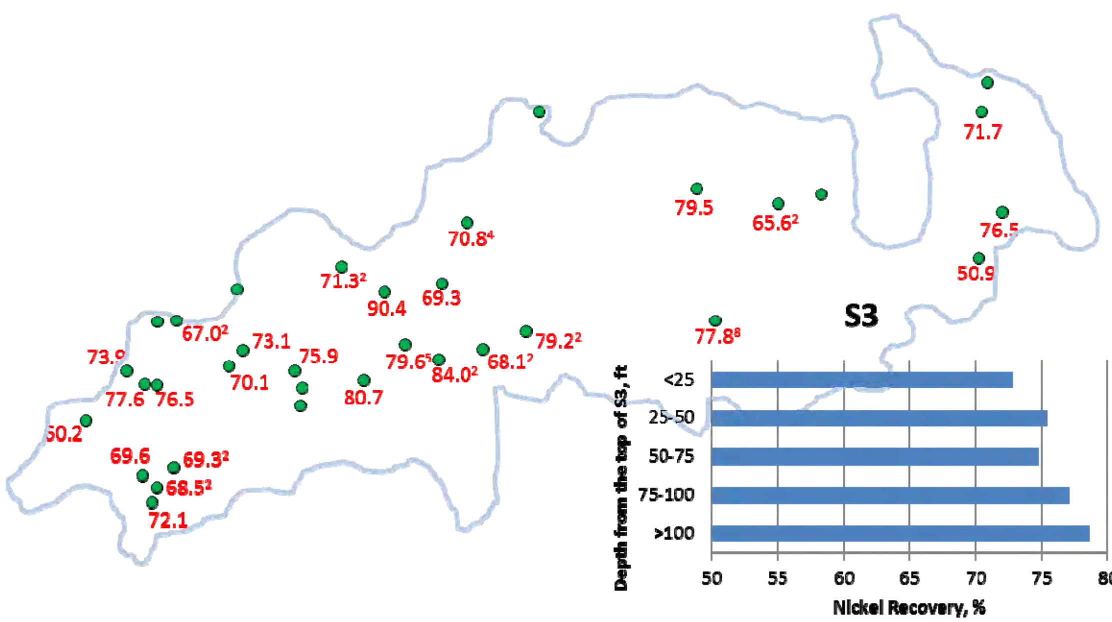
Note: Figure prepared by Blue Coast, 2014.

Figure 13-16:Distribution in Copper and Nickel Rougher Recoveries.

(a) Distribution in Copper Rougher Recoveries, Maturi S3 Variability Samples



(b) Distribution in Nickel Rougher Recoveries, Maturi S3 Variability Samples



Note: Figure prepared by Blue Coast, 2014.

13.7.6 Locked-cycle Testing

A variability program of locked-cycle tests using the base case flowsheet was conducted on the SDC. While strictly the variability program is limited to the SDC, for the sake of completeness the entire locked-cycle dataset is described here.

13.7.6.1 Flowsheet and Conditions used in Locked-cycle Testing

The flowsheet shown in Figure 13-17 was used for all tests, except that reagent addition points were altered for the pyrrhotite rejection test.

The primary grind size was 120 µm for all tests, and the regrind sizes were estimated at 40 µm. Process water was circulated in each of the circuits and cleaner collector and frother doses were usually dialed back in later cycles due to recirculation of flotation reagents.

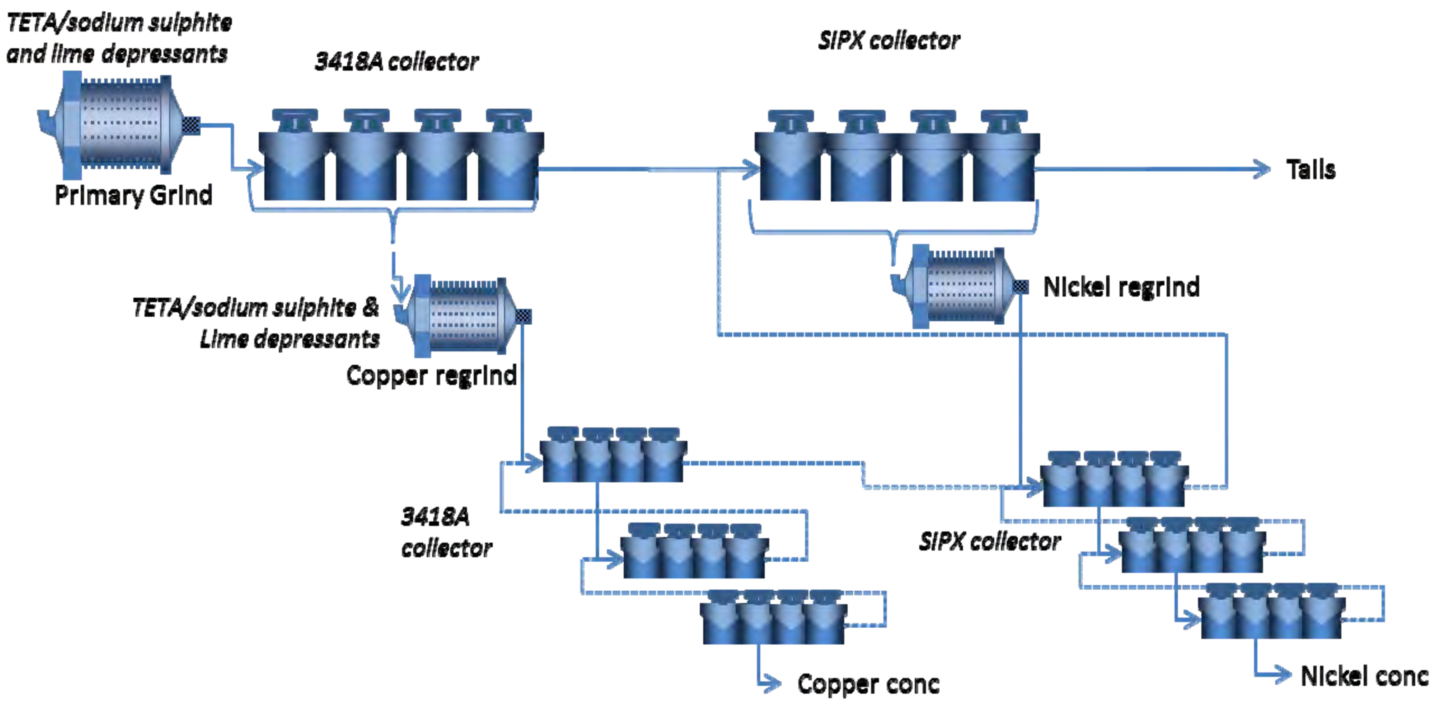
Some 19 locked-cycle tests were run using the base case flowsheet and 6 tests on the pyrrhotite rejection flowsheet, using optimal or near-optimal conditions. The key data from these tests are summarized in Table 13-13. In addition, five tests employed the pyrrhotite rejection circuit and performances are presented in Table 13-14.

On average, the locked-cycle tests (base case and pyrrhotite rejection flowsheets) yielded a copper concentrate assaying 25.1% copper and 0.75% nickel, at a copper recovery of 85%. The recovery of nickel to the nickel concentrate was 56.4%, to a concentrate that assayed 9.1% nickel. This concentrate also contained 3.8% copper, with the additional 9.1% copper recovery bringing the total copper recovery to 94%.

Table 13-15 shows the degree of improvement made in the performance of the flowsheet between the ALS and Blue Coast programs. Although the LCTs were all not performed on the same samples, the data populations (10 ALS tests and 25 Blue Coast tests) are sufficient, and the range of samples tested broad enough in both cases to conclude that the changes are indeed due to process improvements. Substantial improvements were made in both the copper and nickel circuits.

The recoveries of gold, platinum and palladium from the Blue Coast locked-cycle tests, where PGE balances are available, are shown in Figure 13-18. Mean concentrate gold, platinum and palladium grades and associated recoveries are shown in Table 13-16. The mean concentrate minor element analyses from the locked-cycle test concentrates are shown in Table 13-17 for the copper concentrates and Table 13-18 for the nickel concentrates.

Figure 13-17: General Flowsheet for Baseline Locked-cycle Flotation Tests



Note: Figure prepared by Blue Coast, 2014.

Table 13-13: Summarized Results from Locked-cycle Testing of Various TMM Samples – Base Case Tests

Base Case Composite/Test	Copper Cleaner Concentrate				Ni Cleaner Concentrate				Ni:Cu Grade
	Cu Rec (%)	Cu Grade (%)	Ni Rec (%)	Ni Grade (%)	Cu Rec (%)	Cu Grade (%)	Ni Rec (%)	Ni Grade (%)	
S2									
D-S2-H	81.7	25.3	3.9	0.47	9.5	3.4	46.9	6.6	1.9
D-S2-L/M	83.0	25.9	6.8	0.79	9.5	4.2	55.6	9.2	2.2
DE-S2-M	85.1	26.1	8.5	0.73	7.4	4.5	42.9	7.3	1.6
Average	*82.9*	*25.1*	*7.1*	*0.68*	*9.7*	*4.7*	*50.9*	*8.0*	*1.8*
S3									
S-S3-H	86.2	25.7	10.1	1.01	6.8	4.1	52.6	10.7	2.6
S-S3-M	87.5	24.3	6.1	0.65	6.7	2.9	56.3	9.0	3.1
D-S3-H	86.8	25.5	6.1	0.64	7.6	3.3	62.1	9.7	2.9
D-S3-M	80.7	25.1	6.9	0.86	10.9	4.4	54.8	8.9	2
DE-S3-H	83.8	24.8	7.5	0.78	8.7	3.7	58.7	8.7	2.3
DE-S3-M	83.6	25.9	9.3	0.95	8.7	3.7	58.7	8.1	2.2
DE-S3-L	85.3	25.5	7.7	0.66	7.7	3.7	60.1	8.3	2.2
Average	*84.8*	*25.3*	*7.7*	*0.79*	*8.2*	*3.7*	*57.6*	*9.1*	*2.5*
PP-3									
LCT-1	84.6	25.3	7.7	0.83	n/a	n/a	n/a	n/a	n/a
LCT-2	86.1	24.6	8.9	0.9	8.4	3.0	62.7	7.9	2.6
LCT-3	86.3	24.8	7.5	0.77	7.8	3.2	64.3	9.4	2.9
LCT-4	84.2	25.3	7.0	0.74	10.4	4.2	62.3	8.8	2.1
LCT-7	84.1	24.7	6.8	0.7	9.9	4.2	61.4	9.1	2.2
Average	*84.8*	*25*	*7.3*	*0.76*	*9.3*	*3.9*	*62.7*	*9.1*	*2.4*
Life of Mine									
SCT-LOM (baseline)	87.6	24.8	8.5	0.87	6.3	2.6	57.8	8.7	3.3
0-3 years	85.9	24.6	6.1	0.56	7.7	3.6	55.5	8.4	2.3
4-8 years	87.2	24.0	7.2	0.67	6.9	2.8	57.6	7.9	2.8
9-19 years	86.3	25.5	7.5	0.72	6.8	2.9	57.4	7.9	2.7
Mean LCT performance	**85.1**	**25.1**	**7.4**	**0.75**	**8.2**	**3.8**	**57.1**	**8.6**	**2.4**

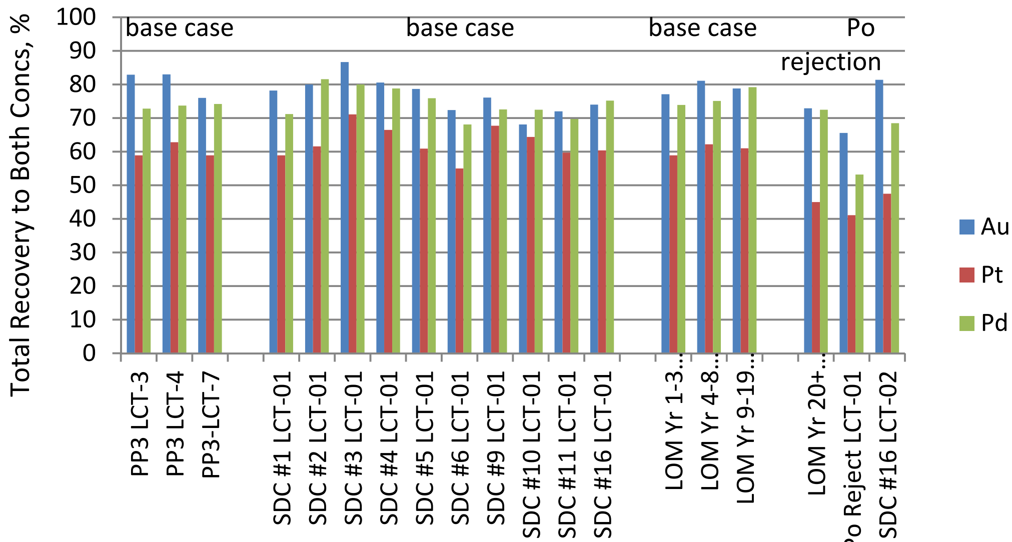
Table 13-14: Summarized Results from Locked-cycle Testing of Various TMM Samples – Pyrrhotite Reject Testing

Pyrrhotite Reject Comp/Test	Copper Cleaner Concentrate				Ni Cleaner Concentrate				Ni:Cu Grade
	Cu Rec (%)	Cu Grade (%)	Ni Rec (%)	Ni Grade (%)	Cu Rec (%)	Cu Grade (%)	Ni Rec (%)	Ni Grade (%)	
SDC - S-S2-L/M	78.8	22.3	7.3	0.66	12.2	6.7	52.5	9.2	1.4
SDC - D-S2-L/M	81.9	25.0	7.1	0.86	10	4.7	53.4	10.1	2.1
Po Rejection	87.6	28.1	6.3	0.68	7.3	4.1	60.7	11.5	2.8
LOM - 20-32 years	87.9	24.6	6.6	0.68	4.8	3.3	51.2	12.9	3.9
SCT - LOM	86.2	25.1	7.0	0.73	6.7	3.5	54.9	10.3	2.9
PP-3 - LCT-8	85.2	23.6	9.3	0.90	8.2	4.3	54.2	9.8	2.3
Mean LCT performance	**84.6**	**24.8**	**7.3**	**0.75**	**8.2**	**4.4**	**54.5**	**10.6**	**2.6**

Table 13-15: Mean LCT Results from the Blue Coast and ALS Locked-cycle Programs

Test program	Copper Cleaner Concentrate Cu Rec (%)	Cu Grade (%)	Ni Rec (%)	Ni Grade (%)	Ni Cleaner Concentrate Cu Rec (%)	Cu Grade (%)	Ni Rec (%)	Ni Grade (%)	Ni:Cu Grade
BCR	84.9	25.1	7.3	0.75	8.2	3.9	56.4	9.1	2.5
ALS	83.2	22.5	7.8	0.72	6.5	3.9	51.0	6.5	1.7

Figure 13-18:PGE Recoveries to Locked-Cycle Test Concentrates



Note: Figure prepared by Blue Coast, 2014.

Table 13-16:Mean Concentrate Au, Pt and Pd Grades and Recoveries

	PGE Recovery, % Au	Pt	Pd	PGE Grade, g/t Au	Pt	Pd
Base case circuit						
Copper conc	64.1	23.4	36.7	2.50	1.49	5.50
Nickel conc	14.0	38.0	37.7	0.78	3.75	8.70
Pyrrhotite rejection circuit						
Copper conc	59.8	22.9	39.4	2.24	1.34	4.84
Nickel conc	13.6	21.7	25.3	1.68	3.32	7.68

Table 13-17: Copper Concentrate Minor Element Analyses

	Cu %	Ni %	Au g/t	Ag g/t	Pt g/t	Pd g/t	Co ppm	Fe %	S %	Sb ppm	As ppm	Bi ppm	Cl ppm
Mean	25.0	0.72	2.6	54	1.6	5.8	174	33.0	30.4	2.0	7.9	35.1	286
20th percentile	24.5	0.65	2.0	50	1.1	4.3	143	31.8	29.0	1.7	5.6	4.0	200
80th percentile	25.7	0.80	2.9	58	2.0	7.1	225	34.2	31.8	2.3	9.6	97.0	340
	F ppm	**Pb ppm**	**Zn ppm**	**Hg ppb**	**Se ppm**	**Te ppm**	**Cd ppm**	**Mn ppm**	**Mo ppm**	**SiO_2 %**	**Al_2O_3 %**	**CaO %**	**MgO %**
Mean	<100	54.7	544	55	97	17	8.0	138	4.8	4.4	1.5	1.0	1.2
20th percentile	<100	43.6	458	36	85	16	6.4	112	2.6	3.2	1.2	0.8	0.7
80th percentile	<100	65.0	603	74	109	18	9.6	156	5.8	5.5	1.8	1.2	1.6

Table 13-18: Nickel Concentrate Minor Element Analyses

	Cu %	Ni %	Au g/t	Ag g/t	Pt g/t	Pd g/t	Co ppm	Fe %	S %	Sb ppm	As ppm	Bi ppm	Cl ppm
Mean	3.9	8.8	0.7	23.4	4.1	9.4	2083	30.8	21.1	10.1	24.1	6.5	264
20th percentile	3.3	7.9	0.6	19.8	3.1	7.3	1626	29.3	20.1	2.3	8.6	3.0	200
80th percentile	4.2	9.3	0.8	28.2	5.1	12.3	2346	32.4	22.3	15.4	17.4	11.0	340
	F ppm	**Pb ppm**	**Zn ppm**	**Hg ppb**	**Se ppm**	**Te ppm**	**Cd ppm**	**Mn ppm**	**Mo ppm**	**SiO_2 %**	**Al_2O_3 %**	**CaO %**	**MgO %**
Mean	<100	81.5	458	28.0	75.8	8.0	2.2	428	6.3	17.9	6.2	1.0	3.9
20th percentile	<100	60.6	258	15.2	73.0	5.6	1.5	387	5.0	16.0	5.6	0.8	3.0
80th percentile	<100	87.4	518	36.8	79.0	10.2	2.9	504	8.0	19.6	7.0	1.2	4.8

A feature of all the testwork at ALS and Blue Coast was the limited degree of water recycling in the locked-cycle tests. The ALS tests did not use recycling of water, while Blue Coast tests employed separate recycling of copper and nickel cleaner circuit waters (effectively simulating the in-circuit thickener in the ALS pilot plant).

The problem of xanthate recycle through use of a single process water reticulation system to the copper circuit was evaluated extensively at ALS and never solved. The move, however, to shorter chain xanthates, and the substantial drop in xanthate dose at Blue Coast may suggest that these waters may be more easily recycled back to the copper circuit (as is practiced at reference sites such as Kevitsa). This needs to be studied further in the next phase of work as it potentially impacts the design of the plant (which for the PFS did not include any in-circuit thickening).

13.7.7 Pilot Plant Testing

A total of 28 pilot plant runs were conducted using the sequential copper/nickel flotation flowsheet developed at ALS. The ALS flowsheet is the only flowsheet tested by pilot plant, and that the upside associated with the Blue Coast flowsheet previously described is not reflected in these pilot plant results.

The pilot plant was typically run at 200 kg per hour throughout the program; the runs being on average 7.2 hours long. Runs 1–14 employed a bulk flotation flowsheet which was later discarded for the purposes of the prefeasibility work. The early stages of the sequential test program were devoted to addressing problems of collector-rich water circulating from the collection-intensive nickel flotation circuit to the copper circuit where starvation doses were needed to effect good separation from the nickel. Re-circulating the water yielded copper concentrates with 2–4% nickel. Some 25–30% of the nickel was found to be floating to the copper final concentrate. This was not metallurgically acceptable, and required modifications to the water system.

Four follow-up runs were completed using the same treatment scheme but were completed entirely on fresh water, to demonstrate the viability of the basic process with the factor of water quality eliminated. Nickel misplacement to the copper concentrate dropped to 2–4% and nickel grades in the copper concentrate to 0.4–0.5%. Copper recoveries to copper concentrates were in the range of 69–77% with a further 16–23% of the copper reporting to the subsequent nickel concentrate (combined copper recoveries were 92–94%). Nickel recovery to nickel concentrate was 66–67%. While this demonstrated the process could be effective in continuous mode, the use of fresh water is not a practical option for full-scale plant operations.

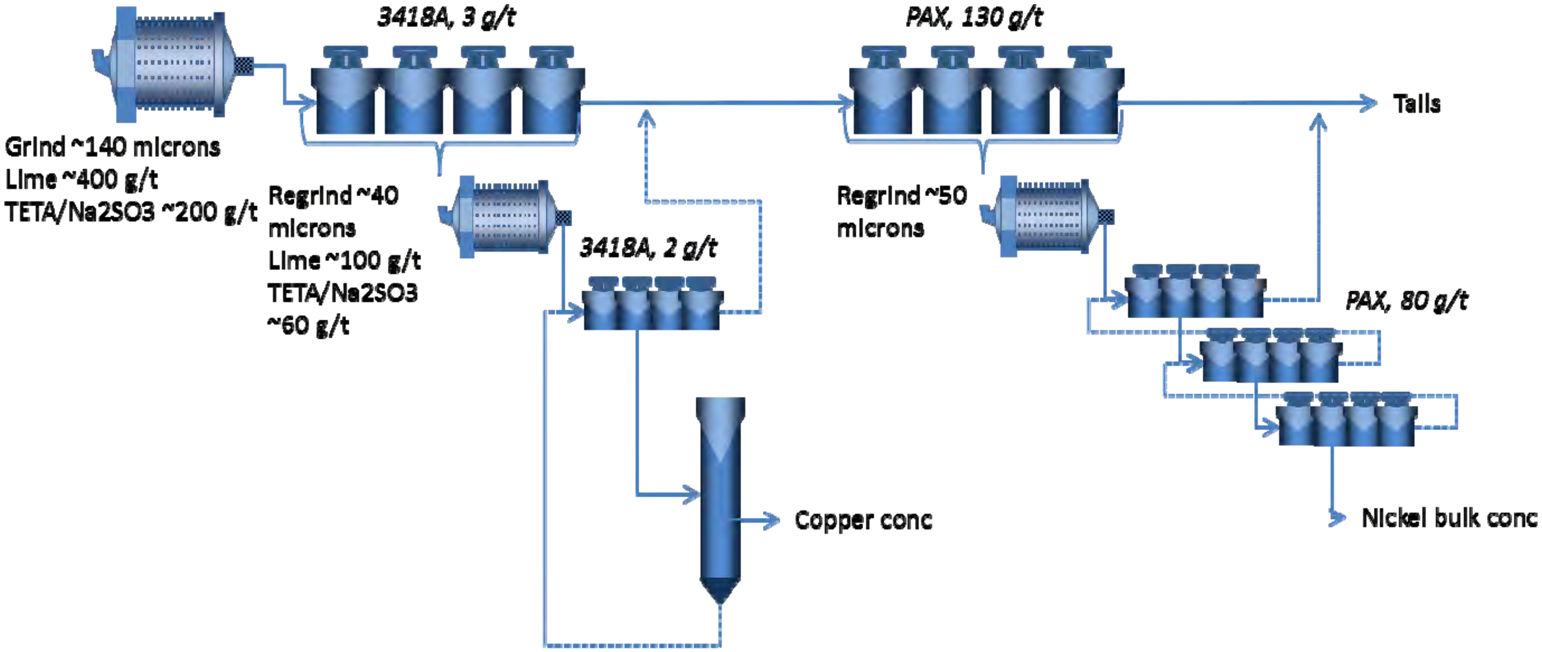
A thickener was installed in the circuit from run 19 onwards, located between the copper and nickel bulk circuits. This thickener allowed for the collector-poor water from the copper circuit to be circulated back to the primary grind while, more importantly, allowing collector-rich water from the nickel bulk tails to be circulated back to the nickel bulk rougher feed, thereby avoiding the collector-sensitive copper circuit. The flowsheet used in runs 1–18 is shown in Figure 13-19, and the modified flowsheet incorporating the thickener between circuits is shown in Figure 13-20.

Five runs were employed to establish the appropriate reagent scheme using this flowsheet, with the following nine runs operated using variations of essentially optimal conditions (using the ALS flowsheet).

Runs P27 and P28 were run with a greater focus on nickel recovery, and yielded concentrate nickel grades of 8.1–8.8% nickel, at recoveries of 64–65%. The ensuing runs, P29 to P35 focused on achieving the target Ni grade of over 10% by slowing down the nickel float and forcing better gangue rejection in the froth.

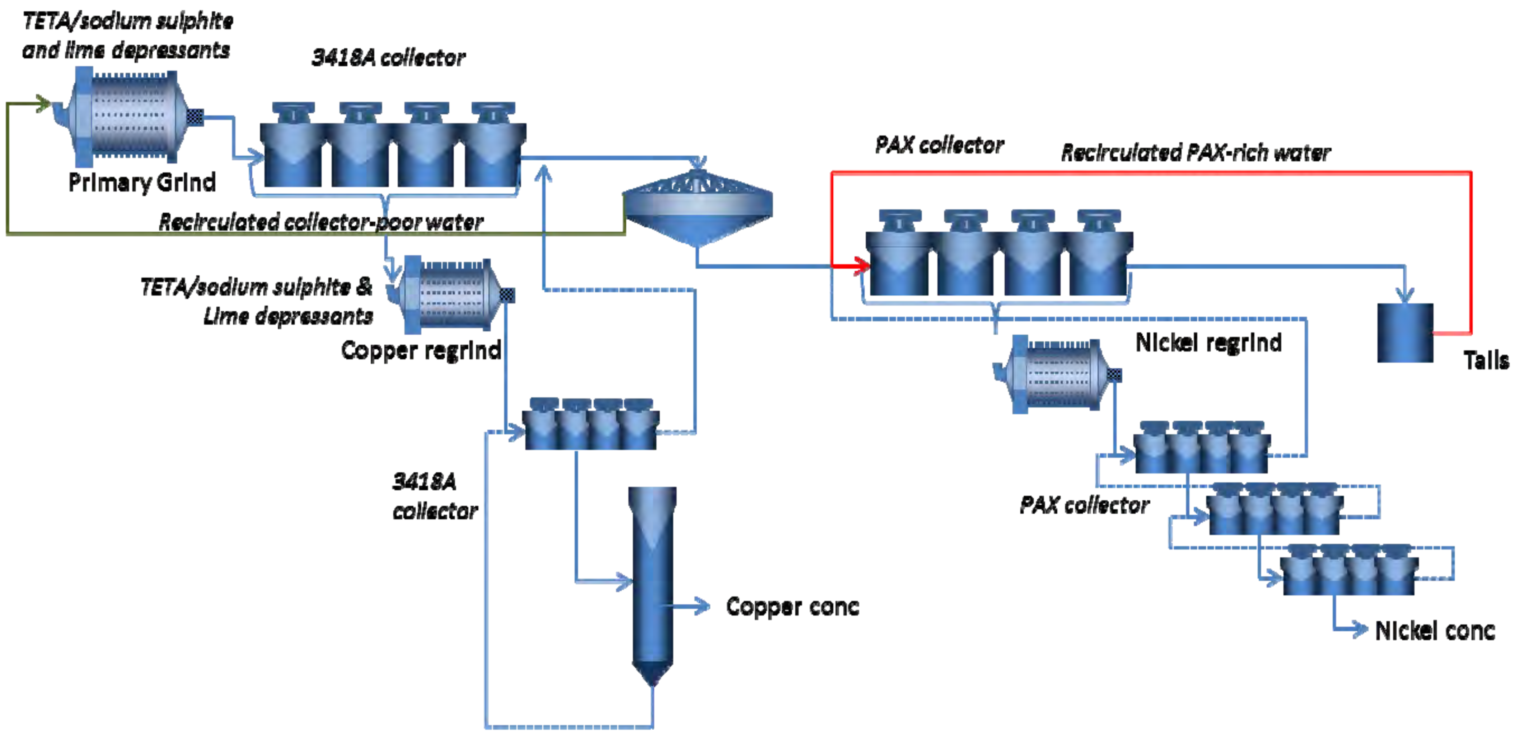
Seven of the pilot plant runs were operated under what would be considered optimal conditions. These runs were: P27, P28, P29, P30, P32, P33 and P34. These runs yielded a mean copper recovery of 83% and a mean nickel recovery of 5% to the copper concentrate. The copper concentrate assayed 25.5% copper and 0.6% nickel.

Figure 13-19: Flowsheet Employed in Early Testing of the Sequential Cu/Ni Flowsheet



Note: Figure prepared by Blue Coast, 2014.

Figure 13-20: Sequential Copper/Nickel Flotation Circuit with In-Circuit Thickener



Note: Figure prepared by Blue Coast, 2014.

The gold, platinum and palladium recoveries to the copper concentrate were 68%, 22% and 43% to grade 2.4 g/t, 1.3 g/t and 6.2 g/t respectively.

The nickel circuit yielded concentrates assaying 11.1% nickel and 4.4% copper, at nickel and copper recoveries of 60% and 10% respectively. The gold, platinum and palladium recoveries were 12%, 36% and 33% to grade 0.7 g/t, 3.4 g/t and 7.5 g/t respectively.

Accordingly, the total metal recoveries were 93% for copper, 65% for nickel (of which 60% would generate revenue), and 80%, 58% and 76% for gold, platinum and palladium.

The key metallurgical performance statistics from the runs are shown in Table 13-19 (copper and nickel) and Table 13-20 (PGEs). They reflect much better performance than the equivalent ALS locked-cycle tests, with a more nickel-free copper concentrate produced by the pilot plant column, and far better nickel grades and recoveries to the nickel concentrate. Should the same scale-up effects be observed when the Blue Coast flowsheet is piloted, further improvements in nickel metallurgy over those seen in either the Blue Coast locked-cycle program or the ALS pilot plant program can be expected. No upside potential was included in the metallurgical forecast used in the PFS.

13.8 Comments on Section 13

All metallurgical work described in Section 13 has been conducted using standard industry methods at reputable testing laboratories. All metallurgical assays have been conducted using quality control systems that are consistent with normal industry practice.

The results described in Section 13 are representative of the current level of understanding of the likely metallurgical response of Twin Metals mineralization. The process as developed is a conventional process quite typical of what has been successfully implemented for several major sulfide copper/nickel projects in operation worldwide.

The reader should be aware that replication of the final flowsheet and final flowsheet results still needs to be done and should be done in the next phase of testing. Specifically, the need and operation of in-circuit thickening needs to be firmed-up with the optimized flowsheet, while the pyrrhotite rejection process needs further development to enhance nickel recoveries, and pilot plant confirmation testing. The other major source of metallurgical process risk lies in the processing of Maturi South west mineralization, as this is very poorly understood at the present time.

Table 13-19: Pilot Plant Copper and Nickel Metallurgy

	Copper Final Concentrate				Nickel Final Concentrate			
	Cu Recovery (%)	Cu Grade (%)	Ni Misplaced (%)	Ni Grade (%)	Cu Recovery (%)	Cu Grade (%)	Ni Recovery (%)	Ni Grade (%)
P27	80.1	24.6	4.7	0.56	13.8	4.5	65.5	8.1
P28	84.7	24.2	6.7	0.74	9.7	3.5	63.7	8.8
P29	84.5	24.0	6.0	0.63	8.6	4.4	58.2	10.9
P30	80.4	25.1	4.9	0.58	13.7	8.0	57.7	12.6
P32	85.8	26.1	5.1	0.60	7.3	3.7	59.2	11.6
P33	80.9	25.7	4.4	0.51	13.5	6.0	58.8	9.6
P34	83.4	26.4	4.9	0.62	8.9	3.7	64.2	10.7
Average	*82.8*	*25.2*	*5.3*	*0.61*	*10.8*	*4.8*	*61.0*	*10.3*

Table 13-20: Pilot Plant Gold, Platinum and Palladium Metallurgy

	Copper Final Concentrate						Nickel Final Concentrate					
	Grades (g/t)			Recoveries (%)			Grades (g/t)			Recoveries (%)		
	Au	Pt	Pd	Au	Pt	Pd	Au	Pt	Pd	Au	Pt	Pd
P27	2.83	1.45	6.65	79.9	24.0	47.4	0.37	2.27	4.90	10.0	35.7	33.2
P28	2.84	1.42	6.66	72.5	27.1	51.5	0.46	2.55	4.76	9.4	39.0	29.4
P29	2.58	1.33	7.03	75.6	26.6	52.6	0.54	3.25	6.50	8.8	36.1	27.1
P30	1.70	1.26	6.03	56.3	22.4	43.8	0.59	3.73	8.58	10.6	35.5	33.3
P32	2.43	1.35	6.69	79.0	22.2	44.3	0.63	2.57	6.73	12.2	25.4	26.7
P33	3.34	1.28	6.34	75.5	20.9	41.3	0.93	3.41	7.64	14.9	39.6	35.4
P34	2.17	1.12	4.91	54.7	16.1	34.2	0.82	4.05	8.24	15.7	44.6	43.7
Average	*2.56*	*1.32*	*6.33*	*70.5*	*22.8*	*45.0*	*0.62*	*3.12*	*6.76*	*11.7*	*36.5*	*32.7*

Metallurgical upside exists from (1) improving the rejection of non-sulfide gangue from the nickel concentrate and (2) piloting the fine-tuned baseline and pyrrhotite rejection processes, as experience to date suggests that piloting Maturi material leads to significantly better metallurgy than testing the same material in the laboratory. A third area of potentially significant upside lies in enhancing the payability of the precious metals in the concentrate.

Further work will be needed to enhance the geometallurgical model used for the project, so nickel recoveries can be more accurately predicted, together with the use of the pyrrhotite rejection process.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

Mineral Resources have been estimated for the Maturi, Maturi Southwest, Birch Lake, and Spruce Road Cu–Ni–PGE deposits. This Report was prepared, in part, to support updated Mineral Resource estimates for the Maturi deposit that were based on additional data collected in 2013–2014. The Maturi Southwest and Birch Lake Mineral Resource estimates and the re-tabulation of Mineral Resources for the Spruce Road deposit remain unchanged from the Parker and Eggleston (2014) report.

The Maturi, Maturi Southwest, and Birch Lake Mineral Resource estimates were prepared under the supervision of Dr. Harry Parker, RM SME, of AMEC. All three estimates used Vulcan software and ordinary kriging (OK) interpolation.

The 2014 Maturi resource model was completed by Douglas Reid, P.Eng., AMEC Principal Geological Engineer. The 2014 Maturi geological model was constructed using 554 holes (1,435,990 ft; 437,689.8 m) that were drilled between 1960 and 2014.

The Maturi Southwest resource model was also completed by Douglas Reid. The Maturi Southwest geological model was constructed using 143 drill holes (177,900 ft) that were drilled between 1960s and 2013 that included not only holes from Maturi Southwest but holes from the Maturi and Birch Lake areas. Many of these holes were well outside the grade estimation area, but were used to help guide the construction of the outer edges of the geological model. The Maturi Southwest resource estimate was based on a subset of these holes comprising 42 TMM drill holes and seven legacy holes.

The 2012 Birch Lake resource model was completed by Tim Kuhl, RM SME, AMEC Principal Geologist. This model update was completed with 115 drill holes (288,781.5 ft; excluding wedges) that were drilled between the 1970s and 2012.

Scott Wilson Roscoe Postle Associates Inc. (SWRPA) produced a resource estimate for the Spruce Road deposit in 2007 (Routledge and Cox, 2007) for Franconia. AMEC reviewed and accepted the SWRPA model and recast the resource estimate based on underground mining assumptions. The Spruce Road resource estimate is based almost entirely on largely unverified ACNC legacy data. Details of that resource estimate are included in Routledge and Cox (2007).

14.2 Database Adjustments

14.2.1 Un-sampled Intervals

At Maturi and Birch Lake, numerous intervals were not sampled for a variety of reasons. At Maturi, un-sampled intervals were assigned lower detection limit values for all elements. In cases where core was not recovered in a mineralized interval, missing intervals were purposely left blank. This allows the estimation algorithm to estimate across intervals with no core recovery so that grades are not affected because of the missing data.

Early campaigns in the Birch Lake area did not sample the full extent of the BMZ. In 2011, AMEC recommended that, where possible, those non-sampled intervals be recovered and properly sampled. In 2012, TMM recovered 733 samples in previously non-sampled intervals and had those samples analyzed at ALS Chemex. Remaining missing values were assigned lower detection limit values.

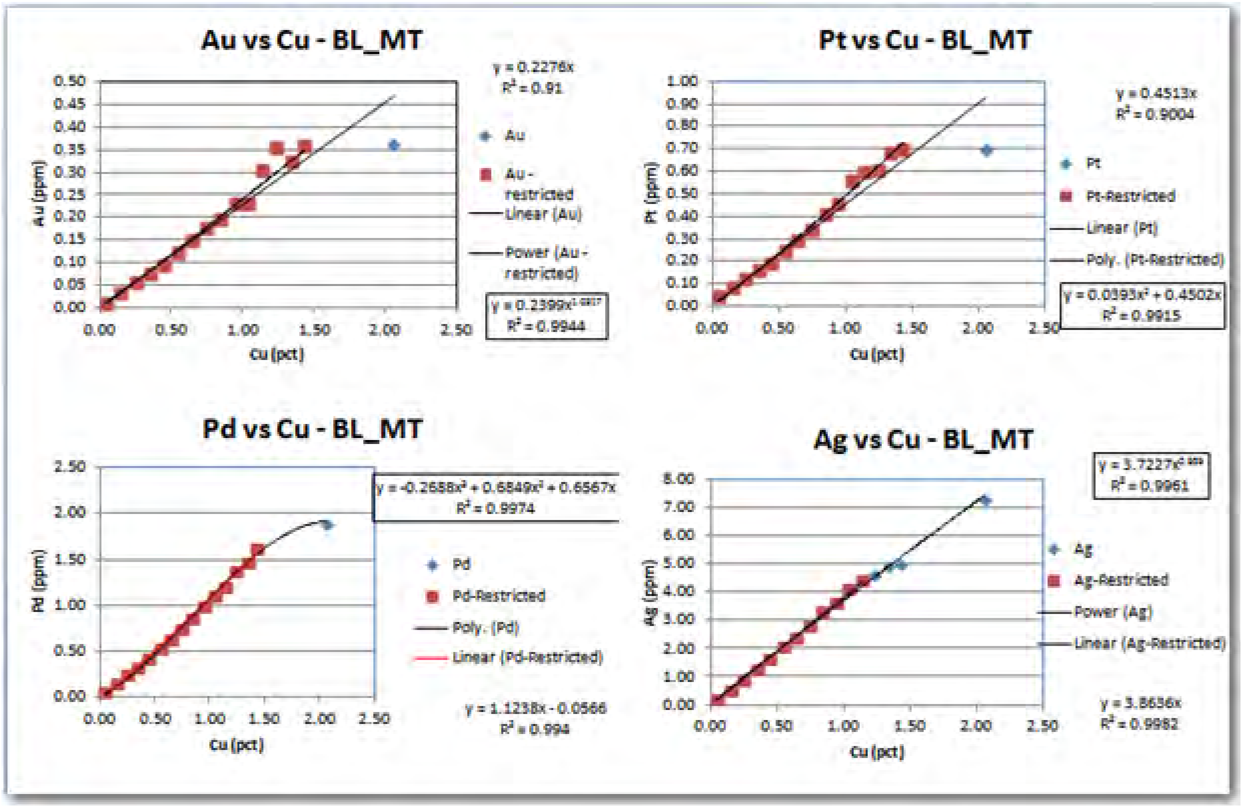
14.2.2 Regressions for PMs

Legacy drill holes at Maturi, Maturi Southwest, and Birch Lake were only assayed for copper and nickel. To aid in grade estimation in areas populated by the legacy drilling, AMEC developed regression equations for Pt, Pd, Au, Co, Ag, S, Cr, and Mg using either copper or nickel grades, depending on which pairing of the dependent variable had the highest correlation with copper or nickel. This is an accepted practice used in similar deposits located in the Sudbury basin. Figure 14-1 shows an example of regressions for Birch Lake.

14.2.3 Wedge Group Drill Hole Construction

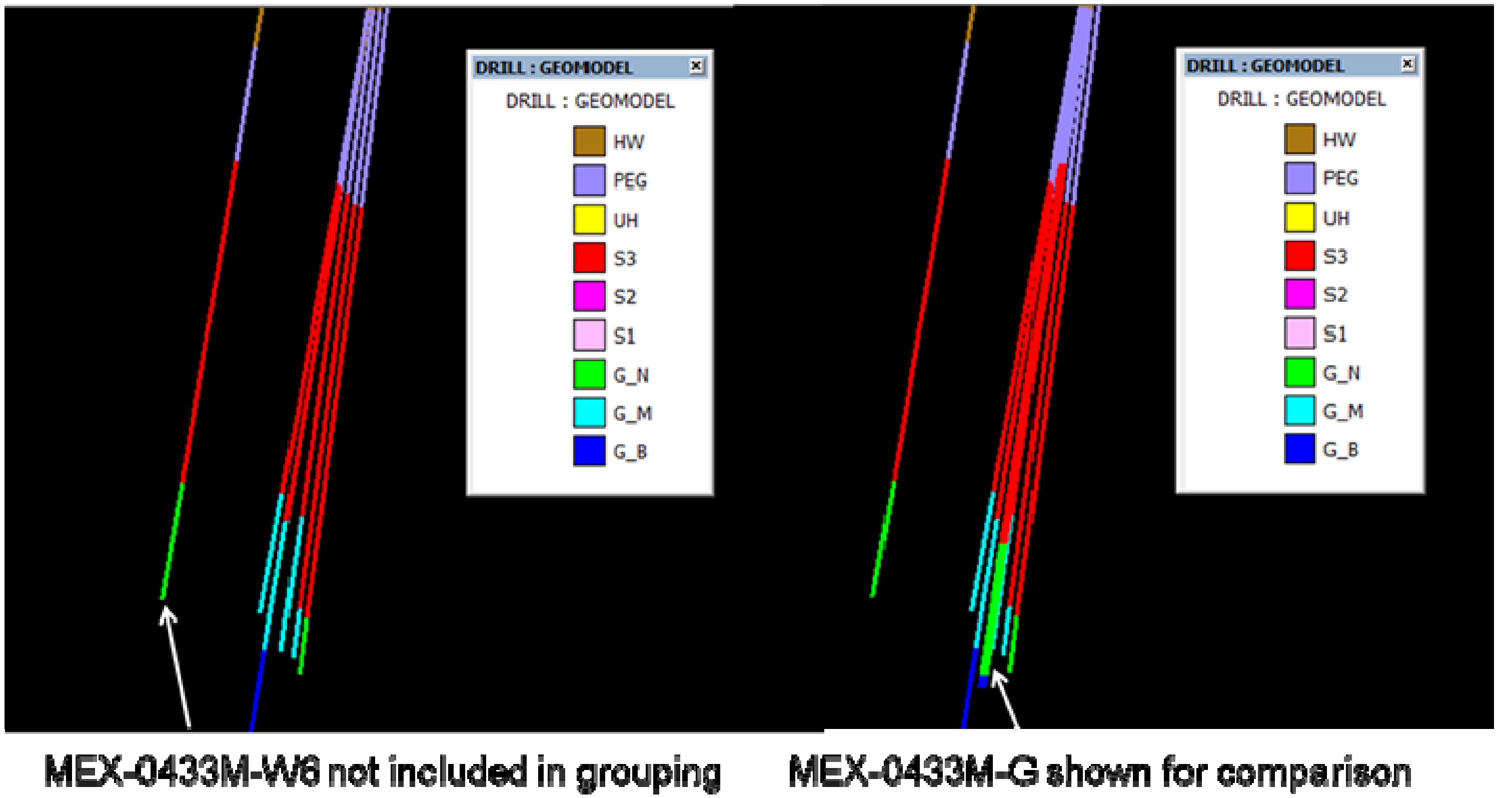
The Maturi and Birch Lake data include numerous wedge holes that were drilled for the purpose of confirming grades and/or collecting material for metallurgical testing. Due to declustering difficulties and geological surface modeling issues, each pilot and associated wedge holes were combined into “group” holes at Maturi and Birch Lake. The assays and downhole locations from the pilot and included wedge holes were then averaged to generate assays and location data for the “group” hole. Geologic unit intervals were assigned to the “group” hole. If an individual wedge hole was greater than 25 ft from the pilot or other wedge holes, it was excluded from the grouping. Figure 14-2 is an example of a wedge group from Maturi.

Figure 14-1: Grade Regression – Pt vs Cu



Note: Figure prepared by AMEC, 2014

Figure 14-2: Comparison of Group to Pilot and Wedges at Maturi (MEX-0433M-G)



Note: Figure prepared by AMEC, 2014

14.3 Geological Models

The geological models at Maturi, Maturi Southwest, and Birch Lake were constructed by AMEC using geological picks provided by TMM and reviewed by AMEC. Topographic surfaces were generated using LIDAR data with a 2 ft contour interval supplied by TMM. Gridded surface models for each of the stratigraphic units were generated using X, Y and Z of drill hole intercepts using Vulcan Grid Calc modeling functions. A grid cell size of 50 x 50 ft was used at Maturi. A grid cell size of 25 ft x 25 ft was used over the model area at Maturi Southwest and Birch Lake. The grid surfaces were converted to surface triangulations using spot elevations from the drill holes. These triangulations were used to back tag composites and for the construction of the block model.

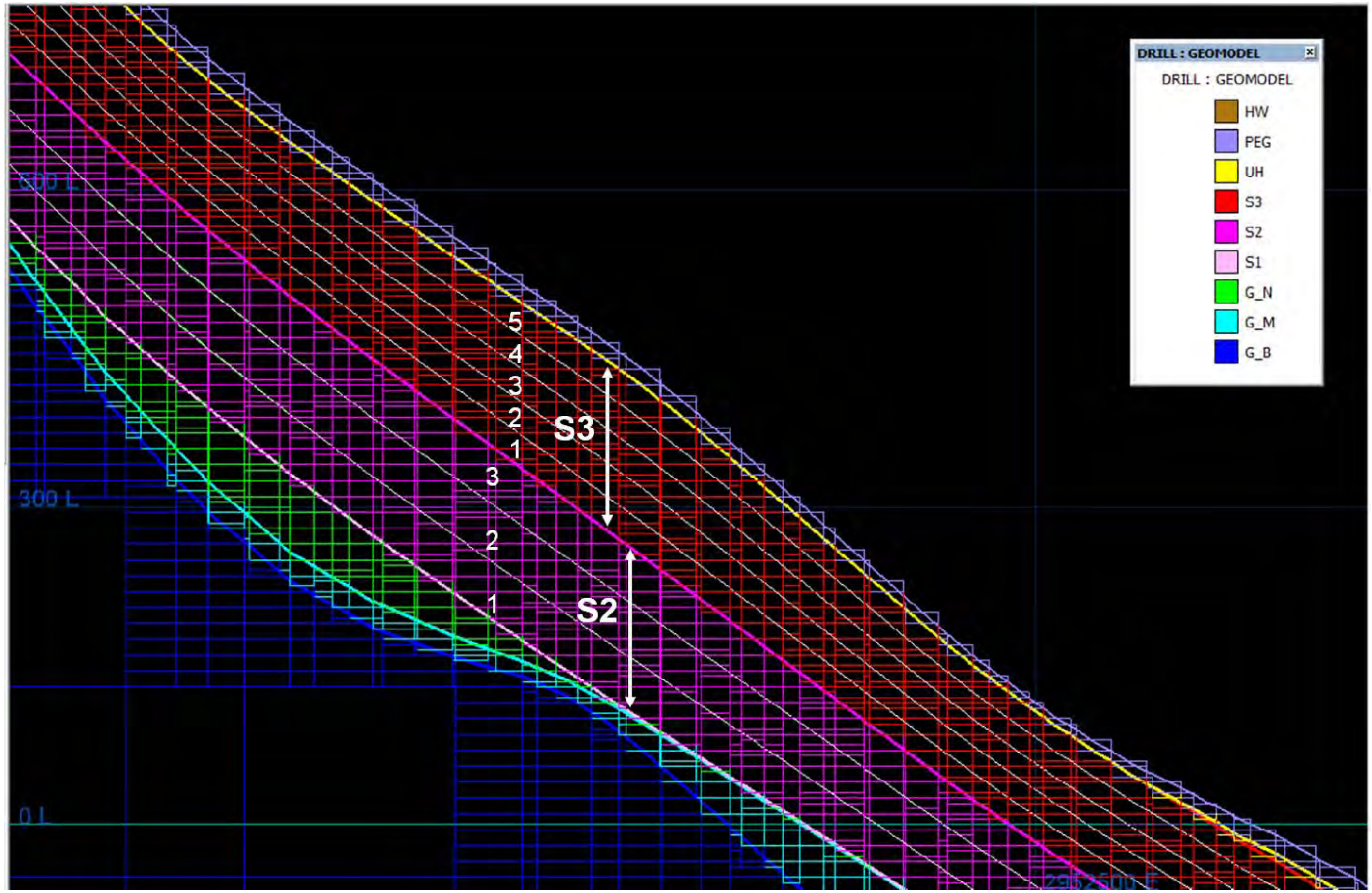
In order to maintain the stratigraphic location of mineralization within the units which typically exhibit vertical gradients, i.e. mineralization located along the upper contact of the S3 unit at Maturi, for example, would be constrained to the upper stratigraphic levels rather than be smeared vertically within the unit, a stratigraphic model of the major units were developed by dividing the major units into three to five layers of equal thickness proportional to the thickness encountered in individual drill holes.

At Maturi, AMEC divided the S3 unit into five stratigraphic layers and the S2 unit into three stratigraphic layers (Figure 14-3). In addition to the stratigraphic layers, AMEC created four domains based on the structural orientation of the deposit to better constrain variography and estimation.

At Maturi Southwest, AMEC divided the S3 Unit into five stratigraphic layers and the S2 Unit into three stratigraphic layers. The geological model was flattened by hanging the BMZ on the HW to remove the variable dip seen in the original model. Variography and grade estimation were performed within the UH, S3, S2, S1, and GM units in the flattened or transformed co-ordinate system. Flattening was used to eliminate the need for domains.

For Birch Lake, BL_MT was subdivided into five stratigraphic levels and BL_T was divided into three stratigraphic layers. Wireframes were constructed for dikes in the southern part of the Birch Lake model area. Two structural domains were created to better constrain variography and estimation.

Figure 14-3: Typical Modeled Unit Geology Showing S3 and S2 Stratigraphic Layers at Maturi



Note: Figure prepared by AMEC, 2014

14.4 Composites

At Maturi, AMEC generated 15 ft composites for Cu, Ni, Pt, Pd, Au, Ag, Co, Cr, S and Mg. The composites were broken by the geological units: UH, S3, S2, S1, GN and GB. Composites with lengths less than 7.5 ft were merged with adjacent composites within the same geological unit where possible. Codes for the stratigraphic layers in S3 and S2 were added to the composite file by flagging the samples using the stratigraphic triangulations.

At Maturi Southwest, AMEC generated 20 ft composites for Cu, Ni, Pt, Pd, Au, Ag, Co, Cr, S and Mg. The composites were broken by the geological units: UH, S3, S2, S1, GM. Composites with lengths of less than 10 ft were merged with adjacent composites within the same geological unit where possible. Codes for the stratigraphic layers in S3 and S2 were added to the composite file by flagging the samples using the stratigraphic triangulations. A 20 ft composite length was chosen to reflect the stope sublevel height under consideration at the time.

The Birch Lake assay data were composited to 15 ft equal length composites. The final composite in each drill hole was stitched into the previous composite if its length was <7.5 ft. The composites were coded with the majority code from the lithology table and were also coded from the geological surfaces and wireframes (AGT, BL_MT, BL_T, BL_HX, GRB_M, GRB_B and dikes).

14.5 Exploratory Data Analysis (EDA)

Information is summarized from Parker and Eggleston (2014), and a more detailed discussion of the EDA performed, including example plots and matrices, can be found in that technical report.

14.5.1 Assays

AMEC created boxplots of Maturi assay data to examine the behavior of each metal separated by unit. Assay intervals were tagged with unit geology codes from the drill hole.

For Maturi Southwest assay data, AMEC created box plots of assay data to examine the behavior of each metal by unit. Assay intervals were tagged with unit geology codes from the drill hole.

14.5.2 Composites

AMEC created box plots for the 15 ft Maturi composites and 20 ft Maturi Southwest composites. These were used to evaluate characteristics of the geological units and stratigraphic layers within S3 and S2 units. This assisted with identification of possible grouping of units and stratigraphic layers for each metal. Proposed groupings were then refined using contact plots. Box plots were also created for Birch Lake 15 ft composites

Histograms and probability plots were constructed for Birch Lake 15 ft composites for BL_MT, BL_T, BL_HX, GRB_M for each of the elements to be estimated (copper, nickel, platinum, palladium, gold, silver, cobalt, chromium, magnesium and sulfur). Histograms and probability plots were also constructed for the stratified horizons of the BL_MT and BL_T.

14.5.3 Contact Profiles

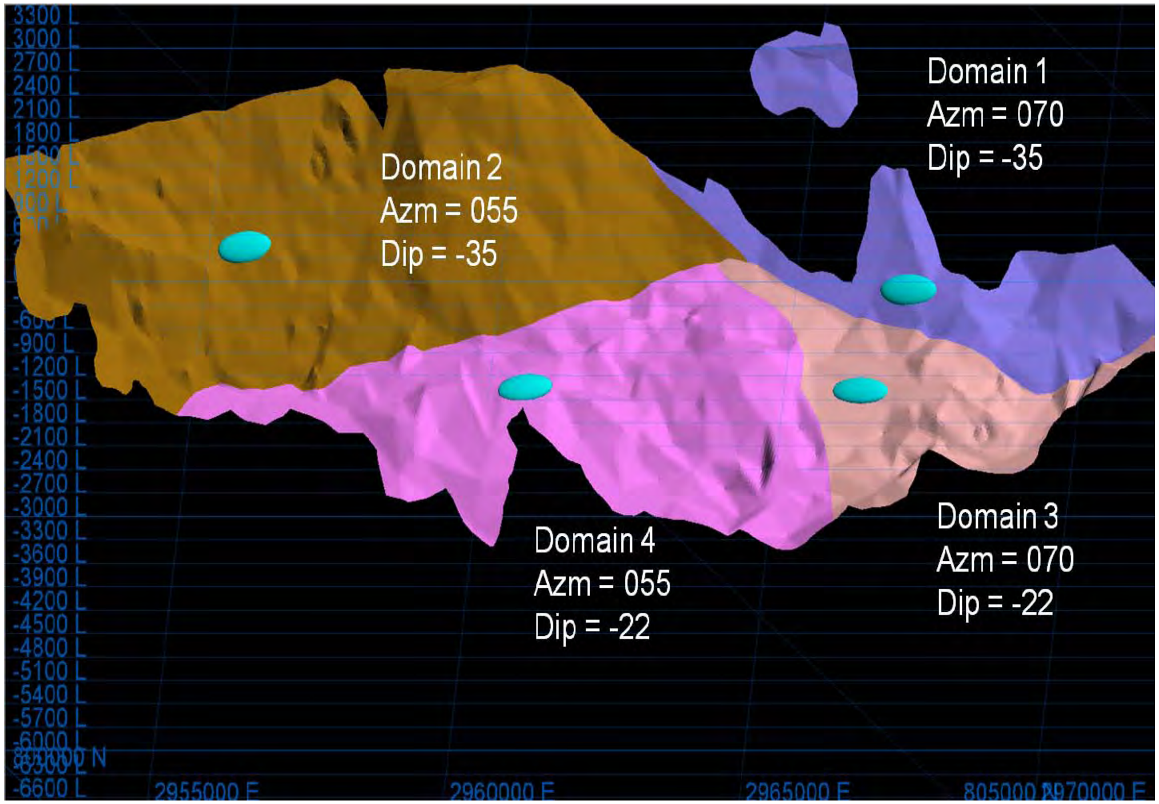
Contact profiles were completed on composites at Maturi, Maturi Southwest, and Birch Lake to evaluate the nature of the contacts between the various geological units. The S3/S2 and BL_MT/BL_T contacts were considered "hard", unless a particular layer was missing in the composite file. This approach was refined to consider the contact as soft, firm or hard (SFH) for each element. A soft contact allows composites to be selected on either side of a contact; a firm contact allows composites within a specified distance of the contact to be selected; while a hard contact does not allow composite selection across the contact.

14.5.4 Variography

AMEC performed variography for each element (Cu, Ni, Pt, Pd, Au, Ag, Co, Cr, S and Mg), and for each unit to be interpolated. Calculations were performed on uncapped grades due to generally low CV values. Values derived from regression equations were not included. AMEC utilized Sage2001 software to assist with the variogram modeling.

At Maturi, unfolding was not applied; instead, the deposit was broken into four domains, (Figure 14-4) of fairly consistent strike and dip, based on the orientation of the modeled top of the S3 unit for variography and estimation purposes. The search ellipse and variograms were oriented to match the domain orientation. Domain boundaries were considered soft; thus composites were shared across domain boundaries. Insufficient data are contained in Domains 1 and 4 to generate reasonable variograms. As a result, the variogram from Domain 2 was rotated and applied to Domain 1 and the variogram for Domain 3 was rotated and applied to Domain 4. Directional correlograms were calculated in the along-strike and down-dip directions. For all units, AMEC assumed the orientation to be equivalent to the top of the S3 unit.

Figure 14-4: Variography Domains at Maturi showing Ellipse Orientations (blue ellipses; surface represents the top of the S3 unit; view looking northeast)



Note: Figure prepared by AMEC, 2014

Variography at Maturi Southwest was performed on the major elements (Cu, Ni, Pt, Pd and Au) and minor elements (Ag, Co, Cr, Mg and S) within the BMZ and GM units using the correlogram approach. Due to the limited number of GM data available, AMEC applied the S1 unit model to the GM unit estimation. Downhole variograms were modeled using 5 ft composites, the nugget (C0) and C1 and C2 (sills of 2 structures) were determined. These values and ranges of both structures were regularized to suit 20 ft composites. This method incorporated the variability seen in drill hole assays into the variograms used in grade estimations. The regularized nugget and sill values were used to model the variograms generated from the 20 ft composites.

Variograms were modeled for BL_MT and BL_T at Birch Lake. Because of the paucity of data in the BL_HX and GRB_M, these units were combined for variography. The Birch Lake deposit generally strikes northeasterly and dips 15^{o} to the east. However, in the central portion of the deposit, the strike direction is north. Variogram domains were identified to accommodate the change in strike (Figure 14-5).

Variogram domain 1 includes the northern and southern portions of the deposit where the strike is northeasterly. Variogram domain 2 is in the central portion of the deposit where the strike is generally north. Variogram models were completed with correlograms by variogram domain for the BL_MT and BL_T.

14.6 Density

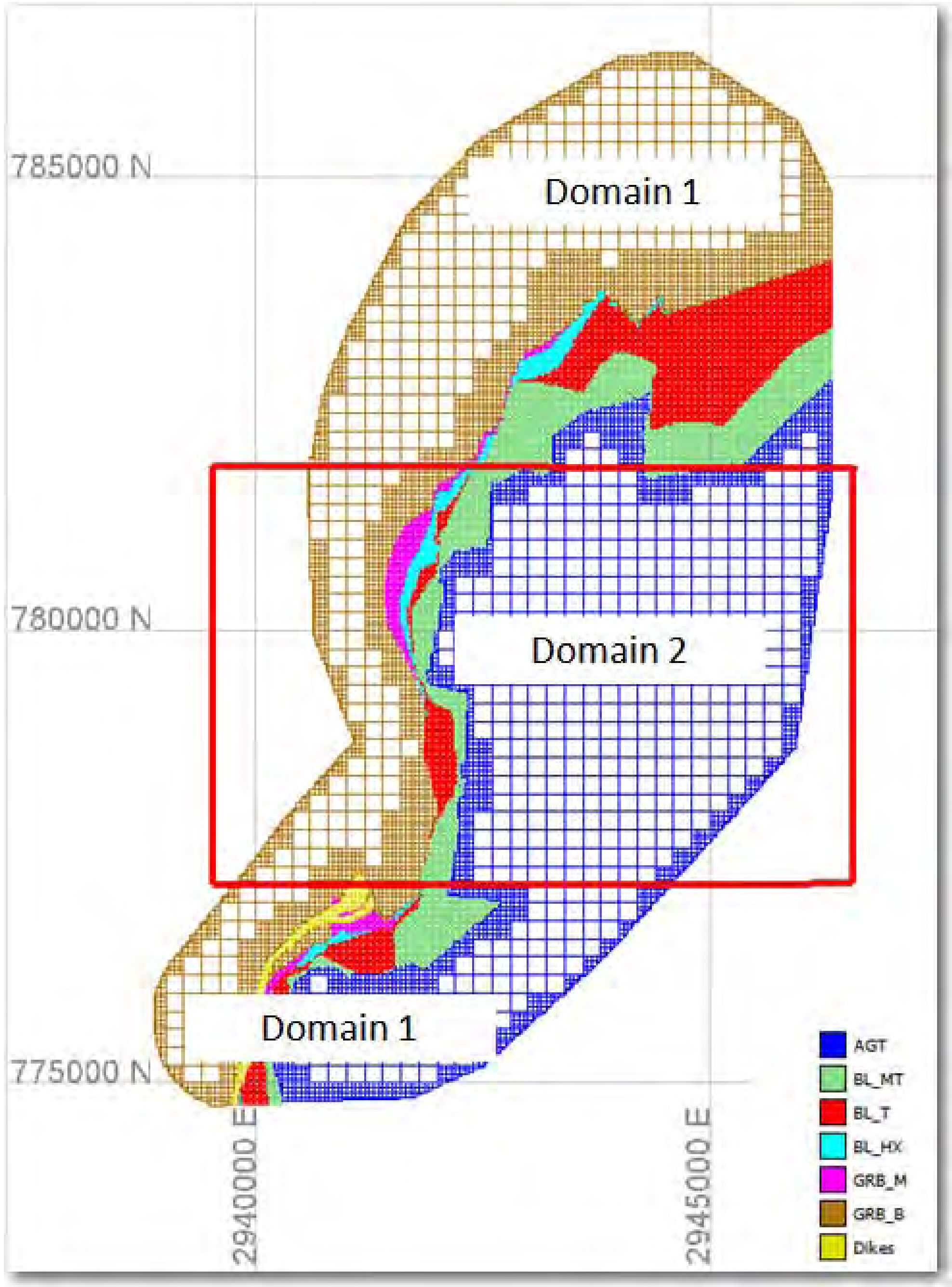
14.6.1 Maturi

A total of 24,644 density measurements were recorded at Maturi. Density data for the S3 and S2 units were refined by the stratigraphic layer. The density for the geological units and stratigraphic subdivisions is shown in Table 14-1. AMEC used the mean density value calculated for each of these groups to derive the tonnage factors assigned in the block model.

14.6.2 Maturi Southwest

At Maturi Southwest, a total of 1,391 density determinations were available. Density for the S3 and S2 units was further refined by the stratigraphic layer. Density values used for the geological units and stratigraphic subdivisions are shown in Table 14-2. AMEC used the mean density calculated for each of these groups to assign a tonnage factor used in the block model.

Figure 14-5: Variogram Domains at Birch Lake



Note: Figure prepared by AMEC, 2014

Table 14-1: Maturi Mean Density Values by Unit and Stratigraphic Layer

Unit	No. Determinations	Mean Density (g/cm^3)	Tonnage Factor (st/ft^3)
PEG	885	2.95	0.09208
UH	777	3.02	0.09427
S3	5,882	3.02	0.09427
S3_5	1,220	3.00	0.09364
S3_4	1,206	3.01	0.09395
S3_3	1,165	3.02	0.09427
S3_2	1,170	3.03	0.09458
S3_1	1,121	3.05	0.09520
S2	4,241	3.05	0.09520
S2_3	1,475	3.07	0.09583
S2_2	1,383	3.06	0.09551
S2_1	1,383	3.04	0.09489
S1	1,261	3.01	0.09395
G_N	474	2.82	0.08802
G_M	2,574	2.78	0.08677
G_B	1,931	2.74	0.08553

Table 14-2: Maturi Southwest Mean Density Values by Unit and Stratigraphic Layer

Unit	No. Determinations	Mean Density (g/cm^3)	Tonnage Factor (st/ft^3)
UH	117	2.99	0.0933
S3	405	2.99	0.0933
S3_5	86	2.96	0.0924
S3_4	76	2.98	0.0930
S3_3	84	2.99	0.0933
S3_2	78	3.00	0.0937
S3_1	81	3.02	0.0943
S2	364	3.03	0.0946
S2_3	127	3.03	0.0946
S2_2	124	3.03	0.0946
S2_1	113	3.02	0.0943
S1	222	3.01	0.0940
G_M	38	2.75	0.0858

14.6.3 Birch Lake

Birch Lake density data (4,344 determinations) were coded with the unit code and the mean of the density values for each unit was assigned for the Unit density. The results are summarized in Table 14-3.

14.6.4 Spruce Road

An average density of 3.02 g/cm^3 was used for Spruce Road which is consistent with the average density data from Maturi.

Table 14-3: Birch Lake Density Determinations

Unit	No. Determinations	Mean Density (g/cm^3)	Tonnage Factor (st/ft^3)
AGT	1,582	2.921	0.09117
BL_MT	1,167	3.042	0.09496
BL_T	569	3.036	0.09476
BL_HX	412	3.004	0.09377
GRB_M	234	2.775	0.08661
GRB_B	271	2.783	0.08686
BL_DI	109	3.033	0.09467

14.7 Block Model

14.7.1 Estimation

14.7.1.1 Maturi

Grade estimates for the 2014 Maturi resource model update were completed for copper, nickel, palladium, platinum, gold, silver, cobalt, chromium, magnesium and sulfur. Estimates were completed for each element independently. Geological units were each estimated independently. Grade estimates were not completed for the HW, PEG and GB units (below the GM unit). Each element was estimated independently in multiple passes with expanding searches for each pass within the unit. Estimation passes are shown in Table 14-4. A restrictive pass (Pass 0) was used for estimation of PGE to reduce the smearing of higher grades. This pass used a smaller number of composites within a reduced search ellipse to eliminate contribution of distant drill holes. Thereafter the estimation passes were applied.

The large number of passes shown in Table 14-4 was required to accommodate the various combinations of search ellipses, geological units, and stratigraphic layers. At the completion of grade estimation, a Vulcan script was run to fill any unestimated blocks with the average grade for the particular domain, unit, and stratigraphic level. There were no unestimated blocks within the Measured, Indicated and Inferred Mineral Resource classifications.

14.7.1.2 Maturi Southwest

Grade estimates were completed for copper, nickel, palladium, platinum, gold, silver, cobalt, chromium, magnesium and sulfur. The estimates were completed for each element independently. The geological units were each estimated independently. Grade estimates were completed for the UH, S3, S2, S1, and GM units. Each element was estimated independently in multiple passes with expanding searches for each pass within the unit. Estimation passes are shown in Table 14-5.

Table 14-4: Maturi Estimation Search Strategy

Pass	Search (ft)			Composites		
	X	Y	Z	Min	Max	Max per Hole
0*	500	500	200	5	9	3
1	1000	1000	200	5	12	3
2	2500	2500	200	5	12	3
3	5000	5000	500	5	12	3

* Pass 0 was applied to Pt, Pd and Au only.

Table 14-5: Maturi Southwest Estimation Search Strategy

Pass	Search (ft)			Composites		
	X	Y	Z	Min	Max	Max per Hole
1	750	750	750	5	12	3
2	1500	1500	1500	5	12	3
3	2500	2500	2500	5	12	3

The three-pass search was required to accommodate the various combinations of search ellipses, geological units, and stratigraphic layers.

At the completion of grade estimation, a Vulcan script was run to fill any unestimated blocks with the average grade for the particular domain, unit, and stratigraphic level. There were no unestimated blocks within the Indicated and Inferred Mineral Resource classifications.

14.7.1.3 Birch Lake

Grade estimates were completed for copper, nickel, palladium, platinum, gold, silver, cobalt, chromium, magnesium and sulfur. Elements in each of the main units (BL_MT, BL_T, BL_HX, GRB_M) were estimated independently. Grade estimation was not completed for the AGT and GRB_B units. Each element was estimated independently in four passes with expanding searches for each pass. The search and sample selection is summarized in Table 14-6.

14.7.1.4 Spruce Road

At Spruce Road, Scott Wilson Roscoe Postle Associates (SWRPA) estimated Mineral Resources at cutoff grades appropriate for underground mining (0.4% Cu) and for open-pit mining (0.26% Cu equivalent) in accordance with the requirements of NI 43-101 and the definitions set out by the CIM Definition Standards for Mineral Resources and Mineral Reserves adopted by the CIM Council on December 11, 2005 (2005 CIM definitions).

Table 14-6: Birch Lake Search Strategy for Grade Estimation

Pass	Search (ft)			Composites		
	X	Y	Z	Min	Max	Max per Hole

1	750	750	750	5	12	3
2	1500	1500	1500	5	12	3
3	2500	2500	2500	5	12	3
4	15000	15000	1500	3	12	3
Note: Pass 4 was a fill pass to populate unestimated blocks located near the periphery of the model area.						

The resource estimate was based on core sampling data and employs 3D computer block modeling with inverse distance squared (ID2) interpolation for the resource amenable to underground mining methods and OK for the resource amenable to open-pit mining methods. Block dimensions were 30 x 15 x 10 m and rotated 28^{o} to be parallel with regional strike and inclined to be parallel with the base of the BMZ.

AMEC used the SWRPA OK model for the re-tabulation of the Mineral Resources at Spruce Road.

14.7.2 Metal at Risk

14.7.2.1 Maturi

AMEC examined probability plots and histograms (logarithmic and arithmetic) of 15 ft composites for each element separated by geological unit. Due to the low coefficient of variation (CV), a very light or no cap grade was selected. Table 14-7 summarizes the grade capping that was applied.

Metal removed from the BMZ units is summarized in Table 14-8. The metal removed was determined by comparing grades between the unrestricted kriged model and the final kriged model where the grade capping was applied. Within the Measured Mineral Resources, the metal removed for copper and nickel is relative 0.2% and 0.0% respectively. Metal removed for platinum, palladium and gold is relative 0.0%, 0.0% and 1.3% respectively. Within the Indicated Mineral Resources, metal removed was 0.20% for copper, 1.2% for nickel, 0.0% for platinum 0.3% for palladium and 0.0% for gold. Within the Inferred Resources, metal removed was 0.0% for copper, 1.8% for nickel, 0.0% for platinum and 0.5% for palladium and 2.0% for gold. The low to no metal removed is due to the very light capping applied.

AMEC considers the level of metal removed from the resource to be reasonable for the Measured, Indicated, and Inferred Mineral Resource classifications. Differences in metal removed between this and the previous model are largely due to the improved geological model.

Table 14-7: Maturi – Grade Capping Levels

Unit	Grade Capping				
	Cu (%)	Ni (%)	Pt (ppm)	Pd (ppm)	Au (ppm)
UH	—	0.25	0.30	0.60	0.15
S3	2.00	0.60	—	2.00	2.00
S2	—	0.50	0.40	—	0.30
S1	0.80	0.25	0.25	0.40	0.10
GN	1.00	0.80	0.30	0.60	0.15
GM	1.00	—	—	0.90	0.25

Table 14-8: Maturi – Metal Removed by Capping

Measured				
Metal	**Units**	**OK Uncapped**	**OK Capped**	**Metal Removed (OK Uncapped vs OK Capped)**
Copper	%	0.60	0.60	0.2%
Nickel	%	0.19	0.19	0.0%
Platinum	ppm	0.142	0.142	0.0%
Palladium	ppm	0.331	0.331	0.0%
Gold	ppm	0.081	0.080	1.3%
Indicated				
Metal	**Units**	**OK Uncapped**	**OK Capped**	**Metal Removed (OK Uncapped vs OK Capped)**
Copper	%	0.50	0.50	0.2%
Nickel	%	0.16	0.16	1.2%
Platinum	ppm	0.137	0.137	0.0%
Palladium	ppm	0.309	0.308	0.3%
Gold	ppm	0.073	0.073	0.0%
Inferred				
Metal	**Units**	**OK Uncapped**	**OK Capped**	**Metal Removed (OK Uncapped vs OK Capped)**
Copper	%	0.36	0.36	0.0%
Nickel	%	0.12	0.11	1.8%
Platinum	ppm	0.097	0.097	0.0%
Palladium	ppm	0.220	0.219	0.5%
Gold	ppm	0.050	0.050	0.0%

14.7.2.2 Maturi Southwest

AMEC examined probability plots and histograms (logarithmic and arithmetic) of assays and 20 ft composites for each element separated by geological unit. Due to the low coefficient of variation (CV) observed, a very light or no cap grade was selected. Assays above the capping levels were capped prior to compositing. Table 14-9 summarizes the grade capping that was applied.

Table 14-9: Maturi Southwest – Grade Capping Levels

Unit	Grade Capping									
	Cu (ppm)	Ni (ppm)	Pt (ppm)	Pd (ppm)	Au (ppm)	Ag (ppm)	Co (ppm)	Cr (ppm)	Mg (%)	S (%)
UH	—	—	0.10	-	0.15	3.0	600	—	—	1.0
S3	—	—	0.50	1.00	0.40	5.0	1000	200	—	2.0
S2	—	—	0.30	-	0.20	4.0	500	200	—	-
S1	—	—	0.15	0.25	0.10	3.5	600	—	—	—
GM	—	—	-	0.20	0.06	2.0	200	100	—	1.0

Metal removed from all units at Maturi Southwest is summarized in Table 14-10. The metal removed was determined by comparing grades between the kriged uncapped composites and the final kriged model where capped composites were used. Copper and nickel were not capped; thus there was no metal was removed for those elements. Globally, the metal removed for platinum, palladium and gold is 0.76%, 0.27% and 1.76% respectively. In the S2 and S3 units the metal removed for platinum, palladium and gold is 0.53%, 0.23% and 1.82% respectively.

AMEC considers the level of metal removed from the resource to be reasonable for the Indicated and Inferred Mineral Resource classifications.

14.7.2.3 Birch Lake

AMEC addressed Birch Lake metal at risk by capping 15 ft composites. Grade capping levels are summarized in Table 14-11. Metal removed is based on a 0.0% Cu cutoff.

Table 14-12 summarizes metal removed from the BL_MT and Table 14-13 summarizes metal removed from BL_T. AMEC considers the level of metal removed from the resource to be reasonable.

14.7.2.4 Spruce Road

Grades at Spruce Road were not capped.

14.7.3 Model Validation

Model validation consisted of visual inspection of cross-sections and plan-sections comparing estimated grades to the composites. Box plots and swath plots were used to compare grade estimates to nearest-neighbor (NN) grades and composite grades. Contact plots were also generated comparing the block estimates and composite grades across the geological contacts.

Table 14-10: Maturi Southwest – Global Metal Removed by Capping

Metal	Unit	OK Uncapped	OK Capped	Metal Removed (OK Uncapped vs OK Capped)
Indicated				
Copper	%	0.37	0.37	0.00%
Nickel	%	0.13	0.13	0.00%
Platinum	ppm	0.062	0.061	0.92%
Palladium	ppm	0.142	0.141	0.36%
Gold	ppm	0.037	0.036	2.36%
Inferred				
Metal	**Unit**	**OK Uncapped**	**OK Capped**	**Metal Removed (OK Uncapped vs OK Capped)**
Copper	%	0.26	0.26	0.00%
Nickel	%	0.10	0.10	0.00%
Platinum	ppm	0.042	0.042	0.28%
Palladium	ppm	0.101	0.101	0.00%
Gold	ppm	0.026	0.026	0.00%

Table 14-11: Birch Lake – Capping Levels (ppm unless otherwise specified)

Metal	Unit	Capping Level by Unit			
		BL_MT	BL_T	BL_HX	GRB_M
Copper	%	1.10	0.75	0.70	0.80
Nickel	%	0.50	0.30	0.45	0.80
Palladium	ppm	2.00	0.75	0.90	0.60
Platinum	ppm	1.20	0.40	0.45	0.25
Gold	ppm	0.40	0.20	0.20	0.17
Silver	ppm	6.50	2.80	2.70	2.80
Cobalt	ppm	200	150	250	300
Chromium	ppm	2000	1000	400	200
Magnesium	%	None	11.0	11.0	6.0
Sulfur	%	2.0	2.0	4.0	4.0

Table 14-12: Birch Lake – Global Metal Removed by Capping – BL_MT

Metal	Units	Uncapped Mean	Capped Mean	Relative Metal Removed
Copper	%	0.42974	0.42943	0.1%
Nickel	%	0.13894	0.13894	0.0%
Palladium	ppm	0.33843	0.33842	<0.1%
Platinum	ppm	0.16467	0.16463	<0.1%
Gold	ppm	0.08027	0.07982	0.6%
Silver	ppm	1.59172	1.59172	0.0%
Cobalt	ppm	97.60	97.60	0.0%
Chromium	ppm	300.74	296.56	1.4%
Magnesium	%	6.56	6.56	0.0%
Sulfur	%	0.69744	0.69675	0.1%

Table 14-13: Birch Lake – Global Metal Removed by Capping – BL_T

Metal	Units	Uncapped Mean	Capped Mean	Relative Metal Removed
Copper	%	0.17154	0.17154	0.0%
Nickel	%	0.06154	0.06126	0.5%
Palladium	ppm	0.08810	0.08672	1.6%
Platinum	ppm	0.04324	0.04314	0.2%
Gold	ppm	0.02285	0.02282	0.1%
Silver	ppm	0.64934	0.64632	0.5%
Cobalt	ppm	68.74	68.55	0.3%
Chromium	ppm	175.36	173.49	1.1%
Magnesium	%	4.18	4.18	0.0%
Sulfur	%	0.42094	0.41298	1.9%

A summary of the results from Parker and Eggleston (2014) is included in the following sub-sections, and additional information and example plots can be found in that technical report.

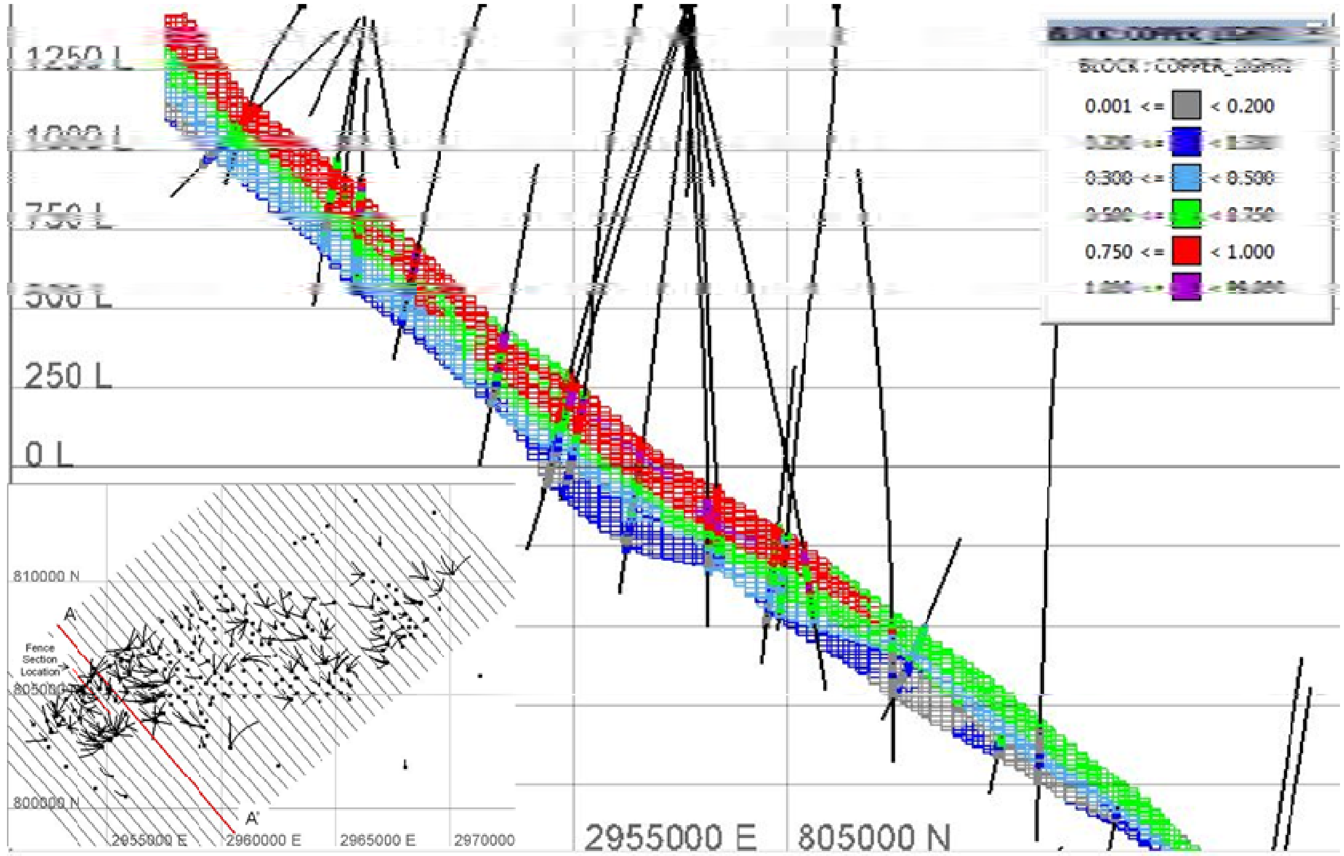
14.7.3.1 Nearest Neighbor (NN) Model

NN models were completed for model validation for Maturi, Maturi Southwest, and Birch Lake. The NN model provides a declustered distribution of grades, wherein a block is assigned the grade of the closest composite. Kriged models use multiple composites to interpolate grades into blocks. While this theoretically provides more accurate local estimates, sometimes artifacts are introduced related to selection of composites from areas with different mean grades or assigning too much weight to some composites and too little to others. The NN model is a benchmark used to check for problems in the kriging process. The NN models utilized the same search criteria as the OK estimates and were used for comparison of summary statistics in box plots and swath plots. Model validation was completed using blocks classified as Indicated for the BMZ units and blocks within 250 ft of a drill hole for the GN and GM units, as these were classified as Inferred.

14.7.3.2 Visual Inspection

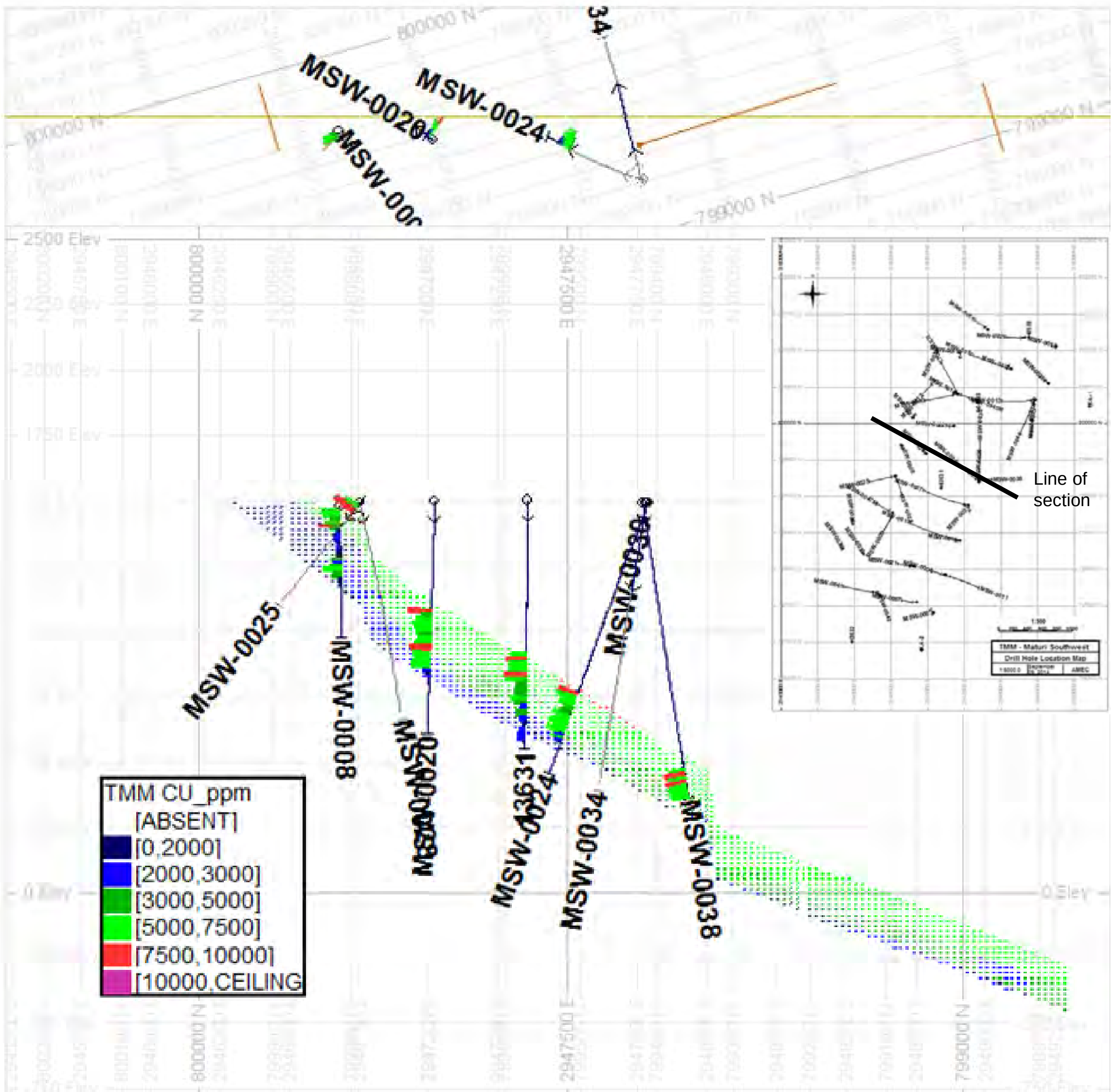
Visual inspection on sections and plans comparing estimated block grades to the composite grades was completed for all elements at Maturi (Figure 14-6), Maturi Southwest (Figure 14-7), and Birch Lake (Figure 14-8). Figure 14-9 is a cross section at Spruce Road. In all cases, AMEC noted good correlation between composite data and block grades. In general AMEC observed good grade and thickness continuity between drill holes; however, in some areas, additional drilling is required to reduce the distance between existing drill holes. This additional drilling is required to increase the confidence in the resource model and to reduce the reliance on legacy drill holes.

Figure 14-6: Maturi Copper Grades for Block and Composites – Section A-A' (looking Northeast) Detail View



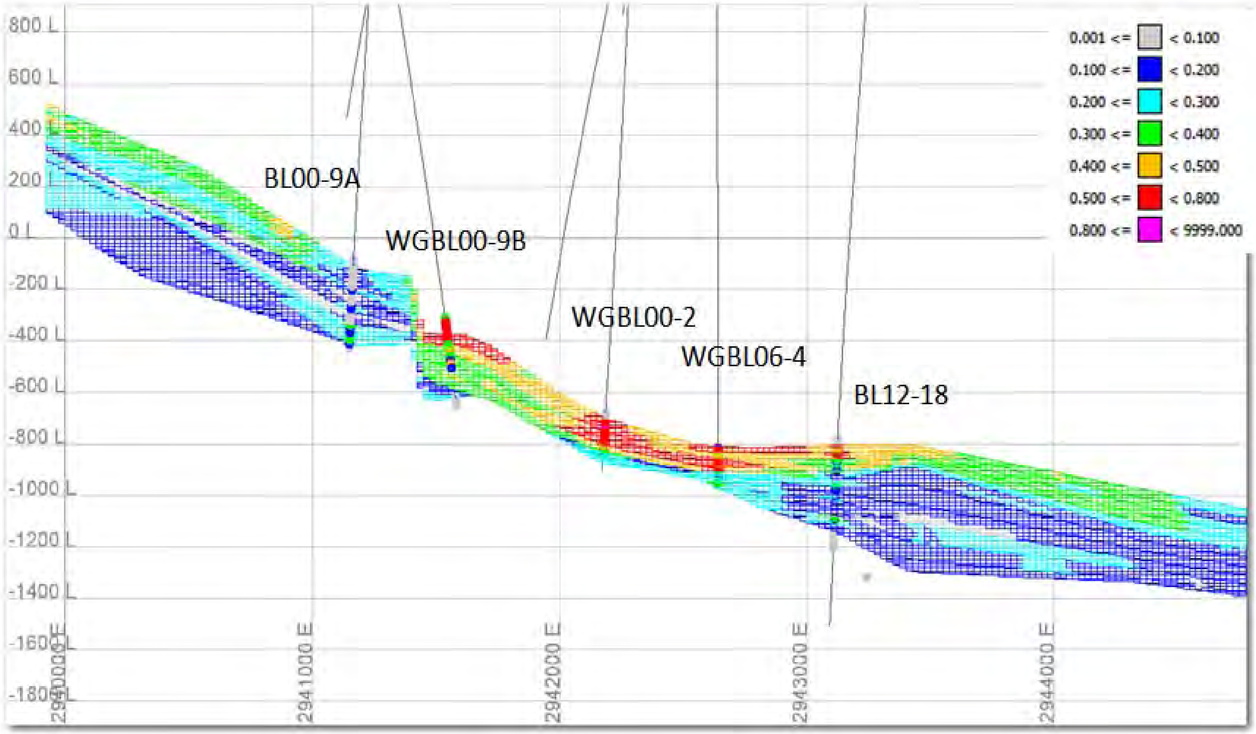
Note: Figure prepared by AMEC, 2014

Figure 14-7: Maturi Southwest Copper Grades for Block and Composites



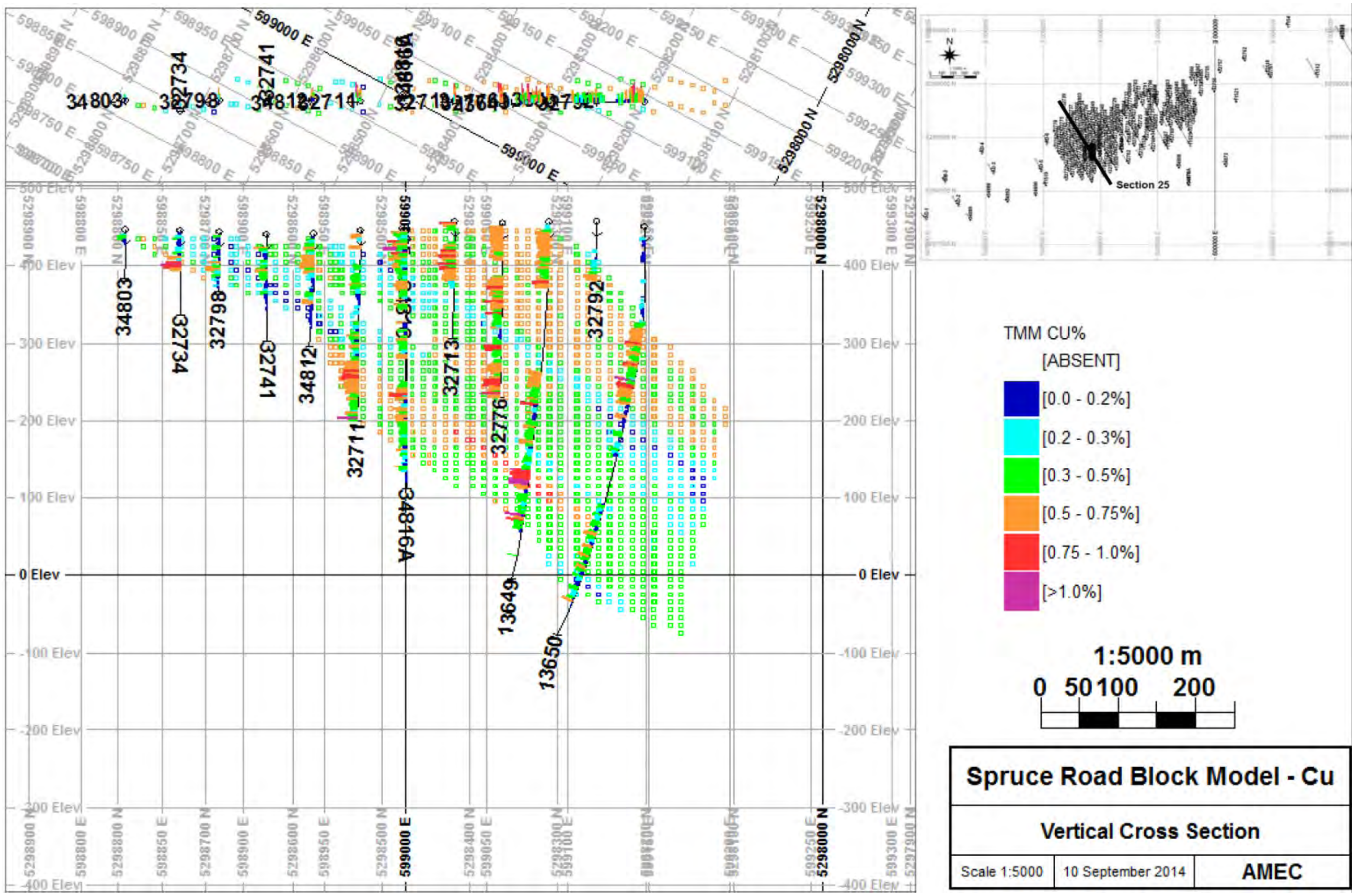
Note: Figure prepared by AMEC, 2014

Figure 14-8: Birch Lake Copper Cross Section 777400 N (Drill Hole Projection 200 ft; units % Cu)



Note: Figure prepared by AMEC, 2014

Figure 14-9: Spruce Road Section 2



Note: Figure prepared by AMEC, 2014

14.7.3.3 Boxplots

At Maturi, Maturi Southwest, and Birch Lake, boxplots were completed for each element comparing the composites, NN, and kriged estimates by unit and by stratigraphic group for each metal. In all cases, boxplots generally show very good agreement of average grades of composites with NN and kriged estimates indicating that globally, the kriging process gives the same results as the NN (declustered) model.

14.7.3.4 Swath Plots

Swath plots were constructed for a combination of units and stratigraphic levels in Maturi, Maturi Southwest, and Birch Lake. Swath plots compare the OK grade estimates to the NN grades and the grades of the composites in swaths across the model. Swath intervals were 500 ft in the easterly and northerly directions and 100 ft in the vertical direction. Swaths generally show good agreement with the exception of areas where data become sparse.

14.7.3.5 Block Contact Profiles

Contact grade profiles were constructed across the contacts between the various units and across the stratigraphic levels within Maturi, Maturi Southwest, and Birch Lake. These contact profiles compare the OK grade estimates and the grades of the 15 ft composites as they approach geologic contacts. The composites tend to be slightly higher grade than the block estimates near the contact, but this is not seen in the swath plots. Likely there was some clustering in the composites in higher grade areas; the block values are declustered.

14.7.4 NSR Calculation

A net smelter return (NSR) was calculated for each block using a Vulcan script. Metal prices used are based on industry-consensus surveys of long-term metal prices used for cash flows and Mineral Reserves with an approximate 15% uplift for evaluation of reasonable prospects for eventual economic extraction of the resources.

14.7.4.1 Maturi and Maturi Southwest

Metal prices used in the Maturi NSR calculations were mutually agreed upon by TMM, Antofagasta and AMEC on 4 February, 2014 (Table 14-14) and are based on a flotation-only process option.

Table 14-14: 2014 Maturi and Maturi Southwest NSR Parameters (US$; Source, TMM, 4 February 2014)

Metal	Price (US$)	Recovery Global	Payable
Copper	$3.30/lb	93.4%	75.2%
Nickel	$10.00/lb	63.9%	48.8%
Gold	$1,350/troy oz	78.2%	56.6%
Palladium	$850/troy oz	76.2%	57.4%
Platinum	$2,000/troy oz	61.3%	43.3%
Silver	$21/troy oz	66.9%	31.1%

14.7.4.2 Birch Lake

Criteria for the Birch Lake NSR calculation are summarized in Table 14-15 and are based on a flotation–hydrometallurgical process option. The metal prices used in the NSR calculation were mutually agreed upon by TMM, Antofagasta and AMEC on 7 December 2011.

14.7.4.3 Spruce Road

Spruce Road NSR parameters (refer to Table 14-15) are extracted from the 2012 Maturi results which are based on a flotation–hydrometallurgical process option. The metal prices used in the NSR calculation were mutually agreed upon by TMM, Antofagasta and AMEC on 7 December 2011.

14.7.5 Reasonable Prospects for Eventual Economic Extraction

During 2012, TMM evaluated a number of conceptual mining scenarios using the 2007 SWRPA PEA and recent process testwork as a basis for a conceptual analysis of likely mining and processing options and costs (Berenguela, 2012, pers. comm.). These studies indicated that a number of large-tonnage throughput rates could be economically attractive and that both a flotation-only and a concentrate with hydrometallurgical recovery processes are viable. This work clearly showed that the Maturi, Maturi Southwest, Birch Lake, and Spruce Road deposits have reasonable prospects for eventual economic extraction under a number of scenarios.

TMM used the Berenguela (2012) study and recent process testwork and mining studies as the basis for NSR and cutoff grade calculations. NSR assumptions are summarized in Section 14.7.4. Mining, process, and G&A cost assumptions are summarized in the following subsections.

Table 14-15: 2011 Birch Lake and Spruce Road NSR Parameters (US$; Source TMM; 7 December 2011)

Metal	Price (US$)	Recovery Concentrate	Recovery Hydromet	Recovery Global	Payable
Copper	$3.00/lb	94.3%	96.3%	90.8%	100.0%
Nickel	$9.38/lb	60.0%	95.6%	57.4%	80.0%
Platinum	$1,840/troy oz	93.0%	59.4%	55.2%	80.0%
Palladium	$805/troy oz	90.0%	70.7%	63.6%	80.0%
Gold	$1,050/troy oz	85.0%	74.5%	63.3%	80.0%
		Spruce Road Parameters			
Metal	**Price (US$)**	**Recovery Concentrate**	**Recovery Hydromet**	**Recovery Global**	**Payable**
Copper	$3.00/lb	94.3%	96.3%	90.8%	100.0%
Nickel	$9.38/lb	72.2%	95.6%	68.8%	80.0%

Processes and procedures for permitting a mine in Minnesota are well understood. AMEC believes that there is a reasonable expectation that permits will be obtained to mine the deposit; however, even though the processes and procedures are in place, AMEC considers permitting a significant risk because of proximity to the Boundary Waters Canoe Wilderness Area.

AMEC believes that the Maturi, Maturi Southwest, Birch Lake, and Spruce Road deposits have reasonable prospects for eventual economic extraction.

14.7.5.1 Maturi and Maturi Southwest

Recent work shows that a flotation-only option where two concentrates are produced, a primary copper-rich concentrate and a combined Ni–Cu–PGE concentrate from the tails of the primary concentrate, may potentially be the best option at Maturi.

Estimated production costs are as follows:

- Mining = $12.54/st
- Process = $5.96/st
- G+A costs = $3.16/st.

The total production cost is estimated to be $21.66/st. This equates to a break-even NSR cost of about $22/st which approximates a copper cutoff grade of about 0.3%.

14.7.5.2 Birch Lake and Spruce Road

Large-scale underground mining with concentration and hydrometallurgical processing was the most attractive mining scenario for Birch Lake and was the basis for the following mining and process cost assumptions:

- Mining costs - $16/st
- Process costs - $12/st

- G+A costs - $2/st.

These sum to a total operating cost of $30/st (TMM, 7 December 2011) which indicates a breakeven NSR of approximately $30/st. Resources meeting an NSR cutoff of $30/st approximately equate to a copper cutoff of 0.3%.

Spruce Road results are based on Maturi 2012 testwork and NSR parameters because of geological and mineralogical similarities. The NSR parameters and mining, process, and G&A costs for Maturi as stated in 2012 indicate a copper cutoff of about 0.3% is appropriate.

14.8 Resource Classification

14.8.1 Maturi

In 2013, AMEC reviewed the Maturi drill data and, using geostatistical tools and internal protocols, estimated the drill hole spacing to support Measured and Indicated Mineral Resources at the Maturi deposit (Reid, 2013). Based on the confidence limit study, AMEC estimated that the maximum drill hole spacing required to meet the 90% confidence level of ±15% on a quarterly production increment (Measured Mineral Resources) was approximately 325 ft. The maximum drill hole spacing required to meet the 90% confidence level of ±15% for an annual production increment (Indicated Mineral Resources) was undefined. AMEC maintained the 500 ft spacing recommendation for Indicated Mineral Resources as stated in the 2011 drill hole spacing study (Reid, 2011).

Mineral Resources are classified as Measured when a block is located within 250 ft to the nearest composite and two composites from two additional drill holes are within 360 ft. Under these conditions, the drill hole spacing for Measured Mineral Resources broadly corresponds to a 325 ft grid. Indicated Mineral Resources are supported by a drill hole spacing of 500 ft. This typically requires one composite located within 390 ft and one composite from an additional drill hole located within 550 ft from the block centroid.

AMEC calculated the distance from each block to the closest three drill holes. The drill holes were selected based on matching stratigraphic units (a block coded as S3 would require selected drill holes to contain an S3 interval). Blocks that met the following criteria were assigned a code 1 (Measured Mineral Resources):

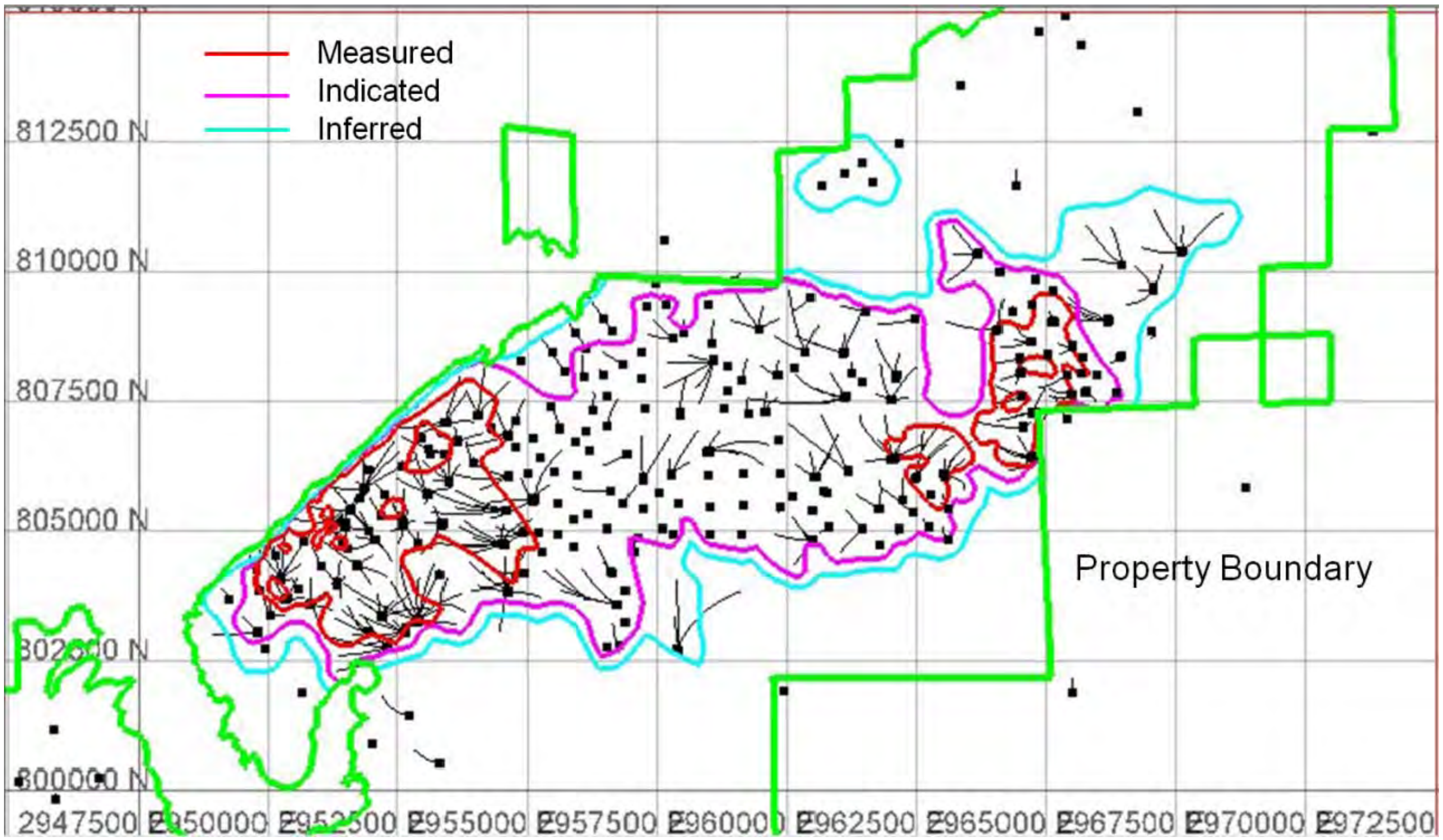
- Block must meet the distance criteria described above
- Block must have been classified as Indicated in 2012
- Block must be coded as a S3 or S2 unit. S1 and GRB units were not considered for Measured Mineral Resource classification.

Areas defined primarily by legacy drilling are not included in the Indicated Mineral Resource outline, and were downgraded to an Inferred Mineral Resource classification. This downgrade was due to uncertainty in collar location, downhole location, lack of QA/QC to support assays, and the use of regressed data for Pt, Pd, Au, and the minor elements.

To identify regions where the estimates were largely influenced by legacy drilling, AMEC created an indicator model using legacy composites flagged as 1 and MEX (current) composites flagged as 0. These indicators were kriged into blocks. Blocks within the Measured Mineral Resource between surface and the 775 ft elevation with an estimated indicator of over 0.30 were re-classified as Indicated. Blocks below the 775 ft elevation with an estimated indicator over 0.25 were re-classified as Indicated. The elevation criterion was based on a review of the downhole deviation of the legacy drill holes. The estimated indicator threshold was selected based on AMEC's experience with other estimates and classification dealing with a mixture of legacy and current drill holes. Blocks within the Indicated classification with an estimated indicator of over 0.50 were used to refine the Inferred classification outline. In 2013, only the blocks identified as Indicated were considered as candidates for Measured Mineral Resources and the Inferred Mineral Resource outline remained the same as for 2012.

Using the results of this work as a basis, AMEC reclassified the 2012 Maturi Mineral Resources into Measured, Indicated, and Inferred Mineral Resources (Figure 14-10). AMEC considers this reclassification reasonable in the light of the study discussed above.

Figure 14-10: Plan View Showing the Maturi Resource Classification



Note: Figure prepared by AMEC, 2014.

A 400 ft thick safety pillar was removed from classification and represents an estimate by AMEC of the amount of material required for a safety zone that separates the contemplated underground mine workings from significant ground and surface water as well as to protect nearby housing from the effects of mining. This material will be left in place, and thus cannot be included in the resource estimate as it would not be mined as defined by the CIM (2014) Definition Standards incorporated by reference in NI 43-101. The material left in place is based on studies conducted by Itasca (2014) and may be refined as additional studies are conducted.

14.8.2 Maturi Southwest

The Maturi Southwest Mineral Resource was classified based an underground Mining option and only Indicated and Inferred Mineral Resources have been classified. The Indicated Mineral Resource boundary was generated based on drill hole spacing within individual geologic units.

There were two criteria applied:

- First drill hole within 275 ft and a second drill hole within 550 ft; or
- First drill hole within 389 ft, second drill hole within 550 ft and a third drill hole within 550 ft

Material located within 100 ft of the East Fault was classified as Inferred Mineral Resources until additional drilling better defines the location and nature of the fault. Although the resource classification is based on an underground mining scenario, the deposit will likely be mined to the surface at some point in the future; thus a 15 ft thick skin at the surface was removed from classification to account for the average overburden over the area.

The Inferred Mineral Resource boundary typically extends 500 ft from well drilled areas showing geological continuity. The drill hole spacing averages 500 ft. The GM unit was classified entirely as Inferred Mineral Resources. No material east of the East Fault was considered for classification. This material is considered to be a target for additional exploration.

Areas defined primarily by legacy drilling are not included within the Indicated Mineral Resource outline, and were downgraded to an Inferred Mineral Resource classification. This downgrade was due to uncertainty in collar location, downhole location, lack of QA/QC to support assays, and the use of regressed data for Pt, Pd, Au, and the minor elements. To identify regions where the estimates were largely influenced by legacy drilling, AMEC created an indicator model using legacy composites flagged as 1 and TMM-Maturi Southwest composites flagged as 0. These indicators were kriged into blocks, and blocks with an estimated indicator of over 0.50 were used to refine the Inferred Mineral Resource classification outline. A plan map showing the mineral resource classification outline is shown in Figure 14-11. Drill hole collars used in the resource estimation are shown for reference.

14.8.3 Birch Lake

To determine the mineral resource classification, AMEC reviewed the continuity of grade and the continuity of thickness between drill holes. The process included constructing 247 cross-sections between paired drill holes. Each section was given a continuity letter grade that was the basis for classification (see following). The final resource classification boundaries are presented in Figure 14-12.

The grading classification included:

A. Continuity of grade and thickness observed.
B. Continuity of thickness observed
C. Poor continuity of grade and thickness
D. No continuity

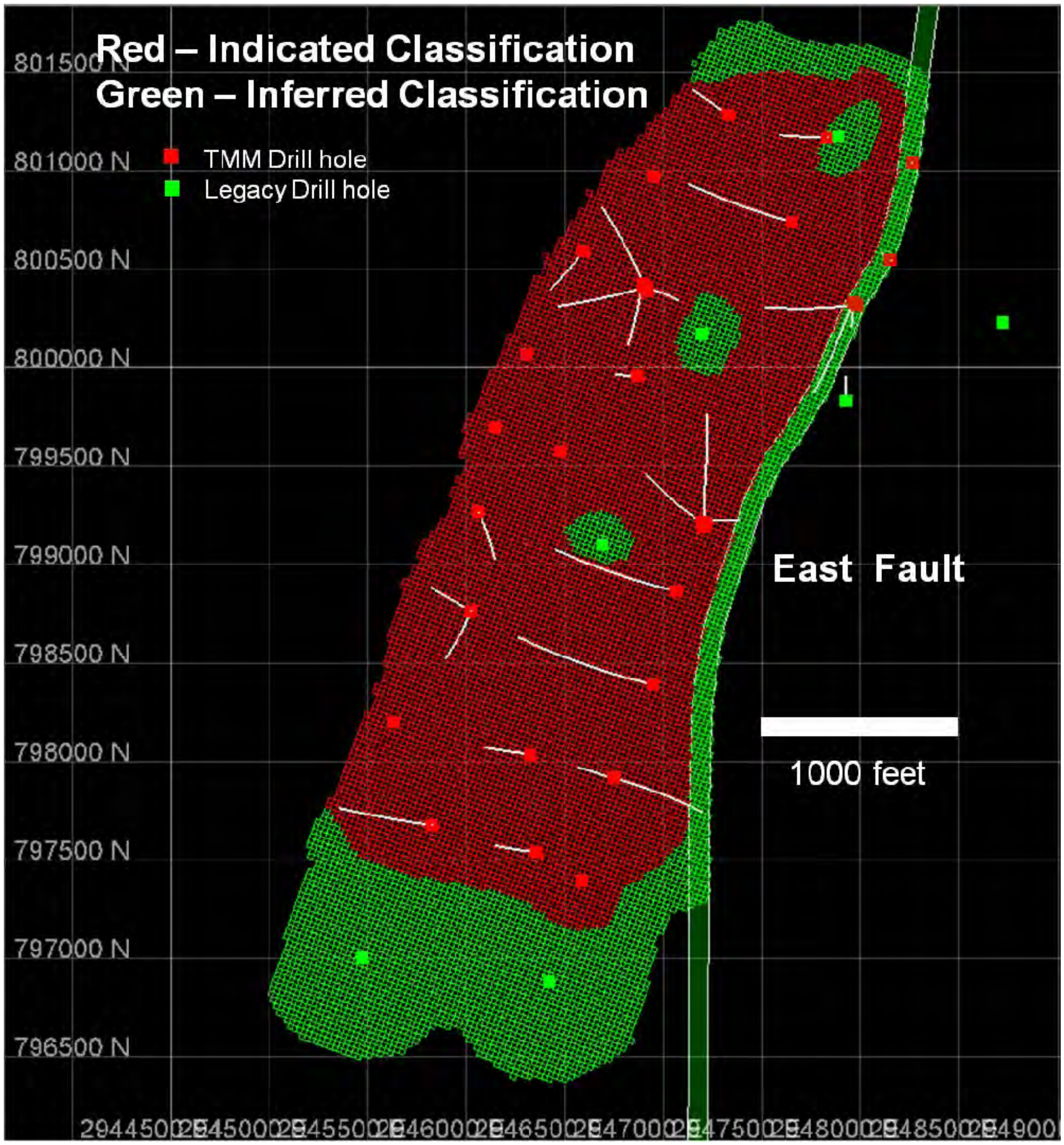
Note: If a fault separated the drill holes on the paired section the classification was downgraded one level.

The Inferred classification was determined by a polyline 250 ft beyond the drilling envelope. The Indicated classification was determined by reviewing the paired sections. Paired sections grading A or B were considered to be Indicated Mineral Resources.

14.8.4 Spruce Road

The Spruce Road resource estimate is based almost entirely on legacy ACNC data that are largely unverified. Wallbridge drilled a single hole (WM_001) in 1999 and that hole is included in the model. The Inco core shed and offices were destroyed by fire, and all original physical records of the Spruce Road deposit were in those facilities; thus the data used for the resource estimate are unverified. Recent drilling at Maturi has largely verified legacy ACNC data from the same era. Based on this work, AMEC concludes that it is reasonable to assume that the data at Spruce Road will be verified when twin holes are completed and thus that the ACNC data are appropriate to use for resource estimation at an Inferred Mineral Resource level.

Figure 14-11: Plan View of the Maturi Southwest Resource Classification (AMEC figure, 2014)



Note: Figure prepared by AMEC, 2014.

14.9 Mineral Resource Tabulations

Mineral Resources are reported using the 2014 Canadian Institute of Mining and Metallurgy Definition Standards (the 2014 CIM Definition Standards) as incorporated by reference in NI 43-101. The Qualified Person for the estimates is Dr. Harry Parker, RM SME, an AMEC employee.

Mineral Resources are reported in million short tons (Mst) and are reported inclusive of Mineral Reserves on a 100% basis. AMEC cautions that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Mineral Resources are stated on an in situ basis, and exclude application of planned and unplanned contact dilution and mining recovery factors, which are discussed in Section 15.7.

The Maturi, Maturi Southwest, Birch Lake, and Spruce Road Mineral Resource estimates are tabulated using cumulative copper cutoff grades. In the tabulations, the basecase, 0.30% Cu, is gray-shaded. The remaining cases are sensitivity cases included to show the sensitivity of the Mineral Resource estimates to changes in cutoff grade. Below 0.2% Cu, the grade is too low to support mining operations. Above 0.6% Cu, current models show the deposits breaking up into numerous pods that may be difficult to mine.

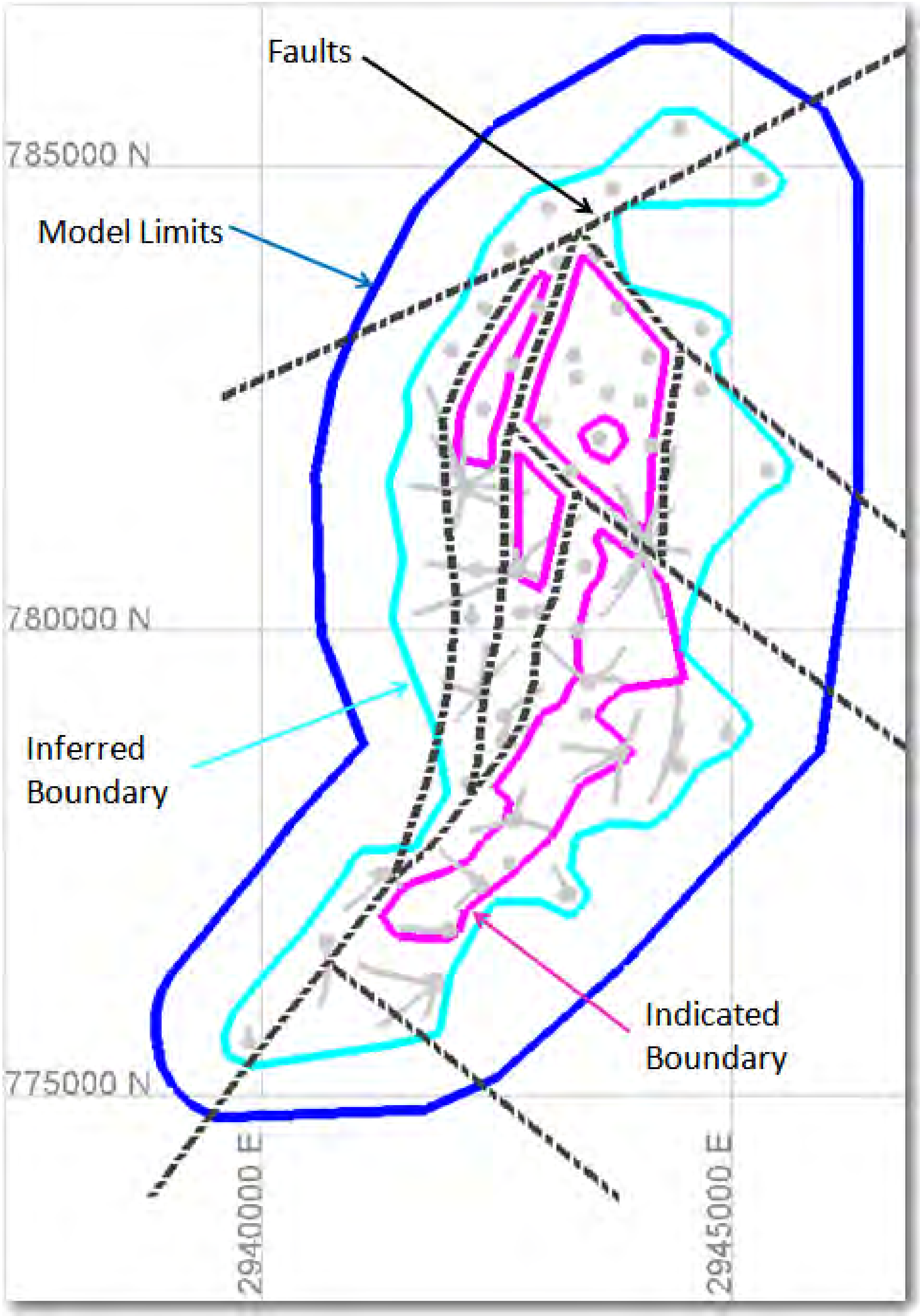
14.9.1 Maturi

Table 14-16 summarizes the Measured, Indicated, and Inferred Mineral Resources for the BMZ and GRB Units. Table 14-17 summarizes Maturi S3+S2 Measured, Indicated, and Inferred Mineral Resources. Information in Table 14-17 is a subset of Table 14-16 and is not additive to that table. These tabulations assume a 400-ft-thick safety pillar above the Mineral Resource.

14.9.2 Maturi Southwest

Table 14-18 tabulates the Indicated and Inferred Mineral Resources for the BMZ and GM Units. Table 14-19 summarizes the Indicated and Inferred Mineral Resources within the S3 and S2 units. Information in Table 14-19 is a subset of Table 14-18 and is not additive to that table. These Mineral Resources are tabulated based on a 15 ft allowance for overburden and no safety pillar.

Figure 14-12: Plan View of the Birch Lake Resource Classification



Note: Figure prepared by AMEC, 2014.

Table 14-16: Maturi Mineral Resources by Copper Cutoff (basecase is highlighted)

Category	Cutoff (Cu %)	Short Tons (Mst)	Grade Cu (%)	Grade Ni (%)	Grade Pt (ppm)	Grade Pd (ppm)	Grade Au (ppm)	Grade Ag (ppm)	Grade Co (ppm)	Grade Pt (oz/st)	Grade Pd (oz/st)	Grade Au (oz/st)	Grade Ag (oz/st)	Contained Metal Cu (Mlb)	Contained Metal Ni (Mlb)	Contained Metal Pt (Moz)	Contained Metal Pd (Moz)	Contained Metal Au (Moz)	Contained Metal Ag (Moz)	Contained Metal Co (Mlb)
Measured	0.2	327.4	0.61	0.20	0.141	0.328	0.080	2.19	104.8	0.004	0.010	0.002	0.064	3,987	1,277	1.3	3.1	0.8	20.9	69
	0.3	308.1	0.63	0.20	0.146	0.339	0.083	2.26	106.6	0.004	0.010	0.002	0.066	3,883	1,245	1.3	3.0	0.7	20.3	66
	0.4	274.9	0.66	0.21	0.155	0.359	0.088	2.38	109.8	0.005	0.010	0.003	0.069	3,651	1,166	1.2	2.9	0.7	19.1	60
	0.5	236.7	0.70	0.22	0.165	0.383	0.093	2.50	112.7	0.005	0.011	0.003	0.073	3,305	1,056	1.1	2.6	0.6	17.3	53
	0.6	182.5	0.74	0.24	0.177	0.411	0.100	2.66	116.3	0.005	0.012	0.003	0.077	2,705	862	0.9	2.2	0.5	14.1	42
Indicated	0.2	881.4	0.56	0.18	0.148	0.336	0.080	2.02	101.9	0.004	0.010	0.002	0.059	9,783	3,138	3.8	8.6	2.1	52.0	180
	0.3	821.8	0.58	0.19	0.155	0.350	0.083	2.10	103.7	0.005	0.010	0.002	0.061	9,484	3,041	3.7	8.4	2.0	50.3	171
	0.4	716.1	0.61	0.20	0.166	0.375	0.089	2.22	106.1	0.005	0.011	0.003	0.065	8,736	2,793	3.5	7.8	1.9	46.4	152
	0.5	546.9	0.66	0.21	0.186	0.420	0.099	2.41	108.8	0.005	0.012	0.003	0.070	7,208	2,286	3.0	6.7	1.6	38.5	119
	0.6	379.2	0.71	0.22	0.205	0.461	0.108	2.60	111.0	0.006	0.013	0.003	0.076	5,354	1,699	2.3	5.1	1.2	28.7	84
Measured	0.2	1208.7	0.57	0.18	0.146	0.334	0.080	2.07	102.7	0.004	0.010	0.002	0.060	13,770	4,414	5.2	11.8	2.8	72.9	248
+	0.3	1130.0	0.59	0.19	0.153	0.347	0.083	2.14	104.5	0.004	0.010	0.002	0.063	13,366	4,286	5.0	11.4	2.7	70.7	236
Indicated	0.4	991.0	0.62	0.20	0.163	0.371	0.089	2.26	107.1	0.005	0.011	0.003	0.066	12,387	3,958	4.7	10.7	2.6	65.4	212
	0.5	783.6	0.67	0.21	0.180	0.409	0.097	2.44	110.0	0.005	0.012	0.003	0.071	10,513	3,342	4.1	9.3	2.2	55.7	172
	0.6	561.7	0.72	0.23	0.196	0.445	0.105	2.62	112.7	0.006	0.013	0.003	0.076	8,059	2,560	3.2	7.3	1.7	42.9	127
Inferred	0.2	767.6	0.42	0.13	0.116	0.262	0.059	1.58	80.5	0.003	0.008	0.002	0.046	6,417	2,057	2.6	5.9	1.3	35.4	124
	0.3	530.6	0.49	0.16	0.138	0.314	0.070	1.81	97.6	0.004	0.009	0.002	0.053	5,242	1,730	2.1	4.9	1.1	28.0	104
	0.4	357.5	0.57	0.19	0.167	0.376	0.083	2.04	109.7	0.005	0.011	0.002	0.060	4,040	1,323	1.7	3.9	0.9	21.3	78
	0.5	235.2	0.63	0.20	0.202	0.449	0.099	2.27	111.6	0.006	0.013	0.003	0.066	2,944	931	1.4	3.1	0.7	15.6	52
	0.6	126.5	0.69	0.21	0.246	0.545	0.118	2.51	110.6	0.007	0.016	0.003	0.073	1,748	539	0.9	2.0	0.4	9.3	28

Notes:

1. The Mineral Resource effective date is 4 February 2014. Dr. Harry Parker, RM SME, AMEC Consulting Geologist and Geostatistician, is the QP for the estimate and is a Professional Geologist licensed in Minnesota.
2. Mineral Resources are reported inclusive of Mineral Reserves and are reported on a 100% basis. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. The Mineral Resource estimates are based on a US$21.66/st NSR that in turn assumes a mining cost of $12.54/st, a process cost of $5.96/st and general and administrative charges of $3.16/st; global metallurgical recoveries of 93.4% (Cu), 63.9% (Ni), 78.2% (Au), 76.2% (Pd), 61.3% (Pt) and 66.9% (Ag); and long-term consensus metal prices of $3.30/lb Cu, $10.0/lb Ni, $1,350/troy oz Au, $850/troy oz Pd, $2,000/troy oz Pt, and $21.00/troy oz Ag.
4. The NSR equates to an approximate 0.3% Cu cutoff grade.
5. Figures have been rounded and may not sum.

Table 14-17: Maturi S3+S2 Mineral Resources by Copper Cutoff (basecase is highlighted)

Category	Cutoff (Cu %)	Short Tons (Mst)	Grade Cu (%)	Grade Ni (%)	Grade Pt (ppm)	Grade Pd (ppm)	Grade Au (ppm)	Grade Ag (ppm)	Grade Co (ppm)	Grade Pt (oz/st)	Grade Pd (oz/st)	Grade Au (oz/st)	Grade Ag (oz/st)	Contained Metal Cu (Mlb)	Contained Metal Ni (Mlb)	Contained Metal Pt (Moz)	Contained Metal Pd (Moz)	Contained Metal Au (Moz)	Contained Metal Ag (Moz)	Contained Metal Co (Mlb)
Measured	0.2	322.3	0.61	0.20	0.142	0.329	0.081	2.21	105.0	0.004	0.010	0.002	0.064	3,958	1,263	1.3	3.1	0.8	20.8	68
	0.3	306.7	0.63	0.20	0.146	0.339	0.083	2.27	106.7	0.004	0.010	0.002	0.066	3,876	1,239	1.3	3.0	0.7	20.3	65
	0.4	274.6	0.66	0.21	0.155	0.359	0.088	2.38	109.8	0.005	0.010	0.003	0.069	3,647	1,164	1.2	2.9	0.7	19.1	60
	0.5	236.7	0.70	0.22	0.165	0.383	0.093	2.50	112.7	0.005	0.011	0.003	0.073	3,305	1,056	1.1	2.6	0.6	17.3	53
	0.6	182.5	0.74	0.24	0.177	0.411	0.100	2.66	116.3	0.005	0.012	0.003	0.077	2,705	862	0.9	2.2	0.5	14.1	42
Indicated	0.2	830.3	0.57	0.18	0.153	0.347	0.082	2.09	103.1	0.004	0.010	0.002	0.061	9,498	3,039	3.7	8.4	2.0	50.5	171
	0.3	806.0	0.58	0.19	0.156	0.353	0.084	2.12	104.0	0.005	0.010	0.002	0.062	9,366	2,999	3.7	8.3	2.0	49.7	168
	0.4	713.0	0.61	0.20	0.166	0.376	0.089	2.22	106.2	0.005	0.011	0.003	0.065	8,713	2,781	3.5	7.8	1.9	46.2	151
	0.5	546.8	0.66	0.21	0.186	0.420	0.099	2.41	108.8	0.005	0.012	0.003	0.070	7,207	2,286	3.0	6.7	1.6	38.5	119
	0.6	379.2	0.71	0.22	0.205	0.461	0.108	2.60	111.0	0.006	0.013	0.003	0.076	5,354	1,699	2.3	5.1	1.2	28.7	84
Measured	0.2	1152.6	0.58	0.19	0.150	0.342	0.082	2.12	103.6	0.004	0.010	0.002	0.062	13,456	4,302	5.0	11.5	2.7	71.2	239
+	0.3	1112.7	0.60	0.19	0.153	0.349	0.084	2.16	104.7	0.004	0.010	0.002	0.063	13,242	4,237	5.0	11.3	2.7	70.0	233
Indicated	0.4	987.6	0.63	0.20	0.163	0.371	0.089	2.27	107.2	0.005	0.011	0.003	0.066	12,360	3,945	4.7	10.7	2.6	65.3	212
	0.5	783.6	0.67	0.21	0.180	0.409	0.097	2.44	110.0	0.005	0.012	0.003	0.071	10,512	3,342	4.1	9.3	2.2	55.7	172
	0.6	561.7	0.72	0.23	0.196	0.445	0.105	2.62	112.7	0.006	0.013	0.003	0.076	8,059	2,560	3.2	7.3	1.7	42.9	127
Inferred	0.2	458.4	0.50	0.16	0.143	0.325	0.073	1.83	109.6	0.004	0.009	0.002	0.053	4,593	1,494	1.9	4.3	1.0	24.5	100
	0.3	411.2	0.53	0.17	0.151	0.345	0.077	1.92	112.4	0.004	0.010	0.002	0.056	4,350	1,406	1.8	4.1	0.9	23.1	92
	0.4	325.7	0.58	0.18	0.172	0.387	0.086	2.09	114.4	0.005	0.011	0.003	0.061	3,745	1,192	1.6	3.7	0.8	19.8	74
	0.5	227.9	0.63	0.20	0.203	0.452	0.099	2.28	113.3	0.006	0.013	0.003	0.066	2,863	898	1.3	3.0	0.7	15.1	52
	0.6	125.5	0.69	0.21	0.247	0.545	0.118	2.51	111.1	0.007	0.016	0.003	0.073	1,735	535	0.9	2.0	0.4	9.2	28

Notes:

1. Mineral Resources in this table are a subset of the Mineral Resource estimates in Table 14-16 and are not additive to that estimate.
2. The Mineral Resource effective date is 4 February 2014. Dr. Harry Parker, RM SME, AMEC Consulting Geologist and Geostatistician, is the QP for the estimate and is a Professional Geologist licensed in Minnesota.
3. Mineral Resources are reported inclusive of Mineral Reserves and are reported on a 100% basis. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
4. The Mineral Resource estimates are based on a US$21.66/st NSR that in turn assumes a mining cost of $12.54/st, a process cost of $5.96/st and general and administrative charges of $3.16/st; global metallurgical recoveries of 93.4% (Cu), 63.9% (Ni), 78.2% (Au), 76.2% (Pd), 61.3% (Pt) and 66.9% (Ag); and long-term consensus metal prices of $3.30/lb Cu, $10.0/lb Ni, $1,350/troy oz Au, $850/troy oz Pd, $2,000/troy oz Pt, and $21.00/troy oz Ag.
5. The NSR equates to an approximate 0.3% Cu cutoff grade.
6. Figures have been rounded and may not sum.

Table 14-18: Maturi Southwest Mineral Resources by Copper Cutoff (basecase is highlighted)

Category	Cutoff (Cu %)	Short Tons (Mst)	Grade Cu (%)	Grade Ni (%)	Grade Pt (ppm)	Grade Pd (ppm)	Grade Au (ppm)	Grade Ag (ppm)	Grade Co (ppm)	Grade Pt (oz/st)	Grade Pd (oz/st)	Grade Au (oz/st)	Grade Ag (oz/st)	Contained Metal Cu (Mlb)	Contained Metal Ni (Mlb)	Contained Metal Pt (Moz)	Contained Metal Pd (Moz)	Contained Metal Au (Moz)	Contained Metal Ag (Moz)	Contained Metal Co (Mlb)
Indicated	0.2	131.1	0.43	0.15	0.071	0.164	0.042	1.42	103.1	0.002	0.005	0.001	0.041	1,118	394	0.3	0.6	0.2	5.4	27
	0.3	102.6	0.48	0.17	0.080	0.185	0.048	1.58	108.1	0.002	0.005	0.001	0.046	976	340	0.2	0.6	0.1	4.7	22
	0.4	71.4	0.53	0.18	0.093	0.217	0.055	1.77	112.3	0.003	0.006	0.002	0.052	757	260	0.2	0.5	0.1	3.7	16
	0.5	40.4	0.59	0.20	0.108	0.256	0.064	2.02	116.3	0.003	0.007	0.002	0.059	478	162	0.1	0.3	0.1	2.4	9
	0.6	15.9	0.67	0.22	0.124	0.294	0.071	2.28	120.0	0.004	0.009	0.002	0.066	211	71	0.1	0.1	0.0	1.1	4
Inferred	0.2	57.5	0.35	0.13	0.052	0.126	0.033	1.19	92.9	0.002	0.004	0.001	0.035	401	145	0.1	0.2	0.1	2.0	11
	0.3	32.3	0.43	0.15	0.065	0.157	0.041	1.43	102.2	0.002	0.005	0.001	0.042	281	97	0.1	0.1	0.0	1.3	7
	0.4	16.4	0.51	0.17	0.082	0.197	0.050	1.72	107.2	0.002	0.006	0.001	0.050	167	57	0.0	0.1	0.0	0.8	4
	0.5	7.2	0.60	0.20	0.102	0.251	0.063	2.11	113.7	0.003	0.007	0.002	0.062	86	29	0.0	0.1	0.0	0.4	2
	0.6	3.2	0.66	0.22	0.115	0.279	0.069	2.39	117.7	0.003	0.008	0.002	0.070	42	14	0.0	0.0	0.0	0.2	1

Notes:

1. The Mineral Resource estimate effective date is 15 June 2013. Dr. Harry Parker, RM SME, AMEC Consulting Geologist and Geostatistician, is the QP for the estimate and is a Professional Geologist licensed in Minnesota.
2. Mineral Resources are reported inclusive of Mineral Reserves and are reported on a 100% basis. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. The Mineral Resource estimates are based on a US$21.66/st NSR that in turn assumes a mining cost of $12.54/st, a process cost of $5.96/st and general and administrative charges of $3.16/st; global metallurgical recoveries of 93.4% (Cu), 63.9% (Ni), 78.2% (Au), 76.2% (Pd), 61.3% (Pt) and 66.9% (Ag); and long-term consensus metal prices of $3.30/lb Cu, $10.0/lb Ni, $1,350/troy oz Au, $850/troy oz Pd, $2,000/troy oz Pt, and $21.00/troy oz Ag.
4. The NSR equates to an approximate 0.3% Cu cutoff grade.
5. Figures have been rounded and may not sum.

Table 14-19: Maturi Southwest S3+S2 Mineral Resources by Copper Cutoff (basecase is highlighted)

Category	Cutoff (Cu %)	Short Tons (Mst)	Grade Cu (%)	Grade Ni (%)	Grade Pt (ppm)	Grade Pd (ppm)	Grade Au (ppm)	Grade Ag (ppm)	Grade Co (ppm)	Grade Pt (oz/st)	Grade Pd (oz/st)	Grade Au (oz/st)	Grade Ag (oz/st)	Contained Metal Cu (Mlb)	Contained Metal Ni (Mlb)	Contained Metal Pt (Moz)	Contained Metal Pd (Moz)	Contained Metal Au (Moz)	Contained Metal Ag (Moz)	Contained Metal Co (Mlb)
Indicated	0.2	116.5	0.45	0.16	0.075	0.174	0.045	1.49	105.9	0.002	0.005	0.001	0.043	1,048	368	0.3	0.6	0.2	5.1	25
	0.3	101.6	0.48	0.17	0.080	0.186	0.048	1.58	108.3	0.002	0.005	0.001	0.046	969	338	0.2	0.6	0.1	4.7	22
	0.4	71.3	0.53	0.18	0.093	0.217	0.055	1.77	112.4	0.003	0.006	0.002	0.052	756	260	0.2	0.5	0.1	3.7	16
	0.5	40.4	0.59	0.20	0.108	0.256	0.064	2.02	116.3	0.003	0.007	0.002	0.059	478	162	0.1	0.3	0.1	2.4	9
	0.6	15.9	0.67	0.22	0.124	0.294	0.071	2.28	120.0	0.004	0.009	0.002	0.066	211	71	0.1	0.1	0.0	1.1	4
Inferred	0.2	36.2	0.41	0.14	0.062	0.149	0.039	1.36	100.3	0.002	0.004	0.001	0.040	300	104	0.1	0.2	0.0	1.4	7
	0.3	32.0	0.44	0.15	0.065	0.157	0.041	1.43	102.4	0.002	0.005	0.001	0.042	279	96	0.1	0.1	0.0	1.3	7
	0.4	16.4	0.51	0.17	0.082	0.197	0.050	1.72	107.2	0.002	0.006	0.001	0.050	167	57	0.0	0.1	0.0	0.8	4
	0.5	7.2	0.60	0.20	0.102	0.251	0.063	2.11	113.7	0.003	0.007	0.002	0.062	86	29	0.0	0.1	0.0	0.4	2
	0.6	3.2	0.66	0.22	0.115	0.279	0.069	2.39	117.7	0.003	0.008	0.002	0.070	42	14	0.0	0.0	0.0	0.2	1

Notes:

1. Mineral Resources in this table are a subset of the Mineral Resource estimates in Table 14-18 and are not additive to that estimate.
2. The Mineral Resources effective date is 15 June 2013. Dr. Harry Parker, RM SME, AMEC Consulting Geologist and Geostatistician is the QP for the estimate and is a Professional Geologist licensed in Minnesota.
3. Mineral Resources are reported inclusive of Mineral Reserves and are reported on a 100% basis. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
4. The Mineral Resource estimates are based on a US$21.66/st NSR that in turn assumes a mining cost of $12.54/st, a process cost of $5.96/st and general and administrative charges of $3.16/st; global metallurgical recoveries of 93.4% (Cu), 63.9% (Ni), 78.2% (Au), 76.2% (Pd), 61.3% (Pt) and 66.9% (Ag); and long-term consensus metal prices of $3.30/lb Cu, $10.0/lb Ni, $1,350/troy oz Au, $850/troy oz Pd, $2,000/troy oz Pt, and $21.00/troy oz Ag
5. The NSR equates to an approximate 0.3% Cu cutoff grade.
6. Figures have been rounded and may not sum.

14.9.3 Birch Lake

Table 14-20 tabulates the Indicated and Inferred Mineral Resources for Birch Lake by cumulative copper cutoffs to show sensitivity of the estimate to variations in cutoff grade. The basecase, 0.30% Cu, is gray-shaded. The Indicated and Inferred Mineral Resources are stated in million short tons (Mst).

At Birch Lake, the Mineral Resources are located at least 600 ft below the surface. AMEC considers that depth sufficient to not require additional allowances for a safety pillar.

14.9.4 Spruce Road

Table 14-21 summarizes the mineral resources at Spruce Road. AMEC assumed a 164 ft (50 m) safety pillar. All blocks below the safety pillar with a NSR value of $30/st, or more, were tabulated as an Inferred Mineral Resource.

14.10 Targets for Additional Exploration

Canadian disclosure standards under NI 43-101 allow the estimated quantities of a target for additional exploration to be disclosed as a range of tons and grade.

AMEC cautions that the potential quantity and grade are conceptual in nature, and that there has been insufficient exploration to define the exploration targets as a Mineral Resource. It is uncertain if additional exploration will result in the target(s) being delineated as a Mineral Resource.

14.10.1 Maturi

The area inside the Maturi model perimeter surrounding the Indicated and Inferred Mineral Resources was divided into two targets for additional exploration, Maturi North and Maturi South (Figure 14-13).

For the Maturi North and South targets, grade and tonnage ranges were based on estimated blocks within the model that were not classified as either Indicated or Inferred.

Table 14-20: Birch Lake Mineral Resources by Copper Cutoff (basecase is highlighted)

Category	Cutoff (Cu %)	Short Tons (Mst)	Grade Cu (%)	Grade Ni (%)	Grade Pt (ppm)	Grade Pd (ppm)	Grade Au (ppm)	Grade Pt (oz/st)	Grade Pd (oz/st)	Grade Au (oz/st)	Contained Metal Cu (Mlb)	Contained Metal Ni (Mlb)	Contained Metal Pt (Moz)	Contained Metal Pd (Moz)	Contained Metal Au (Moz)
Indicated	0.2	111.9	0.49	0.15	0.220	0.481	0.108	0.006	0.014	0.003	1,097	342	0.7	1.6	0.4
	0.3	99.7	0.52	0.16	0.235	0.515	0.115	0.007	0.015	0.003	1,037	319	0.7	1.5	0.3
	0.4	85.4	0.55	0.17	0.248	0.543	0.120	0.007	0.016	0.004	936	287	0.6	1.4	0.3
	0.5	54.9	0.60	0.18	0.269	0.591	0.130	0.008	0.017	0.004	658	200	0.4	0.9	0.2
	0.6	22.8	0.67	0.21	0.285	0.630	0.140	0.008	0.018	0.004	307	94	0.2	0.4	0.1
Inferred	0.2	313.0	0.41	0.13	0.156	0.320	0.076	0.005	0.009	0.002	2,560	839	1.4	2.9	0.7
	0.3	239.0	0.46	0.15	0.180	0.370	0.087	0.005	0.011	0.003	2,189	707	1.3	2.6	0.6
	0.4	158.0	0.51	0.16	0.203	0.423	0.098	0.006	0.012	0.003	1,621	512	0.9	1.9	0.5
	0.5	77.0	0.58	0.18	0.228	0.480	0.111	0.007	0.014	0.003	895	279	0.5	1.1	0.2
	0.6	23.0	0.66	0.20	0.274	0.569	0.131	0.008	0.017	0.004	305	94	0.2	0.4	0.1

Notes:

1. Mineral Resource effective date is 15 September 2012. Dr. Harry Parker, RM SME, AMEC Consulting Geologist and Geostatistician, is the QP for the estimate and is a Professional Geologist licensed in Minnesota.
2. Mineral Resources are reported on a 100% basis.
3. The Mineral Resources estimates are based on a US$30/st NSR that in turn assumes a mining cost of $16/st, a process cost of $12/st and general and administrative charges of $2/st; global metallurgical recoveries of 90.8% (Cu), 57.4% (Ni), 63.3% (Au), 63.6% (Pd) and 55.2% (Pt); and long-term consensus metal prices of $3.00/lb Cu, $9.38/lb Ni, $1,050/troy oz Au, $805/troy oz Pd and $1,840/troy oz Pt.
4. The NSR equates to an approximate 0.3% Cu cutoff grade.
5. Figures have been rounded and may not sum.

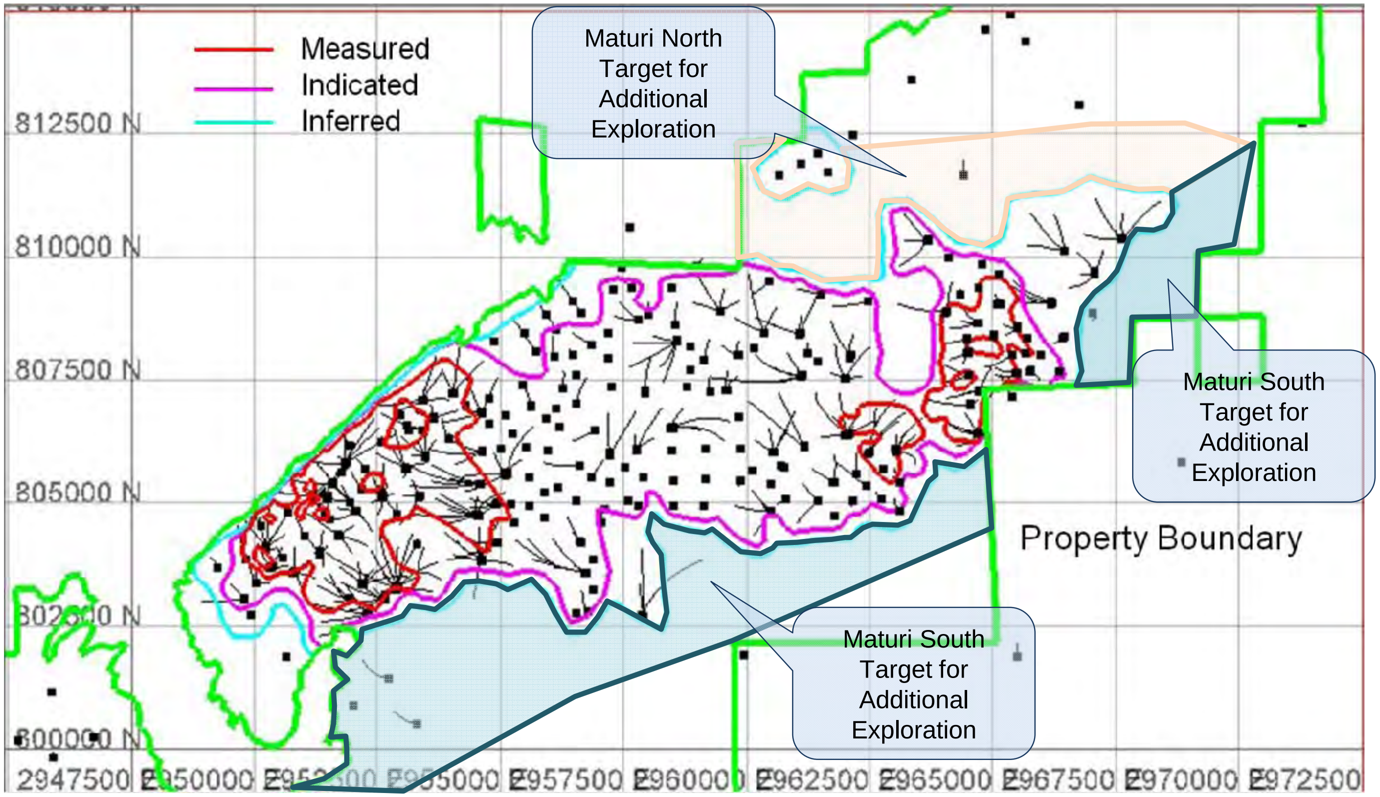
Table 14-21: Spruce Road Mineral Resources by Copper Cutoff (basecase is highlighted)

Category	Cutoff (Cu %)	Short Tons (Mst)	Cu (%)	Ni (%)	Contained Metal Cu (Mlb)	Contained Metal Ni (Mlb)
Inferred	0.2	674	0.38	0.14	5,122	1,887
	0.3	480	0.43	0.16	4,128	1,536
	0.4	254	0.50	0.18	2,540	914
	0.5	101	0.57	0.21	1,151	424
	0.6	24	0.66	0.24	317	115

Notes:

1. The Mineral Resource estimate effective date is 15 September 2012. Dr. Harry Parker, RM SME, AMEC Consulting Geologist and Geostatistician, is the QP for the estimate and is a Professional Geologist licensed in Minnesota.
2. Mineral Resources are reported on a 100% basis.
3. The Mineral Resource estimates are based on a US$30/st NSR that in turn assumes a mining cost of $16/st, a process cost of $12/st and general and administrative charges of $2/st; global metallurgical recoveries of 90.8% (Cu), 68.8% (Ni); and long-term consensus metal prices of $3.00/lb Cu, and $9.38/lb Ni.
4. The NSR equates to a 0.3% Cu cutoff grade.
5. Figures have been rounded and may not sum.

Figure 14-13: Maturi Targets for Additional Exploration



Note: Figure prepared by AMEC, 2014.

Tonnage and grades of the Maturi North target for additional exploration could range from 290 to 435 Mst grading 0.41 to 0.61% Cu, 0.14 to 0.21% Ni, 0.10 to 0.15 ppm Pt, 0.23 to 0.34 ppm Pd and 0.05 to 0.08 ppm Au.

Tonnage and grades of the Maturi South target for additional exploration could range from 330 to 500 Mst grading 0.42 to 0.62% Cu, 0.13 to 0.19% Ni, 0.14 to 0.21 ppm Pt, 0.31 to 0.46 ppm Pd and 0.07 to 0.10 ppm Au.

14.10.2 Maturi Southwest

The target for additional exploration within the Maturi Southwest permit boundaries is calculated based on results from 11 drill holes. Of those, seven are legacy holes and four are recent TMM holes. These holes are shown in Figure 14-14.

Internal to the Maturi Southwest target for additional exploration is a low-grade area (below 0.30 % copper); this area was excluded from the target estimates. The southern boundary of the Maturi Southwest target for additional exploration was truncated against the target for additional exploration identified in the Birch Lake area.

The tonnage and grades of the Maturi Southwest target for additional exploration could range from 500 to 825 Mst (million short tons) grading 0.43 to 0.55% Cu, 0.14 to 0.18% Ni, 0.08 to 0.10 ppm Pt, 0.17 to 0.22 ppm Pd and 0.046 to 0.053 ppm Au.

14.10.3 Birch Lake

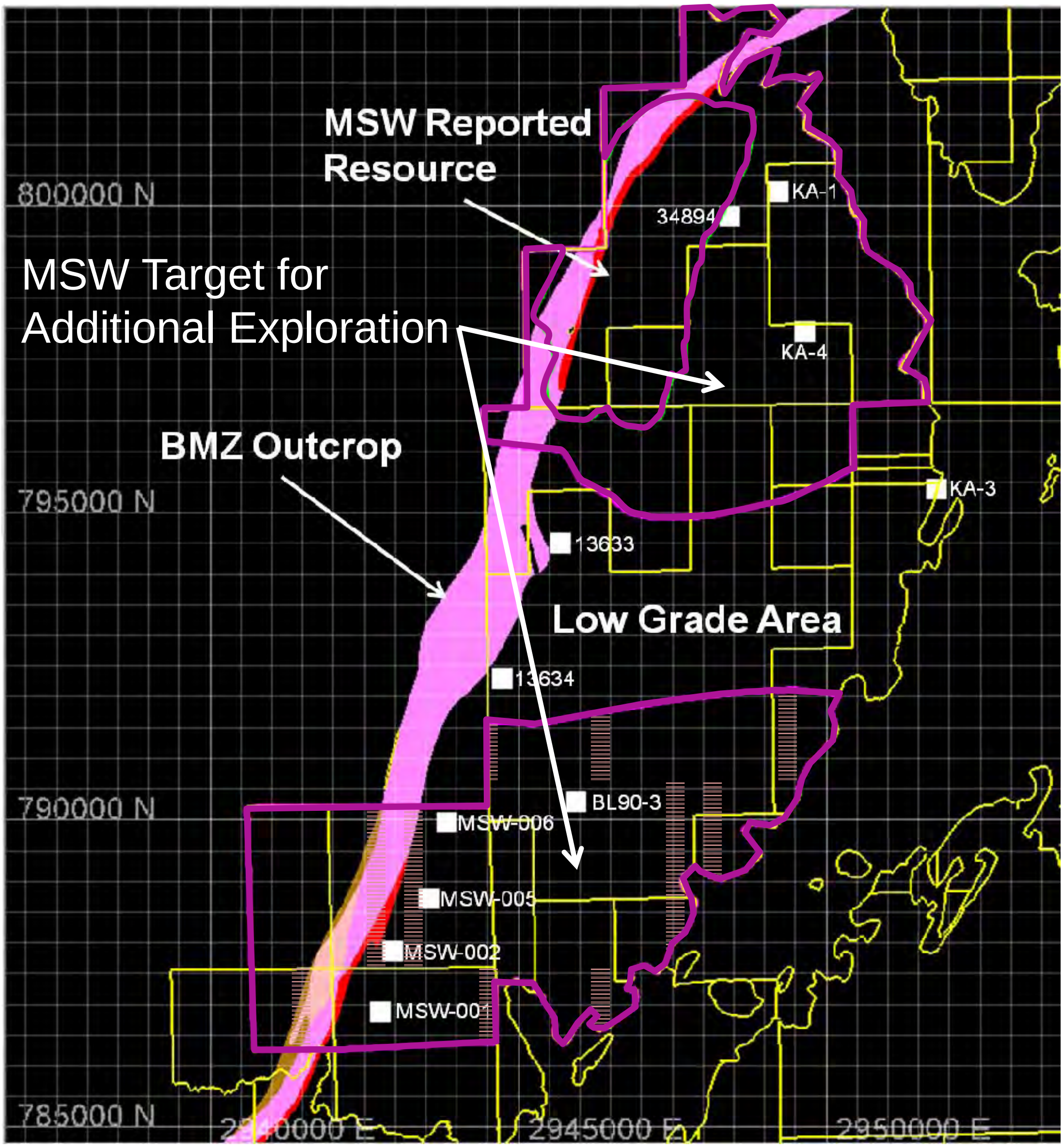
At Birch Lake, blocks with extrapolated grades outside the area classified as Inferred are considered to be targets for additional exploration. Figure 14-15 shows the location of the Indicated and Inferred Mineral Resources and the target for additional exploration.

The target for additional exploration is in the range of 220 to 330 Mst and may contain 0.33 to 0.50% Cu, 0.11 to 0.16% Ni, 0.11 to 0.16 ppm Pt, 0.22 to 0.33 ppm Pd, and 0.05 to 0.08 ppm Au.

14.10.4 Spruce Road

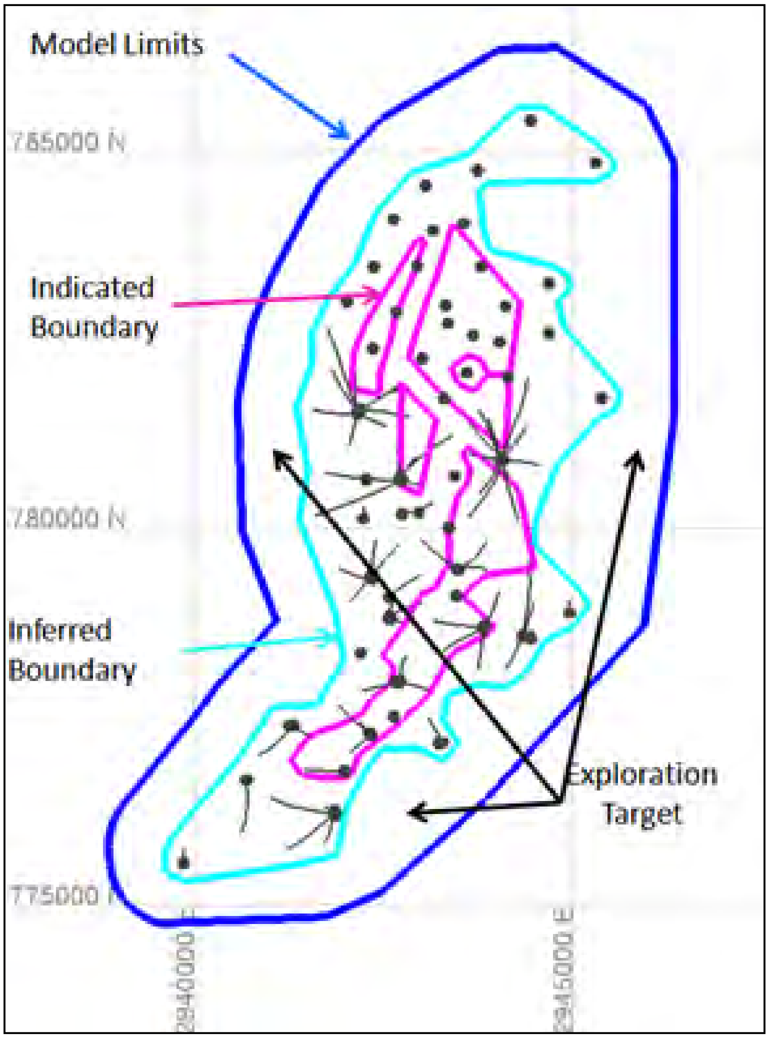
No targets for additional exploration were identified at Spruce Road.

Figure 14-14:Maturi Southwest Target for Additional Exploration



Note: Figure prepared by AMEC, 2014.

Figure 14-15: Birch Lake Target for Additional Exploration



Note: Figure prepared by AMEC, 2014.

14.11 Comment on Section 14

14.11.1 Maturi

The Maturi deposit is currently classified as Measured, Indicated, and Inferred Mineral Resources. Additional drilling will be required to support conversion of material currently classified as Indicated to Measured Mineral Resources. Additional mineralization is likely to be encountered by drilling down dip and along strike. Although no specific drill plan is proposed, generally, "five-spotting" the current drill pattern is likely to support conversion of Indicated to Measured Mineral Resources.

Conversion of remaining Inferred Mineral Resources to higher confidence classes may require significant amounts of drilling.

14.11.2 Maturi Southwest

Maturi Southwest is currently classified as Indicated and Inferred Mineral Resources. Limits of the mineralization down dip and along strike have not yet been defined, but additional mineralization is likely to be encountered by drilling down dip and along strike. Potential conversion of Indicated to Measured Mineral Resources would require, at a minimum, "five-spotting" the current drill pattern. The mineralization is generally lower grade than Maturi, but, like Maturi, it is very continuous over long distances.

14.11.3 Birch Lake

Birch Lake is currently classified as Indicated and Inferred Mineral Resources. Significant issues that affect the Mineral Resource classification at Birch Lake are the faults which potentially will affect underground mining operations and the location of the magma channel that controls the higher grades and greater thicknesses of mineralization.

14.11.4 Spruce Road

AMEC believes that the SWRPA estimate is adequate for a preliminary resource estimate. Re-tabulation of the results of the estimate was utilized for this Report. This estimate is based on largely unverified data. SWRPA verified the data as well as those data can be verified; however, the lack of original collar and down-hole surveys, assay certificates, and drill logs is detrimental to the Project, and the lack of verifiable information can only be resolved by drilling at least 10% twin holes to verify the data.

As Duluth is focusing on the evaluation of the better explored Maturi deposit, no additional drilling is planned for Spruce Road in the near term.

15.0 MINERAL RESERVE ESTIMATES

15.1 Introduction

The PFS assumes that the Maturi and Maturi Southwest deposits will be mined. The PFS does not consider mining the Spruce Road and Birch Lake deposits.

Measured and Indicated Mineral Resources were converted to Mineral Reserves by applying appropriate mining dilution and recovery factors to the triangulations that were created during the mine design stage. The undiluted tonnage and grade of each triangulation is based on the block model that was provided to SRK by AMEC. All Mineral Reserve tonnages are expressed as "dry" tons (i.e., no moisture) and are based on the density values stored in the block model.

While some triangulations consist entirely of Measured and Indicated Mineral Resources, other triangulations may include small amounts of Inferred Mineral Resources and unclassified material. Where Inferred and unclassified material has been included in a triangulation, such material has been assigned a grade of zero.

Where appropriate, a "development allowance" was applied to certain types of triangulations to account for re-muck bays, fan cut-outs, etc. In some cases this development allowance was in ore; however this amount was negligible.

The Maturi and Maturi Southwest deposits are planned to be mined using a combination of two mining methods. These were selected because they were able to produce at a high throughput rate and had the ability to be adjusted to the specific geometries (dip and thickness) of the deposits:

- Post-pillar cut-and-fill is a man-entry mining method. It recovers the ore in horizontal slices, starting from a bottom level and advancing upwards. A level will be extracted by developing a horizontal slot[3] (room) from footwall to hanging wall, followed by cross-cuts perpendicular in both directions from the slot, which are mined on retreat. Unmined pillars will remain between the slots to provide local geomechanical stability. After the slot and cross-cuts have been extracted, a bulkhead will be installed and the mined-out area will be backfilled. Mining will continue with a new level mined immediately above the backfilled level. Pillars typically extend vertically through several levels. The pillars have been designed to yield underneath working levels where they are confined by backfill.
- Long-hole stoping is a traditional blast hole stoping method where extraction and drilling drifts will be developed within the orebody. A slot raise will be mined between the drilling and extraction drifts to create a void for blasted material. Ore will be drilled from either the drilling (upper) drift or extraction (lower) drift, then

[3] A horizontal opening driven in ore and perpendicular to strike in a post-pillar cut-and-fill stope

loaded with explosives, and blasted towards the slot raise. Broken ore will be mucked both manually and remotely from the extraction drift. After a stope has been mined out, it will be backfilled with low-strength paste backfill. Stope walls will not be vertical but rather will be angled at 45° to create a diamond shape stope. This will allow for the use of lower-strength fill material, which will be engineered to stand at a 45° angle, and will conform the stope shape to the dip of the deposit. Stopes will be mined from the bottom of the panel upward.

15.2 Geomechanical Considerations

Geomechanical input was provided by Itasca and Golder. A summary of this work is included as Section 16.1.9 to 16.1.10.

15.3 Hydrogeological Considerations

Hydrogeological design guidelines were provided by Itasca. Additional details on the hydrogeological evaluation are included in Section 16.2.

15.4 Throughput Rate and Supporting Assumptions

Trade-off studies were completed to determine the most appropriate throughput rate for the mine plan, and a throughput capacity of 50,000 st/d of ore was selected for the PFS, which supported a life-of-mine (LOM) production plan of approximately 30 years.

15.5 Net Smelter Return

For the purposes of mine design, a net smelter return (NSR) calculation was used that takes into account revenue for five elements (Cu, Ni, Au, Pd, and Pt). Plant recoveries assumed in the NSR equation were based on current testwork for concentrate production. The NSR was evaluated for each block in the block model. The estimated marginal cutoff grade for the mine plan is an NSR of $25/st.

Applying the $25.00 NSR cutoff at the Mineral Reserves reporting and mine scheduling stages ensures that no sub-economic stopes are included in the production schedule. It also ensures that development mining is counted as ore whenever such material meets the $25.00 NSR cutoff. That $25.00 NSR value was selected based estimated average LOM operating cost for the Project of $23.53/st.

Table 15-1 summarizes NSR input assumptions. Table 15-2 presents the operating costs assumptions used in the NSR calculations. Table 15-3 shows an example NSR calculation for an individual block.

Table 15-1: Mine Design NSR Parameters

Parameter	Cu	Ni	Au	Pd	Pt
Recovery (%)	94.0	60.8	82.3	36.1	42.5
Payable (%) *	76.4	70.8	45.0	68.6	69.3
Price	US$3.00/lb	US$9.50/lb	US$1,200/oz	US$700/oz	US$1,650/oz

Note: * = includes refining costs.

Table 15-2: Operating Costs Used for Mineral Reserves NSR Cutoff

Item	Estimated Costs (US$/st)
Mining	13.80
Processing	5.02
Paste backfill	1.28
Water management	0.21
Tailings	0.06
G&A	2.44
Technical services	0.45
Financial assurance	0.27
Total	*23.53*

Table 15-3: Example Mine Design NSR Block Calculation

	Volume	Density	Cu (%)	Ni (%)	Au (oz/st)	Pd (oz/st)	Pt (oz/st)
Input Block	9,000	0.09296	0.56	0.35	0.006	0.005	0.005
Calculate Contained Metal							
Cu	9,370	lbs					
Ni	5,856	lbs					
Au	5	oz					
Pd	4	oz					
Pt	4	oz					
Calculate saleable metal - discount by recovery and payability							
Cu	8,508	lbs					
Ni	4,029	lbs					
Au	3	oz					
Pd	2	oz					
Pt	2	oz					
Calculate block dollar value for each metal - subtracting refining charges							
Cu	23,398	US$					
Ni	34,249	US$					
Au	2,796	US$					
Pd	1,490	US$					
Pt	3,048	US$					
Total block value	*64,981*	*US$*					
Total block value per short ton	*77.70*	*US$/st*					

15.6 Mine Design

The completed mine design is illustrated in Figure 15-1.

15.6.1 Maturi

The orebody typically dips to the southeast at approximately 35º. Areas less than ~2,400 ft in depth are generally the steepest dipping, averaging approximately 40º and locally up to 60º. Below this to a depth of approximately 2,700 ft from surface the mineralization flattens to approximately 20º. Thereafter at depth, the mineralization steepens again to approximately 30º. As a result, the deposit was subdivided into four tiers to address geometric characteristics and productivity opportunities.

Tier 1 (400–1,200 ft below surface, where the 400 ft figure represents the base of the crown pillar) will be mined using post-pillar cut-and-fill methods. Tier 1 supports most of the ramp-up period, and also carries higher copper and nickel grades. Slots are designed at 46 ft width by 40 ft height, with square pillars of 34 ft width by 40 ft height. The 46 ft slot width and 34 ft pillar width are at the outer bounds of allowable dimensions and additional analysis and evaluation are required..

Tier 2 (1,200–2,500 ft depth below surface), will be mined using long-hole stoping. This method takes advantage of the dip in this area and is the less expensive mining method. Stopes are designed at 150 ft along strike and will be separated by 50 ft panel rib pillars[4].

Tier 3 sits in the flatter portion of the deposit at a depth of 2,500 ft to 3,000 ft below surface. In this area, a post-pillar cut-and-fill method was chosen, but was modified in comparison to Tier 1 to account for a higher stress condition at depth. This resulted in the design of smaller slots, 26 ft wide by 20 ft height and 20 ft wide by 20 ft high square pillars.

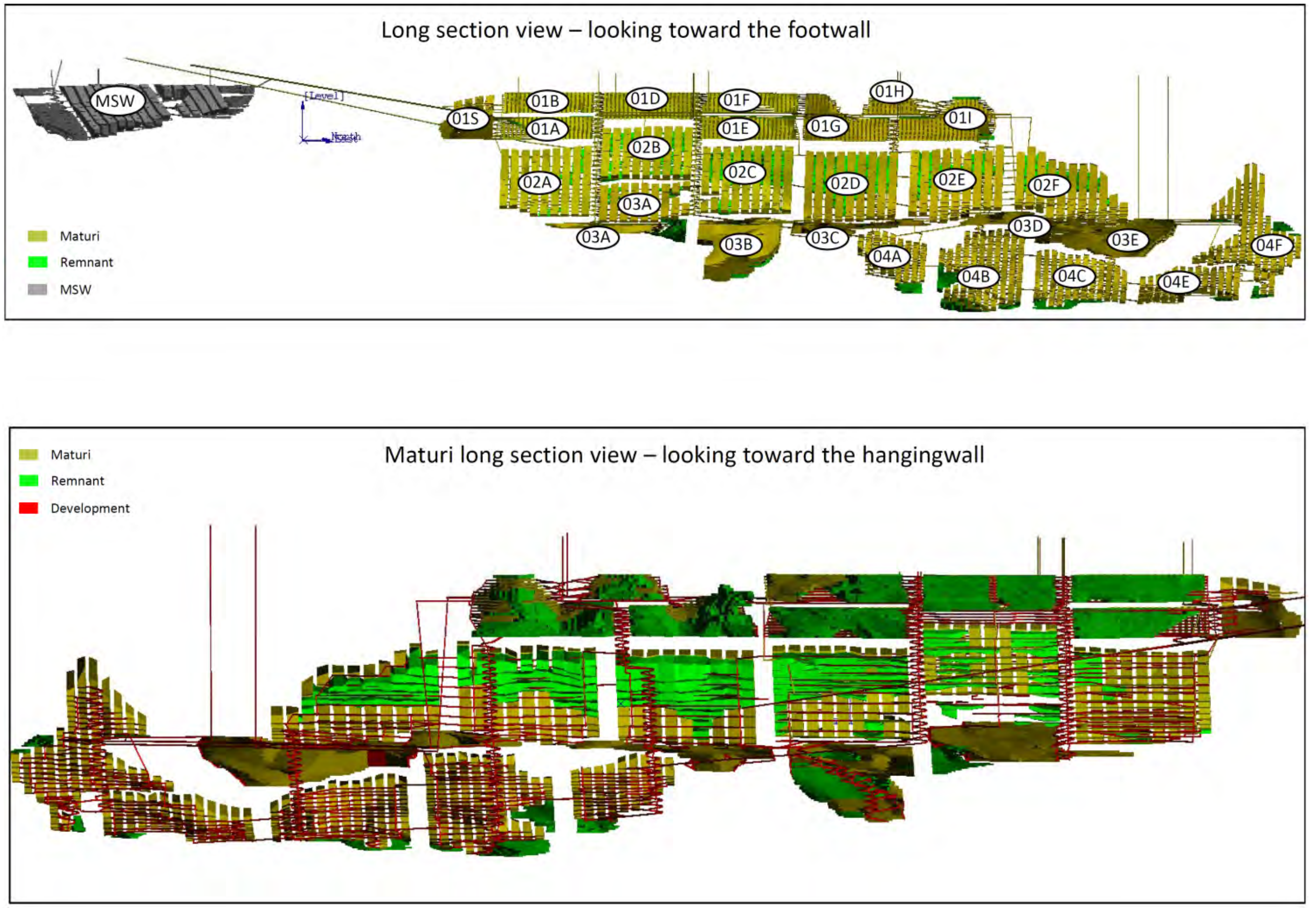
Tier 4 is at a depth greater than 3,000 ft and is again at a steeper angle. Long-hole stoping is selected with stope size modified for depth. The stopes are 100 ft wide along strike and separated by 50 ft panel rib pillars.

15.6.2 Maturi Southwest

The Maturi Southwest deposit covers a smaller area, though is similar in dip and orientation to Maturi. The mining methods planned to be used in Maturi Southwest are the same as those used in Tier 2 long-hole stoping and Tier 3 post-pillar cut-and-fill types from Maturi.

[4] A dip pillar within a long-hole stoping panel which separates stopes

Figure 15-1: Completed Mine Design



Note: figure prepared SRK, 2014. In the top figure, numbers starting with 01 = Tier 1; 02 = Tier 2; 03 = Tier 3 and 04 is Tier 4. Figures are schematic and not to scale.

15.7 Dilution and Mine Losses

Ore dilution and mining recovery was calculated based on detailed designs of the mining areas and recommendations in regards to paste and hanging wall dilution.

15.7.1 Post-Pillar Cut-and-Fill

The post-pillar cut-and-fill mining dilution and recovery factors were developed by reviewing the mine designs for multiple mining panels and selecting representative cross-sections for evaluation.

Primary dilution is expected from the backfill on the level below and from the hanging wall contact. Hanging wall dilution of 0.5 ft at zero grade was used, and the slot configuration along the hanging wall included minimizing the mining height for one round. Backfill dilution from the stope below was assumed at 0.5 ft. The overall dilution for the post-pillar cut-and-fill design was determined, depending on the area, to range between 3% and 4%.

Unrecoverable material is expected to occur in the transition from the footwall to the slot, along the hanging wall contact, and from the backfill in the slot below. One ft of loss was assumed in the backfill. The overall recovery of ore inside the design not considering loss of pillars and other losses outside the design is 92–95%, depending on area.

15.7.2 Long-Hole Stoping

The long-hole stoping mining dilution and recovery factors were developed by reviewing the stope designs for multiple mining panels, and selecting a representative cross-section for evaluation.

The two zones of primary dilution will be the hanging wall and the backfill material. A hanging wall overbreak of 3 ft was used over the entire exposed stope face. Backfill dilution from the stope below was assumed at 2 ft.

The overall volumetric dilution for the long-hole stopes was determined to be 5% using a zero grade. This number is lower than typically expected for the long-hole stoping method; however, with the wide orebody thickness, good rock mass quality and specific drill patterns modified to address the hanging wall dilution/stresses, this is considered to be a reasonable assumption for the PFS.

Mining recovery factors were calculated for the recovery inside the stope and do not consider leaving pillars or ore losses outside the mined stope area, since only planned mined volumes are adjusted for losses. Recovery calculations were modeled by a methodology similar to the dilution calculation using the typical stope geometry. Unrecoverable material is expected to occur on the 45º footwall, assumed as a 3 ft loss, and on the 45º backfill wall, assumed as a 2 ft loss. The overall recovery of ore inside the planned stopes, not considering loss of pillars and other losses outside the stopes, is 95%.

15.8 Ventilation

Ventilation is discussed in detail in Section 16-4.

15.9 Surface Topography

Topography was obtained from the Minnesota DNR LiDAR data, and is contoured at 2 ft intervals.

15.10 Mineral Reserves Statement

Mineral Reserves were classified using the 2014 CIM Definition Standards. The QP for the estimate is Ms. Joanna Poeck, RM SME of SRK. Mineral Reserves are as summarized in Table 15-4.

15.11 Factors That May Affect the Mineral Reserve Estimate

Factors that may affect the Mineral Reserve estimates include:

- Metal price and exchange rate assumptions
- Assumptions relating to geomechanical and hydrogeological parameters used in mine design
- Assumptions that go into defining the NSR cutoff used to constrain Mineral Reserves
- Appropriate dilution control being able to be maintained
- Assumptions as to the paste backfill strengths and quantities required
- Mining and metallurgical recovery assumptions
- Changes to capital and operating cost estimates
- Changes to royalty payment assumptions
- Variations to the permitting, operating or social license regime assumptions.

Table 15-4: Mineral Reserves Statement

Deposit	Classification	Tons (Mst)	Cu (%)	Ni (%)	Pt (ppm)	Pd (ppm)	Au (ppm)	Ag (ppm)	Pt (oz/st)	Pd (oz/st)	Au (oz/st)	Ag (oz/st)
Maturi	Proven	130	0.65	0.21	0.155	0.344	0.092	2.31	0.004	0.010	0.003	0.067
	Probable	354	0.59	0.19	0.158	0.371	0.096	2.16	0.005	0.011	0.003	0.063
	Combined Proven and Probable	*484*	*0.60*	*0.19*	*0.159*	*0.373*	*0.090*	*2.20*	*0.005*	*0.011*	*0.003*	*0.064*
Maturi Southwest	Proven	0	0.00	0.00	0.000	0.000	0.000	0.00				
	Probable	43	0.48	0.17	0.069	0.206	0.034	1.61	0.002	0.006	0.001	0.047
	Combined Proven and Probable	*43*	*0.48*	*0.17*	*0.069*	*0.206*	*0.034*	*1.61*	*0.002*	*0.006*	*0.001*	*0.047*
Total Maturi and Maturi Southwest	Proven	130	0.65	0.21	0.155	0.344	0.092	2.31	0.004	0.010	0.003	0.067
	Probable	397	0.58	0.19	0.148	0.353	0.089	2.10	0.004	0.010	0.003	0.061
	Total Combined Proven and Probable	*527*	*0.59*	*0.19*	*0.154*	*0.350*	*0.090*	*2.15*	*0.004*	*0.010*	*0.002*	*0.063*

Area	Classification	Tons (Mst)	Contained Cu (Blbs)	Contained Ni (Blbs)	Contained Pt (Moz)	Contained Pd (Moz)	Contained Au (Moz)	Contained Ag (Moz)
Maturi	Proven	130	1.7	0.5	0.6	1.3	0.3	8.8
	Probable	354	4.2	1.3	1.6	3.8	1.0	22.3
	Combined Proven and Probable	*484*	*5.8*	*1.9*	*2.2*	*5.1*	*1.3*	*31.1*
Maturi Southwest	Proven	0	0.0	0.0	0.0	0.0	0.0	0.0
	Probable	43	0.4	0.1	0.1	0.3	0.0	2.0
	Combined Proven and Probable	*43*	*0.4*	*0.1*	*0.1*	*0.3*	*0.0*	*2.0*
Total Maturi and Maturi Southwest	Proven	130	1.7	0.5	0.6	1.3	0.3	8.8
	Probable	397	4.6	1.5	1.7	4.1	1.0	24.3
	Total Combined Proven and Probable	*527*	*6.2*	*2.0*	*2.4*	*5.4*	*1.3*	*33.1*

Notes to Accompany Mineral Reserves Table

1. The Qualified Person for the Mineral Reserve estimate is Ms. Joanna Poeck, an employee of SRK Consulting (U.S.), Inc. Mineral Reserves have an effective date of 1 July, 2014.
2. Mineral Reserves are contained within mine designs based on Measured and Indicated Mineral Resources, and assume a mining rate of 50,000 st/d of ore over a 30 year mine life. Underground mining will utilize conventional post-pillar cut-and-fill and long-hole open stoping methods. Paste backfill will be employed. The mine plan includes the mining of remnant ore, which is ore that is above the marginal cutoff grade, but is left behind during the first pass mining of higher-grade material.
3. Mineral Reserves are contained within Measured and Indicated mine designs using the following net smelter return (NSR) calculation inputs. Recovery assumptions used in the calculations were 94.0% for Cu, 60.8% for Ni, 82.3% for Au, 36.1% for Pd and 42.5% for Pt. Payability assumptions were 76.4% for Cu, 70.8% for Ni, 45% for Au, 68.6% for Pd and 69.3% for Pt. Metal price assumptions were US$3.00/lb for Cu, US$9.50/lb for Ni, US$1,200/oz for Au, US$700/oz for Pd and US$1,650/oz for Pt. Operating cost assumptions used in the NSR equations total $23.53/st mined and include mining costs of $13.80/st, process costs of $5.02/st, paste backfill costs of $1.28/st, water management costs of $0.21/st, tailings costs of $0.06/st, general and administrative costs of $2.44/st; technical services costs of $0.45/st and financial assurance costs of $0.27/st.
4. Mineral Reserves are reported using an NSR cutoff of $US25.00/st.
5. Mineral Reserves are reported according to CIM Definition Standards for Mineral Resources and Mineral Reserves (May 10, 2014).
6. Mineralization that was either not classified or assigned to the Inferred Mineral Resource category was set to waste within the above NSR cutoff mining shapes. Mine design incorporates geotechnical and hydrogeological considerations that take into account paste and hanging wall dilution. Dilution is allocated in the mine design based on the mining method, and ranges from 3–5%. Recovery of the planned mine design is assumed at 95%.
7. Tonnage figures are reported as million US short tons (st); grade figures as parts per million (ppm) or percent (%); contained copper and nickel are reported in billion pounds (B lb), contained platinum, palladium, gold and silver are reported in million troy ounces (M oz). Contained metal is reported as in situ metal content and does not include any adjustments for recoverability.
8. Rounding as required by reporting guidelines may result in apparent summation differences between US short tons, grade and contained metal content

16.0 MINING METHODS

16.1 Geomechanical Considerations

16.1.1 Basis of Geomechanical Considerations

The modeling results presented in this section represent interpretations and results of work completed to date. Additional data collection, testwork and analysis will be required in support of more detailed studies.

16.1.2 Rock Mass Strength Parameters

Rock mass strength was estimated stochastically for all domains using the geologic strength index (GSI) and Hoek-Brown failure criterion for each combination of tier and domain and select combinations of tier and sub-domain. These estimates were derived from the following parameters:

- Intact rock strength (from point load and laboratory testing)
- Joint characterization (from geotechnical core logging)
- Joint orientation (from ATV logging)
- Fracture frequency (from exploration drill hole logging)
- Intact rock material constant m_i (derived from laboratory test results).

16.1.3 Rock Mass Quality and Strength—Maturi

The Maturi rock mass has been characterized for each of three strength domains and seven strength sub-domains in four mining tiers. A decrease in overall fracture frequency is strongly correlated with depth.

Uniaxial compressive strength (UCSi) measurements have been conducted in the laboratory on 134 samples from Maturi. Typical UCSi values based on the 30th percentiles of large scale domains range from 124 to 181 MPa, and the 30th percentile GSI values range from 73 to 98.

16.1.4 Rock Mass Quality and Strength—Maturi Southwest

Point load testing in the Maturi Southwest deposit indicates that typical 30th percentile UCSi values range from 123 to 156 MPa and 30th percentile GSI values range from 65 to 76 on a large scale domain basis.

16.1.5 In Situ Stresses

16.1.5.1 Maturi

The estimate of the orientation of Maturi in situ stress is based on borehole breakouts found in 24 boreholes and upon remote Sigra testing (overcoring method) completed by Agapito Associates, Inc. (Agapito).

Horizontal in situ stresses are approximately two to 2.5 times the vertical stress. This stress regime is expected to lead to fairly significant shear stresses in the plane of the orebody.

16.1.5.2 Maturi Southwest

No stress measurements have been conducted at Maturi Southwest. For the purposes of the PFS, the same stress regime as measured at Maturi was used, i.e. two to 2.5 times horizontal to vertical in-situ stresses.

16.1.6 Regional (Barrier) Pillar

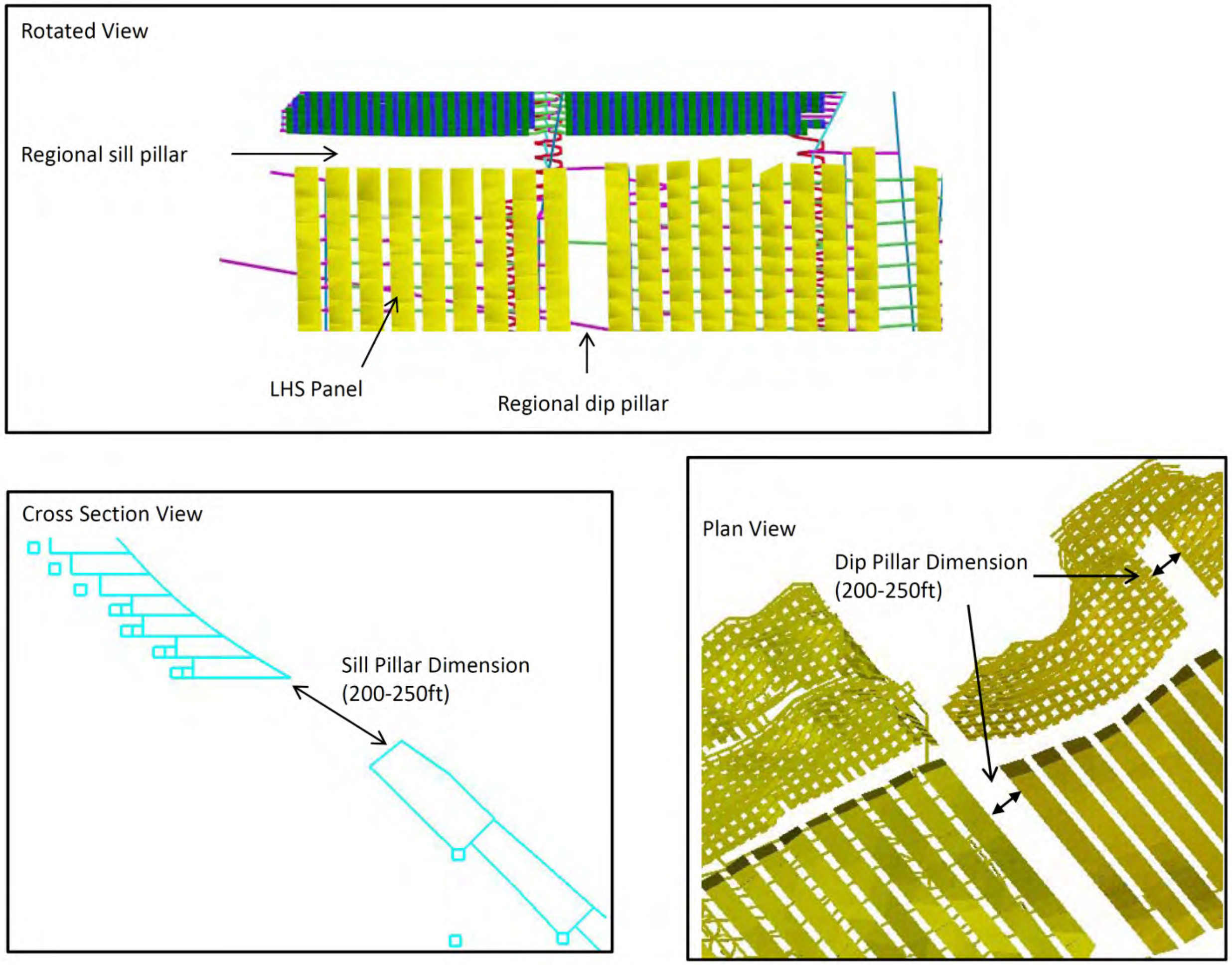
FLAC3D software models were used to study both regional (barrier) pillar and panel stability using various orientations and dimensions for the pillars and panels. Regional pillar design recommendations apply both to the Maturi and Maturi Southwest areas.

The deposits will be divided into panel areas with regional pillars in between. Pillar spacing will be 1,700 ft along strike and approximately every 1,700 ft along dip for a maximum hydraulic radius of 425 ft. The distance along strike is the controlling distance with a fixed maximum requirement. The dip dimension can be modified provided the overall hydraulic radius does not exceed that of a typical 1,700 ft x 1,700 ft panel.

Regional pillar sizes are 200 ft wide on average in the Tier 1 area and 250 ft on average in all other tiers. For regional dip pillars, the measurement of the pillar width is given in plan view along strike and for regional sill pillars the measurement is given parallel to the dip of the orebody as shown in Figure 16-1.

The regional pillars are the support mechanism for the mine. Pillars left within a panel are designed to yield and are relied on for local, not global support.

Figure 16-1: Pillar Measurement Orientations



Note: Figure prepared by SRK, 2014. LHS = long-hole stoping.

16.1.7 Crown Pillar

Modeling results currently suggest a minimum crown pillar thickness of 400 ft for Maturi and 300 ft for Maturi Southwest, as these provide a factor of safety of three and 2.5 respectively. It also satisfies the guidelines of Babcock and Hooker (1977) for mining adjacent to surface water. The model results currently suggest that crown pillar stability is sensitive to rock mass strength, and is linked to the stability of the barrier pillars that separate the panels. The results are based on the interpretations of testwork completed to date, and further testwork and analysis is required.

16.1.8 Panel Design Parameters

Panel design parameters are based on designs using FLAC3D software. Each panel will be mined from the bottom upwards.

16.1.8.1 Post-Pillar Cut-and-Fill

The extraction parameters for the post-pillar cut-and-fill areas were modeled based on depth from surface and deposit geometry. Recommended panel design parameters are summarized in Table 16-1.

A larger slot size of 46 ft wide x 40 ft high with 34 ft wide x 40 ft tall pillars was studied at a high level. These larger slot dimensions and smaller pillar dimensions appear to be within acceptable ranges; however, they require further study, and modified ground support may be needed. There is a higher risk associated with these larger stopes and narrower pillars. The larger stope size was used for mine design purposes and provided approximately 10 Mst of additional production tonnage when compared to using the smaller slot dimensions, or approximately 2% of the total Mineral Reserve.

Pillars in post-pillar cut-and-fill areas cannot be open on more than three sides at any time. This means that a primary/secondary extraction methodology must be used to ensure backfilling is complete in adjacent slots prior to mining. The pillars must be vertically stacked (aligned).

An 80 ft thick sill pillar can be left in place within a Tier 1 panel to allow for simultaneous mining on two levels.

Table 16-1: Post-Pillar Cut-and-Fill Panel Extraction Parameters

Tier	Approximate Depth from Surface (ft)	Stope Size (ft) *	Pillar Size (ft)	Overall In-Panel Extraction Ratio (%)
1 #	400–1,200	40 w x 40 h	40 w x 40 h	75
1 *	400–1,200	46 w x 40 h	34 w x 40h	83
3	2,500–3,000	26 w x 20 h	20 w x 20 h	81

Note: *Used for mine design purposes; # not used in the PFS.

16.1.8.2 Long-hole Stoping

Rib pillars will need to be sized such that they are stable on exposure and can provide support to the hanging wall locally down dip, even if they ultimately yield. It was recommended that a 50 ft rib pillar be left between each stope along strike. Stopes need to be sequenced in a way that avoids exposure of ribs on both sides and allows for backfilling as soon as possible. Recommended panel design parameters are as summarized in Table 16-2.

From a geomechanical standpoint, the preferred scheduling order for long-hole stopes in Tier 4 should be a chevron pattern where a single stope is mined on the level above and multiple stopes are mined on the levels below. This approach will transfer stresses up and out to the pillars.

16.1.8.3 Maturi Southwest

Tier 2 long-hole stoping and Tier 3 post-pillar cut-and-fill parameters from Maturi were applied to mine design at Maturi Southwest.

16.1.8.4 Remnant Mining

Additional analyses conducted with FLAC3D software suggest that it should be feasible to conduct second-pass (remnant) mining of low-grade ore (refer to Section 16.3.2.4). In Tier 2, this can be conducted using a drift-and-slash approach in which sections of the footwall along strike are developed with 20 ft x 20 ft primary drifts, and then slashed on retreat to a 40 ft x 40 ft profile.

Ground conditions may deteriorate when remnant mining occurs in the immediate vicinity of a panel rib pillar. This is accounted for in the extraction ratio applied to the mine design.

Table 16-2: Long-Hole Stoping Panel Extraction Parameters

Tier	Approximate Depth from Surface (ft)	Stope Size (ft)	Pillar Size (ft)	Overall In-Panel Extraction Ratio (%)
2	1,200–2,500	150 (along dip) x 150 (along strike)	50	75
4	3,000+	100 (along dip) x 100 (along strike)	50	67

16.1.9 Ground Support—Itasca

Itasca provided ground support recommendations for production mining areas in Tiers 1 and 3; this included slots and cross-cuts. Itasca's recommendations for the ground support for Tiers 1 and 3 are included as Table 16-3.

16.1.10 Ground Support—Golder

Golder provided ground support recommendations for life-of-mine access ramps and declines (long-term development openings), as well as medium-term development openings such as footwall accesses. The recommendations are as summarized in Table 16-4.

A 5% "poor rock" allowance has been applied to the ground support cost estimate to account for the limited information that is currently available regarding the location and extent of structural faults. The poor rock allowance includes additional costs for bolts and shotcrete.

16.1.11 Ground Support—Sweco

Sweco Norge AS (Sweco), a Norwegian tunneling firm, developed the ground support designs for the main declines for LNS (Chile) SA (LNS) for use in the PFS.

The ground support designs prepared by Sweco used the Q-system to develop the required ground support in different categories of rock masses and used an excavation support ratio (ESR) of "1" for the main declines, which are permanent infrastructure required for the life of the mine. The ground support system includes 3 m long 20 mm diameter resin end anchored bolts, fully grouted deformed steel bars and fully expension-shell bolts and with shotcrete. The bolt spacing and shotcrete thickness are dependent on the rock mass category and the percentage of each rock mass category was developed by LNS, based on geotechnical information provided to them by TMM in 2013. The bolt requirement estimates range from three to 16 bolts per meter of development. Provisions have been made for addressing ground conditions in fault zones and areas with spalling, slabbing and bursts. The quantities of ground support elements were estimated based on the type of ground support required in each rock mass category and the total length of the declines in each rock mass category.

Table 16-3: Itasca Slot Ground Support Recommendations

Location	Duration	Section	Recommendation: Backs	Recommendation: Walls
Slots – Tier 1*	Months	12.0 x 12.0 m 40 x 40 ft	Systematic bolting and screen 1.8 m x 1.8 m spacing 3.0 m long bolts PM 24H	#6 gauge screen to within 7' of the floor, held in place with nominal support (split sets) on 1.8 m x 1.8 m pattern.
Slots – Tier 3	Months	12.0 x 12.0 m 26 x 20 ft	Systematic bolting and screen 1.8 m x 1.8 m spacing 3.0 m long bolts PM 24H	#6 gauge screen to within 7' of the floor, held in place with nominal support (split sets) on 1.8 m x 1.8 m pattern.

Note: *The same ground support regime was applied to 46 x 40 ft slots in the Tier 1 mine design.

Table 16-4: Golder Ground Support Recommendations

Location	Recommended by	Duration	Section	Recommendation: Backs
Ramps and Declines	Golder	LOM	8.5 x 6.5 m 28 x 21 ft	1.5 x 1.5 m spacing 2.4 m long resin grouted #6 rebar Welded wire mesh to 1 ft below shoulder
Ramps and Declines	Golder	LOM	6.0 x 6.0 m 20 x 20 ft	1.2 x 1.2 m spacing 1.8 m long resin grouted #6 rebar Welded wire mesh to 1 ft below shoulder
Footwall drifts	Golder	Medium Term	6.0 x 6.0 m 20 x 20 ft	1.2 x 1.2 m spacing 1.8 m long resin grouted #6 rebar Welded wire mesh to 1 ft below shoulder

AMEC was provided with the data package that TMM supplied Sweco for ground support that included a May 2014 Itasca report (Itasca, 2014a). AMEC reviewed the ground support recommended by LNS for the domains expected to be intersected within the declines, and based on information provided by Golder, the LNS assumptions are likely conservative.

16.1.12 Main Ramps and Underground Infrastructure

16.1.12.1 Design Recommendations

Ideally, the ramps should not be located in high-stress areas. The regional pillars, particularly at depth, will experience high stress loads and therefore any ramps located adjacent to pillars at depth would likely require substantial ground support and maintenance. To maximize the efficiency of the ramp placement, it was determined that, in the shallow areas where pillars are not as heavily stressed, ramps can be located adjacent to (behind) the regional pillars. A 300 ft standoff distance was recommended for the ramps to minimize damage and rehabilitation work over time. Long-hole stoping footwall access drifts should use a 100 ft stand-off distance.

The proposed location and size of permanent infrastructure excavations for crushers, conveyors, etc. were also verified by Itasca to ensure the long-term stability of the openings required for those installations.

16.1.12.2 Design Parameters

SRK completed the design with a 300 ft offset from first pass mining (which is based on higher NSR cutoffs), but when the lower-grade remnant material is mined later in the mine life (Years 18 onward), the final ramp offset can be as little as 75 ft.

16.2 Hydrogeological Considerations

Four hydrogeological investigations for mine water and mine operations purposes have been conducted since 2008 at the proposed Maturi mine site. No hydrogeological studies have been completed to date for environmental purposes and these mine-related studies are not intended for use for prediction of environmental impacts.

Barr conducted the first investigation in 2008 (Barr 2008), and FracFlow Consultants, Inc, (FracFlow) as a contractor to Itasca, conducted the final three of these four investigations of the Maturi site from 2012 to 2013. Two phases of hydrogeological evaluations of the Maturi site were completed in 2012. Investigations by FracFlow in early 2013 focused on the vicinity of the Inco shaft (FracFlow 2013c). All of these investigations were focused on the immediate mine area, and were not intended to include the overall Project area or to be a regional hydrogeological model.

A total of 1,717 borehole logs were provided to Itasca for the Maturi and Maturi Southwest deposits. Most of those boreholes are located in the vicinities of the two deposits, and were drilled in order to assess the extent and grades of the orebodies.

As part of its three hydrogeological investigations, FracFlow performed slug tests, packer tests, and one aquifer test in existing exploration boreholes that are located at the Maturi site for the purposes of assessing hydrogeological properties, primarily hydraulic conductivity (*K*).

The conceptual model of a groundwater flow and hydrologic system is an interpretation or working description of the characteristics and dynamics of the physical hydrogeological system (ASTM 2006).

The major hydrogeological components that control the potential groundwater inflow rates to underground mine workings at the Maturi site include:

- Geologic setting: The hydraulic parameters of geologic units control the estimated groundwater inflow rates to the proposed mine
- Recharge: The main recharge to the groundwater system is precipitation

- Groundwater inflows to the mine: Groundwater inflows are the estimated water that seeps into the proposed underground mining area during mining activities. The prediction of the groundwater inflow rates to potential underground workings was the main objective of the groundwater flow modeling effort.

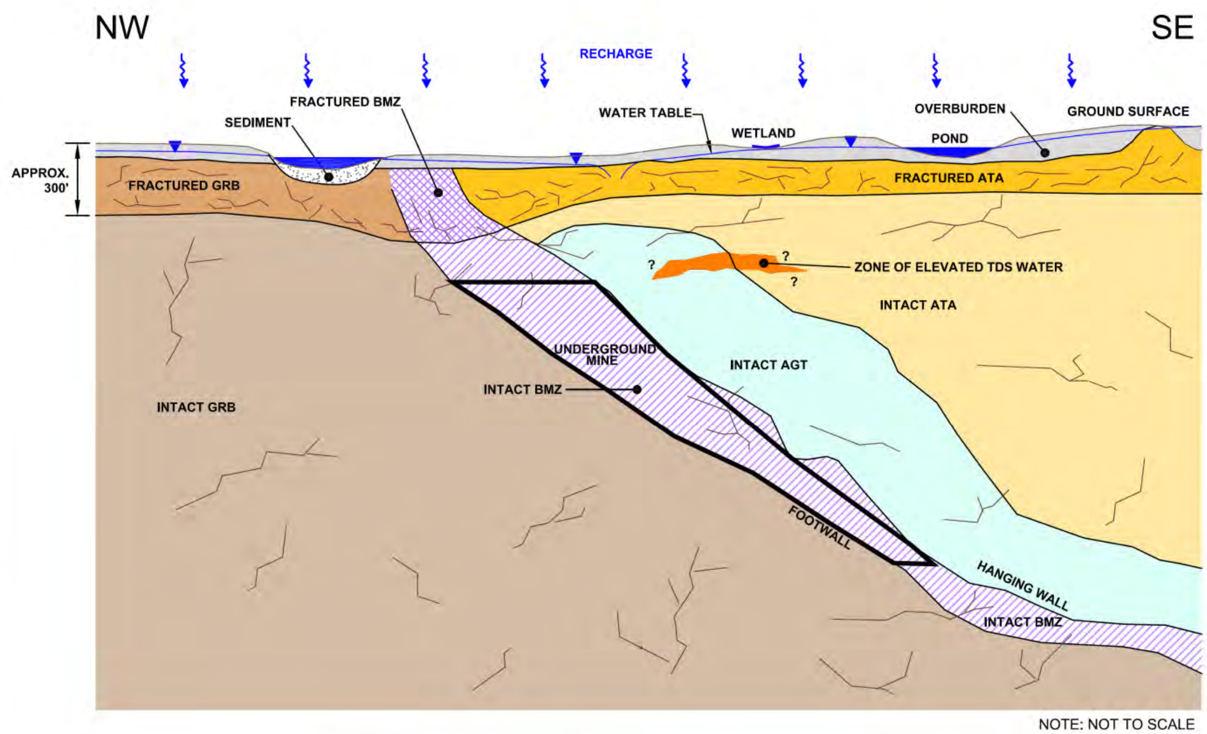
Figure 16-2 summarizes the interpreted groundwater flow and hydrologic system.

The geologic and water-level data obtained from the boreholes and the hydrogeological data collected from the boreholes were used as inputs for the groundwater flow model for the Maturi deposit.

The available calculated values of *K* for fractured and unfractured bedrock vary by approximately five orders of magnitude: 10^{-5} to 10^{-10} cm/s. However, the highest values for *K* suggest that the faults or fracture zones in the area of the proposed mine are neither very permeable nor connected. As such, discreet faults or fractures were not incorporated into the groundwater flow model. The hydrogeological data indicate relatively low *K* values, even for fracture zones. In addition, current data suggest that the discontinuities (fracture zones) are not well connected to sources of recharge. A similar conceptual hydrogeological model is applicable to the Maturi Southwest deposit.

Although a model was undertaken for Maturi Southwest, the assumption in that model was that mining would be by open pit methods, rather than the current underground mine plan. The model remains to be updated for underground mining.

Figure 16-2: Maturi Site Conceptual Hydrogeological Model



PROJECT NO.	1973
BY	JX
CHECKED	RJS
DRAWN	SAC
DRAWING NAME	CONCEPTUAL
DRAWING DATE	27 AUG 2013
REVISION DATE	23 DEC 2013

ITASCA™ Denver, Inc.

Conceptual Hydrogeologic Model

CLIENT: Twin Metals Minnesota, LLC

FIGURE NO. 2

Based on the data analysis, groundwater flow-model calibration, and the predictive simulations, Itasca had the following preliminary conclusions regarding the Maturi deposit. These estimated groundwater inflow rates are for the mining panels and do not include groundwater inflows to declines or other underground infrastructure:

- The preliminary maximum inflow rate assumed in the basecase scenario is 550 gal/min
- The predicted groundwater inflow rates to the mine are sensitive to the assumed *K* value of the intact bedrock, which has not been assessed to the degree required for feasibility-level groundwater inflow predictions.

16.3 Design Assumptions and Design Criteria

A 3D mine design was completed using Maptek's Vulcan software. The basis for the mine design was the resource estimate and block model discussed in Section 14. In addition, a model update that included grades in all blocks along the hanging wall was developed by AMEC for mine design purposes. The Mineral Reserve estimate includes these hanging wall blocks.

16.3.1 NSR Cutoff Strategy

The NSR cutoff calculation is discussed in Section 15.5.

A NSR cutoff strategy was employed to maximize the optimal net present value (NPV) for the deposits. The cutoff grade strategy prioritizes a higher NSR cutoff in the early years of the mine plan and uses a lower NSR cutoff in later years.

Material above marginal cutoff grade, located in the footwall behind high cutoff grade panels, is referred to as remnant material. Remnant material is included in the mine design and production schedule once targeted cutoff grade material is depleted.

16.3.2 Mining Methods

16.3.2.1 Tier 1

From the footwall drift a 46 ft wide x 40 ft high "slot" will be developed to the hanging wall. Initial rounds will serve as a transition from the 20 ft x 20 ft footwall drift to the larger opening used for the slot. Once the full 40 ft stoping height is exposed, a V-shaped drilling pattern will be used with an assumed advance rate of 18 ft/round. After the location of the hanging wall has been confirmed, a lower face height may be drilled (20 ft high) to minimize dilution from the hanging wall in the last round. The entire slot and the transition areas will be fully ground supported, and it is assumed that mucking in the slot will be performed with a manned load-haul-dump unit (LHD). The cross-cuts will then be mined on retreat from the hanging wall to the footwall drift, and are assumed to be unsupported and to require remote mucking. Mucked ore will be loaded directly into a waiting truck and hauled to the crusher.

Once all of the cross-cuts have been mined, a bulkhead will be built in the 20 ft x 20 ft transition to the slot and the area will be backfilled with paste backfill. Subsequent levels will be mined above the filled areas and therefore mobile mining equipment will be driving on paste backfill when operating within the stope. The dilution and recovery calculations assume that, for each level that is mined, 1.0 ft of in situ rock will be lost in the backfill and 0.5 ft of backfill will be sent as dilution to the crusher with the ore.

A typical post-pillar cut-and-fill mining panel will comprise a series of 46 ft wide rooms separated by 34 ft wide pillars. A series of 21 rooms and pillars will be located between regional dip pillars along the strike of the orebody. Panel length along dip typically will be less than the maximum distance recommended by Itasca due to the location of the post-pillar cut-and-fill areas and due to the need to leave a limited number of sill pillars to establish more active working levels. Increasing the number of active working levels was necessary to achieve the desired production rates.

On-ore[5] footwall drifts will be mined parallel to strike and will generally follow the footwall of the orebody. After the footwall drift has been developed along the full length of the panel and ventilation has been established, 46 ft x 40 ft slots will be mined from the footwall drift to the hanging wall. The first 20 ft of each slot will be developed at 20 ft wide x 20 ft high, and then transitioned to 46 ft wide x 40 ft high. This will also allow a smaller bulkhead to be constructed prior to back-filling as the bulkhead can be positioned in the narrower 20 ft wide x 20 ft high part of the slot. After the slots have been mined, cross-cuts developed perpendicular to the slots will be mined on retreat.

16.3.2.2 Tier 3

Mining of Tier 3 will be similar to the approach that has been described for Tier 1; however, Tier 3 will not require a transition to a taller slot dimension. Footwall drifts on the levels above the active production area must be developed either after backfilling the current level or must take care to ensure the footwall drifts have sufficient horizontal offset so as to not interfere with production mining.

16.3.2.3 Long-hole Stoping

Stope sizes will vary depending on the local dip of the orebody; however, a typical stope will be 120 ft wide, 150 ft long and 150 ft high. A stope with these typical dimensions will yield approximately 270,000 ore tons. After the top and bottom stope accesses (20 ft x 20 ft) are established, extraction and drilling drifts will be developed

[5] The term “on-ore” refers to development openings that are positioned within the orebody

the full length of the stope. The drilling drift location will vary based on the geometry of the mineralization, but will generally follow the hanging wall.

A slot raise (11 ft x 11 ft) will be developed between the extraction and drilling drifts at the far end of the stope. The steps required to mine the slot raise include drilling a 6" diameter hole with an ITH drill, followed by reaming to a 9" diameter hole and finally, boring to a 30" diameter hole with a Machines Roger boring head. Additionally, several 6" diameter holes will be drilled within the slot "box". Drilling will continue with the ITH drill using a fan shaped pattern. One ITH rig will drill one or two rings ahead of the slot while other ITH rigs will drill from the slot back to the entrance to the stope. The drill pattern will be 13 ft spacing x 15 ft burden. In a typical stope, holes will be drilled downward from the drilling drift; however, it may also be necessary to drill some up-holes from the extraction drift.

Holes will be loaded with bulk ANFO and stope blasting will commence in the slot. The toes of the slot will be blasted first, followed by further slot blasts until the slot has been removed. While blasting the slot, neighboring ring holes will also be blasted. Blasting continues from the slot, up and to the back of the stope. Up-holes from the extraction drift will also be blasted along with the down-holes as required. The size of each blast will be restricted by the amount of void that is available and by the need to limit blasting induced effects on nearby infrastructure. Holes may be loaded from the drilling drift while mucking below, however controls must be in place to prevent loading holes near or in the extraction drift when mucking is underway.

The lower portion of the stope will be trough-shaped to facilitate ore flow. It is anticipated that a portion of the stope (estimated 58%) can be mined using a manned LHD unit, but the remainder will need to be remote mucked (42%). Remote mucking is required when the LHD operator cannot remain behind the brow of the stope to effectively remove the broken ore.

The stope ore will initially be placed in a muck-bay, and then later rehandled into a haul truck and transported to the crusher. A one-way 150 ft LHD tram distance to the re-muck chamber was assumed.

Occasionally, blasted ore will be too large to be placed in the crusher. When this occurs, the oversize ore will be placed in a "blasting chamber" where it will be fragmented by a block-holer secondary breaker.

Once the stope is completely mucked out and surveyed, a shotcrete bulkhead will be placed in the lower stope access and a dump barricade will be placed in the upper stope access. The stope void will then be filled with paste backfill. Fill pipes and breather pipes will be inserted through the barricade to the back of the stope. If required and depending on the stope orientation and scheduling order, the backfill pipes could be inserted from the stope above.

A typical long-hole stoping mining panel comprises nine stopes aligned along strike that will be separated by 50 ft panel rib pillars for geomechanical support (and to prevent vertical backfill exposures) and approximately 10 stopes aligned along dip that do not require pillars for support. Stopes will be diamond-shaped with a 45º minimum wall angle to facilitate muck flow and maintain backfill stability. The stope wall on the footwall side of the orebody can be steepened as required. To further ensure the stability of the backfill, a small pillar will be left between the extraction drift of each stope and the top of the backfilled stope that is located immediately down dip.

A primary/secondary stoping sequence will be used in the long-hole stoping panels in Maturi Tier 2 and at Maturi Southwest, while Maturi Tier 4 will not employ secondaries. The primary/secondary stoping sequence dictates that, on any given level, primary stopes must be separated by a secondary stope. Full extraction of the secondary stope can only occur after the two immediately adjacent primary stopes have been mined and backfilled. Backfilling will be an integral part of the long-hole stope mining cycle and all stopes will be backfilled to maintain the long-term stability of the mining areas.

Off-ore[6] footwall drifts will be established to access the long-hole stoping panel at various levels. The footwall drift on each level will be connected to the ramp system and to the long-hole stoping panel by stope accesses that will be mined at a gradient of between 0% and 15%. A single stope access will be shared by two adjacent long-hole stoping stopes.

On-ore extraction and drilling drifts will be developed within each long-hole stope. These drifts will be 20 ft high by 20 ft wide and will be oriented parallel to the strike of the orebody.

Ventilation raises will intersect the ends of the footwall drifts on each level. Another ventilation raise, which will also serve as a secondary egress, will be located near the ramp and will be accessed at various ramp elevations. The ramp system will be located approximately 300 ft from the orebody and will have a maximum grade of 15%.

16.3.2.4 Remnant Ore Mining

Remnant mining is second pass mining which allows for extraction of lower-grade ore that is above the NSR cutoff used for Mineral Reserves reporting (US$25.00), but is below the NSR cutoff selected for first pass mining. In Tier 1 this remnant ore is located in the footwall behind the first pass mining. Tier 2 also has some remnant ore material located in the footwall behind the first pass mining. In all other areas, remnant ore is typically located in new mining areas. The locations of the remnant ore were

[6] The term “off-ore” refers to development openings that are positioned outside of the orebody

included in Figure 16-3. Overall remnant mining makes up approximately 17% of the Mineral Reserve.

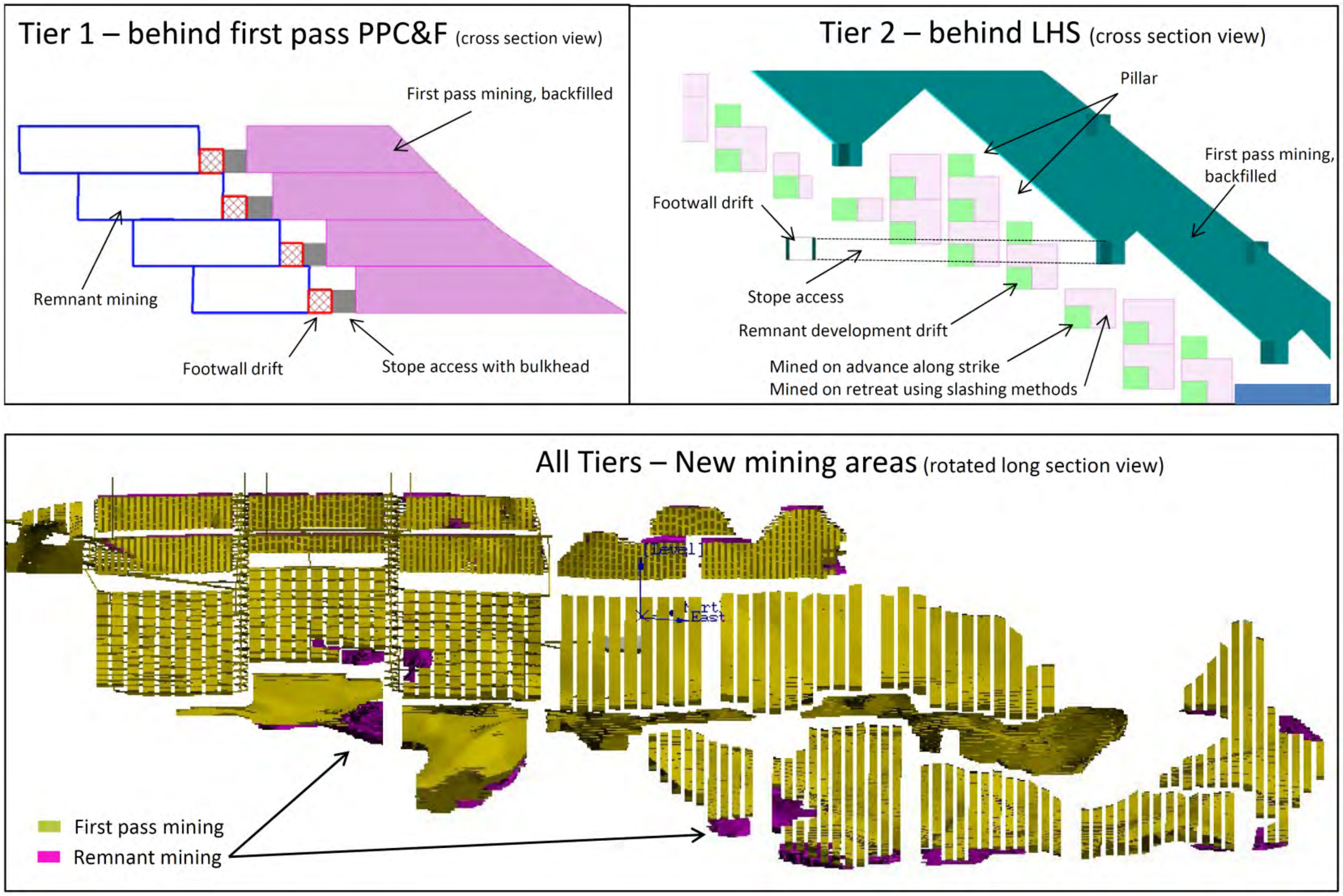
Remnant mining behind the Tier 1 post-pillar cut-and-fill areas will use the same bottom up post-pillar cut-and-fill areas method that was used for the first-pass mining. Overall in-panel extraction will be 75% based on 46 ft wide slots and 34 ft wide pillars. Access to each level will be from the same footwall drift that was used for first pass mining. No additional development will be required.

Behind the long-hole stoping areas a cut, slash, and fill method will be used. To begin with, a 20 ft x 20 ft remnant development drift will be mined from an existing stope access, along strike for approximately 300 ft. Once the full length of the remnant development drift has been established, the drift will be slashed on retreat back to the stope access. Slashing will always occur in the area between the remnant development drift and the backfilled long-hole stope. The objective is to extract the remnant ore without exposing miners to unsafe ground conditions. When slashing, the pillar between the remnant development drift and the backfilled stope will become narrow (but not less than 20 ft). By mining the slash on retreat and using remote mucking, the risk associated with the narrow pillar will be minimized.

Remnant mining in new mining areas will be by the post-pillar cut-and-fill method, and will require new footwall drifts and stope access. Mining recovery will range between 75% and 80% depending upon the tier in which the remnant mining occurs.

The parameters discussed for the above three remnant mining scenarios are applied to mine design accordingly for the estimation of Mineral Reserves. Additional waste development was included as appropriate.

Figure 16-3: Remnant Ore Locations



Note: Figure prepared by SRK, 2014. Note: PPC&F = post-pillar cut-and-fill; LHS = long-hole stoping

16.3.2.5 Blasting

ANFO has been assumed for the PFS as current hydrogeological testwork suggests low water inflows. Mines that experience wet blast holes typically use emulsion in preference to ANFO because of its greater resistance to water damage. If water is found to be more prevalent than expected during development of the mine, emulsion would likely replace ANFO for in some or all blasting applications. The result would be higher blasting costs as emulsion is more expensive than ANFO.

A fragmentation analysis was performed for the post-pillar cut-and-fill and long-hole stoping mining methods and their respective drill patterns to estimate the ability to deliver a fragmentation suitable for crusher feed. Results indicate that a minimum amount (<2% by weight) should be expected above 4 ft long. A size of 4.2 ft is the maximum rock size considered feeding the gyratory crushers.

16.3.3 Ramps and Mine Access

16.3.3.1 Underground Ramps

Footwall drifts and stope accesses will be 20 ft x 20 ft with an arched back. Perimeter blasting control will be used when ground conditions dictate that it is necessary.

All ramps have been designed at a gradient of 15%. The ramp offset in the Maturi area is 300 ft from first pass mining and approximately 150 ft from remnant mining. The ramp offset in the Maturi Southwest area is 100 ft. To account for re-muck bays, safety bays, etc., that have not yet been designed, an additional 10% was added to the development footage. This is based on a re-muck bay spacing of 700 ft assuming a length of 60 ft/re-muck bay, along with an additional minor allowance for other excavations.

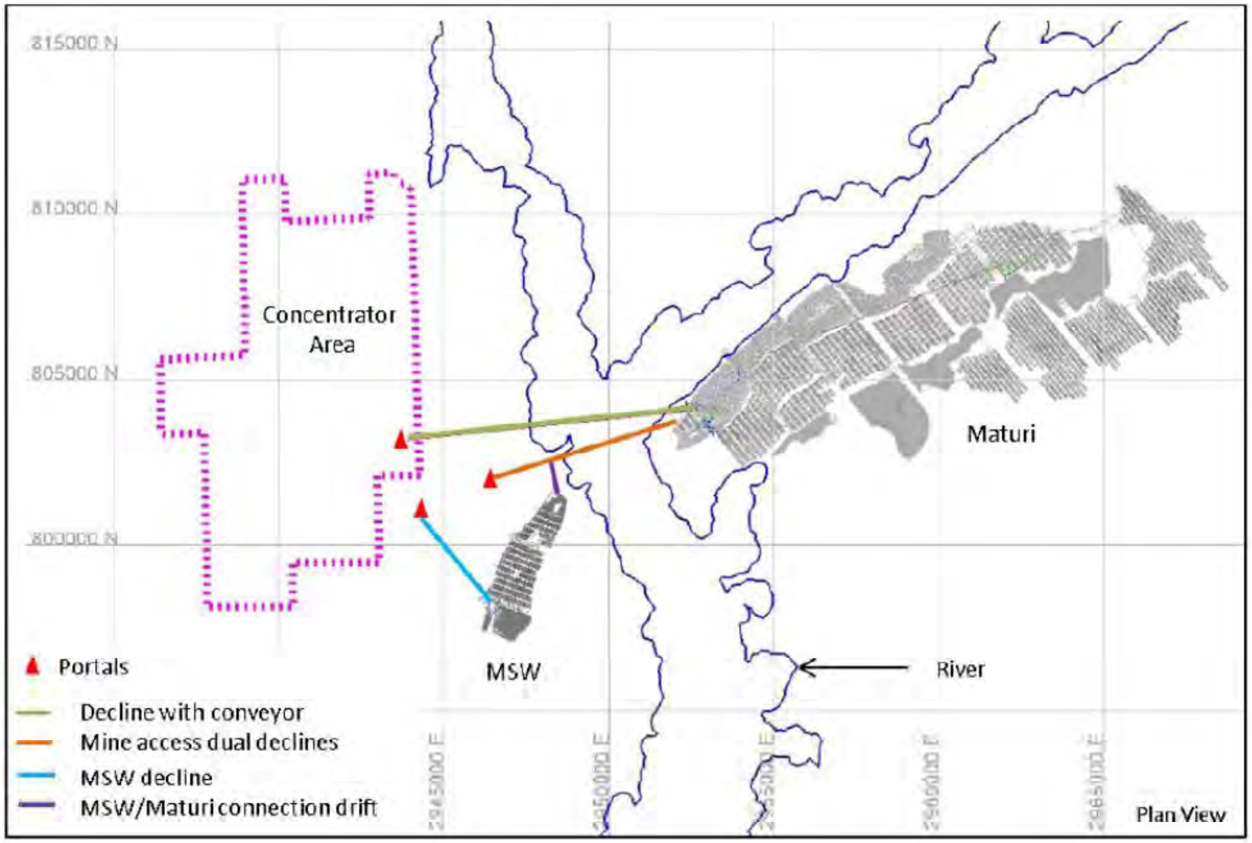
Ramp system dip is constant at approximately 36º, and should be optimized during a more detailed design stage so that it more closely follows local changes in dip.

A turning radius of 75 ft was used to design ramps and other development openings based on the specifications for an MT85 haul truck as provided by Atlas Copco.

16.3.3.2 Mine Access

The underground will be accessed via four declines from surface, three to Maturi and one to Maturi Southwest (Figure 16-4). For the purpose of the PFS, it was assumed that the declines from the surface and the initial internal ramp system will be developed by a mining contractor. Total ramp development footage is estimated at 348,800 ft.

Figure 16-4: Proposed Portal Location Plan



Note: Figure prepared by SRK, 2014. MSW = Maturi Southwest. Grid indicates scale. Map north is to top of plan.

The conveyor decline to Maturi will be located in the concentrator area and will have a conveyor installed to deliver material from the underground crushing station to the coarse ore stockpile.

The two service declines for Maturi will be twin declines located southeast of the concentrator near the Little Lake Road. The twin declines will support access and egress of rubber-tired mobile equipment for the provision of miners, equipment, and materials to operate the underground mine. Traffic will be one-way in each decline, with one decline serving as access and the other as egress.

The access decline to Maturi Southwest will be located south of the proposed concentrator site. Maturi Southwest will also be connected to the Maturi declines from underground.

The conveyor and service declines are planned at 22 ft high by 28 ft wide. The decline profile allows for two lanes of traffic. Within the mine, a series of drifts will provide travel-ways to and from the working areas. These drifts will be used by haul trucks to transport ore and waste material around the mine. Ore will be hauled to one of two underground crushing stations, while waste will be primarily hauled and dumped into mined-out production stopes.

16.3.4 Raises

Vertical development in the form of raises is required throughout the mine for ventilation purposes. Two methods, raiseboring and slot raises[7], will be used to develop the ventilation raises depending on the required raise size, length, and dip/orientation. Slot raises will be developed using a Machines Roger V-30 and then blasted to create the final dimensioned raise.

The raises are summarized in Table 16-5.

16.3.5 Grade Control

Additional infill drilling/grade control program will be required. Cost of the infill drilling is included in the general and administrative (G&A) cost.

[7] A slot raises is a short raise mined in ore between the extraction and drilling drifts in a long hole stope to create the initial void prior to commencing production blasting

Table 16-5: Raise Lengths*

Type	Quantity	Length (ft)
Raisebored underground (18.5 ft diameter)	10	4,375
Raisebored to surface from upper plenums, Maturi Southwest (18.5 ft diameter)	8	5,405
Raisebored to surface from lower plenums (18.5 ft diameter)	2	5,567
Raisebored transfer between plenums (18.5 ft diameter)	2	4,090
Raise slot (21 x 21 ft square)	398	72,368

*Includes Maturi and Maturi Southwest

16.3.6 Underground Haulage

Haulage distances were determined for each panel and level from loading points to the nearest crusher. Ore from the panels below the upper crusher will be delivered via a haulage loop to the crusher. The lower crusher also has a haulage loop to the crusher.

Waste rock haulage distances will be 11,000 ft (one-way) during the pre-production years to account for the fact that waste rock will have to be hauled to the surface. During the production years, the haulage distance will be 4,000 ft (one-way) because waste rock will be used as backfill in mined-out stopes.

All stope ore will be hauled with Atlas Copco MT85 trucks, which are rated at 85 metric tonnes (94 st). MT85 haul trucks are currently being tested in an Atlas Copco test mine and are expected to enter the broader market in a few years. For development rock, both ore and waste, Caterpillar AD60 haul trucks have been assumed. The Caterpillar AD60 has a 66 st payload capacity. Some of the 66 ton trucks will be fitted with ejector buckets. This will allow for dumping into stopes with 20 ft high backs. Yearly average hauling distances for each type of truck are tracked in the production schedule to allow for sizing of the fleet.

Fleet sizing assumes flat hauling speeds of between 16 and 17 mph and a ramp hauling speed of 6.2 mph. These assumed speeds are at the high end of the range of possible speeds for the selected equipment but are not unreasonable provided TMM management is diligent in maintaining the roadways, dispatching and controlling traffic, and sequencing stopes in a way that minimizes traffic congestion.

The PFS fleet size calculations assumed no truck interference. During review of the mine cost model, interference delay time of five minutes per cycle was added to the truck cycle calculations, which resulted in an increase in truck fleet size and operating costs.

16.3.7 Dewatering

A mine dewatering system capable of discharging an average of 1,000 gal/min at the conveyor portal was developed. The planned dewatering system consists of thirteen

400 hp skid-mounted pump stations: 10 at Maturi and three at Maturi Southwest. The pump station locations are planned to be 500 ft apart vertically.

Water from the production areas will be collected in mining level sumps placed strategically around the mine. A total of 52 of sumps will be required over the LOM (45 for Maturi, seven for Maturi Southwest). The majority of the mine production units are planned to have three mining sumps (although the number varies from one to four). It is estimated that no more than 25 of the 52 sumps will be operating at any one time.

Discharge water will be staged out of the mine. The bottom station will pump water to the tank feeding the station above, which in turn will feed the station above it, and so on until the water reaches the conveyor decline portal.

16.3.8 Ore Stockpiling and Waste Rock Storage

All ore mined during the three years of pre-production will be trucked to the surface and placed in a stockpile. The maximum surface ore stockpile size during the pre-production period is expected to be 1.75 Mt.

The cost and design of the surface ore and waste stockpiles was completed by Barr based on a total estimated capacity of 5.8 Mt. The ore stockpile will be crushed on surface and fed to the mill a rate of 5,000 st/d later in the mine life.

A total of 599 kt of low-grade ore mined in Years 1 to 4 is also assumed to be stockpiled underground. This material would be stored underground in a previously mined area and would be fed to the underground crushers when required.

Development waste rock generated during the pre-production period will also be deposited on surface and consists of 1.3 Mt from contractor mining and 2.8 Mt from Owner crews, giving a total surface waste rock storage facility size of approximately 4.1 Mt. All waste material mined after the commencement of production will be deposited underground in mined-out stopes.

16.3.9 Trade-off Studies

A number of trade-off studies were completed:

- Evaluation of throughput rates
- To verify the NSR, the stope optimizer in Maptek's Vulcan software was used to determine the economic material targeted by the cutoff grade strategy
- Several crusher locations were studied. Trade-offs considered locations of the first main crushing stations underground and on surface. The evaluations were coupled with truck fleet estimates and duration and cost of access construction
- Portal locations were determined after a site visit conducted with consultants

- Use of 94 st haul trucks vs 66 st trucks for production haulage
- Primary crusher comparison where three types of primary crushers were reviewed: gyratory, low speed sizer, and jaw crusher
- Evaluation of different heating options: propane, LNG, diesel, electric, natural gas, and the option of not heating as well
- Two different electrical distribution systems were analyzed, 13.8 kV and a 34.5 kV cases.

16.3.10 Underground Infrastructure

Information on planned underground infrastructure is included in Section 18.8. The surface crusher requirements are discussed in Section 18.9.4.

Infrastructure items included in the mine design include development footage for first aid, laydown, sumps, substations, refuge stations and toilets, totaling approximately 14,000 ft.

16.4 Ventilation

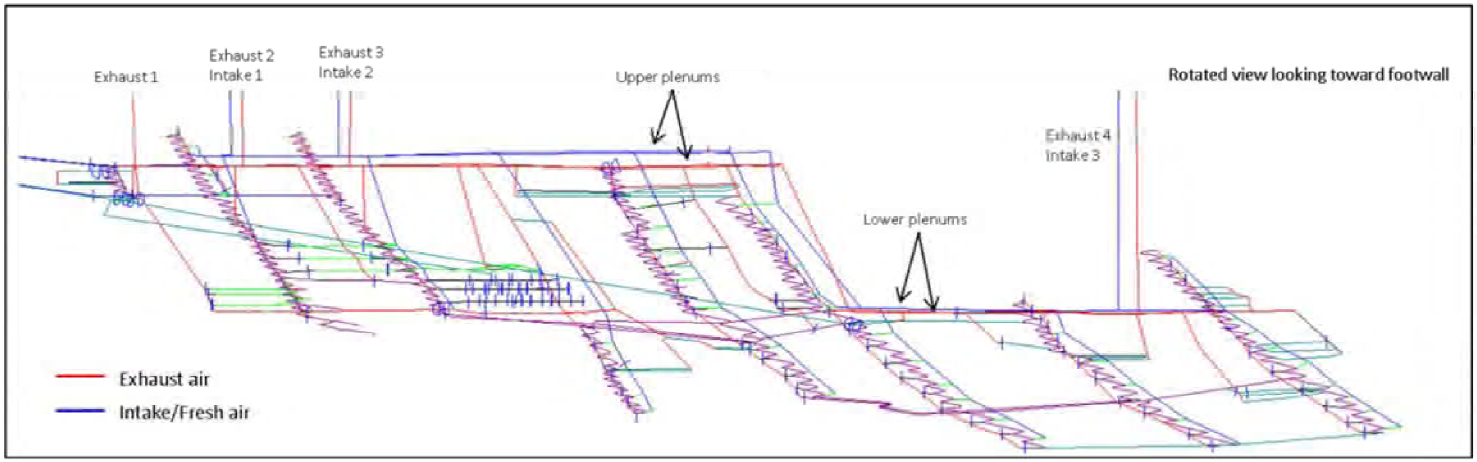
16.4.1 Mine Models

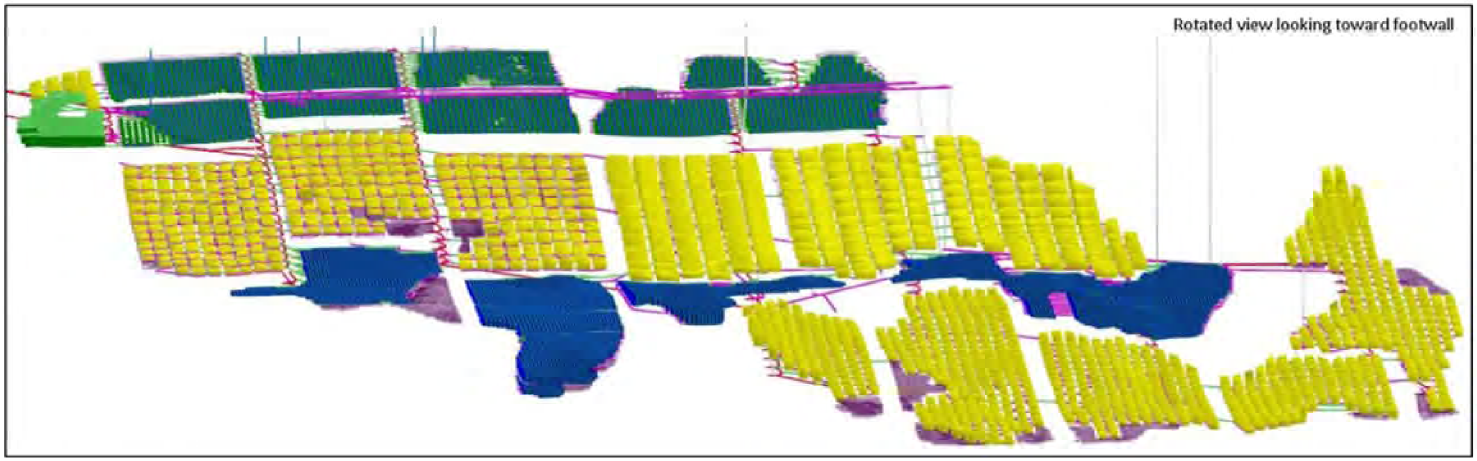
Ventilation models were developed for the mine at various stages throughout the life of the mine. Initially the production areas are located in very close proximity to each other; however, as the mine progresses the production and development areas become more spread out which results in increased leakage and additional airflow demands.

Plenums will be used for both the fresh air and exhaust air to bring airflow to and from the mining areas (Figure 16-5).

The plenums will be developed at an elevation near the top of the orebody and will continue along strike. Raises will be developed from the fresh air and exhaust air plenums to the mining areas and to the surface. As the mine progresses at depth, lower plenums will be required and internal raises will connect the plenums.

Figure 16-5: Planned Ventilation Plenums and Raises to Surface, Maturi





Note: Figure prepared by SRK, 2014. Figures are schematic and not to scale.

16.4.2 Ventilation Requirements

For each development and production area, airflow was estimated based on equipment requirements and diesel engine power as shown in Table 16-6. An airflow requirement of 100 cubic feet per minute per brake horsepower (cfm/bhp) of diesel equipment power was used in the calculations, which is based the use of Tier 4 engines for all equipment and the use of ultra-low sulfur fuel (ULSF).

The maximum air velocity within a raise has been assumed to be 4,000 ft/min. This velocity is near the maximum velocity usually found in unlined raises. If future work determines that ventilation requirements increase, the required size of the vent raise will increase. Future work will include optimization of airflow and raise sizes.

Including the crusher and infrastructure areas, a total airflow of 3.25 million cfm will be required, which is approximately equivalent to 65 cfm/st based on a 50,000 st/d ore production rate. As a frame of reference, the typical airflow for a block cave operation is 45 cfm/st and the typical airflow for a stoping operation is 240 cfm/st.

The Project will have an operating airflow similar to a block cave mine due to the bulk mining techniques that will be used. Because of these bulk mining techniques, less equipment will be required to achieve a high production rate, and the equipment will be operating in larger development openings and production stopes. In contrast, in many less massive stoping operations, more equipment is required to achieve a high production rate, and the equipment is often operating in smaller development openings and production stopes.

Ventilation infrastructure requirements are summarized in Table 16-7.

16.4.3 Maturi

16.4.3.1 Pre-production

During initial mine development the service declines will be developed in parallel, with cross-cuts between the ramps; with one decline serving as a fresh air intake and one decline serving as an exhaust airway in this early-development stage. The lower single decline will be ventilated using vent tubing for approximately three quarters of its full length with exhaust air returning up the ramp.

Once the first exhaust raise to the surface is in place, up to 1,200 kcfm will be exhausted to the surface thereby significantly increasing airflow through the mine. At this point, the both service declines and the internal ramp will be converted to intake airways.

Table 16-6: Airflow Requirements for Development and Production Areas

Area	Notes	Truck (hp)	LHD (hp)	Scaler (hp)	Bolter (hp)	Drill (hp)	Loading Truck (hp)	Total (hp)	Airflow Required* (cfm)
Post-pillar cut-and-fill development level	Supplied by auxiliary ventilation system to ramp exhaust raise	650	250	150				1,050	105,000
Post-pillar cut-and-fill production level	Flow through ventilation system to perimeter/central level exhaust raise	1,300	250	150	138			1,838	183,800
Long-hole stope drill level	Flow through					300	276	576	57,600
Long-hole stope development level	Flow through	650	250	150	276			1,326	132,600
Long-hole stope initial development level	Supplied by auxiliary ventilation system to ramp exhaust raise	650	250					900	90,000
Long-hole stope production level	Flow through	1,300	750	150	138			2,338	233,800

Note: Based on 100 cfm/bhp. PPC&F = post-pillar cut-and-fill mining method; LHS = long-hole stoping mining method; cfm = cubic feet per minute

Table 16-7: Ventilation Infrastructure Requirements

Item	Description	Quantity
Maturi	Airlock doors	15
Bulkheads	Duct adaptors initially transition to standard bulkhead	260
Bulkhead with regulators	Regulator orifice size is 26 to 56 ft^2	290
Maturi Southwest	Airlock doors	6
Bulkheads	Duct adaptors initially transition to standard bulkhead	20
Bulkhead with regulators	Regulator orifice size is 26 to 56 ft^2	20

The first set of intake and exhaust raises on the plenum will then be developed, providing a total airflow of approximately 1,600 kcfm. Plenum development will continue northward.

16.4.3.2 Steady-State

The ventilation models developed by MVS included four exhaust raises and three intake raises over the life of the mine. Intake raise nos. 1 and 2 and exhaust raise nos. 2 and 3 connect to the upper air plenum. Intake 3 and exhaust 4 connect to the lower air plenum. The upper air plenum requires parallel drifts for intake and exhaust. The lower air plenum is modeled with three airways, one intake and two exhausts.

16.4.3.3 Fan Requirements

From the models and the production schedule, it was determined that the airflow requirements for stoping will be close to the ventilation system capacity. The models developed typically had four to five areas where active stoping can be supported while development mining continues towards the next part of the orebody. However, starting in Year 4, the number of active stoping areas will increase and the amount of development mining will also increase. The number of development headings will need to be limited to ensure the ventilation system can support the overall airflow demand.

As the number of fans increase, the total airflow through fans 1 through 3 decreases. There are significant variations between the required pressures. These fluctuations can be correlated to infrastructure installations such as the new raises and fans.

Axial flow fans were selected as the main mine fans for the Maturi mine. Utilizing the fan duty point information provided by the ventilation manufacturer, a fan performance parameter and physical locations were determined. All four main fans will act as exhaust fans, located underground at the base of a dedicated exhaust ventilation raise.

16.4.4 Maturi Southwest

A single, full production scenario ventilation model was created for the Maturi Southwest area base on the maximum production rate of 20,000 st/d.

There are two exhaust raises planned for the Maturi Southwest area and a single intake raise. All ramps will be intake airways.

Ventilation raises in the Maturi Southwest area will have the same 18.5 ft diameter as those in the Maturi area. Axial flow fans were selected as the main mine fans for the operation.

16.4.5 Face Ventilation

The ventilation design for development (20 by 20 ft) headings assumes a 60 in., 275 hp fan, capable of providing 80,000 cfm of ventilation for a maximum distance of 1,000 ft per fan. As development progresses, the vent ducting will be extended so that it does not terminate more than 80 ft from the active mining face.

The ventilation design for the post-pillar cut-and-fill headings assumes a 60 in., 275 hp fan capable of providing approximately 130,000 cfm of ventilation for a maximum distance of 400 ft. One fan will provide ventilation for three headings through the use of controls.

Ventilation for the long-hole stoping areas assumes a 60 in., 275 hp fan capable of providing 130,000 cfm of ventilation for a maximum distance of 500 ft.

16.4.6 Heating

Heating will be by direct-fired natural gas burners in buildings that handle all or a portion of the allocated intake airflow. Based on climatic data from Ely, Minnesota, it is estimated that a total of eleven 40 MMBtu/h heating units will be required. At Maturi, six of the units will be installed on the surface near the collars of the four intake raises, and two at secondary access portals. For Maturi Southwest, two units will be to be installed at the top of the intake raise, and one at the portal. Because intake airflow will remain relatively constant through the LOM, even as additional intake raises are brought online, some heaters will require relocation.

16.4.7 Emission Control

A preliminary model, without a DPM mitigation strategy, was completed for the contaminants that are expected to flow to the atmosphere through the exhaust raises (Table 16-8). An initial calculation of contaminant production was made based on 24 hours of continuous operation and an assumed equipment utilization factor. Refinement of the equipment operating hours assumption will likely reduce the estimated air contaminant production for the mine.

A water spray and fogging system was selected for crusher no. 1 and no. 2 and all transfer conveyor areas for dust control.

16.5 Emergency Considerations

During initial development of the service declines, there will be two ways out of the mine for people working in those areas; but for the first 18 months of the operation, there will be only one way out for people working in the conveyor decline. Soon after development begins, a 12-man refuge chamber will be installed in the conveyor decline. This chamber will be moved down the conveyor decline as development progresses.

When the mine goes into production, there will be three exits. The service declines will be the primary escape routes for all mine personnel, and will be fresh-air ventilation intakes. Production and development vehicles will be able to drive to the surface through the service declines. The conveyor decline will be the secondary escape route, and will be a ventilation exhaust.

Table 16-8: Contaminant Production Expectations on an Annual Basis

	NOx (kg)	CO (kg)	PM (kg)	CO_2 (kg)	SO_2 (kg)
Value	171,290	770,740	8,567	152,848,393	1,437

Note: based on 19.5 hours of effective operating time. PM = particulate matter.

Radio will be the primary means of communication during an emergency, and all mine equipment will be radio-equipped. The secondary means of communication will be though the hard-wired mine telephones providing backup voice communications. This system will allow general paging and handset party-line conversations from each phone. Radios and mine telephones will be installed at all first-aid stations and refuge chambers.

A stench gas system will be used as a backup emergency warning method.

Underground fire protection design is based generally on standards and regulations of the National Fire Protection Association (NFPA), Factory Mutual (FM) and Underwriter Laboratories (UL).

A centralized vehicle dispatch system will be used to track and manage all vehicle movement and location in the underground mine development and production areas.

16.6 Backfill

16.6.1 Paste Plants

There will be four paste plants located on the surface over the life of the mine. Three paste plants will provide paste backfill for the Maturi mine area and one paste plant will be required at the Maturi Southwest mine area. Paste will be delivered by a combination of gravity and pumping via a system of boreholes with multiple pipes into the working areas. Distribution bays would be constructed from off the ramps at various levels and connected to each other with inter-level boreholes. From the borehole distribution bay, piping would be routed down inter-level boreholes to each mine level. At the completion of backfilling for a given mine level, backfill piping may be salvaged and utilized at other locations.

Initially two plants would be constructed with the third and fourth plants coming online in new locations (one plant at Maturi and one plant at Maturi southwest) when required. Each plant at the Maturi mine area will be designed to deliver up to 420,000 ft^3/d of backfill to the mine, the plant planned for Maturi Southwest will be designed to deliver up to 210,000 ft^3/day.

For mine scheduling purposes, it was assumed that an average of 750,000 ft^3/d of paste will be delivered from two operating plants. At any given point in the mine life, three main mining areas will be producing ore and, therefore, backfill was scheduled at a rate of 250,000 ft^3/day into each area. Based on the production schedule, the average demand on the backfill system over the course of a year will be approximately 480,000 ft^3/d.

In the long-hole stoping areas, a 28 day curing time was assumed prior to mining the up-dip or adjacent stopes in the panel. In the post-pillar cut-and-fill areas an overall 20 day curing time was assumed prior to mining adjacent to the fill. The 20-day period provides for a two-stage pour where approximately half of the void is filled, the paste is allowed to rest/cure for three days, and then the remainder of the void is filled and allowed to cure for 14 days.

The minimum percentage of binder content in the backfill was recommended by Golder to vary from 1.5% to 5% depending on mining method (i.e., 1.5% minimum to prevent liquefaction in lower portions of stopes, 5% cement cap in post-pillar cut-and-fill areas to facilitate driving on paste).

The binder content used in each of the mining methods was determined by TMM. For the purposes of the financial analysis, the binder was assumed to be 1% cement and 1% fly ash on average.

Additional information on the backfill plants, backfill distribution system and testwork is provided in Section 18.6.

16.7 Production Schedule

Scheduling was undertaken with the goal of providing 18.25 Mt/a of ROM ore to the process plant (50,000 t/d). To ramp-up as quickly as possible, three years of pre-production mining will be required to develop ramp systems, footwall drifts, stope accesses, ventilation raises, and other mine infrastructure. Because multiple working areas will be developed and numerous production faces will be exposed during the pre-production phase, it is expected that the mine will be able to achieve full ore production (i.e., 50,000 t/d) in Q2 of Year 1. This is a highly-optimized ramp-up schedule and, accordingly, the following measures will be implemented to minimize the ramp-up risk:

- There will be multiple ramp accesses to the mine. The planned dual service decline arrangement from the surface will allow for fewer logistical delays while a single decline arrangement will house the conveying system
- The service decline has been scheduled at an assumed advance rate of 20 ft/d per decline based on a single crew. The advance rate can be increased to 29 ft/d per decline by using a second crew for a slightly higher cost (5% overall). To be conservative, the single crew advance rate has been used for scheduling purposes and the two-crew cost has been used in the Project cost model
- The ventilation plenum design will provide flexibility to route air to various areas of the mine without the need for additional raises to the surface

- Many panels will have multiple connections to the internal ramp system allowing for haulage loops and multiple accesses for men/materials to a working area.

16.7.1 Productivity

Productivity estimates for mining long-hole and post-pillar cut-and-fill stopes and associated development were generated using a first principles methodology that considered:

- A shift work time of 19.5 hours per day
- Mechanical availability from manufacturers/experience
- A 17% delay (in addition to the total operating time).

The time to complete each activity within the mining cycle was then calculated to determine overall productivities for various activities. The total cycle time was then benchmarked against other mining operations that employ industry-leading practices.

To determine the cycle time for a multiple-heading scenario, the activities and subsequent travel delays were entered into a simulation model. Based on the time it takes to complete each activity, factoring in travel delays, and accounting for the development advance in ft for each heading, a total estimated advance rate in ft per day was generated for each multiple heading scenario. Where possible, additional equipment was assumed to reduce the time for the longest duration activity and maximize heading productivity.

16.7.1.1 Post-Pillar Cut-And-Fill

Productivities for post-pillar cut-and-fill are shown in Table 16-9. The productivities listed are on the high side when compared to operations with similar ground conditions.

Considerations that support higher assumed productivities for the Project are:

- Blasting will be “on demand” with no negative effect on other operations; larger blasts can occur during shift change
- There will be an experienced and skilled workforce.

Table 16-9: Productivity Rates Specific to Heading Types

Type	Size (ft)	No. of Headings (ft/d)*		
		1	2	Multiple
Ramps	20 x 20	16.9	28.8	55.7
Footwall drifts	20 x 20	16.9	28.8	48.8
Stope accesses	20 x 20	16.9	28.8	48.8
Slot Tier 1	46 x 40	9.9	19.9	56.1
Cross-cut Tier 1	46 x 40	16.3	30.9	75.3
Slot Tier 3	26 x 20	14.4	27.5	78.2
Cross-cut Tier 3	26 x 20	23.2	44.4	78.2
Remnant development drifts	20 x 20	16.9	28.8	48.8
Raisebored raises	varies	8.0		
Slot raises	varies	7.1		

*Used in production schedule. All rates based on 365 days/a.

- Operating time will be high at 19.5 hours per day
- Management will work diligently to avoid lengthy unforeseen delays that could affect productivities over an extended period of time
- Development drill holes will be 21 ft in length, which is longer than is typical for the industry
- In multiple heading scenarios, management will strive to have headings ready to cycle in equipment without any significant delays
- Equipment packages that are ordered will include maximum mobility with minimum equipment interference.

Achieving and maintaining the relatively high productivities that have been estimated for the Project will require constant vigilance on the part of management and supervisory personnel.

16.7.1.2 Long-hole Stoping

Long-hole stope mining productivities were also generated based on first principles. Long-hole stoping activities include the following:

- Stope surveying
- Slot raise drilling and blasting
- Stope drilling
- Stope loading and blasting
- Stope mucking.

Productivities include 83% mechanical availability and 17% delays. The time to complete each activity within the mining cycle was then calculated to determine overall productivities for various activities.

Ore mucked from the stope will be brought into a re-muck bay. The length of time to muck out a stope was calculated by summing the loading, hauling and dumping times and was based on a 15 yd^3 LHD. Where remote mucking is required, additional time was allocated to the cycle. Approximately 42% of all long-hole stoping mucking will be remote. A factor to compensate for handling oversize material within the stope was included in the mucking productivity calculation.

It was assumed in the long-hole stoping productivity calculations that the activities of mucking and blast hole loading will overlap 80% of the time. It was also assumed that the stopes will be cleared quickly of blast smoke, thus reducing re-entry time.

Table 16-10 summarizes the long-hole stoping estimated productivities.

The long-hole stoping productivity used focuses on mucking a stope as quickly as possible. This reduces interference from equipment and eliminates truck wait times. A second LHD is then used to remove the muck from the re-muck and dump into a truck without concern from stope productivity. This de-couples the stope production from truck loading. The second LHD has a much higher daily productivity then the stope mucking unit, even with truck wait delays, and is shared with other re-muck bays.

On average it is expected that it would take approximately 50 days to drill and 60 days to muck out a long-hole stope.

16.7.2 Production Schedule

All scheduling work was completed using Minemax's iGantt software (iGantt). The following parameters were used when creating the schedule:

- Quarterly ramp-up of the mine production rate (30% in January, 60% in February and 90% in March , i.e. beginning in Q1 of Year 1 and reaching capacity in Q2 of Year 1)
- Surface-stockpiled ore will be fed into the mill when required
- Long-hole stoping areas will be mined using a primary/secondary methodology. Mined-out stopes must be filled and cured prior to mining adjacent stopes
- Due to higher stresses in the Tier 4 area, a chevron-type mining front was recommended by Itasca to help transfer stresses up the panel and towards the regional pillars.

Table 16-10: Productivity Parameters, Long-Hole Stopes

Type	Size (ft)	No. of Headings (ft/d)			Rate
		1	2	Multiple	
Footwall drifts	20 x 20	16.9	28.8	48.8	
Stope accesses	20 x 20	16.9	28.8	48.8	
Long-hole stope drilling					5,670 t/d/stope
Long-hole stope mucking					4,830 t/d/stope
Long-hole stope backfilling					250,000ft^3/d/panel

- A 28 day backfill delay was used for all long-hole stoping areas. This constraint applies to mining adjacent stopes as well as to mining stopes that are up-dip of a backfilled stope
- A 20 day backfill delay was used for all post-pillar cut-and-fill areas. This allowed for a two-stage pour and a 14 day cure time after completion of pouring.

A detailed iGantt schedule was completed for the Maturi area with a particular focus on the early years. Remnant mining areas and the Maturi Southwest area were scheduled in less detail using a Microsoft Excel spreadsheet. For the spreadsheet schedule, production ramp-up, waste development timing and ore quantities were estimated based on the knowledge gained from scheduling the Maturi area in iGantt. For the next level of study the remnant and Maturi Southwest areas should be included in an iGantt (or similar) schedule.

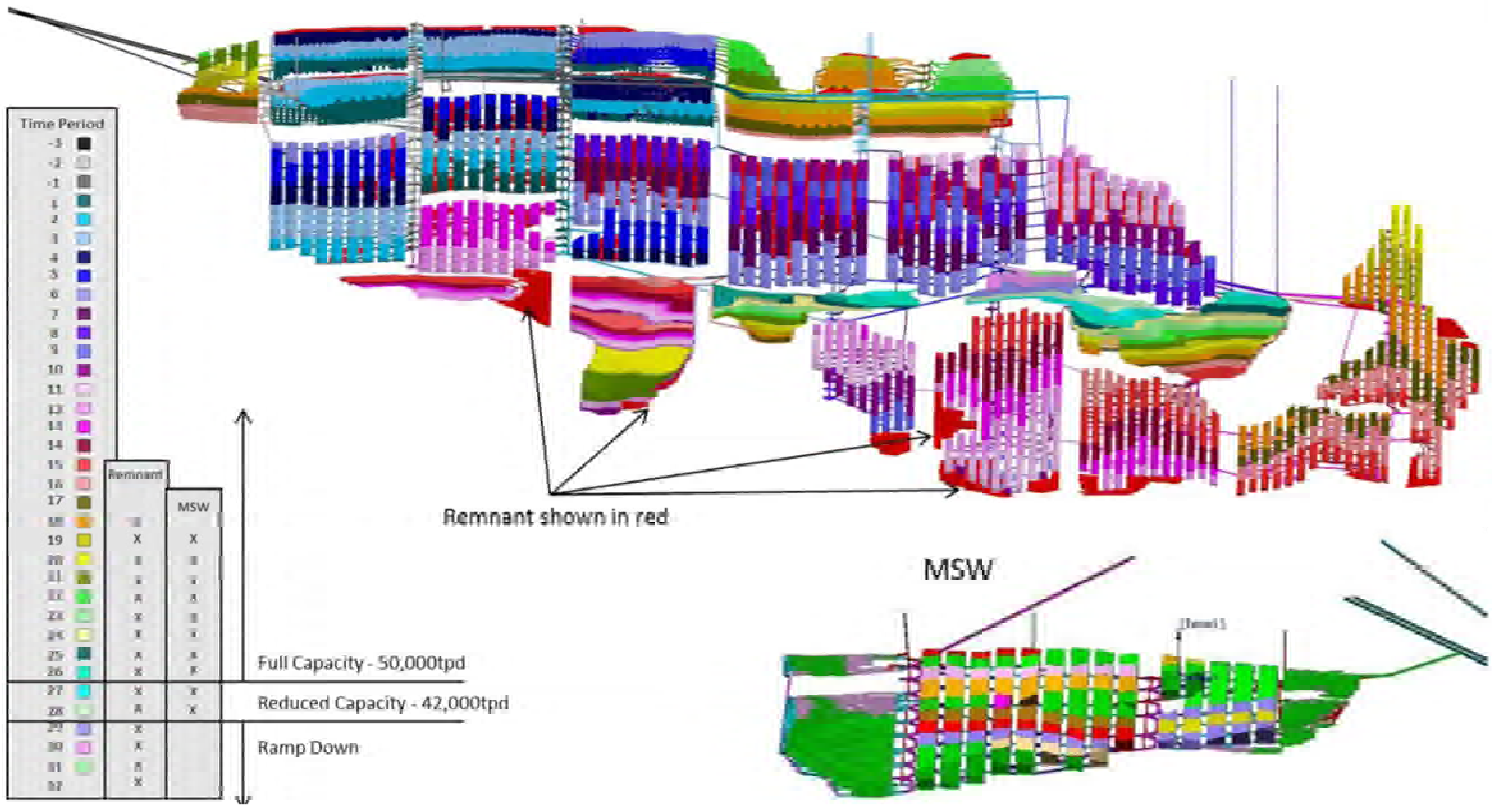
The planned stope mining schedule is shown in Figure 16-6, and labeling by tier was included in Figure 15-1 in Section 15. The overall production schedule is provided in Table 16-11.

Ore mined in Years -3 through -1 will be stockpiled on the surface and fed into the mill in Year 27. The maximum surface ore stockpile size during the pre-production period will be 1.75 Mt. 599 kt of additional low-grade material is assumed to be stockpiled underground in previously mined cut and fill areas.

Waste rock generated by development mining during the pre-production period will also be stockpiled on surface and consists of 1.3 Mt from contractor mining and 2.8 Mt from TMM crews resulting in a total surface waste rock stockpile size of approximately 4.1 Mt. All waste rock from Year 1 onward will be retained underground in mined-out stopes.

For costing/cashflow purposes material mined in Years 31 and 32 was added to Year 30. This assumes that material can be mined faster than currently shown in the production schedule by developing additional headings/working areas.

Figure 16-6: Stope Mining Sequence Plan



Note: Figure prepared by SRK, 2014. Figure looks west, and is schematic. MSW = Maturi Southwest. Time period indicated in legend is in years.

Table 16-11:Production Schedule

Year	Ore Tonnage (kst)	Cu (%)	Ni (%)	Pt (oz/st)	Pd (oz/st)	Au (oz/st)	Ag (oz/st)	Waste Tonnage (kst)
-3		—	—	—	—	—	—	576
-2	540	0.526	0.165	0.003	0.006	0.002	0.055	1,659
-1	1,215	0.564	0.176	0.003	0.006	0.002	0.058	1,825
1	16,471	0.712	0.233	0.004	0.009	0.002	0.073	1,250
2	18,494	0.700	0.230	0.004	0.009	0.002	0.072	1,752
3	18,481	0.719	0.238	0.004	0.009	0.002	0.074	1,235
4	18,327	0.705	0.237	0.004	0.010	0.002	0.073	1,164
5	18,250	0.674	0.231	0.004	0.010	0.003	0.071	1,566
6	18,250	0.647	0.220	0.005	0.011	0.003	0.068	1,037
7	18,247	0.668	0.223	0.005	0.011	0.003	0.071	944
8	18,253	0.666	0.219	0.005	0.010	0.003	0.072	1,191
9	18,250	0.654	0.215	0.005	0.011	0.003	0.072	1,380
10	18,250	0.649	0.202	0.005	0.012	0.003	0.071	1,185
11	18,250	0.610	0.183	0.006	0.013	0.003	0.068	1,492
12	18,231	0.584	0.182	0.006	0.013	0.003	0.062	862
13	18,245	0.611	0.199	0.006	0.013	0.003	0.063	1,860
14	18,253	0.609	0.184	0.006	0.014	0.003	0.064	1,357
15	18,272	0.605	0.181	0.007	0.016	0.004	0.065	666
16	18,250	0.625	0.187	0.007	0.015	0.003	0.069	544
17	18,251	0.634	0.190	0.006	0.015	0.003	0.069	939
18	18,251	0.594	0.185	0.005	0.012	0.003	0.064	524
19	18,251	0.565	0.177	0.004	0.010	0.002	0.061	604
20	18,250	0.547	0.173	0.004	0.009	0.002	0.059	667
21	18,250	0.527	0.175	0.004	0.008	0.002	0.056	609
22	18,250	0.513	0.169	0.003	0.008	0.002	0.054	823
23	18,250	0.509	0.168	0.004	0.008	0.002	0.053	492
24	18,250	0.506	0.167	0.004	0.008	0.002	0.052	484
25	18,250	0.497	0.164	0.003	0.007	0.002	0.050	401
26	18,250	0.483	0.158	0.003	0.007	0.002	0.049	307
27	15,660	0.457	0.149	0.002	0.006	0.001	0.046	330
28	15,073	0.442	0.144	0.002	0.006	0.001	0.045	306
29	10,906	0.449	0.148	0.002	0.006	0.001	0.046	156
30	10,174	0.451	0.153	0.002	0.006	0.003	0.047	181
Total	*526,843*	*0.593*	*0.191*	*0.004*	*0.010*	*0.002*	*0.063*	*30,368*

Note: Ore mined in years -3 through -1 and select low grade material in Years 1 to 4 is stockpiled on underground or on surface and fed into the mill in later years. The waste tons in Years -3 and -2 include waste tons mined by the contractor.

The LOM production schedule includes:

- Years 1 to 5: Production mining will occur in Tiers 1 and 2, with ore initially coming from five post-pillar cut-and-fill panels and a single long-hole stoping panel. Continued development of the ramp system and footwall drifts will allow a second long-hole stoping panel to be brought into production in Year 2. All mining will take place in the southern portion of the Maturi deposit in these early years and all ore will be sent to a crusher that will be located at the base of the service declines (crusher 1)
- Years 6 to 10: Early in the period the majority of production will be from long-hole stoping stopes in Tier 2. Development mining and a small amount of ore mining

will take place in Tier 3. In the later years, production will shift northward to the central/northern area of the Maturi deposit. In Year 6 a second crusher (crusher 2) will be commissioned at a location near panel 03d. Both crushers will be used during this period and haul trucks will be routed to the nearest crusher. Paste Plant #3 will be commissioned in Year 6 to support the expanded Maturi mine footprint. Development will continue northward with the establishment of the lower ventilation plenums and additional raises to the surface.

- Years 11 to 15: Mining will begin in Tier 4 moving northward. Long-hole stoping stopes will provide most of the mill feed; however, post-pillar cut-and-fill areas in Tier 3 will also contribute some ore. Tier 2 long-hole stoping panels will be exhausted early in this period and development will continue northward at depth. All ore will be trucked to the lower crusher
- Years 16 to 20: Mining will take place in the northern part of the deposit with most production being sourced from Tier 4 long-hole stoping areas and Tier 1 post-pillar cut-and-fill areas. Some ore production will also come from Tier 3. Development mining in the Maturi areas will taper off during the period and Tier 4 long-hole stoping production also will taper off during the latter part of the period. Remnant ore mining will begin in year 18 and will contribute approximately 10% of the production. Development mining will begin in the Maturi Southwest area. Most ore will be trucked to the lower crusher; however, Tier 1 production will be sent to the upper crusher. The Maturi Southwest paste plant will be commissioned in Year 20
- Years 21 to 25: Production from the Maturi area will end as the Tier 4 long-hole stoping and Tier 1 post-pillar cut-and-fill areas are mined out. The percentage of remnant ore mining will increase and the Maturi Southwest area will contribute up to 40% of the mill feed during this period. Ore will be sent to both underground crushers and to the crusher located on the surface. The haulage distance between the Maturi Southwest northern area and the surface crusher is approximately the same as the haulage distance between the Maturi Southwest northern area and the upper crusher
- Years 26 to 30: The mill feed requirement will be reduced to an average of approximately 40,000 st/d. The Maturi Southwest area will begin ramping down and all remaining ore will be sourced from remnant mining areas in the Maturi area. It has been assumed that ore will be sent to all crushers during the period; however, further optimization may show that it would be more economical to shut one of the crushers down
- Ore mined in Years -3 through -1 will be stockpiled on the surface on the SRSF. The bulk of the stockpile will be fed to the mill in 4Q Year -1 to support bedding, commissioning, and start-up of the concentrator and tailings system. The maximum surface ore stockpile size during the pre-production period will be

1.75 Mst. The balance of the low-grade stockpile will be fed to the mill in Years 26 and 27. A total of 599 kst of additional low-grade material is assumed to be stockpiled underground in previously-mined cut-and-fill areas

- Waste rock generated by development mining during the pre-production period will also be stockpiled on surface and consists of 1.3 Mst from contractor mining and 2.8 Mst from Owner crews resulting in a total surface waste rock stockpile size of approximately 4.1 Mst. All waste rock from Year 1 onward will be retained underground in mined-out stopes.

16.8 Mining Equipment

16.8.1 Primary Equipment

Primary equipment includes all mobile equipment required to do direct mining in development headings and stopes. The fleet summary is provided in Table 16-12.

16.8.2 Secondary Equipment

The secondary equipment fleet was selected to support the development and stoping operations. The fleet will include haul trucks, ground support equipment, mine service vehicles, and personnel transport equipment. Equipment requirements are provided in Table 16-13.

16.9 Comments on Section 16

The larger slot size of 46 ft wide x 40 ft high with 34 ft wide x 40 ft tall pillars used in Tier 1 post-pillar cut-and-fill panels carries a higher risk than the originally recommended 40 ft x 40 ft slots. This increase in extraction accounts for approximately 10 Mt, or 2%, of the overall Mineral Reserve. The slot size should be further evaluated for the next phase of work. TMM advises that they intend to undertake additional work during more detailed engineering phases to verify all stope dimensions and ground support plans.

An Arena software simulation (or similar) should be undertaken to determine what equipment interference can be expected in the production haulage system.

SRK notes that any early access to the underground, in conjunction with a bulk sample or other program, would provide invaluable information for mine design, productivities, costs, etc. It is also suggested that consideration be given to recruiting key underground technical and management staff in support of optimization during detailed design phases.

Table 16-12: Primary Equipment List

Type	Manufacturer	Description	Use	Yr. 1	Peak Years
2-boom jumbo	Atlas Copco	21 ft feed, electric hydraulic	Development	5	8
3-boom jumbo	Atlas Copco	21 ft feed, electric hydraulic	Production	3	3
LHD	Caterpillar	15 yd^3	Development and Production	14	18
Medium Bolter	Atlas Copco	Mechanized	Development and Production	9	11
Large Bolter	Atlas Copco	Mechanized	Production	5	5
Powder truck	Atlas Copco	ANFO, boom and basket	Development and Production	10	14
ITH drill	Sandvik Cubex	Aries series	Production	4	10
Slot raise borer	Machines Roger	V30, 30 in. dia. up/down holes	Production	2	3
ANFO charger	McLean AC-3	3,000 lbs	Production	3	8
Haul truck	Atlas Copco	MT85, 94 st truck	Production	20	28
Haul truck	Caterpillar	AD60, 66 st truck	Development	6	6

Note: Table does not include nine trucks assumed to be transferred to the Owner by rapid development contractor

Table 16-13: Secondary Equipment Requirements

Type	Manufacturer	Description	Year 1	Peak Years
Shotcrete transmixer	Atlas Copco	5.9 yd^3 transmixer	2	2
Shotcrete sprayer	Atlas Copco	Maxima boom	2	2
Cable bolter	Atlas Copco	Cable bolting	1	2
Scalers	Atlas Copco	Diesel hydraulic	4	5
Scissor truck	Atlas Copco	Tilting platform	5	8
Pipe handler	Getman	A64 SL Hanger	4	4
Fan handler	Marcotte	M60 4 in 1	5	5
Personnel carrier	Atlas Copco	28 person transporter	3	5
Haul truck	Caterpillar	AD30, 33 st truck	2	4
LHD	Atlas Copco	ST14, 8.4 yd^3	2	2
LHD	Caterpillar	4 yd^3	1	2
Jumbo	Sandvik	Twin boom, 16 ft feed	2	2
Remote blockholer	MacLean	BH3	1	3
Flatbed	Atlas Copco	Hiab with boom	6	7
Grader	Caterpillar	12 ft blade	3	4
Backhoe	Kabota		1	1
Fuel truck	MacLean		3	4
Service vehicle	Toyota		34	30
Utility truck	Marcotte	DT3 mobile compressor	2	2
Calcium spreader	Tracks and wheels		1	2

Golder notes that the binder assumed for paste backfill would consist of a combination of cement and fly ash. Golder has completed strength testing for paste with cement binder; however, no paste backfill strength testing with a combination of cement and fly ash binder has been performed to date. Testing of paste backfill strength properties with fly ash should be performed in future studies to validate the assumed paste backfill strengths.

Golder notes the backfill scheduling requirements have been developed based upon peak backfill rates. These scheduling requirements result excess capacity and low utilization of the paste plants. Excess capacity provides for flexibility in the backfill scheduling; however, future studies should consider optimization of the paste backfill system, which could also result in optimization of the slurry tailings transport system.

17.0 RECOVERY METHODS

17.1 Process Flowsheet

A simplified schematic of the proposed process flow sheet is included as Figure 17-1, a more detailed flowsheet for the plant in Figure 17-2 and a general 3D schematic layout for the process plant and facilities is shown in Figure 17-3.

17.1.1 Coarse Ore Handling and Crushing

The primary crushing plant consisting of a single 60 in. x 89 in. gyratory crusher will be located in a central location underground within the Maturi deposit and will process run-of-mine feed in open circuit. Primary crushed ore will be conveyed to the surface via an overland conveyor and fed to the coarse ore primary crushed stockpile.

The stockpile will have a live capacity of 33,000 st or approximately 16 hours of continuous operation. The stockpile will discharge onto three operating (one on standby) feeders located within the reclaim tunnel, which then will feed by conveyor to the primary semi-autogenous grind (SAG) mill.

17.1.2 Grinding and Classification

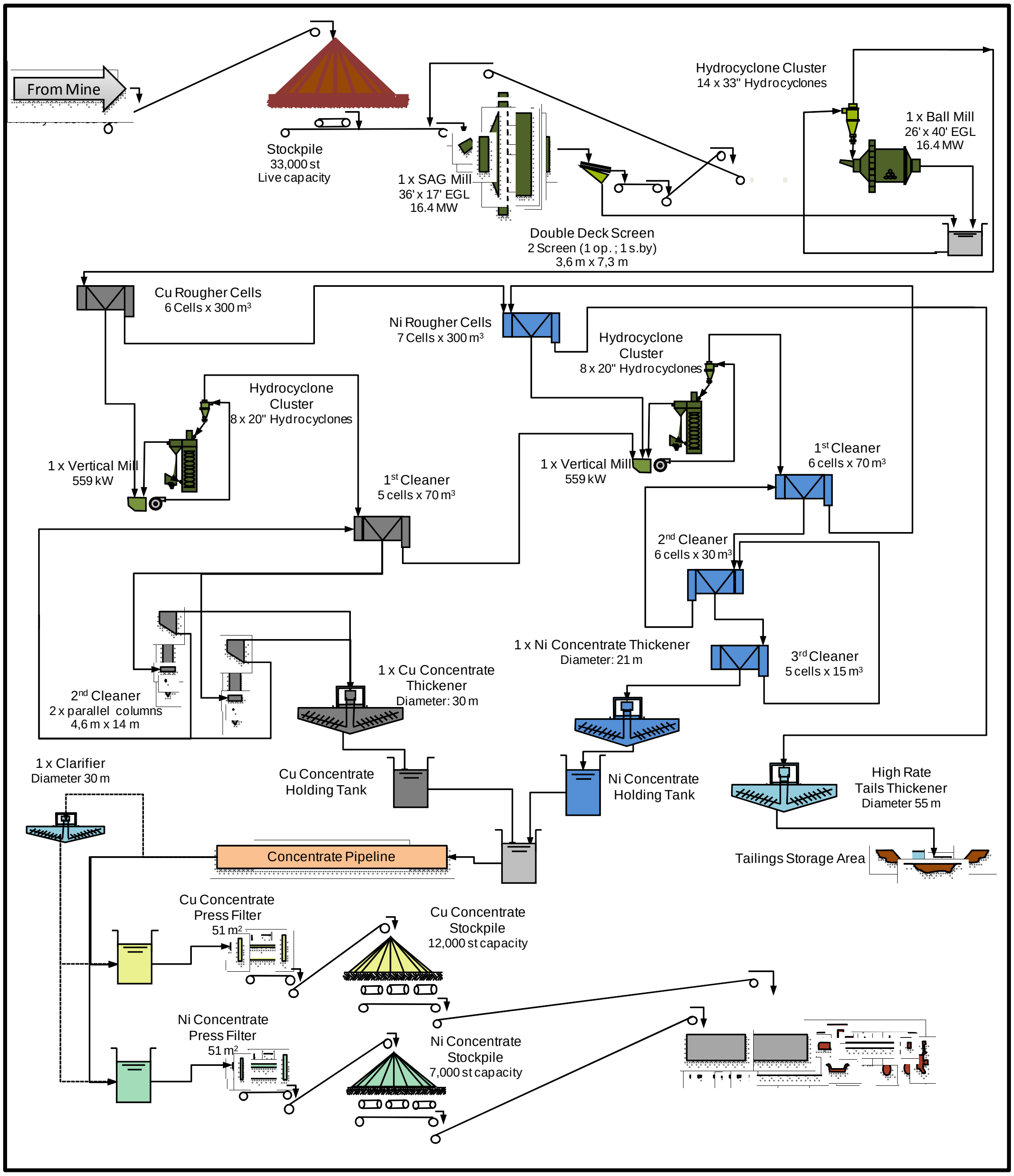
The primary grinding circuit will consists of a single 36 ft x 17 ft effective grinding length (EGL) SAG mill and a single 26 ft x 40 ft EGL ball mill. The SAG mill will be operating in closed circuit with a trommel and vibrating screen classification. The screen's oversize stream will recirculate back to the SAG mill feed. The screen's underflow stream will be the SAG circuit's final product feeding the ball mill grinding circuit; no pebble mill is incorporated into the design. The ball mill will be operating in closed circuit with a hydrocyclone classification cluster consisting of 14 x 33 in. diameter cyclones, each cluster fed by a dedicated cyclone feed pump. The cyclone overflow product P80 will be 120 µm feed to the flotation circuit.

17.1.3 Sequential Flotation Circuit

The flotation circuit essentially comprises two separate flotation circuits inside a single building producing separate copper and nickel/copper concentrates.

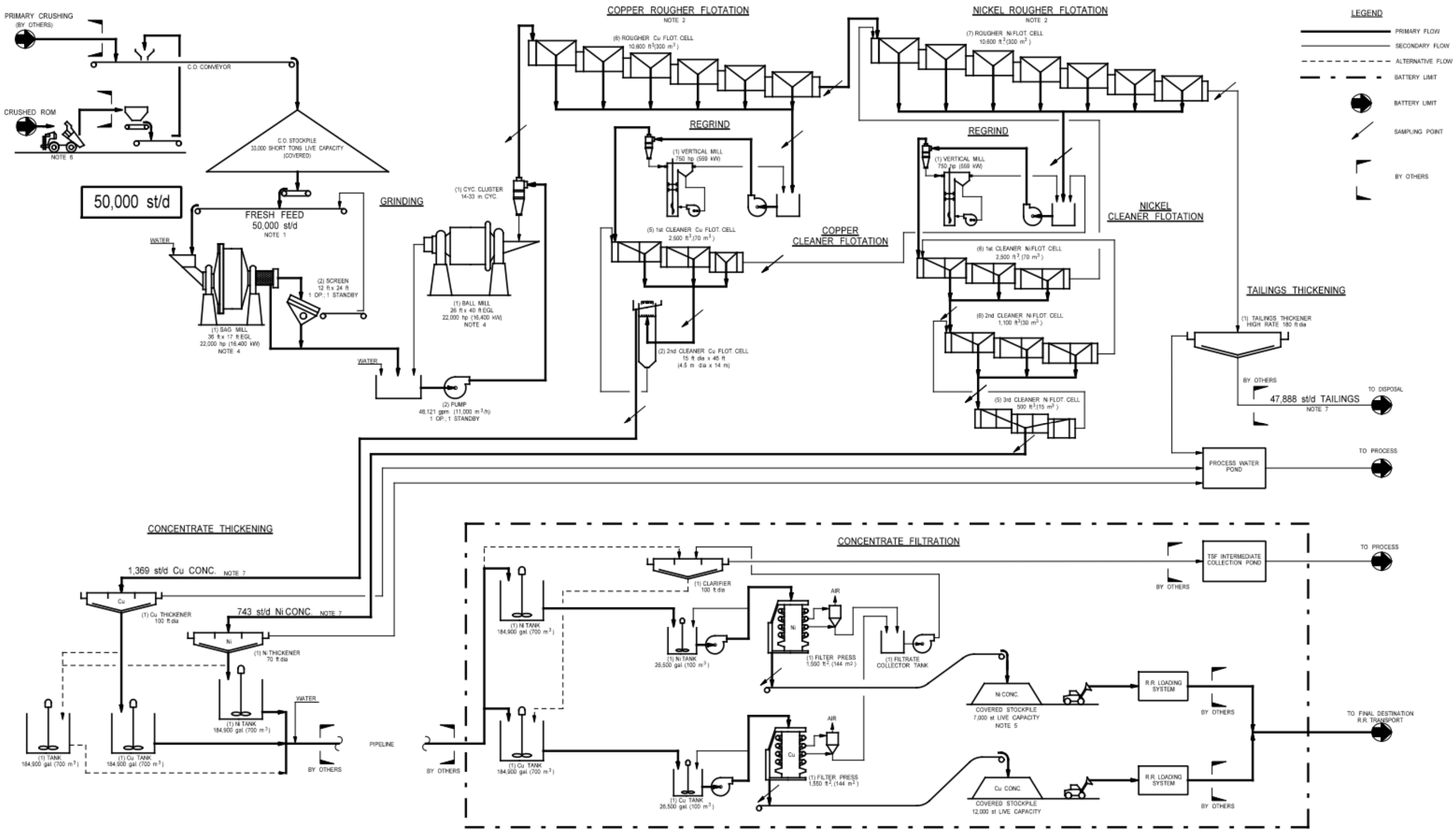
Cyclone overflow pulp will be fed into a single line of six 300 m^3 rougher flotation tank cells. Combined rougher concentrate will be pumped to a bank of 8 x 20 in. hydrocyclones where it will be classified. Cyclone underflow product will be reground in a 750 hp vertical mill. Cyclone overflow, at a P80 of 45 µm will then be fed to the first cleaner flotation circuit consisting of single bank of five 70 m^3 tank cells.

Figure 17-1: Proposed Prefeasibility Schematic Flowsheet



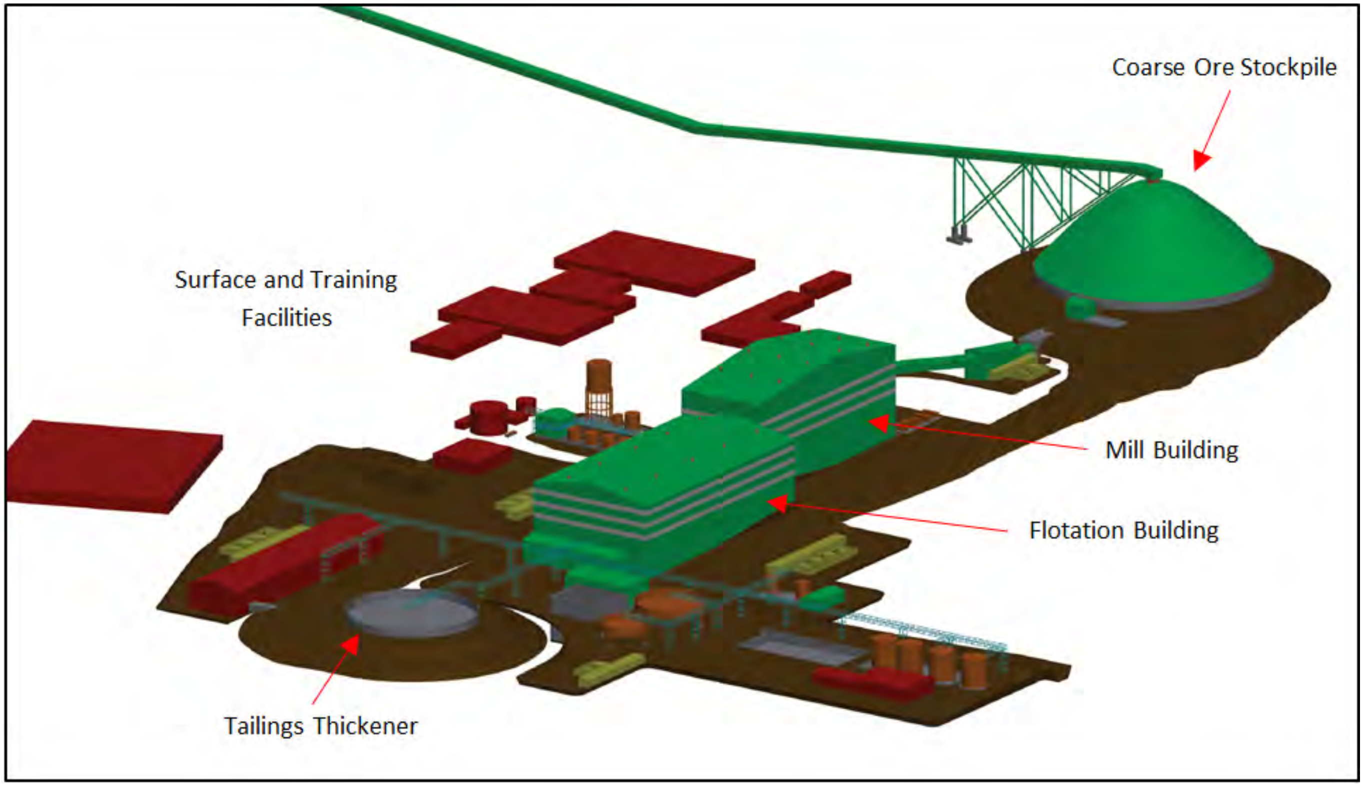
Note: Figure prepared by AMEC, 2014

Figure 17-2: Proposed Process Plant Detailed Process Flowsheet



Note: Figure prepared by TMM's Independent Engineer, 2014

Figure 17-3: Process Plant and Facilities General Arrangement



Note: Figure prepared by TMM's Independent Engineer, 2014

The first cleaner concentrate will be combined and pumped to a second cleaner flotation consisting of two parallel 15 ft diameter x 46 ft high column flotation cells. First cleaner flotation tails will report to the nickel flotation circuit regrind mill feed box. Second cleaner flotation concentrate will be the final copper concentrate product whilst the second cleaner flotation tails will report back to first cleaner flotation feed. Rougher flotation tailings will be pumped to the nickel flotation circuit.

The first stage of nickel flotation will be the roughing flotation stage, consisting of a single line of seven 300 m^3 rougher flotation tank cells with the rougher tailings being final plant tailings. Combined nickel flotation concentrate will report to the nickel flotation circuit regrind mill feed box feeding the nickel regrind circuit classification cyclones. In this feed box the pulp will be combined with the tailings pulp from the first stage of copper cleaner flotation. This combined pulp will be pumped to a bank of eight 20 in. diameter hydrocylones. Cyclone underflow product will be reground in a 750 hp vertical mill.

Cyclone overflow, at a P80 of 45 µm will then be fed to the first cleaner flotation circuit, which will consist of a single bank of six 70 m^3 tank cells. Combined first cleaner concentrate will be combined and pumped to the second stage of nickel cleaning flotation, consisting of six 30 m^3 tank cells. First cleaner flotation tailings will report back to the feed of nickel rougher flotation. Combined second cleaner flotation concentrate will report to the feed of the third stage of nickel cleaner flotation which will consist of five 15 m^3 tank cells. Second cleaner stage tailings will report back to the feed of the first flotation cleaners. The third cleaner stage combined concentrate will also be the final concentrate, whilst the third cleaner stage tailings will report back to the feed of the second cleaner flotation stage.

17.1.4 Thickening

The thickening stage will consist of two separate copper and nickel concentrate thickening stages. Final copper concentrate will flow by gravity to a 30 m diameter thickener where the concentrate will be thickened to 65 % solids w/w and then pumped to a single 31.6 ft diameter and 41 ft high holding tank. Final nickel concentrate will reports to a 21 m diameter thickener where it will also be thickened to 65% solids w/w where it will then be pumped to an identical holding tank for the copper concentrates. The concentrates will then be pumped down a single concentrate pipeline in packets to a common filter facility. A water flush tank of 30 ft diameter and 35 ft high will be used to flush the line between copper and nickel concentrates. A concrete-lined emergency pond of 1,500 m^3 capacity will also be available when required.

17.1.5 Concentrate Filtration, Storage and Rail Load-out Facilities

The filter plant will consist of two separate press-type filter circuits of 51 m^2 capacity, each dedicated to a single concentrate product (copper or nickel) and functioning in a batch cycle process mode.

Filter cake product will discharge onto a collection conveyor located underneath each filter press. Each of the filtered concentrates will then be conveyed to its respective concentrate storage feed stockpile. Filtrate from both filters will be combined in a common collection tank before being pumped to a 30 m diameter clarifier. Clarifier underflow will report back to the filtration stage.

This facility would comprise the storage of both copper and nickel concentrates in two separate buildings, each with different storage methods and sized for 10 days of production. The concentrates would be transferred from the filter plant building to storage buildings through two separated conveyor belts. Copper concentrate would be stored in an enclosed conical stockpile (12,000 st capacity) and nickel concentrate would be stored in an enclosed steel building (7,000 st capacity).

These facilities are an integral part of both copper and nickel concentrate storage buildings and copper and nickel train-loading zones. Both the copper and nickel concentrate load-out facilities are designed to use open top (covered during transportation) gondola-type rail cars that will be filled by a front-end loader to a maximum shipping weight of 100 st. The load-out area will be divided into three separate zones:

- Cover removal (Zone 1)
- Filling and cover replacement (Zone 2)
- Car washing (Zone 3).

The site is designed to receive a train length of 20 gondola cars plus the locomotive which would pass through the load-out building to terminate on the spur line. To begin the loading sequence, the railcars advance into the load-out area and stop at Zone 1 where the cover is removed using the bridge crane fitted with a powered cover removal assembly. The train then advances to Zone 2 until the open car is over the weigh scale. Once the railcar has been filled and its weight verified, the crane reinstalls the cover and return to Zone 1 to remove the next railcar cover. Once complete, the filled car advances to Zone 3, the car-washing zone, to clean off any spillages that had occurred during the filling process. The car-washing zone will contain two wash-down hose stations, one located on each side of the rail car. The floor area will slope towards a sump with an installed sump pump to reclaim the washing water.

17.1.6 Reagent and Supply Plants

The reagent facilities will be located in an open area to the north of the grinding facility, and will include distribution loops and day tanks, with metering pumps as required for each reagent type. The storage tank for each reagent will be located in a separate containment area containing an open sump that will be accessible to portable pumps as and when required. An enclosed building (insulated and heated) will house preparation/mixing equipment for bulk reagents delivered in either maxi-sacks or containers.

The equipment for reagents supply such as milk-of-lime preparation and distribution, flocculant, and flotation reagents, have been estimated according to plant capacity and consumptions from preliminary testwork results. Reagents utilized in this facility include copper collector (3418A), nickel depressants (TETA and Na_2SO_3), MIBC frother, nickel and PGE collector (SIPX), and two systems for spare reagents.

The main elements of this facility would include:

- Five separated agitated storage tanks for reagents
- One lime storage silo with 830 st capacity
- One lime slaking preparation tank
- One lime distribution tank, insulated with heat tracing and agitator
- Electrical room and transformers
- One bulk reagent preparation area in an enclosed and insulated structural steel building.

17.1.7 Tailings Thickening

This facility comprises the installation receiving flotation tailings. The final tailings from the thickeners will be thickened in a 180 ft diameter high rate tailings thickener to 73% solids w/w and pumped to either tailings disposal or the mine area paste plants. A 12,000 m^3 capacity concrete emergency pond will be provided in the event of thickener failure or a slime overflow event.

Tailings thickener overflow solutions together with the copper and nickel concentrate thickener overflow solutions will be collected and pumped to the process water pond.

17.1.8 Plant Auxiliary Facilities

The air compressor system and flotation air blowers will be located in a common noise-controlled room attached to the flotation building. Hoist beams with hoists will be installed to service the equipment. The main elements of this facility will include:

- Three plant/instrument air compressors with 1,840 scfm 109 psig capacity

- One plant air receiver
- One instrument air dryer and receiver
- One column air compressor rated at 110 psi and one 25 m^3 air receiver
- Flotation blowers.

The other services that allow connection and execution of all previous facilities in the concentrator will include the following:

- Site work activities: in-plant circulation roads, pipe racks, and duct banks
- Sewage system
- Makeup water system;
- Potable water system
- Fire water system
- Thickener reclaim (process) water
- Process water system
- Natural or propane gas distribution
- Power distribution system
- Electrical emergency system
- Fire protection and detection system
- Concentrator plant communication.

17.2 Plant Design

17.2.1 Design Basis and Criteria

The design criteria defined were based on information provided to AMEC as follows:

- Concentrator Design Basis Memorandum, December 12, 2013
- Meeting Minutes at St. Paul, Review of Design Basis, Prefeasibility Study, 11 and 12 December, 2013 (10-41)
- Report PFS Comminution Program Millpower2000 Simulations 7 March 2013 and 19 June 2013.

However, these have been adjusted for the inclusion of Maturi Southwest with Maturi material into the LOM plan provided. The figures displayed in this sub-section are the operational figures over the project life. Treatment of Maturi Southwest material commences in the 19th year of operation according to the latest LOM plan.

The major criteria defined are defined in Tables 17-1 to 17-4.

17.2.2 Metallurgical Forecasting

Metallurgical projections for the prefeasibility financial model have been created through the sequential use of rougher kinetics testing, locked cycle testing and pilot plant testing. The question of scale-up from pilot plant to production plant will be discussed briefly in this section though no attempt will be made to modify the laboratory/pilot plant based model to predict actual plant performance. Models provide predictions of rougher flotation recoveries, and the performance of the cleaner circuit in the processing of the rougher concentrate which are derived from metallurgical testing and are based on input parameters available in the resource model.

The rougher model has been built using information from the 98 rougher variability tests. This information has specifically been used for rougher flotation prediction because of the robust data population, and because the samples tested span the entire spectrum of grades of material expected to be fed to the mill through the life of the mine. This results in a prediction of metal recovery to the combined copper and nickel rougher concentrate.

At this point in time the distribution of metal to either the copper or nickel rougher concentrates is not defined. This is due to the following:

- Most of the nickel floated to the copper rougher concentrate ultimately deports to the nickel final concentrate in any case
- The principal driver behind the copper split between the copper and nickel final concentrates is in fact the copper regrind and, especially, the first copper cleaner circuit
- A small but significant portion of the copper floated to the copper rougher concentrate is diverted to the nickel concentrate.

No attempt was made to model the circuit based on specific metal recoveries to specific rougher concentrates. The key criterion in roughing was the metal recovery to the combined concentrates. Whether the metal floated to the rougher concentrates reports to the copper final concentrate, nickel final concentrate or the final tails (through the cleaner tails) is dictated by the cleaner metallurgy. This can only be predicted through actual locked-cycle metallurgy when the circuit is in stable operation. Batch cleaner tests provide some cursory insight but are poor predictors of closed-circuit response. Accordingly, a dataset comprising locked-cycle information on some 18 different composites has been created to predict cleaner metallurgy.

Table 17-1: Process Design Criteria—Operational Time

Item	
Operating days per year	365
Operating hours per day	24
Availability %	92

Table 17-2: Process Design Criteria—Throughput and Production

Stream	Throughput (st/a) #	Throughput (st/day) #
Plant feed	18,250,000	50,000
Copper concentrate	369,344	1,012
Nickel concentrate	185,969	509
Final tailings	17,694,687	48,479

Note: # - average values during peak production period

Table 17-3: Process Design Criteria—Concentrate Grades

Item	%
Feed Copper Grade	0.59
Copper Concentrate Copper Grade	25.4
Feed Nickel Grade	0.19
Nickel Concentrate Nickel Grade	10.5

Table 17-4: Process Design Criteria—Metal Recoveries

Item	Recovery (%)*
Total Copper Recovery	93.4
Total Nickel Recovery	54.7

Note: * Includes recovery to both copper and nickel concentrates over the LOM. This includes copper recovery to copper concentrate of 85.5% and copper recovery to nickel concentrate of 7.9%. Nickel recovery is to nickel concentrate only

The methodology described above therefore provides solid bench-scale information on the closed-circuit behavior of a wide variety of different material types from the Maturi deposit. Bench-scale testing in itself has limitations, especially when predicting the ultimate response of a relatively challenging selective float such as Cu/Ni separation, and the cleaning behavior of pentlandite. As has been described earlier, copper–nickel separation in the copper circuit is achieved using a combination of (mostly) nickel-selective depressants (TETA and sodium sulfite) together with starvation doses of collector. In reality, the difference in selectivity is best exploited using column flotation, which cannot be simulated in the laboratory but has been demonstrated in the study through piloting.

Some of the samples tested contain as little as 0.13% Ni, of which only 60–70% was in sulfide form (0.08 to 0.09% NiS). With such low concentrations of a relatively weak floating mineral, laboratory testing struggles to achieve concentrate grades typically achieved in the plant. A pilot plant is the best way to predict actual grades, and in this project piloting proved able to manipulate nickel concentrate grade and recovery in a way not achieved at bench scale. Consequently, the laboratory-based metallurgical data have been compared with pilot plant data on the same material, allowing for incorporation of the effect of column flotation on copper–nickel separation, as well as the effect of continuous operation on nickel cleaning.

17.2.2.1 Rougher Flotation Prediction

Rougher flotation batch tests were conducted on 98 samples. Of these 98 samples:

- 41 were from the Shallow zone, 36 from the Deep zone and 21 from the Deep East zone
- 58 samples were from the S3 geological unit and 40 samples were from S2
- Ranged in grade from 0.2 to 1.3% Cu.

The test work and results are described in Section 13. The average total copper rougher recovery was 96.4%. Irrespective of head grade, copper recovery was remarkably consistent in all 98 tests. There was very little evidence of a copper head grade/recovery relationship for the deposit as a whole (Figure 17-4).

The difference in recovery by head grade for S2 and S3 samples is shown in Figure 17-5.

There is a weak correlation between copper head grade and recovery for S2. Although an R-squared correlation of 0.26 is not strong, there is sufficient evidence of a weak head grade/recovery relationship to use it for recovery forecasting. The S2 copper recovery algorithm is:

*Copper Recovery = 2.5231*Ln (Cu head grade, ppm) + 74.147.*

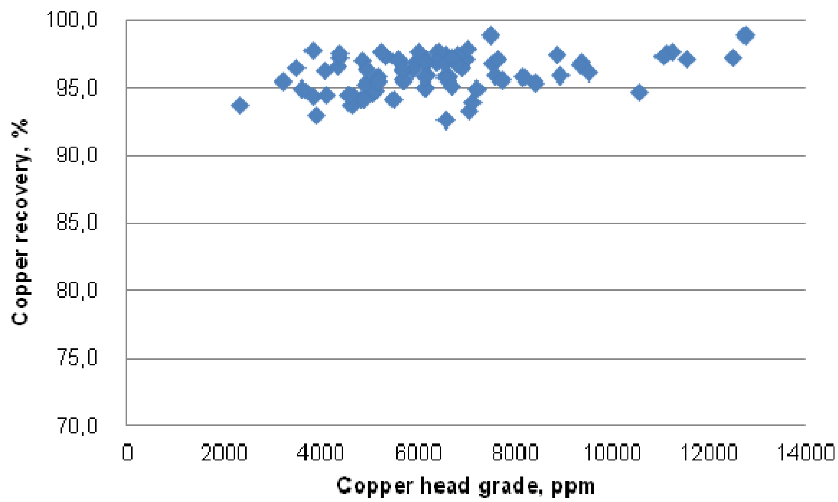
The mean S3 copper recovery was 96.4% and is assumed to be fixed.

Nickel recovery is driven by the proportion of nickel present as pentlandite. As most of the remainder is in olivine, the dominant magnesium-bearing silicate, the recovery of nickel is related to the Mg content in the sample, with the higher Mg content being linked to lower nickel recoveries. Perhaps the most logical algorithm would include both Mg and Ni, in the form of Mg/Ni; however, in reality a better fit occurs when the algorithm is a function of Mg/S. The R-squared fit on this is 0.73 as shown in Figure 17-6.

The curve best fitting this relationship is:

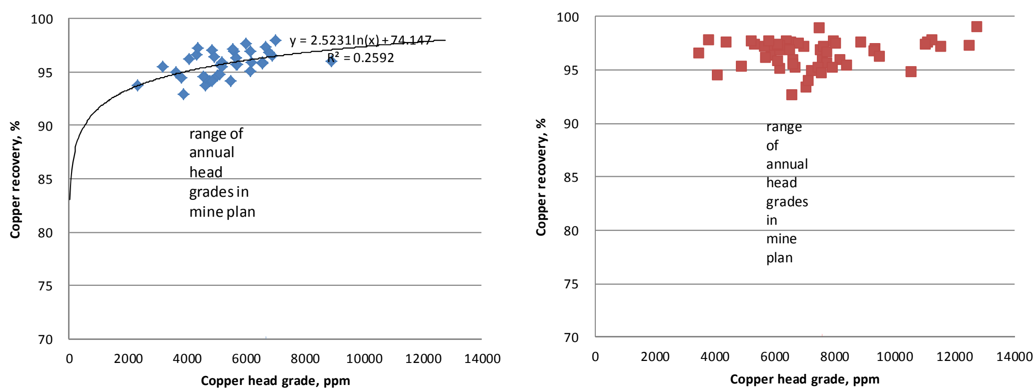
Nickel recovery = -15.72 Ln (Mg %/S %) + 99.247.

Figure 17-4: Copper Grade versus Copper Recovery



Note: Figure prepared by TMM, 2014

Figure 17-5: Copper Grade versus Copper Recovery - S2 and S3 Geological Units



Note: Figure prepared by TMM, 2014

The potential for specific geological unit or location-driven correlations within this relationship has been explored. None of the S2 or S3 sample sets, in any of the three locations yielded recovery relationships significantly different from the relationship described above, so it has been assumed that this relationship would indeed apply to all samples in the Maturi deposit.

17.2.2.2 Cleaner Flotation Prediction

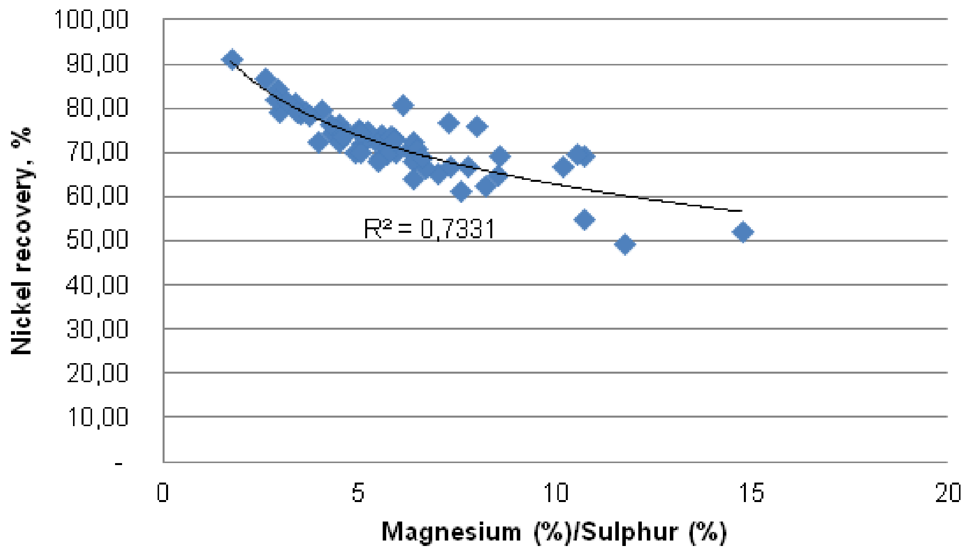
The data and methodology employed to arrive at predictions of laboratory-based cleaner performance are described. This includes recoveries from rougher concentrate to copper and nickel final concentrates, as well as losses from the rougher concentrates to final tails. Also included is the rationale used to predict the grades of the copper and nickel concentrates, as well as the resulting cleaner performance predictions for the economic model. Only locked-cycle metallurgy operating under equilibrium conditions can be used for this prediction as the metallurgical behavior of circulating loads has substantial influence on process performance.

The database comprising locked cycle tests on the 18 composites, each operating under stable conditions, has been examined to allow for determination of future cleaner metallurgy. The composites tested in locked-cycle mode were:

- Pilot plant composite (PP-3): The primary purpose of testing this composite was to allow for direct comparison with the ALS pilot plant operation
- SDC: These composites were sourced from the two dominant geological units (S2 and S3) and from the three broad locations in the mineable area (Shallow, Deep and Deep-East). The SDCs were designed to span a range of pyrrhotite:pentlandite abundance (Po:Pn) ratios as it was strongly suspected that high Po:Pn ratio materials would yield poor quality nickel concentrates
- LOM Composites (LOM): These composites were mostly a blend of the variability samples, but adjusted for grade to represent four discrete phases in the life of the mine (1–3, 4–8, 9–19 and 20–32 years)
- Supercycle composite: prepared from SDCs to represent a LOM average composite.
- Pyrrhotite rejection composite.

Any differences in cleaner metallurgy arising from geological unit or head grade can be established, so linking with the equivalent data from the mine plan to allow for a prediction of cleaner performance on an annual basis. This, linked with the rougher data in the previous subsection, then provides the complete picture on expected bench-scale metallurgy. This does not apply to the pyrrhotite rejection flowsheet, used in the later years of the mine life. Here, a paucity of data does not allow for the same approach to predicting the performance of the pyrrhotite rejection circuit so an alternative mineralogical-based approach, described later in this sub-section, has been adopted.

Figure 17-6: Correlation between Nickel Recovery and Mg/S



Note: Figure prepared by TMM, 2014

17.2.2.3 Locked Cycle Data

Relationships developed on the locked cycle test data revealed the following.

- Copper cleaner distribution to the copper, nickel or combined concentrates is entirely unrelated to copper, nickel or sulfur head grade
- Copper recovery to the copper concentrate proved to be relatively consistent from sample to sample
- Copper recovery to the nickel concentrate showed a greater variability but was not linked to any of the head grades or lithology.

The following can be concluded for metallurgical forecasting purposes:

- Copper cleaner recovery to copper concentrate is a constant at 86.1%, irrespective of material type or head grade
- There is no relationship between any metal head grade and copper recovery to the nickel concentrate. With no clear trend in copper recovery to the nickel concentrate, this has been assumed to be fixed at 8.5%.

Relationships were developed and describe trends in cleaner circuit nickel distribution using the Base Case flowsheet. No strong relationships exist, except nickel recovery to the nickel cleaner concentrate appears to be mostly constant.

The following conclusion has therefore been drawn:

- Nickel cleaner recovery to the nickel concentrate is constant at 78.4%.

Based upon relationships developed, the copper concentrate grade proved to be mostly consistent from sample to sample, with only two samples yielding grades substantially different from 25% Cu. This is not a surprising result, as although copper is present as both cubanite and chalcopyrite, the ratio of abundance between the two, and hence the mineralogical-limiting concentrate grade is relatively consistent. The only regression of any quality is the link between nickel head grade and nickel concentrate grade. This is difficult to explain from a mineralogical context and is most likely a reflection of the greater ability of the laboratory cell to reject gangue when there is more pentlandite to crowd the gangue out of the froth. The regression coefficient is quite strong and was adopted for metallurgical forecasting.

Conclusions drawn for metallurgical forecasting from these data include:

- Copper concentrate grade is fixed at 25.4%
- Nickel concentrate grade linked to nickel head grade by:

Nickel concentrate grade = 20.705 x Ni head grade + 4.2545

The average cleaner metallurgy by geological unit and source location of the material tested, is shown in Table 17-5. Recoveries are individual cleaner circuit stage recoveries to final concentrates, based on the combined copper/nickel rougher concentrate.

The results indicate a drop in nickel recovery and the nickel grade in the copper concentrate. As no head grade/recovery relationship adequately describes this, the assumption has been made that S2 lithology tends to yield copper concentrates with less nickel than S3. The averages shown in Table 17-5 of 0.67% and 0.79% for S2 and S3 respectively, have been assumed to be constant for metallurgical forecasting purposes.

17.2.2.4 Pyrrhotite Rejection

The pyrrhotite rejection flowsheet only affects the metallurgy of the nickel circuit and therefore the copper circuit metallurgical forecast is as described previously. Side-by-side tests using the base case and pyrrhotite rejection flowsheets were run on three samples, of which one can be rejected as the pyrrhotite rejection flowsheet was far from optimal. The average drop in nickel recovery was 4.5% versus the base case flowsheet, and this has been used to predict the nickel recovery to the nickel concentrate.

Table 17-5: Mean Locked Cycle Test Results by Geological Unit and Location

	Copper Concentrate				Nickel Concentrate					
	Cu Rec (%)	Cu Grade (%)	Ni Rec (%)	Ni Grade (%)	Cu Rec (%)	Cu Grade (%)	Ni Rec (%)	Ni Grade (%)	Ni:Cu Grade (ratio)	Number of Samples
	Shallow									
S2	84.0	22.3	10.0	0.66	13.0	6.7	74.0	9.2	1.4	1
S3	91.0	25.0	12.0	0.83	7.0	3.5	81.0	9.9	2.9	3
	Deep									
S2	85.0	25.7	8.0	0.63	12.0	3.8	75.0	7.9	2.1	2
S3	88.0	25.3	9.0	0.75	10.0	3.9	78.0	9.3	2.5	2
	Deep East									
S2	89.0	26.9	13.0	0.73	8.0	4.5	66.0	7.3	1.6	1
S3	89.0	25.5	11.0	0.80	9.0	3.7	78.0	8.5	2.3	3
	Deposit Wide									
S2	86.0	25.5	11.0	0.67	11.0	4.7	73.0	8.1	1.8	4
S3	89.0	25.3	9.0	0.79	9.0	3.7	80.0	9.1	2.5	8

The concentrate grade achieved is a function of the amount of pyrrhotite in the feed. The pyrrhotite rejection flowsheet is particularly effective in the rejection of the iron sulfides from high-pyrrhotite materials, and while the Po:Pn ratio has no effect on copper or nickel recovery using the flowsheet, it has a marked effect on pyrrhotite recovery as shown in Figure 17-7.

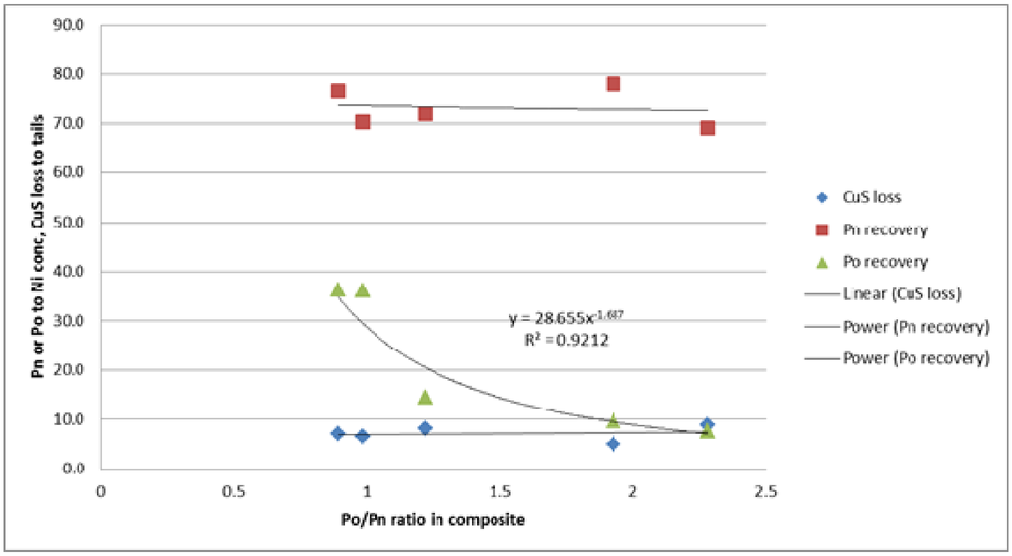
Accordingly, the performance of the process at rejecting pyrrhotite has been forecast based upon the mineralogical content of the feed and recoveries of copper and nickel as described above and the algorithm dictating recovery of pyrrhotite as shown in the graph. This allows for calculation of the mix of sulfides in the nickel concentrate. On average, the pyrrhotite rejection nickel concentrates contained 38% non-sulfides. The remaining 62% as sulfides were therefore split into copper, nickel and iron sulfides which were then used to convert these to metal grades.

17.2.2.5 Pilot Plant Scale Up

Six locked-cycle tests were completed at Blue Coast and ALS, and four investigative pilot plant runs were completed at ALS; the optimal performance conditions on the PP-3 sample have been used to compare bench-scale with continuous pilot-scale performance.

The “comparative” results allow for an indication of the effect of scale-up from 2 kg batch charges to 200kg/hr continuous operation. The flowsheet tested in locked cycle mode at BCR was substantially superior to that which was tested in the laboratory or piloted at ALS.

Figure 17-7: Mineral Recoveries as a Function of Po/Pn Ratio in the Feed



Note: Figure prepared by TMM, 2014

There were two other key equipment differences between the treatment schemes:

- The pilot plant copper circuit at ALS employed a column for final stage copper cleaning to enhance nickel rejection
- The nickel cleaners employed froth crowding, which would be expected to enhance gangue rejection.

The key results from the locked cycle and pilot plant runs are shown in Table 17-6. For each metallurgical parameter, the individual test result is shown together with the average from locked cycle testing and the piloting. The mean difference between the two is also shown, together with the student's t-value describing the statistical significance of the difference.

Scaling up to pilot plant had no significant effect on copper metallurgy. Copper concentrates from cycle testing and piloting assayed 24.6 and 25.2% Cu respectively, while the copper recovery in the pilot plant to the copper concentrate was slightly, but insignificantly, lower. The nickel grade, however, was lower, as well as the nickel misplacement to the copper concentrate. Both these differences, when examined using T-test statistics, were highly significant. Accordingly a drop in nickel grade in the plant can be expected versus the bench scale projection.

T-test statistics also have been used to test the differences between the two datasets with respect to the nickel cleaner circuit. The copper grade and recovery to the nickel concentrate were almost the same in the locked cycle tests and in the pilot plant. Any differences are statistically insignificant so the assumption was made that there would be no scale-up effects on copper recovery to any product.

Table 17-6: Comparison of Locked Cycle and Pilot Plant Optimized Performance on PP-3 Composite

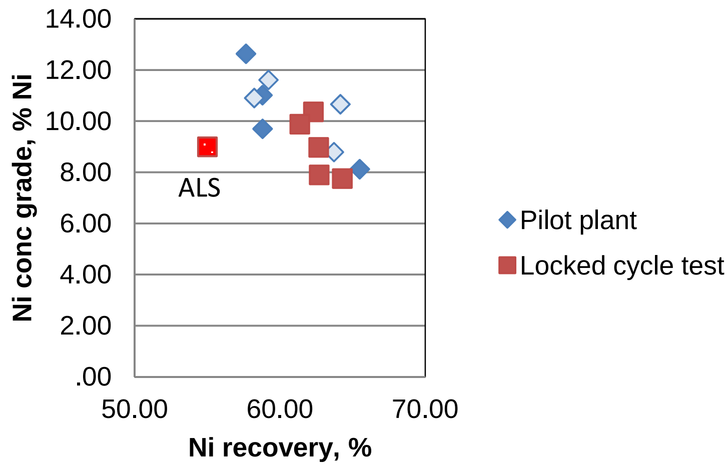
	Copper Cleaner Concentrate				Nickel Cleaner Concentrate				
	Cu Rec (%)	Cu Grade (%)	Ni Rec (%)	Ni Grade (%)	Cu Rec (%)	Cu Grade (%)	Ni Rec (%)	Ni Grade (%)	Ni:Cu (Ratio)
Bench Scale									
LCT 1	84.6	25.3	7.7	0.83	N.A	N.A	N.A	N.A.	N.A.
LCT 2	86.0	24.3	8.9	0.89	8.5	3.0	62.7	7.9	2.6
LCT 3	86.3	24.8	7.5	0.77	7.8	3.2	64.3	9.4	2.9
LCT 4	84.2	25.3	7.0	0.74	10.4	4.2	62.3	8.8	2.1
LCT 7	84.1	24.7	6.8	0.70	9.9	4.2	61.4	9.1	2.2
ALS-T81	87.0	23.1	7.0	0.77	8.0	3.3	55.0	9.0	2.7
Avg.	85.4	24.6	7.5	0.80	8.9	3.6	61.1	8.8	2.5
Pilot Plant									
P28	84.7	24.2	6.7	0.74	9.7	3.5	63.7	8.8	2.9
P29	84.5	24.0	6.0	0.63	8.6	4.4	58.2	10.9	2.5
P32	85.8	26.1	5.1	0.60	7.3	3.7	59.2	11.6	3.1
P34	83.4	26.4	4.9	0.62	8.9	3.7	64.2	10.7	2.9
Avg.	84.6	25.2	5.7	0.65	8.6	3.8	61.3	10.5	2.3
Diff.	-0.8	0.6	-1.8	-0.13	-0.3	0.2	-0.2	1.7	-0.2
T-Value	1.0	0.9	3.5	3.2	0.4	0.7	0.1	2.8	0.1

Nickel cleaner flotation in the pilot plant fell into two broad modes of operation, one yielding high-grade nickel concentrates at lower recoveries, and a second yielding nickel concentrates not dissimilar in grade to the locked-cycle test, but at higher nickel recoveries (Figure 17-8).

The four pilot plant runs used for the side-by-side analysis are illustrated by the open diamonds in Figure 17-8. The difference in performance between the five Blue Coast locked-cycle tests, all clustered at 62 to 64% nickel recovery, and the one ALS test at 55% nickel recovery, demonstrates the degree of enhancement achieved for the Blue Coast program. The ALS cycle test shown was chosen at it was the best of the three locked-cycle tests run on the ALS flowsheet.

The difference between ALS pilot plant performance and the single ALS point may be a truer reflection of the benefits of continuous operation for this material versus locked cycle testing, but as no pilot plant has been run on the Blue Coast flowsheet, the respective improvements through piloting the Blue Coast flowsheet could not be proven or quantified. This ability to manipulate grade was likely the consequence of its ability to crowd and fully drain the froth in the investigative pilot plant, something that cannot be replicated in the laboratory due to the low weights of concentrate being floated.

Figure 17-8: Nickel Grade versus Recovery, Pilot Plant versus Locked Cycle Test



Note: Figure prepared by TMM, 2014

It is assumed that the plant will operate to yield higher nickel grades (10.8 %Ni in the pilot plant versus 9.1% Ni on the bench. This difference withstands statistical scrutiny (T = 2.8) and can be assumed to be possible at continuous pilot or commercial scale (a relative increase in nickel grade of 19%). As there was no difference in mean recovery between the Blue Coast locked-cycle tests and the ALS pilot plant, it has been assumed that scale-up would have no effect on nickel recovery to the nickel concentrate. This discounts the potential for significant improvements in nickel recovery with the use of the Blue Coast flowsheet operated in continuous pilot (or commercial) mode.

Accordingly, the scale-up factors to continuous operation have been assumed to be:

- Copper concentrate
  - Copper grade and recovery, no change
  - Nickel grade down 17%, misplacement calculated based on lower grade
  - Mass pull, no change.
- Nickel concentrate
  - Copper recovery, no change
  - Nickel grade, up (relative) 19%
  - Nickel recovery, unchanged;
  - Mass pull, calculated from nickel balance
  - Copper grade, calculated from mass pull and copper distribution.

The algorithms and predictions presented are for the Maturi deposit. Limited bench testwork and no pilot scale testwork on the Maturi Southwest meant that the algorithms developed cannot be used for production figure predictions for this material even though they are feed-grade based. It appears that the mineralogical characteristics of the Maturi and Maturi Southwest deposits are similar but not identical. Based upon the limited bench-scale locked-cycle tests completed and geological observations, minor discounts have been applied to the nickel recovery to the nickel concentrate and nickel concentrate grade achieved over and above the algorithm estimates.

The S3 units at Maturi and Maturi Southwest host the bulk of the mineralization. The same is true at Maturi Southwest, but S3 at Maturi Southwest contains slightly more Mg, most likely in the form of olivine. Overall, the known geological differences are small and may be insignificant. Once this is proven using pilot-scale testing the existing algorithm for Maturi can be proven for Maturi Southwest or a new algorithm can be established for Maturi Southwest.

17.2.2.6 By-Product Metallurgical Predictions

Relationships exist between the head grade and recovery to combined concentrates, for gold, platinum and palladium. Although not strong, all three provide a better indication of gold, platinum and palladium recovery than simply using a fixed recovery, so the associated linear regressions as shown for each metal have been used (Figures 17-9 to 17-11).

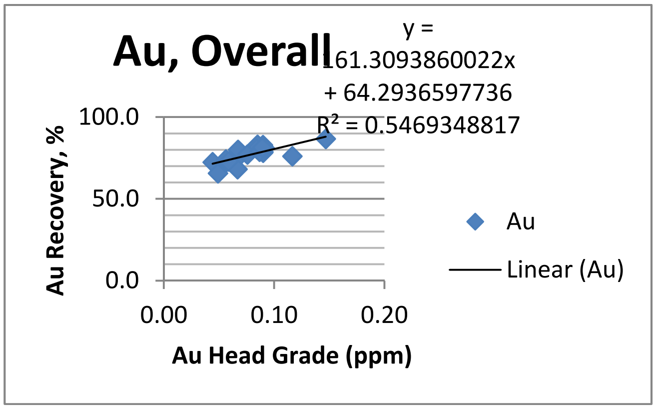
The distributions of recovered platinum and palladium to the individual concentrates are essentially unaffected by head grade, therefore they have been fixed at 38% and 52% to the copper concentrate respectively, the remainder to the nickel concentrate. Gold shows only a very weak trend in favor of higher distributions to the copper concentrate as a function of head grade, and this has also been fixed at 83% to the copper concentrate, and 17% to the nickel concentrate.

In each case, the concentrate grades of these metals have been calculated as a function of the annual head grades, their recoveries to each concentrate and the concentrate mass pull rates.

There was no measureable effect of pyrrhotite rejection flowsheet on the recovery of gold and palladium, while the recovery of platinum to the nickel concentrate dropped, on average, 15% using the pyrrhotite rejection flowsheet.

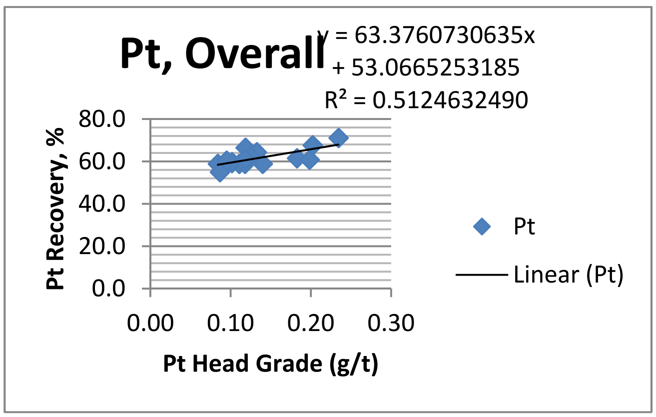
Locked cycle metallurgical balances on silver and cobalt are not available, so the metallurgy of these two metals has been predicted using pilot plant metal balances. Once again, their recoveries have been assumed to be constant and the concentrate grades have been calculated in a similar way to the other by-products.

Figure 17-9: Head Grade versus Recovery Relationships for Au



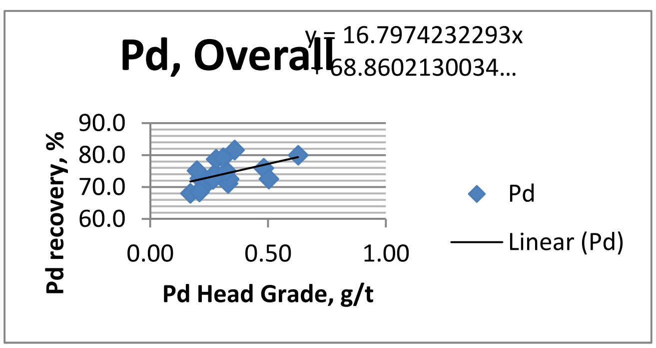
Note: Figure prepared by TMM, 2014

Figure 17-10: Head Grade versus Recovery Relationships for Pt



Note: Figure prepared by TMM, 2014

Figure 17-11: Head Grade vs Recovery Relationships for Pd



Note: Figure prepared by TMM, 2014

17.2.3 Production Plan

17.2.3.1 Throughput Predictions

At this stage insufficient physical characterization data exists on a yearly basis to make annual estimates of throughput predictions. Therefore a fixed daily throughput rate of 50,000 st/d has been assumed after ramp-up has been achieved. There will be variation on throughput with the use of a primary SAG milling in the comminution circuit and at a targeted grind size.

17.2.3.2 Ramp-Up

Ramp-ups have been assumed with respect to throughput, recoveries for both copper and nickel and also copper and nickel concentrate grades based upon industry experience for similar polymetallic operations producing separate concentrate products. A commissioning period of three months has been allowed prior to the full operational period of Quarter 1 which is scheduled for Year 1. These ramp-up factors or percentages of optimal production levels have been applied across the LOM production plan (Table 17-7).

After throughput ramp-up is achieved at the end of the first quarter after commissioning (six months from start up) maximum throughput rates are maintained through for 26 years through until Year 27 (Table 17-8).

Copper and nickel head grades fed to the plant are highest at the beginning of operations and reduce through until the end of operations in Year 30. The feed grades are matched by the copper and nickel concentrate production figures which are highest after the second year of operation and reduce gradually through until the end of LOM, thereby maximizing the Project value. The production figures are illustrated in Figures 17-12 and 17-13.

Excess material mined over the 18.25 Mst per year will be stockpiled on surface and fed back to the plant when excess mill capacity becomes available. Ore becomes available two years before process plant production begins, due to the mine development work and is stockpiled on the surface. A three-month period of commissioning commences with this surface stockpile material treated through the plant. A surface stockpile of around 2.3 Mst is generated in Years 1 to 4 and will be fed into the plant when shortfalls in mine production occur. All of this material is fed back to the plant prior to the final year of operation.

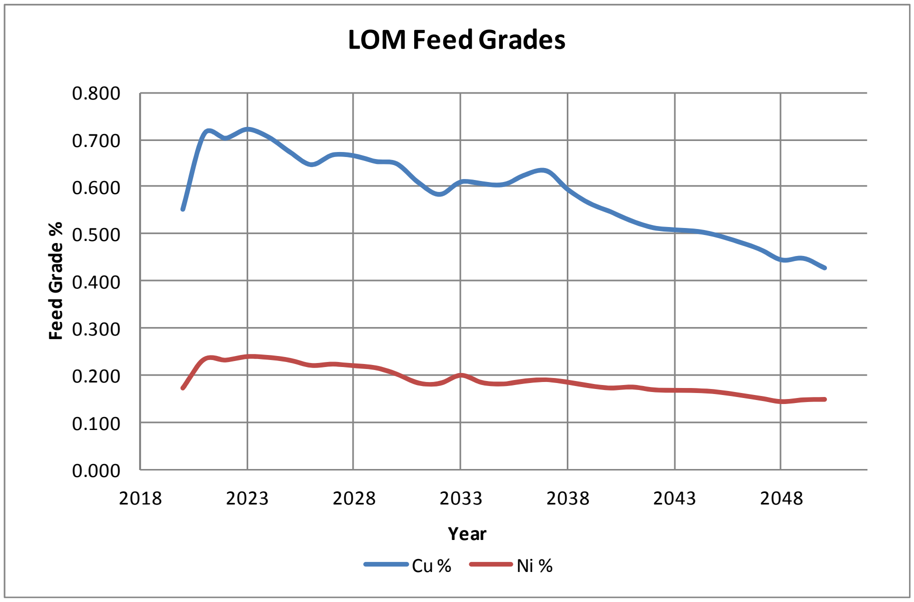
Table 17-7: Ramp-Up Factors

	Q4 Year -1	Q1	Q2	Q3	Q4	Year 1
Throughput ramp-up	40%	60%	100%	100%	100%	90%
Ramp-up Recovery Factor - Cu	50%	80%	95%	100%	100%	94%
Ramp-up Recovery Factors - Ni	50%	60%	75%	90%	100%	81%
Ramp-up Conc Factor - Cu	90%	95%	100%	100%	100%	99%
Ramp-up Conc Factors - Ni	90%	90%	95%	100%	100%	96%

Table 17-8: Production Figures

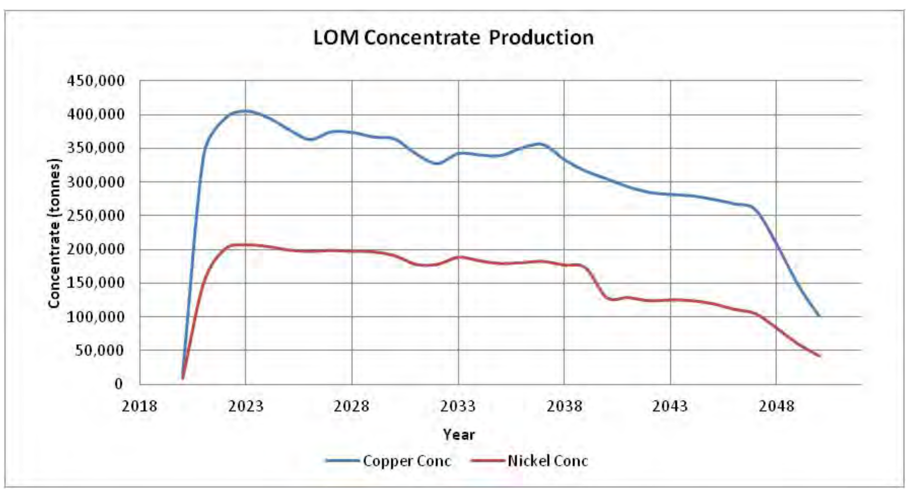
	Units	Calendar Year Total	2018 -3	2019 -2	2020 -1	2021 1	2022 2	2023 3	2024 4	2025 5	2026 6	2027 7	2028 8	2029 9	2030 10	2031 11	2032 12	2033 13	2034 14	2035 15	2036 16	2037 17	2038 18	2039 19	2040 20	2041 21	2042 22	2043 23	2044 24	2045 25	2046 26	2047 27	2048 28	2049 29	2050 30
Ore Milled																																			
Total Mill Feed	kdst	526,844			1,753	16,425	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	16,264	15,073	10,906	10,174
Copper Grade	%	0.592%			0.552	0.713	0.703	0.722	0.706	0.674	0.647	0.668	0.666	0.654	0.649	0.610	0.584	0.611	0.607	0.605	0.625	0.634	0.594	0.565	0.547	0.527	0.513	0.509	0.506	0.497	0.483	0.460	0.442	0.449	0.451
Nickel Grade	%	0.191%			0.173	0.233	0.231	0.239	0.237	0.231	0.220	0.223	0.219	0.215	0.202	0.183	0.182	0.199	0.184	0.181	0.187	0.190	0.185	0.177	0.173	0.175	0.169	0.168	0.167	0.164	0.158	0.150	0.144	0.148	0.153
Gold Grade	g/t	0.084			0.059	0.082	0.080	0.084	0.085	0.088	0.094	0.094	0.089	0.089	0.096	0.105	0.099	0.103	0.109	0.122	0.116	0.112	0.093	0.078	0.074	0.067	0.065	0.068	0.066	0.059	0.056	0.052	0.050	0.051	0.052
Palladium Grade	g/t	0.350			0.218	0.322	0.317	0.323	0.327	0.338	0.370	0.378	0.360	0.367	0.412	0.449	0.428	0.461	0.483	0.534	0.531	0.503	0.400	0.326	0.309	0.280	0.274	0.286	0.277	0.245	0.226	0.207	0.198	0.197	0.200
Platinum Grade	g/t	0.154			0.093	0.137	0.136	0.140	0.142	0.149	0.165	0.170	0.160	0.164	0.183	0.201	0.195	0.204	0.212	0.233	0.231	0.223	0.178	0.143	0.136	0.120	0.118	0.125	0.121	0.107	0.098	0.089	0.086	0.085	0.088
Silver Grade	g/t	2.145			1.957	2.519	2.495	2.552	2.507	2.426	2.336	2.426	2.453	2.456	2.424	2.340	2.131	2.174	2.175	2.237	2.365	2.375	2.208	2.093	2.012	1.909	1.851	1.814	1.779	1.712	1.671	1.604	1.555	1.583	1.599
Unit Costs																																			
Processing	US$/d mt milled	4.40			4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40	4.40
Cu Concentrate																																			
Recoveries																																			
Cu Recovery in Cu conc	%	85.46			42.92	80.69	86.06	86.08	86.08	86.08	86.07	86.06	86.07	86.07	86.05	86.07	86.05	86.06	86.06	86.07	86.07	86.05	86.02	85.84	85.54	85.29	85.15	84.96	84.88	84.79	85.11	85.21	85.13	85.25	85.31
Ni Recovery in Cu conc	%	6.64			3.57	6.40	6.69	6.65	6.61	6.48	6.52	6.63	6.73	6.73	7.09	7.36	7.07	6.77	7.29	7.41	7.41	7.35	7.06	6.75	6.63	6.38	6.41	6.16	6.05	5.95	5.96	5.91	5.90	5.82	5.69
Gold Recovery in Cu conc	%	65.26			34.01	61.06	64.14	64.57	64.76	65.19	66.00	65.98	65.29	65.22	66.28	67.47	66.61	67.13	67.93	69.66	68.89	68.40	65.87	63.94	64.04	63.35	63.47	64.11	63.71	62.30	61.11	60.46	60.07	60.18	60.31
Pd Recovery in Cu conc	%	39.06			20.95	36.66	38.58	38.63	38.67	38.76	39.04	39.11	38.95	39.02	39.41	39.73	39.55	39.84	40.02	40.48	40.45	40.20	39.30	38.71	38.74	38.60	38.63	38.82	38.69	38.25	37.85	37.64	37.56	37.53	37.55
Pt Recovery in Cu conc	%	24.18			12.45	22.27	23.43	23.54	23.58	23.74	24.13	24.25	24.03	24.13	24.58	25.01	24.86	25.08	25.28	25.77	25.74	25.53	24.45	23.67	23.73	23.48	23.52	23.81	23.66	23.12	22.60	22.33	22.25	22.22	22.29
Silver Recovery in Cu conc	%	64.37			35.88	61.31	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58	64.58
Cu Concentrate grades																																			
Dry Concentrate Total	dst	10,504,550			18,182	376,501	434,817	446,645	436,581	416,923	400,258	412,874	411,974	404,503	401,235	377,098	361,071	377,591	375,242	374,193	386,759	392,036	367,411	348,778	336,305	323,201	314,066	310,613	308,576	302,705	295,496	251,198	223,509	164,179	154,031
Moisture content	%	8.00			8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00
Cu grade	%	25.38			22.86	25.08	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40	25.40
Ni Grade	%	0.64			0.59	0.65	0.65	0.65	0.65	0.66	0.65	0.65	0.65	0.65	0.65	0.65	0.65	0.65	0.65	0.65	0.65	0.65	0.65	0.63	0.62	0.63	0.63	0.61	0.60	0.59	0.58	0.57	0.57	0.57	0.58
Gold Grade	g/t	2.74			1.92	2.18	2.17	2.21	2.31	2.52	2.84	2.75	2.58	2.61	2.91	3.44	3.33	3.34	3.59	4.14	3.77	3.58	3.06	2.61	2.58	2.38	2.41	2.56	2.49	2.23	2.12	2.05	2.01	2.04	2.07
Pd grade	g/t	6.85			4.41	5.15	5.14	5.09	5.29	5.73	6.58	6.54	6.21	6.47	7.38	8.64	8.56	8.88	9.40	10.55	10.13	9.41	7.80	6.61	6.51	6.11	6.15	6.53	6.34	5.65	5.29	5.04	5.01	4.90	4.95
Pt Grade	g/t	1.86			1.12	1.33	1.33	1.35	1.40	1.54	1.81	1.82	1.71	1.79	2.05	2.43	2.45	2.47	2.61	2.92	2.81	2.65	2.16	1.77	1.75	1.60	1.61	1.75	1.69	1.49	1.36	1.28	1.28	1.26	1.30
Ag Grade	g/t	69.25			67.70	67.37	67.61	67.35	67.67	68.56	68.80	69.26	70.17	71.55	71.19	73.12	69.55	67.85	68.30	70.45	72.07	71.39	70.83	70.73	70.50	69.61	69.46	68.81	67.95	66.65	66.66	67.05	67.71	67.92	68.22
Ni Concentrate																																			
Recoveries																																			
Cu Recovery in Ni conc	%	7.91			8.17	8.19	8.19	8.19	8.19	8.19	8.19	8.19	8.19	8.19	8.19	8.19	8.19	8.19	8.19	8.19	8.19	8.19	8.19	8.15	7.30	7.20	7.16	7.14	7.13	7.11	7.26	7.32	7.32	7.35	7.36
Ni Recovery in Ni conc	%	54.74			27.98	45.44	56.50	57.31	56.91	56.09	56.84	56.91	56.92	57.28	57.41	56.25	56.41	57.04	57.85	57.01	56.50	56.48	55.74	55.15	51.24	50.59	50.32	50.38	50.54	50.71	52.75	53.48	53.18	54.21	54.44
Gold Recovery in Ni conc	%	13.26			12.54	13.17	13.14	13.23	13.26	13.35	13.52	13.51	13.37	13.36	13.57	13.82	13.64	13.75	13.91	14.27	14.11	14.01	13.49	13.01	12.77	12.27	12.13	12.28	12.16	11.74	12.14	12.28	12.19	12.33	12.35
Pd Recovery in Ni conc	%	35.79			34.81	35.65	35.61	35.66	35.69	35.78	36.03	36.10	35.95	36.02	36.37	36.68	36.51	36.77	36.95	37.36	37.33	37.11	36.27	35.53	34.92	33.93	33.50	33.73	33.51	32.71	33.98	34.48	34.39	34.64	34.66
Pt Recovery in Ni conc	%	39.07			36.57	38.27	38.23	38.40	38.47	38.74	39.37	39.57	39.21	39.36	40.10	40.81	40.56	40.92	41.25	42.04	41.99	41.65	39.90	38.35	37.60	35.97	35.38	35.97	35.60	34.16	35.57	36.07	35.93	36.26	36.36
Silver Recovery in Ni conc	%	12.17			12.46	12.46	12.46	12.46	12.46	12.46	12.46	12.46	12.46	12.46	12.46	12.46	12.46	12.46	12.46	12.46	12.46	12.46	12.46	12.28	11.77	11.15	10.87	10.82	10.76	10.62	11.78	12.26	12.25	12.46	12.46
Ni Concentrate grades																																			
Dry Concentrate Total	dst	5,219,703			10,097	167,224	221,587	228,060	225,703	219,860	217,735	219,262	217,713	217,085	210,706	196,448	196,379	208,086	202,389	197,912	199,449	201,127	195,366	190,836	142,257	142,381	137,190	138,421	137,394	132,368	123,167	102,083	89,994	67,132	64,293
Moisture content	%	8.00			8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00
Cu grade	%	4.73			7.84	5.73	4.74	4.73	4.68	4.58	4.44	4.55	4.57	4.50	4.60	4.64	4.45	4.39	4.48	4.57	4.69	4.71	4.55	4.41	5.12	4.87	4.89	4.79	4.79	4.87	5.20	5.37	5.42	5.36	5.25
Ni Grade	%	10.55			8.38	10.40	10.76	10.94	10.90	10.75	10.49	10.55	10.47	10.37	10.04	9.58	9.55	9.98	9.60	9.53	9.67	9.75	9.61	9.36	11.35	11.33	11.31	11.14	11.23	11.50	12.36	12.74	12.78	13.03	13.19
Gold Grade	g/t	1.12			1.28	1.06	0.87	0.89	0.91	0.98	1.07	1.06	1.00	0.99	1.13	1.35	1.25	1.24	1.37	1.60	1.50	1.43	1.18	0.97	1.22	1.05	1.05	1.10	1.07	0.96	1.01	1.02	1.01	1.02	1.01
Pd grade	g/t	12.63			13.20	11.27	9.31	9.21	9.45	10.03	11.16	11.37	10.84	11.13	12.98	15.31	14.54	14.87	16.08	18.41	18.14	16.93	13.54	11.09	13.87	12.18	12.20	12.73	12.32	11.05	11.40	11.36	11.40	11.07	10.94
Pt Grade	g/t	6.06			5.93	5.14	4.27	4.30	4.40	4.78	5.43	5.59	5.27	5.44	6.37	7.63	7.34	7.32	7.90	9.02	8.89	8.42	6.63	5.24	6.57	5.55	5.56	5.93	5.72	5.03	5.15	5.09	5.15	5.03	5.07
Ag Grade	g/t	26.35			42.34	30.83	25.60	25.45	25.26	25.09	24.40	25.17	25.62	25.73	26.16	27.09	24.68	23.76	24.44	25.70	26.97	26.85	25.71	24.58	30.39	27.29	26.77	25.87	25.44	25.07	29.16	31.32	31.90	32.05	31.54

Figure 17-12: Mine Plan Feed Grade Distribution



Note: Figure prepared by AMEC, 2014

Figure 17-13: Mine Plan Distribution of Concentrate Tonnes



Note: Figure prepared by AMEC, 2014

Maturi Southwest material is introduced to the plant in the 19th year of operation following commissioning and is fed through until the 28th year of operation. This material represents 8.2% of the feed source to the plant over the LOM. A pyrrhotite rejection circuit is used during the treatment of Maturi Southwest material to maintain nickel concentrate grades despite the lower nickel feed grades and higher pyrrhotite to pentlandite ratios in the Maturi Southwest material.

17.3 Comments on Section 17

Separate copper and nickel concentrates will be produced throughout the Project LOM. Material will be stockpiled for three years during the mining development period and the plant commissioned in the last quarter of 2020 according to the production plan, at reduced recovery and throughput ramp-up rates. Optimal throughput mill rates are achieved after six months of operation and optimized copper and nickel recoveries and concentrate recoveries after 12 months.

The total LOM production of copper concentrate is estimated to be 10.50 Mst at a copper recovery of 85.5% to the copper concentrate at and grade of 25.4% copper. The total LOM production of nickel concentrate is 5.22 Mst at a nickel recovery of 54.4% to the nickel concentrate at a grade of 10.5% nickel.

Further bench and pilot plant testwork is required to optimize copper and nickel concentrate grades and recoveries, particularly for the Maturi Southwest material, which has had limited bench scale testwork and no pilot plant testwork conducted to date.

Additional pilot plant testwork is also required to understand whether separate water sources are required for the copper and nickel flotation circuits. This investigation includes the need to include a thickener or thickeners between the copper and nickel roughing flotation cells and also another set of thickeners between the first copper cleaner tail cells and the nickel cleaning cells.

18.0 PROJECT INFRASTRUCTURE

18.1 Introduction

Proposed primary project infrastructure on a site-by-site basis is summarized in Table 18-1. Infrastructure such as roadway ditches and culverts, stormwater runoff collection ponds, traffic control signals, rail crossings, fencing and other minor infrastructure will also be required. Figure 18-1 to Figure 18-5 provide details of the planned infrastructure layout for the major Project components.

18.2 Transport and Logistics

18.2.1 Roads

Project development would include three surface areas, each having separate ingress locations. Access would be via existing local and regional state and trunk highways and include purpose-built on-site roads required for mine and surface facility access and operations. New roads would be required at the concentrator, TSF, and mine sites. Existing roads within the Project boundaries may require new water crossings, such as box culverts and/or bridges, depending on the local ordinances or state regulations.

The main access for the concentrator and mine surface facilities would be via Birch Lake Road. A new 1-mile main entrance road would be constructed from Birch Lake Road to the concentrator site.

Access to the surface mine site area and paste plants would be via Minnesota State Highway 1. A new gravel-surfaced road extending approximately 2.5 miles via MN Hwy 1 would connect the paste plants and ventilation shaft locations.

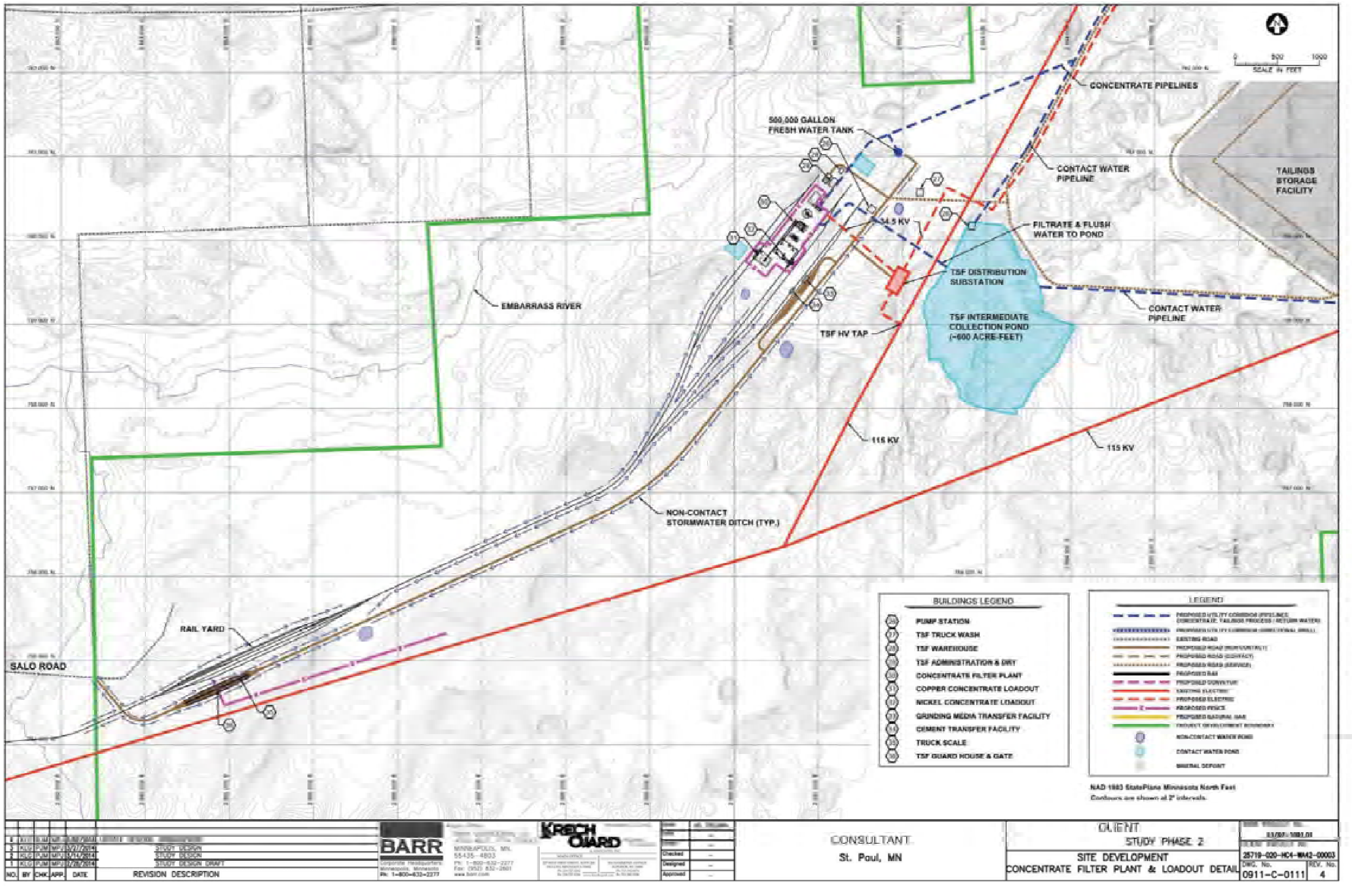
Main access to the TSF would be via County Road 615.

Each road was designed as a primary, secondary, or tertiary road, depending on projected traffic volume and functional use. The conceptual design included approximately 3.0 miles of primary roads, 3.1 miles of paved secondary roads, 6.5 miles of unpaved secondary roads, and 23.5 miles of tertiary roads. All main access roads would be bituminous paved. Secondary internal roads would be gravel surfaced. Tertiary roads would connect Project infrastructure and would be paved or gravel surfaced as appropriate.

Table 18-1: Proposed Infrastructure

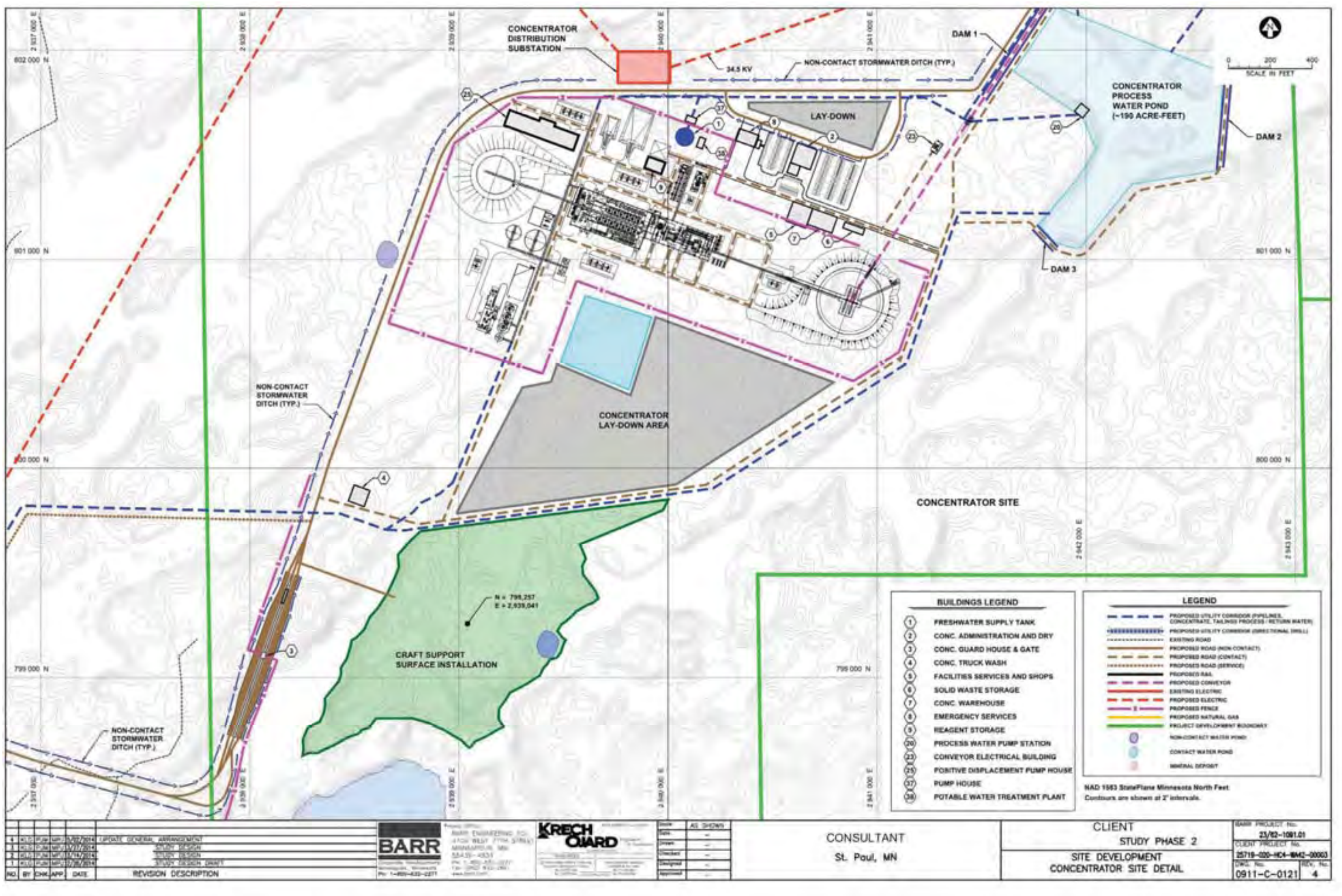
Mine Site	Concentrator Site	TSF Site
Portals	Concentrator	Tailings Storage Facility
Air Intake and Exhaust Shafts	Process Water Pond	TSF Admin and Dry
Paste Plants	Power Distribution Substation	Concentrate Filter Plant
Backfill Distribution System	Concentrator Admin and Dry	Copper Concentrate Load-out
LNG Storage Facility	Facility Services and Shops	Nickel Concentrate Load-out
Heater Building	Surface Rock Storage	Grinding Media Transfer Facility
Backup Power System	Solid Waste Storage	Cement Transfer Facility
Truck Wash	Warehouse	Backup Power System
Fuel Storage	Reagent Storage	Truck Scale
Guard House and Gate	Backup Power System	Truck Wash
Pipelines	Emergency Services	Warehouse
Roadways and Bridge	Potable Water Treatment	Process Water Pond
Power Lines	Pump Houses/Stations	Tailings Pump Station
	Vehicle Maintenance and Fueling	Return Water Pump Station
	Labor Transportation Station	Power Distribution Substation
	Concrete Batch Plant	Fresh Water Supply Tank
	Truck Wash	Guard House and Gate
	Facility Services and Shops	Pipelines
	Fresh Water Supply Tank	Roadways
	Guard House and Gate	Rail Lines
	Pipelines	Power Lines
	Roadways	
	Power Lines	

Figure 18-1: Concentrate Filter Plant and Load-out



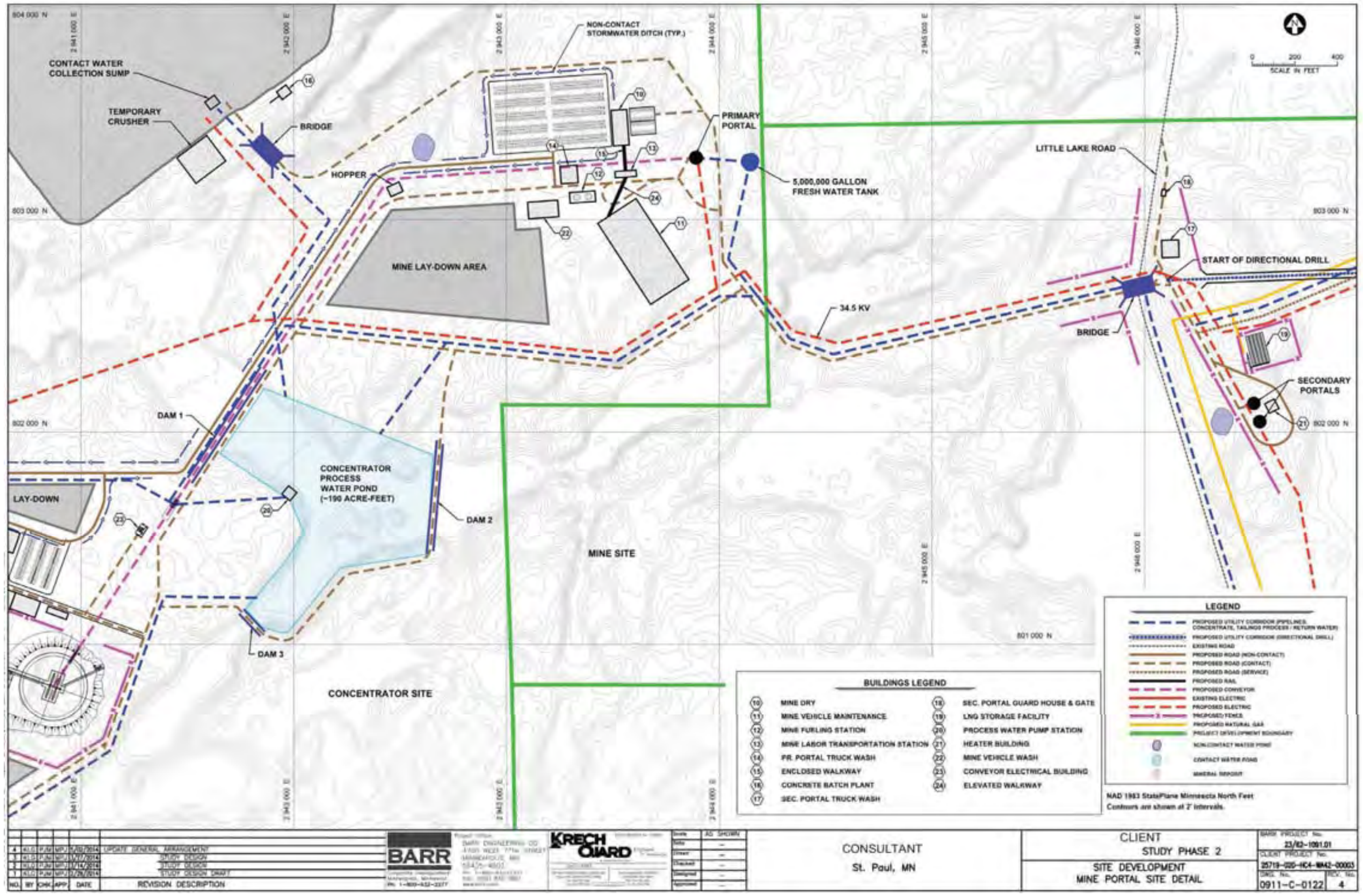
Note: Figure prepared by Barr, 2014.

Figure 18-2: Concentrator Site



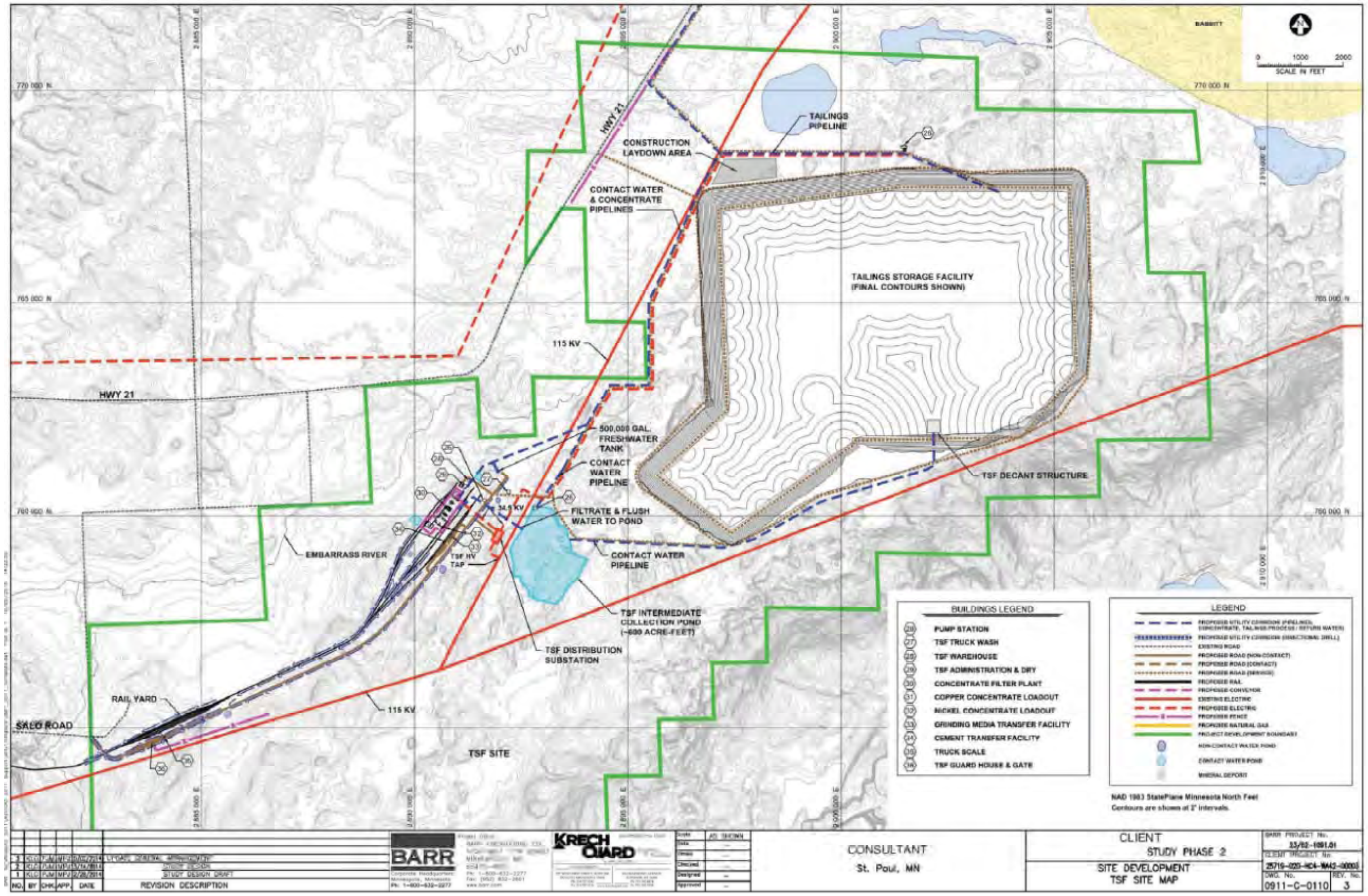
Note: Figure prepared by Barr, 2014.

Figure 18-3: Mine Portal Site



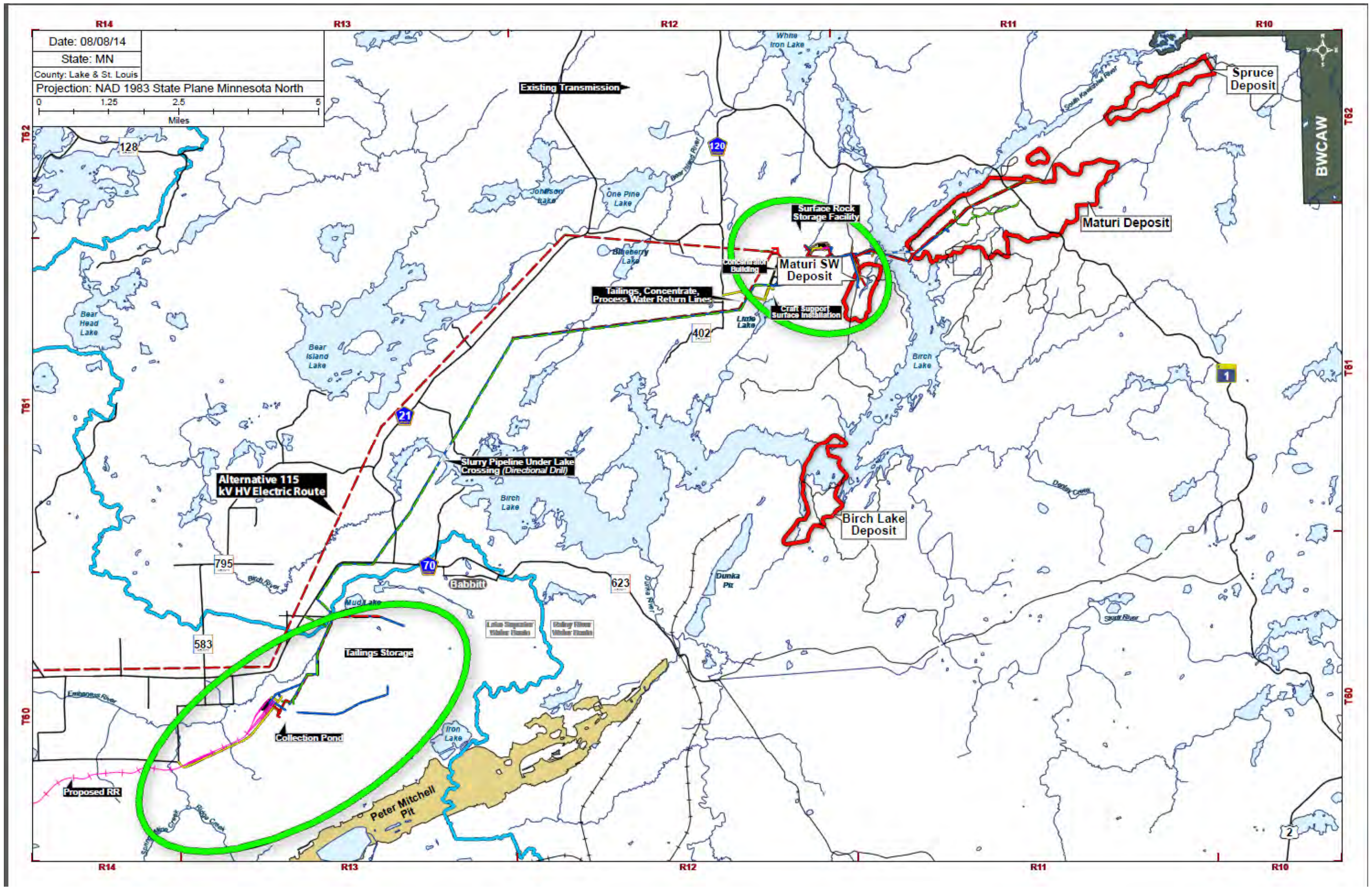
Note: Figure prepared by Barr, 2014.

Figure 18-4: Tailings Storage Facility



Note: Figure prepared by Barr, 2014.

Figure 18-5: Utility Corridors



Note: Figure courtesy Duluth, 2014. Map north is to top of plan. Green areas highlighted on the plan are proposed locations for the TSF (lower left) and the concentrator and associated infrastructure (upper center).

To simplify stormwater treatment, separate roads would be designated for "non-contact" and "contact" vehicles.

Non-contact roads would be used only by vehicles not exposed to ore, concentrates, process waste, or process water, or after such vehicles have been washed. Non-contact roads would generally be located from the point of Project entry and would extend to the personnel vehicle parking lots. The non-contact roads, due to expected high traffic volumes during shift change, would be paved to minimize maintenance and control dust. Noncontact roads would include longitudinal ditching for collection and conveyance of non-contact stormwater runoff to the non-contact stormwater management systems.

Contact roads would be used by vehicles exposed to ore, concentrates, process waste, or process water. Contact roads would be gravel surfaced and would include longitudinal ditching for collection and conveyance of contact stormwater runoff to the contact water management systems.

18.2.2 Rail

Rail would be used to supply the site with bulk commodities and for delivering concentrates to port facilities. CN would provide access from its existing rail network to the TSF site via a new rail line extended from CN Hinsdale Branch mile post 6.5 (present day end-of-track) to the concentrate filter plant at the TSF site.

The Project rail yard would store railroad cars when they were not being loaded or unloaded. Within the TSF site a single track would be extended directly to the cement transfer facility and another single track would extend directly to the grinding media transfer facility. Tracks would be extended from the rail yard to the filter plant. Copper concentrate would be loaded out from one side of the plant and nickel concentrate would be loaded out from the other.

18.2.3 Air Transport

The closest regional airline service is located in Duluth, Minnesota, which has daily direct flights to Minneapolis–Saint Paul International Airport. Ely airport has no regularly-scheduled commercial flights, but is available for civil aviation and charter flights. No Project-specific airstrip is planned.

18.2.4 Port

Recommended ports that have service to support concentrate transport to the preferred smelter locations would include:

- Port of Montreal to Europe and South America
- Port Metro Vancouver to Asia
- Port of New Orleans to South America.

Each of these ports has services to support bulk transport of concentrate product.

18.2.5 Pipeline/Utility Corridors and Under Lake Crossings

Two pipeline corridors would be used to transport tailings, concentrate, and process water between the concentrator area, the mine area, and the TSF area. Pipeline corridors would require two crossings of Birch Lake which would be developed as horizontal, directionally-drilled tunnels below the lake. The horizontal, directionally-drilled tunnels would require approximately 1.8 miles of tunnel which would pass a minimum of 100 ft beneath the bed of Birch Lake.

All pipelines would be located within the selected utility corridors and buried with a minimum depth of cover of 3 ft. Pipeline excavations would be backfilled with the native soil materials screened to remove cobbles and boulders.

18.3 Stockpiles

During the construction period, any mined ore will be stored at a surface stockpile. Most of this stockpiled ore will be used as mill feed during the fourth quarter of Year -1 to test and commission the concentrator. After the construction period is completed, most underground ore at Maturi will be hauled to the underground crushing station and transported via conveyor to the ROM stockpile. A small portion of ore in the early years in Maturi (0.6 Mst) will be stockpiled on surface until Year 27, or as needed during operations.

During Years 19 through 28, surface crushing of Maturi Southwest ore is planned on the surface. A ROM surface crushing surge pile with a maximum capacity of 24,000 st is planned to be located near the conveyor transfer point between the underground conveyor extension and the main overland conveyor.

Beyond the construction period, the waste rock remains underground as it is hauled to an empty stope and used as backfill. This reduces the required dimensions for a surface stockpile and greatly reduces the haulage distances for trucks in the mine. Additionally, this reduces the overall truck fleet and paste backfill requirements for the stope. Waste rock from initial mine development will be tested, and when appropriate, used for construction purposes such as concrete and shotcrete aggregate, backfill material for construction, and road base for underground haul roads.

18.4 Waste Rock Storage Facilities

Waste rock would be generated throughout the LOM, and will have varying physical and chemical compositions. During the first three years of mine development, waste rock and ore would be transported to the surface and stored at the surface rock storage facility (SRSF). Some of the surface storage waste rock, if determined to be non-acid generating (non-ARD) and meeting environmental compliance guidelines, would be used for surface site development actives such as structural fill, surface water erosion control, and road surfacing. Some of the waste rock would be transported back underground and used for mine stope backfill. Ore stored within the SRSF would be processed after the start of concentrator operation. Based upon the current mine plan, the SRSF is expected to be operated through Year 27.

The parameters in Table 18-2 were used in design of the SRSF. Geochemical characterization of waste rock and ore from the Maturi deposit is ongoing.

The SRSF location was selected near the proposed mine portal at the concentrator site in order to minimize transportation of ore and waste rock materials. The SRSF footprint is approximately 48 acres, having a maximum build-out height of 120 ft. The SRSF was designed to accommodate 5.8 Mst of ore and waste rock. Ore and waste rock would be segregated into distinct zones within the SRSF pad. The size of the pad has been developed to allow for deposition of ore and waste rock through truck haul. Additionally, the SRSF pad has been sized to allow for simultaneous deposition and removal of waste rock and ore on the pad. Soil and geomembrane liners are incorporated in the design.

It was assumed that all meteoric precipitation falling on the SRSF would be collected by an overliner drainage system. The contact water collected by the overliner pipes and the drainage layer would be conveyed toward the perimeter sumps. Contact water in the sumps would be pumped to the process water storage pond.

Emissions from the SRSF would predominately be either particulate via air airborne and/or particulate via surface water runoff. Dust control methods will be applied. Surface water runoff will be collected by the contact water collection system and routed to the process water storage pond.

18.5 Tailings Storage Facilities

18.5.1 Design Storage Requirements

The tailings management systems have been designed based upon a nominal tailings throughput of 50,000 st/d. The average split between tailings required for paste backfill and tailings that would be stored in the TSF was estimated for design purposes to be 55% to paste backfill and 45% to the TSF. Based on these assumptions, the TSF would accept 234 Mst of tailings. Additional tailings can be placed underground as paste backfill in access drifts and openings, as portions of the mine are exhausted and closed.

Table 18-2: Physical Properties for Rock used for SRSF Design

Property	Unit	Value
Specific Gravity, SG	—	3.0
Dry Density, pdry	pcf	130
Porosity	—	30.6%
Maximum Particle Size	in,	36
Nominal Particle Size	in,	5.5
Minimum Particle Size	in,	<2
Hydraulic Conductivity	ft/h	>12
Drained Shear Strength Friction Angle	degree	45°
Angle of Repose	H:V	1.4H:1V

Note: pcf = pounds per cubic foot; H:V = horizontal:vertical.

Based on the number of paste backfill plants operating (up to three plants may be operating at any given time), the TSF would receive tailings at typical rates of nil (three paste backfill plants operating), 16,667 st/d (two paste backfill plants operating), 33,333 st/d (one paste backfill plant operating), or 50,000 st/d (no paste backfill plants operating). Actual daily throughputs would vary depending upon both mill throughput and the number of paste backfill plants operating and would be reduced by the mass of concentrate produced.

18.5.2 Testwork

18.5.2.1 Geochemical Testwork

Tailings samples were collected from the hybrid concentrate tests for chemical characterization and for archival purposes. To provide an initial assessment of the potential for acid rock drainage (ARD), static testing was conducted on two final tailings samples. The chemical composition of the tailings samples was also determined. Kinetic testing of the two tailings samples was initiated in July 2013.

The results of the geochemical characterization are summarized as follows:

- The acid–base accounting (ABA) results indicate that the final hybrid tailings are unlikely to generate ARD. Kinetic testing results also indicate a low to non-existent potential for ARD. The low sulfide content of the tailings and the presence of olivine, which provides neutralization potential, are the primary reasons for this classification
- Acidic conditions were not established in either of the humidity cell test (HCT) cells. Metal concentrations were low in kinetic testing leachates. Following the initial flush, copper and aluminum were the only metals that exceeded Project design criteria in cell leachates
- Nutrient concentrations were low in cell leachates at the conclusion of the kinetic testing program

- None of the 87 fibers identified in the tailings solids were classified as occupational or asbestos fibers. None of the 45 fibers identified in the tailings process water samples were classified as asbestos, while seven were classified as occupational fibers

18.5.2.2 Geotechnical Testwork

Geotechnical testing was carried out on tailings samples to define a range of index properties, and to evaluate the strength, permeability, and consolidation behavior of the tailings. Tailings classify as a well-graded sandy silt in accordance with the Unified Soil Classification System, and are non-cohesive. When deposited as a slurry, tailings settle under self weight to an average in-place density of approximately 90 lb/ft^3. Under the stress conditions expected to be present within the TSF over the LOM, tailings are anticipated to consolidate to an average in-place dry density of approximately 106 pcf.

18.5.2.3 Thickening and Rheological Testwork

Dewatering and rheological testing programs were conducted to evaluate tailings properties to support the design of the TSF, slurry tailings transport system (from the concentrator area to the mine and TSF areas), and paste backfill system.

Dewatering tests conducted by Golder, FLSmidth, Diemme and Delkor indicate that conventional thickeners can produce an underflow density of up to 75% solids w/w. A density of 73% solids w/w was selected for transport of tailings from the concentrator to the paste plants and TSF. Rheology testwork indicates that tailings thickened to 73% solids can be pumped as a thickened slurry using positive displacement pumps.

18.5.3 Facility Design

The TSF would be constructed with starter cells to maximize the use of available borrow from within the footprint. An external borrow area would be developed to supply borrow to construct the complete TSF. Four starter cells would be constructed and operated sequentially over the initial eight years of mine operations until the ultimate perimeter of the TSF is reached. Once the ultimate perimeter is reached, the west, north, and east perimeter dams would be raised via centerline raise construction methods to the final crest elevation. As the tailings are deposited from west, north, and east dam segments, the reclaim pond would be relocated to its permanent location against the south dam, requiring the south dam to be raised via downstream raise construction methods. The TSF would be constructed as a zoned earthfill dam and would include a composite (soil and geomembrane) liner system for seepage containment.

A conceptualized two-dimensional seepage model was completed to provide a preliminary estimate of potential seepage losses through the TSF dam and through bedrock. This resulted in selection of a composite liner for the PFS design.

The stability of the TSF dam was evaluated using a typical dam cross-section for the centerline and downstream raise portions of the dam. The stability analyses incorporated the estimated phreatic surfaces generated in the seepage modeling. The analysis considered both static and seismic loading conditions and both upstream and downstream potential failure surfaces. All scenarios analyzed indicate adequate factors of safety can be maintained.

Consolidation modeling was performed for the TSF to estimate the average long-term in-place dry density of the deposited tailings. This analysis was used to estimate the magnitude of long-term consolidation settlements for post-closure conditions. The results of the consolidation modeling indicate that the final average dry density of the tailings is estimated to be approximately 106 pcf.

18.5.4 Tailings Deposition and Water Recovery

Tailings would be received at the tailings distribution pump station located on the north side of the TSF. Tailings would be piped to the crest of the dam where they would be uniformly spigotted from the perimeter of the TSF dam to form a tailings beach. The tailings beach would form against the dam and supernatant water would be collected in the reclaim pond. The reclaim pond would migrate over the life of the mine based upon tailings deposition, reaching its final permanent location against the south dam. A decantation system would be used to remove supernatant water from the surface of the reclaim pond.

Supernatant water would be discharged to the TSF intermediate collection pond and then returned to the process water storage pond at the concentrator.

18.6 Backfill

18.6.1 Testwork

Dewatering/settling tests were performed to determine the process design values including flocculant type, flocculant dosage, feed slurry solids concentration, settling rate, and filterability for thickening and dewatering. Rheological testing was carried out on the tailings samples to evaluate flow and handling properties.

Vacuum filtration tests conducted by Golder, FLSmidth and Delkor indicate that tailings can be filtered to 83% solids w/w using vacuum disc filters within the paste plants in order to produce a uniform consistency for mixing with cement, fly ash, and water for production of paste backfill.

Paste samples (tailings and Portland cement) were measured to have a slump ranging between 7 and 10 in. at solids contents ranging between 78.9 wt% solids and 77.7 wt% solids, respectively. UCS testing was done on paste samples with 7 and 10 in. slump consistencies to determine the achievable strength with differing binder addition quantities (cement only) that would support the underground mining operations. Itasca defined the target 28-day UCS strength for the paste backfill as 15–51 psi (100 to 350 kPa). Paste samples with 1–3% Portland cement binder demonstrated 28-day strengths ranging from 12–35 psi (80 to 240 kPa). Both 10 in. and 7 in. slump samples were used in the UCS testwork.

No testing was performed to confirm paste strengths for binders containing both cement and fly ash. Further tailings test work needs to be conducted evaluate paste strengths with the addition of fly ash as binder.

18.6.2 Backfill Plant Design

Backfill plants are located remotely from the concentrator, on surface above the ore bodies. Tailings would be pumped to the paste backfill plants located at the Maturi and Maturi Southwest mine areas. The paste backfill plants would produce paste backfill material from dewatered tailings and binder (cement and fly ash). Paste would be used as the backfill method of choice for underground mining operation. The paste backfill facilities would include four above ground paste backfill plants. Three plants would provide backfill for the Maturi operation, and one paste plant would provide backfill for Maturi Southwest.

The construction of paste plants would be sequenced based on the mine development and backfill requirements. The paste backfill plants constructed for the Maturi orebody would each be designed for the nominal daily throughput of approximately 420,000 ft^3/d. The paste backfill facility at Maturi Southwest would be designed for a nominal daily throughput of 210,000 ft^3/d.

The paste backfill plants would receive tailings from the tailings transport pipeline at a solids content of 73 wt%. Vacuum disc filters would be utilized to dewater the tailings to approximately 83 wt% solids. The dewatered tailings would be mixed with binder (cement and fly ash) and slurry to create a paste of uniform consistency, with an average solids content of 79 wt% (7 in. slump). The amount of binder required within the paste backfill would vary depending upon the mine operational configurations. The paste backfill would be distributed to the mining units and stopes via the backfill distribution system.

The paste plants would include all equipment necessary to produce paste from tailings, water, and binder. Major equipment for each paste plant would include vacuum disc filters, tanks, agitators, pumps (centrifugal and positive displacement), slurry/water handling systems, dust collection systems, and compressed air systems.

The paste backfill plants would also include binder storage silos, receiving systems, and feed systems required for cement and fly ash, which would deliver the binder required for paste backfill.

The binder storage portion of the paste backfill plants would include all equipment necessary to receive, store, and distribute binder including dust collection systems, rotary valves, aeration pads, weigh belt feeders, and screw conveyors. Binder would be received at the facility by bulk transport trucks and stored in dry form in dedicated silos for cement and fly ash. Cement and fly ash would be fed into the mixer by a screw conveyor.

Paste would be fed from the mixer to the paste pumps for transport to the mine openings through the paste backfill distribution system which will consist of a network of surface and underground boreholes, and interlevel pipes.

The paste backfill plant would include an electrical room where the switchgear and programmable logic controller (PLC) would reside. An area on the top floor of each paste backfill plant has been designated as the control room.

Filtrate water would be collected in a tank where it would be pumped back to the process water storage area, located at the concentrator, via the process water return system.

18.6.3 Backfill Distribution System

The paste backfill distribution system includes cased boreholes from the surface paste backfill plants to distribution bays located within the mining units. Within each mining unit, there will be a network of inter-level distribution and mine level piping that will transport paste backfill to the stopes.

Mine level backfill piping will be 8 in. schedule 80 carbon steel pipe installed with other utilities during drift development. Piping from the mine level header to the stopes will be of lighter material, generally HDPE. These are both included in the mine operating cost.

The borehole from the surface will be angled between 50º to 70º as required to intersect the borehole terminus distribution bay near the top of the spiral access ramps. The bay will be the main distribution area for the paste backfill plant supplying the specific borehole. In the distribution bays, removable spool pieces and swing elbows will be used to route the paste pipelines to other levels. The distribution bay will contain monitoring instrumentation, safety devices, flushing connections, and an emergency sump to hold discharged fluids.

18.7 Water Management

18.7.1 TSF Surface Water Management

There would be two types of water that must be handled around the TSF: non-contact and contact water.

Non-contact water is the surface water that does not come into contact with mining/extraction operations or materials and may be released to the surrounding environment in accordance with permit requirements. Around the TSF, the non-contact water comes from the upgradient, undisturbed watersheds and would be diverted around the TSF.

Contact water is assumed to be unsuitable for direct release into the environment and includes seepage through the TSF dams, runoff in contact with the TSF, and carriage water discharged into the TSF. Contact water collected around the toe of the dam area would remain within the closed circuit water system of the TSF and would be pumped back to the TSF reclaim pond. From there, it would be pumped to the TSF intermediate collection pond to be reused in the process as reclaim water.

18.7.1.1 Non-Contact Water Diversion

There are four planned non-contact water diversions. The non-contact water diversion channels have been located to divert as much of the natural runoff from undisturbed areas around the TSF as practical. The diverted water would be discharged directly to receiving streams using surface drainage ditches. Minimum channel depths were determined to convey the 10 year/24 hour flow with 6 in. of freeboard.

18.7.1.2 Seepage Collection System and Sumps

The TSF seepage collection system will consist of a filter zone and drainage zone downstream of the low-permeability soil core of the TSF dam. The filter and drainage zones would also extend beneath the TSF dam to the ultimate toe of the facility. The filter zone would be 10 ft wide and would provide compatibility between the low-permeability soil core and the drainage zone. The drainage zone would also be 10 ft wide and is designed to collect and convey seepage through the TSF dam to the collection drain located at the toe. The toe collection drain would convey seepage and precipitation infiltration from the downstream face of the dam to one of five sumps located around the perimeter of the TSF.

Seepage collection sumps would provide temporary storage of contact water. There would be five seepage collection sumps located around the TSF. The sumps would be generally located at low points along the perimeter of the TSF or at a location to allow for positive drainage from the toe drain. The sump capacity is designed to contain 24 hours of seepage through the TSF dam plus the expected surface runoff for the 10 year/24 hour design event.

18.7.1.3 TSF Decant and Water Reclaim System

A decant system would be used to remove water from the TSF reclaim pond for return to the concentrator via the TSF intermediate collection pond. The TSF intermediate collection pond would be located approximately two miles west of the TSF decant structure. The water in the TSF intermediate collection pond would be returned to process water storage pond for reuse in the concentrator.

Four starter cells would be constructed and operated sequentially starting with Cell 1 in the northeast and moving counter-clockwise until Cell 4 is constructed. As the tailings are deposited, the water reclaim pond would migrate against the center point where the four cells meet. At the ultimate configuration, the tailings would be deposited from the west, north, and east dam segments and the water reclaim pond would be relocated to its final location against the southern section of the dam (in Cell 4).

The water reclaim system would consist of decant stop log structures, reclaim pumps, and an aboveground pipeline that would transfer the water to the TSF intermediate collection pond.

As tailings deposition continues, the TSF reclaim pond surface would rise as solids within the tailings settle. Stop logs would be placed within the guides of the structure to control the depth and rate of flow of water from the TSF reclaim water pond into the decant structure. Upon completion of a starter cell, stop logs would be added to a level in order to enable management of the pond elevation while the other cells are active. The pond fluctuations in the non-active cells would be from precipitation, contact water from the perimeter sumps, and evaporation. The ultimate water reclaim structure would be located along the southern dam of Cell 4 and managed in a similar fashion, with the exception that the pumps would be installed on a moveable platform that could be raised along the dam as the TSF reclaim pond elevation increases.

18.7.2 Process and Waste Water

Contact stormwater and non-contact stormwater would be handled separately. Non-contact stormwater would be directed off-site by site grading, berms, and ditches. Contact stormwater from the mine and concentrator sites would be collected and conveyed to the concentrator process water pond. Wash water would be routed to the contact water management systems.

Non-contact stormwater would be routed to sedimentation ponds prior to discharge. Ditches and ponds that collect contact water would be lined to reduce seepage to groundwater. Water from the TSF intermediate collection pond would be used for concentrator process makeup water.

The concentrator process water pond, located at the concentrator site, would store approximately 190 acre-ft of process water. Process water would be pumped from the concentrator process water pond to the concentrator at a rate of approximately 25.73 M gal/d.

The TSF intermediate collection pond, located at the TSF site, would store approximately 650 acre-ft of process water. Return water would be pumped from the TSF intermediate collection pond to the concentrator process water pond at a variable rate dependent upon concentrator demand.

Water from the concentrator process water pond would be cycled back to the concentrator, used as mine service water, or lost to evaporation. Water from the concentrator is either delivered to the TSF with tailings, the pastefill plants with tailings, or is contained in the concentrate. No discharge of process liquids is planned.

18.8 Underground Infrastructure

The underground ore handling system for the Maturi deposit, from the discharge of run-of-mine (ROM) ore trucks into the crusher dump boxes to the surface stockpile feed overland conveyor, is designed to handle 50,000 st/d of crushed ore. Crushing stations and transfer conveyors were located in the footwall and do not intersect the orebody. The underground ore handling system consists of two gyratory crushers and the requisite conveying systems to transfer crushed ore to the stockpile feed overland conveyor at the surface.

A portion of the ore from Maturi Southwest will be hauled to the temporary surface crushing plant, while the remainder will be hauled to the underground crushing station.

18.8.1 Crushing and Conveying

The underground crushing and conveying system will consist primarily of crushing stations and associated transfer belt conveyors:

- Crushing and conveying system no. 1 (for crusher no. 1) is scheduled to be commissioned at the end of Year -1. Ore will be crushed at the beginning of Year 1 at an initial rate of 16,425,000 st/a.
- Crushing and conveying system no. 2 (for crusher no. 2) is scheduled to be commissioned at the end of Year 5. Ore will be crushed by crusher 2 at the beginning of Year 6 with an initial crushed ore production rate of 9,400,000 st/a.

The crushing and conveying systems will have a nominal capacity of 3,200 st/h and a design capacity of 4,320 st/h, providing a surge capacity of approximately 30%.

The covered coarse ore stockpile at the concentrator will have a 24-hour storage capacity. The transfer conveyors' design capacity of 4,320 st/h includes approximately a 30% surge capacity (catch-up capacity) to maintain 24-hour ore storage at the surface stockpile.

Crushing and conveying system nos. 1 and 2 are designed to operate simultaneously at variable production rates, but cannot exceed the transfer conveyor no. 1 design capacity of 4,320 st/h.

ROM ore will be delivered to each crushing station with 93.7 st (85 t) end-dump haul trucks. At each crushing station, trucks can dump the ore into either of two crusher dump boxes, oriented 180° apart. The crusher installations are planned with scalping grizzlies to bypass the crushers.

A 60 by 89 in. gyratory crusher was determined to meet the production requirements and design criteria. At each side of the crusher dump box, there will be a live capacity of one-and-a-half ROM ore trucks (140 st). The dump box will have a total live capacity of three ROM ore trucks (280 st). Crushed ore and ore bypassed through the grizzlies will discharge into a 226 st surge bin, equivalent to 2.24 ROM ore trucks.

Crushed ore will be reclaimed from each surge bin by an 84 in. wide apron feeder with a design capacity of 4,320 st/h and equipped with a 375-hp variable frequency drive (VFD). The apron feeder will discharge onto a sacrificial belt conveyor equipped with a 75-hp VFD.

Each sacrificial belt conveyor will have a design capacity of 4,320 st/h running at a nominal speed of 350 feet per minute (fpm). The sacrificial conveyors will be horizontal and relatively short. The sacrificial conveyors will discharge crushed ore onto 60 in. wide inclined transfer conveyors, each having a nominal capacity of 3,200 st/h and a design capacity of 4,320 st/h. Transfer conveyors will have 60 in. belts for the design capacity of 4,320 st/h. Conveyor nos. 1 and 3 will run at 800 fpm due to high belt tensions; transfer conveyor no. 2 will run at 650 fpm. Conveyor drifts will be 22 ft wide and 20 ft high. At one side of the conveyor, 2 to 6 ft of walkway space will be provided for maintenance access to the idlers. On the opposite side of the conveyor, there will be sufficient room for maintenance vehicles and forklifts.

For a portion of Maturi Southwest production and for the ore stockpiled material on surface, it has been assumed that two 6,000 st/d capacity jaw crushers and rehandling equipment and facilities would be required. Each surface crushing unit will consist of a feed grizzly, crusher feed bin, crusher, and short output conveyor. Both short crusher output conveyors will feed a 100 st crushed ore feed bin. The crushed ore feed bin will feed a 36 in., 115 ft conveyor that will terminate at a transfer point on the main overland conveyor.

18.8.2 Electrical

The underground electrical distribution will be a radial system with feeders originating at the main underground substations and routed throughout the mine using cable supported on messenger cables and cable tray where required. Two main underground substations are planned at Maturi, one near each crusher station. Generally, main substation no. 1 will provide power to the western half of Maturi, while main substation 2 will provide power to the eastern half, and the lower levels. One main underground substation is planned for Maturi Southwest.

Power cable will be routed between main levels in purpose drilled boreholes. The LOM electrical borehole requirement is 42,000 ft, including 36,500 ft for Maturi and 5,500 ft for Maturi Southwest. Boreholes will be uncased and 6 in. diameter. Borehole drilling will be an ongoing activity as the mine is developed.

Equipment utilization voltages will be obtained from step-down transformers. The electrical distribution will consist of switchgear, transformers, “smart” starters and feeder breakers for the motor and non-motor loads in common line-ups. Lighting and small power applications will be fed from transformers and power panels as required, and will be located in the electrical rooms. Cables will be armored, jacketed type, shielded/non-shielded (as required) copper conductors with ground wire. Transformers will be located in the electrical rooms and will be dry type. Electrical coordination will be completed to minimize power interruption on operation of power system protective relay operation.

The main power supply for the Maturi mine will be located adjacent to the conveyor portal. This substation will provide power to the drive house for conveyor no. 1 on the surface, and the Maturi underground mine. Main feeder cables (13.8 kV) will be routed down the conveyor decline from the portal to main underground substation no. 1 near crusher no. 1. One of the main feeder cables is fully redundant. Alternative power supply cables will also connect to main underground substation no. 1 through disconnect switches using Kirk-key interlocks.

In Phase Two, the distribution system will be extended to the eastern half of Maturi. A 13.8-kV main power feeder will be installed down the conveyor decline from substation no. 1 to substation no. 2, located near crusher no. 2. Transfer conveyor no. 2 will be powered from main underground substation no. 1, since it is located close to this substation. Power feed to main underground substation no. 2 will power an intermediate transfer substation that will power the transfer conveyor no. 3 drives.

Seventy-five mining area substation locations are planned for Maturi, and 10 for Maturi Southwest. Each will be installed in a 20 ft wide by 60 ft long excavated station. Only 42 are expected to be active at any one time (32 in Maturi and 10 in Maturi Southwest).

Mining area substations will power 1,000 V auxiliary ventilation fans, and mobile equipment such as stope drills, face drills and rock bolters. The 13.8 kV feeder cables from the nearest main underground substation will supply power to the mining area substations, which will step the voltage down to 1,000 V. The number of 13.8 kV circuits feeding the mining area substations will vary with the mining sequence, but will be around five per main substation. One to three mining area substations will be connected to each 13.8 kV feeder circuit.

18.8.3 Magazines

A single underground explosives magazine will be located near the bottom of the access declines. ANFO will be the primary explosive. No surface magazine is planned.

The explosives magazine will be supplied with fresh air directly from the secondary declines. Exhaust air from the magazine will be routed down a dedicated 6 ft diameter vent raise to the conveyor decline, and then up the conveyor decline and out of the mine.

18.8.4 Refuges

Mine Safety and Health Administration regulations require that refuge chambers be located so that mine personnel can reach them within 30 minutes from their work area. Refuge chamber locations are planned to be located no more than a one mile walking distance from work areas.

Refuge chambers will provide a safe atmosphere for up to 36 hours. These critical life-safety chambers are portable, and will be relocated as mining progresses. There will be a total of 17 refuge stations, with a maximum of 13 in use at any one time.

18.8.5 Fuel and Lubrication Stations

Diesel fuel is to be transferred underground daily in batches. The batches will supply four 10,000 gallon underground storage tanks. Diesel fuel will be delivered underground using a dry line fuel system from the surface facility to the underground storage facility. .

18.8.6 Compressed Air

The mine is planned without a centralized, mine-wide distribution system. It will rely on a mix of onboard mobile equipment compressors, portable compressors, and smaller package compressors in localized areas such as the gyratory crusher complexes, to meet the compressed-air needs for fixed plant maintenance and instrumentation.

18.8.7 First-Aid Stations

First-aid stations will be strategically located in active areas of the mine. There will be 52 first-aid station excavations: 45 in Maturi and seven in Maturi Southwest. It is estimated that 25 stations will be in use at any one time.

18.9 Surface Infrastructure

18.9.1 Geotechnical and Hydrogeological Investigations

Site hydrogeological and geotechnical features were reviewed for the planned mine, concentrator and TSF sites. No site-specific geotechnical or hydrogeological explorations have been completed for the Project. Preliminary geotechnical and hydrogeological characterization of subsurface conditions at the proposed TSF, concentrator and mine sites were developed by Barr, based upon publicly-available data.

Similar geotechnical conditions were found to be present at each of the three sites. The ground surface is typically mantled by glacial till soils, which have a makeup that will allow use for site construction purposes. Exposed bedrock is present at the concentrator, TSF, and mine sites.

Groundwater is typically present in the glacial till at a depth range of approximately of 5 to 20 feet below the ground surface. However, in wetland areas, groundwater is present near or at the ground surface. Wetlands are irregular and isolated, mainly confined to topographic lows.

18.9.2 Roads

New roads would be required at the proposed mine, concentrator, TSF, and paste backfill plant sites (see discussion in Section 18.1).

18.9.3 Maintenance

Two separate maintenance and service buildings would be provided for operations.

The mine vehicle maintenance building would be located near the decline and would be used to provide above-ground, on-site, vehicle maintenance and repair capability (service bays, shop area, and welding bay) and associated workforce operations and lunchroom space. This shop will service underground vehicles based on the surface, including most maintenance service vehicles, light vehicles and some utility vehicles. No permanent underground shops are planned. Although some minor maintenance will be performed underground in unused muck bays by service vehicles, most scheduled and unscheduled maintenance will require mobile equipment to be brought to the surface: haulage trucks, powder trucks, and services vehicles will drive out of the mine using their own power. Others, such as development and production drills, loaders, bolters, and scalers, will be transported out on low-boy trailers.

A separate services building and shop would be located adjacent to the concentrator building to service concentrator operation vehicle fleets. It would consist of multiple vehicle maintenance bays, weld shops, parts storage offices, and a break room.

18.9.4 Surface Crushers

It is planned that 31.8 Mst of ore will be crushed on the surface over the mine life. This constitutes 6% of the 527 Mst of LOM process plant feed. It includes 1.4 Mst of pre-production ore used to commission the concentrator in Year -1, 0.6 Mst of ore mined in the pre-production phase at Maturi, and 30.1 Mst mined at Maturi Southwest during Years 19 through 28. Because of the proximity of the deposit to the surface and shorter haul distance, 70% of Maturi Southwest ore (30.1 Mst) is planned to be crushed on the surface. Approximately 30% of Maturi Southwest ore (12.9 Mst) is planned to be crushed underground at Maturi crusher no. 1. During Years 21 through 25, the surface crushing operation will crush 4.4 Mst/a (24% of the annual feed to the process plant). During Years 19 to 20 and 26 to 28, between 0.7 and 3.2 Mst/a will be crushed on the surface.

During Year -1 a temporary crusher will be used to crush the pre-production ore. During Years 21 through 25, two jaw crushers will crush 6,000 st/d each (12,000 st/d total). During Years 19 to 20 and 26 to 28, only one surface jaw crusher will be used. The crushers will be on surface near the conveyor transfer point between the underground conveyor extension and the main overland conveyor, which is approximately 4,000 ft northeast of the secondary decline portals and 1,400 ft east of the conveyor decline portal. Each surface crushing unit will consist of a feed grizzly, crusher feed bin, crusher, and short output conveyor. Both short crusher output conveyors will feed a 100 st crushed feed bin. The crushed feed bin will feed a 36 in., 115 ft conveyor that terminates at a transfer point on the main overland conveyor.

During the full production years (Years 21 to 25), the surface crushing operation will be a two-shift, seven-days-a-week operation. The crusher will work the same schedule as the mine crews, which is a four-crew, 12-hour shift rotation.

The surface crushing system is not designed to accommodate ROM ore trucks dumping directly into the crushers. ROM crusher feed will be dumped on the ground and rehandled with front-end loaders. Underground haul trucks will deliver ROM ore to the surface crusher site 19.5 hours per day (9.75 hours per shift) on average. The maximum size for the ROM surface crushing surge pile was assumed to be 24,000 st, due to the limited space available near the conveyor decline portal.

18.9.5 Buildings

Building and surface installations will include the following:

- Surface mine site installations:
  - Mine vehicle maintenance
  - Conveyor and service portal truck washes
  - Service portal guard house
  - Mine fueling station
  - Personnel shelters and travelways
  - Concrete batch plant
  - Main fan and air-heater installations.
- Paste backfill plants:
  - Binder storage and distribution
  - Electrical building.
- Concentrator surface installations:
  - Dry and administrative building
  - Concentrator guard house
  - Concentrator truck wash
  - Solid waste collection station
  - Concentrator warehouse
  - Emergency services
  - Reagent storage.
- Concentrator filtration and load-out:
  - Concentrator filtration facility
  - Concentrator barns (Cu and Ni)
  - Rail load-out facility.
- TSF surface installations:
  - TSF administrative dry building
  - TSF guard house
  - TSF truck wash
  - TSF warehouse
  - TSF tailings pump station
  - Cement transfer facility
  - Grinding media transfer facility.

- Common to all surface installations:
  - Fencing and lighting
  - Contact water ponds
  - Non-contact drainage.
- Water-handling facilities:
  - Ponds
  - Pumps
  - Pipelines.

Stores and warehouse would be provided at the concentrator site and the TSF site. The concentrator warehouse would provide general operations shipping and receiving for time critical spare parts as well and consumables. The reagent storage building would provide storage for reagents that would be used in the ore beneficiation process.

18.9.6 Accommodation

The PFS assumes that the Project workforce (both construction and operations) would reside locally and commute to the site. No Project accommodation camp is proposed.

18.10 Solid and Domestic Wastes

18.10.1 Solid Waste Other than Tailings

Solid waste handling and recycling would include management of hazardous, universal, and general waste as well as recyclable materials generated at the site, with the exception of concentrator tailings. Recyclables and solid, universal, and hazardous waste would be segregated by type immediately after generation. Recyclables and general solid waste would be placed in dumpsters located near generation areas, while hazardous and universal wastes would be transferred to the solid waste storage building. Hazardous and universal waste will be disposed of in a suitable facility.

18.10.2 Sewage

There will be no on-site sewerage treatment facility. Sewage treatment would include the following features:

- Four sewage holding systems at the mine portal area
- Four sewage holding systems at the concentrator site
- Two sewage holding systems at the TSF site
- Four sewage holding systems at the paste plants.

Sewage would be periodically pumped from these systems and transported for discharge at a permitted publically-owned treatment works nearby.

18.11 Power and Electrical

18.11.1 Design

The total projected load for the Project is expected to be near 121,000 kW. Projected loads and demands by area are summarized in Table 18-3.

High voltage (HV) power supply from the Minnesota electric transmission grid would convey extensions of existing HV transmission lines to the Project site, including installation of HV towers and substations as needed. The HV power supply would be developed by Great River Energy (GRE) and Lake Country Power (LCP), using a looped configuration for redundancy.

The discussion in this sub-section is provisional. The Minnesota Public Utilities Commission (PUC) is the state's agency responsible for the regulation of certain power suppliers such as GRE. The PUC would have ultimate approval authority for final routing and configuration of the Project's HV electric power transmission.

Transmission improvements would include two new 115 kV transmission lines and three rebuilt 115 kV transmission lines. The 115 kV transmission line from Hay Lake to the concentrator site would consist of a new, 26 mile long transmission line. It would provide a redundant transmission segment in order to create a looped configuration to provide high reliability. The 115 kV transmission line from Babbitt to the concentrator site would consist of a new, 10 mile long, transmission line. It would provide electrical energy to the concentrator and complete the looped configuration.

Electrical power for mining operations would need to be transformed from transmission high voltage to site distribution medium voltage (MV). HV power supply would also include all equipment necessary to transform power from the HV substation to MV levels for distribution to facilities that would require electrical power at the mine site, concentrator site, and the TSF site.

Battery limits would include new and rebuilt 115 kV HV transmission lines and HV substations to deliver power to the TSF site and concentrator site. Battery limits would also include the portions of the Project distribution substations that would step the voltage down to 34.5 kV. Step-down transformers would be provided by Lake Country Power. HV power supply would also include new 34.5 kV transmission lines to the mine site. HV power supply would provide power to the surface power substation and distribution installations.

Table 18-3: Power Distribution Requirements by Major Area

Area	Load (kW)	Demand Load (kW)
Concentrator	60,743	53,567
TSF	4,326	3,504
Mine site	92,661	64,238

In detail, the existing 115 kV transmission lines from Embarrass to Babbitt, from Embarrass to the Laskin Energy Center near Hoyt Lakes, and from Embarrass to Virginia, would be upgraded with installation of higher-capacity conductors and updated structures, as needed, to handle the existing loads, plus the amount of energy required by the new mining operations.

Substation improvements would also be necessary. A new HV substation would be constructed at Hay Lake to enable a 115 kV take-off from the existing 115 kV line between Embarrass and Tower for the transmission line to the concentrator site. The Hay Lake HV substation would consist of a 115 kV breaker station with 3-position ring bus and three 115 kV breakers. The existing Babbitt HV substation, which currently feeds Babbitt and Ely, would be upgraded with a 115 kV line outlet and a 115 kV breaker.

A new HV substation would be constructed at the concentrator site at the terminus of the new 115 kV transmission line from Hay Lake to the concentrator site. The concentrator HV substation would provide an access point for the concentrator distribution substation and provide a breaker station for segmenting the transmission system. The concentrator HV substation would consist of a 115 kV substation with switchable 23 million volt-ampere reactive (MVAR) cap bank, four fixed 10 MVAR cap banks, and six 115 kV breakers. A new 115 kV outlet line would feed a new concentrator distribution substation. At this substation, step-down transformers would step voltage down to 34.5 kV and feed distribution switchgear. The battery limit would be located between the step-down transformers and the switchgear.

For the TSF site, a new HV tap would be installed to provide an access point for the TSF distribution substation. The TSF HV tap would consist of a three-way switch and 115 kV breaker. Downstream of the TSF HV tap, a step-down transformer would step voltage down to 34.5 kV and feed distribution switchgear. The battery limit would be located between the step-down transformer and the switchgear.

The surface power substation and distribution installations would include above-ground secondary substations, cables, and switchgear, as well as the electrical distribution lines that route power from the distribution substations to Project facilities.

In detail, distribution substations at the concentrator and TSF sites would house step-down transformers that would be installed by LCP and would provide metal-clad switchgear lineups and breakers to feed the overhead distribution systems. The distribution voltage would be 34.5 kV, a standard voltage which would be able to deliver the needed power at reasonable amperage levels, using reasonable cable sizes.

The overhead distribution system would consist of overhead power lines with fused group-operated switches at take-off points. Standard overhead wood pole construction would be used for the overhead lines. Group-operated switches would feed pad-mounted distribution transformers at the concentrator, mine, and TSF site take-off points as applicable.

At the concentrator site, the distribution substation would be fed from the adjacent HV substation by GRE/LCP. The HV transmission substation would feed new 40 MVAR 115 kV:34.5 kV step-down transformers owned by LCP. A total of four 40 MVAR transformers would be included in the LCP concentrator distribution substation. Two of these transformers would feed the concentrator load, and two would feed the mine load. Power would be delivered from LCP to the concentrator site via two feeders, which would terminate into a metal-clad switch-gear lineup. The metal-clad switch-gear lineup would include a breaker to interface it with the concentrator site generator station as well as two feeder breakers to feed the concentrator overhead distribution system.

At the TSF site, the distribution substation would be fed from a 115 kV tap from the adjacent HV transmission line by GRE. The HV tap would in turn feed a 10 MVAR, 115 kV-to-34.5 kV step-down transformer owned by LCP. Power would be delivered from LCP to the TSF site via two feeders that would terminate into a metal-clad switch-gear lineup. The metal-clad switchgear lineup would include a breaker to interface it with the TSF site generator station, as well as two feeder breakers to feed the TSF overhead distribution system.

At the mine site, the distribution system would be partitioned to serve the mine portal area and the paste backfill plant area. Mine portal area distribution would originate from a metal-clad switch-gear lineup located at the concentrator site (similar to the lineup for the concentrator, but smaller). Power would be distributed to mine portal area buildings and to pad-mounted switch-gear located near the primary and secondary portals, which would provide power for the mine underground power distribution.

The paste plant area would be served by two 40 MVAR power transformers with dual output for looped distribution which would be co-located at the concentrator substation. From these transformers, a 34.5 kV underground double-circuit, owned by LCP, would traverse from the concentrator distribution substation to the mine surface distribution substation via a cable placed on the lake bottom. The mine surface distribution substation would not require any transformers, but would include switchgear to feed the loads from the paste backfill plants, the ventilation shafts, and other loads in the paste backfill plant area.

18.11.2 Trade-off Studies

A trade-off study was performed that reviewed looped versus radial transmission. The criteria used to evaluate the alternates were cost, ease of permitting, and reliability. Despite higher construction costs and the greater time and expense required for permitting, the looped configuration was recommended for the PFS because it would provide much higher reliability than the radial system.

18.11.3 Back-up

Back-up power would be required for critical mine and surface operations including mine ventilation, mine dewatering pumps, paste pumps, tailings pumps, and concentrator pumps and agitators. The back-up power facility would comprise all equipment to produce and distribute electricity during power outages including diesel generators, diesel fuel storage, and connecting infrastructure to the electrical distribution system.

Three back-up power locations are envisaged:

- One concentrator site generator station
- One TSF site generator station
- Three mine site generator stations.

Backup power systems would be designed to provide 1.9 MW to the concentrator site, 2.5 MW to the TSF, and 11.6 MW to the mine site. The load would include 6.8 MW for the mine portal area, and 4.8 MW for the paste plant area. Generator stations would be designed based on a modular strategy, with each generator station housing one or two diesel generators and room for expansion.

Back-up power would be distributed to critical mine and surface installations via the surface power substation and distribution installations and the mine underground power distribution installation.

18.12 Controls

The control system for the Project would be a fully integrated, centralized control system with a minimum of local controls. This system would be a standalone remote control system and would be designed to provide supervisory control and data acquisition, and continuous monitoring and control of the following processes:

- Mine site:
  - Personnel location

  - Traffic
  - Mining equipment status
  - Ventilation and heating
  - Dewatering/pumping
  - Underground crushing stations
  - Conveyor to transport crushed ore from underground to the coarse ore stockpile
  - Paste backfill plants and underground distribution systems.
- Concentrator site:
  - Coarse ore stockpile
  - Coarse ore reclaim system
  - Grinding
  - Flotation
  - Thickening
  - Concentrate transport
  - Concentrate filtration
  - Concentrate stockpiles and load-out systems
  - Tailings transport.
- TSF site:
  - Pump houses
  - Tailings distribution at the TSF
  - Reclaim system.
- Water systems:
  - Makeup water supply
  - Recirculation of process water to the concentrator
  - Process and contact water ponds.
- Power distribution system
- Monitoring of environmental variables
- Security, warehouse, and inventories.

The process control system (PCS) would include process controllers, input and output modules, cabinets, and HMI to monitor, control, and supervise the production process. The PCS would also manage process information to create reports and graphical displays of data according to the user's needs (i.e., operations, maintenance, engineering, or supervision). The control system would include all necessary installations and equipment to implement the required control and instrumentation systems.

18.13 Communications

A fiber-optic network connecting the main Project areas would provide communication services to multiple systems. The communications network would be based on a multiservice fiber-optic system, over which a data and voice dedicated network would also be implemented, mainly for administrative and telephony purposes.

The fiber-optic system would be connected to all major buildings and consist of a 48 fiber cable network.

Vehicle communications would be provided by short-wave radio in surface vehicles for communications to Project operations.

Communications in the underground mine will be by means of a leaky feeder radio system, hard-wired telephones, and fiber-optic system.

Off-site communication would be performed primarily by cell phone. This area has good cell coverage which would be enhanced with the recent addition of new cell phone towers.

18.14 Fuel

The mine fuel storage facility would be a centralized fueling facility to service both mine and concentrator fleets. The facility would be located near the main portal, and provide separate fueling for gasoline and diesel fleets. This facility will feed into the underground fuel supply distribution as discussed in Section 18.8.5.

18.15 Water Supply

The Project is estimated to need an average of about 4 M gal/d of water to be used at the concentrator. After the initial appropriation of this water, approximately 3.2 M gal/d of the total daily water needs will be obtained by recycling water from processing operations and mine dewatering in a “closed-loop system” between the concentrator and TSF. The remaining 0.8 M gal/d must be obtained from a makeup water source.

TMM has not yet selected a preferred water source for the Project. Five potential sources of makeup water supply have been reviewed: Dunka Pit, Birch Lake, groundwater aquifers, Dunka River, and mine pits that are part of the existing Peter Mitchell iron ore operation by Northshore Mining (an affiliate of Cliffs Natural Resources). TMM has determined that all of these potential makeup water sources are technically viable for its projected makeup water needs, but has not yet made its final selection as to the preferred source. Sufficient funds have been included in the capital cost estimate to construct a water supply system from the Dunka pit, likely the most expensive of the options.

Appropriation of makeup water from the selected water source would require TMM to secure DNR approval of a water appropriation permit and comply with other DNR requirements relating to selection of a water source for operations.

18.15.1 Makeup Water Supply

The makeup water supply for the Project is based upon a site-wide water balance description provided by TMM's Environmental Consultant. The water balance model assumed makeup water supplied from a water source. AMEC has reviewed the inputs to the water balance, but was not able to verify the actual model and therefore the results. Additional work will be required to develop an updated and more robust water balance as the Project advances.

Makeup water supply would include water intake equipment from a water source, conveying, holding, and distribution systems, and treatment systems for potable uses, including pumps, pipelines, valves, tanks, and controls. The makeup water supply main would be routed between the water source and the concentrator site. Power would be received from the surface power substation and distribution installations. Details of these installations will vary, depending on the water source(s) selected

The potable water supply for the TSF site and the guard house would be from wells. Design and construction of these wells would be typical of small wells in the vicinity of the Project site.

Makeup and potable water would be provided to the following facilities:

- Makeup water supply mains (2)
- TSF water well(s)
- Guard-house water wells
- Makeup water supply tanks at the concentrator site and TSF site
- Potable water treatment systems
- Fire water systems
- Makeup water supply to the concentrator process water pond
- Road dust control water supply
- Truck wash water systems.

Potable water for human consumption would be stored in tanks at the water treatment system location or in individual buildings.

Fire water systems would supply water for firefighting at the concentrator site, the mine portal area, and the TSF site.

18.15.2 Process Water Storage

Process water storage would provide facilities for process water storage and distribution. Water would be received from contact water collection, slurry carriage water, TSF reclaim, mine discharge, and, when needed, makeup water and delivered to the process water storage facility to be located at the concentrator site.

Process water storage would include the following features:

- Concentrator process water pond
- Concentrator process water pond dams
- Concentrator process water pond liner
- Intake structure and pump house for pumping concentrator makeup water
- Discharge structures for receiving inflows.

The concentrator process water pond, having a planned capacity of approximately 190 acre-ft, would store the water necessary for the concentrator process. It would be constructed in an existing low area adjacent to the concentrator. Three earth and rock fill dams would tie to natural topographic high ground. The concentrator process water pond would be lined to minimize seepage.

The TSF can also be used as a storage facility for process water, as can the TSF intermediate collection pond.

18.15.3 Water and Recycle Water Systems

Water from mine dewatering and contact water from the mine site and TSF would be collected to be used as mine water supply and/or piped to the concentrator process water pond. This installation would comprise all equipment required for water storage and distribution, such as lined ponds, pumps, controls, valves, tanks, and distribution piping.

Mine site contact water would not be discharged to the environment. The surface water management would include the following features:

- Mine site contact water collection system: The mine site contact water collection system would include five contact water ponds (one at each paste plant and one near the service portal)
- Mine dewatering system: Mine dewatering would pump groundwater out of the mine to the surface at the conveyor portal. The dewatering drainage pipe would convey dewatering water from the conveyor portal to the concentrator process water pond

- Concentrator site surface water management: Contact water at the concentrator site would be collected and conveyed to the lined concentrator process water pond for use as process water
- TSF site water management: Contact water at the TSF site would be collected and conveyed to the lined intermediate collection pond. The TSF intermediate collection pond would provide temporary process water holding to balance flows between the TSF and the concentrator process water pond
- SRSF contact water: Contact water from the SRSF will be collected and routed to the concentrator process water pond.

18.16 Comments on Section 18

18.16.1 Golder

Golder notes that no subsurface investigations have been performed at the TSF site, at paste backfill plant sites, or along the pipeline/utility corridors to evaluate subsurface conditions. Subsurface characterization has been generalized based upon publically available, regional geologic surface and subsurface maps. Differences in the subsurface conditions from those assumed in the facility design could result in changes to the facility designs, configurations and costs from those described in this report.

Golder notes that to produce a stable paste backfill material, the percentage of tailings particles smaller than 20 µm must typically be greater than 15%. The percent of material particles smaller than 20 µm is approximately 20%. This is acceptable but close to the typical minimum threshold for paste. Further tailings test work should to be conducted during future studies to evaluate the potential variation in tailings particle size distribution.

Golder notes that tailings test work described in this report was conducted on a single bulk tailings sample. While multiple laboratories provided corroboration of thickening and filtration test results, variations in tailings properties during production could result in future changes to the process design requirements for tailings and paste backfill dewatering, handling and transport. Further tailings test work should be conducted during future studies to evaluate the potential variation in tailings properties.

18.16.2 AMEC

AMEC notes that detailed design of surface infrastructure will depend on the results of geotechnical and hydrogeological investigations, additional design and engineering of the primary facilities, and requirements established during the permit process.

Although the current work is acceptable for a PFS-level study, detailed modeling of the site-wide water balance must be completed before the final Project water system design can be finalized.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

Wood Mackenzie assessed the proposed products from the Project on behalf of Duluth, and confirmed their suitability for sale into the custom nickel and copper concentrate markets.

19.1.1 Nickel

Global nickel demand was 1.8 Mt in 2013, up from 1.7 Mt in 2012 driven by China where demand increased from 760 kt to 913 kt. Despite this increase the market was in a significant surplus in 2013 which contributed to the decline in the average annual nickel price to US$6.81/lb from US$7.95/lb in 2012. However, Wood Mackenzie expects demand to exceed basecase supply from 2015 onwards, when a deficit of 95 kt is forecast, driven by production lost following the ban on Indonesian ore export and continued growth in Chinese demand.

Wood Mackenzie believes that a long-term price (i.e. from 2020 onwards) of US$11.46/lb (real 2014 dollars) is required to incentivize the development of nickel projects needed to retain a balanced market.

Third-party trade of nickel concentrate is a growing market which Wood Mackenzie estimates has increased from 31 kt Ni in 1996 to 186 kt Ni in 2013. New sources of concentrate will be necessary by 2019, despite basecase smelter requirements remaining flat, as mine depletion impacts current supply of integrated and custom feed. By 2020 Wood Mackenzie forecasts a 31 kt nickel in concentrate deficit, increasing to 250 kt by 2025.

The customers for the TMM nickel concentrate will likely be nickel smelters in North America, Europe, Russia, and China.

While there are no global benchmark treatment and refining charges set for nickel concentrates and each contract is negotiated on an individual basis, Wood Mackenzie has supplied indicative payment terms that a seller may expect to be offered from a third-party smelter both in China and the rest of the world.

19.1.2 Copper

In the medium to long term, Wood Mackenzie expects global refined copper consumption to grow at an average rate of 2.9% p.a. from 2013 to 2030 (20.7 Mt to 32.5 Mt) with China's share of copper demand forecast to increase from 44% to 53% during the period driven by increased electricity consumption and urbanization.

By 2023 demand for copper from mines will exceed basecase mine production by 6 Mt. Discounting brownfield capacity additions the market will require 3.8 Mt of new greenfield capacity over this period.

By 2020 a long-term copper price of US$3.50/lb (real 2014 dollars) is necessary in order to incentivize sufficient greenfield mine capacity to satisfy market requirements.

Measured on a copper content basis, the custom concentrate market has grown at an average rate of 4.5% p.a. since the mid-1990s contrasting with stagnant growth in the integrated sector over the same period. In 2013 it is estimated that 59% of all copper-in-concentrate production (8.4 Mt contained copper) was sold to third parties, compared with 41% consumed in integrated facilities. China's share of the custom concentrate market has risen steadily and is now substantially larger than Japan, having more than trebled from just 13% in 2000, to 42% in 2013.

As a result, China will be a potential market for TMM copper concentrate, along with other custom smelters in Europe and Asia.

19.1.3 Concentrate Quality

Wood Mackenzie does not expect any penalties will be payable on the nickel concentrate based on the specification of the concentrations of deleterious elements being lower than typical free limits observed in the market. The nickel content in the copper concentrate could attract a minor penalty which usually is applied at 0.5% Ni+Co and the typical nickel content of the proposed concentrate is 0.65%Ni. The penalty is not expected to be material but could restrict the marketability of this material. This will depend upon the average nickel content of other concentrates consumed by individual custom smelters and the ability to remove nickel from the electrolyte in the tank house. Table 19-1 shows the expected concentrate element ranges, based on metallurgical testwork results.

Indicative copper and nickel treatment charges, based on a typical concentrate sold in the market, are included in Table 19-2 and Table 19-3 respectively.

19.1.4 Conclusions

Wood Mackenzie confirms that the quality of TMM concentrate is suitable for the custom concentrate market and therefore would attract standard commercial terms, including benchmark copper treatment and refining charges (TC/RCs) refining charges for contained silver and gold, and payable metal percentages. The copper concentrate may attract a minor penalty for nickel based on the typical assay of 0.65% Ni.

Table 19-1: Estimated Concentrate Element Ranges

Element Name	Symbol	Unit	Nickel Concentrate Typical	Range	Copper Concentrate Typical	Range
Copper	Cu	%	5.5	4.0–7.0	25	23.5–26.0
Nickel	Ni	%	10.5	10.0–14.5	0.65	0.5–0.70
Gold	Au	g/t	1.1	1.0–1.5	2.7	2.0–5.0
Silver	Ag	g/t	35	25–50	60	40–80
Platinum	Pt	g/t	5.0	4.0–6.5	1.2	1.0–2.0
Palladium	Pd	g/t	11	9.5–13.5	4.5	4.0–7.0
Cobalt	Co	ppm	2000	1800–2500	160	150–250
Iron	Fe	%	27	26–28	33.5	32.0–35.0
Sulfur	S	%	30	28–32	31	30.0–32.0
Antimony	Sb	ppm	1.5	1.2–1.7	1.2	1.1–1.3
Arsenic	As	ppm	200	<1000	330	<2000
Bismuth	Bi	ppm	13	<150	20	<150
Chlorine	Cl	ppm	50	<100	50	<100
Fluorine	F	ppm	80	<300	60	<300
Lead	Pb	ppm	55	<5000	60	<5000
Zinc	Zn	ppm	400	<5000	600	<5000
Mercury	Hg	ppm	<1.0	<1.0	<1.0	<1.0
Selenium	Se	ppm	80	60–150	120	<200
Tellurium	Te	ppm	7.0	<70	10	<70
Cadmium	Cd	ppm	5.0	<40	10	<40
Manganese	Mn	ppm	300	<600	100	<600
Molybdenum	Mo	%	0.2	<0.3	0.2	<0.3
Magnesium Oxide	MgO	%	4.0		1.5	

Note: Data from Duluth, values are based on metallurgical testwork.

Table 19-2: Copper Concentrate Terms (based on typical concentrate assay)

Contract Variable	Units	Terms	Result
Cu TC/RC	US$/t/USc/lb	benchmark treatment/refining reported as treatment charge (real 2014 US$/t)/ refining charge (real 2014 c/lb)	65.7/6.6 (2012); 71.3/7.1 (2013); 92.0/9.2 (2014); 95.0/9.5 (2015); 105.0/10.5 (2016); 110.0/11.0 (2017); 102.0/10.2 (2018); 100/10 (2019 forward)
Au RC	US $/oz	Prevailing market	3.50–6.00
Ag RC	US$/oz	Prevailing market	0.30–0.50
Cu Payables	%	Deduct 1% and pay balance	96
Au Payables	%	Deduct 1 g/t and pay balance	63
Ag Payables	%	Deduct 30 g/t and pay balance	50

Note: Gold and silver payabilities shown above are typical of European and North American contracts. Asian smelters would pay 90% for Au and Ag above 1 g/t and 30 g/t respectively. Tonnage figures are in metric tonnes.

Table 19-3: Nickel Concentrate Terms (based on typical concentrate assay)

Contract Variable	Units	Example 1	Example 2	Example 3
TC	US$/t	370.0	275.0	N/A – No treatment charge
Ni RC	US$/lb	0.8 + 5% if price over $8/lb	1.6 + 8% if price above $5/lb	Incl. in Ni payability
Cu RC	US$/lb	0.6		Incl. in Cu payability
Ni Payables	%	92	95	69% (Ni price ≤$10,000/t) up to 76% (Ni price >$40,000/t)
Cu Payables	%	88 1	90	45% (Cu price ≤$3,000/t) up to 50% (Cu price >$6,000/t)
Pt	N/A	Less 1 g/t or pay 75%	Less 1 g/t or pay 70%	25% (Pay for PGM > 1 g/t)
Pd	N/A	Less 1 g/t or pay 75%	Less 1 g/t or pay 70%	25% (Pay for PGM > 1 g/t)
Au	N/A	Less 1 g/t or pay 75%	Less 1 g/t or pay 70%	25% (Pay for Au > 1 g/t)

Note: Tonnage figures are in metric tonnes

Benchmark copper TC/RCs are expected to rise from their 2014 level over the next three years as mine supply relative to smelter utilization trends towards surplus, reaching a peak of 31.8 c/lb in 2017 (real 2014 dollars). From 2020 onwards, long-term TC/RCs of 28.9 c/lb ($100/st and 10.0 c/lb) in real 2014 dollar terms are anticipated.

19.2 Commodity Price Projections

AMEC performed a review of metal pricing forecasts from a number of different sources, including:

- Wood Mackenzie review of metal marketing and metal pricing prepared for Duluth
- Analyst and bank forecasts
- Three-year trailing average metal prices
- Metal pricing used by peers in technical reports filed on Sedar
- Spot metal prices as at 1 July, 2014.

The final prices as agreed to by Duluth are summarized in Table 19-4.

19.3 Contracts

No contracts are currently in place for any production from the Project.

Most copper concentrate is traded on the basis of term contracts. These frequently run for terms of one to 10 years, although many long-term contracts are treated as evergreen arrangements which continue indefinitely with periodic renegotiation of key terms and conditions. Generally, a term contract is a frame agreement under which a specified tonnage of material is shipped from mine to smelter, with charges re-negotiated at regular intervals (e.g. annual or biannual).

Table 19-4: Metal Price Forecasts used In Financial Analysis

Element	Metal Price (US$)	Unit
Cu	3.50	per pound
Ni	9.50	per pound
Au	1,300	per ounce
Pt	1,680	per ounce
Pd	815	per ounce
Ag	21.50	per ounce

Spot contracts are normally one-off arrangements with a merchant for the sale of concentrates. The material is paid for in much the same way as a concentrate shipped under a term contract. Merchant business is a mixture of single consignments of concentrates and one-off contracts with smelters and long term contracts with both miners and smelters.

There are no global benchmark treatment and refining charges set for nickel concentrates and each contract is negotiated on an individual basis. As a result, it is generally difficult to find details relating to concentrate off-take agreements, these being confidential agreements between the buyer and seller.

19.4 Comments on Section 19

The Wood Mackenzie marketing opinion on concentrate terms does not include consideration of the PGMs that will be present in the copper concentrate. Additional Project-specific marketing studies should be conducted to determine what payabilities could be expected for PGMs in the copper concentrates, and if the concentrates would incur any additional treatment charges for these elements.

AMEC recognizes that some copper smelters may pay for PGMs in copper concentrates, and recommends that Duluth seeks indicative terms for the Project concentrate from such smelters. There is potential upside for the Project if payment for the PGMs from copper concentrate can be included in Project economics.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Existing Studies and Planned Studies

20.1.1 Climate

The Project area is dominated by a marked continental climate. Annual precipitation in the area is 28 inches, of which about 60 percent falls between April 1 and November 1. About 40% of the precipitation is received as snow. Stormwater runoff requires management under State of Minnesota permitting framework.

Regional weather and climate data are available from the State of Minnesota DNR website.

Project personnel established a meteorological monitoring site and began collected site specific baseline ambient air quality and meteorological data in 2012. TMM plans to collect meteorological data this site (Station No. 1). TMM may also establish and monitor meteorological conditions at two to three additional monitoring stations. TMM intends to perform a snow survey within the study area over one winter season. TMM has indicated during stakeholder presentations their intention to perform modeling and impact evaluations that would include greenhouse gas evaluation.

20.1.2 Hydrological Data

Water resources within the footprint of the proposed mine and associated infrastructure consist of lakes, reservoirs, larger rivers, medium-sized perennial streams, smaller perennial to intermittent tributaries, and variously sized wetlands. Water features are contained within two major drainages: Lake Superior and Hudson's Bay, which are separated by the Laurentian Continental Divide.

Most of the study area including the proposed mine site, the concentrator site, and a portion of the utility corridor are located north of the Laurentian Divide within the Hudson's Bay drainage (this area is also referred to as the Rainy River drainage basin in this Report). The proposed TSF site and portions of the utility corridor near the TSF site are south of the Laurentian Divide and lie within the Lake Superior drainage (also referred to in this Report as the Great Lakes drainage basin). The major receiving waters in the study area north and south of the Laurentian Divide are the South Kawishiwi River and the Embarrass River, respectively (MPCA, 2013). The South Kawishiwi River is tributary to the Rainy River which drains to Hudson's Bay. The Embarrass River is tributary to the St. Louis River which drains to Lake Superior.

Available hydrologic surface water and groundwater data include historical and current information obtained from the U.S Geological Survey (USGS) and existing Minnesota DNR gauging stations. Both agencies have monitored surface water and groundwater at several locations in and near the study area as a part of their routine monitoring activities. The DNR has re-established USGS stream gauging stations on several creeks specifically to support TMM's surface water monitoring activities. The groundwater below the Project site has not been well characterized to date, and information has been inferred from reports on regional conditions.

TMM has prepared a work plan for a hydrogeological study to characterize the hydrogeology of the Project area using site-specific data. The study proposed the phased installation of up to 400 wells or piezometers at up to 100 locations to document the baseline hydrogeological conditions across the Project area, including water quality, occurrence, and hydraulic controls on groundwater flow. The phased work plan would start with about a third of the wells being installed, at which time the work plan would be reassessed to determine if adequate technical information to model potential impacts would be able to be gathered in this phase. The study would also include predictive modeling to estimate the effect of mining activities on the groundwater occurrence, quality and flow.

TMM intends to document surface water flow, seasonal variations, and the hydrologic regime. TMM also intends to continue collection of water quality information at locations on lakes, streams, and creeks, with monitoring across summer and winter seasons. TMM intends to prepare a baseline surface water quality information report to present an analysis of data collected from 2012 through 2015.

20.1.3 Water Quality Data

Basic information on water quality in the Embarrass River is provided in MPCA (2013). A substantial amount of water quality information for the lower Embarrass River extending upstream to near the southwest portion of the planned TSF site has been collected to evaluate baseline conditions and to model projected water quality for PolyMet's NorthMet mine plant site (Barr, 2013a; 2013b).

Baseline surface water data have been reviewed to define surface water quality conditions at various locations within the potential area of influence. The dataset includes legacy data collected by the former Franconia Mineral Corporation and publically available data collected by Cliffs-Erie to fulfill National Pollution Discharge Elimination System/State Disposal System (NPDES/SDS) permit requirements of the former Dunka mine. Surface water quality data have also been collected by Project personnel beginning from 2008 and extending to 2013 at 18 sites. The total number of sites and specific sampling periods has varied slightly since 2008 as a few sites have been added and others deleted.

Field data include parameters necessary to characterize dynamic variables at the locations selected for sampling such as flow, temperature, dissolved oxygen, specific conductance, pH, and turbidity. Chemical constituents analyzed include parameters as indicated by MPCA (2014a). In general, all field measurements were typical of high quality, natural surface waters in northeast Minnesota that are not affected by consistently high sediment or organic wastewater inflows that would be associated with urban and/or agricultural areas.

Information regarding groundwater quality will be gained via the proposed hydrogeological study referred to in the previous section.

20.1.4 Stream Morphological Data

Available data include baseline stream morphology data collected by Project personnel to characterize the nature and current stability of select streams and evaluate the physical sensitivity of streams to changes in flow characteristics. Information includes channel cross-section and profile measurements, water surface elevation measurements, bank-full flow elevation measurements, vegetation characteristics, flow rates, and substrate composition. In general, stream morphology data were collected near locations sampled for surface water quality (Barr, 2008a; 2012a). TMM intends to perform stream morphology data collection for streams within and near the proposed mine and associated infrastructure which have the potential to be affected during mine construction and operation.

20.1.5 Wetlands

Publically available wetland data include information from the website of the U.S. Fish and Wildlife Service (USFWS) National Wetlands Inventory (USFWS, 2014b). Desktop and intermediate wetland evaluations have been performed for the majority of the mine support facilities area in 2007 and 2008. To date, desktop wetland evaluations have been completed using aerial photographs, LiDAR topography, and published maps for selected parts of the surface water resource study area. Intermediate-level wetland evaluations were completed on portions of the study area in 2005, 2007, 2008, and 2011 (Barr, 2012b). Data for wetlands examined in the field include approximate boundaries; various wetland classifications (Eggers and Reed, 1997; Shaw and Fredine, 1956; and Minnesota DNR, 2003), and a wetlands functions analysis developed by the Minnesota Board of Water and Soil Resources (Minnesota BWSR, 2014). No detailed delineations associated with possible project infrastructure locations were performed. Detailed wetland delineations to date have been limited to permitting required for drilling programs prior to ground disturbance. Wetlands appear to be less prevalent in the proposed TSF site when compared to the planned mine site.

TMM intends to perform a detailed wetland delineation identifying and documenting the complete jurisdictional wetland boundary. TMM also plans to perform a detailed Minnesota Routine Assessment Method wetland functional analysis for all wetlands directly impacted, for determination of wetland mitigation requirements for direct impacts. This analysis will include preparation of a Wetland Delineation and

Functional Assessment Report. TMM intends to undertake the classification of all wetlands within an area of potential effect according to the Minnesota Native Plant Community Classification system or other agency-designated classification system.

20.1.6 Sediment in Wetlands and Lakes

Sediment sampling has been conducted in some areas of northeastern Minnesota by the MPCA. However, the MPCA sediment study areas were primarily in the St. Louis Watershed in the estuary of Lake Superior approximately 70 miles south of the planned mine site and the proposed TSF area. TMM intends to perform a sediment study for the Project site.

20.1.7 Geology/Paleontology

Data available and used in preparation of the PFS include numerous geological field guides and textbooks. The likelihood of significant paleontological resources is low. Most fossils occur in sedimentary rock formations and not in the rock types typically found in the study area.

20.1.8 Waste Rock, Backfill and Tailings Characterization

The geochemical characterization of materials that will be encountered in the development of the Project will be required, as part of the State of Minnesota permitting process. Minnesota non-ferrous mining regulations require the MDNR to approve the test scope for material characterization. Some limited preliminary test work has been performed on TMM metallurgical test tailings. No formal work for geochemical characterization or leaching characteristics has been done for Project materials (ore, backfill, or waste rock) to date.

In the future, TMM intends to submit work plans for tailing, backfill, and borrow material characterization, obtain MDNR's approval of those plans with co-operating agency consultation as necessary, and implement those studies. Some of the testing that will likely be required may take multiple years to complete.

20.1.9 Vegetation

Information available from the DNR includes descriptions of native plant communities of Minnesota, conservation status ranks for native plant communities and subtypes (DNR, 2009), lists of state endangered, threatened, and special concern plant species (DNR, 2013a), and other material. The USFS has a separate list of sensitive plant species for the SNF (USFS, 2011b). Descriptive information about sensitive plant species is available from the DNR (no date), and the Biological Evaluation for the Forest Plan Revision for Chippewa and Superior National Forests (USFS, 2004). Descriptions of ecological regions and plant communities are available from the DNR (DNR 2014a, b). Lists and information about non-native invasive species potentially applicable to the study area are available through the Minnesota Department of Agriculture (Minnesota Department of Agriculture, 2013), DNR (DNR, 2014c), Superior National Forest (USFS, 2008), and U.S. Department of Agriculture (USDA, 2012). There are approximately 90 state endangered, threatened, or special concern plant species in St. Louis and Lake Counties (DNR, 2014i).

The Minnesota Natural Heritage Program (NHP) within the DNR has data on occurrence of rare plants and plant communities.

Project personnel conducted surveys for sensitive plants and mapped vegetation communities in the proposed mine site area and surrounding waterways in several reports from 2005, 2007, 2008, and 2011. TMM surveys have not been conducted for the biological study area west of Birch Lake, or for the concentrator site, TSF site, and utility routes. Project personnel have not conducted specific surveys for invasive non-native plant species. However, some invasive plant species are included on lists of species observed in the biological study area during the 2005 to 2011 surveys.

Current data suggests that there are no federally listed endangered, threatened, or candidate plant species with potential to occur in the biological study area (USFWS, 2013). Surveys conducted by Project personnel for sensitive plant species found two Minnesota listed endangered, one proposed endangered, and five special concern plant species in the planned mine site area east of Birch Lake and near the former Dunka mine site. No surveys have been completed for much of the current Project configuration, and additional occurrences of Minnesota listed species could be expected.

Regional forester sensitive species (RFSS) are applicable to federal lands managed by the SNF, which occur in the proposed mine site area and along the planned utility routes. RFSS are not formally protected, but their habitat and resource requirements are taken into consideration for management of USFS lands. A number of RFSS have been found during Project surveys completed to date in the planned mine site area east of Birch Lake and near the former Dunka mine. Additional occurrences may be found in biological study areas that have not been surveyed to date including the proposed mine site area that is west of Birch Lake and the planned utility routes. There are 45 vascular plant RFSS and 12 moss and lichen RFSS in the SNF (USFS, 2011b).

Wild rice is a sensitive plant species due to its importance to Native American Communities. Birch Lake has been identified by the 1854 Treaty Authority (2009) as a wild rice resource with good potential to produce harvestable quantities of rice. It is one of 284 wild rice waters identified by the 1854 Treaty Authority; but it is one of only a relatively few waters identified within the biological study area with good rice potential. Project personnel conducted surveys for wild rice distribution in Birch Lake, White Iron Lake, South Kawishiwi River, and related waters from 2009 through 2013.

The reports included a compilation of available DNR lake/stream survey results and other public sources of information. Information on other plant species potentially harvested in the 1854 Ceded Territory is provided in the cultural resources analysis of the NorthMet Mining Project and Land Exchange Supplemental Draft Environmental Impact Statement (SDEIS; PolyMet, 2013).

Areas of high biodiversity are determined by the Minnesota Biological Survey (MBS). However, the MBS has not done an evaluation of biodiversity significance for the area where the Project mine and concentrator areas are to be located (MBS, 2013). Based on preliminary data from the area around the proposed TSF location, no areas of outstanding, high, or moderate biodiversity significance were discovered in the proposed TSF site location. The NorthMet SDEIS (PolyMet, 2013) includes an analysis of areas of high biodiversity significance and imperiled and vulnerable plant communities. TMM can expect that a survey of biodiversity will be required during the EIS process.

The MBS also has sensitivity ratings for plant communities across the state. To date, insufficient information has been collected to assess whether imperiled and vulnerable communities are present, and the MBS has not published reports or completed studies for St. Louis and Lake Counties. Some relevant information may be available from the Minnesota NHP.

TMM intends to perform an initial desk top evaluation to identify threatened, endangered, and special concern species and associated specific seasons that would dictate the field survey schedule for each of these. This task will include preparation of a work plan and DNR input to gain concurrence on methodologies and field techniques. TMM intends to prepare rare plant survey work plan with coordination with USFS and DNR input to gain concurrence on methodologies and field techniques. Wild rice will be surveyed during the rare plant survey discussed above, and a report will be prepared, summarizing the field survey plus all past data collection efforts and including a statistical analysis of the data.

20.1.10 Wildlife

Agency information on species of interest is available from the DNR and USFS. Information on state-listed endangered, threatened, and special concern species is also available from the DNR, including the state list of endangered, threatened, and special concern species (DNR, 2013), lists of species by region and county (DNR, no date), and descriptive information for each species (DNR, no date). The Minnesota State Wildlife Action Plan (DNR, 2006) identified species of greatest conservation need (SGCN) by eco-region and habitat, along with 10-year goals, strategies and priority conservation actions, and biology of these species. The current list of federal endangered, threatened, and proposed species for St. Louis and Lake counties includes two species of interest to the TMM Project; Canada lynx (federal threatened) and northern long-eared bat (proposed endangered) (USFWS, 2013).

Additional wildlife issues identified for the region include wildlife corridors and culturally important wildlife species. The cumulative impact assessment in the NorthMet SDEIS (PolyMet, 2013) includes information regarding wildlife corridors through the historic mining areas. Information on big game (including deer, bear, and moose) are available from the DNR. The cultural resources sections of the NorthMet SDEIS (PolyMet, 2013) provide information and analysis for other culturally important wildlife species.

Project personnel conducted field studies and prepared a Canada lynx assessment in 2008 for a biological study area centered on the Birch Lake deposit east of Birch Lake and an area adjacent to the former Dunka mine site. Field surveys were conducted in 2007 and 2008 within a six mile radius, which included most of the current study area except the proposed TSF site. The proposed mine site, TSF site, and associated infrastructure are located within the designated critical habitat, and the other biological study areas are within or near critical habitat.

The northern long-eared bat is a proposed endangered species in both Lake and St. Louis counties (USFWS, 2013). It is not known whether there is specific information on occurrence of this species in the biological study area. The presence of bats may need to be assessed and a mitigation plan developed if rare, threatened, or endangered species are detected.

For other species of interest, Project personnel conducted winter and spring/early summer habitat and wildlife surveys in 2007 and 2008 for portions of the Birch Lake Deposit. Habitat and wildlife surveys have not been conducted for other portions of the biological study area. Wildlife studies have been conducted for other projects in the vicinity, including the NorthMet Project, for which the results are described in the NorthMet SDEIS (PolyMet, 2013).

Raptors are protected under various state and/or federal laws. It is not known if a specific raptor survey has been performed. Further, the information provided on performed wildlife surveys does not specifically discuss Bald and Golden Eagles, which are protected under the Bald and Golden Eagle Protection Act. It is likely that one or both species are present in the Project area. As part of the EIS process, it is likely that an assessment of the presence of and potential Project impacts to these species will be required. Typically, the lead agency will defer to USFW with regards to survey and mitigation. If nesting locations are present, a raptor monitoring plan may be required.

TMM intends to prepare a wildlife species field survey work plan with input from DNR and USFS to obtain concurrence of the plan if possible. Surveys will determine the presence or absence of species of interest, including threatened, endangered, special concern species, and RFSS in the biological study area. TMM intends to prepare a wildlife assessment report summarizing the results of wildlife surveys. TMM also intends to perform a desk top assessment of migratory waterfowl use in the biological study area. Further, TMM intends to coordinate with USFWS and USFS on the possible need for a Canada lynx report.

20.1.11 Aquatic Biota

The distribution, quantity, and quality of aquatic biota are parameters used by agencies to establish the nature and health of an aquatic resource to support aquatic life, and together are primary stream descriptive datasets. Available aquatic biota data include inventories performed by Project personnel in 2008 in the Dunka River, Flamingo Creek, North Nokomis Creek, and Filson Creek (Barr, 2008b). Methods for the aquatic biota survey followed a work plan approved by the MPCA in June 2008. Additional information on aquatic biota in Unnamed Creek east of the Dunka Mine was collected by Cliffs–Erie in 2001. TMM performed aquatic biota inventories in Dunka River, Flamingo Creek, North Nokomis Creek and Filson Creek in 2008. Future work may include expansion and modification of the previous aquatic biota inventories as needed and as indicated by agency consultation.

20.1.12 Fisheries and Aquatic Resources

Information on current fish sampling and fishery management is available for Birch Lake, White Iron Lake, and Little Lake (edge of concentrator area) from the DNR (DNR 2014d), but not for Mud Lake in the TSF site area. Trout streams and lakes are mapped by the DNR (DNR 2014e).

Fish reported for Birch and White Iron Lakes include black crappie, northern pike, rock bass, cisco, walleye, white sucker, and yellow perch (DNR, 2014d). Little Lake adjoins the concentrator area, occupies 61 acres, and has northern pike, walleye, and white suckers. All fish species in these lakes have fish consumption advisories for mercury. At the time of this report, no fishery information is available for Mud Lake, Hay Lake, or Iron Lake, which are near the planned TSF site. The DNR lake database does not provide information on non-game fish or macroinvertebrates, but it does include information on aquatic plant surveys and invasive species for some lakes. There are no identified trout streams or lakes in or near the biological study area (DNR, 2014h).

Information on state-listed endangered, threatened, and special concern aquatic species is available from the DNR (DNR, 2013a), as well as lists by region and county (DNR 2014i), and links to descriptive information for each species (DNR, 2014i). The Minnesota NHP within the DNR has data on occurrence of sensitive aquatic species. There are three sensitive fish species in the Rainy River basin and six in the Lake Superior basin (DNR 2014i). Additionally, several state threatened or species of special concern may be present in the Rainy River basin, (three mussels and one amphibian) and in the Lake Superior basin (three mussels, three wetland and aquatic reptiles, and one amphibian).

Many of the lakes and some rivers in northeast Minnesota have mercury concentrations in fish tissue that are higher than the state's human health chronic standard. These waters include Birch Lake, White Iron Lake, Little Lake, Bear Island Lake, and many others in the Rainy River watershed, and Embarrass Lake in the Lake Superior basin. Most of the mercury in fish tissue in northeast Minnesota is from atmospheric deposition, with most of the mercury deposition originating out of state (MPCA, 2013).

Project personnel conducted monitoring for macroinvertebrates and fish in the Dunka River and three creeks in 2008. Macroinvertebrates and vertebrate (fish) surveys have been conducted since 2001 at several locations near the former Dunka mine site in Unnamed Creek. According to the 2012 Work Plan for Aquatic Resources (Barr, 2012e), additional agency survey data for streams and lakes may be available from the USFS and MPCA.

TMM intends to perform a future sampling in the study area that may include baseline aquatic biota sampling in representative reaches of each stream/river for fish and mussels, and sampling of macroinvertebrates.

20.1.13 Cultural Resources

Cultural resources include a wide range of archaeological, historic, and natural resources. Cultural resources include items over 50 years of age, so the potential for discovery of resources of relatively recent eras could be high. Archaeological sites could include campsites, trails, rock art, chipped stone, and pottery scatters. Historic sites could include buildings (or building remains), campsites, artifact scatters, logging camps, railroads, historic landscapes, and mines.

Natural resources could include geographical and geological features, natural landscapes, plant/resource gathering areas, and locations where events important to Native Americans took place. These types of resources have been identified in the region and in the general location of the Project.

Various Federal and State laws apply to the discovery, evaluation and protection of cultural resources, and include, but are not limited to, Section 106 of the National Historic Protection Act, Archaeological Resources Protection Act of 1979, Native American Graves Protection and Repatriation Act, and the Antiquities Act.

Locations of cultural resources specific to Project features have not yet been identified. Archeological and historic surveys have been conducted in limited portions of the study area. TMM informally participated in one quarterly tribal mining meeting, and has also informally engaged in discussions relating to cultural resources.

TMM intends to conduct field surveys of all areas of direct disturbance including defined corridors and all planned mine facility locations. Cultural resources will be assessed via standard regulatory-prescribed cultural resources surveys of the entirety of the potentially-affected area, once the “Area of Potential Effect” is determined. Cumulative, distant, and “later-in-time” indirect impacts to historic/architectural resources will be also assessed (e.g. visual impacts). TMM intends to continue informal discussions with tribal government officials and knowledgeable elders.

20.1.14 Air Quality

Published data include measured ambient air quality levels for criteria pollutants and has been used to designate the compliance status of the study area for each of the criteria pollutants. Project personnel collected site specific baseline ambient air quality and meteorological data in 2013. The location of the monitoring site and selection of equipment was determined with informal assistance from MPCA. The data were collected on ground held by TMM adjacent to the Dunka property. These data have not been compiled. A technical work plan for monitoring will be prepared by TMM. TMM may establish and monitor air quality and/or meteorological conditions at two to three additional yet-to-be constructed monitoring stations. An additional work plan may be prepared for air quality modeling efforts (Anne Williamson, phone communication 07/23/14). A review of the surveyed parameters for the monitoring station indicate that the monitored conditions are likely adequate for the PFS stage of the Project (Stephen Ochs, P.E., AMEC email message, 07/28/2014).

The existing ambient air quality of the study area is good. Criteria pollutant levels are below applicable National Ambient Air Quality Standards (NAAQS) and Minnesota Ambient Air Quality Standards (MAAQS) in both Lake and St. Louis Counties. Both counties have been designated as "attainment" by the USEPA for all criteria pollutants.

20.1.15 Noise

The land use in the SNF is mostly forestry. The region surrounding the Project area has traditionally supported various mining activities, as well as logging, on federal, state, county, and private forest lands. Noise sources associated with mining activities include drills, explosives, dump trucks, excavators, crushers, and power generators. Existing noise and/or vibration information applicable to this Project are publically-available information for similar projects and environments, as well as anecdotal information from TMM staff.

Regionally proximate noise-sensitive natural areas include the Boundary Waters Canoe Area Wilderness, wildlife corridors, and wild rice waters/beds that are known to be used by tribal members for harvesting.

TMM has collected only spot noise data for exploration drilling noise at several receptor locations. TMM intends to conduct noise monitoring throughout the study area; include ambient noise measurements taken at numerous existing and potential sensitive receptors during all four seasons.

20.1.16 Socioeconomic Studies

TMM intends to collect data for the study area, including population, income, public finance, housing, public service/infrastructure, commercial/retail centers, recreational facilities, public gathering places, community structure and other religious/cultural groups. For this analysis the study area is defined as 100 mile radius of the proposed decline sites, to ensure inclusion of all three water basins and key Minnesota ports on Lake Superior. Evaluation would address the local community data needs required for the Mine Plan of Operations, such as information regarding population demographics, and significant employers. The assessment would identify minority and low-income populations, and illustrate racial distribution, describe household income character and persons below the poverty level, compare racial and income data for the adjacent regional area and within Minnesota as a whole, and assess access and availability to health care by sensitive populations. TMM would evaluate cultural context as part of the socioeconomic study.

TMM would also identify the historical context and current conditions as related to Eastern Europeans and early mining communities, recent immigrants, and other cultural and/or religious groups.

20.1.17 Land Use and Recreation

The study area includes: the SNF, managed by the Land and Resource Management Plan; Minnesota state lands, managed by the DNR Forest Management Plan; and lands managed by the Lake County Management Plan and St. Louis County Comprehensive Land Use Plans. The Project is located within the territory ceded under the 1854 Treaty of LaPointe, which reserves specific off-reservation hunting, fishing, and gathering rights to the Grand Portage Band of Lake Superior Chippewa, Bois Forte Band of Chippewa (1854 Treaty Authority, 2014), and the Fond du Lac Band of Lake Superior Chippewa..

The study area and adjoining federal lands are a popular and highly valued destination for recreation. Recreational activities that occur within the land use and recreation study area include all-season sports and activities. Mining and timber harvesting activities also occur in the area. Local, state, federal, and tribal management frameworks regulate the use of these lands.

No specific data has been collected to date by TMM regarding land use and recreation. Available published data include federal, state and county land use plans. Future planned work is to include collection of existing documents and plans to evaluate existing and planned uses. GIS data and planning documents would be collected for important resources including, but not limited to, federally, state, and locally managed parks, wilderness areas, scientific and natural areas, wildlife management areas, designated trout streams and lakes, boat/lake access areas, and campgrounds. Timber resources will also be considered. TMM intends to prepare a land use report to summarize collected land use and recreations surveys and data collection.

20.1.18 Visual Resources

The Project area is surrounded by wetlands and mixed deciduous and coniferous upland forests. Key observation points where the view of the proposed mining activity would be most revealing may be present at: clusters of rural homes near Babbitt and Ely, within the town of Babbitt, at cabins near Birch Lake, around fishing areas, and on forest roads within the SNF.

Bois Forte, Fond du Lac, and Grand Portage Bands of Lake Superior Chippewa tribal members exercise rights to hunt, fish, and gather on SNF lands, including lands near the planned mining area. The frequency with which tribal members exercise these rights in portions of the SNF with views of the mining area is not known. Because the SNF is considered a culturally-important location, visual intrusions from the Project are a significant consideration.

The USFS has a management plan for primitive management areas (PMAs) within the Boundary Waters Canoe Area Wilderness. PMAs are managed to provide an experience "relatively free from the sights and sounds of humans." The nearest PMA within the Boundary Waters Canoe Area Wilderness is approximately four miles from the TMM property boundary (Barr, 2011). The purpose of the management plan, in part, would be to protect these areas from visual elements that result in intrusions to the visual experience for users of the Boundary Waters Canoe Area Wilderness.

TMM intends to conduct a visual assessment to define the visual character of the study area and to identify visual receptors. Modeling of visual impacts during the course of construction and operation, and over seasonal variations, is planned.

20.2 Waste Management

20.2.1 Waste Rock and Tailings Storage Facility Requirements

TMM intends to temporarily store waste rock and mined ore in an above-ground engineered storage facility, the SRSF. The majority of storage will be on a temporary basis, until the ore can be processed through the concentrator. A portion of the pre-production waste material will be used in construction where environmentally suitable; and some waste material will be used as mine backfill. However, some of the waste material may be permanently stored in the SRSF. TMM intends to permanently store processed tailings in the TSF, which will be constructed and eventually closed in accordance with regulation and approved designs. Designs for the SRSF and TSF were developed based on the appropriate regulations given in Minnesota Administrative Rules, Chapter 6132. Additional information on the storage facilities is provided in Section 18.

20.2.2 Solid Waste and Hazardous Waste

Solid waste would be generated during construction and operation of the mine, concentrator, and TSF sites. Solid waste handling and recycling would include management of hazardous, universal, and general waste, as well as recyclable materials generated at the site. Management of tailings is not included in the solid waste management program. Any biohazard waste generated by emergency services would be managed by the emergency services facility due to the specialized handling required.

Solid waste handling and recycling will likely include the following features:

- A formal solid waste and recycling program;
- Hazardous waste management program: The TMM Project would likely qualify as a small quantity hazardous waste generator (SQG), and depending upon the amount of hazardous waste generated, may qualify as a conditionally exempt small quantity generator (CESQG). Hazardous waste would be generated primarily by maintenance of vehicles and equipment. Universal waste (light bulbs, batteries, etc.) would also require segregation and would be collected separately from general waste and appropriately stored prior to disposal. Hazardous waste would be stored at the point of generation in suitable containers including cabinets and barrels, and would then be transferred to appropriate storage until a qualified hazardous waste contractor removes the waste for disposal. The hazardous waste management program would include training for staff so that hazardous waste is managed correctly.

20.2.3 Liquid Waste

20.2.3.1 Process and Waste Water

Process and waste water streams would include process water storage in the concentrator process water pond and the TSF intermediate collection pond. The expected chemical composition of process water in the concentrator process water pond is not well known at this time. Water from the concentrator process water pond would be cycled back to the concentrator, lost to evaporation, or conveyed to the TSF with tailings. No discharge of process liquids is presently planned for the Project. Both process water ponds described would be lined to minimize seepage. Liners would be designed to meet seepage limits as per MPCA regulatory guidance.

20.2.3.2 Concentrator Process Water Pond

The concentrator process water pond, to be located at the concentrator site, would store approximately 190 acre-ft of process water. Process water would be pumped from the concentrator process water pond to the concentrator at a rate of approximately 25.7 M gal/d.

20.2.3.3 TSF Intermediate Collection Pond

The TSF intermediate collection pond, to be located at the TSF site, would store approximately 650 acre-ft of process water. Water reclaimed from the TSF would be pumped from the TSF intermediate collection pond to the concentrator process water pond at a variable rate dependent upon concentrator demand.

20.2.3.4 Petrochemical Wastes

Petrochemical wastes within the Project would be generated by maintenance activities, including maintenance of the surface and underground vehicle fleets, and will include used oil and lubricants drained from machinery and mobile equipment. On-site storage, pickup and transfer facilities would be designed, constructed, and operated in accordance with applicable law and would include secondary containment as required. TMM intends to maximize recycling of petrochemical wastes.

At the current stage of development, there is not enough specific information to accurately describe the type of petrochemical wastes, or the quantity and discharge rate of site emissions from petrochemical wastes.

20.2.3.5 Liquid Wastes

Liquid wastes will include non-contact stormwater, contact stormwater, and wash water.

Non-contact stormwater is defined as runoff that does not come in contact with mined materials. It is expected to have characteristics typical of rural runoff. Non-contact stormwater would originate from non-contact roads. Contact stormwater is runoff that does come in contact with mined materials, such as ore, waste rock, concentrates, process waste, or process water. Contact stormwater would be collected from contact roads and other portions of the mine, concentrator, and TSF sites by the respective contact water management system. Return water from vehicle washes is generally assumed to have a composition similar to contact stormwater due to contact with road dust. Wash water would be generated at the mine vehicle wash, primary portal truck wash, the secondary portal truck wash, the concentrator truck wash, and the TSF truck wash.

Contact stormwater and non-contact stormwater would be handled separately. Non-contact stormwater would be directed off-site by site grading, berms, and ditches. Contact stormwater from the mine and concentrator sites would be collected and conveyed to the concentrator process water pond. Contact stormwater from the TSF site would be collected and conveyed to the TSF intermediate collection pond for use in the process. Wash water would be routed to the contact water management systems.

20.2.4 Water Management Requirements

According to the most recent water requirement estimates, the TMM Project will use an average of about 4 M gal/d of water at the concentrator. After the initial appropriation of this water, approximately 3.2 M gal/d of the total daily water needs will be obtained by recycling water from processing operations and mine dewatering in a “closed-loop system” between the concentrator and TSF. The remaining 0.8 M gal/d must be obtained from a makeup water source. The Project does not intend to discharge process water to the environment, but based on the available water balance data this condition cannot be verified. Losses of water from the process circuit will be through evaporation from ponds, water entrained in the tailings and mining paste backfill, and the moisture entrained in the shipped concentrate. Should further study of the water balance indicate that discharge of process water will be required for years with high precipitation or low evaporation, TMM will likely be required to obtain an NPDES/SDS permit for the discharge of process water. TMM will likely be required to install a water treatment plant if the zero-discharge condition for the life of the mine cannot be verified.

TMM has not yet selected a preferred water source for the TMM Project. TMM has evaluated five potential sources of makeup water supply: Dunka Pit, Birch Lake, groundwater aquifers, Dunka River, and mine pits that are part of the existing Peter Mitchell iron ore operation by Northshore Mining (an affiliate of Cliffs Natural Resources). TMM has determined that all of these potential makeup water sources are technically viable for its projected need of 0.8 M gal/d of makeup water. Appropriation of makeup water from the selected water source will require Minnesota DNR approval of a water appropriation permit and compliance with other specific Minnesota DNR requirements.

Water management requirements are described in more detail in Section 18.

20.3 Environmental Management Plan

The Project would be subject to numerous environmental requirements in addition to those imposed in the permits that would be obtained prior to construction and operation. To address the various environmental requirements, TMM would develop an Environmental Management Plan for Operations.

The majority of these requirements address the handling, storage, disposal, and reporting of hazardous and other materials. Protocols for transportation, handling, and storage of hazardous materials would be included in the Environmental Management Plan for Operations. The Environmental Management Plan would also include any monitoring and mitigation measures identified during the Project design and preparation of the Mine Plan for Operations (MPO). Environmental data gathering, permitting, and management and monitoring measures for the closure phase would also be developed at the conceptual level for inclusion with the MPO, and in detail prior to final closure.

Drill permits are applied for as needed, and to date these approvals have been granted for the exploration programs upon application. At the current stage of Project development, TMM has not yet applied for the necessary permits required for construction, operation and/or closure of the Project. Currently the only permits, bonds, and fees that would impact Project development are related to mineral or land tenure.

20.4 Environmental Liabilities

Current liabilities associated with the mineral exploration program would be related to abandonment of boreholes and drill pad and road reclamation. Reclamation bonds have been posted with the BLM and the Minnesota Department of Transportation (MnDOT).

Historical mine features on the Project site include two former bulk sample sites; an underground shaft and workings developed in 1968 at Maturi and a surface excavation developed in 1974 at Spruce Road. These sites were developed by the former lease-holder.

The Maturi shaft is approximately 1,100 ft deep with approximately 700 ft of drifts. Reclamation work at the shaft site was done by the former lease-holder after completion of the sampling, and included removal of all surface structures and installation of a concrete cap in the shaft. An uncapped small remnant rock stockpile is present near the shaft collar. The stockpile is sparsely vegetated, and no indication of impacted surface runoff is present (AMEC, 2012). Sulfide-bearing rock has been visually identified in the remnant stockpile. Excavations used for settling ponds during the bulk sample operations have not been backfilled.

The surface excavation bulk sample site at Spruce Road is approximately 20 yards in diameter and 10 ft deep. The former lease-holder reclaimed the site by capping it through backfilling and grading it to mimic the original topography. Several subsidence areas are present in the capped area. Seepage emanating from the site contains elevated concentrations of sulfate, copper, and nickel. In 2010, the USFS studied water quality in the seep and surrounding surface waters. That study concluded that background loading of surface waters in the area was naturally high in sulfate, copper, and nickel, that the seep had no measurable negative impact on the watershed, and that no additional management action or testing would be needed (USFS, 2010).

TMM has reclamation responsibilities under applicable leases, and may be responsible for additional reclamation of the bulk samples sites if required (Mark Hall, AMEC email communication, August 25, 2014). However, no specific reclamation has been requested by any agencies to TMM's knowledge and no reclamation plans had been developed by TMM at the Report effective date.

Ongoing liabilities at the adjacent Dunka property include permitted discharges from a sulfide-bearing rock stockpile and wetland treatment system, and permitted discharges of untreated mine pit water.

Some general environmental commitments have been presented to stakeholders, including communities, elected officials, and government agencies. Several documents related to the proposed hydrogeological plan have been submitted to agencies, making them publicly-available. The documents include land access applications and a technical study work plan.

20.5 Environmental Risks

The environmental risks of highest consequence would be related to:

- Contamination of surface water and soils due to a containment failure at stockpiles, ponds, pipelines, TSF, or other facilities
- The possibility of discharge of process water during years of high precipitation, which would likely require the installation of a water treatment facility
- Refusal of permits for backfill with additives such as fly ash or slag (refer to Section 18.6) due to the potential for unacceptable environmental impacts
- Unanticipated fugitive dust emissions from stockpiles, roads, and TSF
- Unanticipated impacts to surface waters due to mine dewatering
- Unanticipated impacts to sensitive receptors, including, but not limited to, the Boundary Water Canoe Area Wilderness, and federally-listed endangered species.

These risks would be investigated during the MPO and, as part of more detailed studies, additional engineering, environmental testing, and mitigations would be developed.

20.6 Closure Plan

As the Project is in very early stages of development, the information that follows is based on the current Project understanding, which could change as the Project moves forward through development of the MPO and as a result of more detailed studies. Definition of closure requirements is expected to begin during MPO development.

During MPO preparation, initial regulatory closure requirements would be defined for the Project. Facility closure plans would be defined in permits and required to be annually updated with the annual report. Recently, local communities have been instrumental in developing and executing closure plans. Examples of community closure integration have been developed for Kennecott's Flambeau Mine in neighboring Wisconsin, where selected project structures were donated to the county highway department, as well as Lundin's Eagle Mine in Michigan, where designated buildings are planned for donation to the county. The development of the Project's closure plan would also be subject to public input during environmental review.

Closure obligations, as described in Table 20-1, are based upon interpretation of statutes and regulations and related mining project experience within Minnesota and the US.

Project components that must be addressed by regulation in the closure plan include roads, parking areas, storage pads, high voltage transmission lines, pipelines, railroads, and all other equipment, facilities and structures. Land rehabilitation would follow decommissioning of plants, salvaging of equipment, and demolition of structures. Land surfaces would be graded to topography close to pre-mining land surface topography (excluding the TSF site) and vegetated using native species. In addition, facility closure would require landscape restoration pertinent to all surface facilities.

20.6.1 Planning-Level Closure Costs

Preliminary plans for the closure of the TSF will be construction of an engineered cover to close the TSF in place. As part of the PFS process, TMM contracted a TSF reclamation review, which was completed by TMM's Environmental Consultant. The unit closure costs were based upon historical closures practices within Minnesota and the US. The recommended planning-level cost estimate closure method for the TSF was estimated to be approximately $151 M by TMM's Environmental Consultant. This estimate covered the TSF and did not include closure costs that would be associated with closure of the concentrator and mine surface area.

In the closure of all mine facilities, compliance with surface water and groundwater standards would be required. Site maintenance and monitoring activities would be required for a period of time, as defined by facility permits and until successful reclamation has been established for the facility.

Table 20-1: Key Closure Obligations

Closure Obligation	Description
Mining Operations	Closure of all underground mining operations would include removal of salvageable equipment, backfilling stopes and laterals to drifts, flooding of openings, and sealing of raises. Compliance with surface water and groundwater standards would be required.
Concentrator Operations	Closure would require cleaning and removal of process equipment, demolition of buildings and structures, and site grading. Compliance with surface water and groundwater standards would be required.
TSF and SRSF Operations	TSF and SRSF closure would include capping or otherwise stabilizing the TSF and the SRSF, establishing grades and other features as required for stormwater control and be in compliance with surface water and groundwater standards.
Surface Infrastructure Operations	Surface infrastructure closure would include salvage and removal of designated equipment and surface structures and site grading for stormwater control. Compliance with surface water and groundwater standards would be required.

Presently, TMM has no plans to release properties post-closure and it is TMM's intent to maintain control of the properties after closure. Specific activities have not yet been identified by TMM for post-closure periods. It is expected that TMM will identify these actions during the MPO study phase and during more detailed Project studies. The plans will outline post-closure goals and maintenance and monitoring actions, as well as specify resources required to conduct the activities. These plans would be developed by the TMM closure team and would be subject to the approval of federal and state regulatory authorities. A management plan would also be developed as part of the Project closure policy. The management plan would describe the management team's responsibilities for project closure. Where expertise for a specific closure management action is required, outsourcing of teams or individuals may be considered by TMM.

For the purposes of the financial analysis in Section 22, as no closure costs were included in the PFS, AMEC included a conceptual site-wide closure cost allocation in this financial model of $210 M, based on benchmarking with similar projects. The closure cost estimate does not include any allocations for post-closure monitoring.

Because the final size and configuration of the SRSF may be subject to change during permitting process, and closure requirements are not known, costs associated with closure of the SRSF have not been estimated.

AMEC notes that the final closure cost estimate will depend on the MPO phase, when the Project design is optimized, and will also depend on the conditions that may be imposed on TMM during permitting.

20.6.2 Financial Assurance Requirements:

The Project is subject to several overlapping federal and state financial assurance requirements that are tied to successful closure and reclamation of a mine. These financial assurance requirements must be in place before the necessary permits and approvals, including the Minnesota Permit to Mine and the BLM's approval of the MPO, would be issued. These financial assurance requirements are created both by mining-specific statutes and regulations, and by general environmental laws governing the issuance of various environmental permits and approvals.

The ultimate amount of the financial assurance required for the Project would depend, in part, on the cost of permit compliance, closure and post-closure obligations, and other reclamation activities. TMM would be able to estimate this amount with more accurately during the development of the MPO and Permit to Mine application.

Any such financial assurance cost estimates are, of necessity, preliminary at this time, and will be subject to change both as the design for the TMM Project is refined and the applicable regulatory agencies consider and impose financial- assurance requirements associated with the specific required permits. The financial assurance requirements relate to the following permits and authorizations: DNR Permit to Mine, BLM Federal Mineral Leases, CWA Section 404 Permit, NPDES/SDS Permit, Wetlands Conservation Act Permit, and the Underground Injection Control Permit.

In general, the federal and state financial assurance requirements applicable to the Project are designed to ensure that the responsible agencies could access sufficient funds to ensure completion of all reclamation (including closure and post-closure) requirements and contingencies in the event that TMM, for any reason, would be unable to fulfill its permit obligations. The assurance requirements are based on federal and state wage laws, and represent the estimated costs if the applicable agency would have to hire contractors at prevailing wages. Because of both the federal and state requirements, TMM would likely need more than one financial assurance instrument, although a single surety bond may possibly cover the majority of the required financial assurance. In general, sureties such as a corporate guarantee are not permitted for the majority of financial assurance requirements.

It may be possible to combine at least some of the various financial assurance obligations as required under federal and state law. It may also be possible for TMM to consolidate various financial assurance requirements when it pursues an MOU with the lead federal and state agencies for the Project.

The USEPA has announced that it is preparing to issue "Hardrock Mining Financial Assurance Requirements" under the Comprehensive Environmental Response, Compensation and Liability Act (CERCLA), which is the federal statute governing the cleanup of releases of hazardous substances. These CERCLA financial assurance requirements may prove redundant of other requirements summarized above, but the USEPA has suggested that the proposed rule would take into consideration the parallel financial responsibility requirements imposed through other federal and state programs (USEPA, 2009). USEPA has announced that it expects to publish this proposed hardrock financial responsibility rule sometime in 2016 (USEPA, 2014).

20.7 Environmental Review and Permitting

The Environmental Review and permitting process is a highly-regimented process that would require a high degree of communication between TMM and the relevant federal and state agencies, as well as the tribes and local governments in the vicinity of the Project (e.g., Lake and St. Louis Counties and the cities of Ely and Babbitt). During TMM's preparation of the MPO and the actual filing of the MPO, which triggers the Project EIS and subsequent permitting, TMM would undertake a number of activities that require federal and state agencies to make decisions under various statutes and regulations. TMM would need to obtain federal and state agency approval for most activities undertaken for the Project. There would be at least eight federal and state agencies involved in reviewing the Project, with varying degrees of overlapping authority. It will be important to identify the lead agencies in these cases.

The Project would be subject to NEPA at the federal level, and the MEPA at the state level. Under NEPA and MEPA, the Project would be subject to environmental review by multiple state and federal agencies. The Environmental Review process is a critical preliminary regulatory step for agency approval of almost any activity proposed by TMM. It should be noted that the environmental review and permitting process, including the development and issuance of an EIS is likely to take several years, and the final decisions regarding the EIS and permits are subject to appeal. This could cause significant delays to the commencement of the project.

After finalizing the MPO, TMM would be required to file it with the BLM and equivalent documentation with the DNR, which would commence the Project EIS/permitting phase. The MPO filing would trigger a joint federal-state Environmental Review process in which the BLM and DNR, as the agencies responsible for federal and state minerals, respectively, and likely the USFS, as the federal surface manager, would act as the lead agencies in developing an extensive Project EIS that would take several years to complete.

Concurrently with the Project EIS process, TMM intends to file applications for a wide variety of permits with federal and state agencies, which would initiate other regulatory review procedures. There is some risk to this concurrent review, since findings of the EIS may necessitate changes to the mine design, lengthening the permit application review time and efforts. Permits for the construction and operation of the Project would only be issued after the finalization of the EIS.

20.7.1 Environmental Assessment Worksheet

The MEPA process begins with Project personnel preparing and submitting an Environmental Assessment Worksheet (EAW) to the agencies. The EAW is a brief description of the potential impacts of a project. The EAW contains a series of questions about impacts on air quality, water quality, wildlife habitat, traffic patterns, etc. The EAW is used by the regulatory agencies to determine whether the environmental impacts from a project are potentially significant. The EAW would be used by the agencies in helping to define the scope of the EIS.

The studies described in Section 20.1 would be undertaken to identify preliminary impacts as a result of the proposed Project. The detailed scopes of these studies, with the exception of the hydrogeology study, have not yet been developed. Results of modeling and impact assessments would be reported in the EAW. The lead federal and state agencies would further define impact assessments during the environmental review, as prescribed in NEPA.

The methodology and approach to predictive impact analysis and modeling to be performed would be based on NEPA and MEPA guidelines, federal and state requirements, knowledge of other agency missions and expectations, and lessons learned from other mine projects of similar magnitude within the geographic region. The lead agencies would be responsible for the implementation of the NEPA and MEPA compliance processes. Therefore, their decisions would be based on an independent review of the data provided, and predictive impact analysis. If the agencies determine that additional analysis would be needed, it would be within their authority to collect their own data and conduct the analysis for consideration in the environmental review process

20.7.2 Environmental Impact Statement

Under Minnesota law, the proposed TMM facilities must categorically develop an EIS. NEPA and MEPA processes are similar for the preparation and evaluation of an EIS. The lead agency will undertake the following major steps required in development of the Project EIS:

- Identify the lead agencies
- Scope the EIS, including 1) a preparation plan, 2) a strategy for public involvement and interagency/intergovernmental coordination and consultation, 3) an identified proposed action, 4) identify the purpose and need, alternatives to be considered and impacts to be analyzed, 5) identify information and data needs, 6) identify cooperating agencies, 7) determine contracting needs, staffing and budget need and a proposed schedule
- Conduct the analysis and prepare the Draft EIS
- Issue the Draft EIS for public review and comment, including holding public meetings/hearings as necessary
- Analyze comments and prepare the Final EIS, including reevaluate and revised the preferred alternative or proposed action based on comments received

- Issue the Final EIS
- Reach and record the decision (Record of Decision (NEPA) and Determination of Adequacy (MEPA)).

Once the Determination of Adequacy and the Record of Decision has been issued, appeals of the decision can be made, either by TMM or other affected parties.

20.7.3 Permits

Before constructing the Project, TMM would have to obtain a number of federal, state, and local permits. The permitting process would be strongly influenced by the information obtained and the alternatives considered during the Environmental Review process of the EIS.

The primary permits that TMM would be required to obtain are identified in Table 20-2.

Table 20-2 does not include, with the exception of the MPO filed with the BLM, agency authorizations of plans of operations or other approvals not characterized as permits.

TMM would prepare a permit application based on the mine design that would be incorporated into the federal MPO and the state Permit to Mine submittals. TMM expects that the permit review process would generally proceed in tandem with the Environmental Review process, although final permits would not be issued until the Project EIS has been completed.

Using the information obtained and the mitigation measures and other stipulations and recommendations identified in the Project EIS, the federal or state agency may require modification of the permit applications before or after issuing a draft permit. The agency is frequently required to provide notice and an opportunity to comment on the draft permit to members of the public. The agency is also generally required to consult with tribal entities and other federal and state agencies regarding cultural and historic resources in order to meet federal NHPA and comparable state requirements. The agency is also required to consult with other federal and state agencies with respect to endangered and threatened species to satisfy federal ESA and similar state obligations. The agency would then evaluate the information obtained during these comment and consultation processes, and decide whether to grant the permit, include conditions, or deny the permit. Decisions regarding permits can be appealed by TMM or other affected parties.

Table 20-2: Key Permits

Regulatory Requirement	Jurisdictional Agency
Mining-Specific Permits	
Permit to mine	State/DNR
Federal MPO	Federal/BLM (with USFS input)
Environmental Permits	
NPDES/SDS for process water and storm water discharges	State/MPCA; Federal/EPA
Injection of underground fluid	Federal/EPA
Discharge of dredged and fill materials/wetlands conservation	Federal/USACE, USEPA; State/DNR
Water appropriation	State/DNR
Public waters work permit	State/DNR
Dam safety	State/DNR
Air emissions control	State/MPCA; Federal/EPA, USFS
Resource Conservation and Recovery Act (RCRA)/solid waste storage	State/MPCA; Federal/EPA
HV transmission line	State/Minnesota Public Utilities Commission (MPUC)
Gas pipeline	State/MPUC
Special use and road use permits	Federal/USFS
Local Permits	
Conditional use	County
Building	County

20.7.4 Water Rights

The proposed mine and concentrator are located in the Rainy River drainage basin. While the water source is likely to be within the Rainy River water basin, TMM may, during future studies, evaluate whether the water source could be located within the Great Lakes Water Basin. If the water source is located in the Rainy River water basin, no inter-basin transfer would be involved in transporting water from the water source to the concentrator. If, however, the water source were situated in the Great Lakes water basin, studies would need to be initiated to confirm permitting and other regulatory requirements in support of such transfers.

The TSF is planned to be located within the Great Lakes water basin. Slurries from the concentrator would transfer from the Rainy River water basin to the Great Lakes water basin. Likewise, if the water source is located in the Rainy river basin, but water entrained in the tailings is transferred to the Great Lakes water basin, this too could be construed as an inter-basin transfer. Studies would need to be initiated to confirm permitting and other regulatory requirements in support of such a transfer.

Use of the makeup water in the water source would require both rights to use the water itself as well as control of the properties surrounding the water (i.e., the riparian property rights). Riparian property rights are necessary for both physical infrastructure and to qualify for a DNR water appropriation permit. Comparable rights would be necessary for selection of groundwater as the water source.

20.8 Considerations of Social and Community Impacts

The history, culture, and landscape of Minnesota's Iron Range have been shaped by mining, and mining continues to be the region's major employer and economic sector. Mining enjoys a long history in the region. The northeastern region of Minnesota, where the Project is located, is home to a significant iron ore industry that has spanned more than 100 years of continuous production. In the early 2000s, companies such as Duluth Metals, Franconia, and others began to pursue copper–nickel mining in the region. Currently, several companies in addition to Duluth have mineral interests in the area and are pursuing projects in various stages of development.

20.8.1.1 Demographics

The Iron Range consists of seven counties – Aitkin, Carlton, Itasca, Koochiching, Lake, Cook, and Saint Louis. These counties encompass slightly more than 18,000 square miles, or 23% of Minnesota's land area. The Iron Range economy has historically relied on iron ore and taconite mining, forestry, and tourism.

The Iron Range is Minnesota's most prominent four-season outdoor recreation region. Iron Range counties include more than 20 state parks and recreation areas, more than a dozen state forests, the Superior and Chippewa national forests, the Voyageurs National Park, and the Boundary Waters Canoe Area Wilderness.

The Project falls within the jurisdiction of two Iron Range counties, St. Louis and Lake. While the population in St. Louis County, the Iron Range's largest county, has stabilized over the past 30 years at approximately 200,000, the county's population has declined approximately 14% since 1960. The population of Lake County has seen a drop of 22% since 1960. Populations in cities of closest proximity to the Project have also declined.

20.8.2 Stakeholder Identification

Project stakeholders are likely to include local, state, or federal government elected bodies or regulatory agencies, state and local business interests, educational institutions, local community interests, local Indian bands, and non-government organizations (NGOs). While some informal discussions have been undertaken, to date no formal stakeholder consultations have been initiated.

20.8.3 Native American Interests

The United States has a unique legal and political relationship with Native American tribes as provided in the US Constitution, treaties, and federal statutes. These relationships extend to the federal government's historic preservation activities, mandating that federal consultation with Native American tribes be meaningful, in good faith, and entered into on a "government-to-government" basis. The phrase "government-to-government" reflects the unique status Native American tribes hold as sovereign nations with extensive powers of self-government.

Native American tribes play an important and high-profile role in the environmental review and permitting process for mining projects, as described in the federal National Historic Preservation Act (NHPA), the National Environmental Policy Act (NEPA), and other statutes, regulations, executive orders, and federal policies. Section 106 of the NHPA requires that federal agencies consult with tribal governments to assess the effects of federal undertakings on historic properties, and requires federal agencies to consult with any tribe that attaches religious and cultural significance to historic properties that may be affected by those undertakings. Tribal entities can also be official "cooperating agencies" in the development of an Environmental Impact Statement (EIS) for mining and other projects that are subject to review under NEPA and its state-based statute, the Minnesota Environmental Policy Act (MEPA).

In Minnesota, there are seven Anishinaabe (also known as Chippewa or Ojibwe) reservations and four Dakota (Sioux) communities. TMM's land and mineral interests are not on tribal reservation land, but are located in a region where the hunting and gathering rights of certain tribal entities are ensured and protected under federal treaty. There are three tribal bands of Minnesota Anishinaabe (aka Chippewa or Ojibwe) in close proximity to the Project, and who have treaty-protected hunting and gathering rights in the region:

- Bois Forte Band of Chippewa
- Grand Portage Band of Lake Superior Chippewa
- Fond du Lac Band of Lake Superior Chippewa.

These three tribal bands are likely to have the greatest input from the Native American perspective to the Project's environmental review process.

The 1854 Treaty Authority is an inter-tribal natural resource management organization that manages the off-reservation hunting, fishing, and gathering rights of the Grand Portage Band of Lake Superior Chippewa and Bois Forte Band of Chippewa in the territory ceded under the 1854 Treaty of LaPointe. These rights also apply to the Fond du Lac Band of Lake Superior Chippewa, but they are not represented by the 1854 Treaty Authority.

Tribal rights under the 1854 Treaty of LaPointe apply to a land area of more than 10,000 square miles in northeastern Minnesota. All of the proposed Project land and mineral interests fall within the territory covered by the 1854 Treaty of LaPointe.

The 1854 Treaty Authority is governed by a 10-member board of directors which consists of the duly elected officials of the Grand Portage and Bois Forte Tribal Councils.

TMM has not initiated any formal discussions with Native American stakeholder groups or commenced formal consultations with tribal governments at this Project stage.

20.8.4 Boundary Waters Canoe Area Wilderness

The Boundary Waters Canoe Area Wilderness is part of the National Wilderness Preservation System and is managed by the USFS. The Boundary Waters Canoe Area Wilderness is located in the northern third of the SNF in northeastern Minnesota. It extends nearly 150 miles along the International Boundary adjacent to Canada's Quetico Provincial Park and is bordered on the west by Voyageurs National Park. The Boundary Waters Canoe Area Wilderness is the largest designated wilderness in the eastern U.S. and is the only lake-land wilderness of its kind and size in the National Wilderness Preservation System. Visitors are able to canoe, hike, portage, and camp within the Boundary Waters Canoe Area Wilderness.

Protection of the Boundary Waters Canoe Area Wilderness is the one of the key environmental issues raised in the current public debates over the future of copper–nickel mining in Minnesota.

20.8.5 Public Opinion

TMM monitors public issue and public opinion surveys completed by other Minnesota business organizations that included survey questions on mining issues focusing primarily on copper–nickel mining in Minnesota.

20.8.6 Communications

Duluth recognizes that maintaining an environment of support for the Project within Minnesota will require TMM to have a consistent and intense commitment and investment to stakeholder engagement, public official outreach and general public relations and stakeholder communication.

In conducting public affairs and government relations activities, Duluth and TMM must be cognizant of, and comply with, legal requirements in several relevant areas (e.g. state and federal government public information disclosure, lobbying regulations, etc.).

The various strategic plans developed by TMM during the course of Project development in the areas of Project communication, community/stakeholder engagement, and state and federal government affairs have to date generally complied with TMM’s foundational values and principles.

TMM supports local communities in the Project region. The company bases its community involvement on five "pillars" of community support:

- Health and wellness
- Economic development

- Education
- Environmental stewardship
- Safety.

Community involvement and giving activities are overseen by a TMM employee committee.

A broader and more extensive "Community Investment Plan" may be developed later in the MPO development phase, when greater Project detail would be known.

20.9 Discussion on Risks to Mineral Resources and Mineral Reserves

Environmental risks that could impact TMM's ability to develop the Project at a PFS stage of project knowledge include the following.

20.9.1 Water Rights

Transferring water from one watershed to another is sometimes suggested as a method to supply water, and TMM has considered such an option for operation of the Project at this PFS-level of evaluation. However, inter-basin water transfers in the State can be problematic. Eight states including Minnesota and two Canadian provinces, all surrounding the Great Lakes, have a charter that addresses notification and consultation on requests for interbasin transfers out of the Great Lakes Basin. The State of Minnesota has expressed concerns about such interbasin water transfers (DNR, 2000). Instead, the state supports "the sustainable use of existing resources and encourages water users to live within the means of their naturally occurring water supply" (DNR, 2000).

There is some environmental risk associated with uncertainty regarding the water rights for the water supply necessary for operation of the TMM Project. Should TMM not secure the appropriate permits for transfer of water between the basins, it may be necessary to modify the locations and/or operation of the various Project components, in order to comply with any inter-basin transfer prohibitions.

20.9.2 Refusal of Permits:

There is some environmental risk associated with uncertainty regarding permits to construct and operate the Project. While TMM has been proactive in addressing likely areas of concern that the various permitting authorities may have, the issuance of necessary permits is not guaranteed. The permitting authorities could impose restrictions on the construction and/or operation of the Project that could result in substantial alterations to the proposed Project.

20.10 Comments on Section 20

In light of the stringent regulatory requirements, numerous environmental studies, and close public scrutiny of the Project, there are a number of areas that will likely require further attention, and could affect the timeline for bringing the project to construction and production. These are discussed in some detail in Section 24.2.12, and include:

- Discussions with regulatory agencies on the suitability of TMM collected baseline environmental data for inclusion in the EIS environmental studies
- The possibility of inter-basin transfer of water, and the appropriate authorizations from regulatory agencies that may be necessary
- Proposed hydrogeological study plan
- The need for further study of the Project water balance, and the possibility of the requirement to obtain a permit for the discharge of treated process water in the event the Project cannot be shown to be zero-discharge
- Tailings, waste rock, and paste backfill characterization
- Long lead time for draft EIS studies
- Comments on the draft EIS
- Permit and EIS appeals.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

21.1.1 Basis of Estimate

The capital cost estimate for the Project was developed by TMM's Independent Engineer, with input from consultants for specific areas as indicated in Table 21-1.

TMM's Independent Engineer produced the estimate for the process plant and concentrate filtration plant. Barr produced the estimate for the surface infrastructure. Golder produced the estimate for the tailings storage facilities and paste backfill plants. Estimates for the underground mine capital and operating costs were developed by SRK. AMEC produced the estimate for underground mine infrastructure.

All estimates were produced under the direction of TMM's Independent Engineer.

TMM's Independent Engineer compiled the estimate from detailed estimates provided by all consultants. All consultants participated in the overall execution strategies. Regular meetings were held with the consultants during the estimate development to establish a clear understanding of the scope and how the direct costs relate to their reported indirect costs. The costs were reported by TMM's Independent Engineer at a prefeasibility level of accuracy where the estimate accuracy range is defined as +25%/-20% including contingency and are consistent with an AACE International (formerly Association for the Advancement of Cost Engineering or AACE) Class 4 Estimate. Vendors' pricing was obtained for long-lead items, mining consumables, and mobile equipment, and the consultants actively participated in obtaining appropriate-level budget quotations. Budget quotes and in-house historical cost data were applied by the consultants to the prefeasibility study estimate where applicable and appropriate.

Costs in the TMM prefeasibility study were reflective of Q3 2013 market conditions. TMM's Independent Engineer and its consultants assessed overall construction personnel requirements, material availability and logistics, work methods, and risks. Escalation was excluded from all estimates.

Table 21-1: Consultants Contributing to TMM Capital Cost Estimates

Area	Consultant Responsible
Process plant	TMM Independent Engineer
Surface infrastructure	Barr and Associates
Paste backfill plant and facilities	Golder
Tailings storage facility	Golder
Underground infrastructure and facilities	AMEC
Underground Mining Equipment	AMEC
Pre-production and mine development	SRK

Note: TMM Independent Engineer noted in this table was unable to be identified under the terms of their contract with Twin Metals Minnesota

21.1.2 Labor Assumptions

Approximately 5.54 M direct and indirect man-hours were estimated to be required for construction. Mining man-hours are not included. Duration for the capital cost portion of this project is three years. Work rotation assumptions for on-site personnel were a two weeks on/one week off rotation with a standard workweek comprising ten hours per day and six days per week. Crew rotations will be staggered such that work can be performed on a continuous basis.

Craft labor rates were established based on TMM's Independent Engineer's labor surveys of prevailing wage rates for construction activities in the vicinity of the jobsite and provided to all consultants. AMEC reviewed the rates build-up and considered that they are appropriate for this level of study.

Camp and catering costs were not considered for this Project. A per diem allowance was included in the indirect cost portion of the estimate.

Travel costs were based on labor being sourced in-state (25%), regionally (37.5%) and inter-state (37.5%). A mileage allowance for travel was included in the indirect cost portion of the estimate.

Productivity factors were developed for each discipline and applied to the base man-hour units where applicable. All the consultants were responsible for their base unit hours and development of productivity factors for their portion of the Project.

21.1.3 Material Costs

Average all-inclusive construction equipment rates were estimated by discipline by each consultant.

All permanent material, including bulk costs (i.e., concrete, steel, cables etc.), are covered under material costs. Bulk material costs were based on prices, budget quotes, and historical in-house sources from all of the engineering contractors. The cost of temporary and consumable materials used during construction of direct works (i.e., drill steels, formwork, welding consumables, temporary supports, etc.) was also included in the material costs.

For all major equipment, budget quotations were obtained. These vendor quotations were reviewed for completeness and technical adequacy, and where vendors' quotations were incomplete an appropriate factor was applied before the final costs were incorporated into the estimate.

Freight costs for Project material were included by the consultants as part of their indirect costs. Generally the allowance was 4%. All sales taxes and import duties were excluded.

21.1.4 Contingency

The original capital cost estimate included an average contingency allocation of 15.3%. This was subsequently been increased to 18%.

21.1.5 Engineering, Procurement, and Contract Management (EPCM)

Engineering, procurement, and contract management (EPCM) costs for initial capital are included in the estimate by TMM's Independent Engineer and by AMEC. TMM's Independent Engineer included the EPCM services cost at 15% of direct costs. This would cover the process plant, surface infrastructure (Barr), paste backfill plants (Golder) and tailings management (Golder).

AMEC's cost for engineering and procurement is based on 8% of direct costs for capital expenditure. AMEC's contract management costs were estimated according to staffing requirements using in-house data and were based on 60 hours per work week.

SRK did not include EPCM as a separate allowance. The costs for mining EPCM are built up in their estimate and the general and administrative (G&A) estimate.

Table 21-2 summarizes the EPCM allocations.

21.1.6 Owner's Costs

Owner's costs were developed by TMM and reviewed by AMEC. Costs included provision for Project technical and support staff and supplies, Project Owner's Team, site operating and maintenance costs. Also included were costs related to community relations and for compliance to local and national governing and regulatory agencies.

Table 21-2: EPCM on Initial Capital

Consultant	Area of Work	Scope	Value (US$ M)
TMM Independent Engineer	Process plant	EPCM on capital costs	$180.6
Barr	Surface infrastructure	EPCM on capital costs	
Golder	Paste backfill plants	EPCM on capital costs	
Golder	Tailings management	EPCM on capital costs	
AMEC	Underground infrastructure and facilities	Engineering and procurement on capital costs	$19.0
AMEC	Underground infrastructure and facilities	Contract management on capital costs	$21.6
SRK	Pre-production mining and development	EPCM on capital costs	inc in costs
Overall	*Project – Capital Costs*	*EPCM Initial Capital*	*$221.2*

21.1.7 Sustaining Capital

Sustaining capital cost estimate was compiled by TMM's Independent Engineer and was based on information provided by the consultants, indicated in Table 21-3, as applicable to their scope, for the LOM. The sustaining capital costs were assumed to be executed by both independent contractors and the Owner. Cost areas considered in the estimate included:

- Mine development over the LOM
- Replacements of the mining fleet and major components over the LOM
- Expansion of underground infrastructure and facilities
- Development of the Maturi Southwest deposit (Year 16 of the mine life)
- Construction of additional lifts to expand capacity of the TSF
- Maintenance and expansion of paste backfill distribution system
- Two additional paste backfill plants
- Reclamation costs at the end of mine life.

21.1.8 AMEC Review of Capital Cost Estimate

AMEC performed a detailed estimate review of the capital cost estimate developed by TMM's Independent Engineer in June–July 2014. As a result, some changes were made by AMEC to the estimate.

The capital cost estimate that was provided to AMEC in June 2014 was in Excel format. High-level cost analyses were performed on major cost centers to determine if the costs were reasonable when compared to AMEC in-house data. Detailed line by-line estimate reviews were performed where questions arose.

Table 21-3: Consultants Contributing to TMM Sustaining Cost Estimates

Area	Consultant Responsible
Paste backfill plant and facilities	Golder as modified by TMM's Independent Engineer
Tailings storage facility	Golder as modified by TMM's Independent Engineer
Underground infrastructure and facilities	AMEC
Underground Mining Equipment	AMEC
Pre-production and mine development	SRK

AMEC noted the following from the review:

- The work breakdown structure (WBS) was sufficient to break down the costs into reasonable portions
- The construction labor rates provided by TMM's Independent Engineer to all consultants for their portions of the estimate have been built-up in a reasonable manner for a prefeasibility study and are in an acceptable range of costs per hour
- TMM's Independent Engineer supplied common foreign exchange rates to be used by all consultants. It was noted that the exchange has changed since the rates were set. The impacts are not considered to be significant since non-US costs have reduced in price slightly
- The application of design development allowances, applied by TMM's Independent Engineer, to the process plant is reasonable. No other consultants included this allowance. AMEC's opinion is that design development allowances are not necessary for a prefeasibility level estimate
- The review of TSF estimate indicated to AMEC that the earthwork costs were underestimated. Labor productivity and equipment costs were significantly lower than AMEC's review for the same estimate quantities. The cost impact of these areas is $63.84 M. A total of $57.1 M is allocated to construction equipment costs
- As a result of the above review, AMEC also found a difference in the excavation costs for the two paste backfill plants included in the original estimate. The cost impact for this is $1.3 M
- Earthworks in other areas of the estimate were acceptable for a prefeasibility study.
- Concrete pricing for the project was provided to all consultants by TMM's Independent Engineer. The costs were produced by Barr from local sources that have worked with Barr in past projects. Consultants were responsible for building up their in-place costs using this common concrete supply price
- The review of the process plant estimate prepared by TMM's Independent Engineer indicated to AMEC that the concrete work was underestimated. In discussion with TMM's Independent Engineer regarding the estimating philosophy

for concrete work, AMEC was told that a productivity factor of 1 was used on their base productivity units. AMEC reviews with the same quantities indicate a cost impact of $16.23 M. This reflects a variance of 24% above TMM's Independent Engineer's productivity estimate. AMEC's evaluation of the productivity factor was 25%

- Aggregate supply cost for the project was provided to all consultants by TMM's Independent Engineer. The costs were produced by Barr
- The material pricing methodology is sufficient for a prefeasibility study
- Building costs were developed mainly from consultants' in-house data with the exception of obtaining a quotation for some specific structures. This approach is acceptable for a prefeasibility study
- Common bulk pricing for primary structural steel was provided by TMM's Independent Engineer and was developed from regional supplier quotes. The costs included supply, detailing, paint, fabrication and freight to site. The material pricing and installation man-hours methodology is sufficient for a prefeasibility study
- The estimating methodology for mechanical equipment, piping, electrical and instrumentation is reasonable for a prefeasibility study
- The underground infrastructure and facilities estimate costs were provided by AMEC from recent projects, budget quotations or in-house data. Mass rock excavation costs were from TMM, which provided unit costs. The material pricing and installation man-hours methodology is reasonable for a prefeasibility study
- Estimates for the primary mine access declines were provided in a proposal format from a mining contractor
- The pre-production mining estimate included development and procurement of the mobile equipment fleet. This is extracted from the complete mining costs estimate, which includes all mine development, production, maintenance, and infrastructure operating costs
- The Project indirect costs for the process plant, TSF, paste plants, and surface infrastructure were estimated by TMM's Independent Engineer. Indirect costs for the underground infrastructure were estimated by AMEC. All indirect costs except for mining were built from individual items, and are not based on a percentage of direct costs. Indirect costs for mining are included in unit costs and the direct cost estimate

- The percentage of indirect costs to direct costs is within an expected range for a project of this magnitude. The review found the indirect estimate methodology generally to be very good for a prefeasibility-level study
- The Owner's costs include typical cost centers such as personnel, corporate costs, environmental programs and insurance. The Owner's costs are approximately 5.1% of the Project costs, not including pre-production mining and mining equipment. AMEC's in-house benchmarking suggests Owner's costs could be in the order of 5–15% of direct costs. AMEC assessed the Owners Costs as being reasonable for a prefeasibility-level estimate
- Process plant and site infrastructure contingencies were set at 15%. Paste plant and TSF contingencies were set at 20%. Pre-production and production mining had a 20% allocation for the mobile equipment procurement, and no contingency was applied to the pre-production development work. Underground infrastructure and facilities had 22% contingency applied. The overall average contingency across all areas is 15.3%
- AMEC suggests that a 15.3% contingency is low for a project of this scope. AMEC recommends, based on its analysis, that a contingency of 18% be used. AMEC applied contingency to mine capital development costs. The cost impact of the change in contingency applied to AMEC's assessment of the prefeasibility study is $74.78 M.

21.1.9 Capital Cost Summary

AMEC's restated capital cost estimate is shown in Table 21-4. This is a net increase in initial capital of $182.5 M over that reported in the TMM PFS. The work areas where cost increases were included by AMEC are as shown in Table 21-5.

21.1.10 AMEC Review of Sustaining Capital Estimate

AMEC performed a detailed estimate review of the sustaining capital cost estimate compiled by TMM's Independent Engineer. As with the capital cost estimate, some changes were subsequently made by AMEC to the estimate.

AMEC reviewed the sustaining capital cost estimate produced by Golder and modified by TMM's Independent Engineer for the TSF and two additional paste backfill plants. Similar differences were found as in the capital cost estimate: labor productivity and equipment costs were significantly lower than AMEC's estimate for the earthwork using the same quantities. The cost impact of the assessment is $98.41 M.

Table 21-4: Initial Capital Cost Estimate (restated)

Description	US$ (M)
Mine	794.0
Process	955.6
Tailings and paste	546.6
Surface infrastructure	378.7
Owners costs	100.0
Total initial capital	*2,774.9*

Table 21-5: Cost Estimate Increases by Area

Description	Increase (US$ M)
Mine	66.0
Process	-2.1
Tailings and paste	133.1
Surface infrastructure	-14.1
Owners costs	0
Total initial capital increase	*183.0*

Sustaining capital costs for the underground infrastructure and facilities were estimated by AMEC. Overall scope for the sustaining capital costs are the same as the capital cost estimate. Costs include expansion of facilities concurrent with mining activities and the construction of the Maturi Southwest operation, beginning in Year 16. Labor and material costs are applied as in the capital cost estimate. Sustaining capital costs for the process plant are included in the operating costs.

Sustaining capital costs for the mine includes major development, raises to surface, mobile equipment rebuild and replacement. This is extracted from the complete mining costs estimate, which includes all mine development, production, maintenance, and infrastructure operating costs. The mining cost estimate is discussed separately under Section 21.2. AMEC considers the estimating method reasonable for a prefeasibility study.

AMEC noted that reclamation costs were not included in the PFS cost estimate. AMEC included reclamation costs in the sustaining capital cost estimate.

AMEC's restated sustaining capital cost estimate is shown in Table 21-6. This reflects an increase in sustaining capital of $424.3 M over the PFS estimate.

Table 21-6: Sustaining Capital Cost Estimate (restated)

Description	US$ (millions)
Mine	1,800.4
Tailings and paste	835.2
Total sustaining capital	*2,635.6*

Note: Reclamation costs are included in the mining category; surface infrastructure sustaining capital costs are included in the tailings category. Sustaining capital costs for the process plant and concentrator are included in the process operating cost estimate.

21.2 AMEC Review of Mining Cost Estimate

AMEC performed a detailed estimate review of the mining cost estimate presented in the PFS. During the course of the review, a number of items were identified that required the mining costs to be restated for inclusion in the Technical Report.

The mining cost estimate was developed by SRK and included both capital and operating costs. AMEC and TMM provided input and recommendations to SRK. Permanent development such as ramps, declines, ventilation raises, excavations for permanent infrastructure are capitalized; development within the orebody for production is reported as operating cost, as are production activities themselves. Infrastructure operating costs are reported as operating costs after the commencement of production.

The mining costs for the Project were developed using a bottom-up first principles method. All direct and indirect mining costs were calculated from this method using mine activity performance and a cost modeling process. Once the productivities and cost basis were established, a process of verification and validation was undertaken using benchmark costs from similar operations and input from TMM. Unit costs were generated from budgetary quotations from industry suppliers. Labor and utility costs were provided by TMM.

21.3 Operating Cost Estimates

21.3.1 Labor Rates

Labor rates for operations personnel have been derived from current labor agreements in the Minnesota Iron Range mining district (Table 21-7). These rates have been adjusted to include appropriate burdens, planned overtime (where needed), and vacation pay. The rates in Table 21-7 have not been adjusted to include allowances for items such as absenteeism, or sickness. Each operating cost area subsequently included additional personnel to account for items such as vacation, sickness, absenteeism, and training (VSAT) as applicable.

Table 21-7: Labor Rates

Category	Base $/Hr	Base $/Yr (incl. Vacation)	Total Burdened Annual Rate	Example Type
P-1	$86.54	$180,000	$207,549	Mine Manger
P-2	$54.81	$114,000	$137,969	Senior Department Heads
P-3	$36.54	$76,000	$95,442	Senior Technical and Supervisors
P-4	$26.44	$55,000	$71,135	Support Technical
P-5	$34.13	$71,000	$89,407	Tradesmen and Miner A (Drill/Bolt)
P-6	$31.25	$65,000	$83,755	Construction and Miner B
P-7	$28.85	$60,000	$77,120	Logistics and Miner C
P-8	$21.77	$45,275	$63,219	Mine Helpers
P-9	$19.23	$40,000	$52,080	Security

21.3.2 Mine Operating Costs

The operating cost model includes allocations for operating and maintenance labor, equipment operating costs, material and supply costs and a 10% allowance.

The operating cost summary is shown in Table 21-8 and the following sub-sections discuss items included in each section of the operating cost. The yearly composition of production from the various mining methods is shown graphically in Figure 21-1.

Approximately 50% of the ore will be mined by long-hole stoping, 23% by post-pillar cut-and-fill in the upper area of the deposit and 27% by post-pillar cut-and-fill in the lower area of the deposit.

21.3.2.1 Post-Pillar Cut-And-Fill

The operating cost for post-pillar cut-and-fill production includes mining both slots (transverse rooms) and cross-cuts between slots. Development of select footwall laterals to be open for the life of mine is capitalized.

Operating costs for post pillar cut and fill includes:

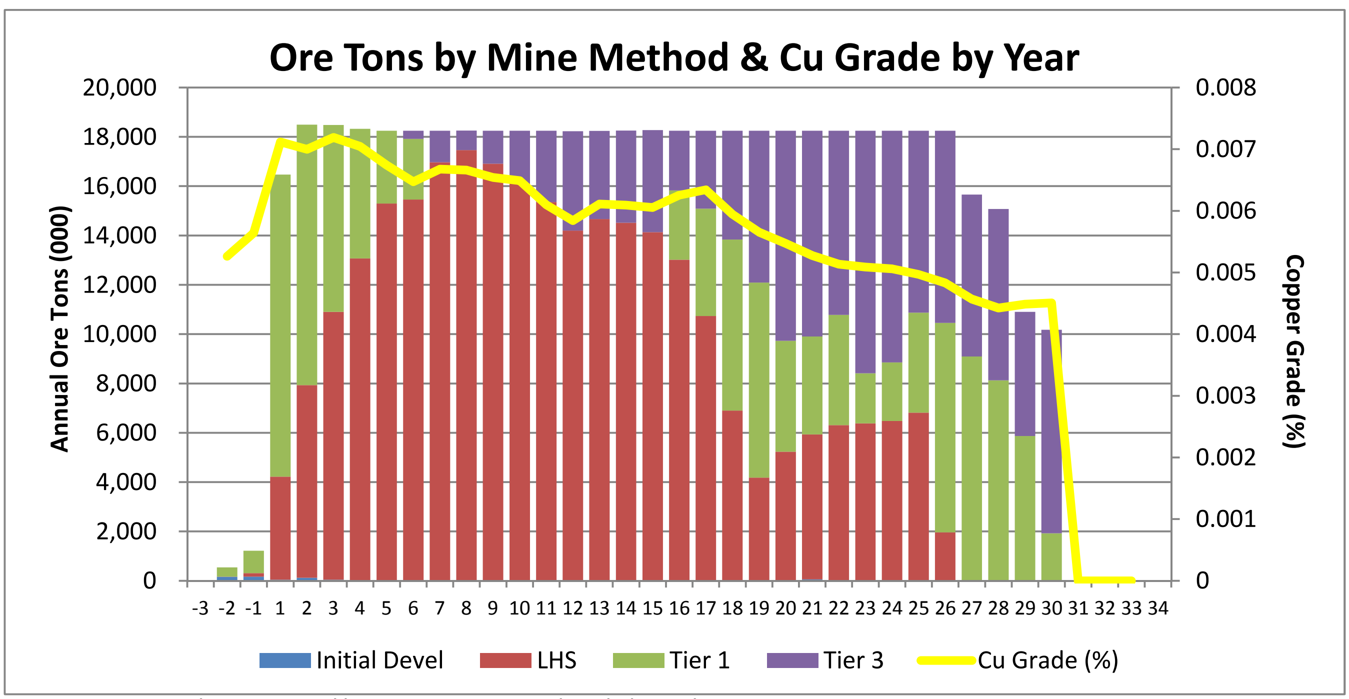
- Labor: labor quantity and cost for mining slots and cross-cuts
- Equipment cost for mining slots and cross-cuts: includes drilling, blasting, mucking, ground support, utilities, etc
- Materials cost for mining slots and cross-cuts: includes service items such as mine service water, dewatering, ventilation, electrical, and roadbeds. Also includes supply costs such as drilling consumables, explosives and blasting supplies, ground support, backfill lines, etc. The cross-cuts are not furnished with services and have reduced ground support requirements

Table 21-8: Underground Mine Operating Cost Summary

Description	Mining Operating Cost (US$ x 1,000)	Unit Cost* (US$)	Equivalent Unit Cost* (US$)	Units
Development				
Ground support multiple headings (4); 2 bolters	825,482	$704	$1.56	$/t-RoM
Production				
Post-pillar cut-and-fill (Tier 1) - 4 headings	451,755	$4.07	$0.85	$/t-RoM
Post-pillar cut-and-fill (Tier 3)	1,261,679	$10.87	$2.40	$/t-RoM
Long-hole stoping	714,283	$2.61	$1.35	$/t-RoM
Raises - Slots (Machines Roger)	5,660	$545	$0.01	$/t-RoM
Truck haulage	1,100,347	$2.00	$2.09	$/t-RoM
Services				
Mine construction backfill bulkheads	193,361	$0.37	$0.37	$/t-RoM
Mine construction reconditioning	170,183	$0.32	$0.32	$/t-RoM
Mine services and support	393,491	$0.75	$0.74	$/t-RoM
Mine technical services and management	109,431	$0.21	$0.21	$/t-RoM
Mine maintenance	498,344	$0.95	$0.94	$/t-RoM
AMEC mine dewatering	51,286	$0.10	$0.10	$/t-RoM
AMEC ventilation and heating	240,775	$0.46	$0.46	$/t-RoM
AMEC crushing and conveying	346,236	$0.66	$0.66	$/t-RoM
AMEC transport of people, equipment, materials and misc.	253,061	$0.48	$0.48	$/t-RoM
Total Operating Costs	*$6,615,376*		*$12.56*	*$/t-RoM*

Note: *Unit cost is based on tons for the particular item (i.e. Tier 1 cost divided by Tier 1 tons). Equivalent unit cost is based on overall ROM tons (i.e. Tier 1 cost divided by LOM ore tons). Totals may not sum due to rounding.

Figure 21-1: Annual Ore Tons by Mine Method



Note: Figure prepared by SRK, 2014. LHS = long-hole stoping.

- Preparation for, and placement of, backfill is not included in the mine operating costs.

Direct unit costs for Tier 3 post-pillar cut-and-fill are higher than the Tier 1 post-pillar cut-and-fill areas largely due to smaller opening sizes in Tier 3 to satisfy the geomechanical requirements at deeper mining depths. Based on cross-sectional area, the opening size in Tier 3 is less than 30% of the size of the Tier 1 slots, meaning less tonnage is produced per cycle of advance.

Tier 3 mining costs are applied to all post-pillar cut-and-fill mining in Maturi Southwest.

21.3.2.2 Long-Hole Stoping

Long-hole stoping costs are summarized into the following items:

- Slot raise development: includes drilling of the slot using an ITH drill with a Machines Roger reamer, blasting of the slot, and mucking the blasted material to a remuck bay. Includes equipment, labor, and materials costs
- Stope drilling: drilling of the entire stope using ITH drills. Includes equipment, labor, and material costs
- Stope blasting and mucking: blasting of the entire stope, initially blasting partial rings and reloading holes and, as the stope is developed, blasting full rings. Also, mucking the stope from the extraction drift. Includes equipment, labor and materials costs
- Preparation for, and placement of, backfill is not included in the mine operating costs
- Secondary breakage cost with blockholer at the stope.

Long-hole stopes in Maturi Southwest attract the same operating costs as detailed for Maturi.

21.3.2.3 Slot Raises (Ventilation)

Short ventilation raises between levels will be developed using a slot raise methodology. The cost for Machines Roger slot raises is based on a 30 inch bored raise, enlarged to a 21 ft x 21 ft opening. The cost includes drilling, blasting, and mucking and all labor, equipment, and materials costs.

21.3.2.4 Underground Haulage

Underground truck haulage includes hauling development waste, development ore, and production ore. Haulage distances from the production schedule were used for cost calculations. Ore is hauled to the underground crusher, waste to stopes as backfill. Costs include labor, equipment and materials. Reloading of long-hole stope ore is included in this cost.

21.3.2.5 Underground Mine Services and Support

Underground mine services and support includes:

- Construction of bulkheads for backfill
- Reconditioning/rehabilitation of development openings and waste rehandling
- Mine services and support categories, including construction, dewatering, ventilation, and road service crews
- Mine management includes general supervisors and foremen in development, mining and maintenance supervision and planning for mobile & fixed plant. Mine management and technical services are included in the G&A costs
- Mine maintenance – mobile and fixed Maintenance of equipment, power, communications systems, and infrastructure
- Mine logistics (transportation of personnel, equipment and materials)
- Labor and equipment for delivery of mining consumables and supplies.

21.3.2.6 Crushing and Conveying

Crushing and conveying operating costs include crushing and conveying from both underground crushers and the surface crusher based on the yearly split from the production schedule. The cost of surface crushing 31.8 Mst through the crusher has an estimated cost of $2.39/st. The majority of the crushed material, 491.3 Mst, is handled underground at an estimated cost of $0.34/st. Power costs are reported separately. Total operating costs, including power, are $0.55/st.

21.3.2.7 Ventilation and Heating

Ventilation and heating operating costs include operation (power cost) and maintenance of main and auxiliary fans, ventilation control structures, and heating intake air to 20°F.

21.3.2.8 Dewatering

Dewatering operating costs include operation (power cost) and maintenance of main and collection sumps and pump stations.

21.3.2.9 Labor

Project standard labor classifications and rates used are discussed in Section 21.3.1. The majority of technical services and senior management personnel are included in G&A. All other labor costs are included within the various operating cost categories. The labor cost makes up 23% of the total underground mine operating cost.

21.3.2.10 Restated Mine Costs

During the review of the mine cost model, AMEC made the following observations:

- The PFS cost estimate did not include any allowance for on-shift underground supervision. AMEC and SRK have included supervision for all crews on all shifts
- Ground support plans used in the PFS estimate did not agree with the approved ground support plan for primary development areas. The model was revised to ensure that the estimate agreed with the approved plan. (The impact of this is mostly to the capital and sustaining capital estimates)
- Discrepancies in the PFS model relating to load factors and double-counting of efficiency factors were corrected
- There were no allowances on the production truck haulage calculations for traffic interference and delays. These were included in the restated model
- Materials and labor allowances were increased to a minimum of 10% in all areas.

As a result of these and other adjustments to the mine operating cost model, the net increase in mine operating costs is $0.68/st when compared to the PFS estimate.

21.3.3 Process Operating Costs

The process plant operating cost estimate has a targeted accuracy of ±25%. However, the estimating methodology is in accordance with a Class 4 estimate in ADS Minimum Standard ADS_MS_013 wherein it states that by the specified methods one can attain a typical accuracy range of ±10% to ±15% if based on known operations or ±15% to ±20% if the facility includes novel technology. The operating costs for all surface facilities have been based on similar, currently-operating facilities.

Within the logical groupings, operating costs have been calculated by commodity including: operating labor, power, fuel, reagents, facility specific consumables (e.g., grinding media in the concentrator), maintenance labor, maintenance supplies, and other. Labor rates for supervisors have been determined from recent labor surveys of executive and supervisory personnel. Labor rates for operations personnel have been derived from current labor agreements in the Minnesota Iron Range mining district.

These rates have been adjusted to include appropriate burdens and bonuses, but do not include account for VSAT.

Unit costs for fuels are based on current experience at the Project site. Costs for major reagents and consumables were obtained from informal quotes on a delivered basis. It is assumed that raw water would be drawn from a source with no purchase cost.

The following items are excluded from the operating cost estimate:

- Amortization
- Depreciation
- Taxes
- Escalation
- Research and development
- Fees
- Financing of inventories
- Currency fluctuations
- Sustaining capital
- Royalties.

21.3.3.1 Labor

Labor rates for supervisors have been determined by TMM from recent labor surveys of executive and supervisory personnel. Labor rates for operations personnel have been derived from current labor agreements in the Minnesota Iron Range mining district. These rates do not account for vacation, sickness, absenteeism, and VSAT.

For the operational labor force for the surface process facilities, staffing levels were determined by position title within the facilities and then placed into a category. As operation of the process facilities does not require continuous intercession by operators, sufficient redundancy between positions has been included. An allowance for additional staff during an annual shutdown of the concentrator for major maintenance such as relining of the grinding mills was also included based upon 100 additional persons per shift for three days at a Category 5 level. The requirement for operating personnel from different surface process facilities was also reviewed, as to proximity of work place, in order to determine whether workforce positions could be combined. Staffing for supervisory positions in the surface process facilities was determined by position, taking into account that all shifts would be covered. The annual cost for operating labor was calculated by multiplying the number of personnel in each of the nine categories by the annual all in rates for each of these categories.

With respect to maintenance labor workforce in the surface process facilities, staffing levels were determined by position and then placed into a category, taking into account the number and types of equipment to be maintained. As non-mobile equipment maintenance has a high percentage of planned maintenance, these positions are scheduled only for day shifts with the opportunity for call out on night shift, as required. No VSAT allowance was added. As many of the fixed equipment items require similar maintenance capabilities, the requirement for maintenance personnel from different surface process facilities was also reviewed in order to determine whether workforce positions could be combined. Staffing for supervisory positions in the surface maintenance departments was determined by position, taking into account that all shifts would be covered. The annual cost for maintenance labor was calculated by multiplying the number of personnel in each category by the annual rates.

Process workforce operating costs are as summarized in Table 21-9.

Surface labor costs total $10.05 M or $0.55/st/a.

21.3.3.2 Power

Electric power consumption was determined from the average connected load for each facility operating at nominal rates associated with the full ore production rate of 50 kst/d ore. This value was adjusted over the projected mine life to take into account changes in mine configuration. The annual cost for electricity was calculated by multiplying the average consumption by the power cost shown in Table 21-10.

Total power consumption of process facilities including filtration is estimated to be 19,852 kWh/st, and a unit power cost of $48.90 per MWH: this equates to $17.716 M per year.

21.3.3.3 Reagents and Consumables

Reagents will include lime, flotation reagents (primary collector, secondary collector, depressant, frother, $NaSO_3$) and flocculants. Consumption rates for each reagent were calculated based on throughput, feed grade, recovery, metallurgical testwork and benchmarking. Unit costs for major reagents and consumables were obtained from informal quotes on a delivered basis.

The annual reagent cost is estimated to be $16.5 M or $0.90/st (Table 21-11).

Table 21-9: Process Operating Costs – Surface Operations Workforce

Area	Category	Position	Number	Salary (US$)	Salary Cost (US$)
Surface Operations Workforce	2	Superintendent	1	137,975	137,975
	3	Supervisor	5	95,450	477,250
	4	Technical personnel	1	71,150	71,150
	6	Process operators	16	83,734	1,339,744
	7	Operators assistant	8	77,118	616,944
		Shutdown personnel			294,000
	Sub-Total		*31*		*2,937,063*
Surface Maintenance Workforce	2	Maintenance superintendent	1	137,975	137,975
	3	Supervisor	6	95,450	572,700
	4	Instrument analyst	1	71,150	71,150
	5	Skilled millwrights	25	89,391	2,234,775
	5	Electricians	8	89,391	715,128
	6	Repairmen	11	83,734	921,074
	8	Boiler welder	4	63,218	252,872
	8	Control systems	4	63,218	252,872
	Sub-Total		*60*		*5,158,546*
Surface Filter Plant Workforce	3	Filter plant supervisor	4	95,450	381,800
	6	Filter plant operator	4	83,374	334,936
	7	Filter plant assistant	16	77,118	1,233,888
	Sub-Total		*24*		*1,950,624*

Table 21-10: Surface Power Costs

Area Code	Area	Consumption (MW)	MWH/a	kWh/st
220	Ore transport to concentrator	0.55	4,433	0.243
240	Coarse ore reclaim	0.72	5,803	0.318
310	SAG mill	15.58	125,562	6.880
310	Ball mill	13.94	112,345	6.156
310	Comminution and flotation feed pumps	2.80	22,566	1.236
320	Pebble collection	0.11	887	0.049
330	Cu flotation and regrind	3.5	28,207	1.546
332	Ni flotation and regrind	4.4	35,460	1.943
340	Cu concentrate and thickening	0.13	1,048	0.057
342	Ni concentrate and thickening	0.10	806	0.044
392	Reagents and supply install	0.38	3,062	0.168
393	Air compressor plant	0.64	5,158	0.283
394	Concentrator service facilities	0.23	1,854	0.102
510	Tailings thickening	0.11	887	0.049
	Filtration	1.76	14,220	0.779
Total		*44.95*	*362,297*	*19.852*

Table 21-11: Surface Reagent Costs

Chemical	g/mt	US$/kg	US$/st	US$/a
3418A	8	4.20	0.030	556,286
SIPX	105	2.10	0.200	3,650,625
TETA	60	7.10	0.386	7,052,908
MIBC	30	3.60	0.098	1,788,061
Sodium sulfite	60	0.50	0.027	496,684
Quicklime	1000	0.12	0.104	1,903,954
Flocculant	20	3.20	0.057	1,038,400
Total			*0.903*	*16,486,917*

Note units use metric tonnes.

21.3.3.4 Grinding Media and Liners

Grinding media and liners includes liners and ball requirements for crushers and mills. Grinding media and liner requirements are estimated to total $22.02 M/a or $1.21/st.

21.3.3.5 Operating Supplies

Operating supplies include considerations of wear items costs (hydrocyclones and screens), fuel costs for the process plant, filter cloth costs, and operating supplies costs for the tailings and water reclaim.

Maintenance materials have been estimated by factoring at US$0.25/st or US$4.563 M/a, which would cover replacement wear items such as the hydrocyclones, screens, wear plates.

An allowance has been made for heating of $10/st or $1.825 M/a. An allowance of $0.01/st or $0.216 M/a has been estimated for the replacement of filter cloths.

Total annual operating costs for the concentrator and filter section are $72.87 M or $3.99/st (Table 21-12).

21.3.4 Infrastructure Operating Costs

Surface infrastructure operating costs were built up from first principals by Barr and Golder. All are based on common labor, power, and material unit costs. Costs include operating and maintenance labor and materials, reagents (including cement and fly ash), equipment operating costs, and power costs.

Included in the infrastructure operating costs are tailings transport (slurry lines) to the TSF and paste plants, TSF operation, paste plant operations, underground paste distribution system, surface water management, and general site operations costs.

As with other areas, AMEC reviewed the infrastructure operating costs presented in the PFS. During the review, it was determined that ongoing operating costs for the underground paste distribution system had been omitted from the PFS cost model.

Table 21-12: Operating Cost Estimate (by area)

Area	US$/a	US$/st milled
Concentrator		
Operations labor	2,937,063	0.16
Maintenance labor	5,158,546	0.28
Power	17,020,958	0.93
Grinding media and liners	22,016,258	1.21
Reagents	16,486,917	0.90
Maintenance materials	4,563,000	0.25
Other (heating)	1,825,000	0.10
Concentrator Sub-Total	*70,007,742*	*3.84*
Concentrate Filtration		
Operations labor	1,950,624	0.11
Maintenance labor	0	0
Power	695,300	0.04
Operating supplies	216,000	0.01
Maintenance materials	0	0
Concentrate Filtration Sub-Total	*2,862,004*	*0.16*
Total	***72,869,746***	***3.99***

Note: Concentrate filtration maintenance labor and materials are included in the Concentrator costs.

As a result of this and other adjustments to the infrastructure operating cost model, the net increase in infrastructure operating costs is $0.76/st.

The recast infrastructure operating costs are shown in Table 21-13.

21.3.5 General and Administrative Operating Costs

G&A costs were built up from first principles, and were done in much greater detail than is normally expected for a PFS. G&A allocations include management, site services, administrative support functions, safety department, and the technical services group.

The technical services group will include mining and process engineers, mine geologists, and operating costs for those teams, including delineation drilling and sample and contract assay costs.

During AMEC's review of G&A operating costs, it was determined that insufficient allowance had been made for grade control and metallurgical sampling. After correcting for this, the recast G&A operating costs showed an increase of $0.13/st over the PFS estimate.

Annual recast G&A costs for the operation at steady state are shown in Table 21-14.

Table 21-13: Infrastructure Operating Costs (restated)

Area	Annual Cost (US$ x 1,000)	US$/st
Tailings	12,773.2	$0.70
Backfill	30932.2	$1.69
Process water	767.4	$0.04
Concentrate transport	365.0	$0.02
Surface infrastructure and utilities operating costs	571.3	$0.03
TOTAL	*45,409.0*	*$2.49*

Table 21-14: G&A Operating Costs (restated)

Area	G&A Labor Head Count	Annual Cost (US$ x 1,000)	US$/st	G&A Other Annual Cost (US$ x 1,000)	US$/st	G&A Total Annual Cost (US$ x 1,000)	US$/st
Management	9	2,146.1	0.12	1,000.0	0.05	3,146.1	0.17
Site services	19	1,761.4	0.10	3,864.5	0.21	5,625.8	0.31
Finance	13	1,657.7	0.09	7,551.4	0.41	9,209.1	0.50
IT	7	857.3	0.05	3,221.0	0.18	4,078.3	0.22
Environmental	6	910.6	0.05	2,020.0	0.11	2,930.6	0.16
Financial assurances	0			4,750.0	0.26	4,750.0	0.26
Safety	5	618.5	0.03	986.2	0.05	1,604.7	0.09
Government affairs, public relations and legal	4	636.9	0.03	3,800.0	0.21	4,436.9	0.24
Human resources	4	466.8	0.03	3,269.3	0.18	3,736.0	0.20
Technical services	45	5,473.2	0.30	2,749.4	0.15	8,222.6	0.45
Total	*112*	*14,528.5*	*0.80*	*33,211.7*	*1.82*	*47,740.2*	*2.62*

21.3.6 Corporate Operating Costs

Corporate operating costs have not been included in this analysis.

21.3.7 Operating Cost Summary

Table 21-15 summarizes the operating costs by key area.

21.4 Comments on Section 21

When sustaining capital ($2,635.63 M), which includes closure costs ($210 M) are incorporated, the total Project capital cost estimate as restated by AMEC is $5,410.49 M.

Estimated operating costs over the LOM are $6,615.4 M, and mining costs average $12.56/st mined. Mining costs do not include operation of the paste backfill system.

Table 21-15: Operating Costs Summary- LOM (restated)

Area	Costs (US$ x 1,000)	Unit Cost	Units
Mining	6,615.4	$12.56	US$/st ROM
Processing	2,103.0	$3.99	US$/st milled
G&A	1,421.0	$2.70	US$/st milled
Surface Infrastructure	1,311.0	$2.49	US$/st milled
Total	11,450.3	$21.73	US$/st milled

22.0 ECONOMIC ANALYSIS

The cautionary statements in Section 1.3 should be read in conjunction with this section.

22.1 Methodology Used

The project has been evaluated using a discounted cash flow (DCF) analysis. Cash inflows consist of annual revenue projections for the mine. Cash outflows such as capital, including the two years of preproduction costs, operating costs, taxes, and royalties are subtracted from the inflows to arrive at the annual cash flow projections.

To reflect the time value of money, annual net cash flow (NCF) projections are discounted back to the project valuation date using several discount rates. The discount rate appropriate to a specific project depends on many factors, including the type of commodity; and the level of project risks, such as market risk, technical risk and political risk. The discounted, present values of the cash flows are summed to arrive at the project's net present value (NPV).

In addition to NPV, internal rate of return (IRR) and payback period are also calculated. The IRR is defined as the discount rate that results in an NPV equal to zero. Cash flows are taken to occur at the end of each period. Capital cost estimates have been prepared for initial development and construction of the project, and ongoing operations (sustaining capital).

The resulting net annual cash flows are discounted back to the date of valuation end-of-year 2014 dollars, and totaled to determine NPVs at the selected discount rates. The payback period is calculated as the time needed after the start up of operations to recover the initial capital spent.

22.2 Financial Model Parameters

The financial analysis was based on the Mineral Reserves presented in Section 15, the mine and process plan and assumptions detailed in Sections 16 and 17, the projected infrastructure requirements outlined in Section 18, the permitting, social and environmental regime discussions in Section 20, and the capital and operating cost estimates detailed in Section 21.

Table 22-1 presents the metal prices used for the purposes of the financial analysis of the Project.

Table 22-1: Metal Price Projections Used in Economic Analysis

Metals	Units	LOM
Copper	US$/lb	3.50
Nickel	US$/lb	9.50
Gold	US$/oz	1,300
Palladium	US$/oz	815
Platinum	US$/oz	1,680
Silver	US$/oz	21.50

Two concentrates will be produced at site, a copper concentrate and a nickel concentrate. The copper concentrate will receive credits for copper, gold and silver. The nickel concentrate will receive credits for copper, nickel, gold, platinum and palladium. The current model includes deduction for treatment charges and refining charges but no deductions for penalties. The nickel smelter payables and charges assumed in the analysis are those shown for Example 1, Table 19-3. Additional information on smelter contracts is included in Section 19.

There are various royalties applicable to the Project depending on the origin of the ore mined. These royalties are to be paid to either: federal government, state government or private entities. The royalty rates vary from 1.15% to 4.80%. Royalties are calculated based on various metrics including: portion of revenue from payable metal, net return value (NSR before freight cost) and net distributable earnings. Additional details regarding the royalties calculations can be found in Section 4 of this Report. Current base case royalties payments are estimated at $1,266 M over the LOM.

Working capital cash outflow and inflows are included in the model. The calculations are based on the assumption that accounts payable will be paid within 60 days and accounts receivable within 30 days.

22.3 Taxes

Duluth Metals Limited engaged PwC to perform a Federal income tax and applicable Minnesota state taxes analysis for the Report. The Project is currently held by a limited liability company that is treated as a partnership for U.S. tax purposes. However, for purposes of the tax calculations it was assumed that the Project is held and operated by a US corporation that is subject to Federal income taxes and state taxes in Minnesota.

The following general tax regime was recognized as applicable at 13 August, 2014, the date at which the final draft model and tax narrative was prepared for inclusion in the financial analysis.

22.3.1 US Federal and State Taxation Regime

For US federal income tax purposes, in accordance with the Internal Revenue Code (IRC), a taxpayer is required to calculate taxes under both the regular corporate tax system and the Alternative Minimum Tax (AMT) system and pay whichever method results in the higher amount of income taxes in a given taxation year.

The statutory US federal income tax rate is 35% and the tax rate under AMT is 20%. The applicable Minnesota tax rates are as follows:

- Minnesota net proceeds tax: 2%
- Minnesota occupation tax: 2.45%.

All the above Minnesota state taxes are deductible for federal income tax purposes.

US Federal net operating losses generated in a given year may be carried forward for 20 years and applied to taxable income when it arises, or carried back two years and applied against taxable income from the Project in those years. The IRC also allows mining companies to claim certain deductions related to their investment in mining properties, e.g. depletion and development expenditures.

22.3.2 Depletion

For federal income tax purposes, two forms of depletion are allowed: cost depletion and percentage depletion. The taxpayer is required to use the method that will result in the greatest deduction.

22.3.3 Cost Depletion

The first step of this method is to determine the number of units (as of the beginning of each year), which comprise the deposit. The unit can be any measure of production such as tonnes of ore, barrels of oil, board ft of timber, etc. The taxpayer must be consistent from year to year in the type of unit being used to calculate depletion to ensure uniformity. The second step takes the cost or adjusted basis of the property, which pertains to the deposit and divides this basis by the total number of units to obtain the depletion cost per unit. The depletion cost per unit is multiplied by the total units sold during the year to arrive at cost depletion.

22.3.4 Percentage Depletion

Under the percentage depletion method, a prescribed percentage of adjusted gross income from the activity is used to calculate the depletion allowance. The amount of percentage depletion calculated using the prescribed percentage may however be restricted based on the company's "adjusted taxable income". The deduction for depletion cannot exceed 50% of the adjusted taxable income from the activity. The adjusted taxable income from the property is computed without allowance for depletion. The amount of the deduction allowable under percentage depletion is not limited by the basis of the property. Thus, even though the basis of the property is reduced by the amount of depletion taken, if the basis becomes zero, the depletion based on the percentage of adjusted gross income may continue. However, if cost depletion in a given taxation year yields a higher deduction, it must be used in the calculation of taxable income.

22.3.5 Minnesota Mining Occupation Tax

The Minnesota Constitution mandates that the state impose an occupation tax (Minnesota Mining Occupation Tax or MOT) on the business of mining. In order to meet this constitutional requirement, the occupation tax is generally computed in accordance with the Minnesota corporate franchise (income) tax. The occupation tax is paid in lieu of the corporate franchise tax on revenue from mining; therefore, revenue from mining is exempt from corporate income tax. However, any non-mining revenue earned in Minnesota would still be subject to the corporate franchise tax.

In 2006, the legislature amended M.S. 298.01, subd. 3, and defined all sales as Minnesota sales, so 100% of net income is assigned to Minnesota. The tax rate is 2.45%. This change is effective for tax years beginning after December 31, 2005.

Generally, occupation tax is determined in the same manner as the corporate franchise tax imposed by M.S. Section 290.02 with the following exceptions:

- The tax is non-unitary because it applies only to the Minnesota mine and plant
- Percentage depletion is allowed
- Alternative minimum tax (AMT) is not applicable
- The applicable rate of tax is different.

As at 13 August, 2014, no taxpayer in Minnesota was paying the non-ferrous MOT. As this is the case, there have not been any regulations, judicial reviews, DOR opinions, forms, instructions, or other guidance beyond the constitution and statutes. As with all areas of tax, the laws could change prior to the payment of tax and therefore, caution should be exercised when reviewing the tax positions taken in respect of MOT for the Project.

22.3.6 Minnesota Net Proceeds Tax

Effective 1987, a person engaged in the business of mining shall pay to the State of Minnesota for distribution a net proceeds tax (NPT) equal to 2% of the net proceeds from mining in Minnesota. The tax applies to all ores, metals and minerals mined, extracted, produced or refined within the State of Minnesota, except for sand, silica sand, gravel, building stone, crushed rock, limestone, granite, dimension granite, dimension stone, horticultural peat, clay, soil, iron ore and taconite concentrates. Net proceeds are the gross proceeds from mining less allowable deductions. When a metal or mineral product is sold by the producer in an arm's-length transaction, the gross proceeds are equal to the proceeds from the sale of the product. Allowable deductions are deductions applied to the mining, production, processing, beneficiation, smelting, or refining of metal or mineral products. No other credits or deductions apply to this tax, and the carry-back or carry-forward of deductions is not allowed.

As at 13 August, 2014, no taxpayer in Minnesota was paying the NPT. As this is the case, there have not been any regulations, judicial reviews, DOR opinions, forms, instructions, or other guidance beyond the constitution and statutes. As with all areas of tax, the laws could change prior to the payment of tax and therefore, caution should be exercised when reviewing the tax positions taken in respect of NPT for the Project.

22.4 Financial Results

The after-tax NPV at an 8% discount rate over the estimated mine life is $753 million. The after-tax IRR is 11.4%. Payback of the initial capital investment is estimated to occur in 7.2 years after the start of production.

A summary of the financial analysis in US$ is presented as Table 22-2 and Table 22-3.

Results of the financial analysis are provided on an annual basis in Table 22-4. Years shown in Table 22-4 are for illustrative purposes only, as statutory permits and Board approval from TMM and Duluth are required to be granted prior to mine commencement.

Table 22-2: Summary Financial Analysis Results

Pre Tax	**Units**	**LOM**
Cumulative Cash flow Pre Tax	US$M	7,913
NPV 6%	US$M	2,231
NPV 8%	US$M	1,358
NPV 10%	US$M	732
Payback period	Years	6.4
IRR before tax	%	13.6%
After Tax	**Units**	**LOM**
Cumulative Cash flow After Tax	US$M	6,003
NPV 6%	US$M	1,449
NPV 8%	US$M	753
NPV 10%	US$M	257
Payback period	Years	7.2
IRR after tax	%	11.4%

Table 22-3: Financial Metrics Summary

Production Statistics						
Metal Price	**Units**	**Year 1**	**Year 2**	**Year 3**	**Avg. Y1–10**	**LOM**
Copper	US$/lb	3.50	3.50	3.50	3.50	3.50
Nickel	US$/lb	9.50	9.50	9.50	9.50	9.50
Gold	US$/oz	1,300	1,300	1,300	1,300	1,300
Palladium	US$/oz	815	815	815	815	815
Platinum	US$/oz	1,680	1,680	1,680	1,680	1,680
Silver	US$/oz	21.50	21.50	21.50	21.50	21.50
Copper	klbs	208,046	241,910	248,490	230,315	5,826,868
Cu Concentrate	klbs	188,870	220,885	226,893	210,188	5,332,942
Ni Concentrate	klbs	19,176	21,025	21,597	20,127	493,926
Nickel	klbs	39,669	53,333	55,692	50,771	1,235,014
Cu Concentrate	klbs	4,899	5,643	5,789	5,404	133,670
Ni Concentrate	klbs	34,770	47,690	49,903	45,367	1,101,345
Gold	koz	29.1	33.1	34.7	36.4	1,011
Cu Concentrate	koz	23.9	27.5	28.8	30.2	841
Ni Concentrate	koz	5.2	5.6	5.9	6.2	171
Palladium	koz	111.5	125.4	127.6	138.4	4,022
Cu Concentrate	koz	56.5	65.2	66.4	71.8	2,099
Ni Concentrate	koz	54.9	60.2	61.3	66.6	1,923
Platinum	koz	39.6	44.5	46.1	51.2	1,493
Cu Concentrate	koz	14.6	16.9	17.5	19.4	571
Ni Concentrate	koz	25.1	27.6	28.6	31.8	922
Silver	koz	890	1,023	1,047	994	25,230
Cu Concentrate	koz	740	857	877	833	21,218
Ni Concentrate	koz	150	165	169	161	4,012
Cash Flow Statistics						
Metal Revenue	**Units**	**Year 1**	**Year 2**	**Year 3**	**Avg. Y1–10**	**LOM**
Total Revenue	000 US$	1,031,373	1,253,084	1,295,059	1,211,109	30,698,594
Operating Costs						
On Site Costs	000 US$	332,645	369,105	352,908	351,007	11,450,323
Off Site Costs	000 US$	157,851	195,385	200,905	187,697	4,658,849
Royalties	000 US$	36,550	54,051	63,177	53,507	1,265,699
Operating profit	000 US$	504,328	634,542	678,069	618,898	13,323,723
Taxes, Capex and Working Capital						
Taxes	000 US$	18,094	29,280	34,080	72,307	1,910,283
Capex	000 US$	207,322	139,409	125,387	137,744	5,410,489
Changes in Working Capital	000 US$	(183,174)	(33,588)	(7,785)	(20,938)	(0)
Metal Revenue	**Units**	**Year 1**	**Year 2**	**Year 3**	**Avg. Y1–10**	**LOM**
Cash Flow						
Cash Flow Pre Tax	000 US$	113,832	461,546	544,897	460,216	7,913,233
Cash Flow After Tax	000 US$	95,738	432,266	510,817	387,909	6,002,950
Operation Statistics						
	Units	**Year 1**	**Year 2**	**Year 3**	**Avg. Y1–10**	**LOM**
Metal Equivalent						
Copper payable (Cu revenue)	klbs	189,966	221,252	227,271	210,616	5,331,701
Nickel payable (Ni revenue)	klbs	29,013	39,794	41,640	37,855	918,993
Copper equivalent (Cu + Ni revenue)	klbs	268,717	329,264	340,293	313,366	7,826,110
Copper equivalent (all metals revenues) *	klbs	294,678	358,024	370,017	346,031	8,771,027
Operating Costs and Profit margins per lbs of Cu						
Copper price	US$/lb	3.50	3.50	3.50	3.50	3.50
C1 costs / lb Cu **	US$/lb	0.65	0.39	0.24	0.31	0.76
Operating Margin / lb Cu	US$/lb	2.85	3.11	3.26	3.19	2.74
Operating Costs & Profit Margins per lbs of CuEq						
Copper price	US$/lb	3.50	3.50	3.50	3.50	3.50
C1 costs / lb CuEq ***	US$/lb	1.49	1.41	1.32	1.36	1.64
Operating Margin / lb CuEq	US$/lb	2.01	2.09	2.18	2.14	1.86
Operating costs and Profit margins per dst milled						
Revenue / dst milled	US$/dst	62.79	68.66	70.96	66.99	58.27
Operating cost / dst milled	US$/dst	29.86	30.93	30.35	29.82	30.58
Operating Margin / dst milled	US$/dst	32.93	37.73	40.62	37.17	27.69

Notes: * metal revenues do not include any payments for nickel and PGMs contained in the copper concentrate; please see Section 19 of the report for the discussion on payabilities for the concentrates; ** C1 Cu cost = (onsite costs + offsite cost – royalties – revenue from (Ni, Au, Ag, Pt, Pd))/ (Cu revenue/Cu price), where the units are US$/lbs of Cu; *** C1 CuEq cost = (onsite costs + offsite cost – royalties – revenue from (Au, Ag, Pt, Pd))/ ((Cu revenue/Cu Price)+(Ni revenue/Cu price)) where the units are US$/lbs of CuEq; dst = dry short ton; Avg = average; LOM = life-of-mine..

Table 22-4: Cashflow Analysis (note some figures reported in metric units)

Year	Units	LOM	2018	2019	2020	2021	2022	2023	2024
Project Time Line			-3	-2	-1	1	2	3	4
Mining									
Ore to Crusher	kmt	475,814	—	—	—	14,901	16,556	16,556	16,556
Ore to Stockpile	kmt	2,135	—	490	1,100	44	221	210	70
Stockpile to Crusher	kmt	2,135	—	—	1,590	—	—	—	—
Waste	kmt	27,879	523	1,505	1,655	1,134	1,590	1,120	1,056
Mill Feed									
Mill feed	kdmt	477,945	—	—	1,590	14,901	16,556	16,556	16,556
Cu grade	%	0.592	—	—	0.552	0.713	0.703	0.722	0.706
Ni Grade	%	0.191	—	—	0.173	0.233	0.231	0.239	0.237
Gold Grade	g/mt	0.08	—	—	0.06	0.08	0.08	0.08	0.09
Pd grade	g/mt	0.35	—	—	0.22	0.32	0.32	0.32	0.33
Pt Grade	g/mt	0.15	—	—	0.09	0.14	0.14	0.14	0.14
Ag Grade	g/mt	2.15	—	—	1.96	2.52	2.49	2.55	2.51
Copper Conc Revenue									
Copper revenue	000 US$	17,400,284	—	—	27,000	615,939	720,720	740,324	723,643
Gold revenue	000 US$	691,227	—	—	635	16,748	19,148	20,376	21,517
Silver revenue	000 US$	253,508	—	—	421	8,650	10,054	10,256	10,111
Total	000 US$	18,345,019	—	—	28,056	641,337	749,922	770,956	755,272
Nickel Conc Revenue									
Nickel revenue	000 US$	8,730,430	—	—	13,417	275,626	378,041	395,580	389,987
Copper revenue	000 US$	1,260,669	—	—	4,039	48,944	53,663	55,122	53,880
Platinum revenue	000 US$	1,160,764	—	—	2,198	31,543	34,717	35,988	36,492
Palladium revenue	000 US$	1,174,196	—	—	2,374	33,554	36,742	37,411	37,974
Gold revenue	000 US$	27,516	—	—	106	369	0	0	0
Total	000 US$	12,353,575	—	—	22,135	390,036	503,162	524,103	518,333
Total Revenue	*000 US$*	*30,698,594*	—	—	*50,190*	*1,031,373*	*1,253,084*	*1,295,059*	*1,273,605*
Operating Costs on site									
Mining	000 US$	6,615,376	—	—	19,242	180,812	205,460	187,678	170,042
Processing	000 US$	2,102,957	—	—	6,998	65,563	72,845	72,846	72,848
G & A	000 US$	1,421,005	—	—	4,767	44,664	44,573	46,156	46,159
Surface operating costs	000 US$	1,310,985	—	—	4,441	41,605	46,227	46,227	46,228
Total on site operating cost	*000 US$*	*11,450,323*	—	—	*35,448*	*332,645*	*369,105*	*352,908*	*335,276*
Operating Costs off site									
Copper Conc									
Treatment charge	000 US$	952,004	—	—	1,648	34,121	39,407	40,478	39,566
Freight costs	000 US$	1,307,075	—	—	2,262	46,848	54,104	55,576	54,324
Total	000 US$	2,259,079	—	—	3,910	80,969	93,511	96,054	93,890
Nickel Conc									
Treatment charge	000 US$	1,750,285	—	—	3,386	56,074	74,303	76,474	75,683
Freight costs	000 US$	649,485	—	—	1,256	20,808	27,572	28,377	28,084
Total	000 US$	2,399,770	—	—	4,642	76,881	101,875	104,851	103,767
Total off site operating cost	*000 US$*	*4,658,849*	—	—	*8,552*	*157,851*	*195,385*	*200,905*	*197,657*
NSR	000 US$	26,039,745	—	—	41,638	873,523	1,057,699	1,094,154	1,075,948
Royalties	000 US$	1,265,699	—	—	1,131	36,550	54,051	63,177	62,390
Operating profit	*000 US$*	*13,323,723*	—	—	*5,060*	*504,328*	*634,542*	*678,069*	*678,281*
Taxes	000 US$	1,910,283	—	—	—	18,094	29,280	34,080	44,761
Capital costs									
Initial Capital	000 US$	2,774,860	572,716	1,228,565	973,578	—	—	—	—
Sustaining capital	000 US$	2,635,630	—	—	—	207,322	139,409	125,387	184,821
Total Capital Cost	*000 US$*	*5,410,489*	*572,716*	*1,228,565*	*973,578*	*207,322*	*139,409*	*125,387*	*184,821*
Working Capital									
Accounts receivable	000 US$		—	—	8,250	169,541	205,986	212,886	209,360
Change in accounts receivable	000 US$		—	—	8,250	161,290	36,446	6,900	(3,527)
Accounts payable	000 US$		47,073	100,978	83,730	61,846	64,704	63,818	67,800
Change in accounts payable	000 US$		47,073	53,905	(17,248)	(21,883)	2,858	(885)	3,982
Change in working capital	*000 US$*	*0*	*47,073*	*53,905*	*(25,499)*	*(183,174)*	*(33,588)*	*(7,785)*	*7,509*
Valuation indicators									
Pre Tax									
Cash flow	000 US$	7,913,233	(525,644)	(1,174,660)	(994,018)	113,832	461,546	544,897	500,970
Cumulative Cash flow Pre Tax	000 US$	7,913,233	(525,644)	(1,700,303)	(2,694,321)	(2,580,489)	(2,118,943)	(1,574,046)	(1,073,076)
After Tax									
Cash flow	000 US$	6,002,950	(525,644)	(1,174,660)	(994,018)	95,738	432,266	510,817	456,209
Cumulative Cash flow After Tax	000 US$	6,002,950	(525,644)	(1,700,303)	(2,694,321)	(2,598,583)	(2,166,317)	(1,655,499)	(1,199,291)

Year	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Project Time Line	5	6	7	8	9	10	11	12	13	14
Mining										
Ore to Crusher	16,556	16,556	16,553	16,559	16,557	16,556	16,557	16,539	16,551	16,559
Ore to Stockpile	—	—	—	—	—	—	—	—	—	—
Stockpile to Crusher	—	—	—	—	—	—	—	—	—	—
Waste	1,421	941	856	1,098	1,300	1,129	1,391	782	1,687	1,272
Mill Feed										
Mill feed	16,556	16,556	16,556	16,556	16,557	16,556	16,557	16,557	16,556	16,556
Cu grade	0.674	0.647	0.668	0.666	0.654	0.649	0.610	0.584	0.611	0.607
Ni Grade	0.231	0.220	0.223	0.219	0.215	0.202	0.183	0.182	0.199	0.184
Gold Grade	0.09	0.09	0.09	0.09	0.09	0.10	0.11	0.10	0.10	0.11
Pd grade	0.34	0.37	0.38	0.36	0.37	0.41	0.45	0.43	0.46	0.48
Pt Grade	0.15	0.16	0.17	0.16	0.16	0.18	0.20	0.19	0.20	0.21
Ag Grade	2.43	2.34	2.43	2.45	2.46	2.42	2.34	2.13	2.17	2.17
Copper Conc Revenue										
Copper revenue	691,059	663,437	684,349	682,858	670,473	665,056	625,049	598,483	625,866	621,973
Gold revenue	23,932	27,809	27,232	24,538	24,511	28,895	34,721	31,765	33,302	36,747
Silver revenue	9,885	9,547	9,965	10,175	10,333	10,160	9,998	8,780	8,787	8,836
Total	724,876	700,793	721,546	717,571	705,317	704,111	669,768	639,028	667,955	667,557
Nickel Conc Revenue										
Nickel revenue	374,866	361,954	366,793	361,356	356,793	335,264	298,367	297,425	329,133	307,888
Copper revenue	51,454	49,398	50,955	50,844	49,922	49,518	46,539	44,561	46,600	46,310
Platinum revenue	38,565	43,444	45,001	42,146	43,357	49,241	54,995	52,931	55,895	58,687
Palladium revenue	39,277	43,285	44,405	42,051	43,023	48,702	53,558	50,840	55,122	57,962
Gold revenue	0	574	496	0	0	1,072	2,622	1,893	1,893	2,799
Total	504,162	498,656	507,650	496,397	493,094	483,796	456,082	447,650	488,643	473,647
Total Revenue	*1,229,038*	*1,199,449*	*1,229,195*	*1,213,968*	*1,198,411*	*1,187,907*	*1,125,851*	*1,086,679*	*1,156,598*	*1,141,203*
Operating Costs on site										
Mining	162,703	191,043	191,856	177,123	198,450	199,980	221,937	231,980	239,399	226,785
Processing	72,846	72,847	72,846	72,847	72,849	72,847	72,849	72,849	72,847	72,845
G & A	46,790	46,790	47,748	47,732	47,740	47,740	47,740	47,789	47,754	47,731
Surface operating costs	46,227	46,227	46,227	46,228	46,229	46,228	46,229	46,229	46,228	46,227
Total on site operating cost	*328,566*	*356,906*	*358,676*	*343,929*	*365,267*	*366,795*	*388,754*	*398,847*	*406,228*	*393,589*
Operating Costs off site										
Copper Conc										
Treatment charge	37,785	36,274	37,418	37,336	36,659	36,363	34,176	32,723	34,220	34,007
Freight costs	51,877	49,804	51,374	51,262	50,332	49,925	46,922	44,928	46,983	46,691
Total	89,662	86,078	88,792	88,598	86,991	86,288	81,098	77,651	81,204	80,699
Nickel Conc										
Treatment charge	73,724	73,011	73,524	73,004	72,794	70,654	65,874	65,850	69,776	67,866
Freight costs	27,357	27,093	27,283	27,090	27,012	26,218	24,444	24,435	25,892	25,183
Total	101,081	100,104	100,806	100,094	99,805	96,872	90,318	90,286	95,668	93,049
Total off site operating cost	*190,743*	*186,183*	*189,598*	*188,692*	*186,797*	*183,161*	*171,415*	*167,937*	*176,872*	*173,748*
NSR	1,038,295	1,013,266	1,039,597	1,025,276	1,011,615	1,004,746	954,435	918,742	979,726	967,456
Royalties	56,466	51,699	54,373	53,472	51,806	51,082	44,103	42,836	47,100	44,039
Operating profit	*653,263*	*604,661*	*626,549*	*627,875*	*594,542*	*586,869*	*521,579*	*477,060*	*526,399*	*529,828*
Taxes	114,567	80,900	103,414	103,025	97,449	97,503	78,780	70,811	81,221	87,987
Capital costs										
Initial Capital	—	—	—	—	—	—	—	—	—	—
Sustaining capital	225,145	181,095	67,343	142,991	55,804	48,120	132,471	26,107	62,895	88,696
Total Capital Cost	*225,145*	*181,095*	*67,343*	*142,991*	*55,804*	*48,120*	*132,471*	*26,107*	*62,895*	*88,696*
Working Capital										
Accounts receivable	202,034	197,170	202,060	199,556	196,999	195,272	185,071	178,632	190,126	187,595
Change in accounts receivable	(7,326)	(4,864)	4,890	(2,503)	(2,557)	(1,727)	(10,201)	(6,439)	11,494	(2,531)
Accounts payable	75,246	70,420	63,567	68,393	62,229	61,369	67,029	58,072	63,642	64,772
Change in accounts payable	7,445	(4,825)	(6,853)	4,825	(6,163)	(860)	5,660	(8,958)	5,571	1,130
Change in working capital	*14,771*	*39*	*(11,743)*	*7,328*	*(3,606)*	*867*	*15,861*	*(2,519)*	*(5,923)*	*3,660*
Valuation indicators										
Pre Tax										
Cash flow	442,889	423,605	547,463	492,212	535,132	539,616	404,968	448,434	457,581	444,792
Cumulative Cash flow Pre Tax	(630,187)	(206,582)	340,881	833,093	1,368,225	1,907,841	2,312,809	2,761,243	3,218,824	3,663,616
After Tax										
Cash flow	328,322	342,705	444,049	389,187	437,683	442,113	326,188	377,623	376,361	356,805
Cumulative Cash flow After Tax	(870,969)	(528,263)	(84,215)	304,972	742,655	1,184,768	1,510,956	1,888,579	2,264,940	2,621,745

Year	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044
Project Time Line	15	16	17	18	19	20	21	22	23	24
Mining										
Ore to Crusher	16,576	16,556	16,557	16,557	16,557	16,556	16,556	16,556	16,556	16,556
Ore to Stockpile	—	—	—	—	—	—	—	—	—	—
Stockpile to Crusher	—	—	—	—	—	—	—	—	—	—
Waste	628	556	877	495	548	605	553	747	446	439
Mill Feed										
Mill feed	16,556	16,556	16,556	16,556	16,556	16,556	16,556	16,556	16,556	16,556
Cu grade	0.605	0.625	0.634	0.594	0.565	0.547	0.527	0.513	0.509	0.506
Ni Grade	0.181	0.187	0.190	0.185	0.177	0.173	0.175	0.169	0.168	0.167
Gold Grade	0.12	0.12	0.11	0.09	0.08	0.07	0.07	0.07	0.07	0.07
Pd grade	0.53	0.53	0.50	0.40	0.33	0.31	0.28	0.27	0.29	0.28
Pt Grade	0.23	0.23	0.22	0.18	0.14	0.14	0.12	0.12	0.13	0.12
Ag Grade	2.24	2.37	2.37	2.21	2.09	2.01	1.91	1.85	1.81	1.78
Copper Conc Revenue										
Copper revenue	620,234	641,062	649,809	608,992	578,108	557,433	535,713	520,571	514,849	511,472
Gold revenue	44,274	40,438	38,107	28,535	21,127	20,052	16,881	16,659	18,305	17,327
Silver revenue	9,305	10,004	9,977	9,224	8,734	8,374	7,871	7,618	7,412	7,200
Total	673,812	691,503	697,892	646,750	607,969	585,859	560,465	544,849	540,566	535,999
Nickel Conc Revenue										
Nickel revenue	298,978	305,874	310,933	297,681	283,044	256,006	255,843	245,970	244,580	244,666
Copper revenue	46,181	47,732	48,383	45,344	42,927	37,190	35,396	34,238	33,844	33,605
Platinum revenue	65,507	65,088	62,150	47,582	36,726	34,311	29,003	27,980	30,151	28,874
Palladium revenue	64,910	64,438	60,645	47,105	37,696	35,133	30,887	29,809	31,393	30,158
Gold revenue	4,508	3,758	3,257	1,316	0	1,166	261	271	529	346
Total	480,083	486,891	485,367	439,028	400,394	363,806	351,390	338,268	340,496	337,649
Total Revenue	*1,153,895*	*1,178,394*	*1,183,259*	*1,085,778*	*1,008,363*	*949,665*	*911,855*	*883,117*	*881,062*	*873,649*
Operating Costs on site										
Mining	243,747	214,210	234,556	231,119	247,115	260,934	268,440	264,678	272,495	270,607
Processing	72,847	72,848	72,845	72,847	72,846	72,847	72,847	72,847	72,847	72,847
G & A	47,682	47,740	47,737	47,737	47,737	47,740	47,740	47,740	47,740	47,740
Surface operating costs	46,228	46,228	46,227	46,227	46,081	45,253	44,493	43,302	42,561	41,948
Total on site operating cost	*410,503*	*381,026*	*401,365*	*397,930*	*413,779*	*426,774*	*433,520*	*428,567*	*435,642*	*433,142*
Operating Costs off site										
Copper Conc										
Treatment charge	33,912	35,051	35,529	33,298	31,609	30,479	29,291	28,463	28,150	27,966
Freight costs	46,561	48,124	48,781	45,717	43,398	41,846	40,216	39,079	38,649	38,396
Total	80,473	83,175	84,310	79,014	75,007	72,325	69,507	67,542	66,800	66,362
Nickel Conc										
Treatment charge	66,364	66,880	67,442	65,511	63,992	47,702	47,744	46,003	46,416	46,071
Freight costs	24,626	24,817	25,026	24,309	23,746	17,701	17,716	17,070	17,224	17,096
Total	90,990	91,697	92,469	89,820	87,737	65,403	65,460	63,073	63,640	63,167
Total off site operating cost	*171,463*	*174,873*	*176,779*	*168,834*	*162,745*	*137,728*	*134,967*	*130,616*	*130,439*	*129,529*
NSR	982,432	1,003,522	1,006,481	916,944	845,619	811,937	776,888	752,502	750,623	744,120
Royalties	44,763	45,809	45,500	45,260	41,399	39,241	37,683	36,579	35,855	35,490
Operating profit	*527,166*	*576,687*	*559,616*	*473,754*	*390,440*	*345,922*	*305,685*	*287,356*	*279,125*	*275,488*
Taxes	99,772	122,157	117,153	92,976	73,884	63,295	30,984	39,330	40,251	46,495
Capital costs										
Initial Capital	—	—	—	—	—	—	—	—	—	—
Sustaining capital	37,788	44,813	104,563	77,376	66,810	147,171	31,624	54,549	70,920	17,218
Total Capital Cost	*37,788*	*44,813*	*104,563*	*77,376*	*66,810*	*147,171*	*31,624*	*54,549*	*70,920*	*17,218*
Working Capital										
Accounts receivable	189,681	193,709	194,508	178,484	165,758	156,109	149,894	145,170	144,832	143,613
Change in accounts receivable	2,086	4,027	800	(16,024)	(12,726)	(9,649)	(6,215)	(4,724)	(338)	(1,219)
Accounts payable	62,818	63,179	69,482	64,305	62,352	66,921	54,968	56,683	58,612	54,401
Change in accounts payable	(1,954)	361	6,303	(5,177)	(1,953)	4,569	(11,953)	1,715	1,929	(4,211)
Change in working capital	*(4,040)*	*(3,667)*	*5,503*	*10,848*	*10,773*	*14,218*	*(5,738)*	*6,439*	*2,267*	*(2,992)*
Valuation indicators										
Pre Tax										
Cash flow	485,337	528,208	460,556	407,225	334,404	212,969	268,323	239,246	210,471	255,278
Cumulative Cash flow Pre Tax	4,148,954	4,677,162	5,137,718	5,544,943	5,879,347	6,092,316	6,360,639	6,599,885	6,810,356	7,065,634
After Tax										
Cash flow	385,565	406,050	343,403	314,249	260,519	149,674	237,339	199,916	170,220	208,782
Cumulative Cash flow After Tax	3,007,310	3,413,360	3,756,764	4,071,013	4,331,532	4,481,206	4,718,546	4,918,462	5,088,682	5,297,464

Year	2045	2046	2047	2048	2049	2050	2051
Project Time Line	25	26	27	28	29	30	31
Mining							
Ore to Crusher	16,556	16,556	14,207	13,674	9,893	9,230	—
Ore to Stockpile	—	—	—	—	—	—	—
Stockpile to Crusher	—	—	545	—	—	—	—
Waste	364	279	299	277	141	165	—
Mill Feed							
Mill feed	16,556	16,556	14,755	13,674	9,893	9,230	0
Cu grade	0.497	0.483	0.460	0.442	0.449	0.451	0.000%
Ni Grade	0.164	0.158	0.150	0.144	0.148	0.153	0.000%
Gold Grade	0.06	0.06	0.05	0.05	0.05	0.05	0.00
Pd grade	0.24	0.23	0.21	0.20	0.20	0.20	0.00
Pt Grade	0.11	0.10	0.09	0.09	0.09	0.09	0.00
Ag Grade	1.71	1.67	1.60	1.55	1.58	1.60	0.00
Copper Conc Revenue							
Copper revenue	501,741	489,792	416,367	370,471	272,130	255,311	0
Gold revenue	14,037	12,543	9,952	8,495	6,413	6,204	0
Silver revenue	6,820	6,659	5,722	5,182	3,828	3,619	0
Total	522,598	508,995	432,041	384,148	282,371	265,134	0
Nickel Conc Revenue							
Nickel revenue	241,331	241,307	206,231	182,372	138,662	134,461	0
Copper revenue	32,910	32,696	27,995	24,905	18,355	17,221	0
Platinum revenue	24,446	23,297	19,074	17,017	12,392	11,965	0
Palladium revenue	26,045	25,000	20,654	18,278	13,230	12,533	0
Gold revenue	0	60	95	41	49	35	0
Total	324,732	322,360	274,048	242,613	182,689	176,215	0
Total Revenue	*847,331*	*831,355*	*706,089*	*626,761*	*465,060*	*441,349*	*0*
Operating Costs on site							
Mining	251,935	255,257	230,940	218,765	162,609	183,479	0
Processing	72,847	72,847	64,920	60,164	43,531	40,610	0
G & A	47,740	47,740	43,103	47,740	47,740	47,740	0
Surface operating costs	41,693	43,427	40,048	36,869	27,624	25,771	0
Total on site operating cost	*414,214*	*419,271*	*379,011*	*363,538*	*281,504*	*297,600*	—
Operating Costs off site							
Copper Conc							
Treatment charge	27,433	26,780	22,766	20,256	14,879	13,960	0
Freight costs	37,665	36,768	31,256	27,811	20,429	19,166	0
Total	65,099	63,549	54,022	48,067	35,308	33,126	0
Nickel Conc							
Treatment charge	44,386	41,301	34,231	30,177	22,511	21,559	0
Freight costs	16,470	15,326	12,702	11,198	8,353	8,000	0
Total	60,856	56,626	46,933	41,375	30,864	29,559	0
Total off site operating cost	*125,955*	*120,175*	*100,955*	*89,442*	*66,172*	*62,685*	—
NSR	721,375	711,181	605,134	537,319	398,888	378,664	—
Royalties	34,640	31,126	25,106	21,652	15,878	15,443	—
Operating profit	*272,521*	*260,784*	*201,018*	*152,130*	*101,507*	*65,621*	—
Taxes	48,394	46,030	33,271	8,724	5,695	—	—
Capital costs							
Initial Capital	—	—	—	—	—	—	—
Sustaining capital	32,504	70,748	13,311	932	1,366	176,334	—
Total Capital Cost	*32,504*	*70,748*	*13,311*	*932*	*1,366*	*176,334*	—
Working Capital							
Accounts receivable	139,287	136,661	116,069	103,029	76,448	72,550	—
Change in accounts receivable	(4,326)	(2,626)	(20,592)	(13,040)	(26,581)	(3,898)	(72,550)
Accounts payable	53,894	56,494	45,341	39,804	30,461	45,375	—
Change in accounts payable	(507)	2,601	(11,153)	(5,537)	(9,343)	14,914	(45,375)
Change in working capital	*3,819*	*5,227*	*9,439*	*7,503*	*17,238*	*45,987*	
Valuation indicators							
Pre Tax							
Cash flow	243,836	195,263	197,146	158,702	117,379	(64,726)	
Cumulative Cash flow Pre Tax	7,309,470	7,504,733	7,701,879	7,860,580	7,977,959	7,913,233	
After Tax							
Cash flow	195,443	149,233	163,875	149,978	111,684	(64,726)	
Cumulative Cash flow After Tax	5,492,907	5,642,139	5,806,014	5,955,992	6,067,676	6,002,950	

22.5 Sensitivity Analysis

Sensitivity analysis was performed on the base case net cash flow and examined sensitivity to copper and nickel price, operating costs and capital costs.

Sensitivities are shown in Figure 22-1 and Table 22-5 for the after-tax scenarios. For the purposes of the analysis changes in nickel and copper grades were found to be reasonably represented by the changes in metal prices, and are not shown.

22.6 Comments on Section 22

Under the assumptions presented in this Report, the Project demonstrates positive economics. The after-tax NPV at an 8% discount rate over the estimated mine life is $753 million. The after-tax IRR is 11.4%. Payback of the initial capital investment is estimated to occur in 7.2 years after the start of production. The project is most sensitive to changes in copper prices, less sensitive to changes in operating costs, less sensitive to changes in capital costs and least sensitive to changes in nickel price.

Figure 22-1: NPV Sensitivity After-Tax

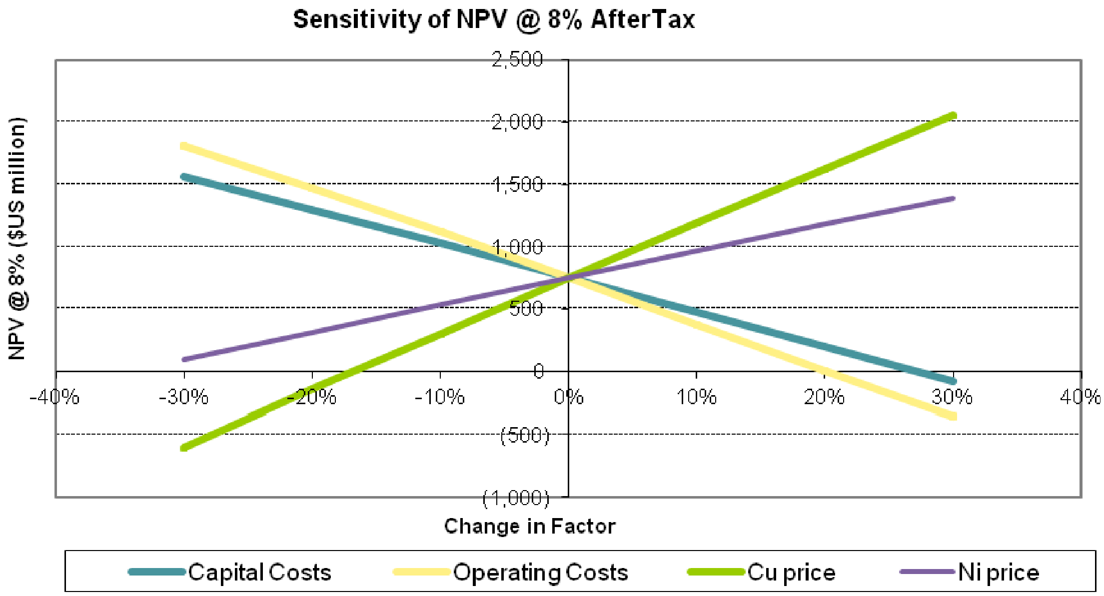


Table 22-5: Sensitivity Table NPV After-Tax (basecase is highlighted)

		Change in Factor						
		-30%	**-20%**	**-10%**	**0%**	**10%**	**20%**	**30%**
Factor	Capital Costs	1,562	1,296	1,027	753	477	199	(82)
	Operating Costs	1,803	1,465	1,118	753	375	7	(362)
	Cu price	(607)	(145)	296	753	1,197	1,628	2,051
	Ni price	93	312	536	753	968	1,179	1,388

23.0 ADJACENT PROPERTIES

This section is not relevant to this Report.

24.0 OTHER RELEVANT DATA AND INFORMATION

While no formal risk and opportunity analysis was completed for the Project, the QPs evaluated potential risks and opportunities in their areas of expertise as follows, by company/QP undertaking the analysis.

24.1 Opportunities

24.1.1 Geology and Mineral Resources (AMEC)

- The PFS utilizes only the Measured and Indicated Mineral Resources estimated for Maturi and Maturi Southwest. Indicated Mineral Resources estimated for the Birch Lake deposit represent upside potential if included in an updated mine plan
- Inferred Mineral Resources for the Spruce Road deposit are also a potential Project upside, but will require additional drilling to increase the confidence category, and assessment as to consideration of the most appropriate mining method, given the proximity to the Boundary Waters Canoe Area Wilderness
- Additional upside is represented in the targets for additional exploration noted for Maturi and Maturi Southwest; however, these targets require significant further drilling before any tonnage–grade estimate can be classified
- Mineralization is open down dip and along strike in all four deposits and there is potential to expand the known mineralization in those directions with additional drilling and supporting studies.

24.1.2 Metallurgy (Blue Coast)

- Piloting the more finely-tuned Blue Coast flowsheet (versus the ALS flowsheet that is used as the basis of the PFS design) may yield a quick and substantial improvement in metallurgy over that predicted in the PFS. This comment also applies to the pyrrhotite rejection flowsheet
- Upside exists in concentrate grade with the use of chemical and/or engineering technologies to minimize the recovery of non-sulfide gangue to the final nickel concentrate
- The use of gravity concentration on the concentrates could allow for the recovery of gold, platinum and palladium to high-grade precious metal concentrates, which should attract better pay than is assumed in this Report
- Further development of the pyrrhotite rejection circuit to enhance nickel recoveries and broader application of the circuit to a great proportion of the life of mine feed could increase the mean LOM nickel concentrate grades

- Downstream hydrometallurgical processing may enhance the net revenue obtained from the flotation concentrate. Processing the entire concentrate stream will enhance recoveries but will be risky and capital intensive. Processing of a Cu–Ni middlings stream, however, may minimize capital costs and project risk while creating copper metal and nickel precipitate, and leaving the remaining copper and nickel concentrates as marketable high-grade products
- Enhancement of fine nickel recoveries may be possible through advanced engineering cell technologies such as pneumatic cells, which are aimed at maximizing the recovery of pentlandite fines
- The use of inert regrinding media has substantially improved cleaner metallurgy and its use in primary grinding may enhance overall recoveries. It may also reduce the need for depressants or (as in the case of the Kevitsa Mine in Finland) completely eliminate the use of TETA
- The use of auxiliary collectors, as commonly practiced in South Africa, may enhance platinum group metal recoveries.

24.1.3 Mine Design (SRK)

- Mine design regional pillar layout should be re-evaluated to optimize extraction ratio and maximize the extracted grade considering current rock mass strength results and maintaining appropriate safety margins. This would likely require iterations between the mine planning group and the geotechnical group. Currently approximately 45% of the tons in the Maturi and Maturi Southwest Mineral Resource estimates are captured in the Mineral Reserve estimate
- Analyze the opportunity of long-hole stoping areas in Tier 1 to benefit from lower operating costs
- Evaluate possible design modification in Tier 1 to standardize equipment throughout the mine in support of potential reductions in capital costs
- Outsourcing to a third-party contractor of mine expansion activities in peak years (i.e., development to second crusher, etc.) thus delaying or even reducing sustaining capital costs
- More detailed mine planning and scheduling to delay the development of ramps and footwall accesses, thus delaying or even reducing sustaining capital costs
- Re-evaluate cutoff grade strategy with current panel designs and mining macro-sequence in support of improved Project economics
- The PFS assumes that automation will be used to perform drilling in long-hole stopes between shifts. As technology advances, there may be an opportunity to use automation for loading and hauling, which would potentially improve safety,

improve productivity and reduce costs. Ventilation on demand, automation of pumping systems, and controls on electrical systems should also be evaluated

- It may be advantageous in certain areas of the mine to use an ore pass system to facilitate material movement. The potential benefit would be likely reduced haulage costs; however, there may be additional development costs incurred
- Optimization of long-hole stope mucking remuck/truck configuration to minimize costs and maximize productivity.

24.1.4 Mine Design (AMEC)

- A 20% risk factor is carried in the capital costs of the main vent raises. More detailed design may result in this risk being reduced
- Re-evaluate cutoff grade strategy for high cost areas such as Tier 3 post-pillar cut-and-fill mining zones to optimize Project economics
- Evaluate the potential to reduce development in waste and low-grade material to determine if sustaining capital costs can be reduced
- The scalping grizzlies at the crusher pocket could be omitted, which would present a cost savings opportunity in excavation, materials and construction costs for the crusher area

24.1.5 Mine Design (Itasca)

- A 250 ft barrier pillar width represents a reasonable average for PFS planning purposes, but in general it appears that the barrier pillars would need to be widened above 250 ft where the orebody is thicker and could potentially be narrowed where the orebody is thinner. During optimization, barriers should be sized locally depending on orebody thickness (and should continue to consider the most appropriate safety margins) and, if possible, placed in thinner or lower-grade sections of the orebodies to maximize recovery
- Feasibility-level evaluations should aim to conduct analyses in which explicit representation of rock mass fracturing and detachment (collapse) is possible so that a more accurate estimate of the likely sloughage depths and associated dilution can be made for pillars and stopes. This may potentially result in a decrease in the dilution allocation
- Once the barrier pillar design is finalized in the feasibility study, the geomechanical models can be re-run with the final configuration, and a ramp offset optimization can be performed.

24.1.6 Infrastructure (Barr)

- Facility siting optimization. New information developed since the TSF siting study could suggest opportunities to optimize Project configuration. Further, acquisition of additional subsurface geotechnical information, in-field reconnaissance and further process and operations plan development may reveal opportunities to optimize overall site plans. For example, property ownership positions and conditions of sale change over time; property ownership constraints present during facility siting may have changed, thereby opening new alternatives for the Project configuration
- Project-wide water balance optimization. The current water balance includes simplifying assumptions (e.g., zero evaporation and zero leakage from the TSF, and other smaller water storage ponds). Future iterations of the Project-wide water balance, including further consideration of water management alternatives, may yield opportunities for optimization of water management system infrastructure design
- High voltage transmission and electricity service providers. While PUC constraints potentially limit competition, PUC rules simultaneously accommodate alternative electric service scenarios, albeit via alternate and potentially more time-consuming administrative processes. Further review and consideration of competing high-voltage transmission and electricity service providers is recommended
- Integration of Project-wide earthwork requirements. Infrastructure earthwork plans and cost estimates have been prepared independently from earthwork plans and cost estimates for the mine, concentrator and TSF. An integrated Project-wide earthwork assessment could provide Project cost optimization, by limiting earthwork haul distances, minimizing double handling of materials, and avoiding unnecessary removal of bedrock or other foundation material
- Transportation of products and consumables. Further consideration of current plans for transportation of the final products to clients as well as for delivery of consumables to the Project is recommended. This could include the development of a more compact site, or pipelines for transport of concentrate slurry to filter plants and rail transloading facilities that are located closer to the existing rail lines. Initial studies of off-site transportation/logistics considered both bulk and sack products for nickel and copper concentrates. Bulk was selected for both transport options and corresponding load-out infrastructure was developed to support this handling method. This decision should be verified after more detailed review of the product chemistry, self-heating properties, third-party terminal capabilities, and environmental regulations at Canadian ports. Changes to the

handling/transportation model may impact the current storage and load-out concepts and associated operations

- Renewable energy and energy conservation. Further evaluation of net present costs of conventional versus renewable energy sources (especially geothermal for building heat) and further exploration of energy conservation measures could yield long-term cost savings. In addition, consideration of Leadership in Energy and Environmental Design (LEED) principles and certifications as well as a sustainability philosophy plan for the overall Project may warrant further consideration and refinement
- Other potential opportunities for Project optimization include but are not limited to:
  - Eliminate TSF intermediate collection pond by increasing TSF water storage capacity, while maintaining appropriate dam stability
  - Contact water quality is unknown; mine site and concentrator site contact water management infrastructure may be overdesigned if contact water quality is good.

24.1.7 Infrastructure (Golder)

- Project configuration and facility siting studies. Further studies regarding the TSF site location could result in optimization of the overall Project configuration by reducing the required transport distances for tailings, water and concentrate between the concentrator and the TSF site
- Mine plan and backfill scheduling. The mine backfill requirements have been developed based upon peak backfill rates required by the development schedule. Further evaluation of the mine plan and schedule in future studies should be considered to optimize the paste backfill and tailings management systems including:
  - Optimize the paste backfill plant capacity or required number of paste plants
  - Optimize the paste backfill distribution system including number of surface boreholes and quantity of inter-level associated boreholes
  - Reduce variability of tailings delivery rates to TSF and paste backfill plants to optimize tailings transport system
  - Reduce the variability in tailings delivery rate to the TSF to enable optimization of alternative tailings management strategies such as dry stack tailings management.
- Tailings management strategy. Additional tailings dewatering studies are recommended in the future to further evaluate dry stack tailings management options for the TSF. Dry stack tailings management options could lead to optimization of the TSF including:

  - Reduction in TSF footprint
  - Reduced freshwater demand for the Project
  - Reduction in the risk of potential contamination of surface water and soils by dewatering tailings prior to deposition
  - Reduction in required quantities and the impact of uncertainty in availability and suitability of construction materials at the TSF
  - Opportunities for progressive closure.
- Paste backfill plants. Within the paste plant, future studies should consider optimization of the binder storage and delivery system (dense phase pneumatic conveying systems for each plant).

24.1.8 Process (AMEC)

- Use larger flotation cells such as the 500 m^3 Outotec or 600 m^3 FLSmidth Tank cells to reduce footprint and tank capital costs and operating costs. This would result in fewer cells, less plant footprint and associated earthworks, and lower power consumption
- Optimize reagent conditions and chemistry to increase flotation rates for copper and nickel minerals, which could reduce the necessary flotation residence time, hence cost of flotation equipment
- Reductions in operating costs may be possible with the introduction of flotation circuit thickeners (in between copper rougher and nickel feed roughers and between the copper first cleaner and nickel first cleaner flotation cells) could be considered an opportunity as well as risk protection, due to the recirculation of chemicals back to the correct flotation area, although the amount of chemical recovered is not expected to be high
- Most major process equipment prices were based on informal quotes from a single supplier. It is possible that with a more competitive bid process, more favorable equipment prices could be negotiated
- There may be opportunities for value engineering of the process plant facilities, in conjunction with an assessment of production risks. The PFS design appears robust, with conservative equipment sizings in some areas. There may be opportunities to scale back sizes of equipment such as conveyors to reduce capital cost with limited production risk. There may be opportunities to reduce costs of the concentrate filtering, storage and load out facilities.

24.1.9 Marketing (AMEC)

- The Wood Mackenzie marketing opinion on concentrate terms does not include consideration of the PGMs that will be present in the copper concentrate.

Additional Project-specific marketing studies should be conducted to determine what payabilities could be expected for PGMs in the copper concentrates, and if the concentrates would incur any additional treatment charges for these elements. AMEC recognizes that some copper smelters may pay for PGMs in copper concentrates, and recommends that Duluth seeks indicative terms for the Project concentrate from such smelters. There is potential upside for the Project if payment for the PGMs from copper concentrate can be included in Project economics.

24.1.10 Environmental and Permitting (AMEC)

- It may be possible for TMM to obtain input from State and Federal agencies to establish baseline data collection and modeling work plans, consistent with established and published data collection protocols, early in the data collection process
- Based on feedback that may be received during the EIS and permitting stage, TMM could determine key areas of interest to stakeholders in the Project and potentially address these in the Project design.

24.1.11 Cost Estimates (AMEC)

- The process plant design, if optimized, may result in capital cost savings
- Unified earthworks planning across the Project may also result in reduced capital costs
- Quantity purchases of bulk commodities may result in lower unit costs
- Preferred vendor agreements for mining equipment, reagents, supplies, and consumables may provide lower unit costs
- A maintenance and repair contract (MARC) for mobile equipment maintenance may result in lower maintenance costs.

24.2 Risks

24.2.1 Mineral Tenure and Surface Rights (AMEC)

- The option over the Dunka pit was exercised, in part, to provide a potential water source for the Project. However, the option has not been closed as there are a number of regulatory reviews that must be completed and regulatory approval of portions of the transaction is required prior to closing. There is a risk that the regulatory authorities may impose requirements on pit water discharges such as water treatment facilities. Any such water treatment costs and costs for a water treatment facility are not included in the Project financial analysis. There is also a risk that state of Minnesota consent/approvals required for the formal transfer of lands will not be forthcoming

- There is a risk to the Project configuration as envisaged in the PFS if lands proposed for infrastructure cannot be obtained in a timely manner, are found to be cost-prohibitive to acquire, or surface rights are unable to be assigned for mining purposes
- The current financial model assumes a total federal royalty rate of 4.8% under the federal leases. The federal leases contain language allowing the Secretary of the Interior for the BLM, at his discretion, to increase the royalty rates at the time the federal leases are renewed. Currently the third renewal for these leases is in process and the BLM has expressed an interest in renegotiating the terms and conditions of the royalty. While the lease allows a maximum royalty of 6% to be applied for the third renewal period, the exact royalty rate will be subject to negotiations with BLM. In the event the BLM adjusts the total rate for royalties payable under the federal leases, the financial model will need to be adjusted to reflect any changes that occur.

24.2.2 Geology and Mineral Resources (AMEC)

- Legacy analytical data have been largely validated by work at Maturi which has a large number of legacy drill holes. The legacy data at Maturi are considered by AMEC to be adequate to support resource estimation; however, there is a small economic risk to the all of the deposits in the Project that legacy data are biased, positively or negatively and that estimates based on those data will not be realized. That risk is largely mitigated by classifying portions of the estimate that rely heavily on legacy data as Inferred Mineral Resources and requiring additional drilling in those areas before the resource classification can be improved
- The classification of all of the deposits is based on assessment of continuity of grade. There has been no detailed assessment of the effect of small faults or undulations of the top and bottom of assay walls. This may affect the ability to perform detailed planning and/or dilution, and may require redefinition of Measured Resources. AMEC recommends that TMM performs a conditional simulation study on the potential impact of small faults and undulations of the assay walls as part of any feasibility study. Several larger faults were recognized and modeled at Maturi Southwest and Birch Lake; however, no faults were identified or modeled at Maturi. Small faults and undulations of the assay wall are likely at Maturi and may have an adverse impact on mine planning.

24.2.3 Metallurgy (Blue Coast)

- Some of the grindability data from two separate laboratories were contradictory. Should the dataset from the laboratory indicating the mill feed would be harder prove to be correct, the primary mill may not be large enough to achieve the design throughput
- To date, Maturi Southwest testing has failed to achieve saleable concentrate grades, especially of the nickel concentrate. While application of the BCR flowsheet and especially the pyrrhotite rejection flowsheet is likely to address this, there is a risk that even these steps will not be enough to allow for creation of marketable products
- The design of the plant excludes in-circuit thickening – which proved the only way to make both the copper and nickel circuits work well in the ALS pilot plant. The assumption has been that the use of a lower dose of a xanthate more prone to degradation will obviate the need for this thickener, but this has not been proven
- The pyrrhotite rejection circuit has not been tested at pilot plant scale, and has thus not been proven to scale up to continuous operation
- Nickel recoveries are highly variable. While the density of sample coverage across the deposit is sufficient for a PFS-level evaluation of the Project, it is insufficient to ensure complete confidence in the predicted (especially nickel) metallurgy on an annual basis. Recovery predictions should be confirmed based on additional sampling and laboratory testwork, particularly focusing on achieving target nickel concentrate grades.

24.2.4 Mine Design (SRK)

- The overall labor count in the PFS is low compared to industry averages. If additional workers are needed to achieve the forecast production, labor costs will be higher than predicted by the PFS
- The productivities and equipment utilizations used are high compared to industry averages and were based upon the assumption that equipment is readily available for use when required without delay. Equipment availability can be a source of bottleneck should there be fluctuations or delays in activity requirements and could affect production
- The truck speeds assumed are relatively high for the underground environment, and would require managing traffic in a way that eliminates congestion and delays
- The Atlas Copco MT85 haul truck used does not have an established operating history in the underground mining industry and prototypes are currently in manufacturer testing. Accordingly, the estimated capital and operating costs used

are less reliable than estimates for other trucks. While the assumption has been that the trucks will be in service at the time of planned mine startup, there remains an uncertainty that the MT85 truck will be available in the market and operating at full availability in the underground environment

- ANFO was chosen as the preferred explosive in accordance with hydrogeological estimates. If the Project encounters more water in the mine than is currently expected, a bulk emulsion may need to be used which would result in higher blasting costs
- 20 ft of advance per development round has been used, which is relatively high even for the large size of the openings that have been designed for the Project. There is a risk that the Project will not be successful in pulling 20 ft rounds. If this happens, development productivity will be reduced and development costs will increase
- For post-pillar cut-and-fill mining, the PFS relies on a V-cut blast design which is not typically used in this type of rock environment. If this blast design proves to be ineffective, there is a risk of increased drilling cycle time as more blast holes will be required
- A high-quality, motivated and well-trained workforce has been assumed. The labor pay rates were estimated based on Iron Range-based mining operations. Miners will need to gain sufficient experience in pre-production years to be proficient in their tasks by the start of Year 1 as it has been assumed that miners will be as proficient at their tasks in Year 1 as in Year 10. If the Project is unsuccessful in attracting, training, and retaining such a workforce, the estimated productivities will not be achieved
- The labor and equipment complement for services and construction is highly optimized. There is some risk that it will not be adequate to support a 50,000 st/d fully-integrated mine with multiple mining methods and high traffic. If the Project has to employ more workers in the services and construction area, costs will increase
- Locally, the position of the orebodies is not well constrained. Local-scale fluctuations may require additional delineation drilling during operations, and may also require additional grade control efforts
- Current location of the dual declines portals is outside of the currently-planned concentrator area footprint and thus may require additional permitting or the relocation of the portal. Extending the length of the declines would negatively impact the cost estimates and construction/development schedule assumptions

- The larger slot size of 46 ft wide x 40 ft high with 34 ft wide x 40 ft tall pillars used in Tier 1 post-pillar cut-and-fill panels carries a higher risk than the originally recommended 40 ft x 40 ft slots. This increase in extraction accounts for approximately 10 Mt, or 2%, of the overall Mineral Reserve. The slot size should be further evaluated for the next phase of work.

24.2.5 Mine Design (AMEC)

- Unplanned dilution may be higher for post-pillar cut-and-fill than that assumed in the Mineral Resource and Mineral Reserve estimates
- There is a risk that the SRSF as designed may not have sufficient capacity to store additional waste material that may be generated during mine ramp-up that cannot be backfilled into mined-out areas.

24.2.6 Mine Design (Itasca)

- It is recommended that additional analyses be conducted in feasibility to better understand the evolution of stresses in the Tier 3 and 4 barriers so that rockbursting potential can be minimized
- A 1,700 ft x 1,700 ft (hydraulic radius = 425 ft) panel size represents a reasonable average for PFS planning purposes, but in general it appears that panels may need to be reduced in size where the orebody is thicker to reduce the volume of yielded ground that develops in the hanging wall
- There is a risk that the 46 ft x 40 ft slot and 33 ft x 33 ft pillar design in the Maturi Tier 1 mining areas may not be appropriate as additional geotechnical information becomes available. Changes to the assumptions could increase mining costs and reduce mining recoveries. This will require more detailed analysis during feasibility-level studies
- Crown pillar stability is sensitive to rock mass strength. A reduction in rock mass strength from the 30^{th} to 10^{th} percentile drops the factor of safety from a value in excess of three to a value of about 1.2. Further analysis of crown pillar stability should be based on additional information that would be generated during more detailed studies and consider estimates of rock mass strength local to the crown pillar volume
- It will be necessary during feasibility-level studies to better understand the risk of a hydraulic connection between the underground workings and surface water sources through the crown pillar. This will require development of a detailed structural model of the Maturi site, geotechnical core logging of the rock mass in the crown pillar region, geomechanical modeling to estimate mining-induced

changes to the rock mass permeability and fault aperture and associated hydrogeological modeling

- The potential for very large, structurally-controlled failures in the stope backs has yet to be assessed. If large wedges can detach from the backs of these stopes, they could result in significant air blasts and cause significant disruption to stope draw. Such analyses can be based on the development of discrete fracture networks that honor the measured joint orientations and spacings
- For the post-pillar cut-and-fill method, it will be necessary to develop proper pillar characterization approaches and re-entry protocols, particularly for deeper tiers
- For any mining project, there are always unknown rock mechanics and hydrogeological conditions that cannot be predicted ahead of actual mining. These unknown conditions, such as faulting, zones of weak rock, or zones of unanticipated water inflow, may only be discovered during mining and may require significant changes to the mining plan. While additional laboratory testing and characterization work should always continue through every stage of project development to help reduce uncertainty, it is never possible to carry out enough drilling/characterization work ahead of time to identify all of the potential risks. This may in turn have significant effects on cost, as relates for example to slope angles, recovery/dilution, caveability, pillar stability etc
- Even when features are known, it is often not possible to explicitly and accurately represent each and every feature likely to affect the behavior of a complex and highly heterogeneous solid system such as a rock mass on a large scale. Firstly, many of these features will never be fully identified and/or characterized, even after mining will have been completed. Many features that have the potential to affect ground stresses and the behavior of the rock mass to mining, such as local geological units and their contacts, geological discontinuities, zones of weaker/ altered rock, local changes in the pre-mining stress field, etc., are simply unknown at the time when even feasibility-level analyses are performed (and often do remain largely unknown throughout the life of the mine). All that can reasonably be expected at this stage is to capture the dominant behaviors and mechanisms of the system being analyzed, and to obtain reliable indications concerning the expected behavior of this system for various sets of conditions and the data collected.

24.2.7 Hydrogeological Considerations (Itasca)

- There may be potential to have a significant groundwater inflow into the underground workings due to mining activity intersecting an undetected water-bearing fault or joint set
- Mine dewatering activities may have an effect on surface waters such that there are reductions in surface water flows, or impacts on wetlands and phreatic surface

levels; there is currently insufficient groundwater information available to quantify any such potential impacts and detailed studies are required.

24.2.8 Infrastructure (Barr)

- Paste and/or dry-stack tailings management. Further consideration of paste and/or dry-stack tailings management is recommended to improve dam safety permitting feasibility for this facility as it will be located near local population centers. Paste or dry-stack tailings management methods would potentially: reduce the TSF footprint; reduce the Project-wide fresh water demand, and decrease earthwork requirements, thereby also potentially affecting related infrastructure locations, requirements and designs. Capital cost and some operating cost estimates may be higher or lower than current estimates if paste and/or dry-stack tailings management is pursued or is required by regulatory agencies
- Site-specific subsurface geotechnical conditions. Limited facility location-specific and utility corridor-specific geotechnical information is currently available. Capital cost and some operating cost estimates may be higher or lower than current estimates, depending on findings from future geotechnical explorations. Current earthwork (soil and rock) estimates are based on limited data, and assumptions have been made to estimate:
  - Relative quantities of soil excavation and backfill vs. rock excavation and backfill
  - Haul distances to and from suitable earthen material borrow source locations.
- Access to pipelines. The current designs at water crossings provide only limited access to tailings and concentrate pipelines via directional borings; consideration of man-way or vehicle tunnels as alternate to provide ready access for pipeline inspection and maintenance at water crossings is recommended
- Other potential Project risks include but are not limited to:
  - Water transport, storage and treatment system infrastructure sizing may change as further detailed water balance studies are performed
  - LNG supply and distribution sizing and configuration may change as the Project mine plan of operation is further refined
  - Power supply and distribution system sizing may change as Project configuration and process designs are further refined.

24.2.9 Process (AMEC)

- There is a potential risk that the SAG mill cannot consistently achieve the design throughput rates throughout the life of mine. This is due to potential limitations to throughput as a result of increased ore competency on occasions and uncertainties in the assumptions used to size the SAG mill. Additional year-by-year physical characterization testing should be performed in support of more detailed studies to quantify mill sizing

- The nickel flotation requires a sustained high level of control over water chemistry to ensure that target nickel concentrate grades can be achieved. By using a single process water stream, there is the potential for interactions between the flotation reagents used to recover copper and nickel in their respective circuits. If the water chemistry becomes unbalanced, unacceptable levels of nickel may report to the copper concentrate, and the nickel concentrate grades may fall short of target, reducing its marketability, and render it potentially unsaleable
- The current PFS flowsheet assumes that there is one process water source for both copper and nickel flotation circuits. The removal of two thickeners from the process flowsheet as a result of design changes during the PFS may affect the plant performance
- While pilot plant testwork has achieved nickel concentrate grades that appear marketable, the higher concentrate grade results have not been consistently replicated in laboratory testwork. Additional testwork should be contemplated and may include further cleaning laboratory and pilot nickel cleaning testwork, and investigation of froth crowding to support nickel recovery.

24.2.10 Infrastructure (Golder)

- Lack of site-specific subsurface geotechnical information. No geotechnical or hydrogeological explorations have been performed to evaluate facility specific subsurface conditions. Capital costs associated with required excavation depths, quantities of unsuitable materials to be removed, availability and suitability of borrow materials for use in construction of key facilities such as the TSF may be higher or lower than current estimates, depending on findings from future geotechnical explorations. Current earthwork (soil and rock) estimates are based on limited data, and assumptions have been made to estimate:
  - Relative quantities of soil excavation and backfill versus rock excavation and backfill
  - Availability and suitability of borrow materials for use in dam construction
  - Quantities of unsuitable materials to be removed from excavations
  - Haul distances to and from suitable earthen material borrow source locations
  - Excavation conditions (rock vs. soil excavation) for buried pipelines.
- Tailings, concentrate, and process water transport systems (pipelines)

  - The primary risk for the tailings transport system is the overall operational complexity of the system required by the mine plan and the transport distances between each facility
  - Testwork for tailings thickening, dewatering and rheology is based upon a single bulk tailings sample. Testwork in future phases should consider potential ranges in variability of the ore to confirm behavior observed in PFS studies. Variability in tailings properties from those observed in the PFS studies could result in changes to the thickening, transport and paste backfill designs
  - Long-term shutdown of pipelines during freezing conditions could result in a frozen, burst, or plugged pipeline
  - There is a risk of external corrosion from the elements or surrounding soils since there has been no investigation into the corrosive properties of the site soils
  - There is a risk of internal corrosion from angular tailings or concentrate particles. The corrosive nature of the tailings have not yet been investigated.

- Paste Backfill

  - The mine plan includes a configuration of paste backfill for long hole open stoping which results in long and steep unsupported paste backfill slopes at angles of approximately 45º. Future studies should be conducted to evaluate paste backfill performance, strength requirements (and associated binder content) under these conditions
  - Binder components (cement and fly ash) and the respective concentrations of each must be evaluated in future studies to confirm assumed paste backfill properties
  - With the high solids content feeding the disc filters there is potential of poor mixing in the disc filter vat which could reduce filtration efficiency. A review should be undertaken on the use of paddle agitators in lieu of e-ductors for tailings mixing
  - The standalone and remote location of the paste backfill plants will require dedicated personnel to operate the plants. Schedule and preventative maintenance planning will be particularly important to enable the plants to efficiently run
  - Future studies should be performed to assess the subsurface conditions at each of the paste backfill plant locations.

- Tailings Storage Facility
  - The availability of borrow materials required for the TSF dam construction is largely unknown due to the lack of site-specific geotechnical investigations and testing and could impact estimated construction costs
  - The cost to process borrow materials has been defined based on limited and general information available on the native soils. The volume of materials to be processed as well as the effort required for material processing could impact the estimated construction costs
  - The current lack of site-specific hydrogeological information could affect the final liner system requirements and design in order to demonstrate non-degradation of groundwater at the designated point of compliance
  - Operational management of the TSF reclaim pond is expected to be difficult due to extreme climate conditions. If the reclaim pond is allowed to drop too low, then the potential for dusting and particulate air contamination will be increased. If the reclaim pond is allowed to get too high, then seepage and stability conditions could be compromised
  - The water quality of the TSF reclaim pond has not yet been defined and could have an impact on pond management and/or the requirement for treatment and discharge.

24.2.11 Infrastructure (AMEC)

- A rigorous pipeline maintenance program will be required. The risks of an inadequate program could include environmental contamination if pipelines rupture; the next program phase should include a comprehensive review of inspection frequencies and maintenance requirements
- Pipelines installed in boreholes that will be drilled beneath surface water bodies may represent a maintenance challenge and in the next design phase, maintenance considerations should be further addressed
- Final location and costs of infrastructure may be influenced by requirements contained in permits that would be issued by federal, state, or local authorities.

24.2.12 Environmental and Permitting (AMEC)

- Water rights. There is some risk regarding the water supply necessary for Project operation. Risks are primarily related to the securing of permits to transfer water between basins, as discussed in Section 20.7.4. Should TMM not secure the appropriate permits for transfer of water between the basins, it may be necessary to modify the locations and/or operation of the various Project components, in order to comply with any inter-basin transfer prohibitions

- Refusal of permits. While TMM has been proactive in addressing likely areas of concern that the various permitting authorities may have, the issuance of necessary permits is not guaranteed. The permitting authorities could impose restrictions on the construction and/or operation of the Project that could result in substantial alterations to the proposed Project
- Hydrogeological study plan. Project assessment for hydrogeological purposes would include water quality, occurrence and hydraulic controls on groundwater flow, as well as predictive modeling to estimate the effects of mining activities on the groundwater resource. There is a risk that additional work may be required over that currently contemplated.
- Tailings, waste rock, and paste backfill characterization. TMM intends to submit work plans for tailings, backfill, and borrow material geochemical characterization, obtain MDNR's approval of those plans, with consultations with cooperating agencies as necessary, and implement those studies, as per regulatory requirements. This testing will likely include additional humidity cell testing (HCT) to assess the acid-generating potential and the mobilization of metal species. These tests require long run times and could have an impact on the projected development timeline
- Closure cost assumptions: The final reclamation and closure costs will be dependent on the final Project configuration, on conditions that may be imposed on TMM by regulatory authorities, on regulatory changes that may affect the Project during operations, and on any future bonding requirements. There is a risk that the actual costs will be higher than estimated in this financial model
- Long lead time for Draft EIS studies. The required Draft EIS will involve assessment of the presence or absence of possible effects on various natural resources. While TMM intends to conduct, or has already conducted baseline studies, the regulatory agencies will likely require further baseline studies to be performed. Many of these baseline studies, particularly for wildlife and vegetation surveys, will necessarily be carried out over a number of seasons, and may even require monitoring for more than a year. The length of the studies outlined in the various accepted assessment protocols could significantly impact the timeline for the Project
- Comments to Draft EIS. TMM can expect a large number of comments to the Draft EIS and draft permit applications. As part of the NEPA/MEPA process, the comments must be addressed before a Final EIS can be issued. Addressing the comments and, if necessary, conducting additional work, could significantly impact the timeline for the Project

- Permit and EIS appeals. Both the NEPA/MEPA process and the permit issuance process provide avenues for appeal of the final decisions.

24.2.13 Cost Estimates (AMEC)

- Unit costs for materials, consumables, and supplies may be higher than estimated
- Labor rates ($/hour) for construction and operating personnel may be higher than anticipated
- Labor productivity may be lower than anticipated
- The PUC is Minnesota's agency responsible for the regulation of certain power suppliers. They have final authority over the routing and capacity of power transmission facilities; and on how they are funded. Their decisions on the transmission system supplying TMM could have an impact on the Project capital and operating costs
- Design standards to meet permit requirements for all areas have the potential to impact capital and operating cost estimates
- Changes in design from that used in the PFS may result in increased capital and operating costs.

25.0 INTERPRETATION AND CONCLUSIONS

In the opinion of the QPs, the following interpretations and conclusions are appropriate to the review of data available for this Report:

25.1 Mineral Tenure, Surface Rights, Royalties

- Legal opinion supports that the mineral tenure and surface rights held by TMM in the areas for which Mineral Resources and Mineral Reserves are estimated is valid
- TMM holds permits, leases, options to purchase, fee title, and fee title to limited mineral rights, to about 27,000 acres of mineral rights across a patchwork of federal, state, and private mineral interests
- Duluth has currently identified 11 unique royalty combination schemes within the proposed mine plan area boundaries that will be payable to federal, state, and private parties
- Duluth should identify all surface and subsurface rights that are required for the Project and identify the holders of these rights. Duluth should subsequently take steps to acquire these rights; options for rights acquisitions could include options, land purchase, or leasing.

25.2 Geology and Mineralization

- The geological understanding of the settings, lithologies, and structural and alteration controls on mineralization in the different zones is sufficient to support estimation of Mineral Resources and Mineral Reserves. The geological knowledge of the area is also considered sufficiently acceptable to reliably inform preliminary mine planning
- The mineralization style and setting is well understood and can support declaration of Mineral Resources and Mineral Reserves.

25.3 Exploration, Drilling, Analysis and Data Verification

- Drilling has primarily consisted of core methods, due to the depth to mineralization. To 4 February 2014, a total of 1,339 pilot and wedge holes have been completed on the Project for a total of approximately 2,083,027 ft. Of this total, 765 holes (1,523,181 ft) were drilled at Maturi, and 71 holes (69,918 ft) at Maturi Southwest
- Core logging is adequate to support resource estimation and preliminary mine planning. Current core sampling conforms to industry-standard practices and is adequate to support Mineral Resource and Mineral Reserve estimation and preliminary mine planning

- Current collar surveying at Maturi, Maturi Southwest and Birch Lake utilizes industry-standard instrumentation and procedures and is adequate to support resource estimation and preliminary mine planning. Collar surveying at Spruce Road is believed to have been performed with theodolites and chains, which was industry-standard practice at the time the holes were drilled, but that has not been confirmed
- Legacy (pre-TMM) downhole surveying was done primarily with acid-tubes which do not provide adequate control on the azimuth of drill holes; AMEC has restricted blocks which are informed predominantly by legacy data to the Inferred category. Current TMM practice is to use gyroscopic tools that are unaffected by magnetic minerals in the rocks
- Density determinations at Maturi, Maturi Southwest, and Birch Lake were performed using standard procedures and are adequate to support resource estimation and preliminary mine planning. No density determinations have been performed at Spruce Road
- Recent sample preparation and assaying was performed at accredited commercial laboratories. Legacy samples were prepared and analyzed at a number of commercial and at least one company laboratory
- Quality assurance and quality control (QA/QC) for legacy samples is not documented. QA/QC for current samples is considered by AMEC to be adequate to support Mineral Resource and Mineral Reserve estimation and preliminary mine planning
- Sample security for legacy samples is not documented. Sample security for modern samples is considered to be sufficient to support Mineral Resource and Mineral Reserve estimation and preliminary mine planning
- The combined Maturi, Maturi Southwest, and Birch Lake database is adequate to support estimation of Mineral Resources without restriction
- AMEC considers that the Spruce Road database is adequate to support estimation of only Inferred Mineral Resources because the data are largely unverifiable.

25.4 Metallurgical Testwork

- Metallurgical testwork and associated analytical procedures were appropriate to the mineralization type, appropriate to establish the optimal processing routes, and were performed using samples that are typical of the mineralization styles found within the Project
- Samples selected for testing were representative of the various types and styles of mineralization. Samples were selected from a range of depths within the deposit.

Sufficient samples were taken so that tests were performed on sufficient sample mass

- Metallurgical forecasting is based on geometallurgical algorithms using the rougher database to link copper and nickel rougher recoveries to parameters in the resource model. The locked-cycle data are then used to predict how the recovered metal is distributed to the two final concentrates and the cleaner tails
- The mean copper recovery from tests using the basecase flowsheet was 85%, to a copper concentrate assaying on average 25% copper and 0.75% nickel. All tests yielded copper concentrates assaying above 24% copper, and only one test yielded a nickel grade in the copper concentrate above 1%, at 1.01%. The nickel metallurgy was also consistently better. On average the nickel circuit yielded a nickel product assaying 8.6% nickel, and 3.8% copper, at 57% nickel recovery. The mean overall copper recovery was 93.3%
- Combined precious metal recoveries to the combined concentrates averaged 78%, 61% and 74% for gold, platinum and palladium respectively
- As the pilot plant tended to yield cleaner copper and nickel concentrates than the respective locked-cycle tests, often at equal or better recoveries, the forecast also assumes some degree of cleaner performance enhancement in continuous mode over the results achieved in the Blue Coast locked-cycle program.

25.5 Mineral Resources

- Mineral Resources are stated on an in situ basis, and exclude application of planned and unplanned contact dilution and mining recovery factors
- Mineral resources have been estimated using ordinary kriging (OK) for the Maturi, Maturi Southwest, Birch Lake, and Spruce Road Cu–Ni–PGE deposits. These resources are estimated assuming underground mining as the preferred option. The Mineral Resource estimate for Spruce Road is a re-tabulation of a 2007 resource estimate produced by Scott Wilson RPA
- Mineral Resources for the Project, which have been estimated using core drill data, have been performed to industry-leading practices, and conform to the requirements of CIM (2014), and are reported on a 100% basis
- Combined Measured and Indicated Mineral Resources (including Mineral Reserves) for Maturi and Maturi Southwest total 1,233 Mst grading 0.58% Cu, 0.19% Ni, 0.147 ppm Pt, 0.334 ppm Pd, 0.080 ppm Au and 2.10 ppm Ag. Inferred Mineral Resources for Maturi and Maturi Southwest total 563 Mst grading 0.49% Cu, 0.16 % Ni, 0.134 ppm Pt, 0.305 ppm Pd, 0.068 ppm Au and 1.79 ppm Ag

- Indicated Mineral Resources at Birch Lake total 100 Mst grading 0.52% Cu, 0.16% Ni, 235 ppm Pt, 0.515 ppm Pd, and 0.115 ppm Au. Inferred Mineral Resources at Birch Lake total 239 Mst grading 0.46% Cu, 0.15% Ni, 0.18 ppm Pt, 0.370 ppm Pd and 0.087 ppm Au. Silver was not estimated
- Inferred Mineral Resources at Spruce Road total 480 Mst grading 0.43% Cu and 0.16% Ni
- Targets for additional exploration were defined at Maturi, Maturi Southwest and Birch Lake.

25.6 Mineral Reserves

- The total Proven and Probable Mineral Reserves for the Maturi and Maturi Southwest deposits are estimated at 527 Mst grading 0.59% Cu, 0.19% Ni, 0.154 ppm Pt, 0.350 ppm Pd, 0.090 ppm Au and 2.15 ppm Ag
- Factors which may affect the Mineral Reserve estimates include: commodity price and exchange rate assumptions; changes to the assumptions used to construct the NSR values used to constrain the estimates; metallurgical recovery and mine recovery and dilution assumptions; changes to the geotechnical and hydrogeological parameters used for stope and mine design; assumptions as to paste backfill strengths and quantities; changes to capital and operating cost estimates; changes to royalty payment assumptions; and variations to the permitting, operating or social license regime assumptions.

25.7 Geomechanical Considerations

- Horizontal in situ stresses are approximately two to 2.5 times the vertical stress. The stress regime is expected to lead to fairly significant shear stresses in the plane of the orebody
- Typical UCSi values for Maturi, based on the 30th percentiles of large scale domains range from 124 to 181 MPa and the 30th percentile GSI values range from 73 to 98. Typical 30th percentile UCSi values at Maturi Southwest range from 123 to 156 MPa and 30th percentile GSI values range from 65 to 76 on a large-scale domain basis
- Geotechnical conditions are sufficiently competent to allow the use of large scale and bulk mining methods.

25.8 Hydrogeological Considerations

- The estimated groundwater inflows, from numerical groundwater modeling, to the Maturi underground mine are estimated under the basecase scenario as 550 gal/min. The primary uncertainty in this value is the water transmitting capabilities of discontinuities that may be encountered within the mine. The estimated values for groundwater inflows do not include estimates for declines or other underground excavations to accommodate infrastructure. Estimated impacts to groundwater from a mining standpoint (e.g. drawdowns in the water table) are not well defined. Estimates of groundwater inflows to Maturi Southwest from a mining perspective have not been made

- No groundwater evaluations from an environmental perspective have been undertaken to date.

25.9 Ventilation

- Including the crusher and infrastructure areas, a total airflow of 3.25 million cfm will be required.

25.10 Mine Plan

- A NSR cutoff strategy was employed to maximize the optimal NPV for the deposits. The cutoff grade strategy prioritizes a higher NSR cutoff in the early years of the mine plan and uses a lower NSR cutoff in later years
- Mining will use a combination of post-pillar cut-and-fill and long-hole stoping methods, which are industry-standard methods, and have been successfully used in similar deposit types
- Underground mine plans are appropriately developed to maximize mining efficiencies, based on the current knowledge of geotechnical, hydrological, mining and processing information on the Project
- Scheduling was undertaken with the goal of providing 18.25 Mt/a of ROM ore to the process plant. To ramp-up as quickly as possible, three years of pre-production mining will be required to develop ramp systems, footwall drifts, stope accesses, ventilation raises, and other mine infrastructure
- It is expected that the mine will be able to achieve full ore production (i.e., 50,000 t/d) in Q2 of Year 1
- Achieving and maintaining the relatively high productivities that have been estimated for the Project will require constant vigilance on the part of management and supervisory personnel
- The equipment provisions include all primary and secondary equipment needed to meet the LOM production schedule requirements
- Production forecasts are achievable with the equipment selected. The MT85 haul truck is not currently commercially available. These trucks are currently being

tested in an Atlas Copco test mine and are expected to enter the broader market in a few years

- The predicted 30 year mine life is achievable based on the projected annual production rate and the Mineral Reserves estimated.

25.11 Recovery Plan

- The concentrator facilities proposed for the Project comprise a process plant with an ore capacity of 50,000 st/d, a single process line using SAG and ball milling with sequential copper and nickel flotation, high-rate tailings thickening, concentrate receiving system, filter plant, concentrate storage, and rail load-out
- The proposed process uses conventional technologies.

25.12 Infrastructure

- The Project would be subdivided into three non-contiguous primary sites consisting of the mine site, the concentrator site, and the TSF site
- Labor, materials, and concentrates would be transported to and from the Project sites by roads (state, county, and local) and via railroad. Supplies arriving via rail would be transferred into trucks and transported to the point of use
- Earthwork costs are based on geotechnical desktop studies, field reconnaissance and knowledge gained from other local infrastructure project development and are appropriate for this phase of project development
- Infrastructure sizing, configuration and functionality is appropriate for support of the proposed project
- Infrastructure designs consider project demands and functionality requirements, local facility design and configuration experience and expectations, and local climate extremes
- Rail and electric power system designs have been developed in coordination with input and assistance from local/regional power utility personnel and local/regional/national rail company personnel.

25.13 Marketing

- The customers for the TMM nickel concentrate will likely be nickel smelters in North America, Europe, Russia and China. China will be a potential market for TMM copper concentrate, along with other custom smelters in Europe and Asia
- The quality of TMM concentrate is suitable for the custom concentrate market and therefore would attract standard commercial terms, including benchmark copper

TC/RCs, refining charges for contained silver and gold, and payable metal percentages. No penalties for deleterious elements are expected

- No contracts are currently in place for any production from the Project.

25.14 Environmental, Permitting and Social License

- The Project is located within the area that was ceded to the United States by Lake Superior Chippewa Bands in 1854. Current land use in the region includes mining, forestry, and recreation on a mixture of private and public lands. The proposed Project area is in near proximity to the Boundary Waters Canoe Area Wilderness
- The proposed Project configuration places components in two separate water drainage basins separated by the Laurentian Continental Divide. The mine facilities, concentrator, and associated appurtenances are to be located in the Rainy River Basin, which drains north to Hudson Bay; the Tailings Storage Facility is to be located in the Lake Superior Basin. It is unknown at this time if the Project will be allowed to transfer water between the two basins
- The Project will be subject to regulatory review under both NEPA and MEPA, and under these frameworks, the Project is subject to review by multiple state and federal agencies
- The Project EIS and permitting process would require extensive co-ordination between TMM and the relevant federal and state agencies, as well as the tribes and local governments in the vicinity of the Project (e.g., Lake and St. Louis Counties and the cities of Ely and Babbitt). The EIS process, which must be completed prior to issuance of the State of Minnesota Permit to Operate, typically takes multiple years to complete. TMM has taken a proactive approach to development of baseline environmental data collection plans, in view of the EIS process. However, if the agencies determine that additional analysis would be needed, it would be within their authority to collect their own data and conduct the analysis for consideration in the environmental review process
- Considering the comments received by other polymetallic projects in the area during their EIS process, it is likely that the Project will receive numerous comments that must be addressed before the EIS can be finalized and permits can be issued. Depending on the complexity of the issues raised and the potential for the requirement for additional study, addressing the comments could significantly extend the EIS timeline. The Project will likely receive significant public scrutiny and possibly appeal during the EIS and permitting process
- There would be at least eight federal and state agencies involved in reviewing the Project. In particular, because of the patchwork of federal, state, and private mineral and surface properties involved, multiple agencies may have jurisdiction

over the same lands and/or closely related regulatory issues. These overlapping and complementary roles would present unique challenges for environmental review and permitting

- As certain agencies act not only as permitting authorities, but also as land and mineral owners, TMM would need to obtain federal and state agency approval for almost every activity undertaken for the Project. The issuance of permits for the various activities is not guaranteed
- The environmental study area would encompass currently proposed Project facilities including: surface lands above the underground mining areas and surface facilities (the mine site), the concentrator site, TSF and ancillary facilities, and utility corridors. The utility corridors would include roads, rail lines, power transmission lines, natural gas pipelines, tailing and concentrate pipelines, and water pipelines. It is likely that the environmental study area for certain resources (i.e., visual resources, noise, water quality, endangered and threatened species, etc.) will be expanded due to the close proximity of sensitive receptors. These sensitive receptors include, but are not limited to, tribal hunting and gathering rights, Boundary Waters Canoe Area Wilderness, and federally or State listed endangered or threatened species
- A number of desktop reviews of publicly-available data, together with some Project-specific studies, have been initiated in support of preliminary preparations for the Project EIS. Numerous additional studies will be required
- Liabilities associated with the mineral exploration program would be related to abandonment of boreholes and drill pad and road reclamation and historic mine features on the Project site. Reclamation bonds have been posted with the BLM and MnDOT for exploration related reclamation. Historical mine features on the Project site include two former bulk sample sites; an underground shaft and workings developed in 1968 and a surface excavation developed in 1974. TMM has reclamation responsibilities under applicable leases, and may be responsible for additional reclamation of the bulk samples sites if required; however, no specific reclamation has been requested by any agencies to TMM's knowledge and no reclamation plans had been developed by TMM at the Report effective date. Ongoing liabilities at the Dunka property, which is part of the TMM holdings, include permitted discharges from a sulfide-bearing rock stockpile and wetland treatment system, and permitted discharges of untreated mine pit water. As the Project is in very early stages of development, closure costs are at a conceptual level of detail. AMEC has included a conceptual closure cost allocation in the financial model of $210 million, based on benchmarking of similar projects. The closure cost estimate does not include any allocations for post-closure monitoring. The final closure cost estimate will depend on the MPO phase, when the Project

design is optimized, and will also depend on conditions that may be imposed on TMM during permitting

- To-date, likely stakeholders would be associated with local, state, or federal government elected bodies or regulatory agencies, state and local business interests, educational institutions, local community interests, or state NGOs
- A thorough socio-economic baseline analysis, analysis of projected and potential socio-economic impacts of the proposed Project, analysis of potential project alternatives (including a "no build" alternative), and a "cumulative impacts" analysis that will include identification and assessment of any known "regional development plans" or economically significant projects, will be required.

25.15 Capital Cost Estimates

- The capital cost estimate for the Project was developed by TMM’s Independent Engineer, with input from consultants for specific areas. The capital cost estimates are based on a combination of quotes, vendor pricing, and experiences with similar-sized operations. The costs were reported by TMM’s Independent Engineer at a prefeasibility level of accuracy where the estimate accuracy range is defined as +25%/-20% including contingency and are consistent with an AACE International (formerly Association for the Advancement of Cost Engineering or AACE) Class 4 Estimate
- AMEC performed a detailed estimate review of the PFS capital cost estimate. AMEC considered that the earthworks, excavation costs, concrete works, and contingencies were underestimated, and made an upward adjustment of approximately $156 M to cover these areas. This increased the capital cost estimate to $2,774.86 M
- A similar review was performed on the PFS sustaining capital estimate, and AMEC noted that the earthworks were underestimated, and made an upward adjustment of approximately $98 M
- When sustaining capital ($2,635.63 M) costs, including closure costs of $210 M, are incorporated, the total Project capital cost estimate as restated by AMEC is $5,410.49 M.

25.16 Operating Cost Estimates

- Mining operating costs over the LOM are estimated to total $6,615.4 M, and average $12.56/st mined. Mining costs do not include paste backfill costs
- The LOM underground infrastructure operating cost of $1.69/st was based on the LOM production schedule. Estimated costs over the LOM total $890 M

- The process plant operating cost estimate has a targeted accuracy of ±25%. Total process operating costs are estimated at approximately US$72.9 M/a, or $3.99/st milled. This total is split between concentrator costs of about $70 M/a (3.84/st milled) and concentrate filtration costs of approximately $2.9/a (0.16/st milled)
- Over the LOM, infrastructure operating costs are estimated to total $45.44 M/a or $2.49/st
- G&A costs, on a LOM basis, are estimated at $49.27 M/a, or $2.70/st.

25.17 Financial Analysis

- The after-tax NPV at an 8% discount rate over the estimated mine life is $753 million. The after-tax IRR is 11.4%. Payback of the initial capital investment is estimated to occur in 7.2 years after the start of production.

25.18 Sensitivity Analysis

- The Project is most sensitive to changes in the copper price, less sensitive to changes in operating costs, less sensitive again to changes in capital costs and least sensitive to changes in the nickel price.

25.19 Conclusions

- Based on the assumptions detailed in this Report, the Project shows positive a financial return and supports the declaration of Mineral Reserves
- Should the Duluth and TMM Boards make such a decision, there is sufficient support from the Report results for progression to a feasibility study.

26.0 RECOMMENDATIONS

26.1 Introduction

A two-phase work program is recommended to complete a MPO, feasibility study, and EIS, and to prepare associated permit applications.

The first phase will provide data support to allow TMM to complete the necessary testwork, engineering, and documentation to support the application for a mine plan of operation (MPO). The application for the MPO describes the configuration of the Project, so must be supported by sufficient engineering to adequately define all major variables, facility locations, and production rates. In the MPO, all land access must be defined in detail. The submission of the MPO will conclude the Phase 1 work program, and will trigger the EIS work program in Phase 2.

The second phase will build on Phase 1, and can be conducted in part concurrently with Phase 1. Phase 2 will provide engineering and data support to allow TMM to complete the required EIS and the feasibility study. The EIS and feasibility study will need to be undertaken concurrently, as the Project design as contemplated in the feasibility study must accommodate the recommendations arising out of the EIS; and the EIS must correctly reflect the proposed Project design.

The budget estimates are restricted to technical work, and no provision has been made in the estimates for items such as corporate overheads, land acquisition, legal and other consulting fees, additional work or program changes that may be required as a result of interactions with regulatory agencies, community and stakeholder consultations, or permit applications and acquisition. AMEC notes that for completion of the MPO phase, TMM must demonstrate that the company either holds the rights to all surface and mineral lands or has the acquisition of those rights in process.

26.2 Phase 1: Mine Plan of Operation

26.2.1 Introduction

The recommendations and budget estimates are presented as the program total, and then the QPs have provided more specific recommendations and budget breakdown estimates for future work in their areas of respective expertise. The QP estimates are included in the overall totals for Phase 1.

The MPO phase is intended to include, but will not be limited to, the following technical scopes of work:

- Commencement of environmental studies, including baseline data collection and impact predictions for hydrogeology and other environmental resources. TMM also intends to prepare additional environmental reports specifically to support the environmental review and permitting process

- Conducting a hydrogeological study. TMM intends through this work to develop an understanding of groundwater movement, connectivity, and chemistry within selected Project locations, and estimate potential hydrogeological impacts on the Project and the surrounding areas
- Geotechnical, hydrogeological and condemnation drilling. The geotechnical drilling would support additional geotechnical data collection associated with the proposed mine (declines, Maturi, Maturi Southwest, portals, ventilation raises, etc) as well as for improved understanding of the ground under proposed surface facilities (TSF, concentrator, shop, etc.). Hydrogeological wells will support data collection for purposes of establishing groundwater flow, predictive flow and chemical transport modeling. Condemnation drilling will be required to ensure that facilities and corridors are appropriately located
- Bulk sample collection and pilot plant program to gather information in support of process design, mine design, and increase geological understanding
- Engineering design. Studies will be completed to confirm the design and layout of surface infrastructure, including, but not limited to confirming locations for and , routing of utilities, pipelines, rail lines, access and project roads; further develop waste rock and TSF storage designs, traffic and logistics study, power study, backfill/binder assessments, and closure planning. Mine planning will be advanced to confirm the location and surface design of portals, raise collars, ventilation infrastructure, paste plants, access roads, and utility corridors.

It is likely that the technical component of the MPO will cost between $70 and $100 million to complete, with the approximate budget estimate allocation by key area being as follows:

- Engineering: $7–10 million
- Bulk sample and pilot plant program: $20–25 million
- Drilling: $8–13 million
- Environmental: $36–49 million.

26.2.2 AMEC

26.2.2.1 Geology and Mineral Resources

- Perform a conditional simulation study on the potential impact of small faults and undulations of the assay walls.

26.2.2.2 Mine Design

- Obtain a bulk sample from the orebody to confirm geological, geotechnical, mining, process, groundwater, environmental, and other assumptions. AMEC's preferred location for this sample would be from the existing historic Inco shaft that was excavated on the Maturi deposit, which could be re-opened to gain access to the orebody at a depth of 1,000 ft below surface
- Optimize cutoff grades for Tier 3 mining and Maturi Southwest areas
- Confirm locations of all portals, ventilation raises, and other surface accesses to the underground mine; drill pilot holes to confirm geotechnical conditions at these locations.
- Further evaluate geotechnical stability of the proposed 46 ft slots in Tier 1 area to confirm safety and stability of the openings
- Update the hydrogeological model of Maturi Southwest to incorporate underground mining assumptions so as to confirm pump calculations
- Refine paste strength testing for various combinations of cement and fly-ash binders
- Repeat and update water balance for the entire Project, as well as each individual water-usage or collection area. Particular attention should be paid to the construction phase and production ramp-up period
- Confirm make-up water quantities and source. Particular attention should be paid to the construction phase and production ramp-up period Update the marketing study for the nickel and copper concentrates, ensuring that by-product elements (including PGEs) are included in the concentrate marketing terms.

This work, including the bulk sampling, is estimated at $25–$35 M.

26.2.2.3 Environmental

- Initiate discussions with regulatory agencies on appropriate study protocols and the suitability of TMM collected baseline environmental data for inclusion in the environmental studies required for the EIS
- Develop and gain agency approval of the waste characterization work plan for development rock, tailings, and the paste backfill. Initiate the work plan to begin waste characterization testing
- Determine the most appropriate water source for facility operation and obtain water rights, as necessary

- Gain confirmation/authorization from appropriate agencies on the approval of TMM to perform inter-basin water transfers, based on the preferred water source selected and as required for efficient operation of the facility
- Initiate the first phase of the hydrogeological study to define surface/subsurface water occurrence and quality, in support of planned hydrogeological modeling

This work, including the hydrogeological drilling, is estimated at $38–$48 M.

26.2.3 Barr

It is recommended that the following additional studies be conducted to support future phases of Project infrastructure development:

- Geotechnical Exploration: an initial phase of geotechnical exploration at the proposed mine site, TSF site, concentrator site and utility corridors is recommended to form the basis for future optimization of infrastructure locations and earthwork cost estimating. Exploration should utilize industry-standard split spoon testing (SPT) and cone penetration testing (CPT) and should accommodate potentially difficult drilling conditions including cobbles and boulders in local glacial till soils. Groundwater levels should be confirmed at select locations where infrastructure elevations may intercept local groundwater. The recommended budget range for an initial phase of geotechnical drilling is $200,000 to $300,000 to accommodate field, laboratory, data interpretation and reporting; culminating in a report recommending general facility geotechnical siting and design criteria and recommendations for future detailed location-specific geotechnical exploration.
- Electric Power Supply: further evaluation of the opportunity to obtain competitive bids for electric power supply is recommended. This should include further technical and potentially legal review of PUC guidelines and constraints on power supply procurement. The recommended budget for further review is less than $100,000; culminating in a report recommending the path forward for permitting and competitive bidding of project power delivery by local electric power utilities and cooperatives
- Project Water Balance: further detailed evaluation of the Project water balance is recommended as a precursor to optimization of designs of the water transport pipeline and pumping systems, and water storage infrastructure, and to confirm that process water treatment is not required infrastructure. The recommended budget for further water balance evaluation is in the range of $100,000 to $200,000; culminating in an updated water balance study confirming the Project-wide water balance/demand, the preferred water supply strategy, and any Project water treatment plant and water discharge system infrastructure and capacity requirements.

- Project Logistics: further review of Project logistics is recommended to confirm project consumables and product types and quantities, preferred suppliers and selected customers, and modes of bulk materials transport. Specifically; local transport of materials by road, rail or pipeline should be confirmed with any planned changes considered relative to impact on Project infrastructure type, sizing and configuration and relative to impact on project NPV. The recommended budget for further Project logistics evaluation is on the order of $100,000; culminating in a report that confirms the logistics management approach already chosen and when possible, recommends alternate management approaches that improve reliability and/or economics.

26.2.4 Blue Coast

The Blue Coast recommendations for additional work include:

- Flowsheet enhancements
  - Integration of gravity PGE recovery from copper and/or nickel concentrates to improve pay for the precious metals. Gravity concentration testwork on copper and nickel concentrates at Blue Coast showed that precious metal recoveries of up to 65% were achievable to low mass pull products. The use of gravity recovery of gold and PGE may allow for the delivery of these precious metals to smelters where pay will be enhanced over the current copper and nickel smelters
  - Nickel cleaner optimization with focus on gangue rejection using 28L cell. Large volume testing is needed to create sufficient concentrate allow for good nickel cleaner testwork. All laboratory testwork to date has been at the 2 kg scale, making nickel cleaner work challenging. Testwork at this larger scale would investigate the effects of regrind, pulp dilution, multi-stage cleaning, collector selection and dose, and gangue depressants on nickel cleaner performance
  - Maturi Southwest flowsheet development (if different from Maturi) and confirmatory locked-cycle testing. Testwork is needed to bring the level of understanding of the Maturi Southwest deposit metallurgy to that of the main Maturi deposit
  - Pyrrhotite rejection scheme optimization and usage optimization. This process is far from optimized, and when it should be used for different feed types has not been optimized. It is likely that a better developed pyrrhotite rejection flowsheet might find broader application through the life of the mine.
  - Resolve issues with crusher work index data. The issues of two divergent datasets on crusher work index need to be resolved

  - Inert media primary grinding. The enhancement in recovery as a result of the use of inert media in regrinding was a relatively late technological breakthrough, and minimal work was conducted to investigate if it could be applied to the primary grind. Further work is needed here
  - Testing auxiliary collectors to enhance PGE recovery, especially with the pyrrhotite rejection circuit. The use of auxiliary collectors to enhance PGE recovery is widespread in the PGE industry and warrants investigation for the Project. Little has been done in this field to date
  - Investigate partial copper regrind. It seems likely that much of the copper rougher concentrate can be cleaned without any regrinding. If so, any associated pentlandite would also avoid regrinding. As fines losses constitute the greatest source of sulfide nickel loss in the circuit, this may limit nickel losses in the nickel circuit.
  - Alternatives to TETA: While industry experience indicates that it seems unlikely that an alternative system for Cu/Ni and pentlandite/pyrrhotite separation can be found, likely sensitivities associated with the use of TETA should be respected and all efforts should be made to investigate alternatives.

- Variability and geometallurgical data enhancements
  - Variability study on Maturi Southwest. A small variability program (small as the resource is not a major player in the NPV of the Project) should be conducted on Maturi South West once the flowsheet has been optimized for the resource. Further, a small program of blend tests with Maturi material should also be conducted.
  - Build more robust Ni recovery predictive system improving the existing geometallurgical parameters.
  - Further variability testing, using Blue Coast conditions
    - Batch roughing
    - Locked-cycle cleaning.

    The current rougher variability study uses a primary grind size that is coarser than the current flowsheet (~145 µm vs 120 µm), and incorrect collector and depressant doses. Past studies optimizing primary grind have indicated that ~1% better copper recoveries were possible at 120 µm vs 150 µm, while the heavy doses of depressant may have adversely have affected nickel recoveries. Further locked-cycle work may well also be needed if the optimization work described above leads to step improvements in overall metallurgical performance.
- Piloting and scale up confirmation

- Pilot plant confirmation of Blue Coast basecase and pyrrhotite rejection circuits: Piloting of the final optimized flowsheets would be needed to confirm their operability in continuous mode – while at the same time investigating the potential for upside due to the scale-up effects observed during the PFS program.
- Confirm whether an in-circuit thickener will be needed with Blue Coast flowsheet: With the lower collector doses of the shorter chain xanthate, the problems of poor selectivity when re-circulating the water back from the nickel tails thickener to the grinding circuit may be less severe, and may allow the circuit to operate without in-circuit thickening.

Blue Coast estimates that the likely budget required to complete the recommended work program is $6–8 million.

26.2.5 Golder

The following studies and tasks are recommended to further define certain aspects of the Project in the next phase of work.

Development of a site-specific geotechnical and hydrogeological subsurface investigation program should be initiated to characterize site conditions for TSF design, pipeline corridor design, and paste backfill plant design. The program should be structured to adequately characterize the depth to bedrock, the depth to the water table and regional hydrogeological conditions, geotechnical index properties, strength properties, consolidation properties, and permeability of the native soils and bedrock.

Following completion of the subsurface investigation program, all civil, geotechnical, and hydrogeological analyses should be re-evaluated with site specific information, including the following:

- Borrow availability and feasibility of soil processing to produce low permeability materials for soil liner, filter materials, and drain rock materials and to evaluate the associated cost estimate for producing these materials
- Borrow availability and quantity for general fill required for tailings dam construction and to confirm the assumed haul distance and cost estimates for TSF construction
- Site specific seepage analysis coupled with the appropriate transient effects from the TSF water balance including climate and tailings deposition in order to demonstrate adequate water quality requirements at the point of compliance
- Finite element based stability and deformation analyses for the TSF using the best available strength information for all construction materials in order to confirm the stability of the TSF and the integrity of the low permeability core under the worst-case stability conditions.

Golder estimates that the completion of the geotechnical and hydrogeological investigations may cost on the order of $300,000–$400,000.

When the ore recovery processes have been defined and finalized, Golder recommends that a suite of geotechnical, dewatering and rheological testing be completed to confirm tailings properties used in design. Once these properties have been confirmed and/or updated, the following tasks should be completed:

- Re-evaluate the selected thickening and filtration technology based on the range of anticipated tailings properties in order to confirm and/or update paste production and underground delivery costs and tailings management alternatives, including re-evaluation and comparison of slurry and dry stack tailings management alternatives for the TSF (estimated to be on the order of $100,000–$200,000)
- Re-evaluate the tailings transport configuration and requirements for pumping and piping based on the anticipated range tailings properties in order to confirm pumping and piping costs (estimated to be on the order of $100,000–$200,000)
- Understand the effect of flyash on UCS for paste backfill. Evaluate the effect of various binder types on the UCS results with full mill tailings. Fly ash and slag are commonly used as cement substitutes in concrete and paste backfill mixes (estimated to be on the order of $50,000)

26.2.6 Itasca

26.2.6.1 Hydrogeology

Itasca has the following recommendations in relation to the mine hydrological modeling for Maturi mine:

- Conduct a pumping test in the intact rock in the mine area during which hydraulic stresses propagate sufficiently far in both vertical and horizontal directions to measure the hydraulic properties of the rock at the depth of the proposed mine. Prior to the test, install multi-level pore-pressure monitoring systems that span various geologic units, and monitor the water levels before, during, and after conducting the pumping test
- Use the test results to estimate K and storage values for the intact bedrock along with the fractured bedrock
- Packer testing or pump tests should target fault zones to assess if they have enhanced or reduced hydraulic conductivity
- Once the additional hydrogeological data, detailed mine design specifications, and mining schedule become available, the groundwater flow model for the planned Maturi mine should be updated and re-calibrated so that updated predictions can

be made. A groundwater flow model for the Maturi Southwest deposit that is applicable to an underground mine should be constructed.

For the Maturi Southwest mine, all of the above items would have to be undertaken as well, including the construct of a groundwater flow model for an underground mine.

The estimated costs to undertake the field and modeling tasks for the Maturi site probably range from approximately $1 M to $1.5 M. The estimated costs for the additional data and analyses for Maturi Southwest may range from $700,000 to $1.3 M. Additional analyses would be required to refine these estimates.

26.2.6.2 Mine Design

Itasca has the following recommendations in relation to the mine design:

- Barrier pillars and panels
  - It is recommended that additional analyses be conducted in feasibility to better understand the evolution of stresses in the Tier 3 and Tier 4 barriers so that rockbursting potential can be minimized
- Crown pillar
  - Further analysis of crown pillar stability should be based on estimates of rock mass strength local to the crown pillar volume
  - It will be necessary at the feasibility level to better understand the risk of a hydraulic connection between the underground workings and surface water sources through the crown pillar. This will require development of a detailed structural model of the Maturi site, geotechnical core logging of the rock mass in the crown pillar region, geomechanical modeling to estimate mining-induced changes to the rock mass permeability and fault aperture and associated hydrogeological modeling. The work should commence during the MPO phase so as to be available for the feasibility study
  - Feasibility assessment of crown pillar stability should include study of additional failure modes (such as structurally controlled failure) and analysis of sensitivity to the in-situ stress conditions (orientation and magnitude) and pore pressures. The work should commence during the MPO phase so as to be available for the feasibility study.
- Long-hole stoping:
  - Studies should aim to conduct analyses in which explicit representation of hanging wall and end wall fracturing and detachment (collapse) is possible so that a more accurate estimate of the likely sloughage depths and associated dilution can be made for stopes
  - The potential for very large structurally controlled failures in the stope backs has yet to be assessed. If large wedges can detach from the backs of these

stopes, they could result in significant air blasts and cause significant disruption to stope draw. Such analyses can be based on the development of discrete fracture networks that honor the measured joint orientations and spacings

  - Stope mining in deeper tiers will be more sensitive to the increase in stresses as mining approaches an up-dip sill pillar. This should be examined closer in detailed design.
- Post-pillar cut-and-fill:
  - Perform additional analysis and study to confirm the safety and stability of the 46 foot slot (room) width included in the mine plan for Tier 1 mining.
  - The backs of the slots appear to be subject to much more shear damage in Tier 3 than in Tier 1 due to the higher induced stresses coming over the backs during up-dip advance. This should be examined with a more detailed model in feasibility (one that can explicitly account for stress-induced fracturing) to gain confidence in the design.
  - The NSR60 shell is quite a bit thinner at these depths and application of the method could be challenging in some areas (due to insufficient number of lifts for the post pillars to yield). This should be investigated further. Furthermore, at these deeper levels one would expect spalling modes of failure to be more prevalent. This puts additional emphasis on the need for alternate approaches to modeling in feasibility (e.g., with explicit representation of spalling-induced fractures) to improve confidence in application of this method at these depths, especially as additional data becomes available for these deeper levels.
- Remnant Mining
  - It will be necessary within feasibility to review the rock properties, stresses, backfilling properties and overall sequence of remnant mining to understand the local-scale issues that would dictate the overall success of this process.
- Underground Infrastructure Placement
  - Once the barrier pillars are put in their final positions within feasibility, the models can be re-run with the final configuration, and a ramp offset optimization can be performed. Some offsets may decrease and others may increase depending on the local conditions.
  - Further work will be required to verify a suitable location for Crusher #3.
- Data collection and testing
  - It will be necessary to drill additional characterization boreholes intersecting the deposit, especially in deeper portions of the orebody. These additional boreholes will be used to collect characterization data from core logging and acoustic televiewer (ATV) logging.

- Core from existing exploration boreholes intersecting the deposit will be core logged and associated boreholes will be ATV logged in order to provide additional characterization data.
- It will be necessary to drill characterization boreholes along the proposed decline alignments and at the proposed decline portal locations to identify possible major faults and provide data for rock mass quality assessments. These boreholes will be used to collect characterization data by core logging and acoustic televiewer logging.
- Insitu stress measurements will be required, especially in deeper portions of the deposit and in Maturi Southwest. Borehole breakout analyses will be performed using the ATV data and will supplement the results of the insitu stress measurements.
- Additional point load testing, uniaxial compressive strength testing and triaxial compression testing will be required to provide a better understanding of the distribution of intact material strengths throughout the site.
- The structural model and characterization of major structures will need refinement. Additional input data will be collected from surface outcrop mapping, test pits targeting major structures and borehole data.

The mine geotechnical program is estimated to cost $4.05 M. Although completed as part of the MPO phase, this is expected to provide sufficient data to support the completion of a feasibility study.

26.2.7 SRK

The SRK recommendations for additional work include:

- Proceed with a more detailed design/schedule including:
  - Complete mine design for remnant material
  - Design all accesses/development for long-hole stope areas
  - Complete an iGantt (or similar) schedule for the full project including remnant and Maturi Southwest.
- Complete a simulation model such as Arena (or similar) to determine equipment interference potential and update productivity/equipment assumptions accordingly
- Create ventilation models with varying raise sizes to optimize airflow/cost balance
- Long-lead time capital equipment should be ordered in a timely manner.

The program is estimated at between $300,000 to $500,000.

26.3 Phase 2

The MPO phase differs from the feasibility study phase in that the MPO represents only those tasks necessary to support initiation of an EIS and permitting. The MPO filing would trigger the preparation of the EIS.

The EIS work phase will include, but will not be limited to:

- Ongoing studies in support of the EIS
- State and federal consultations
- Preparation of draft permit applications
- Development of the draft and final EIS documents

The feasibility study phase, to be conducted in conjunction with the EIS, will partially overlap with the MPO work phase, as a portion of the technical data that was started in the MPO phase will continue to be collected on an ongoing basis.

The feasibility study work phase will include, but will not be limited to:

- Refinements to and optimizations of the mine, infrastructure and process plans and designs incorporated in the plans
- Evaluation of earthworks cut-and-fill quantities
- Material flows and simulations
- Additional metallurgical tests as required
- Geotechnical and hydrogeological considerations, including subsidence evaluations, and water balance refinements
- Power supply, rail, traffic and logistics considerations
- Closure considerations
- Considerations of opportunities identified during the PFS
- Considerations of risk mitigation for risks identified during the PFS
- Formal risk and opportunity analysis based on EIS results and feasibility design

It is likely that the technical component of the EIS and feasibility studies will cost between $57 and $74 million to complete, with the approximate budget estimate allocation by key area being as follows:

- Engineering: $6–8 million. This includes completion of the feasibility study and will incorporate refinements and optimizations to the mine plan, material flows and simulations, fill characterization, and evaluation of earthworks cut-and-fill quantities. The risks and opportunities identified in the PFS phase and in this

Report will be evaluated and the opportunities and mitigations identified included as appropriate into the feasibility design

- Ongoing pilot plant program: $5–10 million. Apart from continued pilot testing, work may include additional metallurgical assessments such as crusher increases, variability testing, and physical characterization testing. Value optimization of the process plant and associated infrastructure and consumables will be performed to support the feasibility design
- Drilling: $11–16 million. For support of feasibility studies, drilling is planned as a combination of underground and surface. Drilling is planned in support of better delineating the top surface of the orebodies and providing additional control for dilution estimates and for modeling. Drilling will also target selected structures to determine the likely displacement across linear features noted at surface that have been interpreted as faults. Additional geotechnical and site condemnation drilling is included. Infill drilling may be performed to support potential upgrades of classification of some of the Indicated material.
- Environmental: $35–40 million. An allocation has been made to support ongoing baseline and targeted environmental studies, since many of these studies have requirements for seasonal data collection. Preparation of the draft and final EIS documents are incorporated. AMEC notes that the estimate for the environmental portion is likely to be the upper end of the potential expenditure. The estimate allocation assumes that third-party data verification for the EIS of the MPO work phase will be required by the regulatory authorities.

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Appendix A

Tenure and Title Information.

Location of Federal Mining Leases

Lease Number	Permitting Or Leasing Authority	Surface Owner	Legal Description					Acres
			County	Section	TWP	Range	Section Subdivision	
MNES-01352	USA (BLM)	USA; South Kawishiwi Association LLC; Allete, Inc.	Lake	3	61	11	Lot 2	2,610.07 (Although MNES-01352 indicates a total of 2,610.07 acres, title searches indicate that the total acres held by USA under MNES-01352 are 2,444.12)
							SW¼SW¼	
							S½SE¼	
				5	61	11	Lots 1-2	
							S½NE¼	
							Lots 6-7	
							NE¼SW¼	
							S½SW¼	
							N½SE¼	
				6	61	11	Lot 13	
							Lots 22-24	
				7	61	11	Lots 1-4	
							Lots 9-10	
							Lot 12	
							Lots 15-16	
							Lot 19	
				8	61	11	Lot 2	
							Lot 6	
				9	61	11	All EXCEPT W½NW¼	
				18	61	11	Lot 2	
							Lot 7	
							Lot 9	
							Lots 12-20	
				19	61	11	Lots 2-5	
							Lots 7-8	
				27	62	11	SE¼SW¼	
				32	62	11	Lot 4	
				33	62	11	Lots 6-7	
				34	62	11	NW¼	
			St. Louis	25	61	12	Lot 2	
							SW¼SW¼	
MNES-01353	USA (BLM)	USA	Lake	19	62	10	All	2,254.71
				20	62	10	SW¼	
				29	62	10	N½	
				30	62	10	N½	
							Lot 3 (NW¼SW¼)	
				24	62	11	Lot 7	
							SE¼SW¼	
							S½SE¼	
				25	62	11	N½	
							W½SW¼ (undiv 1/2 interest)	
							NE¼SE¼	

Lease Number	Permitting Or Leasing Authority	Surface Owner	Legal Description					Acres
			County	Section	TWP	Range	Section Subdivision	
				26	62	11	S½NE¼	
							NE¼SW¼	
							E½SE¼ (undiv 1/2 interest)	

Terms of Federal Mining Leases

Lease Number	Expiry Date	Renewal	Initial Agreement Date	Royalties			Current Annual Carrying Costs		
				Base Royalty	Additional Royalty	Rental or Minimum Royalty	Rental	Minimum Royalty	Total
							2014	2014	2014
MNES-01352	Application for Extension filed on 10/24/2012	Initial 20-year period followed by three 10-year renewal terms, with the possibility of additional 10-year renewal terms under BLM regulations	1-Jun-66	4.5% of the "gross value" of Cu and Ni mined and shipped to the concentrating mill ("gross value" is defined as1/3 of the market prices of a quantity of fully-refined copper and of a quantity of fully-refined nickel equal to the respective quantities of unrefined copper and unrefined nickel contained in said minerals shipped to the concentrating mill). May be subject to escalation.	Additional royalty of 0.3% of the "gross value" of a quantity of fully-refined copper and of a quantity of fully-refined nickel equal to the respective quantities of unrefined copper and unrefined nickel contained in said minerals shipped to the concentrating mill. Additional royalty of 1% of gross value of "associated products" if value of such associated products exceeds 20% of aggregate market price as fully-refined metals of the quantity of copper and nickel contained in the minerals mined under the leases and shipped to the concentrating mill. Following a lease year in which the 1% additional royalty has been paid, if the gross value of such products exceeds 30% of said aggregate market price, the additional royalty shall be subject to renegotiation by The Company and the BLM. Additional (overriding) royalties are due to third parties under the Childers-Whiteside, Longyear, and ACNC agreements. See Sections 4.9.4, 4.9.5, and 4.9.6 of this report.	Rent of $1/acre/year until production; Min. Royalty of $10/acre/year	N/A	$26,110.00 Additional minimum royalties are made to third parties pursuant to the Childers /Whiteside Agreement and Longyear Agreement.	$26,110.00
MNES-01353	Application for Extension filed on 10/24/2012	Initial 20-year period followed by three 10-year renewal terms, with the possibility of additional 10-year renewal terms under BLM	1-Jun-66	4.5% of the "gross value" of Cu and Ni, mined and shipped to the concentrating mill ("gross value" is defined as 1/3 of the market prices of a quantity of fully-refined copper and of a quantity of fully-refined nickel equal to the	Additional royalty of 0.3% of the "gross value" of a quantity of fully-refined copper and of a quantity of fully-refined nickel equal to the respective quantities of unrefined copper and unrefined nickel contained in said minerals shipped to the concentrating mill.	$1.00 per acre per year until production Minimum royalty $10.00/acre/year	N/A	$22,550.00 Additional minimum royalties are made to third parties pursuant to the Childers /Whiteside Agreement	$22,550.00

Lease Number	Expiry Date	Renewal	Initial Agreement Date	Royalties			Current Annual Carrying Costs		
				Base Royalty	Additional Royalty	Rental or Minimum Royalty	Rental	Minimum Royalty	Total
							2014	2014	2014
		regulations		respective quantities of unrefined copper and unrefined nickel contained in said minerals shipped to the concentrating mill). May be subject to escalation.	Additional royalty of 1% of gross value of "associated products" if value of such associated products exceeds 20% of aggregate market price as fully-refined metals of the quantity of copper and nickel contained in the minerals mined under the leases and shipped to the concentrating mill. Following a lease year in which the 1% additional royalty has been paid, if the gross value of such products exceeds 30% of said aggregate market price, the additional royalty shall be subject to renegotiation by The Company and the BLM. Additional (overriding) royalties are due to third parties under the Childers-Whiteside, Longyear, and ACNC agreements. See Sections 4.9.4, 4.9.5, and 4.9.6 of this report.			and Longyear Agreement.	

Location of Federal Prospecting Permits

Permit Name	Permitting Or Leasing Authority	Surface Owner	Legal Description					Net Acres	Expiry Date
			County	Section	Twp	Range	Section Subdivision		
MNES 050652 (PRLA Applied for on 3/21/2013)	Bureau of Land Management	Federal	Lake	4	61	11	SE ¼ and N ½ -NE ¼ (Lots 1 and 2)	865.78	Unknown
				26	62	11	SE ¼ - SW ¼		
				26	62	11	W ½ -SE ¼		
				33	62	11	SW ¼ - SE ¼		
				33	62	11	NE ¼ -SE ¼		
				34	62	11	N ½ - SW ¼		
				34	62	11	W ½ - SE ¼		
				34	62	11	S ½ - NE ¼		
				35	62	11	N ¼ (being the N ½ - NW ¼ and the N ½ -NE ¼)		
				35	62	11	NW ¼ - SW ¼		
MNES 050846 (PRLA Applied for on 3/21/2013)	Bureau of Land Management	Federal	Lake	8	61	11	Lot 1 (includes NE ¼ - NW ¼)	178.50	Unknown
				8	61	11	Lot 3		
				8	61	11	Lot 4		
				8	61	11	NW ¼ -NE ¼		
				8	61	11	SW ¼ - NE ¼		
				8	61	11	NW ¼ - SE ¼		
MNES-53731 (Permit)	Bureau of Land Management	Federal	Lake	30	61	11	N ½ - NE ¼	572.27	12/1/2014
							SW ¼ - NE ¼ (Lot 22)		
							SE ¼ - NE ¼ (Lot 23)		

Permit Name	Permitting Or Leasing Authority	Surface Owner	Legal Description					Net Acres	Expiry Date
			County	Section	Twp	Range	Section Subdivision		
							NE ¼ - SE ¼ (Lot 26)		
							NW ¼ - SE ¼ (Lot 27)		
							S ½ -SE ¼ (Lot 44)		
				31	61	11	NE ¼		
							N ½ - SE ¼		
							SW ¼ - SE ¼ (Lot 14)		
							SE ¼ - SE ¼ (Lot 15)		
MNES-54387 (Permit)	Bureau of Land Management	Federal	Lake	17	61	11	Lot 3	1,293.47	11/1/2014
							Lot 6		
							Lot 7		
							Lot 8		
							Lot 9		
							Lot 10		
							SE ¼ - NE ¼		
							SE ¼ - SE ¼		
				19	61	11	Lots 1, 10, 11, 19		
				20	61	11	SE ¼ - SE¼		
							NE ¼ , Lots 1 and 2		
							S ½ , SE1/4NW1/4		
				29	61	11	E ½		
MNES-52446 (Permit)	Bureau of Land Management	Federal	Lake	19	61	11	Lot 17	708.51	9/16/2006
							Lot 18		
				30	61	11	Lot 19		
							Lot 20		
							Lot 21		
							Lot 28		
							Lot 29		
							Lot 30		
							Lot 31		
							Lot 32		
							Lot 33		
							Lot 34		
							Lot 35		
							Lot 36		
							Lot 37		
							Lot 38		
							Lot 39		
							Lot 40		
							Lot 41		
							Lot 42		
							Lot 43		
				31	61	11	Lot 1		
							Lot 2		
							Lot 3		
							Lot 4		

Permit Name	Permitting Or Leasing Authority	Surface Owner	Legal Description: County	Section	Twp	Range	Section Subdivision	Net Acres	Expiry Date
							Lot 5		
							Lot 6		
							Lot 7		
							Lot 8		
							Lot 9		
							Lot 10		
							Lot 11		
							Lot 12		
							Lot 13		
MNES-50264 (PRLA Applied on 12/13/2006)	Bureau of Land Management	USA	St. Louis	25	61	12	NE ¼ - NE ¼ (Lot 1), all unsurveyed islands	13.75	Unknown
MNES-55301 (Permit)	Bureau of Land Management	USA	Lake	18	61	11	Lot 11	88.70	12/1/2014
							Lot 21		
							Lot 22		
				19	61	11	Lot 9		
MNES-55302 (Permit)	Bureau of Land Management	USA	Lake					9.96	12/1/2014
				17	61	11	Lot 2 (50% US Mineral Interest)		
							Lot 4		
							Lot 5		
MNES-55305 (Permit)	Bureau of Land Management	USA	Lake	29	61	11	W 1/2	320	12/1/2014
MNES-55306 (Application)	Bureau of Land Management	USA	Lake	18	61	11	Lot 3	165.05	Unknown
							Lot 4		
							Lot 5		
							Lot 6		
							Lot 8		
MNES-54050 (Permit)	Bureau of Land Management	USA	Lake	5	61	11		0.50	11/1/2014
							Lot 5 (NE ¼ of SW ¼)		
MNES 054194 (Permit)	Bureau of Land Management	USA	Lake	4	60	11	All	1,780.20	11/1/2014
				5	60	11	All		
				8	60	11	E1/2; NW1/4; N1/2SW1/4; SE1/4SW1/4		
MNES 054195 (Permit)	Bureau of Land Management	USA	Lake	18-19	60	11	Section 18: Lots 2 thru 16, NE1/4, SE1/4 Section 19: All	2,033.70	11/1/2014
MNES 057601 (Application)	Bureau of Land Management	USA	Lake	30	61	11	Lots 2 and 18	60	Unknown
MNES 057681 (Application)	Bureau of Land Management	USA	Lake	18	61	11	Lot 10	37.45	Unknown
MNES 057765 (Application)	Bureau of Land Management	USA	Lake & St. Louis	6 and 7	61	11	Section 6: Lots 19, 20 - 21 Section 7: Lots 5-8, 17, 18	436.8	Unknown
				13	61	12	SE ¼ - NW ¼; SW ¼ - SW ¼; NW ¼ - SW ¼; SE ¼ - SW ¼		
MNES 054196 (Permit)	Bureau of Land Management	USA	Lake	6	60N	11W	1	947.80	11/1/2014
							2		

Permit Name	Permitting Or Leasing Authority	Surface Owner	Legal Description					Net Acres	Expiry Date
			County	Section	Twp	Range	Section Subdivision		
							3		
							4		
							5		
							6		
							7		
							8		
							12		
							13		
							21		
							22		
							S1/2-NE ¼		
							SE ¼		
							Lot 2		
							10		
				7	60N	11W	20		
							NE ¼		
							SE1/4SE ¼; SW1/4SE1/4 (50% US Mineral Interest)		

Terms of Federal Prospecting Permits and Permit Applications

Permit Name	Expiry Date	Date	Property Tax Paid By	Work Requirements		Annual Carrying Costs	Comments
				Yearly Work Commitment Required	Submission of Work Results	Rental 2014-2016	
MNES 050652 (PRLA Applied for on 3/21/2013)	Unknown	Permit 12/1/2001	N/A	No	Quarterly	$433.00	
MNES 050846 (PRLA Applied for on 3/21/2013)	Unknown	Permit 12/1/2001	N/A	No	Quarterly	$89.50	
MNES-53731 (Permit)	12/1/2014	Permit 12/1/2012	N/A	No	Per CFR	$286.50	
MNES-54387 (Permit)	11/1/2014	Permit 11/1/2012	N/A	No	Per CFR	$647.00	Former Permit #50163.
MNES-52446 (Permit)	9/16/2016	Permit 5/1/2006	N/A	No	Per CFR	$354.50	Former Permit # 49258.
MNES-50264 (PRLA Applied for on 12/13/2006)	Unknown	Permit 11/1/2000	N/A	No	Per CFR	$20.00	
MNES-55301 (Permit)	12/1/2014	Permit 12/1/2012	N/A	No	Per CFR	$44.50	
MNES-55302 (Permit)	12/1/2014	Permit 12/1/2012	N/A	No	Per CFR	$20.00	
MNES-55305 (Permit)	12/1/2014	Permit 12/1/2012	N/A	No	Per CFR	$160.00	
MNES-55306 (Application)	Unknown	Application 04/17/2008	N/A	N/A	N/A	N/A	
MNES-54050 (Permit)	11/1/2014	Permit 11/1/2012	N/A	No	Per CFR	$20.00	
MNES 054194 (Permit)	11/1/2014	Permit 11/1/2012	N/A	No	Per CFR	$890.50	
MNES 054195 (Permit)	11/1/2014	Permit 11/1/2012	N/A	No	Per CFR	$1,017.00	
MNES 054196 (Permit)	11/1/2014	Permit 11/1/2012	N/A	No	Per CFR	$474.00	
MNES 057765 (Application)	Unknown	Application 1/23/2013	N/A	N/A	N/A	N/A	
MNES 057681 (Application)	Unknown	Application 10/19/2012	N/A	N/A	N/A	N/A	

Permit Name	Expiry Date	Date	Property Tax Paid By	Work Requirements		Annual Carrying Costs	Comments
				Yearly Work Commitment Required	Submission of Work Results	Rental 2014-2016	
MNES 057601 (Application)	Unknown	Application 8/17/2012	N/A	N/A	N/A	N/A	

Note: Royalties on Federal Prospecting Permits will be negotiated at the time the permits are advanced to Preference Rights Leases.

Location of Minnesota State Mineral Leases

Lease Number	Mineral Lessor/Owner	Surface Owner	Legal Description					Net Acres	Expiry Date
			County	Section	Twp	Range	Section Subdivision		
MM-9132	State of Minnesota	USA	St. Louis	24	61	12	2/3 Interest in N ½ - SE ¼ 2/3 Interest in Lot 3 2/3 Interest in Lot 4	106.17	12/21/2038
MM-9455-N	State of Minnesota	MN (NE ¼ and NE ¼ of SE ¼) RendField Land Co. (NE-NW)	St. Louis	25	61	12	Mineral only SE ¼ - SW ¼	242	6/7/2040
				36	61	12	Surface and Mineral Rights NE ¼ NE ¼ - SE ¼ Mineral only That part of the NE ¼ - NW ¼ lying east of the current natural ordinary high water mark of the Birch Lake Reservoir, more or less, but excepting and excluding the lands, minerals, and mineral rights lying in and directly under the bed of the Birch Lake Reservoir below the current natural ordinary high water mark		
MM-9706-N	State of Minnesota	USA and MN	Lake	19	61	11	S ½ - S ½ - W ½	614.96	10/4/2047
				30			West ½		
				31			N ½ - N ½ - W ½		
			St. Louis	24	61	12	S ½ - SE ¼		
				25	61	12	E ½		
MM-9722-N	State of Minnesota	RendField Land Co. (Lots 3 and 4, SE ¼ of SW ¼) Allete (Part of NW ¼ of SW ¼ and part of SW ¼ of SW ¼); USA (Lot 5); Franconia (Part of NW ¼ of SW ¼ and part of SW ¼ of SW ¼)	St. Louis	25	61	12	Mineral only Lot 4, including the lands, minerals, and mineral rights lying in and directly under the bed of the Birch Lake Reservoir below the current natural ordinary water mark thereof, Section 25; and the lands, minerals, and mineral rights lying in and directly under the bed of the Birch Lake Reservoir below the current natural ordinary high water mark thereof, in SE ¼ -SW ¼, Section 25;	480.25	6/16/2049
		RendField Land Co. (Part of NW ¼ of NW ¼ and SW ¼ of NW ¼) Franconia (SE ¼ of NW ¼; Part of NW ¼ of NW ¼ and SW ¼ of NW ¼); MN (E ½)	St. Louis	36	61	12	Surface and Mineral rights NW ¼ - SE ¼; and S ½ - SE ¼; Mineral only NE ¼ -NW ¼, except that part lying east of the current natural ordinary high water mark of the Birch Lake Reservoir; and NW ¼ - NW ¼; and S ½ -NW ¼; and SW ¼ each of which include the lands, minerals, and mineral rights lying in and directly under the bed of the Birch Lake Reservoir below the current natural ordinary high water mark.		
MM-9724	State of Minnesota	USA & Cliffs	St. Louis	11	60	12	Mineral only	80	6/16/2049

Lease Number	Mineral Lessor/Owner	Surface Owner	Legal Description					Net Acres	Expiry Date
			County	Section	Twp	Range	Section Subdivision		
							SW ¼ - SW ¼ Mineral and Surface NW ¼ - NE ¼		
MM-9725	State of Minnesota	MN	St. Louis	12	60	12	N ½ - NE ¼ SE ¼ - NE ¼ SW ¼ - SW ¼ E ½ - SE ¼	240	6/16/2049
MM-9726	State of Minnesota	USA and MN	St. Louis	13	60	12	NW ¼ - NW ¼ NW ¼ - SW ¼ SE ¼ - SE ¼ NW ¼ - SE ¼	160	6/16/2049
MM-9727	State of Minnesota	MN and Franconia	St. Louis	14	60	12	NW ¼ - NE ¼ (Surface - MN) NE ¼ - NW ¼ (Surface - MN) S ½ - NW ¼ (Surface - MN) SE ¼ - SW ¼ (Surface - MN) NE ¼ - SE ¼ (Surface - MN) S ½ - SE ¼ (Surface - MN) NW ¼ - NW ¼ (Surface - Franconia)	360	6/16/2049
MM-9755	State of Minnesota	MN	Lake	3	61	11	Lots 1, 3 and 4 S½NE¼ S½ NW¼ N½SW¼ SE¼SW¼ N½SE¼	457.58	6/8/2050
MM-9756	State of Minnesota	MN	Lake	34	62	11	S½SW¼ E½SE¼	160	6/8/2050
MM-9764	State of Minnesota	MN	Lake	4	61	11	S½NE¼ Lots 3 - 4 S½NW¼ N½SW¼ SE¼SW¼	348.20	9/7/2050
MM-9812	State of Minnesota	USA	Lake	17	61	11	Undivided 1/2 Interest in NE ¼ - NE ¼	20	12/14/2050
MM-9813	State of Minnesota	County & private	Lake	18	61	11	Lot One, except north 1.063 ft and except south 250 ft; W ½ - NE ¼	80.30	12/14/2050
MM-9814	State of Minnesota	USA and MN	St. Louis	23	60	12	NW ¼ - NE ¼ (Mineral & Surface, if any) NE ¼ - SW ¼ (Mineral & Surface, if any) S ½ - SW ¼ (Mineral & Surface, if any) S ½ - NE ¼ (Mineral only)	240	12/14/2050
MM-9815	State of Minnesota	USA	St. Louis	24	61	12	Mineral only: undivided interest, as follows:	302.12	12/14/2050

Lease Number	Mineral Lessor/Owner	Surface Owner	Legal Description					Net Acres	Expiry Date
			County	Section	Twp	Range	Section Subdivision		
							8568/16128 in NE ¼ -NE ¼; 1065/16128 in NE ¼ - NE ¼; 5210/16128 in NE ¼ - NE ¼; 204120/435456 in NW ¼ - NE ¼; 32589/435456 in NW ¼ - NE ¼; 159426/435456 in NW ¼ -NE ¼; 204120/435456 in SW ¼ - NE ¼; 32589/435456 in SW ¼ - NE ¼; 159426/435456 in SW ¼ - NE ¼; 1065/16128 in SE ¼ - NE ¼; 5210/16128 in SE ¼ -NE ¼; 32589/435456 in NE ¼ -NW ¼; 159426/435456 in NE ¼ - NW ¼; 204120/435456 in SE ¼ -NW ¼; 32589/435456 in SE ¼ - NW ¼; 159426/435456 in SE ¼ - NW ¼; 9072/60480 in NE ¼ - SW ¼, ex easement 1.10 acres; 3291/60480 in NE ¼ - SW ¼, ex easement 1.10 acres; 43458/60480 in NE ¼ - SW ¼, ex easement 1.10 acres; 3291/60480 in NW ¼ - SW ¼; 52530/60480 in NW ¼ -SW ¼; 3291/60480 in Lot 1, ex easement 8.54 acres; 52530/60480 in Lot 1, ex easement 8.54 acres; 3291/60480 in Lot 2, ex easement 14.90 acres; 52530/60480 in Lot 2, ex easement 14.90 acres; -Mineral and surface: undivided interest, as follows: 25/40 in NE ¼ -SW ¼, ex easement, 25/40 in Lot 1, ex easement, 173/200 in Lot 2, ex easement.		
MM-9828	State of Minnesota	MN	Lake	5	61	11	SW¼SE¼	40	12/14/2050
MM-10011-N	State of Minnesota	TMM (SE 1/4 of SW 1/4 and SE 1/4 of NE 1/4); Erie Mining Company (SW 1/4 of SW 1/4); Allete, Inc. (SE 1/4 of NE 1/4); USA (NE 1/4 of SE 1/4)	St. Louis	2	60	12	S ½ -SW ¼ undivided 19/1080 of SE ¼ - NE ¼ undivided 19/1080 of NE ¼ -SE ¼	82.11	6/3/2054
MM-10012-N	State of Minnesota	Erie Mining Co. (SE 1/4 of NE 1/4); Northshore Mining Co. (SE 1/4 of SW 1/4 and SW 1/4 of SE 1/4); Allete, Inc. (SE 14 of SE 1/4)	St. Louis	10	60	12	SE ¼ - NE ¼ SE ¼ - SW ¼ S ½ -SE ¼	160	6/3/2054
MM-10141-N	State of Minnesota	TMM	St. Louis	8	61	11	Undivided 1/2 interest SE ¼ - NE ¼ Undivided 1/2 interest NE ¼ - SE ¼ Undivided 1/2 interest SE ¼ -SE ¼	60	10/16/2057
MM-10142-N	State of Minnesota	TMM	Lake	9	61	11	Undivided 1/2 interest SW ¼ - NW ¼	20	3/21/2057
MM-10144-N	State of Minnesota	Thomas Arendshorst and Sharon Arendshorst, husband and wife (Lot 5) and State of MN (SW 1/4 of SE 1/4)	Lake	8	61	11	Undivided 1/2 interest in Lot 5 and SW ¼ - SE ¼	34.50	10/16/2057

Lease Number	Mineral Lessor/Owner	Surface Owner	Legal Description					Net Acres	Expiry Date
			County	Section	Twp	Range	Section Subdivision		
MM-10146-N	State of Minnesota	USA and State of MN (Lots 17 & 18)	Lake	6	60	11	Lot 17	384.35	12/6/2057
							Lot 18		
							Lot 11		
							Lot 14		
							Lot 9		
							Lot 10		
							Lot 15		
							Lot 16		
							Lot 19		
							Lot 20		
MM-10147	State of Minnesota	MN	Lake	7	60	11	Lot 1 (surface and mineral)	599.30	12/6/2057
							Lot 9 (surface and mineral)		
							Lot 12 (surface and mineral)		
							Lot 3 (surface and mineral)		
							Lot 4 (surface and mineral)		
							Lot 7 (surface and mineral)		
							Lot 8 (surface and mineral)		
							Lot 5 (surface and mineral)		
							Lot 6 (surface and mineral)		
							Lot 13 (surface and mineral)		
							Lot 14 (surface and mineral)		
							Lot 15 (surface and mineral)		
							Lot 16 (surface and mineral)		
							Lot 11 (minerals only)		
							NE-SE (minerals only)		
							NW-SE (minerals only)		
							Undivided 1/2 interest SW ¼ - SE ¼ (minerals only)		
MM-10157	State of Minnesota	USA	Lake	18	60	11	Lot 1	40	12/6/2057
MM-10197-N	State of Minnesota	USA (E 1/2 of S 1/2 of W 1/2); RendField Land Co. (E 1/2 of W 1/2)	Lake	24	61	12	E ½ -S ½ -W ½	91.65	6/21/2058
				25	61	12	E ½ -W ½		

Lease Number	Mineral Lessor/Owner	Surface Owner	Legal Description					Net Acres	Expiry Date
			County	Section	Twp	Range	Section Subdivision		
MM-10206-N	State of Minnesota	USA	Lake	7	61	11	Lot 14	160	9/11/2058
							Lot 20		
							Lot 21		
							SW¼SE¼		
MM-10229	State of Minnesota	State of MN	St. Louis	6	59	12	Lot 10	48.75	3/12/2059

Terms of Minnesota State Leases

Lease Number	Expiry Date	Initial Agreement Date	Property Tax Paid By	Royalties				Work Requirements		Current Annual Carrying Costs	Possible Land Use Restrictions	Comments
				Base Royalty	Additional Royalty	Royalty Escalator Applies	Rental or Advance Minimum Royalty	Yearly Work Commitment Required	Submission of Work Results	2014–2016		
MM-9132	12/21/2038	12/21/1988	Lessee	3.50%	2.70%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum.	$2,654.25	Unknown	Subject to Minn. Stat. § 93.55, Subd. 2. Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-9455-N	6/7/2040	6/7/1990	Lessee	3.50%	2.60%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum	$6,050.00	Yes	Subject to special review of exploration plans by DNR. Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-	0/4/2047	10/4/1997	Lessee	3.50%	2.60%	Yes	Yes	No	Monthly production	$18,448.80	Yes	Lake bottom lease.

Lease Number	Expiry Date	Initial Agreement Date	Property Tax Paid By	Royalties				Work Requirements		Current Annual Carrying Costs	Possible Land Use Restrictions	Comments
				Base Royalty	Additional Royalty	Royalty Escalator Applies	Rental or Advance Minimum Royalty	Yearly Work Commitment Required	Submission of Work Results	2014–2016		
9706-N									reports required, exploration reports are required annually as a minimum			Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-9722-N	6/16/2049	6/16/1999	Lessee	3.95%	0.50%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum	$14,407.52	Unknown	Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-9724	6/16/2049	6/16/1999	Lessee	3.95%	0.23%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum.	$2,400.00	Yes	Excludes lake beds 60-1288P and 69.51P. Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-9725	6/16/2049	6/16/1999	Lessee	3.95%	0.16%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum	$7,200.00	Yes	Excludes lake beds 69-51P, 69-52P. Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.

Lease Number	Expiry Date	Initial Agreement Date	Property Tax Paid By	Royalties				Work Requirements		Current Annual Carrying Costs	Possible Land Use Restrictions	Comments
				Base Royalty	Additional Royalty	Royalty Escalator Applies	Rental or Advance Minimum Royalty	Yearly Work Commitment Required	Submission of Work Results	2014–2016		
MM-9726	6/16/2049	6/16/1999	Lessee	3.95%	0.11%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum.	$4,800.00	Yes	Excludes lake beds 69-51P, 69-52P. Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-9727	6/16/2049	6/16/1999	Lessee	3.95%	0.11%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum	$10,800	Yes	Excludes lake bed 69-51P. Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-9755-P	6/8/2050	6/8/2000	Lessee	3.95%	None	Yes	Yes	No	Monthly and quarterly production reports required, exploration reports are required annually as a minimum	$13,727.40	Yes	Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-9756-P	6/8/2050	6/8/2000	Lessee	3.95%	None	Yes	Yes	No	Monthly and quarterly production reports required, exploration reports are required annually as a minimum	$4,800.00	Yes	Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-9764	9/7/2050	9/7/2000	Lessee	3.95%	0.50%	Yes	Yes	No	Monthly and quarterly production reports required, exploration	$10,446.00	Yes	Lessee must satisfy the following conditions by the end of the 20th year of the Lease:

Lease Number	Expiry Date	Initial Agreement Date	Property Tax Paid By	Royalties				Work Requirements		Current Annual Carrying Costs	Possible Land Use Restrictions	Comments
				Base Royalty	Additional Royalty	Royalty Escalator Applies	Rental or Advance Minimum Royalty	Yearly Work Commitment Required	Submission of Work Results	2014–2016		
									reports are required annually as a minimum			(a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-9812	12/14/2050	12/14/2000	Lessee	3.95%	0.07%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum	$600.00	Yes	Subject to Minn. Stat. 93.55. Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-9813	12/14/2050	12/14/2000	Lessee	3.95%	0.07%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum	$2,409.00	Yes	Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-9814	12/14/2050	12/14/2000	Lessee	3.95%	0.23%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum	$7,200.00	Yes	Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-9815	12/14/2050	12/14/2000	Lessee	3.95%	0.04%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum	$9,063.60	Yes	Subj. to MN Statute 93.55, subd. 2 Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned

Lease Number	Expiry Date	Initial Agreement Date	Property Tax Paid By	Royalties				Work Requirements		Current Annual Carrying Costs	Possible Land Use Restrictions	Comments
				Base Royalty	Additional Royalty	Royalty Escalator Applies	Rental or Advance Minimum Royalty	Yearly Work Commitment Required	Submission of Work Results	2014–2016		
												royalty during any one single calendar year.
MM-9828	12/14/2050	12/14/2000	Lessee	3.95%	0.50%	Yes	Yes	No	Monthly and quarterly production reports required, exploration reports are required annually as a minimum	$1,200.00	Unknown	Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-10011-N	2054	6/3/2004	Lessee	3.95%	0.23%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum	$1,231.64 (2014) $2,155.40 (2015) $2,463.32 (2016)	Unknown	Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-10012-N	2054	6/3/2004	Lessee	3.95%	0.23%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum	$2,400.00 (2014) $4,200.00 (2015) $4,800.00 (2016)	Unknown	Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-10141-N	2057	3/21/2007	Lessee	3.95%	0.23%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum	$517.52	Unknown	Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-10142-N	2057	3/21/2007	Lessee	3.95%	0.23%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum	$300.00	Unknown	Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State

Lease Number	Expiry Date	Initial Agreement Date	Property Tax Paid By	Royalties				Work Requirements		Current Annual Carrying Costs	Possible Land Use Restrictions	Comments
				Base Royalty	Additional Royalty	Royalty Escalator Applies	Rental or Advance Minimum Royalty	Yearly Work Commitment Required	Submission of Work Results	2014–2016		
												at least $100,000.00 in earned royalty during any one single calendar year.
MM-10144-N	2057	10/16/2007	Lessee	3.95%	0.23%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum	$517.50	Unknown	Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-10146-N	2057	12/6/2007	Lessee	3.95%	0.66%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum	$5,765.25	Unknown	Lease excepts and excludes the lands, minerals, and mineral rights lying in and directly under the bed of Stony River below the natural ordinary high water mark thereof Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-10147	12/6/2057	12/6/2007	Lessee	3.95%	0.66%	Yes	Rent.	No	Monthly production reports required, exploration reports are required annually as a minimum	$9,289.52	Yes	Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-10157	12/6/2057	12/6/2007	Lessee	3.95%	0.66%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum	$600.00	Yes	Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State

Lease Number	Expiry Date	Initial Agreement Date	Property Tax Paid By	Royalties				Work Requirements		Current Annual Carrying Costs	Possible Land Use Restrictions	Comments
				Base Royalty	Additional Royalty	Royalty Escalator Applies	Rental or Advance Minimum Royalty	Yearly Work Commitment Required	Submission of Work Results	2014–2016		
												at least $100,000.00 in earned royalty during any one single calendar year.
MM-10197-N	6/21/2058	6/21/2008	Lessee	3.95%	0.50%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum	$1,145.63 (2014) $1,374.76 (2015-2016)	Yes	Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-10206-N	9/11/2058	9/11/2008	Lessee	3.95%	0.50%	Yes	Yes	No	Monthly and quarterly production reports required, exploration reports are required annually as a minimum	$2,400.00	Yes	Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.
MM-10229	3/12/2059	3/12/2009	Lessee	3.95%	0.57%	Yes	Yes	No	Monthly production reports required, exploration reports are required annually as a minimum	$243.76 (2014) $609.37 (2015) $731.24 (2016)	Yes	Lessee must satisfy the following conditions by the end of the 20th year of the Lease: (a) Be actively engaged in mining property in the subject or adjacent townships; and (b) Pay the State at least $100,000.00 in earned royalty during any one single calendar year.

TMM Private Leased Lands Locations

Lease Name	Mineral Lessor/Owner	Surface Owner	County	Legal Description				Net Acres	Expiration Date
				Section	Twp	Range	Section Subdivision		
RGGS Lease	RGGS Land & Minerals, Ltd., L.P.	US and	Lake	33	62	11	NW¼NW¼	561.75	1/1/2026, with provisions for extensions

Lease Name	Mineral Lessor/Owner	Surface Owner	County	Legal Description				Net Acres	Expiration Date
				Section	Twp	Range	Section Subdivision		
01/01/2006		Private					SE¼SW¼		
							NW¼SE¼		
							SE¼SE¼		
							Government Lot 5		
				35	62	11	SW¼NE¼		
							SE¼NE¼		
							SW¼NW¼		
							SE¼NW¼		
							NE¼SW¼		
							SW¼SW¼		
							SE¼SW¼		
							NW¼SE¼		
							SE¼SE¼		
Foster Lease 02/12/08	Goldie I. Foster, a/k/a Goldie I. Parker, a/k/a Goldie I. Mayer, and Walter B. Foster	US	Lake	4	61	11	Undivided 17/81 of SW¼SW¼	33.58	2/12/2028, with provisions for extensions
				5	61	11	Undivided 17/81 of SE¼SE¼		
				8	61	11	Undivided 17/81 of NE¼NE¼		
				9	61	11	Undivided 17/81 of NW¼NW¼		
Maki Lease 03/17/2007	Richard A. and Lavonne Maki; James K. and Linda Maki; Diane J. and Brian Manuszak; David Allen Maki; and Jean Maki	US	Lake	4	61	11	Undivided 4/9 in SW¼SW¼	71.11	3/17/2027, with provisions for extensions
				5	61	11	Undivided 4/9 in SE¼SE¼		
				8	61	11	Undivided 4/9 in NE¼NE¼		
				9	61	11	Undivided 4/9 in NW¼NW¼		
Saint Croix Lumber Co. Lease 12/15/2006	Saint Croix Lumber Company	TMM and State	Lake	9	61	11	Undivided ½ in SW¼ NW¼	118	12/15/2026, with provisions for extensions
				8	61	11	Undivided ½ in SE¼NE¼		
				8	61	11	Undivided ½ in NE¼SE¼		

Lease Name	Mineral Lessor/Owner	Surface Owner	County	Legal Description				Net Acres	Expiration Date
				Section	Twp	Range	Section Subdivision		
				8	61	11	Undivided ½ in SW1/4SE1/4		
				8	61	11	Undivided ½ in SE¼SE¼		
				8	61	11	Undivided ½ in SE¼SW¼ (Government Lot 6) Undivided ½ in NE1/4SW1/4		
St. Croix Lease 04/09/1987	St. Croix Lumber Company, Inc.	USA, State and Private	Lake	19	61	11	Undivided ½ in Lot 6	142.76	4/9/2037 or as long as commercial production (see remarks)
				19	61	11	Undivided ½ in Lot 12		
				19	61	11	Undivided ½ in Lot 13		
				19	61	11	Undivided ½ in Lot 14		
				19	61	11	Undivided ½ in Lot 15		
				19	61	11	Undivided ½ in Lot 16		
				30	61	11	Undivided ½ in Lot 2		
				17	61	11	Undivided ½ in NE ¼ - NE ¼		
				17	61	11	Undivided ½ in Lots 1 & 2		
				13	61	12	Undivided ½ in NE ¼ - NE ¼		
WF Mitchell Lease 06/01/1987	Wells Fargo Bank, MN	USA	St. Louis	24	61	12	Undivided 1/3 in N ½ - SE ¼	53.08	6/1/2037 or as long as commercial production. See remarks.
							Undivided 1/3 in Lot 3 (SW ¼ SE ¼)		
							Undivided 1/3 in Lot 4 (SE ¼- SE ¼)		
Rendrag Lease 7/31/1999	Rendrag, Inc.	USA, MPL and Cliffs	St. Louis	1	60	12	Lot 1	857.11	8/1/2049
							Lot 3		
							Lot 4		
							NW ¼ - SW ¼		
							SE ¼ - NW ¼		
							SW ¼ - NW ¼		
				2	60	12	Lot 2		
							Undivided ½ in NE ¼ -SE ¼		

Lease Name	Mineral Lessor/Owner	Surface Owner	County	Legal Description				Net Acres	Expiration Date
				Section	Twp	Range	Section Subdivision		
							NW ¼ - SE ¼		
							Undivided ½ in SE ¼- NE ¼		
							Undivided ½ in SE ¼ - SE ¼		
							SW ¼ - NE ¼		
							SW ¼ - SE ¼		
				11	60	12	NE ¼ - NE ¼		
							NE ¼ - SE ¼		
							NW ¼ - SE ¼		
							SE ¼ - NE ¼		
							SW ¼ - NE ¼		
				12	60	12	NE ¼ - SW ¼		
							NW ¼ - SW ¼		
							SE ¼ - NW ¼		
							SW ¼ - NW ¼		
				13	60	12	SW ¼ - NE ¼		
RM Bennett Lease 1/1/2000	R.M. Bennett Heirs L.P.	Mesabi Trust	St. Louis	15	60	12	NE ¼	160	12/31/2051
RGGS 08/30/2001	RGGS Lands & Minerals Ltd.	USA, MPL, Mesabi Trust, Cliffs, County	St. Louis	1	60	12	SW ¼ - NE ¼	1,560	8/29/2021; continues indefinitely if minerals are produced (see Section 4).
							SE ¼ - NE ¼		
							NE ¼ - SW ¼		
							SW ¼ - SW ¼		
							SE ¼ - SW ¼		
							NE ¼ - SE ¼		
							NW ¼ - SE ¼		
							SW ¼ - SE ¼		
							SE ¼ - SE ¼		

Lease Name	Mineral Lessor/Owner	Surface Owner	County	Legal Description				Net Acres	Expiration Date
				Section	Twp	Range	Section Subdivision		
			St. Louis	3	60	12	SE ¼ - SE ¼		
			St. Louis	11	60	12	NE ¼ - NW ¼		
							SW ¼ - NW ¼		
							SE ¼ - NW ¼		
							NE ¼ - SW ¼		
							NW ¼ - SW ¼		
							SE ¼ - SW ¼		
							SE ¼ - SE ¼		
			St. Louis	12	60	12	SW ¼ - NE ¼		
							NE ¼ - NW ¼		
							NW ¼ - NW ¼		
							SE ¼ - SW ¼		
							NW ¼ - SE ¼		
							SW ¼ - SE ¼		
			St. Louis	13	60	12	NE ¼ - NE ¼		
							NW ¼ - NE ¼		
							SE ¼ - NE ¼		
							NE ¼ - NW ¼		
							NE ¼ - SE ¼		
			St. Louis	14	60	12	NE ¼ - NE ¼		
			St. Louis	15	60	12	SW ¼ - NW ¼		
							NW ¼ - SW ¼		
							SW ¼ - SW ¼		
			St. Louis	23	60	12	NE ¼ - NE ¼		
							NE ¼ - NW ¼		
							NW ¼ - SW ¼		

Lease Name	Mineral Lessor/Owner	Surface Owner	County	Legal Description				Net Acres	Expiration Date
				Section	Twp	Range	Section Subdivision		
							NE ¼ - SE ¼		
							NW ¼ - SE ¼		
							SW ¼ - SE ¼		
							SE ¼ - SE ¼		
Johnson Lease 06/10/1986	J. Thomas Johnson, Mr. Darryl E. Coons, Duluth-Superior Area Community Foundation, Mr. Harold A. Knutson	USA	St. Louis	25	61	12	Lot 5 (SW ¼ - SE ¼)	62.75	6/09/2016; 06/09/2026 if merchantable ore discovered by 6/09/2016; indefinite if commercial production from the premises
							Lot 6 (SE ¼ - SE ¼)		
Longyear Mesaba 10/01/2000	Longyear Mesaba Co., dba LMC Minerals	USA, Cliffs, Mesabi Trust Company	St. Louis	1	60	12	NW ¼ - NE ¼	1,000	9/30/2050 (9/30/2059 if min. earned royalty paid. Section 27)
			St. Louis	2	60	12	NE ¼ - NE ¼		
							NW ¼ - SW ¼		
			St. Louis	3	60	12	NE ¼ - SE ¼		
			St. Louis	10	60	12	NE ¼ - NE ¼		
			St. Louis	11	60	12	NW ¼ - NE ¼		
							NW ¼ - NW ¼		
							SW ¼ - SE ¼		
			St. Louis	13	60	12	SW ¼ - NW ¼		
							SE ¼ - NW ¼		
							NE ¼ - SW ¼		
							SW ¼ - SW ¼		
							SE ¼ - SW ¼		
							SW ¼ - SE ¼		
			St. Louis	14	60	12	SW ¼ - NE ¼		
							SE ¼ - NE ¼		

Lease Name	Mineral Lessor/Owner	Surface Owner	County	Legal Description				Net Acres	Expiration Date
				Section	Twp	Range	Section Subdivision		
							NE ¼ - SW ¼		
							NW ¼ - SW ¼		
							SW ¼ - SW ¼		
							NW ¼ - SE ¼		
			St. Louis	15	60	12	SE ¼		
			St. Louis	23	60	12	NW ¼ - NW ¼		
Reed 04/15/2010	Dayton Reed	Franconia, USA and Private	Lake	7	61	11	1/7 interest in NW ¼ - SE ¼	24.04	4/15/2060
							1/7 interest in Lot 11		
							1/7 interest in Lot 13		
							1/7 interest in Lot 22		
				18	61	11	1/7 interest in Lot 1		
Carroll Lease 04/15/2011	Robert Carroll	Franconia, USA and Private	Lake	7	61	11	1/7 interest in NW ¼ - SE ¼	24.04	4/15/2060
							1/7 interest in Lot 11		
							1/7 interest in Lot 13		
							1/7 interest in Lot 22		
				18	61	11	1/7 interest in Lot 1		
Coons Lease 01/07/2013	Darryl E. Coons	State & USA	St. Louis	13	61	12	1/8 interest in NE ¼ - NE ¼	20.66	01/07/2043
							1/8 interest in SE ¼		
Johnson Lease 01/22/2013	Jean T. Johnson	State & USA	St. Louis	13	61	12	1/8 interest in NE ¼ - NE ¼	20.66	01/22/2043
							1/8 interest in SE ¼		
Knutson Lease 01/22/2013	Harold A. Knutson	State & USA	St. Louis	13	61	12	1/8 interest in NE ¼ - NE ¼	25.80	01/22/2043
							1/8 interest in SE ¼		
DSACF Lease06/27/2013	Duluth-Superior Area Community Foundation	State & USA	St. Louis	13	61	12	1/8 interest in NE ¼ - NE ¼	20.66	06/27/2043

Lease Name	Mineral Lessor/Owner	Surface Owner	County	Legal Description				Net Acres	Expiration Date
				Section	Twp	Range	Section Subdivision		
							1/8 interest in SE ¼		
Adolfson Lease 10/14/2013	Robert F. Adolfson (17/486 interest); Paula Moser, et vir. Ralph (17/486 interest); Sandra I. Stigar, et vir. Thomas (17/486 interest); Laura Richert (17/486 interest); Earl Hook, et ux. Lola (2/81 interest); Matthew Adolfson (17/486 interest); Robert Rodriguez (17/972 interest); & Kristina Metheny, et vir. John (17/972 interest)	USA	Lake	4	61	11	Undivided interest in SW¼SW¼ (see note next to lessor name)	37.54	10/14/2033, with provisions for extensions
				5			Undivided interest in SE¼SE¼ (see note next to lessor name)		
				8			Undivided interest in NE¼NE¼ (see note next to lessor name)		
				9			Undivided interest in NW¼NW¼ (see note next to lessor name)		

TMM Private Leased Lands Terms

Lease Name	Expiry Date	Renewal Notice (if option exists)	Initial Agreement Date	Base Royalty	Additional Royalty	Royalty Escalator Applies	Rental or Advance Min. Royalty	Yearly Work Commitment Required	Submission of Work Results	Rental 2014	Minimum Advance Royalty 2014	2015	2016	Comments
RGGS 01/01/2006	1/1/2026	7/5/2025 (180 days prior to expiration)	1/1/2006	5%	none	no	$200,000 per year after start of commercial production, payable in quarterly installments	$25,000 in years 1-2; $25,000/year after. Yearly report required, see Section 46.	Monthly: See Sections 15 and 16.	Rent: Greater of $10/acre or $7,500.00	Only if commercial production starts	Rent: Greater of $10/acre or $7,500.00	Rent: Greater of $10/acre or $7,500.00	Renewal terms for 5 years, max of four for a total extension to 01/01/2046. Rent ceases when commercial production begins.
Foster Lease 02/12/08	2/12/2028	08/16/2027 (180 days prior to expiration)	2/12/2008	3%	none	No	Yearly minimum royalty (Section 12); no rent	No	Monthly. See Sections 15 and 16.	None	$5,000.00	Adv. Royalty: $5,000.00	Adv. Royalty: $5,000.00	Renewal terms for 5 years, max of four for a total extension to 02/12/2048.
Maki Lease 03/17/2007	3/17/2027	9/18/2026 (180 days prior to expiration)	3/17/2007	3%	none	No	Yearly minimum royalty (Section 12); no rent	No	Monthly. See Sections 15 and 16.	None	$5,000.00	Adv. Royalty: $5,000.00	Adv. Royalty: $5,000.00	Renewal terms for 5 years, max of four for a total extension to 03/17/2047.
Saint Croix Lumber Co. Lease 12/15/2006	12/15/2026	6/18/2026 (180 days prior to expiration)	12/15/2006	3%	none	No	Yearly minimum royalty (see 2007 amendment to Section 12); no rent	No.	Monthly, See Sections 15 and 16.	None	$7,500.00	Adv. Royalty: $7,500.00	Adv. Royalty: $7,500.00	Renewal terms for 5 years, max of 4 for a total extension to 12/15/2046.
St. Croix Lease 04/09/1987	4/9/2012, 4/9/2037 or as long as commercial production (see remarks)	See remarks	4/9/1987	4% by underground; 5% by pit	none	Yes	Yes	No	Yearly reports. See Section 7.	None	$7,138.00 (before PPI adjustment per Section 5)	Adv. Royalty: $7,138.00 (before PPI adjustment per Section 5)	Adv. Royalty: $7,138.00 (before PPI adjustment per Section 5)	5% Royalty on open pit ores. Initial term automatically extends for an additional 25 years if merchantable ore is discovered, giving an expiration date of 2037. If there is commercial production on the premises by the expiration date in 2037, the lease extends as long as there is commercial production.

Lease Name	Expiry Date	Renewal Notice (if option exists)	Initial Agreement Date	Base Royalty	Additional Royalty	Royalty Escalator Applies	Rental or Advance Min. Royalty	Yearly Work Commitment Required	Submission of Work Results	Rental 2014	Minimum Advance Royalty 2014	2015	2016	Comments
WF Mitchell 06/01/1987	6/1/2012; 6/1/2037 or as long as commercial production. See remarks.	See remarks.	6/1/1987	4% by underground; 5% by pit	none	Yes	Yes	No	Yearly reports. See section 7.	None	Adv. Royalty: $2,654.00 (before PPI adjustment per Section 5)	Adv. Royalty: $2,654.00 (before PPI adjustment per Section 5)	Adv. Royalty: $2,654.00 (before PPI adjustment per Section 5)	5% Royalty on open pit ores. Initial term automatically extends for an additional 25 years if merchantable ore is discovered, giving an expiration date of 2037. If there is commercial production on the premises by the expiration date in 2037, the lease extends as long as there is commercial production.
Rendrag 7/31/1999	8/1/2049		7/31/1999	3.95% (Varies by Net Return Value, See Section 6)	0.1525% of Net Return Value (See Section 6 a)	Yes.	Rent	$100,000	Quarterly report of ore removed (Section 9); Detailed monthly report (Section 13); Additional monthly and annual reports with samples (Section 14)	$25,713.04	None	Rent: $25,713.04	Rent: $25,713.04	Work commitment can be met on adjacent lands. Lessor has the right to cancel in years 26 and 36 if no development or production.
RM Bennett Lease 1/1/2001	12/31/2051		1/1/2001	3.95% (varies by PPI, see Section 7)	0.23%	Yes	Yes (Section 5)	$50,000	Quarterly reports with royalty (section 11); Monthly reports (section 14); Additional annual and monthly reports (Section 15)	N/A	$1,600.00 (before adjustment per Section 5)	Adv. Royalty: $4,000.00 (before adjustment per Section 5)	Adv. Royalty: $4,000.00 (before adjustment per Section 5)	Lessor can terminate if minimum earned royalty payments not made by 2024, 2039, 2049 or 2059. Work commitment can be met on adjacent lands.
RGGS Lease 08/30/2001	8/29/2021; continues indefinitely if minerals are produced (see Section 4).	N/A	8/30/2001	5.00%	none	No	Yes (section 7)	$25,000	Monthly (section 11); Annual report re exploration (Section 12); Annual minimum work commitment report (section 8).	$39,000.00	N/A	Rent: $39,000.00	Rent: $39,000.00	20-yr term and so long thereafter as mining is occurring on a deposit wholly or partially leased lands. This lease is between United States Steel and LEM. United States Steel is now RGGS.

Lease Name	Expiry Date	Renewal Notice (if option exists)	Initial Agreement Date	Base Royalty	Additional Royalty	Royalty Escalator Applies	Rental or Advance Min. Royalty	Yearly Work Commitment Required	Submission of Work Results	Rental 2014	Minimum Advance Royalty 2014	2015	2016	Comments
Johnson Lease 06/10/1986	6/09/2016; 06/09/2026 if merchantable ore discovered by 6/09/2016; indefinite if commercial production from the premises	N/A	6/10/1986	Precious mineral royalty varies by depth of minerals; see remarks	none	No	Yes, Adjusted per PPI (Section 5 and 1998 Amendment)	No	Annual Reports (Section 7)	N/A	$6,275.00 (Before PPI Adjustment)	Adv. Royalty: $6,275.00 (Before PPI Adjustment)	Adv. Royalty: $12,550.00 (Before PPI Adjustment)	Precious minerals (gold, silver, platinum group) and uranium royalties vary by depth: 7% if within 1,000 ft of the surface; 6% if between 1,000 and 2,000 ft of the surface and 5% if greater than 2,000 ft. USFS ownership of surface restricts use without notification and approval. Commercial production" for purpose of lease duration requires production of at least 10,000 short tons per annum.
Longyear Mesaba Lease 10/01/2000	9/30/2050 (9/30/2059 if min. earned royalty paid. Section 27)	N/A	10/1/2000	3.95% (adjusted. See section 7 and Exhibit A)	0.23%	Yes	Advanced Minimum Royalty	$100,000 per year	Quarterly statements (section 11); monthly reports (section 14) and additional monthly and annual reports (section 15)	N/A	$25,000.00 (Adjusted for CPI-U. See section 5)	Adv. Royalty: $25,000.00 (Adjusted for CPI-U. See section 5)	Adv. Royalty: $25,000.00 (Adjusted for CPI-U. See section 5)	Work commitment can be met by work on adjoining lands. Conflict as to ownership with state of NWNE Section 11-60-12. Lessor has right to cancel in years 26 and 36 if no development or production occurring.
Reed Lease 04/15/2010	4/15/2060	N/A	4/15/2010	3.95% (adjusted per section 6)	0.25%	Yes	Rent: $750.00	No	Monthly reports (section 13); Quarterly reports with royalty (Section 10); Additional monthly and annual reports (section 14)	$240.40	N/A	Rent: $360.60	Rent: $360.60	

Lease Name	Expiry Date	Renewal Notice (if option exists)	Initial Agreement Date	Base Royalty	Additional Royalty	Royalty Escalator Applies	Rental or Advance Min. Royalty	Yearly Work Commitment Required	Submission of Work Results	Rental 2014	Minimum Advance Royalty 2014	2015	2016	Comments
Carroll Lease 04/15/2011	4/15/2060	N/A	4/15/2010	3.95% (adjusted per section 6)	0.25%	Yes	Rent: $750.00	No	Monthly reports (section 13); Quarterly reports with royalty (Section 10); Additional monthly and annual reports (section 14)	$240.40	N/A	Rent: $360.60	Rent: $360.60	Legal description of Lot 22 is unclear.
Coons Lease 01/07/2013	01/07/2043	N/A	01/07/2013	Base metals 5% Net Return open pit (4% underground). Precious mineral royalty varies by depth of minerals; see remarks	None	No	Yes, Adjusted per PPI beg. On 6th anniv. date	No	Quarterly reports with royalty payments (Section 3)	N/A	$206.59	$206.60	$413.20	Precious minerals (gold, silver, platinum group) and uranium royalties vary by depth: 7% if within 1,000 ft of the surface; 6% if between 1,000 and 2,000 ft of the surface and 5% if greater than 2,000 ft.
Johnson Lease 01/22/2013	01/22/2043	N/A	01/22/2013	Base metals 5% Net Return open pit (4% underground). Precious mineral royalty varies by depth of minerals; see remarks	None	No	Yes, Adjusted per PPI beg. On 6th anniv. date	No	Quarterly reports with royalty payments (Section 3)	N/A	$206.60	$206.60	$413.20	Precious minerals (gold, silver, platinum group) and uranium royalties vary by depth: 7% if within 1,000 ft of the surface; 6% if between 1,000 and 2,000 ft of the surface and 5% if greater than 2,000 ft.
Knutson Lease 01/22/2013	01/22/2043	N/A	01/22/2013	Base metals 5% Net Return open pit (4% underground). Precious mineral royalty varies by depth of minerals; see remarks	None	No	Yes, Adjusted per PPI beg. On 6th anniv. date	No	Quarterly reports with royalty payments (Section 3)	N/A	$258.00	$258.00	$516.00	Precious minerals (gold, silver, platinum group) and uranium royalties vary by depth: 7% if within 1,000 ft of the surface; 6% if between 1,000 and 2,000 ft of the surface and 5% if greater than 2,000 ft.

Lease Name	Expiry Date	Renewal Notice (if option exists)	Initial Agreement Date	Base Royalty	Additional Royalty	Royalty Escalator Applies	Rental or Advance Min. Royalty	Yearly Work Commitment Required	Submission of Work Results	Rental 2014	Minimum Advance Royalty 2014	2015	2016	Comments
DSACF Lease 06/27/2013	06/27/2043	N/A	06/27/2013	Base metals 5% Net Return open pit (4% underground). Precious mineral royalty varies by depth of minerals; see remarks	None	No	Yes, Adjusted per PPI beg. On 6th anniv. date	No	Quarterly reports with royalty payments (Section 3)	N/A	$206.60	$206.60	$413.20	Precious minerals (gold, silver, platinum group) and uranium royalties vary by depth: 7% if within 1,000 ft of the surface; 6% if between 1,000 and 2,000 ft of the surface and 5% if greater than 2,000 ft.
Adolfson Lease 10/14/2013	10/14/2033	7/16/2033 (90 days prior to expiration)	10/14/2013	3% of the Net Return Values	None	No	Yes, royalty.	No	Monthly reports during active production. See Sections 16 and 17.	N/A	$1,000.00	$2,500.00	$2,500.00	Renewal terms possible for 5 years, max of four for a total extension to 10/14/2053.

Minnesota Power Fee Lands

Township	Range	Section	Parcel	Exceptions	Acreage
60	12	2	Lot 1		39.16
60	12	2	SE-NE-		39.16
60	12	2	NW-SE-		40.00
60	12	2	SE-SW-		40.00
60	12	2	SW-SE/		40.00
60	12	3	Lot 4	also described as NW-NW	29.35
60	12	3	SW-NW		40.0
60	12	3	SE-NW	Except Cliffs ownership	5.1
60	12	3	NW-SW	except Cliffs ownership	34.4
60	12	3	NE-SW/	except Cliffs ownership	0.2
60	12	3	SW-SW/	except Cliffs ownership	9.0
61	12	25	NW-SW	except Birch Portage land plat and Dunka overflow	5.7
61	12	25	SW-SW	except Dunka overflow	5.0
61	12	26	Lot 1	except Birch Portage land plat	10.60
61	12	26	SW-NE	except Birch Portage land plat	21.67
61	12	26	NW-SE		40.0
61	12	26	NE-SE	except Birch Portage land plat	38.85
61	12	26	SW-SE	Undivided ¼ except for Cliffs	35.00
61	12	26	SE-NW	Except platted portion	36.37
61	12	26	NE-SW		40.0
61	12	26	SW-SW		40.0
61	12	26	SE-SW		40.0
61	12	34	NE-NE	except Cliffs ownership	39.50
61	12	34	SW-NE	except Cliffs ownership	39.50
61	12	34	SE-NE	except Cliffs ownership	20.2
61	12	34	NE-SW'	except Cliffs ownership	37.5
61	12	34	SE-SW	except Cliffs ownership	19.2
61	12	34	NW-SE	except Cliffs ownership	10.4
61	12	35	NE-SE	except Cliffs' NW ¼	30.00
61	12	35	SE-SE		40.00
61	12	35	NW-NW	except Cliffs ownership	9.1
61	12	35	NE-NW	except Cliffs ownership	8.8
61	12	36	NW-NW	except Dunka overflow	11.00
61	12	36	SW-NW	except Dunka overflow	38.00
61	12	36	SE-NW		40.00
61	12	36	NW-SW		40.00
61	12	36	NE-SW		40.00
61	12	36	SW-SW		40.00
61	12	36	SE-SW		40.00
60	12	10	NE-SE		40.00
60	12	10	SE-SE		40.00
60	12	11	NE-NW		40.00
60	12	11	SW-NW		40.00
60	12	11	SE-SW		40.00
60	12	11	N1/2 of NW-SW		20.00
60	12	14	NW-NW		38.49
Total					*1,391.25*

Option Agreements

Potlatch Option							
Subquarter	**Quarter**	**Section**	**Township**	**Range**	**Acres**	**Abstract/Torrens**	**PIN**
NE	NE	10	60	13	40	Abstract	105-0080-1460
NW	NE	10	60	13	40	Abstract	105-0080-1470
SE	NE	10	60	13	40	Abstract.	105-0080-1490
NW	NE	11	60	13	40	Abstract	105-0080-1630
SW	NE	11	60	l3	40	Abstract	105-0080-1640
SE	NW	11	60	13	40	Abstract	105-0080-1690
NE	SW	11	60	13	40	Abstract	105-0080-1700
NW	SW	11	60	13	40	Abstract	105-0080-1710
SW	SW	11	60	13	40	Abstract	105-0080-1720
NE	SE	11	60	13	40	Abstract	105-0080-1740

Potlatch Option

Subquarter	Quarter	Section	Township	Range	Acres	Abstract/Torrens	PIN
NW	SE	11	60	13	40	Abstract	105-0080-1750
SW	SE	11	60	13	40	Abstract	105-0080-1760
SE	SE	11	60	13	40	Abstract	105-0080-1770
NE	NE	14	60	13	40	Abstract	105-0080-2100
NW	NE	14	60	13	40	Abstract	105-0080-2110
NW	NW	14	60	13	40	Abstract	105-0080-2150
SW	NW	14	60	13	40	Abstract	105-0080-2160
SE	NW	14	60	13	40	Abstract	105-0080-2170
SW	NE	15	60	13	40	Abstract	105-0080-2280
SE	NE	15	60	13	40	Abstract	105-0080-2290
All	SW	I15	60	13	160	Cert. 318000	105-0080-2340
N ½	SE	15	60	13	80	Cert. 318001	105-0080-2380
SW	SW	17	60	13	40	Cert. 318001	105-00802680
SE	SW	17	60	13	40	Cert. 318001	105-0080-2690
SW	SE	17	60	13	40	Cert. 318001	105-0080-2720
E ½ of SW	NE	19	60	13	20	Cert. 318001	105-0080-2930
SE	NE	19	60	13	40	Cert. 318001	105-0080-2940
NE·	SE	19	60	13	40	Cert. 318001	105-0080-3030
NW	SE	19	60	13	40	Cert. 318001	105-0080-3040
NE	NE	20	60	13	40	Cert. 318001	105-0080--3070
NW	NE	20	60	13	40	Cert. 318001	105-0080-3080
SW	NE	20	60	13	40	Cert. 318001	105-0080-3090
SE	NE	20	60	13	40	Cert. 318001	105.0080-3100
NE	NW	20	60	13	40	Cert. 318001	105-0080-3110
NW	NW	20	60	13	40	Cert. 318001	105-0080-3120
SW	NW	20	60	13	40	Cert. 318001	105-0080-3130
SE	NW	20	60	13	40	Cert. 318001	105-0080-3140
W ½	SW	20	60	13	80	Cert. 318001	105-0080-3160
all	SE	20	60	13	160	Cert. 318001	105-0080-3190
NE	NE	21	60	13	40	Cert. 318001	105-0080-3230
NW	NE	21	60	13	40	Cert. 318001	
SW	NE	21	60	13	40	Cert. 318001	
All	NW	21	60	13	160	Cert.318001	105-0080-3270
NE	SW	21	60	13	40	Cert. 318001	105-0080-3310
NW	SW	21	60	13	40	Cert. 318001	
SW	SW	21	60	13	40	Cert. 318001	
NW	NE	29	60	13	40	Cert. 318001	105-0080-4540
NW	SW	29	60	13	40	Cert. 318001	105-0080-4620
NE	NW	29	60	13	40	Cert. 318001	105-0080-4570
NW	NW	29	60	13	40	Cert. 318001	
SW	NW	29	60	13	40	Cert. 318001	
All except Lot 2		30	60	13	624.11	Cert. 318001	105-0080-4690
Total Acres					*3,084.11*		

Cliffs Erie Owned Lands Under Purchase Option		
Township and Range	**Section**	**Quarter and Sub-quarter**
Township 60 North - Range 12 West	Section 2	NW-NE (Lot 2)
		SW-NE
		NE-NW (Lot 3)
		NW-NW (Lot 4)
		SW-NW
		SE-NW
		NE-SW
		SW-SW
	Section 3	NE-NE (Lot 1)
		NW-NE (Lot 2)
		SW-NE
		SE-NE
		NE-NW (Lot 3)
		SE-NW EXCEPT that part lying W'ly and NW'ly of a line drawn parallel with and distant 200 ft W'ly and NW'ly of the following described line: Commencing at the East quarter comer of said Section 9; thence South 71 degrees 44 minutes 20 seconds West, bearing based on the East line of said Section 9 having a bearing of South 03 degrees 27 minutes 19 seconds East, Saint Louis County Transverse Mercator 1996 projection, a distance of 462.67 ft to the point of beginning of the line to be described; thence NE'ly along a non-tangential curve concave to the East, having a radius of 2925.20 ft, central angle of 46 degrees 35 minutes 13 seconds, the tangent of said curve at this point bears North 22 degrees 35 minutes 37 seconds West a distance of 2378.47 ft to the point of tangency; thence North 23 degrees 59 minutes 36 seconds East a distance of 426.28 ft; thence NE'ly along a tangential curve concave to the SE, having a radius of 1217.20 ft, central angle of 13 degrees 13 minutes 05 seconds, a distance of 280.81 ft to the point of tangency; thence North 37 degrees 12 minutes 41 seconds East, a distance of 1001.36 ft; thence NE'ly along a tangential curve concave to the NW, having a radius of 3780.62 ft, central angle of 32 degrees 51 minutes 39 seconds, a distance of 2463.58 ft to the point of tangency; thence North 04 degrees 21 minutes 02 seconds East a distance of 2244.11 ft, and there terminating.
		Those parts of NE-SW, NW-SW, and SW-SW lying easterly and southeasterly of a line drawn parallel with and distant 200 ft westerly and northwesterly of the first following described line and easterly, southeasterly and southerly of the second following described line:
		First Described Line: Commencing at the east quarter comer of Section 9, Township 60 North, Range 12 West; thence S 71 degrees 44 minutes 20 seconds W, bearing based on the east line of said Section 9 having a bearing of S 03 degrees 27 minutes 19 seconds E, St. Louis County Transverse Mercator 1996 projection, a distance of 462.67 ft to the point of beginning of the line to be described; thence northeasterly along a non-tangential curve concave to the east, having a radius of 2925.20 ft, central angle of 46 degrees 35 minutes 13 seconds, the tangent of said curve at this point bears N 22 degrees 35 minutes 37 seconds W; a distance of 2378.47 ft to the point of tangency; thence N 23 degrees 59 minutes 36 seconds E a distance of 426.28 ft; thence northeasterly along a tangential curve to the southeast, having a radius of 1217.20 ft, central angle of 13 degrees 13 minutes 05 seconds, a distance of 280.81 ft to the point of tangency; thence N 37 degrees 12 minutes 41 seconds E, a distance of 1001.36 ft; thence northeasterly along a tangential curve concave to the northwest, having a radius of 3780.62 ft, central angle of 32 degrees 51 minutes 39 seconds, a distance of 2168.30 ft to the point of tangency; thence N 04 degrees 21 minutes 02 seconds E, a distance of 2244.11 ft; thence northeasterly along a tangential curve concave to the southeast, having a radius of 2866.16 ft, central angle of 49 degrees 14 minutes 53 seconds, a distance of 2463.58 ft to the point of tangency; thence N 53 degrees 35 minutes 54 seconds E, a distance of 664.36 ft; thence northeasterly along a tangential curve concave to the southeast, having a radius of 910.15 ft, central angle of 04 degrees 00 minutes 27 seconds, a distance of 63.66 ft to the point of tangency; thence N 57 degrees 36 minutes 21 seconds E a distance of 1469.17 ft, and there terminating.
		Second Described Line: Commencing at the point of termination of the first above-described line; thence N 32 degrees 23 minutes 39 seconds W a distance of 200 ft to the point of beginning of the line to be described; thence N 06 degrees 23 minutes 50 seconds W a distance of 482.88 ft; thence N 34 degrees 17 minutes 24 seconds E a distance of 1692.54 ft; thence S 77 degrees 26 minutes 00 seconds E a distance of 1541.34 ft; thence N 52 degrees 08 minutes 41 seconds E a distance of 1454.79 ft; thence N 68 degrees 02 minutes 16 seconds E a distance of 148.61 ft; thence N 51 degrees 03 minutes 13 seconds E a distance of 321 ft; thence N 41 degrees 52 minutes 37 seconds E a distance of 459.18 ft, and there terminating.
		SE-SW
		NW-SE
		SW-SE
		SE-SE
	Section 10	SE-NE
		NE-NW
		SE-NW
Township 61 North -	Section 26	SW-SE

Cliffs Erie Owned Lands Under Purchase Option		
Township and Range	**Section**	**Quarter and Sub-quarter**
Range 12 West		
		SE-SE
	Section 34	Those parts of NE-NE, and SW-NElying easterly, southerly and southeasterly of the following described line:
		Beginning atMinnesota State Plane North Zone Coordinates, Northing 777443.63, Easting 2933961.70; thence North 06°52'47" West, bearing based on Minnesota State Plane North ZoneCoordinates, a distance of 482.76 ft to Minnesota State Plane North Zone Coordinates, Northing 777922.92, Easting 2933903.87; thence North 33°48'27" East, bearing based on Minnesota State Plane North ZoneCoordinates, a distance of 1692.26 ft to Minnesota State Plane North Zone Coordinates, Northing 779329.04, Easting 2934845.45; thence South 77°54'58" East, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 1541.08 ft to Minnesota State Plane North Zone Coordinates, Northing 779006.42, Easting 293652.39; thence North 51°39'44" East, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 1454.55 ft to Minnesota State Plane North Zone Coordinates, Northing 779908.67, Easting 2937493.29; thence North 67°33'18" East, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 148.59 ft to Minnesota State Plane North Zone Coordinates, Northing 779965.40, Easting 2937630.62; thence North 50°21'12" East, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 328.68 ft to Minnesota State Plane North Zone Coordinates, Northing 780175.12, Easting 2937883.70; thence North 41<>23'40East, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 451.27 ft to Minnesota State Plane North Zone Coordinates, Northing 780513.65, Easting 2938182.10 and there terminating.
		Those parts of SE-NE, NE-SW, SE-SW and NW-SE; lying easterly and southeasterly of a line drawn parallel with and distant 200 ft westerly and northwesterly of the first following described line and easterly, southeasterly and southerly of the second following described line:
		First Described Line: Commencing at the east quarter comer of Section 9, Township 60 North, Range 12 West; thence S 71 degrees 44 minutes 20 seconds W, bearing based on the east line of said Section 9 having a bearing of S 03 degrees 27 minutes 19 seconds E, st.Louis County Transverse Mercator 1996 projection, a distance of 462.67 ft to the point of beginning of the line to be described; thence northeasterly along a non-tangential curve concave to the east, having a radius of2925.20ft, central angle of 46 degrees 35 minutes 13 seconds, the tangent of said curve at this point bears N 22 degrees 35 minutes 37 seconds W a distance of 2378.47 ft to the point of tangency; thence N 23 degrees 59 minutes 36 seconds E a distance of 426.28 ft; thence northeasterly along a tangential curve to the southeast, having a radius of 1217.20 ft, central angle of 13 degrees 13 minutes 05 seconds, a distance of280.81 ft to the point of tangency; thence N 37 degrees 12 minutes 41 seconds E, a distance of 1001.36 ft; thence northeasterly along a tangential curve concave to the northwest, having a radius of 3780.62 ft, central angle of32 degrees 51 minutes 39 seconds, a distance of 2168.30 ft to the point of tangency; thence N 04 degrees 21 minutes 02 seconds E, a distance of 2244.11 ft; thence northeasterly along a tangential curve concave to the southeast, having a radius of2866.16 ft, central angle of 49 degrees 14 minutes 53 seconds, a distance of 2463.58 ft to the point of tangency; thence N53 degrees 35 minutes 54 seconds E, a distance of 664.36 ft; thence northeasterly along a tangential curve concave to the southeast, having a radius of910.15 ft, central angle of 04 degrees 00 minutes 27 seconds, a distance of 63.66 ft to the point of tangency; thence N57 degrees 36 minutes 21 seconds E a distance of 1469.17 ft, and there terminating.
		Second Described Line: Commencing at the point of termination of the first above-described line; thence N 32 degrees 23 minutes 39 seconds W, a distance of 200 ft to the point of beginning of the line to be described; thence N 06 degrees 23 minutes 50 seconds W a distance of 482.88 ft; thence N 34 degrees 17 minutes 24 seconds E a distance of 1692.54 ft; thence S 77 degrees 26 minutes 00 seconds E a distance of 1541.34 ft; thence N 52 degrees 08 minutes 41 seconds E a distance of 1454.79 ft; thence N 68 degrees 02 minutes 16 seconds E a distance of 148.61 ft; thence N 51 degrees 03 minutes 13 seconds E a distance of 321 ft; thence N 41 degrees 52 minutes 37 seconds E a distance of459.18 ft, and there terminating.
		NE-SE
		SW-SE
		SE-SE
	Section 35	NE-NE
		NW-NE
		SW-NE
		SE-NE
		NE-NW lying easterly and southeasterly of a line drawn parallel with and distant 200 ft westerly and northwesterly of the first following described line and easterly, southeasterly and southerly of the second following described line:
		First Described Line: Commencing at the east quarter comer of Section 9, Township 60 North, Range 12 West; thence S 71 degrees 44 minutes 20 seconds W, bearing based on the east line of said Section 9 having a bearing of S 03 degrees 27 minutes 19 seconds E, St, Louis County Transverse Mercator 1996 projection, a distance of 462.67 ft to the point of beginning of the line to be described; thence northeasterly along a non-tangential curve concave to the east, having a radius of 2925.20 ft, central angle of 46 degrees 35 minutes 13seconds, the tangent of said curve at this point bears N 22 degrees 35 minutes 37 seconds W a distance of 2378.47 ft to the point of tangency; thence N 23 degrees 59 minutes 36 seconds E a distance of 426.28 ft; thence northeasterly along a tangential curve to the southeast, having a radius of 1217.20 ft, central angle of 13 degrees 13 minutes 05 seconds, a distance of 280.81 ft to the point of tangency; thence N 37 degrees 12 minutes 41 seconds E, a distance of 1001.36 ft; thence northeasterly along a tangential curve concave to the northwest, having a radius of 3780.62 ft, central angle of 32 degrees 51 minutes 39 seconds, a distance of2168.30 ft to the point of tangency; thence N 04 degrees 21 minutes 02 seconds E, a distance of2244.11 ft; thence northeasterly along a tangential curve concave to the southeast, having a radius of 2866.16 ft, central angle of 49 degrees 14 minutes 53 seconds, a distance of2463.58 ft to the point of tangency; thence N 53 degrees 35 minutes 54 seconds E, a distance of 664.36 ft; thence northeasterly along a tangential curve concave to the southeast, having a radius of910.15 ft, central angle of 04 degrees 00 minutes 27 seconds, a distance of 63.66 ft to the point of tangency;

Cliffs Erie Owned Lands Under Purchase Option		
Township and Range	**Section**	**Quarter and Sub-quarter**
		thence N 57 degrees 36 minutes 21 seconds E a distance of 1469.17 ft, and there terminating.
		Second Described Line: Commencing at the point of termination of the first above-described line; thence N 32 degrees 23 minutes 39 seconds W a distance of 200 ft to the point of beginning of the line to be described; thence N 06 degrees 23 minutes 50 seconds W a distance of 482.88 ft; thence N 34 degrees 17 minutes 24 seconds E a distance of 1692.54 ft; thence S 77 degrees 26 minutes 00 seconds E a distance of 1541.34 ft; thence N 52 degrees 08 minutes 41 seconds E a distance of 1454.79 .ft; thence N 68 degrees 02 minutes 16 seconds E a distance of 148.61 ft; thence N 51 degrees 03 minutes 13 seconds E a distance of 321 ft; thence N 41 degrees 52 minutes 37 seconds E a distance of 459.18 ft, and there terminating.
		NW-NW, EXCEPT That part lying westerly and northwesterly of a line drawn parallel with and distant 60.960 meters (200.00 ft) westerly of the first following described line and westerly, northwesterly and northerly of the second following described line:
		First Described Line: Beginning at Minnesota State Plane North Zone Coordinates, Northing 233462.499 (meter typical) (765951.54 ft), Easting 892853.239 (meter typical) (2929302.64 ft); thence northeasterly along a tangential curve (chord definition) (the tangent of said curve at this point bears North 23°4'35" west, bearing based on Minnesota State Plane North Zone Coordinates), having a radius of 891.456 meters (2924.72 ft), central angle of 46°35'14", a distance of 724.843 meters (2378.09 ft) to the point of tangency at Northing 234167.530 (768264.63 ft), Easting 892855.913 (2929311.41 ft); thence North 23°30'39" East, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 129.909 meters (426.21 ft) to the point of curvature at Minnesota State Plane North Zone Coordinates, Northing 234286.655 (768655.46 ft), Easting 892907.735 (2929481.43 ft); thence northeasterly along a tangential curve (chord definition) concave to the southeast, having a radius of 370.942 meters (1217.00 ft), central angle of 13°13'04", a distance of 85.573 meters (280.75 ft) to the point of tangency at Minnesota State Plane North Zone Coordinates, Northing 234360.511 (768897.77 ft), Easting 892950.581 (2929622.00 ft); thence North 36°43'43" East, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 305.163 meters (1001.19 ft) to the point of curvature at Minnesota State Plane North Zone Coordinates, Northing 234605.093 (769700.20 ft), Easting 893133.080 (2930220.75 ft); thence northeasterly along a tangential curve (chord definition) concave to the northwest having a radius of 1152.146 meters (3780.00 ft), central angle of 32°51'39", a distance of 660.789 meters (2167.94 ft) to the point of tangency at Minnesota State Plane North Zone Coordinates, Northing 235216.388 (771705.76 ft), Easting 893359.181 (2930962.55 ft); thence North 03°52'04" East, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 683.893 meters (2243.74 ft) to the point of curvature at Minnesota State Plane North Zone Coordinates, Northing 235898.724 (773944.39 ft), Easting 893405.316 (2931113.91 ft); thence northeasterly along a tangential curve (chord definition) concave to the southeast, having a radius of 873.461 meters (2865.68 ft), central angle of 49°14'53", a distance of 750.776 meters (2463.17 ft) to the point of tangency at Minnesota State Plane North Zone Coordinates, Northing 236538.443 (776043.20 ft), Easting 893752.535 (2932253.08 ft); thence North 53°06'57" East, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 202.464 meters (664.25 ft) to the point of curvature at Minnesota State Plane North Zone Coordinates, Northing 236659.964 (776441.89 ft), Easting 893914.479 (2932784.39 ft); thence northeasterly along a tangential curve (chord definition) concave to the southeast having a radius of 277.369 meters (910.00 ft), central angle of 04°00'2711, a distance of 19.401 meters (63.65 ft) to the point of tangency at Minnesota State Plane North Zone Coordinates, Northing 236671.056 (776478.28 ft), Easting 893930.390 (2932836.59 ft); thence North 57°07'2411 East, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 447.734 meters (1468.94 ft) to Minnesota State Plane North Zone Coordinates, Northing 236914.097 (777275.66 ft), Easting 894306.413 (2934070.26 ft), and there terminating.
		Second Described Line: Commencing at the point of termination of first above-described line; thence North 32°52'36" West, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 60.960 meters (200.00 ft) to the point of beginning of the line to be described at Minnesota State Plane North Zone Coordinates, Northing 236965.295 (meter typical) (777443.63 ft), Easting 894273.324 (meter typical) (2933961.70 ft); thence North 06°52'47" West, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 147.145 meters (482.76 ft) to Minnesota State Plane North Zone Coordinates, Northing 237111.383 (777922.92 ft), Easting 894255.697 (2933903.87 ft); thence North 33°48'27" East, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 515.802 meters (1692.26 ft) to Minnesota State Plane North Zone Coordinates, Northing 237539.969 (779329.04 ft), Easting 894542.691 (2934845.45 ft); thence South 77°54'5811 East, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 469.722 meters (1541.08 ft) to Minnesota State Plane North Zone Coordinates, Northing 237441.634 (779006.42 ft), Easting 895002.008 (2936352.39 ft); thence North 51°39'4411 East, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 443.348 meters (1454.55 ft) to Minnesota State Plane North Zone Coordinates, Northing 237716.640 (779908.67 ft); thence North 67°33'1811 East, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 45.290 meters (148.59 ft) to Minnesota State Plane North Zone Coordinates, Northing 237733.932 (779965.40 ft), Easting ('"895391.613 (2937630.62 ft); thence North 50°21 '12" East, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 100.182 meters (328.68 ft) to Minnesota State Plane North Zone Coordinates, Northing 237797.855 (780175.12 ft), Easting 895468.752 (2937883.70 ft); thence North 41°23'40" East, bearing based on Minnesota State Plane North Zone Coordinates, a distance of 137.547 meters (451.27 ft) to Minnesota State Plane North Zone Coordinates, Northing 237901.039 (780513.65 ft), Easting 895559.704 (2938182.10 ft) and there terminating.
		SW-NW
		SE-NW
		NE-SW
		NW-SW
		SW-SW
		SE-SW
		NW-NE-SE
		NW-SE
		SW-SE
Dunka Property Leased Lands		

Cliffs Erie Owned Lands Under Purchase Option		
Township and Range	**Section**	**Quarter and Sub-quarter**
NW ¼ of SW ¼, Section 2, Township 60 North, Range 12 West of the Fourth Principal Meridian.		
NE ¼ of SE ¼, Section 3, Township 60 North, Range 12 West of the Fourth Principal Meridian.		
NE ¼ of NE ¼, Section 10, Township 60 North, Range 12 West of the Fourth Principal Meridian.		
NW ¼ of NW ¼, Section 11, Township 60 North, Range 12 West of the Fourth Principal Meridian.		
Total		*Approximately 1,845 acres*

Minnesota Power Option Lands				
Township	**Range**	**Section**	**Parcel**	**Acreage**
61	12	25	610-0011-03610	16.5
61	12	25	610-0011-03620	23.2
61	12	25	610-0011-03630	19.3
61	12	25	610-0011-03640	11.7
61	12	25	610-0011-03650	28
61	12	36	610-0011-04810	16.6
61	12	36	610-0011-04800	26
Total				*141.3*