Nova Gold Resources Inc.
Donlin Creek Gold Project,
Alaska, USA
NI 43-101 Technical Report

Prepared for Nova Gold Resources Inc. by:

Kirk Hanson P.E.
Gordon Seibel M.AusIMM.
Simon Allard, P.Eng.
Gregory Wortman P.Eng.
Alexandra Kozak P.Eng.

Effective Date: 1 April, 2009
160638

CERTIFICATE OF QUALIFIED PERSON

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I, Kirk Hanson, P.E., am employed as a Principal Engineer with AMEC E&C Services Inc.

This certificate applies to the Technical Report entitled "NovaGold Resources Inc., Donlin Creek Gold Project, Alaska, USA, NI 43-101 Technical Report" dated 1 April 2009 (the Technical Report).

I am a registered Professional Engineer in the states of Idaho (11063) and Nevada (10640).

I graduated with a B.Sc. degree from Montana Tech of the University of Montana, Butte, Montana in 1989 and from Boise State University, Boise, Idaho with a MBA in 2003.

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 Donlin Creek Gold Project on 1 October 2008.

I am responsible for Sections 1, 2, 3, 4, 5, 6, 17.2, 18.1 to 18.8, 19, 20, 21, 22, and 23 and those portions of the summary, conclusions and recommendations that pertain to those sections of the Technical Report.

I am independent of NovaGold Resources Inc. as independence is described by Section 1.4 of NI 43–101.

I have been involved with the Donlin Creek Gold Project since October 2008 as part of preparation of this Technical Report.

I have read NI 43–101 and this Technical Report has been prepared in compliance with that Instrument.

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

"signed and sealed"

Kirk Hanson, P.E.

Dated: 2 June 2009

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I, Gordon Seibel, M.AusIMM, am employed as a Principal Geologist with AMEC E&C Services Inc..

This certificate applies to the Technical report entitled "NovaGold Resources Inc., Donlin Creek Gold Project, Alaska, USA, NI 43-101 Technical Report" dated 1 April 2009 (the Technical Report).

I am a Corporate Member of the Australasian Institute of Mining and Metallurgy (AusIMM # 223092). I graduated from the University of Colorado with a Bachelor of Arts degree in Geology in 1980. In addition, I obtained a Masters of Science degree in Geology from Colorado State University in 1991.

I have practiced my profession for 29 years. I have been directly involved in the development of resource models and mineral resource estimation for gold projects in Colorado, Nevada, California, Canada, Mexico, Peru, and Australia since 1991.

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 Donlin Creek property on 1 October 2008.

I am responsible for Sections 7, 8, 9, 10, 11, 12, 13, 14, 15, 17.1 and those portions of the summary, conclusions and recommendations that pertain to those sections of the Technical Report.

I am independent of NovaGold Resources Inc. as independence is described by Section 1.4 of NI 43–101.

I have been involved with the Donlin Creek Gold Project since October 2008 as part of preparation of this Technical Report.

I have read NI 43–101 and this Technical Report has been prepared in compliance with that Instrument.

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

"Signed"

Gordon Seibel, M.AusIMM

Dated: 2 June 2009

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I, Simon Allard P.Eng, am employed as a Financial Analyst with AMEC Americas Limited.

This certificate applies to the Technical Report entitled "NovaGold Resources Inc., Donlin Creek Gold Project, Alaska, USA, NI 43-101 Technical Report" dated 1 April 2009 (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 five years. I have been directly involved in cash-flow modelling, risk evaluation, real-options valuation, financial analysis, marketing studies and financial review of mines including Araguaia (base metals) in Brazil, Oyu Tolgoi (copper-gold) in Mongolia, Gibellini (vanadium) in Nevada, Bisha (polymetallic) in Eritrea, Gaho Kue (diamonds) in NWT and Gurupi (gold) in Brazil.

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 Donlin Creek Gold Project.

I am responsible for Sections 18.9 to 18.12, and those portions of the summary, conclusions and recommendations that pertain to those sections of the technical report titled "NovaGold Resources Inc., Donlin Creek Gold Project, Alaska, USA, NI 43-101 Technical Report" and dated 1 April 2009, (the "Technical Report").

I am independent of NovaGold Resources Inc. as independence is described by Section 1.4 of NI 43–101.

I have been involved with the Donlin Creek Gold Project since October 2008 as part of preparation of this Technical Report.

I have read NI 43–101 and this Technical Report has been prepared in compliance with that Instrument.

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

“Signed and sealed”

Simon Allard, P.Eng.

Dated: 2 June 2009

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I, Gregory Wortman, P.Eng., am employed as a Technical Director, Process with AMEC Americas Limited..

This certificate applies to the Technical Report entitled "NovaGold Resources Inc., Donlin Creek Gold Project, Alaska, USA, NI 43-101 Technical Report" dated 1 April 2009 (the Technical Report).

I am a member of Professional Engineers, Ontario. I graduated from the Technical University of Nova Scotia in 1968.

I have practiced my profession for 40 years. I have been directly involved in design and operation of a number of pressure oxidation autoclave hydrometallurgical facilities during that period.

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 Donlin Creek Gold Project.

I am responsible for Section 16.1.3 of the Technical Report.

I am independent of NovaGold Resources Inc. as independence is described by Section 1.4 of NI 43–101.

I have been involved with the Donlin Creek Gold Project since October 2008 as part of preparation of this Technical Report.

I have read NI 43–101 and this Technical Report has been prepared in compliance with that Instrument.

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

"Signed and sealed"

Gregory Wortman, P.Eng.

Dated: 2 June 2009

CERTIFICATE OF QUALIFIED PERSON

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I, Alexandra J. Kozak, P.Eng., am employed as Manager, Process Engineering with AMEC Americas Limited.

This certificate applies to the Technical Report entitled "NovaGold Resources Inc., Donlin Creek Gold Project, Alaska, USA, NI 43-101 Technical Report" dated 1 April 2009 (the Technical Report).

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC). I graduated from the University of Alberta with a Bachelor of Science degree in Mineral Process Engineering in 1985.

I have practiced my profession continuously since 1985 and have been involved in operations in Canada and Guyana and preparation of scoping, pre-feasibility, and feasibility level studies for gold, base metals and diamond properties in Canada, United States, Peru, Mexico, Mongolia, Ghana, and New Guinea. I am currently a Consulting Engineer and have been so since September 1996.

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 Donlin Creek Gold Project.

I am responsible for Sections 16.1.1, 16.1.2, and 16.1.4 to 16.3 of the Technical Report and those portions of the summary, conclusions and recommendations that are based on that section.

I am independent of NovaGold Resources Inc. as independence is described by Section 1.4 of NI 43–101.

I have been involved with the Donlin Creek Gold Project since October 2008 as part of preparation of this Technical Report.

I have read NI 43–101 and this Technical Report has been prepared in compliance with that Instrument.

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

"Signed and sealed"

Alexandra J. Kozak, P.Eng.

Dated: 2 June 2009

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To: Securities Regulatory Authority
British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Financial Services Commission
Manitoba Securities Commission
Ontario Securities Commission
Autorité des marchés financiers du Quebec
New Brunswick Securities Commission
Nova Scotia Securities Commission
Securities Office, Prince Edward Island
Securities Commission of Newfoundland and Labrador

Re*: NovaGold Resources Inc., Press Release Dated 28 April 2009 Entitled "Donlin Creek Feasibility Study Adds 14.7 Mozs to NovaGold's Reserves"*

I, Kirk Hanson, P.E. consent to the public filing of Sections 1, 2, 3, 4, 5, 6, 17.2, 18.1 to 18.8, 19, 20, 21, 22, and 23 and those portions of the summary, conclusions and recommendations that pertain to those sections, of the Technical Report entitled "NovaGold Resources Inc., Donlin Creek Gold Project, Alaska, USA, NI 43-101 Technical Report" and dated 1 April 2009, (the "Technical Report").

I consent to extracts from, or a summary of, the Technical Report in the NovaGold Resources Inc. press release entitled "Donlin Creek Feasibility Study Adds 14.7 Mozs to NovaGold's Reserves" (the "Press Release") and dated 28 April 2009.

I confirm that I have read the Press Release and that it fairly and accurately represents the information in the Technical Report that supports the disclosure.

"signed and sealed"

Kirk Hanson | 2 June, 2009

Signed | Dated

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To: Securities Regulatory Authority
British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Financial Services Commission
Manitoba Securities Commission
Ontario Securities Commission
Autorité des marchés financiers du Quebec
New Brunswick Securities Commission
Nova Scotia Securities Commission
Securities Office, Prince Edward Island
Securities Commission of Newfoundland and Labrador

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I, Gordon Seibel, M.AusIMM, consent to the public filing of Sections 7, 8, 9, 10, 11, 12, 13, 14, 15, and 17.1, and those portions of the summary, conclusions and recommendations that pertain to those sections of the technical report titled "NovaGold Resources Inc., Donlin Creek Gold Project, Alaska, USA, NI 43-101 Technical Report" and dated 1 April 2009, (the "Technical Report").

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I confirm that I have read the Press Release and that it fairly and accurately represents the information in the Technical Report that supports the disclosure.

"signed"

Gordon Seibel, M.AusIMM	2 June 2009
Signed	Dated

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To: Securities Regulatory Authority
British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Financial Services Commission
Manitoba Securities Commission
Ontario Securities Commission
Autorité des marchés financiers du Quebec
New Brunswick Securities Commission
Nova Scotia Securities Commission
Securities Office, Prince Edward Island
Securities Commission of Newfoundland and Labrador

Re*: NovaGold Resources Inc., Press Release Dated 28 April 2009 Entitled "Donlin Creek Feasibility Study Adds 14.7 Mozs to NovaGold's Reserves"*

I, Simon Allard, P.Eng. consent to the public filing of Sections 18.9 to 18.12, and those portions of the summary, conclusions and recommendations that pertain to that section of the technical report titled "NovaGold Resources Inc., Donlin Creek Gold Project, Alaska, USA, NI 43-101 Technical Report" and dated 1 April 2009, (the "Technical Report").

I consent to extracts from, or a summary of, the Technical Report in the NovaGold Resources Inc. press release entitled "Donlin Creek Feasibility Study Adds 14.7 Mozs to NovaGold's Reserves" (the "Press Release") and dated 28 April 2009.

I confirm that I have read the Press Release and that it fairly and accurately represents the information in the Technical Report that supports the disclosure.

“signed and sealed”

Simon Allard, P.Eng.	2 June, 2009
Signed	Dated

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To: Securities Regulatory Authority
British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Financial Services Commission
Manitoba Securities Commission
Ontario Securities Commission
Autorité des marchés financiers du Quebec
New Brunswick Securities Commission
Nova Scotia Securities Commission
Securities Office, Prince Edward Island
Securities Commission of Newfoundland and Labrador

Re: *NovaGold Resources Inc., Press Release Dated 28 April 2009 Entitled "Donlin Creek Feasibility Study Adds 14.7 Mozs to NovaGold's Reserves"*

I, Gregory Wortman, P.E. consent to the public filing of Section 16.1.3 and those portions of the summary, conclusions and recommendations that pertain to that section, of the Technical Report entitled "NovaGold Resources Inc., Donlin Creek Gold Project, Alaska, USA, NI 43-101 Technical Report" and dated 1 April 2009, (the "Technical Report").

I consent to extracts from, or a summary of, the Technical Report in the NovaGold Resources Inc. press release entitled "Donlin Creek Feasibility Study Adds 14.7 Mozs to NovaGold's Reserves" (the "Press Release") and dated 28 April 2009.

I confirm that I have read the Press Release and that it fairly and accurately represents the information in the Technical Report that supports the disclosure.

"signed and sealed"

Gregory Wortman	2 June, 2009
Signed	Dated

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To: Securities Regulatory Authority
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Securities Office, Prince Edward Island
Securities Commission of Newfoundland and Labrador

Re*: NovaGold Resources Inc., Press Release Dated 28 April 2009 Entitled "Donlin Creek Feasibility Study Adds 14.7 Mozs to NovaGold's Reserves"*

I, Alexandra Kozak, P.Eng. consent to the public filing of Sections 16.1.1, 16.1.2, and 16.1.4 to 16.3, and those portions of the summary, conclusions and recommendations that pertain to that section of the technical report titled "NovaGold Resources Inc., Donlin Creek Gold Project, Alaska, USA, NI 43-101 Technical Report" and dated 1 April 2009, (the "Technical Report")..

I consent to extracts from, or a summary of, the Technical Report in the NovaGold Resources Inc. press release entitled "Donlin Creek Feasibility Study Adds 14.7 Mozs to NovaGold's Reserves" (the "Press Release") and dated 28 April 2009.

I confirm that I have read the Press Release and that it fairly and accurately represents the information in the Technical Report that supports the disclosure.

“signed and sealed”

Alexandra Kozak, P.Eng. 2 June 2009

Signed Dated

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IMPORTANT NOTICE

CONTENTS

TABLES

FIGURES

1.0 SUMMARY

AMEC Americas Limited (AMEC) was commissioned by NovaGold Resources Inc. (NovaGold), to provide an independent Qualified Person's Review and Technical Report (the Report) for the Donlin Creek Gold Project (the Project) located in Alaska, USA.

AMEC understands that this Report will be used by NovaGold in support of a NovaGold press release dated 28 April 2009, entitled "Donlin Creek Feasibility Study Adds 14.7 Mozs to NovaGold's Reserves".

The Project is a joint venture between NovaGold Resources Alaska, Inc., a wholly-owned subsidiary of NovaGold, and Barrick Gold U.S. Inc., a wholly-owned subsidiary of Barrick Gold Corporation. For the purposes of this report, the name "NovaGold" refers interchangeably to the NovaGold subsidiary and parent companies; and the name "Barrick" refers interchangeably to the Barrick subsidiary and parent companies.

During 2006, Barrick acquired Placer Dome Inc., which had held an interest in the Project between 1995 and 2006 through its wholly-owned subsidiary, Placer Dome US Inc. For the purposes of this report, the name "Barrick (formerly Placer Dome)" refers interchangeably to Placer Dome Inc. and to Placer Dome US Inc. when reference is made to the manager of work undertaken on the Project between 1995 and 2000, and between 2003 and 2006.

On 1 December 2007, NovaGold entered into a limited liability company agreement with Barrick that provided for the conversion of the Project into a new limited liability company, the Donlin Creek LLC (DCLLC), which is jointly owned by NovaGold and Barrick on a 50/50 basis. The limited liability company agreement provides that information obtained in connection with the performance of the agreement, which includes information developed by, or on behalf of the DCLLC, may be disclosed by either NovaGold or Barrick, or affiliates of either, where such disclosure is required by law or to meet stock exchange requirements, including for purposes of a technical report required under National Instrument 43–101, *Standards of Disclosure for Mineral Projects* (NI 43–101).

1.1 Principal Outcomes

- Proven and Probable Mineral Reserves estimated for 29.3 Moz contained gold:
  - Proven Mineral Reserves: 8.4 Mt at 2.59 g/t Au (0.7 Moz contained gold)
  - Probable Mineral Reserves: 375 Mt at 2.37 g/t Au (28.6 Moz contained gold)

- 21 year mine life at 53,500 t/d throughput
- Average annual gold production:
  - 1.25 Moz over the projected life of mine
  - 1.5 Moz over the first full 10 years
  - 1.6 Moz over the first full 5 years
- Total predicted total cash costs:
  - $467/oz[1] Au over the life of mine
  - $442/oz over the first full 10 years
  - $394/oz over the first full 5 years
- Cumulative net present after-tax cash flow (net present value (NPV) 5%)
  - At $725/oz gold price (Base Case) minus $733 million
  - At $900/oz gold price (Alternative Case 1) $829 million
  - At $1,000/oz gold price (Alternative Case 2) $1,674 million
- Average annual cash flow for first full five years of production[2]
  - At $725/oz gold price $521 million
  - At $900/oz gold price $790 million
  - At $1,000/oz gold price $944 million
- At $725/oz gold price, undiscounted cumulative pre-tax net cash flow is $1.5 billion, pre-tax NPV 5% is minus $592 million, with a pre-tax internal rate of return (IRR) of 3%
- At current gold prices of $900/oz, undiscounted cumulative pre-tax net cash flow is $5.9 billion, pre-tax NPV 5% is $1.5 billion with a pre-tax IRR of 9.4%
- At a gold price of $1,000/oz, undiscounted cumulative pre-tax net cash flow is $8.4 billion, pre-tax NPV 5% is $2.7 billion with a pre-tax IRR of 12.3%
- Increase in contained gold ounces of 3.6 Moz in Proven and Probable Mineral Reserve and Measured and Indicated Mineral Resources over the previous Measured and Indicated Mineral Resource estimate of 10 June 2008.

1 All dollar figures quoted in this summary are in US dollars
2 Total revenues minus total operating costs and royalties before interest, taxes, depreciation and amortization.

1.2 Location and Access

Donlin Creek is located in southwest Alaska in the United States of America, approximately 80 km northeast of Aniak, a regional hub, and approximately 20 km north of the village of Crooked Creek on the Kuskokwim River. The Kuskokwim River is a regional transportation route and is serviced by commercial barge lines.

There is no road or rail access to the site and therefore all personnel and supplies are transported by air. An airstrip 1,500 m long is located adjacent the exploration camp and is capable of handling aircraft as large as C-130 Hercules. The project can be serviced directly by charter air facilities out of both Anchorage and Aniak.

At present, the project is isolated from power and all other public infrastructure. Planned infrastructure to support proposed mining operations will include, in addition to the future plant site at the mine, a wind farm, an airstrip, barge terminals at Bethel and Birch Tree Crossing (BTC), construction of major receiving, storage, and transfer facilities at different locations in Alaska en-route to the mine, mine access road development, and a fuel pipeline.

1.3 Tenure and Surface Rights

The Donlin Creek exploration and mining lease currently includes a total of 42 sections leased from Calista Corporation (Calista), a regional Native corporation. Calista holds the subsurface (mineral) estate for Native-owned lands in the region. Title to all these sections was conveyed to Calista by the Federal Government. Calista owns the surface estate on nine of these 42 sections. A separate surface use agreement for access and surface rights is in place with The Kuskokwim Corporation (TKC) that owns the majority of the surface estate of the remaining 33 sections. The surface use agreement grants non-exclusive surface use rights to the DCLLC. All of these sections have now been conveyed to Calista/TKC by the Federal Government.

The currently identified Mineral Resource and the bulk of the primary infrastructure sites (mill and waste rock facilities) are located on leased lands that were conveyed. Lands required for the BTC port site, access road, wind farm power facilities, tailings storage facility in Anaconda Creek, and airstrip are located on a combination of Native-conveyed, Native-selected, and State of Alaska-selected lands, and additional negotiations will be required. A right-of-way will also be required from the State of Alaska for the proposed road alignment where it crosses State lands.

1.4 Agreements

The DCLLC operates under an exploration and mining lease from the Calista Corporation. A separate Surface Use Agreement with the TKC grants non-exclusive surface use rights to the DCLLC. This can be converted to an exclusive use right under the existing agreement.

The Calista agreement includes staged royalty provisions, with payments tied to various stages of Project development and production. Calista also has the right to acquire an equity interest in the Project of up to 15%. Calista shareholders have a hiring preference and Calista has a 5% bidding preference on competitive contracts for all work on or for the project.

The TKC agreement provides for an annual aggregate surface use fee of $50,000. Once exclusive-use lands are identified, payment of an annual exclusive-use fee of 10% of the fair market value of the property is required. At TKC's request, this may be converted into an outright property purchase.

Lyman Resources has existing placer mining leases covering approximately four square miles within the Donlin lease area.

All exploration activities on leased lands are covered under the terms of the lease agreement with Calista and the surface use agreement with TKC. Activities on Native-owned lands not currently within the agreement, or on state and federal lands, are permitted on an individual basis as required. Drilling operations on the project are covered under the Alaska Placer Mining Application process and related permits.

The proposed Donlin Creek operation will require a considerable number of permits and authorizations from both federal and state agencies.

1.5 Environmental Studies

Baseline environmental studies commenced in 1996, comprising water quality studies, meteorology, aquatic studies in the main drainages, wetlands delineation in the areas of the Mineral Resource estimates and some waste rock characterization. During 2003, the baseline program was expanded, and included ambient air monitoring, terrestrial wildlife and avian surveys, groundwater monitoring, detailed aquatic studies, cultural site surveys, detailed waste rock characterization and additional wetlands delineation. Feedback from regulatory and public consultation resulted in additional studies to review the impact of mercury use and the impact of barge traffic on subsistence fishing and river erosion.

The Environmental Management System (EMS) and permit review process will determine the precise number of management plans required to address all aspects of the project to ensure compliance with environmental design and permit criteria.

1.6 Geology and Mineralization

The Kuskokwim region of southwestern Alaska is predominantly underlain by rocks of the Upper Cretaceous Kuskokwim Group that filled a subsided northeast-trending strike-slip basin between a series of amalgamated terranes. Undivided Kuskokwim Group sedimentary rocks and granite porphyry complexes are the main rock units. Greywacke is dominant in the northern part of the area ("northern resource area" comprising Lewis, Queen, Rochelieu, and Akivik), while shale-rich units are common in the southern part of the area ("southern resource area" comprising South Lewis and ACMA). Overall, sedimentary structure in the northern resource area is monoclinal, whereas sedimentary rocks in the southern resource area display open easterly-trending folds.

The Donlin Creek deposits lie between two regional, northeast-trending faults, in an area that contains numerous northeast to east–northeast- and northwest to west–northwest-trending lineaments that probably represent steeply-dipping strike-slip faults. Locally, intermediate composition volcano-plutonic complexes intrude and overlie Kuskokwim Group rocks throughout the region.

Gold deposits are associated with an extensive Late Cretaceous gold–arsenic–antimony–mercury hydrothermal system. Gold-bearing zones exhibit strong structural and host rock control along north–northeast-trending fracture zones and are best developed where those zones intersect relatively competent host rocks. Mineralized material is most abundant in the igneous rocks, but sedimentary rocks are also mineralized within strong fracture zones.

Two distinct styles of gold-rich mineralization (ACMA–Lewis style and Dome–Duqum style) occur within the Donlin Creek trend. The ACMA–Lewis style of mineralization, a later low-temperature, low-sulphidation epithermal system, constitutes the main mineralizing system within the Donlin Creek property. This is the sole style of mineralization within the current resource area. The ACMA–Lewis style consists of sheeted quartz, quartz–carbonate and sulphide-only veins characterized by a gold–arsenic–antimony–mercury geochemical signature. The bulk of the gold occurs in the lattice structure of arsenopyrite. Stibnite, realgar and native arsenic are commonly observed associated with zones of higher-grade gold mineralization but do not appear to host any significant gold mineralization compared to arsenopyrite. Disseminated gold-bearing arsenopyrite can also be found typically adjacent to veins and vein zones.

Mineralization is best developed in all intrusive rocks, and to a lesser extent, sedimentary rocks (mainly greywacke). Sedimentary units in areas of ACMA–Lewis mineralization typically show no contact metasomatic effects.

The Dome–Duqum prospect is best characterized as an early, higher-temperature porphyry style of mineralization with fracture-controlled stockwork, and laminated quartz-only veins containing varying proportions of copper, zinc, bismuth, silver, tellurium, selenium, and local native gold mineralization.

1.7 Exploration History

Gold was discovered in the Donlin Creek area in 1909, and placer production of about 30,000 ounces of gold occurred between 1909 and 1956. From 1956 to 1988, exploration comprised reconnaissance efforts, focusing on first-pass evaluation of ridge tops and outcrops, to determine the lode source of the placer gold.

Exploration in the period 1988 to 2005 comprised airborne geophysical surveys, geological reconnaissance, rock chip, soil and auger sampling, trenching, RC and diamond drilling, environmental studies, petrographic, fluid inclusion and metallurgical studies. Eight prospects, Snow, Dome, Quartz, Carolyn, Queen, Upper Lewis, Lower Lewis, and Rochelieu, were initially identified, followed by the discovery of the ACMA deposit in 1996.

The 2002 work program culminated in a preliminary assessment study, and first time disclosure of Mineral Resources under NI 43–101. An updated Mineral Resource estimate was prepared by Barrick (formerly Placer Dome) during 2005 and SRK Consulting subsequently prepared a preliminary assessment study on behalf of NovaGold, confirming that deposit was amenable to conventional open pit mining operation.

From 2006 to 2008 work programs focused on infill, geotechnical and condemnation drilling, water geochemical studies, peat exploration, wind power generation studies, metallurgical studies, and project economic reviews. The work culminated in an updated feasibility study, completed January 2009 that indicated positive project economics.

1.8 Drilling

Approximately 1,676 exploration and development core (88%) and reverse circulation (RC) (12%) drill holes, totalling 392,937 m were completed from 1988 through 2007. All but 8% (district exploration, carbonate resource, geotechnical, waste rock, condemnation, and hydrology) of this drilling was utilized for the feasibility study resource model. Approximately 50% of the core and 40% of the holes were drilled during the 2006–2007 period. An additional 108 core holes totalling 33,425 m were added in 2008 to explore near-pit expansions and satellite deposits, and for facility-related condemnation and geotechnical studies.

Recoveries were not routinely measured for RC drilling; core recoveries typically ranged between 80% to 100%.

Two specific gravity values have been used, 2.65 for the mineralized intrusive units, and 2.71 for the mineralized sedimentary units, based on wet immersion measurements.

1.9 Sample Preparation and Analysis

The majority of sample preparation for the Donlin Creek Project has taken place at a facility at the Donlin Creek camp. Sample preparation was performed by employees of Barrick or NovaGold, depending on who had project management at the time of sample preparation.

A number of laboratories have performed sample analysis, including Barrick's internal laboratory, and Bondar Clegg/ALS Chemex. The majority of assays in the database were supplied by Bondar Clegg/ALS Chemex.

Gold is typically fire assayed primarily using a fire assay-atomic absorption spectroscopy (AAS) method. The major proportion of trace and major element data for drill holes located within the resource model boundary was acquired prior to the 2005 program by various laboratories using industry standard acid digestions followed by atomic absorption (AA) or inductively coupled plasma (ICP) instrumental determinations.

Standard reference materials were used to monitor the performance of gold analysis. Overall the results were consistent with current industry standards. Blank sample submission indicates limited contamination of samples during the analysis procedures.

1.10 Data Verification

The data verification process has included internal and external reviews.

The drill hole database is considered sufficiently free from error to support Mineral Resource and Mineral Reserve estimation.

1.11 Mineral Resource and Mineral Reserve Estimation

The Mineral Resource block model consists of 6 m x 6 m x 6 m blocks estimated using inverse-distance to the third-power methodology into gold and sulphur discriminator models. The discriminator models were generated by an inverse-distance-squared method to calculate probabilities, which define blocks both internal and external to a probable mineralized envelope. Grades were estimated from multiple passes on each of the major rock groups (intrusive rocks, shale, greywacke), both internal and external to the probable mineralized envelope. Search distances increased with each successive pass.

Variogram ranges were found to be 30 m at 80% of the sill variance and 45 m at 90% of the sill variance. Based on these ranges, the discriminator model, estimation pass, and distance to nearest composite sample were used to classify the blocks to resource confidence categories.

Mineral Resources are based on a Lerchs-Grossmann pit optimized for all Measured, Indicated, and Inferred blocks assuming:

- A gold selling price of US$850/oz
- Mill recoveries in the pit optimization varied by rock type, domain, and degree of oxidation, and ranged from 86.66% to 94.17%
- Administrative costs estimated at $1.56/t
- Refining, freight and marketing (selling costs) were estimated at $0.573/oz recovered
- A royalty of 3.75%, based on the Au price minus the selling cost.

The Mineral Reserves were subtracted from the total Mineral Resources reported from this pit optimization to determine the reported Mineral Resources that are exclusive of Mineral Reserves. During Whittle® pit optimization, incremental cut-offs can be applied to determine whether material within a pit shell is classed as potentially economic mineralization or as waste. The cut-offs assume that all material within a pit will be mined, but that at the top of the exit ramp of a pit, a choice must be made between what will report to the mill as potentially economic mineralization, and what will be sent to dumps as waste. To be considered potentially economic mineralization, the net smelter return (NSR) must pay back the incremental processing cost plus US$0.01/t.

Mineral Resources were classified using criteria appropriate under the 2005 CIM Definition Standards for Mineral Resources and Mineral Reserves by application of the NSR-based cut-off grade that incorporated mining and recovery parameters, and constraint of the Mineral Resources to a pit shell based on commodity prices. Mineral Resources have an effective date of 31 December 2008.

Mineral Resources are summarized in Table 1-1. AMEC cautions that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-1: Mineral Resource Statement, Effective Date 31 December 2008, Gordon Seibel, M.AusIMM

Category	Tonnage (Mt)	Au (g/t)	Contained Au (Moz)
Measured	1.2	2.19	0.08
Indicated	93.4	1.97	5.92
Total Measured and Indicated	**94.6**	**1.98**	**6.01**
Inferred	54.5	2.29	4.02

Note:
1) Mineral Resources are exclusive of Mineral Reserves, and are reported on a 100% basis
2) Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
3) Mineral Resources are reported to an Au price of US$850/oz
4) Sums may not agree due to rounding

Mineral Reserves were estimated based on a series of Lerchs–Grossmann pit shells, established following a number of throughput rationalization studies. The pit shell considered Measured and Indicated Resources. The base case parameters used in the optimizations were:

- Throughput of 53.5 kt/d and 20 year mine life
- Conventional open pit mining using a combined bulk mining (12 m benches) and selective mining (6 m benches) approach
- A long-term gold price assumption of US$725/oz
- Mill recoveries in the pit optimization varied by rock type, domain, and degree of oxidation, and ranged from 86.66% to 94.17%
- Slopes were determined by geotechnical domain, with bench face angle recommendations ranging from 43° to 65°, inter-ramp slope angles from 26° to 50°, and overall slope angles ranging between 26° and 47°
- General and administrative (G&A) costs were estimated at $1.61/t
- Refining, freight and marketing (selling costs) were $0.573/oz recovered
- A royalty of 3.75%, based on the Au price minus the selling cost.

Because of narrow mineralized zones, the deposits were initially modelled with relatively small blocks to ensure that sufficient resolution was available to ensure adequate characterization.

Dilution and ore selectivity was determined using a program "SMUman," developed in-house by Barrick and NCL Ingineria y Construccion S.A. Practical mining areas were designated for selective mining if a significant NSR dollar per tonne benefit over bulk mining was demonstrated. This significant benefit was chosen as being approximately 5%. In general, this benefit occurred in the ACMA deposit, which includes flatter-dipping areas and is less contiguous than the Lewis deposit.

Additional Inferred Mineral Resource tonnes were added to the optimization resource base by to the reclassification of Inferred Mineral Resources to Measured or Indicated Mineral Resources during the "SMUman" 12 m block category allocation. The material was included in the Proven and Probable Mineral Reserve statement, the mining plan, and the financial analysis, and was subtracted from the Mineral Resources tabulations.

The base mining cost (before incremental mining cost with depth) was $1.68/t, the average processing cost was $15.97/t, and the G&A cost was $1.61/t.

Mineral Reserves were classified using criteria appropriate under the 2005 CIM Definition Standards for Mineral Resources and Mineral Reserves, and have an effective date of 31 December 2008. Mineral Reserves are summarized in Table 1-2.

Table 1-2: Proven and Probable Mineral Reserve Statement, Effective Date 31 December 2008, Kirk Hanson, P.E.

Category	Tonnes (Mt)	Au (g/t)	Contained Au (Moz)
Proven	8.4	2.59	0.70
Probable	375.4	2.37	28.57
Total	**383.8**	**2.37**	**29.27**

Note:
1) Mineral Reserves reported to an Au price of US$725/oz
2) Mineral Reserves are reported on a 100% basis
3) Sums may not agree due to rounding

1.12 Open Pit Mine Plan

The mine will be an open pit operation, and is proposed to be mined by a combination of bulk and selective mining methods using owner-operated large-scale equipment.

Two pits are planned, at ACMA and Lewis. A set of 14 mining phases were designed, eight in the ACMA pit and six in the Lewis pit.

This sequence aims to deplete ACMA as early as possible to maximize use of the waste backfill dump designed inside the pit while minimizing deviation from the optimal economic mining sequence. The initial phases of the two pits are independent, but they partially merge later in the mine life.

Open pit mining on both 6 m and 12 m high benches provided the best project economics. About 22% of the total tonnage is planned to be selectively mined on 6 m benches. Blasting will be required.

The operating life-of-mine (LOM) is estimated to be 20 years based on a nominal processing rate of 53.5 kt/d. Mine start-up is assumed to commence in 2014, and cease in 2034. The processing rate is variable from period to period as a function of sulphur grade and ore hardness.

To maximize plant utilization, long-term ore stockpiling is required to balance sulphur feed grades. Short-term stockpiling will also be required to handle crusher downtime and production fluctuations in the pit.

Preproduction covers the first 15 months of the mine plan, when mining activities will focus on providing sufficient ore exposure for plant start-up. Mining is initially focused on the ACMA pit to access the highest-value ore.

Waste rock from open pit mining will be placed in an ex-pit waste rock facility (WRF), in the American Creek valley, east of the pit area, or in a backfill dump in ACMA. Sufficient allocation was made in the WRF design to accommodate non-acid generating (NAG) and potentially acid generating (PAG) rock from the ACMA and Lewis pits. A total of 1.69 Gt of waste will be stored in the WRF and another 404 Mt in the ACMA backfill dump. An engineered rock dam is planned downstream of the WRF to support higher water discharge events.

A proposed tailings storage facility (TSF) in the Anaconda Creek basin will be a fully lined impoundment with cross-valley dams at both the upstream ("upper dam," comprising upper north and upper south) and downstream ("main" dam) ends. The TSF will have an ultimate capacity of 311.43 Mm^3, corresponding to an ultimate impoundment surface area of 549 ha. The total catchment area of the TSF will be 705 ha.

The main objectives of the water management plan for the Donlin Creek project are to minimize or eliminate the need for treatment and discharge of contact water during mine construction, operations, and closure; to achieve the pit-slope depressurization requirements; and to provide adequate quantity and quality of water supply to the mill. Contact water will be stored behind a dam in American Creek, and tailings will be stored in the adjacent Anaconda Creek basin.

Staged diversion structures will be required to divert fresh water out of the project area during construction, operations, and closure.

1.13 Process and Process Plant

Key testwork programs were conducted on Donlin Creek ores at a number of laboratories over a period of approximately eight years, including:

- Mineralogy
- Direct leach/carbon-in-leach (CIL) testwork
- Comminution tests
- Flotation
- Pressure oxidation
- CIL and gold recovery testwork
- Environmental considerations.

The testing has shown that the ores require pre-treatment ahead of cyanidation to recover the gold. The preferred method of pre-treatment is pressure oxidation of the sulphide concentrate produced from flotation. Overall gold recovery is estimated to be 89.5%, based on the combined LOM average recovery of 92.6% from flotation and 96.6% from pressure oxidation of the concentrate.

The process plant is designed to recover a sulphide flotation product and to oxidize the refractory gold concentrate in a pressure oxidation circuit before cyanidation. Key features of the plant are:

- Gyratory crusher feeding a covered stockpile. Design operating times are 65% for the primary crusher and 93% for the process plant.
- Mill–chemical–float–mill–chemical–float (MCF2) grinding and flotation circuit. A single semi-autogenous grind (SAG) mill will operate in closed circuit with parallel cone crushers, followed by a primary ball mill in closed circuit with cyclones. Primary ball mill product reports to primary rougher flotation. Rougher flotation tailings report to the secondary ball mill circuit, while in closed circuit with cyclones. Secondary ball mill product at P_{80} 50 µm reports to secondary rougher flotation. Secondary rougher flotation concentrate reports to cleaner flotation. A cleaner scavenger flotation circuit treats the cleaner flotation tailings.
- Combined flotation concentrates from primary rougher and cleaner flotation are dewatered in a thickener before acidulation and counter-current decant (CCD) washing to remove solubilised ions from the concentrate. It was shown that high levels of certain soluble ions in the feed to the autoclave have detrimental effects

on the pressure oxidation (POX) / CIL gold recovery. The flotation circuit has a concentrate storage option.

- The autoclave circuit includes two autoclaves operating in parallel. Thickened flotation tailings slurry is used as a cooling medium in the autoclave letdown circuit.
- Flotation tailings are combined with the flotation concentrate wash solution product to neutralize the acidic solution before discharge to the tailings storage facility. The carbonate in the flotation tailings slurry will provide primary neutralization. The final pH level will be adjusted by adding slaked lime.
- Flashed and cooled autoclave discharge slurry is cured before POX discharge to CCD. The acidic solution recovered by CCD is recycled to acidulation and flotation feed conditioning.
- POX CCD product slurry is neutralized with lime ahead of cyanidation.
- The CIL circuit will operate at pH 9.0. The CIL tanks are fully enclosed and vented to a caustic scrubbing system to recover cyanide and recycle it back to the CIL circuit. Carbon will be handled with in-tank revolving screens.
- The carbon-handling area for the loaded carbon consists of an acid wash circuit and a modified pressure Zadra circuit for stripping carbon. Carbon is reactivated in an electric kiln.
- Gold is recovered in an electrowinning circuit. The electrowinning sludge is treated in a retort before being melted in an induction furnace. The final product is doré bars.
- Mercury that evolves in the process plant will be captured in a number of mercury abatement systems to treat the following streams: autoclave flash vent, regeneration kiln feed and discharge vents, electrowinning vents, retort furnace exhaust, induction furnace vent, and general refinery area ventilation stream.

1.14 Cost Estimates

The total estimated cost to design and build the Donlin Creek Project is $4,481 million, including an Owner-provided mining fleet and self-performed pre-development (Table 1-3). Sustaining capital requirements total $803 million. The feasibility capital cost estimate was developed in accordance with AACE Class 3 requirements, consisting of semi-detailed unit costs and assembly line items. All costs are expressed in third-quarter (3Q) 2008 U.S. dollars but with a de-escalation allowance applied subsequently to adjust the estimate to fourth-quarter (4Q) 2008 U.S. dollars.

Table 1-3: Summary of Capital Costs by Major Discipline

Discipline	Cost ($000)
Direct Costs	
Civil	383,298
O/L Piping	124,804
Mining	431,636
Concrete	183,043
Structural	181,293
Architectural	105,990
Mechanical	1,104,979
Piping	190,137
Electrical	360,026
Instrumentation	60,641
Coatings	14,986
Total Direct Costs	3,140,833
Indirect Costs	
Owner's Costs	191,921
Project Indirect Costs	925,821
Total Indirect Costs	1,117,742
Subtotal	4,258,575
Contingency @ P50	394,625
Total Project Cost	**4,653,200**
De-escalation @ P50	(172,600)
Net Project Cost	**4,480,600**

No allowances are included for escalation through construction, interest during construction, taxes, or duties. Life-of-mine operating costs, including allocations for mining, processing, administration and refining are estimated at $30.03/t milled, $4.60/t mined, and approximately $440/oz overall (Table 1-4).

Table 1-4: LOM Operating Cost (US$000)

Area	Total LOM	$/t Milled	$/t Mined	$/oz
Mine Operations	5,226,143	13.62	2.08	200
Processing Operations	5,664,194	14.76	2.26	216
Administration	589,596	1.54	0.24	23
Refining	43,858	0.11	0.02	2
Total	**11,523,790**	**30.03**	**4.60**	**440**

The operating cost estimates were assembled by area and component, based on estimated staffing levels, consumables, and expenditures, according to the mine plan and process design. Operating costs were prepared in fourth-quarter 2008 U.S. dollars with no allowances for escalation, sales tax, import duties, or contingency.

1.15 Financial Analysis

The results of the economic analysis represent forward-looking information that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

The overall economic viability of the project was evaluated by both discounted and undiscounted cash flow analyses. The Project is expected to generate net cash flows of $1.1 billion and yield an internal rate of return (IRR) of 2.3%, under a long-term gold price assumption of $725/oz.

The base case after-tax net present value at 5% (NPV 5%) of the Project is a negative $733 million (Table 1-5). At recent gold prices of $900/oz, (Alternative Case 1) the Project has an after-tax NPV (5%) of $829 million and an after-tax internal rate of return (IRR) of 7.7% (Table 1-6). The Project is particularly sensitive to the gold price. For the purposes of the sensitivity analysis, DCLLC assumed that the Project sensitivity to changes in gold grades was mirrored by the sensitivity of the Project to changes in the gold price.

The Project requires a gold price of $670/oz to break even at an oil price of $75/barrel. From the base case of gold at $725/oz and oil at $75/barrel, each $1/barrel increase in the price of oil requires approximately a $1.50/oz increase in the price of gold to offset the impact (Table 1-7).

Table 1-5: Donlin Creek Project Financial Summary (Base Case US$725/oz)

Item	Unit	LOM	$/oz	$/t milled	$/t mined
Total Mined	Mt	2,567.7	-	-	-
Ore Milled	Mt	383.8	-	-	-
Strip Ratio (waste tonnes:ore tonnes)	t:t	5.69	-	-	-
Gold Grade	g/t	2.37	-	-	-
Contained Gold	Moz	29.269	-	-	-
Gold Recovery	%	89.5	-	-	-
Recovered Gold	Moz	26.184	-	-	-
Mine Life	Years	21			
Oil Price	$/barrel	75			
Revenue	$M	18,983	725		
Mining Costs	$M	5,226	200	13.62	2.08
Processing Cost	$M	5,664	216	14.76	2.26
G&A	$M	590	23	1.54	0.24
Refining	$M	44	2	0.11	0.02
Operating Costs	$M	11,524	440	30.03	4.60
Royalties	$M	693	26	1.81	0.28
Total Cash Costs	$M	12,217	467	31.84	4.87
Other Revenue	$M	(156)	(6)	(0.41)	(0.06)
Depreciation (Excluding Sunk Costs)	$M	5,242	200	13.66	2.09
Trust Fund	$M	179	7	0.47	0.07
Total Production Costs	$M	17,481	668	45.55	6.97
Cash Taxes	$M	402	15	1.04	0.16
Working Capital, Net	$M	(2)	-	(0.01)	0.00
Total Costs, Including Taxes and Working Capital*	**$M**	**17,881**	**683**	**46.59**	**7.13**

Table 1-6: Project Sensitivity to Gold Price (US$)

Item	Unit	Base Case	Alternative Case 1	Alternative Case 2
Gold Price	$/oz	725	900	1,000
Oil Price	$/barrel	75	75	75
Undiscounted Cumulative Net Cash Flow Pre-tax	$	1,504	5,915	8,435
Undiscounted Cumulative Net Cash Flow After-tax	$	1,103	4,166	5,876
NPV (5%) Pre-tax	$	(592)	1,525	2,735
NPV (5%) After-tax	$	(733)	829	1,674
IRR Pre-tax	%	3.0	9.4	12.3
IRR After-tax	%	2.3	7.7	10.2
Payback	Years	15	7	5

Note: NPV = net present value, IRR = internal rate of return.

Table 1-7: Project Sensitivity to Oil Price (US$725/oz Au price)

Oil Price ($/barrel)	Net Cash Flow ($M)	NPV @ 5% ($M)	IRR (%)
35	2,106	(236)	4.2
50	1,744	(415)	3.5
75	1,103	(733)	2.3
100	430	(1,069)	0.9

Note: NPV = net present value, IRR = internal rate of return

1.16 Recommendations

A recommended work program was budgeted in two phases, and totals $2.5 million. The work primarily involves additional studies and design to support Project advancement.

2.0 INTRODUCTION

AMEC Americas Limited (AMEC) was commissioned by NovaGold Resources Inc. (NovaGold), to provide an independent Qualified Person's Review and Technical Report (the Report) for the Donlin Creek Gold Project (the Project) located in Alaska, USA (Figure 2-1).

The Report was prepared in compliance with National Instrument 43–101, Standards of Disclosure for Mineral Projects (NI 43–101) and documents the results of an updated feasibility study on the Project. AMEC understands that this Report will be used by NovaGold in support of a press release entitled "Donlin Creek Feasibility Study Adds 14.7 Mozs to NovaGold's Reserves", dated 28 April 2009.

The Project is a joint venture between NovaGold Resources Alaska, Inc., a wholly-owned subsidiary of NovaGold Resources Inc., and Barrick Gold U.S. Inc., a wholly-owned subsidiary of Barrick Gold Corporation.

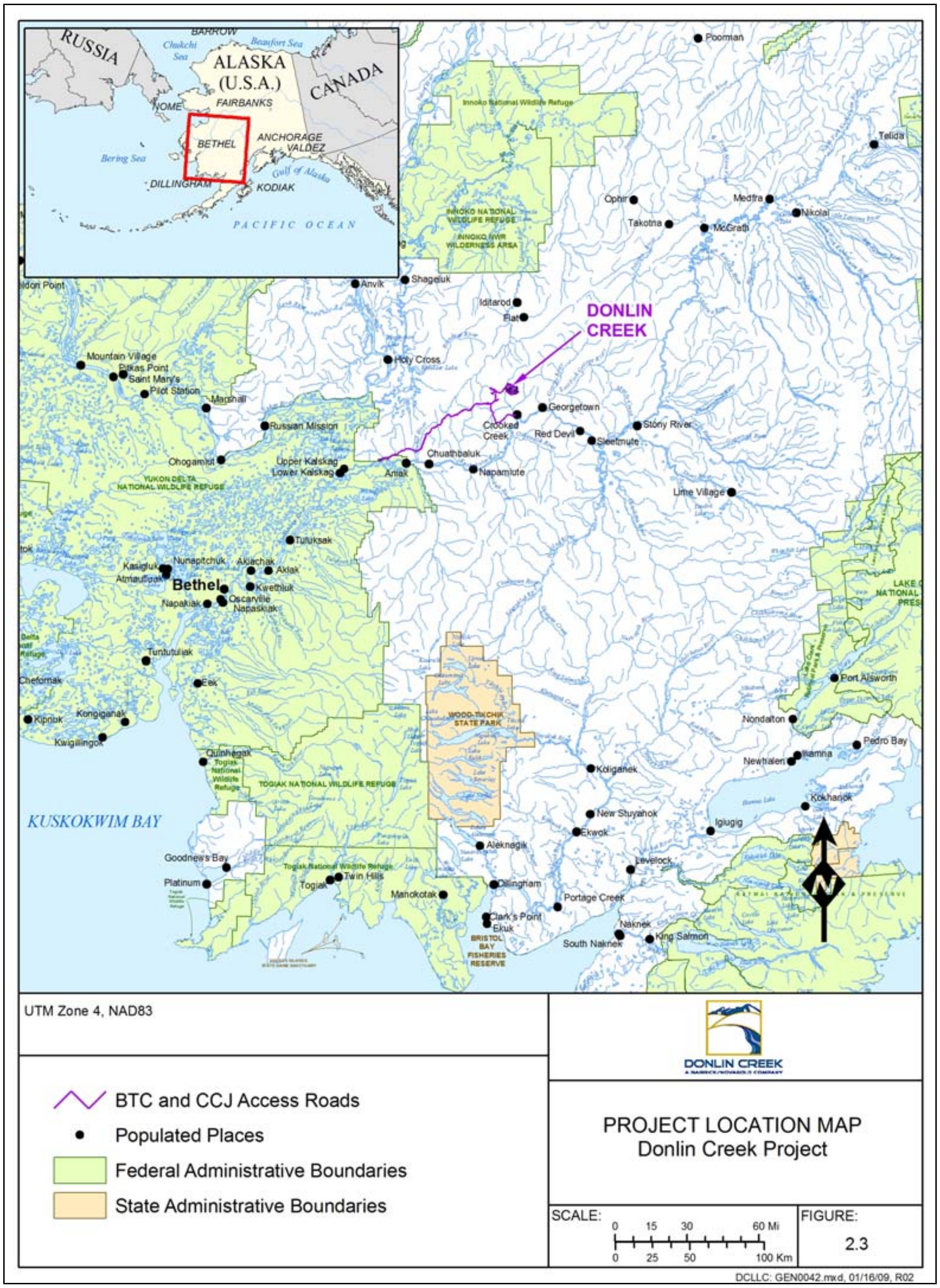
For the purposes of this report, the name "NovaGold" refers interchangeably to the NovaGold subsidiary and parent companies; and the name "Barrick" refers interchangeably to the Barrick subsidiary and parent companies.

During 2006, Barrick acquired Placer Dome Inc., which had held an interest in the Project between 1995 and 2006 through its wholly-owned subsidiary, Placer Dome US Inc. For the purposes of this report, the name "Barrick (formerly Placer Dome)" refers interchangeably to Placer Dome Inc. and to Placer Dome US Inc. when reference is made to the manager of work undertaken on the Project between 1995 and 2000, and between 2003 and 2006.

On 1 December 2007, NovaGold entered into a limited liability company agreement with Barrick that provided for the conversion of the Project into a new limited liability company, the Donlin Creek LLC (DCLLC), which is jointly owned by the NovaGold and Barrick on a 50/50 basis. The limited liability company agreement provides that information obtained in connection with the performance of the agreement, which includes information developed by, or on behalf of the DCLLC, may be disclosed by either NovaGold or Barrick, or affiliates of either, where such disclosure is required by law or to meet stock exchange requirements, including for purposes of a technical report required under NI 43-101.

The Project is located in the USA, which uses U.S. Imperial measurements. Unless specified, all measurements in this Report were converted to the metric system. The Report currency throughout the Report is expressed in U.S. dollars; the Report uses Canadian English.

Figure 2-1: Location Map



Note: Figure courtesy DCLLC

2.1 Qualified Persons

The Qualified Persons (QPs), as defined in NI 43–101 and in compliance with Form 43–101F1 Technical Report, responsible for the preparation of the Report include:

- Gordon Seibel, M.AusIMM., Principal Geologist (AMEC, Reno)
- Kirk Hanson, P.E., Principal Mining Engineer, (AMEC, Reno)
- Simon Allard, P.Eng., Senior Financial Analyst (AMEC, Vancouver)
- Greg Wortman, P.Eng., Technical Director, Process (AMEC, Santiago)
- Alexandra Kozak, P.Eng., Manager, Process Engineering (AMEC, Vancouver)

2.2 Site Visits

AMEC QPs have conducted site visits to the Donlin Creek Project as shown in Table 2-1.

Table 2-1: Dates of Site Visits and Areas of Responsibility

QP Name	Site Visit Date	Area of Responsibility
Kirk Hanson	1 October 2008	Sections 1, 2, 3, 4, 5, 6, 17.2, 18.1 to 18.8, 19, 20, 21, 22 and 23 and those portions of the summary, conclusions and recommendations that pertain to those sections.
Gordon Siebel	1 October 2008	Sections 7, 8, 9, 10, 11, 12, 13, 14, 15, 17.1 and those portions of the summary, conclusions and recommendations that pertain to those sections.
Simon Allard	No site visit	Sections 18.9 to 18.12 and those portions of the summary, conclusions and recommendations that pertain to those sections.
Greg Wortman	No site visit	Section 16.1.3 and those portions of the summary, conclusions and recommendations that pertain to that section.
Alexandra Kozak	No site visit	Sections 16.1.1, 16.1.2, 16.1.4 to 16.3 and those portions of the summary, conclusions and recommendations that pertain to those sections.

2.3 Effective Dates

The Report has a number of effective dates. The effective date for the Mineral Resources and Mineral Reserves is 31 December, 2008. The feasibility study update completion date for financial information included in the Report is 24 February 2009. The date of last supply of significant information to the Report, comprising Project tenure details, was 1 April 2009. The effective date for the Report is therefore 1 April 2009.

There were no material changes to the information on the Project between the effective date and the signature date of the Report.

2.4 Previous Technical Reports

A number of previous Technical Reports were filed on the Donlin Creek Project:

Francis, K., 2007: Donlin Creek Project, NI 43-101 Technical Report, Southwest Alaska, U.S: unpublished NI43-101F1 Technical Report to NovaGold Resources Inc., 8 February 2008

Dodd, S., Francis, K. and Doerksen, G., 2006: Preliminary Assessment Donlin Creek Gold Project Alaska, USA, unpublished NI43-101F1 Technical Report to NovaGold Resources Inc. by SRK Consulting (US), Inc., 20 September 2006

Dodd, S., 2006: Donlin Creek Project 43-101 Technical Report, unpublished NI43-101F1 Technical Report to NovaGold Resources Inc. by NovaGold Resources Inc., 19 January 2006

Juras, S. and Hodgson, S., 2002: Technical Report, Preliminary Assessment, Donlin Creek Project, Alaska, unpublished NI43-101F1 Technical Report to NovaGold Resources Inc. by MRDI, March 2002

Juras, S., 2002: Technical Report, Donlin Creek Project, Alaska, unpublished NI43-101F1 Technical Report to NovaGold Resources Inc. by MRDI, 24 January 2002

AMEC has sourced information from these reports and other reference documents are as cited in the text and summarized in Section 22 of this Report. Additional information was sourced from, and provided by, NovaGold and the DCLLC. AMEC has relied upon other experts in the fields of mineral tenure, surface rights, permitting, and environmental as outlined in Section 3.

2.5 Technical Report Sections and Required Items under NI 43-101

Table 2-2 relates the sections as shown in the contents page of this Report to the Prescribed Items Contents Page of NI 43-101.

Table 2-2: Contents Page Headings in Relation to NI 43-101 Prescribed Items—Contents

NI 43-101 Item Number	NI 43-101 Heading	Report Section Number	Report Section Heading
Item 1	Title Page		Cover page of Report
Item 2	Table of Contents		Table of contents
Item 3	Summary	Section 1	Summary
Item 4	Introduction	Section 2	Introduction
Item 5	Reliance on Other Experts	Section 3	Reliance on Other Experts
Item 6	Property Description and Location	Section 4	Property Description and Location
Item 7	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Section 5	Accessibility, Climate, Local Resources, Infrastructure and Physiography
Item 8	History	Section 6	History
Item 9	Geological Setting	Section 7	Geological Setting
Item 10	Deposit Types	Section 8	Deposit Types
Item 11	Mineralization	Section 9	Mineralization
Item 12	Exploration	Section 10	Exploration
Item 13	Drilling	Section 11	Drilling
Item 14	Sampling Method and Approach	Section 12	Sampling Method and Approach
Item 15	Sample Preparation, Analyses and Security	Section 13	Sample Preparation, Analyses and Security
Item 16	Data Verification	Section 14	Data Verification
Item 17	Adjacent Properties	Section 15	Adjacent Properties
Item 18	Mineral Processing and Metallurgical Testing	Section 16	Mineral Processing and Metallurgical Testing
Item 19	Mineral Resource and Mineral Reserve Estimates	Section 17	Mineral Resource and Mineral Reserve Estimates
Item 20	Other Relevant Data and Information	Section 19	Other Relevant Data and Information
Item 21	Interpretation and Conclusions	Section 20	Interpretation and Conclusions
Item 22	Recommendations	Section 21	Recommendations
Item 23	References	Section 22	References
Item 24	Date and Signature Page	Section 23	Date and Signature Page
Item 25	Additional Requirements for Technical Reports on Development Properties and Production Properties	Section 18	Additional Requirements for Technical Reports on Development Properties and Production Properties
Item 26	Illustrations		Incorporated in Report under appropriate section number

3.0 RELIANCE ON OTHER EXPERTS

The QPs, authors of this Report, state that they are qualified persons for those areas as identified in their respective "Certificate of Qualified Person" attached to this Report. The authors have relied upon and disclaim responsibility for information derived from the following expert reports pertaining to mineral rights, surface rights, and permitting issues.

3.1 Mineral Tenure

AMEC QPs have not reviewed the mineral tenure, nor independently verified the legal status or ownership of the Project area or underlying property agreements. AMEC has fully relied upon legal experts for this information through the following document:

Reeves Amodio LLC, 2009: Title Opinion: unpublished document prepared for Donlin Creek LLC, Barrick Gold US Inc., and NovaGold Alaska Inc., 23 February 2009, 49 p

This information was used in Sections 4.2.2 to 4.2.4 and Section 4.3.6 of the Report.

3.2 Surface Rights, Access, and Permitting

AMEC QPs have fully relied on information regarding the status of the current Surface Rights, Road Access and Permits through opinions and data supplied by legal experts through the following document:

Reeves Amodio LLC, 2009: Title Opinion: unpublished document prepared for Donlin Creek LLC, Barrick Gold US Inc., and NovaGold Alaska Inc., 23 February 2009, 49 p

Donlin Creek Feasibility Study Update, Section 2, Introduction: unpublished report to the DCLLC, 21 May, 2009

Donlin Creek Feasibility Study Update, Section 13 Environmental and Permitting: unpublished report to the DCLLC, 21 May, 2009

This information was used in Sections 4.2.2 to 4.2.4 and Section 4.4 of the Report.

3.3 Environmental and Permitting

AMEC QPs have fully relied on information regarding the environmental and permitting status of the Project through opinions and data supplied by independent experts to the DCLLC as part of the feasibility study update, through the following:

Donlin Creek Feasibility Study Update, Section 2, Introduction: unpublished report to the DCLLC, 21 May, 2009

Donlin Creek Feasibility Study Update, Section 13 Environmental and Permitting: unpublished report to the DCLLC, 21 May, 2009

This information was used in Sections 4.2.2 to 4.2.4, Section 4.4, Section 4.5 and Section 18.8 of the Report.

3.4 Reclamation and Closure

AMEC QPs have fully relied on information regarding the reclamation and closure proposals for the Project through opinions and data supplied by independent experts to the DCLLC as part of the feasibility study update, through the following:

Donlin Creek Project Feasibility Study Update, Section 14 Closure Plan: unpublished report to the DCLLC, 21 May, 2009

This information was used in Sections 4.5 and Section 18.8 of the Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

Donlin Creek is located in southwest Alaska in the United States of America, approximately 80 km northeast of Aniak, a regional hub (see Figure 2-1). The property consists of about 109 km^2 of privately-owned Native Alaskan land. Calista Corporation (Calista), a regional Native corporation, owns the subsurface rights, and The Kuskokwim Corporation (TKC), a village corporation, owns the majority of the surface rights.

The Mineral Resource is located within Township (T) 23 North (N), Range (R) 49. Seward Meridian, Kuskokwim and Mt. McKinley Recording Districts, Crooked Creek Mining District, Iditarod A-5 United States Geological Survey (USGS) 1:63,360 topography map. These areas consist of the ACMA and 400 Zone, Aurora, and Akivik prospects (grouped as ACMA) and the Lewis, South Lewis, Vortex, Rochelieu, and Queen prospects (grouped as Lewis) and shown in Figure 4-1.

4.2 Mineral Tenure

4.2.1 Tenure History

Barrick (formerly Placer Dome) acquired a 20 year lease from Calista in 1995. Subsequently, in 2002, NovaGold joint ventured into the property.

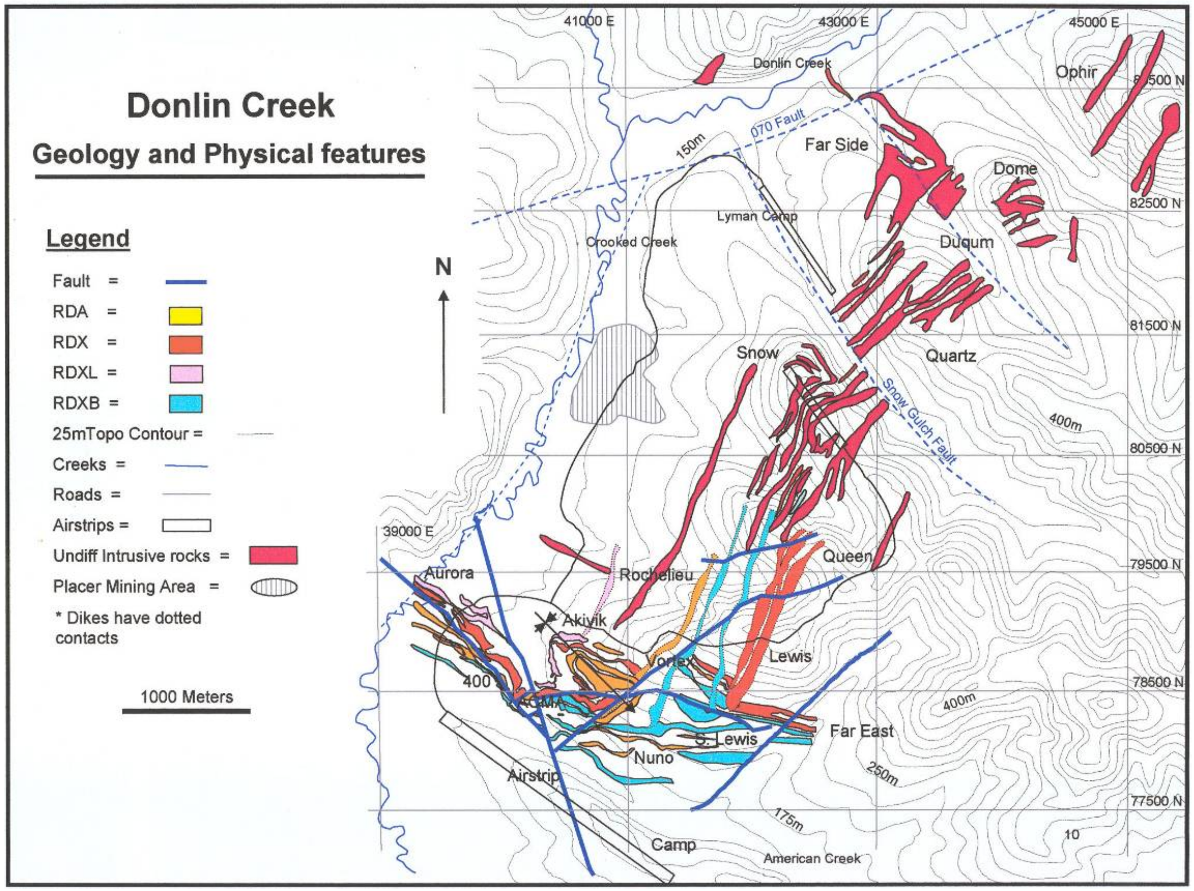
4.2.2 Current Tenure

The current land status held by the DCLLC in the area of the Project is shown in Figure 4-2. Land status in the greater region to Birch Tree Crossing (BTC) is shown in Figure 4-3.

4.2.3 Proposed Mining Operation Area

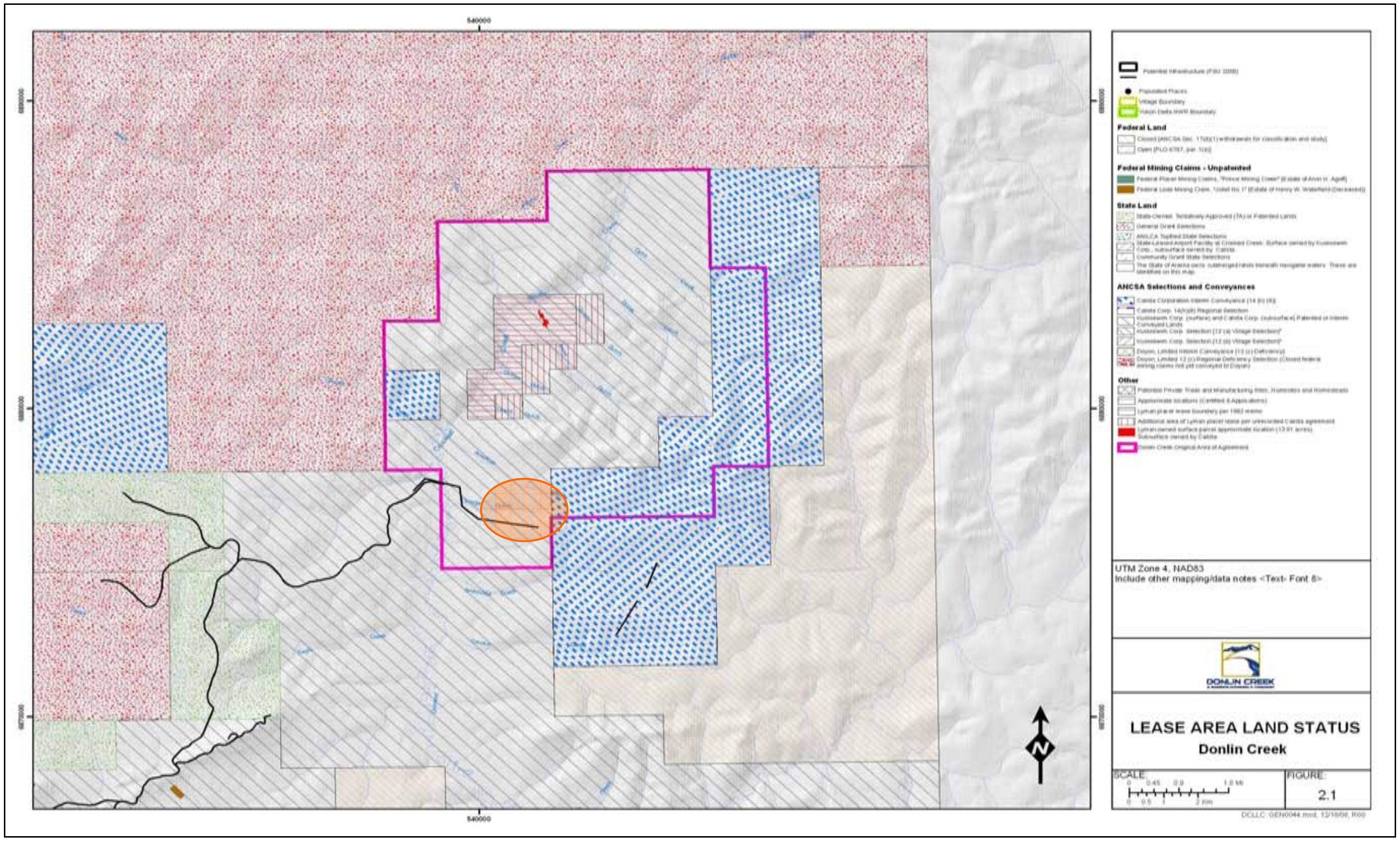
Most of the rights (surface and subsurface) are governed by conditions defined by the Alaska Native Claims Settlement Act (ANCSA). Section 12(a) of ANCSA entitled each village corporation to select surface estate land from an area proximal to the village in an amount established by its population size. Calista receives conveyance of the subsurface when the surface estate in those lands is conveyed to the village corporation. Section 12(b) of ANCSA allocated a smaller entitlement to the regional corporations with the requirement they reallocate it to their villages as they choose. Calista receives subsurface estate when its villages receive 12(b) lands.

Figure 4-1: Deposit and Prospect Location Map



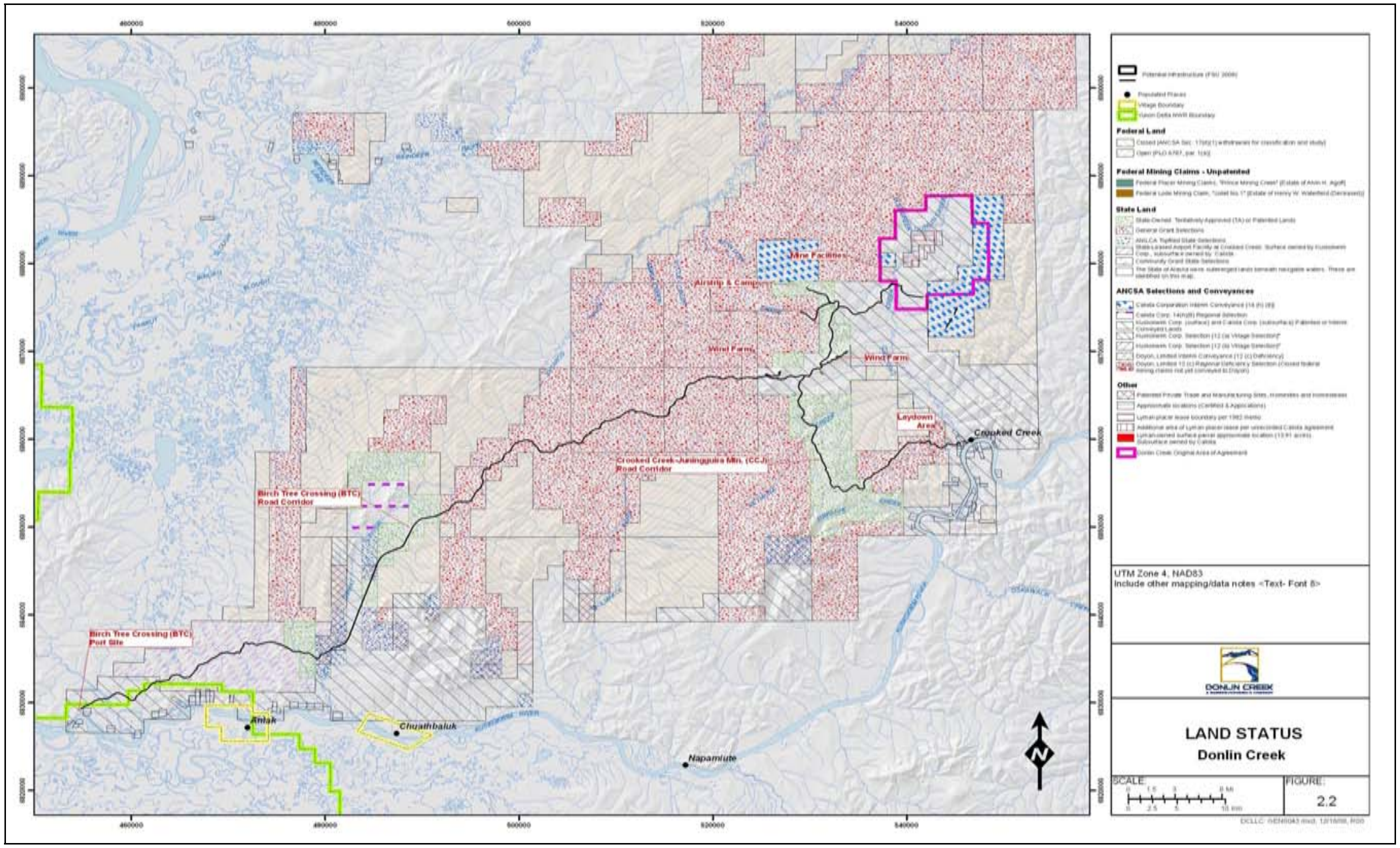
Note: Figure courtesy DCLLC, sourced from Piekenbrock and Petsel 2003. Key: RDA = Aphanitic Porphyry; RDX = Crowded Porphyry; RDXL = Lath-Rich Porphyry; RDXB = Blue Porphyry

Figure 4-2: Project Tenure Map



Note: Figure courtesy DCLLC. Key shading: orange oval marks the approximate position of the ACMA–Lewis deposits in relation to the tenure boundaries. Pink outline on plan is the area of the original Donlin Creek Agreement' blue shading indicates Calista Corporation conveyance; red striping is Lyman lease areas; yellow striping is Federal lands; red stipples are general grant selections; green stipples are state owned, tentatively approved or patented lands.

Figure 4-3: Project Regional Tenure Map Showing Proposed Facility Locations



Note: Figure courtesy DCLLC. Key shading: pink outline on plan is the area of the original Donlin Creek Agreement' blue shading indicates Calista Corporation conveyance; red striping is Lyman lease areas; yellow striping is Federal lands; red stipples are general grant selections; green stipples are State-owned, tentatively approved or patented lands; blue stipples are topfiled State Selections; yellow outline is village lands; green outline is Yukon Delta NWR boundary. Infrastructure noted is proposed.

Calista re-allocated its 12(b) entitlement in 1999 according to a formula based on original village corporation enrolments.

The Donlin Creek exploration and mining lease currently includes a total of 42 sections leased from Calista, an Alaska Native Corporation that holds the subsurface (mineral) estate for Native-owned lands in the region (Table 4-1). Title to all of these sections was conveyed to Calista by the Federal Government. Calista owns the surface estate on nine of these 42 sections.

The Calista Exploration and Lode Mining Lease, dated 11 January 1996 and effective 1 May 1995, between Calista and Placer Dome (now Barrick) covers the following areas of the Seward Meridian, situated within the Kuskokwim Recording District, Fourth Judicial District, State of Alaska:

- Township 22 North, Range 48 West – Sections 5 and 6
- Township 22 North, Range 49 West – Sections 1, 2, 3, 10, and 11
- Township 23 North, Range 48 West – Sections 5, 6, 7, 8, 16, 17, 18, 19, 20, 21, 28, 29, 30, 31, 32, and 33
- Township 23 North, Range 49 West – Sections 1, 10, 11, 12, 13, 14, 15, 21, 22, 23, 24, 25, 26, 27, 28, 33, 34, 35, and 36.

A separate Surface Use Agreement with TKC, an Alaska Native Village Corporation that owns the surface estate of the remaining 33 sections, grants non-exclusive surface use rights to the DCLLC. All of these sections have now been conveyed to Calista/TKC by the Federal Government. The TKC surface use agreement between TKC and Barrick covers the following areas of the Seward Meridian:

- Township 22 North, Range 49 West – Sections 2, 3, 10, and 11
- Township 23 North, Range 48 West – Sections 5, 6, 7, 8, 17, 18, 19, 20, 29, 30, and 31
- Township 23 North, Range 49 West – Sections 1, 10, 11, 12, 13, 14, 15, 21, 22, 23, 24, 25, 26, 27, 33, 34, 35, and 36.

The TKC agreement does not include lands which TKC is required to convey pursuant to Section 14(c) of the Alaska Native Claims Settlement Act.

Calista and TKC agreement lands (the “leased lands”) are approximately 10,858 ha in extent. Land title was verified, and is summarized in Table 4-1.

Table 4-1: Donlin Creek Leased Land

Land Description	Grantee	Conveyance Document	Date Recorded	Kuskokwim Recording District Document No.
T. 22N, R. 48W, Sections 5 and 6 (surface & subsurface estates)	Calista Corporation	Patent No. 50-2007-0148	02/05/2007	2007-000024-0
T. 22N, R. 49W, Section 1 (surface & subsurface estates)	Calista Corporation	Patent No. 50-2007-0148	02/05/2007	2007-000024-0
T. 22N, R. 49W, Sections 2, 3, 10, and 11) (subsurface estate)	Calista Corporation	Patent No. 50-94-0010	01/20/1994	1994-000018-0
T. 22N, R. 49W, Sections 2, 3, 10, and 11) (surface estate)	TKC	Patent No. 50-94-0009	Not recorded	N/A
T. 23N, R. 48W, Sections 17-20, 29-31 (subsurface estate)	Calista Corporation	Patent No. 50-94-0010	01/20/1994	1994-000018-0
T. 23N, R. 48W, Sections 17-20, 29-31 (surface estate)	TKC	Patent No. 50-94-0009	Not recorded	N/A
T. 23N, R. 48W, Sections 5-8 (subsurface estate)	Calista Corporation	Patent No. 50-2007-0214	12/09/2008	2008-001007-0
T. 23N, R. 48W, Sections 5-8 (surface estate)	TKC	Patent No. 50-2007-0213	01/30/2007	2007-000238-0
T. 23N, R. 48W, Sections 16, 21, 28, 32, and 33 (surface & subsurface estates)	Calista Corporation	Patent No. 50-2007-0148	02/05/2007	2007-000024-0
T. 23N, R. 49W, Section 28 (surface & subsurface estates)	Calista Corporation	Patent No. 50-2007-0148	02/05/2007	2007-000024-0
T. 23N, R. 49W, Sections 10, 13-15, 21-27, and 33-36 (subsurface estate)*	Calista Corporation	Patent No. 50-94-0010	01/20/1994	1994-000018-0
T. 23N, R. 49W, Sections 10, 13-15, 21-27, and 33-36 (surface estate)	TKC	Patent No. 50-94-0009	Not recorded	N/A
T. 23N, R. 49W, Sections 1, 11, and 12 (subsurface estate)	Calista Corporation	Patent No. 50-2007-0214	12/09/2008	2008-001007-0
T. 23N, R. 49W, Sections 1, 11, and 12 (surface estate)	TKC	Patent No. 50-2007-0213	01/30/2007	2007-000238-0

Note: * The surface estate within Section 14, SE ¼ and Section 23, NE ¼ is owned by Spencer and Carolyn Motherway Lyman. Spencer and Carolyn Motherway Lyman acquired an additional 2.21 acres of the surface estate within the same lands

The DCLLC holds unsurveyed State of Alaska mining claims within the Seward Meridian, comprising 158 claims (7,808.4 ha) primarily surrounding the leased land in the Kuskokwim and Mt. McKinley recording districts (Table 4-2). These claims have not been legally surveyed. All claims are either 40 acres (16.2 ha) or 160 acres (64.8 ha) in size.

Within the leased lands are a number of encumbrances that are granted to third parties. These are primarily trail, road, right-of-way, and airport easements. One acre site easements also exist along the banks of the Kuskokwim River.

Table 4-2: Donlin Creek State Mining Locations

ADL #	Claim Name	Status	Located/T.A.	Size in (ha)	T.	R.	Sec.	Recording District
578768	DNC 1	T.A.	39308	16.2	24N	47W	31	Kuskokwim
578769	DNC 2	T.A.	39308	16.2	24N	47W	31	Kuskokwim
578770	DNC 3	T.A.	39273	16.2	24N	48W	36	Kuskokwim
578771	DNC 4	T.A.	39273	16.2	24N	48W	36	Kuskokwim
578772	DNC 5	T.A.	39273	16.2	24N	48W	36	Kuskokwim
578773	DNC 6	T.A.	39273	16.2	24N	48W	36	Kuskokwim
578774	DNC 7	T.A.	39273	16.2	24N	48W	35	Kuskokwim
578775	DNC 8	T.A.	39273	16.2	24N	48W	35	Kuskokwim
578776	DNC 9	T.A.	39273	16.2	24N	48W	35	Kuskokwim
578777	DNC 10	T.A.	39273	16.2	24N	48W	35	Kuskokwim
578778	DNC 11	T.A.	39273	16.2	24N	48W	36	Kuskokwim
578779	DNC 12	T.A.	39273	16.2	24N	48W	36	Kuskokwim
578780	DNC 13	T.A.	39273	16.2	24N	48W	36	Kuskokwim
578781	DNC 14	T.A.	39273	16.2	24N	48W	36	Kuskokwim
578782	DNC 15	T.A.	39308	16.2	24N	47W	31	Kuskokwim
578783	DNC 16	T.A.	39308	16.2	24N	47W	31	Kuskokwim
578784	DNC 17	T.A.	39308	16.2	24N	47W	31	Kuskokwim
578785	DNC 18	T.A.	39308	16.2	24N	47W	31	Kuskokwim
578786	DNC 19	T.A.	39273	16.2	24N	48W	36	Kuskokwim
578787	DNC 20	T.A.	39273	16.2	24N	48W	36	Kuskokwim
578788	DNC 21	T.A.	39273	16.2	24N	48W	36	Kuskokwim
578789	DNC 22	T.A.	39273	16.2	24N	48W	36	Kuskokwim
578790	DNC 23	T.A.	39273	16.2	24N	48W	35	Kuskokwim
578791	DNC 24	T.A.	39273	16.2	24N	48W	35	Kuskokwim
578792	DNC 25	T.A.	39273	16.2	24N	48W	35	Kuskokwim
578793	DNC 26	T.A.	39273	16.2	24N	48W	35	Kuskokwim
578794	DNC 27	T.A.	39273	16.2	24N	48W	36	Kuskokwim
578795	DNC 28	T.A.	39273	16.2	24N	48W	36	Kuskokwim
578796	DNC 29	T.A.	39273	16.2	24N	48W	36	Kuskokwim
578797	DNC 30	T.A.	39273	16.2	24N	48W	36	Kuskokwim
578798	DNC 31	T.A.	39308	16.2	24N	47W	31	Kuskokwim
578799	DNC 32	T.A.	39308	16.2	24N	47W	31	Kuskokwim
578800	DNC 33	T.A.	39442	16.2	23N	48W	1	Kuskokwim
578801	DNC 34	T.A.	39442	16.2	23N	48W	1	Kuskokwim
578802	DNC 35	T.A.	39442	16.2	23N	48W	2	Kuskokwim
578803	DNC 36	T.A.	39442	16.2	23N	48W	2	Kuskokwim
578804	DNC 37	T.A.	39442	16.2	23N	48W	2	Kuskokwim
578805	DNC 38	T.A.	39442	16.2	23N	48W	2	Kuskokwim
578806	DNC 39	T.A.	39442	16.2	23N	48W	2	Kuskokwim
578807	DNC 40	T.A.	39442	16.2	23N	48W	2	Kuskokwim
578808	DNC 41	T.A.	39442	16.2	23N	48W	1	Kuskokwim
578809	DNC 42	T.A.	39442	16.2	23N	48W	1	Kuskokwim
578810	DNC 43	T.A.	39442	16.2	23N	48W	2	Kuskokwim
578811	DNC 44	T.A.	39442	16.2	23N	48W	2	Kuskokwim
578812	DNC 45	T.A.	39442	16.2	23N	48W	2	Kuskokwim
578813	DNC 46	T.A.	39442	16.2	23N	48W	2	Kuskokwim
578814	DNC 47	T.A.	39442	16.2	23N	48W	2	Kuskokwim
578815	DNC 48	T.A.	39442	16.2	23N	48W	2	Kuskokwim
578816	DNC 49	T.A.	39442	16.2	23N	48W	11	Kuskokwim
578817	DNC 50	T.A.	39442	16.2	23N	48W	11	Kuskokwim
644952	GROUSE 1	S.S.	38111	64.8	23N	50W	35	Kuskokwim/Mt. McKinley
644953	GROUSE 2	S.S.	38111	64.8	23N	50W	35	Kuskokwim
644954	GROUSE 3	S.S.	38111	64.8	23N	50W	36	Kuskokwim

ADL #	Claim Name	Status	Located/T.A.	Size in (ha)	T.	R.	Sec.	Recording District
644955	GROUSE 4	S.S.	38111	64.8	23N	50W	36	Kuskokwim
644956	GROUSE 5	S.S.	38111	64.8	23N	49W	31	Kuskokwim
644957	GROUSE 6	S.S.	38111	64.8	23N	49W	31	Kuskokwim
644958	GROUSE 7	S.S.	38111	64.8	23N	49W	32	Kuskokwim
644959	GROUSE 8	S.S.	38110	64.8	23N	49W	32	Kuskokwim
644960	GROUSE 9	S.S.	38111	64.8	23N	50W	35	Kuskokwim/Mt. McKinley
644961	GROUSE 10	S.S.	38111	64.8	23N	50W	35	Kuskokwim
644962	GROUSE 11	S.S.	38111	64.8	23N	50W	36	Kuskokwim
644963	GROUSE 12	S.S.	38111	64.8	23N	50W	36	Kuskokwim
644964	GROUSE 13	S.S.	38111	64.8	23N	49W	31	Kuskokwim
644965	GROUSE 14	S.S.	38111	64.8	23N	49W	31	Kuskokwim
644966	GROUSE 15	S.S.	38111	64.8	23N	49W	32	Kuskokwim
644967	GROUSE 16	S.S.	38110	64.8	23N	49W	32	Kuskokwim
644968	GROUSE 17	S.S.	38111	64.8	23N	50W	26	Kuskokwim/Mt. McKinley
644969	GROUSE 18	S.S.	38111	64.8	23N	50W	26	Kuskokwim
644970	GROUSE 19	S.S.	38111	64.8	23N	50W	25	Kuskokwim
644971	GROUSE 20	S.S.	38111	64.8	23N	50W	25	Kuskokwim
644972	GROUSE 21	S.S.	38111	64.8	23N	49W	30	Kuskokwim
644973	GROUSE 22	S.S.	38111	64.8	23N	49W	30	Kuskokwim
644974	GROUSE 23	S.S.	38111	64.8	23N	49W	29	Kuskokwim
644975	GROUSE 24	S.S.	38110	64.8	23N	49W	29	Kuskokwim
644976	GROUSE 25	S.S.	38111	64.8	23N	50W	26	Kuskokwim/Mt. McKinley
644977	GROUSE 26	S.S.	38111	64.8	23N	50W	26	Kuskokwim
644978	GROUSE 27	S.S.	38111	64.8	23N	50W	25	Kuskokwim
644979	GROUSE 28	S.S.	38111	64.8	23N	50W	25	Kuskokwim
644980	GROUSE 29	S.S.	38111	64.8	23N	49W	30	Kuskokwim
644981	GROUSE 30	S.S.	38111	64.8	23N	49W	30	Kuskokwim
644982	GROUSE 31	S.S.	38111	64.8	23N	49W	29	Kuskokwim
644983	GROUSE 32	S.S.	38110	64.8	23N	49W	29	Kuskokwim
644984	GROUSE 33	S.S.	38111	64.8	23N	50W	23	Kuskokwim/Mt. McKinley
644985	GROUSE 34	S.S.	38111	64.8	23N	50W	23	Kuskokwim/Mt. McKinley
644986	GROUSE 35	S.S.	38111	64.8	23N	50W	24	Kuskokwim
644987	GROUSE 36	S.S.	38111	64.8	23N	50W	24	Kuskokwim
644988	GROUSE 37	S.S.	38111	64.8	23N	49W	19	Kuskokwim
644989	GROUSE 38	S.S.	38111	64.8	23N	49W	19	Kuskokwim
644990	GROUSE 39	S.S.	38111	64.8	23N	49W	20	Kuskokwim
644991	GROUSE 40	S.S.	38110	64.8	23N	49W	20	Kuskokwim
644992	GROUSE 41	S.S.	38111	64.8	23N	50W	23	Mt. McKinley
644993	GROUSE 42	S.S.	38111	64.8	23N	50W	23	Kuskokwim/Mt. McKinley
644994	GROUSE 43	S.S.	38111	64.8	23N	50W	24	Kuskokwim/Mt. McKinley
644995	GROUSE 44	S.S.	38111	64.8	23N	50W	24	Kuskokwim
644996	GROUSE 45	S.S.	38111	64.8	23N	49W	19	Kuskokwim
644997	GROUSE 46	S.S.	38111	64.8	23N	49W	19	Kuskokwim
644998	GROUSE 47	S.S.	38111	64.8	23N	49W	20	Kuskokwim
644999	GROUSE 48	S.S.	38110	64.8	23N	49W	20	Kuskokwim
645000	GROUSE 49	S.S.	38111	64.8	23N	50W	14	Mt. McKinley
645001	GROUSE 50	S.S.	38111	64.8	23N	50W	14	Mt. McKinley
645002	GROUSE 51	S.S.	38111	64.8	23N	50W	13	Kuskokwim/Mt. McKinley
645003	GROUSE 52	S.S.	38111	64.8	23N	50W	13	Kuskokwim/Mt. McKinley
645004	GROUSE 53	S.S.	38111	64.8	23N	49W	18	Kuskokwim/Mt. McKinley
645005	GROUSE 54	S.S.	38111	64.8	23N	49W	18	Kuskokwim
645006	GROUSE 55	S.S.	38111	64.8	23N	49W	17	Kuskokwim
645007	GROUSE 56	S.S.	38110	64.8	23N	49W	17	Kuskokwim
645008	GROUSE 57	S.S.	38110	64.8	23N	49W	16	Kuskokwim

ADL #	Claim Name	Status	Located/T.A.	Size in (ha)	T.	R.	Sec.	Recording District
645009	GROUSE 58	S.S.	38110	64.8	23N	49W	16	Kuskokwim
645010	GROUSE 59	S.S.	38111	64.8	23N	50W	14	Mt. McKinley
645011	GROUSE 60	S.S.	38111	64.8	23N	50W	14	Mt. McKinley
645012	GROUSE 61	S.S.	38111	64.8	23N	50W	13	Mt. McKinley
645013	GROUSE 62	S.S.	38111	64.8	23N	50W	13	Mt. McKinley
645014	GROUSE 63	S.S.	38111	64.8	23N	49W	18	Kuskokwim/Mt. McKinley
645015	GROUSE 64	S.S.	38111	64.8	23N	49W	18	Kuskokwim
645016	GROUSE 65	S.S.	38111	64.8	23N	49W	17	Kuskokwim
645017	GROUSE 66	S.S.	38110	64.8	23N	49W	17	Kuskokwim
645018	GROUSE 67	S.S.	38110	64.8	23N	49W	16	Kuskokwim
645019	GROUSE 68	S.S.	38110	64.8	23N	49W	16	Kuskokwim
645020	GROUSE 69	S.S.	38111	64.8	23N	50W	11	Mt. McKinley
645021	GROUSE 70	S.S.	38111	64.8	23N	50W	11	Mt. McKinley
645022	GROUSE 71	S.S.	38111	64.8	23N	50W	12	Mt. McKinley
645023	GROUSE 72	S.S.	38111	64.8	23N	50W	12	Mt. McKinley
645024	GROUSE 73	S.S.	38111	64.8	23N	49W	7	Kuskokwim/Mt. McKinley
645025	GROUSE 74	S.S.	38111	64.8	23N	49W	7	Kuskokwim/Mt. McKinley
645026	GROUSE 75	S.S.	38111	64.8	23N	49W	8	Kuskokwim
645027	GROUSE 76	S.S.	38110	64.8	23N	49W	8	Kuskokwim
645028	GROUSE 77	S.S.	38110	64.8	23N	49W	9	Kuskokwim
645029	GROUSE 78	S.S.	38110	64.8	23N	49W	9	Kuskokwim
645030	GROUSE 79	S.S.	38111	64.8	23N	50W	11	Mt. McKinley
645031	GROUSE 80	S.S.	38111	64.8	23N	50W	11	Mt. McKinley
645032	GROUSE 81	S.S.	38111	64.8	23N	50W	12	Mt. McKinley
645033	GROUSE 82	S.S.	38111	64.8	23N	50W	12	Mt. McKinley
645034	GROUSE 83	S.S.	38111	64.8	23N	49W	7	Mt. McKinley
645035	GROUSE 84	S.S.	38111	64.8	23N	49W	7	Kuskokwim/Mt. McKinley
645036	GROUSE 85	S.S.	38111	64.8	23N	49W	8	Kuskokwim/Mt. McKinley
645037	GROUSE 86	S.S.	38110	64.8	23N	49W	8	Kuskokwim
645038	GROUSE 87	S.S.	38110	64.8	23N	49W	9	Kuskokwim
645039	GROUSE 88	S.S.	38110	64.8	23N	49W	9	Kuskokwim
645040	GROUSE 89	S.S.	38111	64.8	23N	50W	2	Mt. McKinley
645041	GROUSE 90	S.S.	38111	64.8	23N	50W	2	Mt. McKinley
645042	GROUSE 91	S.S.	38111	64.8	23N	50W	1	Mt. McKinley
645043	GROUSE 92	S.S.	38111	64.8	23N	50W	1	Mt. McKinley
645044	GROUSE 93	S.S.	38111	64.8	23N	49W	6	Mt. McKinley
645045	GROUSE 94	S.S.	38111	64.8	23N	49W	6	Mt. McKinley
645046	GROUSE 95	S.S.	38111	64.8	23N	49W	5	Kuskokwim/Mt. McKinley
645047	GROUSE 96	S.S.	38110	64.8	23N	49W	5	Kuskokwim
645048	GROUSE 97	S.S.	38110	64.8	23N	49W	4	Kuskokwim
645049	GROUSE 98	S.S.	38110	64.8	23N	49W	4	Kuskokwim
645050	GROUSE 99	S.S.	38111	64.8	23N	50W	2	Mt. McKinley
645051	GROUSE 100	S.S.	38111	64.8	23N	50W	2	Mt. McKinley
645052	GROUSE 101	S.S.	38111	64.8	23N	50W	1	Mt. McKinley
645053	GROUSE 102	S.S.	38111	64.8	23N	50W	1	Mt. McKinley
645054	GROUSE 103	S.S.	38111	64.8	23N	49W	6	Mt. McKinley
645055	GROUSE 104	S.S.	38111	64.8	23N	49W	6	Mt. McKinley
645056	GROUSE 105	S.S.	38111	64.8	23N	49W	5	Kuskokwim/Mt. McKinley
645057	GROUSE 106	S.S.	38110	64.8	23N	49W	5	Kuskokwim
645058	GROUSE 107	S.S.	38110	64.8	23N	49W	4	Kuskokwim
645059	GROUSE 108	S.S.	38110	64.8	23N	49W	4	Kuskokwim
Total – 158 claims				**7,808.4**				

Note: T = township, R = range, Sec = section, SS = state selected, TA = tentative approval

4.2.4 Additional Lands

Lands classed as “Additional Lands” are located in the vicinity of the Calista and TKC agreement lands (the leased lands), but are not covered by either agreement. Such lands are within the Seward Meridian, and within the Kuskokwim or Mt. McKinley Recording Districts, Fourth Judicial District, State of Alaska:

- Township 19 North, Range 50 West – Sections 4-9, 16-21, 28-33
- Township 19 North, Range 51 West – all Sections
- Township 20 North, Range 49 West – Sections 19-22, 27-34
- Township 20 North, Range 50 West – all Sections
- Township 20 North, Range 51 West – all Sections
- Township 21 North, Range 48 West – Sections 31 and 32
- Township 21 North, Range 49 West – Section 6, 19-36
- Township 21 North, Range 50 West – all Sections
- Township 21 North, Range 51 West – Sections 1 and 12
- Township 22 North, Range 48 West – Sections 1-4, 7-36 (Sections 5 and 6 are among the leased lands)
- Township 22 North, Range 49 West – Sections 4-9, 12-36 (Sections 1, 2, 3, 10, and 11 are among the leased lands)
- Township 22 North, Range 50 West – all Sections
- Township 22 North, Range 51 West – Section 36
- Township 23 North, Range 48 West – Sections 1-4, 9-15, 22-27, 34-36 (Sections 5-8, 16-21, 28-33 are among the leased lands)
- Township 23 North, Range 49 West – Sections 2-9, 16-20, 29-32 (Sections 1, 10-15, 21-28, 33-36 are among the leased lands)
- Township 23 North, Range 50 West – all Sections
- Township 24 North, Range 48 West – Sections 19-36
- Township 24 North, Range 49 West – Sections 25-36

Title to the Additional Lands was verified to the DCLLC, and is held by various parties, including the State of Alaska, Federal Government of the United States, Calista, and TKC (Table 4-3). Some title is “Tentative Approval” (TA), a term used in the federal law to denote Bureau of Land Management (BLM) approval for issuing patent before completion of certain administrative actions such as survey. The federal government treats TA as tantamount to the conveyance of unsurveyed land to the State. All right, title, and interest of the United States in tentatively approved or “TA’d,” lands is deemed to have vested in the State of Alaska as of the date of TA. Upon BLM’s approval of a land survey, BLM patents the TA’d lands to the State.

Table 4-3: Donlin Creek Additional Lands

Land Description	Landowner/ Grantee	Conveyance Document/Note	Date Recorded	Kuskokwim Recording District Document No.
T. 19N, R. 50W, Sections 5-8, 18, 19, 30, and 31	State of Alaska	Tentative Approval 2008-0058	01/15/2008	2008-000012-0
T. 19N, R. 50W, Sections 4, 9, 16, 17, 20, 21, 28, 29, 32, and 33	United States	General Grant State Selection (GS 2893)	N/A	N/A
T. 19N, R. 51W, Sections 4, 9, 16, 20, 21, 32, and 33	State of Alaska	Tentative Approval 2008-0058	01/15/2008	2008-000012-0
T. 19N, R. 51W, Sections 13, 14, 15, 22-30, 34-36 (surface estate)	TKC	Patent No. 50-2005-0281	12/27/2005	2005-000285-0
T. 19N, R. 51W, Sections 13, 14, 15, 22-30, 34-36 (subsurface estate)	Calista Corporation	Patent No. 50-2005-0282	08/02/2005	2005-000131-0
T. 19N, R. 51W, Sections 1-2, 10-12	United States	TKC 12(b) Reallocated Selection; ANILCA Top-Filed State Selection	N/A	N/A
T. 19N, R. 51W, Section 3	United States	TKC 12(b) Reallocated Selection; ANILCA Top-Filed State Selection	N/A	N/A
T. 19N, R. 51W, Sections 5-8, 17-19, and 31	United States	General Grant State Selection (GS 2893)	N/A	N/A
T. 20N, R. 49W, Sections 22 (lot 4), 27 (lots 4-7), 28 (lots 4 & 5), 29 (lots 4 & 5), 30 (lot 3), 31 (lot 3), 32 (lots 3 & 4), and 33 (lot 1) (surface estate)	TKC	Patent No. 50-2007-0683	12/05/2007	2007-001295-0
T. 20N, R. 49W, Sections 22 (lot 4), 27 (lots 4-7), 28 (lots 4 & 5), 29 (lots 4 & 5), 30 (lot 3), 31 (lot 3), 32 (lots 3 & 4), and 33 (lot 1) (subsurface estate)	Calista Corporation	Patent No. 50-2007-0684	09/17/2007	2007-001150-0
T. 20N, R. 49W, Section 34	United States	Community Grant State Selection (CG 160)	N/A	N/A
T. 20N, R. 49W, Sections 19-21	United States	N/A	N/A	N/A
T. 20N, R. 50W, Sections 4-9, 15-32	State of Alaska	Tentative Approval 2008-0129	06/06/2008	2008-000661-0
T. 20N, R. 50W, Sections 1-3, 10-14	State of Alaska	Tentative Approval 2009-0004	10/16/2008	2008-000932-0
T. 20N, R. 50W, Sections 33-36	United States	General Grant State Selection (GS 5378)	N/A	N/A
T. 20N, R. 51W, Sections 25-29, 32-36	State of Alaska	Tentative Approval	11/05/2007	2007-001196-0
T. 20N, R. 51W, Sections 1-24, 30-31	United States	General Grant State Selection (GS 5379)	N/A	N/A
T. 21N, R. 48W, Sections 31 (lots 1 & 2) and 32 (lots 4-7) (surface estate)	TKC	Patent No. 50-2007-0683	12/05/2007	2007-001295-0

Land Description	Landowner/ Grantee	Conveyance Document/Note	Date Recorded	Kuskokwim Recording District Document No.
T. 21N, R. 48W, Sections 31 (lots 1 & 2) and 32 (lots 4-7) (subsurface estate)	Calista Corporation	Patent No. 50-2007-0684	09/17/2007	2007-001150-0
T. 21N, R. 49W, Section 6 (surface estate)	TKC	Patent No. 50-94-0009	Not recorded	N/A
T. 21N, R. 49W, Section 6 (subsurface estate)	Calista Corporation	Patent No. 50-94-0010	01/20/1994	1994-000018-0
T. 21N, R. 49W, Section 24 (lots 1, 2, and 3) (surface estate)	TKC	Patent No. 50-2007-0683	12/05/2007	2007-001295-0
T. 21N, R. 49W, Section 24 (lots 1, 2, and 3) (subsurface estate)	Calista Corporation	Patent No. 50-2007-0684	09/17/2007	2007-001150-0
T. 21N, R. 49W, Sections 25, 34-36	State of Alaska	Tentative Approval 2009-0004	10/16/2008	2008-000932-0
T. 21N, R. 49W, Sections 19-23, 26-33	United States	N/A	N/A	N/A
T. 21N, R. 50W, Sections 1 and 12 (surface estate)	TKC	Patent No. 50-94-0009	Not recorded	N/A
T. 21N, R. 50W, Sections 1 and 12 (subsurface estate)	Calista Corporation	Patent No. 50-94-0010	01/20/1994	1994-000018-0
T. 21N, R. 50W, Sections 2-11 (surface estate)	TKC	Patent No. 50-2007-0213	04/26/2007	2007-000238-0
T. 21N, R. 50W, Sections 2-11 (subsurface estate)	Calista Corporation	Patent No. 50-2007-0214	12/09/2008	2008-001007-0
T. 21N, R. 50W, Sections 13-16, 21-28, 34-36	State of Alaska	Tentative Approval 2008-0061	01/16/2008	2008-000018-0
T. 21N, R. 50W, Sections 17 and 33	State of Alaska	Tentative Approval 2009-0007	11/24/2008	2008-000997-0
T. 21N, R. 50W, Sections 18-20, 29-32	United States	General Grant State Selection (GS 6423)	N/A	N/A
T. 21N, R. 51W, Sections 1 and 12	United States	General Grant State Selection (GS 6424)	N/A	N/A
T. 22N, R. 48W, Sections 4, 7-9, 17-20 (surface & subsurface estates)	Calista Corporation	Patent No. 50-2007-0148	02/05/2007	2007-000024-0
T. 22N, R. 48W, Section 31 (surface estate)	TKC	Patent No. 50-94-0009	Not recorded	N/A
T. 22N, R. 48W, Section 31 (subsurface estate)	Calista Corporation	Patent No. 50-94-0010	01/20/1994	1994-000018-0
T. 22N, R. 48W, Sections 1-3, 10-16, 21-30, and 32-36	United States	N/A	N/A	N/A
T. 22N, R. 49W, Sections 12, 13, and 24 (surface & subsurface estates)	Calista Corporation	Patent No. 50-2007-0148	02/05/2007	2007-000024-0
T. 22N, R. 49W, Sections 14-16, 22, 23, 26-36 (subsurface estate)	Calista Corporation	Patent No. 50-94-0010	01/20/1994	1994-000018-0

Land Description	Landowner/ Grantee	Conveyance Document/Note	Date Recorded	Kuskokwim Recording District Document No.
T. 22N, R. 49W, Sections 14-16, 22, 23, 26-36 (surface estate)	TKC	Patent No. 50-94-0009	Not recorded	N/A
T. 22N, R. 49W, Sections 4-9, 17-21 (surface estate)	TKC	Patent No. 50-2007-0213	04/26/2007	2007-000238-0
T. 22N, R. 49W, Sections 4-9, 17-21 (subsurface estate)	Calista Corporation	Patent No. 50-2007-0214	12/09/2008	2008-001007-0
T. 22N, R. 49W, Section 25	United States	N/A	N/A	N/A
T. 22N, R. 50W, Sections 1, 12, 13, 34, and 35 (surface estate)	TKC	Patent No. 50-2007-0213	04/26/2007	2007-000238-0
T. 22N, R. 50W, Sections 1, 12, 13, 34, and 35 (subsurface estate)	Calista Corporation	Patent No. 50-2007-0214	12/09/2008	2008-001007-0
T. 22N, R. 50W, Sections 2-5, 11, 14, 23-26, 32, and 33	State of Alaska	Tentative Approval 2008-0087	02/25/2008	2008-000032-0 (Mt. McKinley Recording District)
T. 22N, R. 50W, Section 36 (surface estate)	TKC	Patent No. 50-94-0009	Not recorded	N/A
T. 22N, R. 50W, Section 36 (subsurface estate)	Calista Corporation	Patent No. 50-94-0010	01/20/1994	1994-000018-0
T. 22N, R. 50W, Sections 6-10, 15-22, 27-31	United States	General Grant State Selection (GS 6432)	N/A	N/A
T. 22N, R. 51W, Section 36	United States	General Grant State Selection (GS 5942)	N/A	N/A
T. 23N, R. 48W, Sections 3, 4, 9, 10, 15, 22, 27, and 34 (surface & subsurface estates)	Calista Corporation	Patent No. 50-2007-0148	02/05/2007	2007-000024-0
T. 23N, R. 48W, Sections 1, 2, 11, and 12	State of Alaska	Patent No. 50-2008-0112	01/15/2008	2008-000010-0
T. 23N, R. 48W, Sections 13, 14, 23-26, and 35-36	United States	N/A	N/A	N/A
T. 23N, R. 49W, Sections 2-9, 16-20, 29-32	United States	General Grant State Selection (GS 6436)	N/A	N/A
T. 23N, R. 50W, Sections 19-22, 27-34 (surface & subsurface estates)	Calista Corporation	Patent No. 50-2007-0148	02/05/2007	2007-000024-0
T. 23N, R. 50W, Sections 1-18, 23-26, 35-36	United States	General Grant State Selection (GS 6437)	N/A	N/A
T. 24N, R. 48W, Sections 19-36	State of Alaska	Tentative Approval	07/25/2007	2007-000656-0 (Mt. McKinley)
T. 24N, R. 29W, Sections 25-36	State of Alaska	Tentative Approval	07/19/200708/23/2007	2007-000655-0(Mt. McKinley)2007-000792-0 (Kuskokwim)

The Additional Lands area also encompasses a number of encumbrances that are granted to third parties. These are primarily trail, road, right-of-way and airport easements. One acre site easements are also present.

A public lands order (BLM Public Land Order (PLO) 5184) withdrew certain federally-owned lands located within the Project area from location and entry under federal mining laws, and from leasing under the Mineral Leasing Act of 25 February 1920 and its amendments.

PLO 5184 specifies that the withdrawn lands remain subject to federal administration until they are conveyed to the State or to Village/Regional Corporations pursuant to Section 14 of ANCSA. BLM will reject applications for leases under the Mineral Leasing Act until PLO 5184 is modified or the lands are appropriately classified to permit mineral leasing. However, the order does not impair the Interior Secretary's authority to make contracts and to grant permits, rights-of-way, easements, or other leases. Sections include:

- T. 19 N, R. 50 W – Sections 4, 9, 16, 17, 20, 21, 28, 29, 32, and 33
- T. 19 N, R. 51 W – Sections 1-2, 10-12
- T. 19 N, R. 51 W – Section 3
- T. 19 N, R. 51 W – Sections 5-8, 17-19, and 31
- T. 20 N, R. 49 W – Section 34
- T. 20 N, R. 49 W – Sections 19-21
- T. 20 N, R. 50 W – Sections 33-36
- T. 20 N, R. 51 W – Sections 1-5
- T. 21 N, R. 49 W – Sections 19-23, 26-33
- T. 21 N, R. 50 W – Sections 18-20, 29-32
- T. 22 N, R. 48 W – Sections 1-3, 10-16, 21-30, and 32-36
- T. 22 N, R. 49 W – Section 25
- T. 22 N, R. 50 W – Sections 6-10, 15-22, 27-31
- T. 23 N, R. 48 W – Sections 13, 14, 23-26, 35-36
- T. 23 N, R. 49 W – Sections 2-9, 16-20, 29-32
- T. 23 N, R. 50 W – Sections 1-18, 23-26, 35-36

Effective 19 July 1990, PLO 6787 modified prior PLO 5180 and PLO 5184 to open certain federally-owned lands located within the Project area to location and entry under all United States mining laws. The following Additional Lands were opened to mineral location and entry pursuant to PLO 6787:

- T. 20 N, R. 51 W – Sections 6–24, 30–31
- T. 21 N, R. 51 W – Sections 1 and 12

These lands remained closed to appropriation under federal mineral leasing laws.

PLO 6787 opened the following Additional Lands to the mineral leasing laws as well as location and entry under all United States mining laws:

- T. 22 N, R. 51 W – Section 36

The currently identified Mineral Resource and the bulk of the primary infrastructure (mill, tailings and waste rock facilities) are located on leased lands that were conveyed. Lands required for the proposed BTC port site, road, wind power facilities, Crooked Creek offloading, lay-down area, and all-weather road, tailings storage facility in Anaconda Creek, and airstrip are located on a combination of Native conveyed, Native selected, and State of Alaska selected lands. Selected lands were identified by the State of Alaska for possible conveyance from the Federal Government, but are still administered by the Federal Government until such time as the land may be conveyed. A right-of-way would be required from the State of Alaska for the proposed road alignment where it crosses State lands.

Negotiations regarding the additional Native lands are ongoing with both TKC and Calista. Private Native allotments are present throughout the region. The current Project description does not require access to any of these private lands.

4.3 Agreements and Royalties

4.3.1 Calista

Barrick (formerly Placer Dome) acquired a 20-year lease from Calista effective 1 May 1995. The lease agreement contains a provision that extends the lease period beyond 20 years as long as mining or processing operations continue in good faith or good faith efforts are being made to place a mine on the property into production.

The terms for the Calista Exploration and Lode Mining Lease include the following:

- Annual advance minimum royalty of $200,000, increasing to $500,000 upon delivery by the DCLLC of a feasibility study and satisfaction of certain other conditions as set forth in the Exploration and Lode Mining Lease
- Net smelter return of 1.5% (or $500,000, whichever is greater) for the earlier of the first five years following commencement of production or until payback, increasing thereafter to 4.5% (or $500,000 annually, whichever is the greater)
- Right of Calista to acquire an equity interest of up to 15%, all or part of which may be purchased with “in kind contributions”; “in kind contribution” is defined as any

public funding or other funding sources Calista secures to deliver equipment, professional services, or other goods, services, and infrastructure to the Project. They will receive credit only if these good and services are needed for the Project.

- Calista shareholder hire preference and Calista 5% bidding preference on competitive contracts for all work on or for the Project.

4.3.2 TKC

The terms of the TKC Surface Use Agreement include the following:

- Annual Aggregate Surface Use Fee of $50,000
- Once exclusive-use lands are identified, payment of an annual exclusive-use fee of 10% of the fair market value of the property, or
- At TKC's request, purchase the property.

4.3.3 Barrick

On 13 November 2002, NovaGold Resources Alaska, Inc., a wholly-owned subsidiary of NovaGold Resources Inc., earned a 70% interest in the Project by expending US$10 million on exploration and development of the Project. Once the financial commitment was fulfilled, Barrick (formerly Placer Dome) had 90 days to decide on one of three options:

- To remain at 30% interest and participate as a minority partner
- To convert to a 5% net profits interest (NPI)
- To exercise a back-in right to re-acquire a majority interest in the Project (70%) by expending three times the amount expended by NovaGold at the time the back-in is exercised, completing a feasibility study, and making a decision to construct a mine at a production rate of not less than 600,000 ounces of gold per year within a five-year period from the exercise back-in.

On 11 February 2003, Barrick (formerly Placer Dome) exercised its back-in right and assumed management of the continued development of the Project.

On 1 December 2007, NovaGold entered into a limited liability company agreement with Barrick that provided for the conversion of the Project into a new limited liability company, the DCLLC, which is jointly owned by NovaGold and Barrick on a 50/50 basis. As part of the DCLLC, NovaGold agreed to reimburse Barrick over time for approximately US$63.5 million, representing 50% of Barrick's approximately US$127 million expenditures at the Project from 1 April 2006 to 30 November 2007.

NovaGold's reimbursement will be made following the effective date of the agreement, by NovaGold paying the next approximately US$12.7 million of Barrick's share of Project development costs, and the remaining approximately US$50.8 million will be paid out of future mine production cash flow. These amounts were agreed to subject to adjustment upon audit of the US$127 million expenditure. After NovaGold's initial contribution, all funding will be shared by both parties on a 50/50 basis.

Upon delivery of a feasibility study by the DCLLC and satisfaction of certain other conditions, Calista retains a 90 day back-in right to participate in the Project at a level of 5% to 15% by committing to contribute its share of capital. Calista's share would be divided pro rata from Barrick and NovaGold.

4.3.4 Calista Royalty

An advance minimum royalty (AMR) on the Donlin Creek property of US$200,000 is payable by the DCLLC to Calista annually until a feasibility study is delivered by the DCLLC and additional conditions set forth in the Exploration and Lode Mining Lease are met, after which the AMR will increase to US$500,000 per year. Upon commencement of production, a net smelter return royalty on production equal to the greater of 1.5% of the revenues from valuable minerals production and US$500,000 is payable to Calista, until the earlier of the expiry of five years or the payback of all preproduction expenses incurred by the DCLLC. Thereafter, the annual net smelter return royalty on production will be increased to the greater of 4.5% of the revenues from valuable minerals production and US$500,000.

4.3.5 Lyman Resources Royalty

Lyman Resources has existing placer mining leases covering approximately four square miles within the Donlin lease area. The Lyman family also has title to approximately 5.3 ha of surface estate within the Snow Gulch area. This lease area lies immediately to the north of the current open pit shell outline but should not result in any significant conflicts with the pit shell or envisioned infrastructure layout. The Calista Exploration and Lode Mining Lease grants priority to extraction of the lode resource in the event of a conflict of use between lode and placer mining operations, provided that a two-year notice period is provided to Lyman Resources. Negotiations regarding the future of the Lyman holdings are ongoing.

4.4 Permits

4.4.1 Current Permits

The DCLLC has maintained all of the necessary permits for exploration and camp facilities. These permits are active at the Alaska Department of Natural Resources (hard rock exploration, temporary water use), the Corp of Engineers (individual 404 and nationwide 26), Alaska State Department of Conservation (wastewater, drinking water, food handling), the Alaska Department of Fish and Game (title 16 – fish), the Environmental Protection Agency (NPDES) and the Federal Aviation Administration (airport).

4.4.2 Future Permitting

Donlin Creek will require a considerable number of permits and authorizations from both federal and state agencies. Much of the groundwork to support a successful permitting effort is undertaken prior to the submission of permit applications, so that issues can be identified and resolved, supporting baseline data can be acquired, and regulators and stakeholders can become familiar with the proposed Project.

To support successful application for the more than 60 permits, this Project will likely require extensive baseline environmental information, supporting scientific analysis, and detailed engineering design. The DCLLC and predecessors have invested significant money, resources, and time acquiring this information over the last five years, and in some cases over the last 12 years. Designing in line with baseline data in advance of filing permit applications has resulted in a project that affords due consideration to all environmental concerns and is designed to mitigate potential impacts on the environment wherever practicable.

The comprehensive permitting process for Donlin Creek can be divided into three categories, all of which are important to the successful establishment of a future mining operation:

- Exploration-stage permitting – required to obtain approval for exploration drilling, environmental baseline studies, and feasibility engineering studies.
- Pre-application phase – conducted in parallel with feasibility engineering studies. This stage includes the collection of environmental baseline data and interaction with stakeholders and regulators to facilitate the development of a project that can be successfully permitted.

- The National Environmental Policy Act (NEPA) process and formal permit applications – formal agency review and analysis of the Project, resulting in the issuance or denial of permits.

Permit review timelines are controlled by the requirements of the federal NEPA review and State requirements for meaningful public and agency participation to determine if the Project is in the State's best interest.

Table 4-4 provides an overview of the permits that may be required for the Project.

Upon completion of the NEPA review, a positive Record of Decision (ROD), and final issuance of permits and authorizations, the Environmental Management System (EMS), consisting of a number of management and maintenance plans for the Project, will be fully implemented.

Each Federal and State permit will have compliance stipulations that require scrutiny and negotiation that can typically be resolved within 60 days of the ROD. Project delays could occur as a result of public opposition, limitations in regulatory staff resources during regulator review, or Project changes made by the owner.

4.5 Environmental

Baseline environmental studies commenced in 1996, comprising water quality studies, meteorology, aquatic studies in the main drainages, wetlands delineation in the areas of the Mineral Resource estimates and some waste rock characterization.

During 2003, the baseline program was expanded, and included ambient air monitoring, terrestrial wildlife and avian surveys, groundwater monitoring, detailed aquatic studies, cultural site surveys, detailed waste rock characterization and additional wetlands delineation.

Feedback from regulatory and public consultation, resulted in additional studies to review the impact of mercury use and the impact of barge traffic on subsistence fishing and river erosion.

The three primary reasons for collecting baseline data are to inform the design process, to determine environmental controls to mitigate the impacts of exploration activities and future development on the area, and to characterize the Project environment in anticipation of compliance with NEPA and permitting. The environmental baseline data provide a reference point for environmental assessments and facilitate early detection of potential changes that may occur during mine development and operation.

Table 4-4: Federal Agency Permit and Authorizations

Agency	Authorization
Federal	
Bureau of Land Management (BLM)	• Surface Estate Lease (facilities managed lands) • Land Use Permit (activities on BLM managed lands) • Access Right-of-Way (BLM managed lands)
Environmental Protection Agency (EPA)	• CWA Section 402 NPDES Permit (discharges to waters of the U.S.) • Spill Prevention Containment and Contingency (SPCC) Plan • Storm Water Pollution Prevention Plan – Construction and Operations
U.S. Army Corps of Engineers (USACE)	• CWA Section 404 Permit (wetlands dredge and fill) • River and Harbors Act (RHA) Section 10 (structures in navigable waters) • RHA Section 9 (dams and dykes in navigable waters – interstate commerce)
U.S. Coast Guard	• RHA Section 9 Construction Permit (bridge across navigable waters) • Marine Protection, Research, and Sanctuaries Act compliance (ocean dumping requires a permit)
Bureau of Alcohol, Tobacco, and Firearms	• License to Transport Explosives • Permit and License for Use of Explosives
Federal Aviation Administration	• Notice of Landing Area Proposal (existing airstrip) • Notice of Controlled Firing Area for Blasting
U.S. Department of Transportation	• Hazardous Materials Registration
National Marine Fisheries Service	• Marine Mammal Protection Act authorization (IHA/LOA)
U.S. Fish and Wildlife Service	• Section 7 of the Endangered Species Act, Consultation requiring a Biological Assessment or Biological Opinion
State	
Office of Project Management and Permitting	• Alaska Coastal Management Program Consistency Applicability • Determination
Division of Mining, Land, and Water	• Plan of Operations • Reclamation Plan Approval • Mining License • Land Use Permits and Leases • Right-of-Ways, Easements, Material Sales, etc. • Certificate of Approval to Construct a Dam • Certificate of Approval to Operate a Dam • Temporary Water Use Permit • Water Rights Permit/Certificate to Appropriate Water • Tidelands Permit
Office of History and Archaeology/State Historic Preservation Office	• Section 106 Historical and Cultural Resources Protection Act clearance
Office of Habitat Management and Permitting	• Fish Habitat Permit • Culvert/Bridge Installation Permit
Division of Water	• Section 401 Water Quality Certification (CWA 404 permit) • Section 401 Water Quality Certification (CWA 402 permit) • Wastewater Disposal Permits • Non-Domestic Wastewater Disposal Permit • Storm Water Discharge Pollution Prevention Plan • Domestic Wastewater Disposal Permit • Approval to Construct and Operate a Public Water Supply System
Division of Environmental Health	• Solid Waste Disposal Permits • Food Sanitation Permit
Division of Air Quality	• Air Quality Construction Permit (first 12 months) • Air Quality PSD Title V Operating Permit (after 12 months) • Air Quality permit to Open Burn

The EMS and permit review process will determine the precise number of management plans required to address all aspects of the Project to ensure compliance with environmental design and permit criteria. Each plan will describe the appropriate environmental engineering standard (e.g., secondary containment for petroleum products, process solutions, and reagents) and the applicable operations requirements, maintenance protocols, and response actions.

Permits issued by federal agencies constitute "federal actions." Any major federal action requires review under NEPA. All elements of a project and their cumulative effects are considered and evaluated in a NEPA review.

In addition, alternatives to the proposed action are evaluated and potential mitigation measures are identified. For Donlin Creek, NEPA will require the preparation of an environmental impact statement (EIS). Typically, under NEPA the federal agency with the predominant permit is designated the lead agency. The lead agency for this Project has not yet been selected.

Over the nearly 13 years since exploration and environmental baseline data collection began, considerable effort was spent developing support for the Project by fostering local relationships, developing a strong local workforce, educating stakeholders about the Project and mining in general, and providing stakeholders with regular Project updates and site visits. This has enabled the DCLLC to better understand and address the perspectives and concerns of the Project stakeholders and resulted in broad public support for the Project, especially in the upriver region surrounding the immediate Project area. This support has taken the form of resolutions from tribal councils and organizations, participation by individuals and tribal groups in various Project-related forums, and permissions granted to conduct environmental baseline studies on tribal lands.

As a result of comprehensive interaction with regulators and routine informal interaction with individual agencies during exploration permitting, the Project is now well positioned to trigger the NEPA review and move forward with permit applications for construction, operations, and closure. Regulators who will be administering this review now have a solid understanding of the Project and confidence in the manner in which the supporting baseline data were collected and evaluated.

Permit review timelines are controlled by the requirements of the federal NEPA review and state requirements for meaningful public and agency participation to determine if the Project is in the state's best interest. Having engaged in comprehensive dialogue with stakeholders and regulatory agencies, and by moving forward with a well-defined Project description, the DCLLC has positioned itself well for an optimal permit review timeline. The responsiveness of DCLLC and its provision of sufficient resources will help to facilitate the process as it advances. However, the process will be driven by the regulatory agencies and is subject to delays resulting from factors such as regulatory and statutory changes, inadequate agency staffing, competing projects, and delays in public notice publications in local newspapers and the Federal Register.

Upon completion of the NEPA review, a positive ROD, and final issuance of permits and authorizations, the EMS, consisting of a number of management and maintenance plans for the Project, will be fully implemented. The comprehensive permit review process will determine the precise number of management plans required to address all aspects of the Project to ensure compliance with environmental design and permit criteria. Each plan will describe the appropriate environmental engineering standard (e.g., secondary containment for petroleum products, process solutions, and reagents) and the applicable operations requirements, maintenance protocols, and response actions.

Present environmental liabilities are believed to be limited to the exploration camp.

4.6 Socio-Economics

Since exploration and environmental baseline data collection began, considerable effort was spent developing support for the Project by fostering local relationships, developing a strong local workforce, educating stakeholders about the Project and mining in general and providing stakeholders with regular Project updates and site visits. This activity has enabled the DCLLC to better understand and address the perspectives and concerns of the Project stakeholders and has resulted in broad public support for the Project, especially in the upriver region surrounding the immediate Project area. This support has taken the form of resolutions from tribal councils and organizations, participation by individuals and tribal groups in various Project-related forums and permissions granted to conduct environmental baseline studies on tribal lands.

4.7 Comments on Section 4

Mining tenure held by the DCLLC in the area for which Mineral Resources are estimated is valid.

The DCLLC holds a significant portion of the surface rights that will be required to support mining operations in the proposed mining area. Negotiations will be required for surface rights for additional lands including road rights-of-way, the proposed wind farm, airstrip, Crooked Creek, Anaconda Creek and BTC facilities.

Agreements between the members of the DCLLC are typical of such agreements. Agreements exist with Calista and TKC for land and surface rights, respectively.

Current permits have allowed exploration and associated feasibility study supporting testwork to be conducted under appropriate State and Federal laws. Additional permits are required for Project development.

Baseline environmental studies were completed. Present environmental liabilities are believed to be limited to the exploration camp. Environmental permits for Project development have to be secured. The EMS and permit review process will determine the precise number of management plans required to address all aspects of the Project to ensure compliance with environmental design and permit criteria.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

The Donlin Creek property is located in southwest Alaska, approximately 20 km north of the village of Crooked Creek on the Kuskokwim River (see Figure 4-1). The Kuskokwim River is a regional transportation route and is serviced by commercial barge lines.

An airstrip 1,500 m long adjacent the exploration camp is capable of handling aircraft as large as C-130 Hercules (19,050 kg), allowing efficient shipment of personnel, some heavy equipment, and supplies. The Project can be serviced directly by charter air facilities out of both Anchorage and Aniak.

5.2 Climate

The area has a relatively dry interior continental climate with typically less than 50 mm total annual precipitation. Summer temperatures are relatively warm and may reach nearly 30°C. Minimum temperatures may fall to well below -40°C during the cold winter months. Work is possible on the Project year-round.

5.3 Local Resources and Infrastructure

5.3.1 Infrastructure

Current site infrastructure comprises an all-season, soft-sided camp with facilities to house up to 150 people consisting of kitchen, living quarters, equipment shop, drill shack and other buildings required for support of year-around exploration activities.

There is sufficient area within the Project to host an open pit mining operation, including any proposed open pit, waste dumps, tailings, and process facilities. The DCLLC has secured the majority of the surface rights for the areas that may host these facilities.

Crooked Creek has about 140 residents; Aniak has a population of about 570. The workforce for the Project would be sourced from the local area, and from Alaskan regional centres.

5.3.2 Transport

A 24 km long winter road, designated as an Alaska State Highway route and transportation corridor, accesses the property from the barge site at the village of Crooked Creek. The river serves as a regional transport hub, serviced by commercial barge lines. During winter, the frozen river serves as an ice road.

The Project is directly serviced by commercial air services out of both Anchorage, 450 km to the east, and Aniak, 80 km to the west. Flights utilise the existing airstrip on the Property.

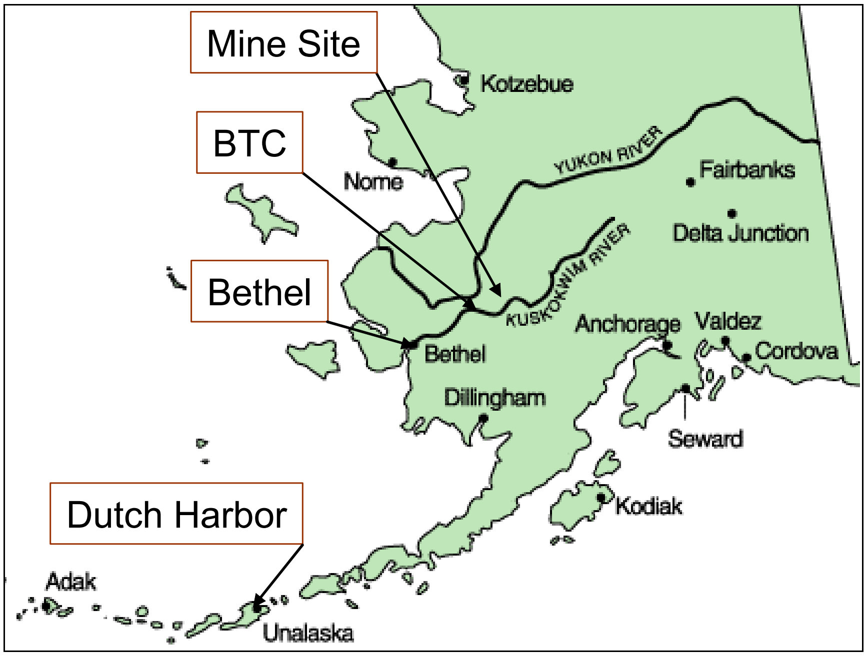
The Port of Bethel is the northernmost medium-draft port in the United States. Ocean barges from Seattle and Anchorage deliver supplies to communities along the Alaskan coast. During the summer, barges bring fuel, construction materials and large consumer goods to the Bethel region. Shipments typically consist of 45% bulk fuel and 55% dry cargo. These barges also work their way up the Kuskokwim River to Bethel. From Bethel, smaller river barges bring fuel, supplies and construction materials to Kuskokwim River villages. Three-fourths of the area's communities use barge services for cargo supplies.

Proposed Shipping Services

The supply chain for mine consumables includes Dutch Harbor, on the Aleutian Islands (Figure 5-1). From Dutch Harbor, ocean barges will travel across the Bering Sea and up the Kuskokwim River to Bethel. From Bethel, river barges will travel to the BTC port site (Figure 5-1), which will be connected by fuel pipeline and road to the planned mine.

The transportation of cargo and supplies to the mine site will entail the construction of major receiving, storage, and transfer facilities at different locations in Alaska en route to the mine. General cargo consolidated in Seattle and Vancouver will be shipped on ocean barges to Bethel. A cargo terminal will be constructed in Bethel to receive the ocean barges. This will comprise an unloading terminal at Bethel, with an area of 6.5 ha, capable of simultaneously unloading ocean barges and with sufficient storage capacity to hold up to 2,750 containers. An estimated 85% of the annual cargo during re-supply for mine operations will be containerized. A diesel power generation facility at the port will provide electricity.

Figure 5-1: Location Map



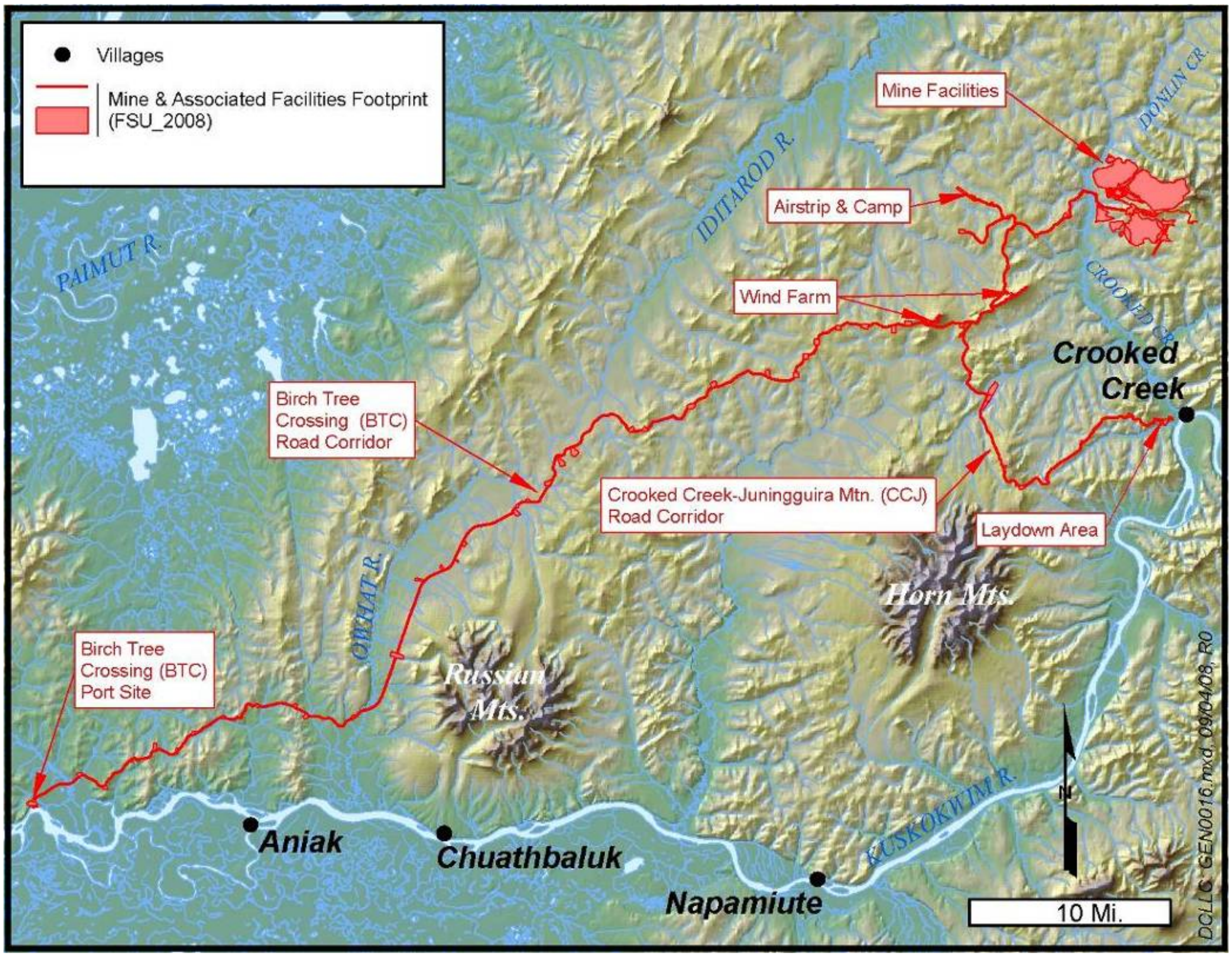
Note: Modified from www.state.ak.us. BTC = Birch Tree Crossing

All river barges will be designed, constructed, and operated in accordance with the requirements of applicable legislation and regulations. All fuel tugs will carry a supply of oil spill containment and recovery equipment on board, and the crews will be trained in the use of this equipment.

Barges discharging at Bethel will be surrounded by a spill containment boom as standard practice. To ensure a safe and secure marine system on the river, each tug will be equipped with modern communication and navigation equipment, including satellite- and radar-based marine chart displays and tracking transmitters.

The cargo will be unloaded and either placed into storage or reloaded onto river barges to be towed upriver to upriver to a port site at BTC (Figure 5-2). The BTC port site on the Kuskokwim River will receive river barges loaded with cargo and fuel from Bethel and be the link for subsequent overland transport. Facilities will include barge berths, a barge ramp, container-handling equipment, storage for containers, break-bulk cargo, and fuel, as well as barge-season camp facilities. Containers and other cargo will be stored at the port terminal and trucked to the mine throughout the year as required. Fuel will be off-loaded from barges and temporarily held in tanks before being pumped by pipeline to the main fuel storage facility at the mine site.

Figure 5-2: Location of Mine Footprint and Off-Site Facilities



Note: Figure courtesy DCLLC

Proposed Access Roads

The mine access road will traverse varied terrain from the mine site to the Kuskokwim River port site at BTC (see Figure 5-2). BTC is 19 km downstream and on the opposite river bank from the town of Aniak; there is no road connection between the two locations. The BTC mine access road to the mine site battery limits[3] will be 119 km long. The entire road will be new construction in an untracked region, with no passage through or near any settlements or communities, and no junctions with any existing road system.

[3] The mine battery limit is defined as the interface point between the off-site infrastructure and on-site infrastructure.

A 5 km long spur road, beginning at BTC route kilometre 8.7, will serve the Project airstrip and permanent accommodations complex. A second 6 km long spur, beginning at BTC route kilometre 18.8, will access the Getmuna Flats area proposed as a source of borrow material. This spur road will also continue on to the village of Crooked Creek and serve as the early construction route. The total length of the Crooked Creek access spur road, from the intersection with the BTC road to the village of Crooked Creek on the Kuskokwim River, is 33 km. Kilometre 16 to 22.5 of the BTC route passes through the northern end of the planned wind farm.

Proposed Airstrip

The planned airstrip will be approximately 14.5 km by road west of the mine site (see Figure 5-2). The airstrip design is based on U.S. Department of Transportation, Federal Aviation Administration standards. The specified aircraft are the DHC Dash-8 and the Hercules C-130. The design was governed by the needs of the Hercules C-130. A gravel runway is suitable for both types of aircraft. A single airstrip was considered sufficient to accommodate the predominant wind directions.

5.3.3 Power

The Project is currently isolated from power and other public infrastructure. The exploration camp has a capacity of 160 persons. Power is provided by diesel generators. Electric power for the Project site is planned to be generated from a diesel oil-fuelled combined-cycle gas turbine power plant and a standby/peaking diesel power plant.

A wind farm consisting of 14 wind turbine generators, each with a nominal peak output of 2.5 MW, will also be installed. Under average conditions, the wind farm will contribute approximately 7.5% of the yearly energy requirements of the Project.

Given their synergistic roles, the gas turbine and diesel power plants will be located adjacent to each other. To minimize electrical distribution costs and load losses, they will be near the two major process electrical loads: the oxygen plant and the grinding building. The wind farm will be installed on Juningguira Mountain, approximately 12 km southwest of the Donlin Creek mine site, and will be connected to the site with a 69 kV transmission line running to a substation located at the mine site.

5.3.4 Water

Water requirements for the planned process facilities depend on mill feed rates and vary annually. Water will primarily be sourced from contact dam/pit dewatering.

However, in years with average and below-average precipitation, the contact water pond and pit dewatering system will not be able to meet the year-round freshwater requirements for the plant. In this case, additional water will be obtained from the north and south freshwater reservoirs upstream of the tailings storage facility (TSF).

The source of water supply for the construction camp and, later, the plant site potable water systems is an array of eight deep wells south of Omega Gulch, near Crooked Creek (Figure 5-3). Potable water for the permanent accommodation complex will be supplied from another array of four wells approximately 2.4 km southwest of the camp.

5.3.5 Communications

Current site communication is via radio.

The planned communications system for the proposed Donlin Creek mine will consist of the following:

- A microwave tower at Anaconda Mountain to link the site to Bethel to establish a microwave/satellite connection to Anchorage, to the airport facilities, to the BTC port facilities, and to Crooked Creek
- Servers, including an e-mail server, Web server and VoIP server
- Support for voice, data, fax, Internet, and video capabilities
- Cellular system for mobile voice/data communications
- Two-way radio communications equipment.

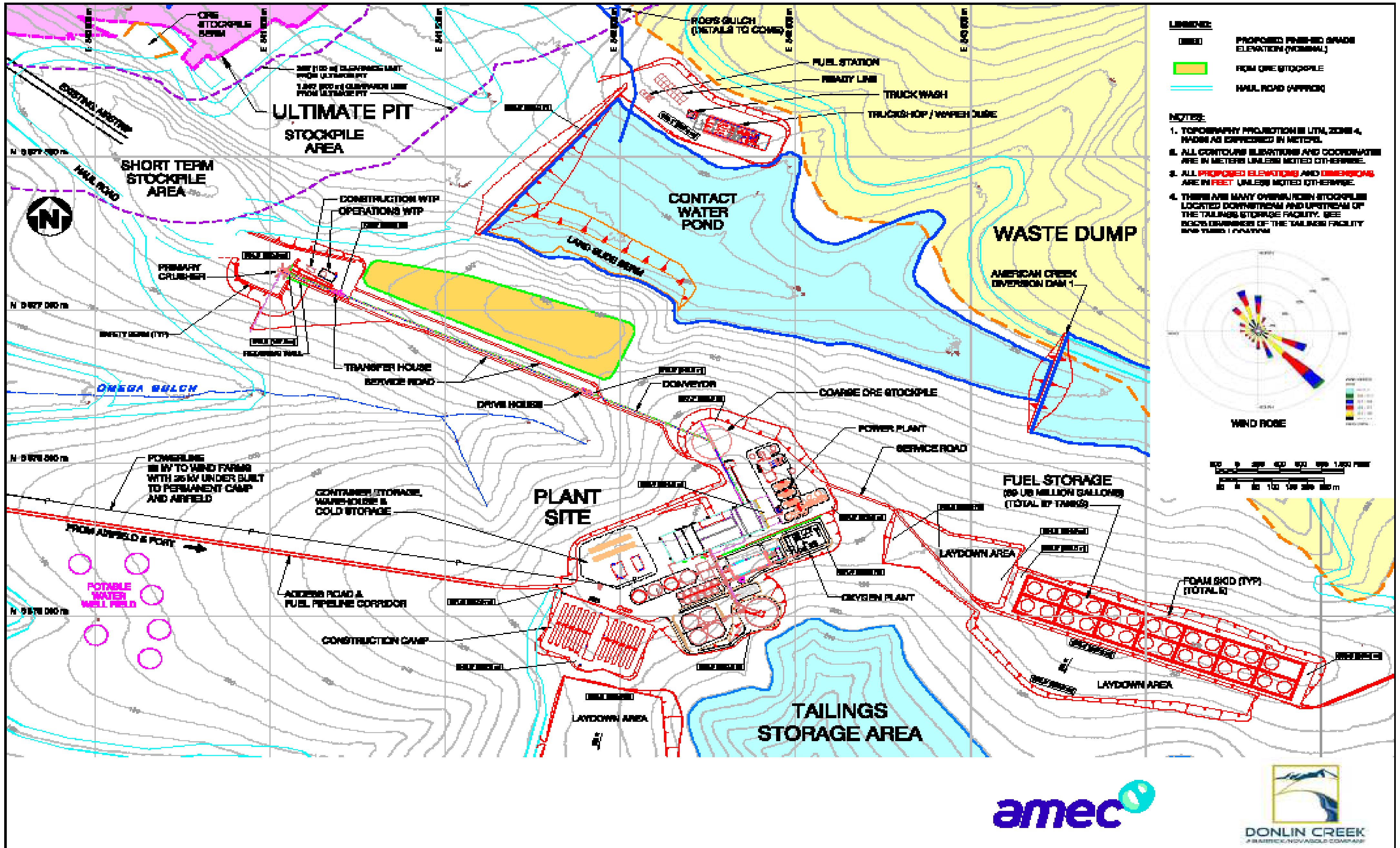
5.3.6 Contractor Camp/Work Force Housing

The planned permanent accommodation complex will be sited approximately 1 km east of the airstrip. The complex will initially house 404 people and be expanded in Year 1 and Year 6 to house 548 and up to a maximum of 608 people, respectively.

5.4 Physiography, Flora, and Fauna

The Project area is one of low topographic relief on the western flank of the Kuskokwim Mountains. Elevations range from 152 m to 640 m. Ridges are well rounded and easily accessible by all-terrain vehicle. Hillsides are forested with black spruce, tamarack, alder, birch and larch. Soft muskeg and discontinuous permafrost are common in poorly-drained areas at lower elevations.

Figure 5-3: Proposed Plant Site Layout



Note: Figure courtesy DCLLC. Grid north approximates geographic north; each grid square on plan is 1,000 ft (305 m).

5.5 Comments on Section 5

Current access methods support exploration-level work programs. During planned mine development, additional access methods will be required including barge transportation, construction of a more robust airfield, and construction of an all-weather access road to site.

Project development plans have considered the availability of staff, power, water, infrastructure and communications facilities to suit the planned development requirements.

There is sufficient area within the Project to host an open pit mining operation, including any proposed open pit, process facilities, waste dumps, and tailings. The DCLLC has secured the majority of surface rights for the area that may host these facilities. Negotiations will be required for surface rights for additional lands including road rights-of-way, the proposed wind farm, airstrip, Crooked Creek, Anaconda Creek and BTC facilities.

6.0 HISTORY

Gold was discovered in the Donlin Creek area in 1909, and placer production of about 30,000 ounces of gold occurred between 1909 and 1956. From 1956 to 1988, exploration comprised reconnaissance efforts, focusing on first-pass evaluation of ridge tops and outcrops, to determine the lode source of the placer gold.

In 1989, Western Gold Exploration and Mining Co. (WestGold) undertook the first modern exploration program. Eight prospects, Snow, Dome, Quartz, Carolyn (Far Side), Queen, Upper Lewis, Lower Lewis and Rochelieu, were identified from airborne and ground geophysical surveys, geological reconnaissance, geological mapping, rock chip, soil and auger sampling, and trenching. WestGold completed 10,423 m of drilling in 125 reverse circulation (RC) drill holes. Metallurgical and petrographic studies were also completed. The company also tested biogeochemical sampling and ground penetrating radar with positive results. This information was used to support a Mineral Resource estimate.

Follow-up exploration efforts by Teck Cominco in 1993 focused on defining the geological and mineralogical characteristics of the deposit, including petrographic, fluid inclusion and metallurgical studies, together with a limited trenching and soil sampling program. Teck Cominco updated the WestGold Mineral Resource estimate.

Barrick (formerly Placer Dome)'s 1995 acquisition of the Project resulted in the following work programs:

- 1996 – camp, airstrip and road construction, baseline environmental studies, 34,995 m ft of drilling in 144 core and RC drill holes, trenching, soil sampling and metallurgical testwork. This identified the American Creek aeromagnetic anomaly (ACMA) deposit
- 1997 – trenching, geological mapping, aerial photography, 23,900 m of drilling in 119 core and RC drill holes, ground electromagnetic and airborne geophysical surveys, soil samples, baseline environmental studies, geological modelling, Mineral Resource estimate
- 1998 – aerial photography (1:20,000 scale), geological reconnaissance, geological mapping, trenching, drilling of 24,131 m of core in 96 drill holes, baseline environmental studies, geological modelling, Mineral Resource estimate
- 1999 – geological mapping (1:10,000 scale), trenching, soil and rock chip sampling, ground induced polarization (IP) and resistivity surveys, 9,189 m of core drilled in 33 holes, baseline environmental studies, geological modelling, Mineral Resource estimate

- 2000 – ground IP and resistivity surveys, 1,403 m of core drilled in seven holes, baseline environmental studies.

Barrick (formerly Placer Dome) completed Mineral Resource estimates and an economic evaluation in 2000, and identified that the Project did not meet corporate targets for economic development.

NovaGold joint ventured into the property in 2001, and completed the following:

- 2001 – trenching, and 7,403 m of HQ core drilled in 42 holes
- 2002 – 51,039 m of core and RC drilling in 348 drill holes, including water bore and geotechnical drilling.

The 2002 work program culminated in a preliminary assessment study, and first time disclosure of Mineral Resources under NI 43-101. The study indicated a conventional open pit mining operation was appropriate for the mineralization outlined to date.

With Barrick (formerly Placer Dome)'s return as operator in 2003, the following work programs were conducted:

- 2003 – iterations of conceptual mining scenarios, updates to the Mineral Resource estimate completed in 2002, investigations of calcareous sandstone units, ground geophysical surveys in where planned facilities were to be situated, commenced monitor well drilling
- 2004 – installation of water monitoring wells, geological sampling for carbonate-rich material, 3,187 m of RC and HQ core drilling in 20 condemnation and geotechnical drill holes, exploration for sand and gravel supplies
- 2005 – auger and test pits for geotechnical purposes, 28,394 m of core and RC drilling in 120 infill, water well, geotechnical, calcium carbonate exploration and condemnation drill holes.

An updated Mineral Resource estimate was prepared by Barrick (formerly Placer Dome) and validated by NovaGold in 2006. SRK Consulting subsequently prepared a preliminary assessment study on behalf of NovaGold, confirming that a conventional open pit mining operation was technically feasible.

During 2006–2008, Barrick completed work, including:

- 2006 – water geochemical studies, peat exploration, wind power generation studies, metallurgical studies, and 92,804 m of infill, geotechnical, metallurgical, condemnation core drilling in 327 drill holes, geotechnical pits and auger holes, updated geology model and Mineral Resource estimates

- 2007 – auger holes and test pits for geotechnical studies, interim geological models and resource estimates, 50,562 m of core and RC drilling in 124 drill holes
- 2008 – 108 HQ/NQ core holes totalling 33,425 m, auger holes and test pits for geotechnical studies, soil, stream sediment and stream concentrate geochemical samples, Mineral Resource estimate update. The 2008 drilling results are not included in the Mineral Resource estimate that is the basis of the feasibility study, and are discussed in Section 11 of the Report. It is unlikely that the 2008 drilling will have a material impact on the Project.

This work resulted in preparation of a feasibility study in 2007 and an update to the feasibility study in 2008, which is the subject of this Report.

7.0 GEOLOGICAL SETTING

7.1 Regional Geology

The Kuskokwim region of southwestern Alaska is predominately underlain by rocks of the Upper Cretaceous Kuskokwim group (Figure 7-1). These include coarse- through fine-grained clastic rocks that reach an estimated thickness of 12 km. Minor basin margin andesitic tuff and flows are also present near the top of the sequence and may represent an initiation of volcanism that later culminated in widespread Late Cretaceous and early Tertiary igneous activity. These basin margin volcanic rocks also suggest that deep penetrating structures controlled basin subsidence.

Kuskokwim Group sediments filled a northeast-trending strike-slip basin that subsided between a series of amalgamated terranes including Mesozoic marine volcanic rocks, Palaeozoic clastic and carbonate rocks, and Proterozoic metamorphic rocks. Kuskokwim Group rocks generally do not display penetrative metamorphic fabric, but they are locally folded.

Igneous activity was coeval with Late Cretaceous sedimentation in the Kuskokwim basin and continued into the early Tertiary. Intermediate composition volcano-plutonic complexes intrude and overlie Kuskokwim Group rocks throughout the region. The igneous rocks are predominantly tuffs, flows, and composite co-magmatic monzonite to granodiorite plutons. Volcanic and plutonic rocks range in age from 76 Ma to 63 Ma and 71 Ma to 66 Ma, respectively. Kuskokwim sedimentary rocks are often display extensive hornfelsing near plutons. Volumetrically minor Late Cretaceous intermediate to mafic intrusive bodies are also common and often associated with mercury and antimony occurrences. Felsic to intermediate hypabyssal granite to granodiorite porphyry dykes, sills, and plugs are also widely distributed and often associated with placer and lode gold occurrences (e.g., Donlin Creek). Many dykes were emplaced within or near northeast-trending extensional zones. Contacts between porphyry igneous rocks and Kuskokwim sedimentary rocks are generally sharp and do not display hornfels margins. Age dates range from 70 Ma to 65 Ma, but a genetic association with the volcano-plutonic complexes is uncertain.

The Donlin Creek area lies between two regional, northeast-trending, right lateral faults: the Denali–Farewell fault system to the south, and the Iditarod–Nixon Fork fault system to the north. The region contains numerous northeast to east–northeast- and northwest to west–northwest-trending lineaments that probably represent steeply dipping strike-slip faults. Fault movement in the Donlin Creek region appears to be right lateral on northeast structures and left lateral on northwest structures.

Figure 7-1: Regional Geology Map of Donlin Creek Area

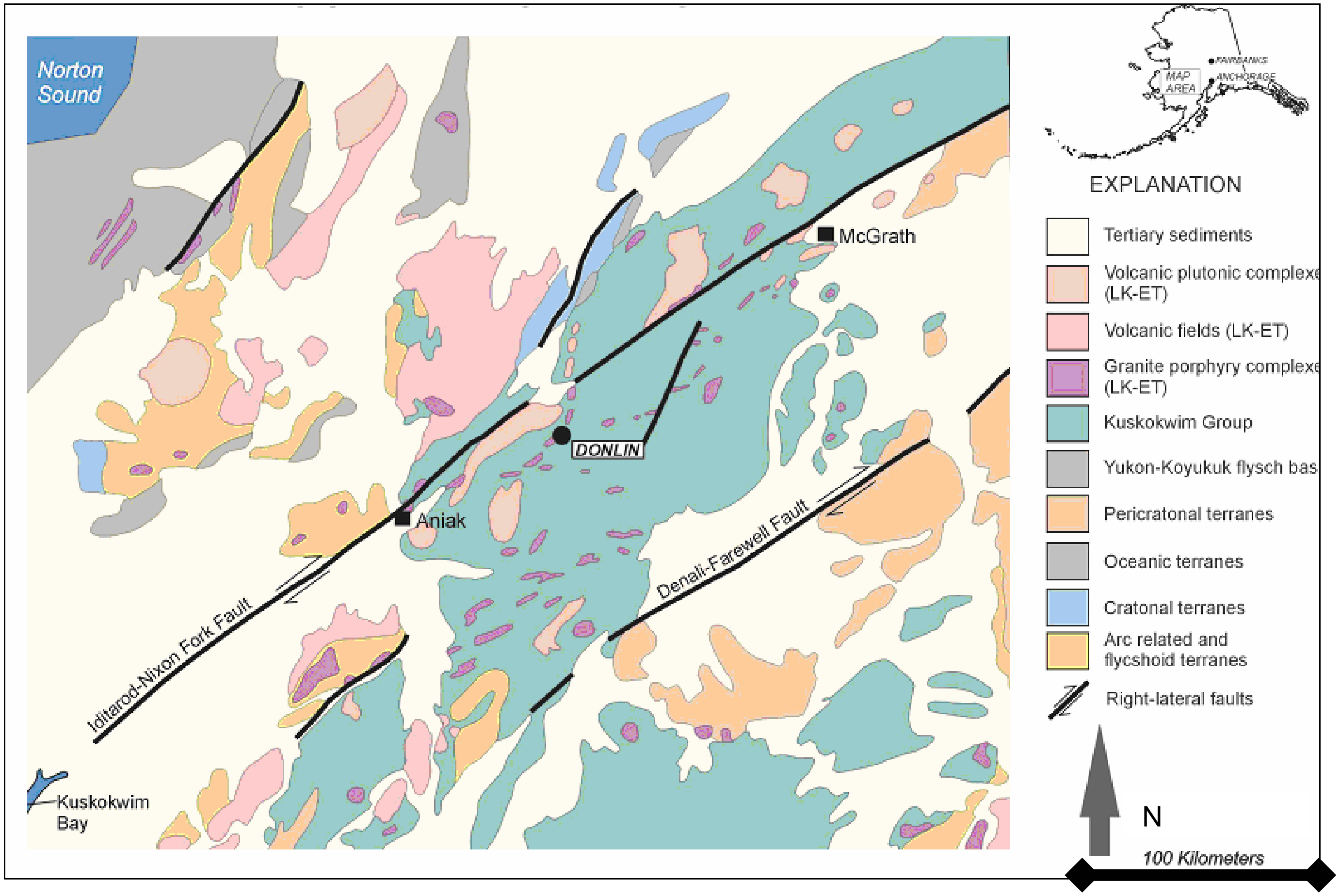


Figure courtesy DCLLC. Map north is to top of plan, and approximates geographic north. Map scale is as shown.

Folding in the region probably occurred soon after sedimentation, since folds are truncated by the volcano-plutonic complexes. East-trending open folds are prominent east of the Donlin Creek area, but appear truncated to the west by the Donlin Creek fault, a splay of the Iditarod–Nixon Fork fault.

7.2 Property Geology

Property scale geology is largely interpreted from trenches and drill holes because outcrop is limited and of generally poor quality. Simplified property scale igneous geology was included in Figure 4-1. Undivided Kuskokwim Group sedimentary rocks and granite porphyry complexes associated with the 70 Ma to 65 Ma igneous event are the main rock units. Greywacke is dominant in the northern part of the area (Lewis, Queen, Rochelieu, Akivik), while shale-rich units are common in the southern part of the area (South Lewis, ACMA). Overall, sedimentary structure in the northern resource area (Lewis) is monoclinal, while sedimentary rocks in the southern resource area (ACMA) display open easterly-trending folds.

The oldest igneous rocks at Donlin Creek are 74 Ma to 72 Ma intermediate to mafic dykes and sills. They are not abundant, but occur widely throughout the property as generally thin and discontinuous bodies. The later, more voluminous 70 Ma to 65 Ma granite porphyry dykes and sills vary from a few feet to 60 m wide and intrude the sedimentary rocks along a 8 km long x 3 km wide northeast-trending corridor. Sills are common in the southern area (shale-dominant), while dykes dominate in the north (greywacke-dominant). The granite porphyry dykes and sills have similar mineralogy and generally display textures indicative of relatively shallow emplacement. Although these rocks belong to the regionally important granite porphyry igneous event, geologists working on the property classify them into five textural varieties of rhyodacite. Rhyodacite is a term normally reserved for volcanic to sub-volcanic rock types, but it is also used informally for igneous rocks emplaced at a shallow depth. These units are chemically similar, temporally and spatially related, and probably reflect textural variations of related intrusive events. Differences include phenocryst size and abundance, groundmass texture, and overall colour.

The earliest deformation on the property (pre-74 Ma through 68 Ma) includes flexural slip folding and southward tilting of the sedimentary sequence. Continued compression further advanced the fold-thrust response and formed east–southeast-trending and plunging folds or monoclinal warps. Shallow to moderately north-dipping reverse or thrust fault structures developed along bedding slips, competency breaks, and ramped through fold hinge areas.

Mafic dykes and sills (74–72 Ma) were intruded into compressional structures prior to the intrusion of rhyodacite dykes and sills (72–70 Ma) into northeast extensional faults and layer parallel weaknesses, respectively. Northeast and northwest-striking oblique slip faults formed near the end of the compression event and displace earlier low angle reverse faults as well as intrusive rocks. As intrusive activity decreased, gold-bearing hydrothermal fluids invaded north—northeast-oriented fractures that formed during east–southeast to west–northwest extension (72–65 Ma, averaging 70 Ma).

The fractures are best developed in the relatively competent igneous rocks and coarser greywacke-dominant sedimentary sequences.

7.3 Deposit Geology

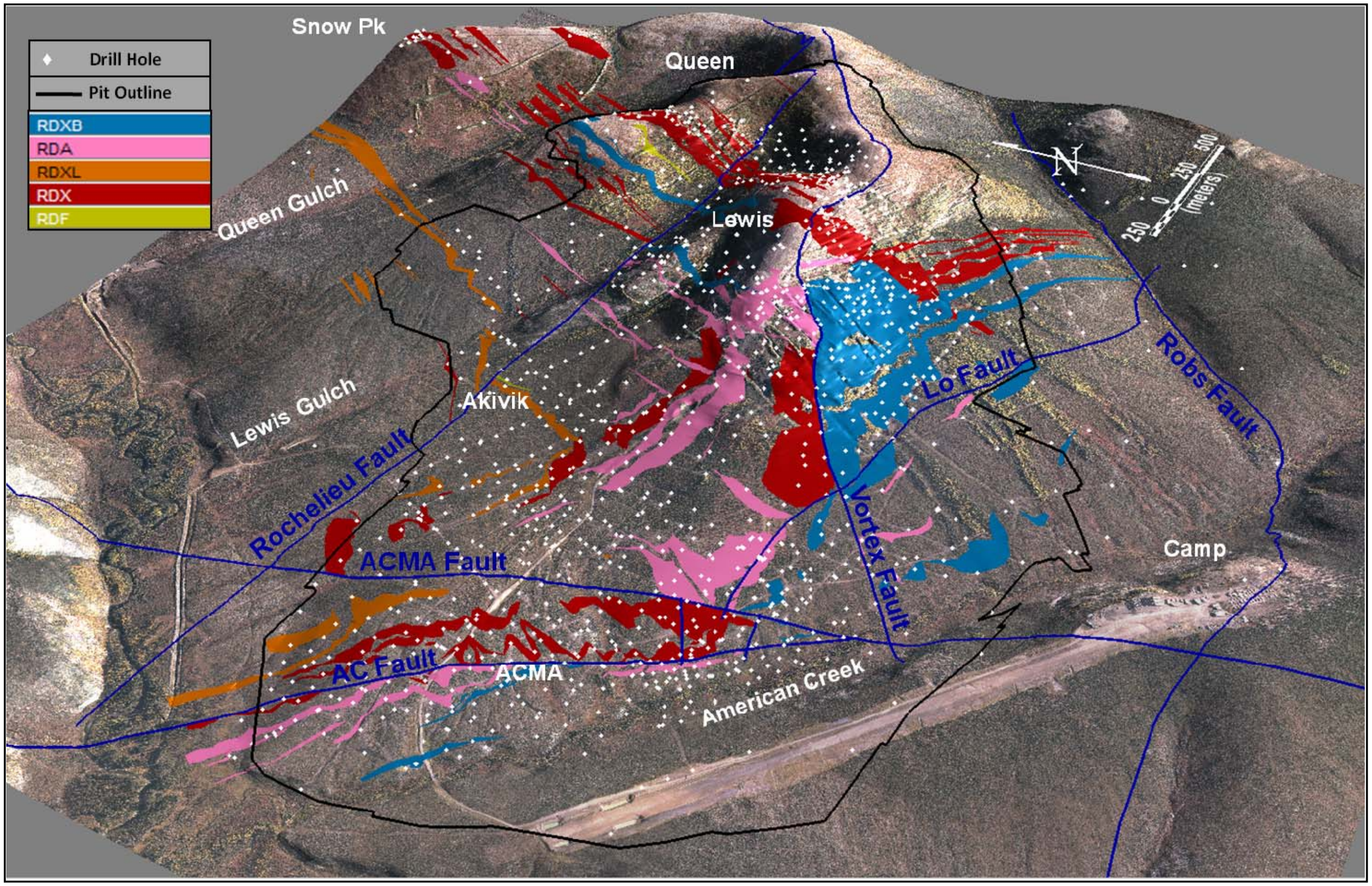
An interpretative geological map of the area is shown in Figure 7-2, and is largely based on trench exposures and drill holes. As with the broader-scale property geology, outcrops are generally poor in quality and limited to ridge lines. Hydrothermally-altered intrusive rock phases may be difficult to interpret in weathered exposures, and the sedimentary stratigraphy is subtle and can be problematic to deduce even with the aid of close-spaced drill holes. The general surface geology, therefore, focused on igneous rocks identified from trench and drill hole data. Detailed trench and other surface geological maps have not yet been compiled into a geological “fact map” for the area of the Mineral Resource estimate.

7.3.1 Sedimentary Stratigraphy

Preliminary stratigraphy for sedimentary rocks in the immediate deposit area is shown in Table 7-1. The stratigraphy in the deposit area consists of multiple deep and shallow water clastic rock sequences with complex transition zones of rhythmically interbedded shales and greywacke (lithic sandstone). Marker beds are not yet recognized, so absolute stratigraphic breaks are difficult to identify.

Thicknesses presented for each unit in Table 7-1 are from the ACMA area. In general, the Main Shale appears to thin to the west, whereas the Upper Siltstone appears to thicken in the same direction. The northern part of the resource area is mostly greywacke, while the southern area is shale rich. The coarse-grained greywacke contains abundant metamorphic lithic fragments and locally abundant igneous and sedimentary clasts. Shale-rich sedimentary rocks contain minor syngenetic pyrite, minor coaly plant debris to thin coal seams, and rare thin volcanic ashfall beds that may be as much as 10 cm thick. The ash beds are restricted to low-energy shale and argillite intervals.

Figure 7-2: Interpretative Surface Geological Map



Note: Figure courtesy Barrick. Figure shows igneous rock units, faults, drill holes, and planned pit outline. Section is an oblique view looking N30E.
Key: RDXB = Blue Porphyry; RDA = Aphanitic Porphyry; RDXL = Lath-Rich Porphyry; RDX = Crowded Porphyry; RDF = Fine-Grained Porphyry.

Table 7-1: Donlin Creek Stratigraphy

Assigned Nomenclature	Principal Rock Type	Apparent Thickness (m)
Upper Greywacke	greywacke	100+
Upper Siltstone	siltstone/shale	50
Main Greywacke	greywacke	80
Main Shale	shale/argillite	up to 140 (with sills)
Basal Greywacke	greywacke	200+

Attempts to use ash beds as stratigraphic markers were unsuccessful. Calcareous horizons and conglomerate beds were useful on a very local scale. Abundant bedding plane faults occur in most shale-rich intervals.

7.3.2 Igneous Rocks

The mafic dykes and sills and the five varieties of rhyodacite recognized in the Donlin Creek deposit are listed from oldest to youngest in Table 7-2, and are described in the following subsections.

Table 7-2: Donlin Creek Intrusive Rocks

Name	Code	Age
Mafic Dykes/Sills	MD	oldest
Fine-grained Porphyry	RDF	-
Crowded Porphyry	RDX	-
Lath-Rich Porphyry	RDXL	-
Aphanitic Porphyry	RDA	-
Blue Porphyry	RDXB	youngest

MD – Mafic Dykes

The earliest intrusive rocks at Donlin Creek are a series of intermediate to mafic dykes and sills (MD). These dykes and sills are thin, 1 m to 3 m, and are normally characterized by intense pervasive carbonate and bright green clay alteration. The mafic rocks are compositionally variable, typically porphyritic, and were compared to lamprophyres. In the transition area between Akivik and ACMA, an area of extremely abundant mafic sills occurs within the Lower Greywacke immediately below the Main Shale. The mafic sills locally host high-grade gold.

RDF – Fine-grained Porphyry

The RDF dykes are the earliest rhyodacite intrusions recognized at Donlin. They are typically fine-grained, felsic porphyries with distinctive small feldspar phenocrysts set in a grey fine-grained matrix. RDF intrusive rocks occur as dykes 5 m to 10 m wide and appear to fill the north–northeast extension fracture zones and the east-striking compressional faults (e.g., Lo fault).

RDX – Crowded Porphyry

The RDX rocks are volumetrically the most significant intrusive phase on the property. The unit is characterized by a homogenous crowded porphyry texture and sharp intrusive contacts with little (<5 cm) or no chill margins. The unit occurs as two 50 m to 100 m wide dyke zones in the eastern edge of the north to north–northeast Lewis/South Lewis mineralized trend. RDX also occurs as sills throughout the southern portion of the property and generally occupies the basal part of the sill sequence. The sills begin as sub-horizontal units in the South Lewis area and follow an apparent syncline/anticline structure as they dip from sub-horizontal to near-overturned at depth in the ACMA area.

RDXL – Lath-rich Porphyry

The RDXL unit is a rhyodacite phase characterized by large elongate plagioclase laths in a population of smaller potassium feldspar phenocrysts. It has significant coarser-grained biotite and seems to be more texturally enhanced by alteration than the other units. RDXL occurs as two important dykes in the Akivik area that strike south into the centre of the ACMA deposit. In the Akivik and ACMA areas, RDXL occurs as a significant sill immediately below the RDX sill. The RDXL sill continues to the west, but pinches out to the east. RDXL dykes also occur within the main Lewis area RDX dyke trend, but these dykes are volumetrically insignificant.

RDA – Aphanitic Porphyry

The RDA unit is a rhyodacite rock is characterized by having a salt-and-pepper texture of fine biotite phenocrysts, variable quartz, and potassium feldspar phenocrysts in an aphanitic matrix with distinctive flow-banded margins. Numerous (up to eight) RDA dykes strike south from the Vortex/Rochelieu area into the East ACMA/ACMA area. The dykes are typically found west of the Vortex fault, but can be found between the Lo and Vortex and below the Lo fault. An extensive sill package of RDA is located immediately above the RDX sills in the ACMA area. In west ACMA, the RDA sills are buttressed against, and locally cross-cut, RDX sills. Another package of RDA sills is found south of the AC fault, in the Aurora domain.

RDXB – Blue Porphyry

The final intrusive event at Donlin Creek is represented by Blue Porphyry (RDXB). This unit is coarsely porphyritic with large blocky feldspars set in a graphite- and sulphide-rich matrix that gives the unit a distinctively dark color that occasionally looks like an alteration product. It is often more intense near faults or appears to have pooled behind fluid barriers such as fractures and veins. Alteration fronts occasionally cut across feldspar phenocrysts, and the darker color near contacts with carbonaceous rocks suggests remobilization of carbon. The RDXB unit locally hosts important high-grade disseminated sulphide material in addition to gold-bearing veins. RDXB occurs as two major dykes, the Lewis Blue Porphyry dyke and the Vortex Blue Porphyry dyke. Extensive thin RDXB sills occur in the uppermost part of the sill sequence in the South Lewis and ACMA areas. The RDXB sills also occur as both distinct sills and co-mingled with RDA in the core of ACMA and in the Aurora domains.

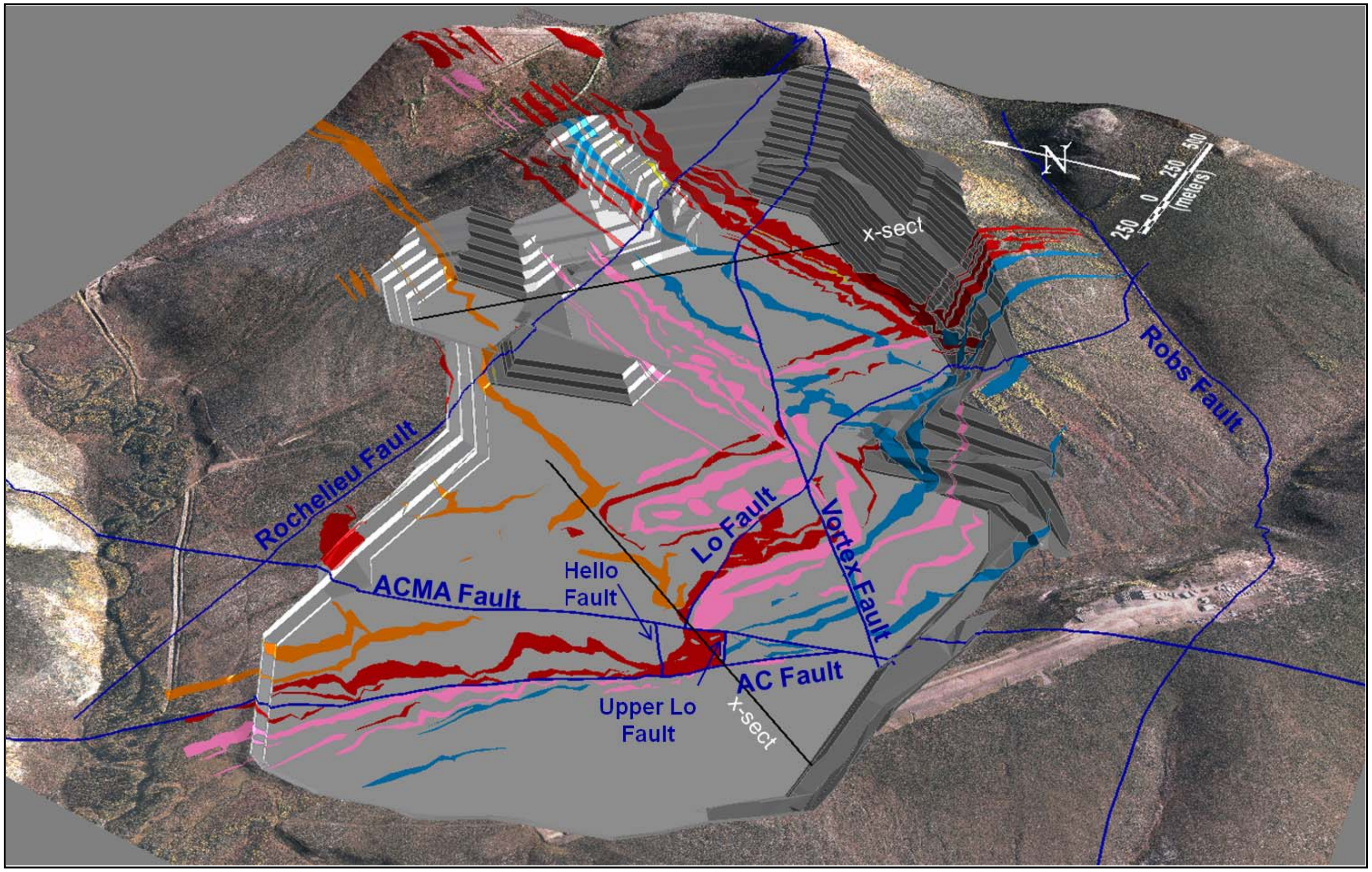
7.3.3 Structural Geology

A projection of the 3D geological model on the 100 m pit bench level (Figure 7-3) illustrates modelled intrusive dykes, sills, and faults on the deposit scale. The morphology of intrusive rocks is largely determined by sedimentary rock stratigraphy and structure.

The generally thick-bedded, massive greywacke is the dominant sedimentary unit in the northern part of the resource area (Lewis and Queen) and is generally monoclinal with average southwest dips of about 10° to 50°. These rocks host planar dykes that intruded northeast-striking, southeast dipping extensional faults. In contrast, southeast plunging open folds (?) and/or monoclinal warps are evident in the pattern of intrusive sills in the more shale rich South Lewis–ACMA transition area. The dykes seem to “feed” into the footwall side of the south-dipping sill sequence, but these relationships remain obscure in drill hole intersections.

Recent drilling shows that the intrusive sills and enclosing sedimentary beds become steeper to the southwest and are vertical to overturned in the main ACMA area near American Creek. This steepening could indicate an asymmetric to overturned anticlinal fold limb or a possible drag fold on a fault sub-parallel to American Creek.

Figure 7-3: 100 m Bench Level Geology Map



Note: Figure courtesy Barrick. Shows intrusive rocks, faults, and proposed DC8 pit. Oblique view looking northeasterly. Key: RDXB = Blue Porphyry (blue); RDA = Aphanitic Porphyry (pink); RDXL = Lath-Rich Porphyry (orange); RDX = Crowded Porphyry (red); RDF = Fine-Grained Porphyry (yellow)

Structural data from deep oriented core holes show an abrupt change from vertical to nearly horizontal bedding attitudes at depth that may also indicate a drag fold on a deep, basal thrust fault beneath the ACMA deposit area.

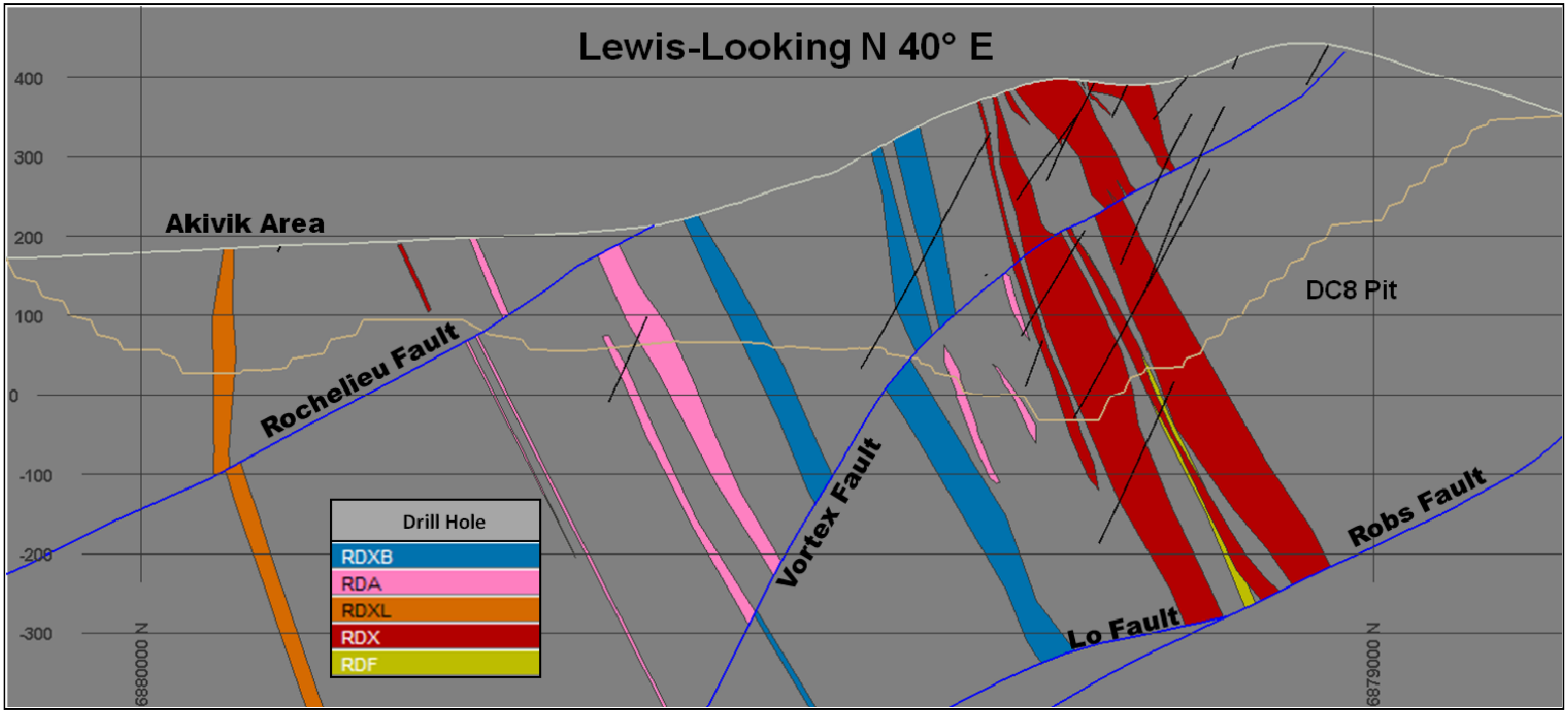
The Lo Fault (a shallow to moderately north-dipping, low-angle reverse or thrust fault) follows the greywacke-shale transition zone in the South Lewis area and may have created a barrier to the southward intrusion of the planar dyke sequence. This barrier and the proliferation of layer parallel faults in the shale-rich South Lewis–ACMA sedimentary section appears to have deflected the northeast-trending Lewis area dykes, and caused them to spread out as west–northwest to east–southeast-trending sills in the ACMA area.

Subsequent high-angle northwest- (AC and ACMA Faults) and northeast-striking (Vortex Fault) oblique slip faults postdate the low angle faults and the intrusive activity, and may have remained intermittently active during and after the hydrothermal event. Local auriferous and sulphide-rich tectonic or hydrothermal breccia bodies indicate that some faults enhanced ground preparation and/or acted as barriers to gold-bearing hydrothermal fluids.

Modeled faults are moderately well documented in drill holes and in rare instances verified in surface exposures. Faults are defined in drill holes by narrow zones of gouge, brecciation, fracturing, and slickenside surfaces. In contrast to the multiple gouge seams up to 2 m wide and the fracture zones up to 150 m wide in typical interbedded greywacke and shale, the faults in intrusive rocks generally produce broad but relatively subtle zones of fracturing or “crackle breccia” and narrow gouge seams. Consequently, definitive drill core evidence for some faults is often absent or ambiguous. In the latter case, only the interpreted juxtaposition of rock units across a fault defines its location and the direction and amount of fault displacement.

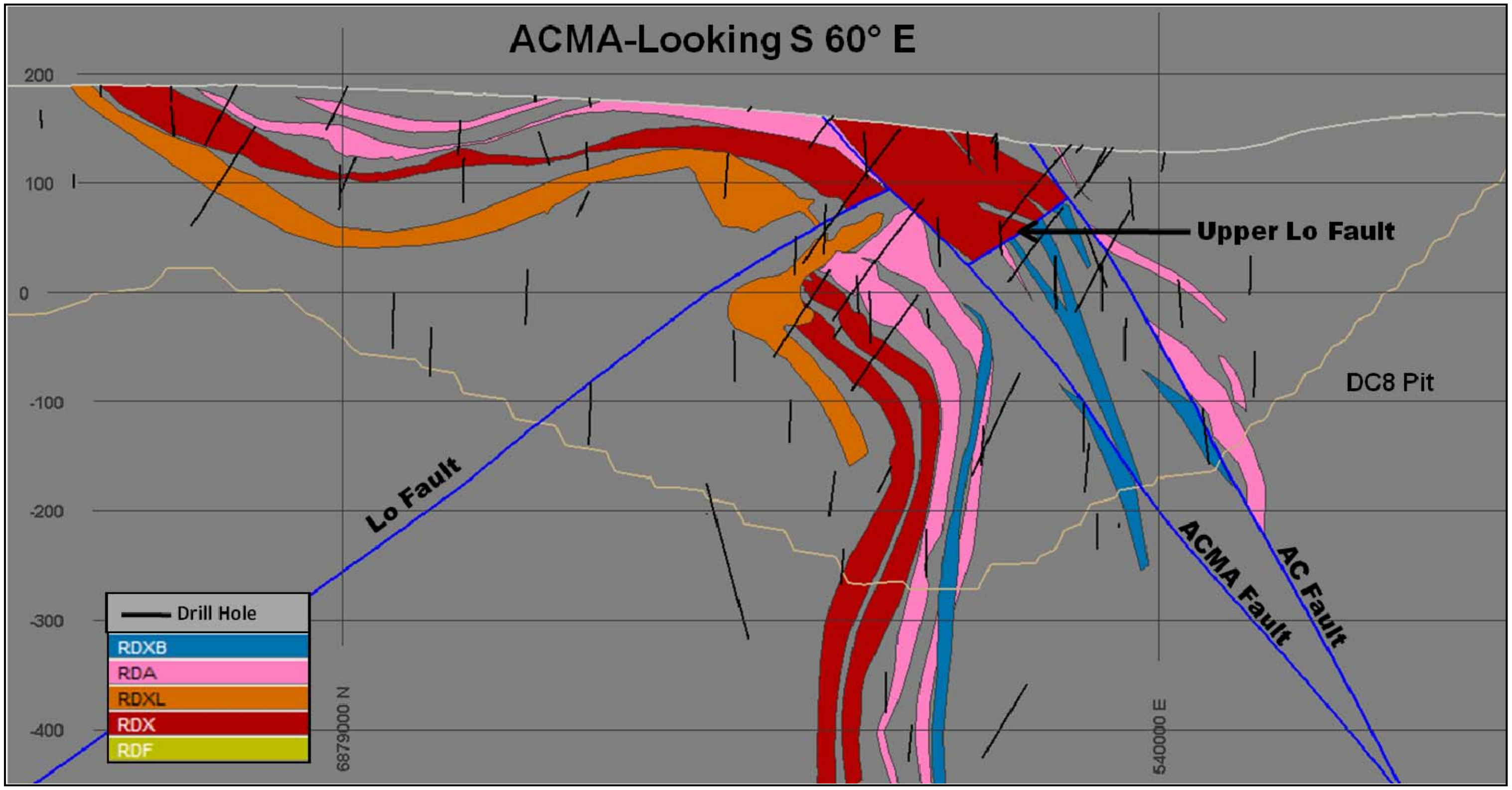
The most important faults in the geological model, from earliest to youngest, are the Lo Fault, American Creek (AC) Fault, Vortex Fault, and ACMA Fault. Rob’s Fault, southeast of the area of the Mineral Resource estimate, is coincident with a strong topographic linear, and remains hypothetical. Figures 7-4 and 7-5, which are vertical sections normal to the Lewis dyke and ACMA sill trends, illustrate the contrasting structural style of intrusive rocks and faults.

Figure 7-4: Lewis Area Section



Note: Figure courtesy Barrick. Pit outline shown is the DC8 open pit, used to constrain Mineral Resources in this Report. Key: RDXB = Blue Porphyry; RDA = Aphanitic Porphyry; RDXL = Lath-Rich Porphyry; RDX = Crowded Porphyry; RDF = Fine-Grained Porphyry

Figure 7-5: ACMA Area Section

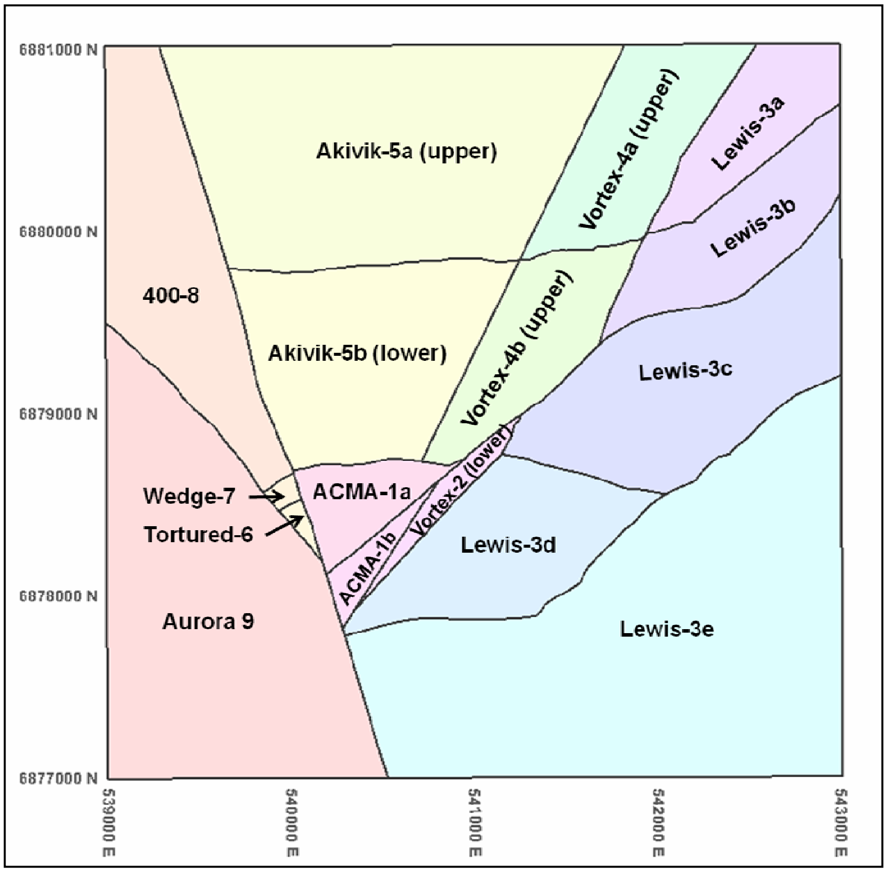


Note: Figure courtesy Barrick. Pit outline shown is the DC8 open pit used to constrain Mineral Resources in this Report. Key: RDXB = Blue Porphyry; RDA = Aphanitic Porphyry; RDXL = Lath-Rich Porphyry; RDX = Crowded Porphyry; RDF = Fine-Grained Porphyry

7.4 Geological Domains

For the purposes of geological modelling and Mineral Resource estimation, nine geology and mineral domains with up to five sub-domains were established (Figure 7-6). Domains were established using lithological and structural criteria.

Figure 7-6: Geological Domain Boundary Map



Note: Figure courtesy DCLLC. North is to top of page. Grid spacing is 1 km x 1 km

7.5 Comments on Section 7

The deposit settings, lithologies, and structural and alteration controls on mineralization are well understood, and the geological understanding is sufficient to support Mineral Resource and Mineral Reserve estimation.

8.0 DEPOSIT TYPES

Two distinct styles of mineralization occur within the Donlin Creek trend as discussed below. Exploration programs were tailored to best explore these styles.

The mineralization in the northern part of the district (Dome–Duqum prospect) is characterized by copper- and gold-bearing stockwork veinlets in hornfels while the southern part of the district (ACMA–Lewis deposit) is characterized by auriferous arsenopyrite-bearing quartz and sulphide-only veins associated with felsic intrusive rocks. The age of Dome–Duqum mineralization was established to be older than the ACMA–Lewis style mineralization using cross-cutting relationships observed during trench mapping in the 1999 field season; however, the genetic relationship between the two types of mineralization is unknown.

The Dome–Duqum mineralization is best characterized as a high-temperature, porphyry-style mineralization with fracture-controlled stockwork, and laminated quartz-only veins containing varying proportions of copper, zinc, bismuth, silver, tellurium, selenium and local native gold mineralization. Silicification is locally associated with the veins, and contact metamorphism (hornfelsing) of the sedimentary rocks adjacent to host intrusive rocks is common in areas containing this style of mineralization.

The ACMA–Lewis style of mineralization is best characterized as a low-temperature, low-sulphidation epithermal system, and is the dominant style of mineralization within the current resource area. The ACMA–Lewis style consists of sheeted quartz, quartz–carbonate and sulphide-only veins characterized by a gold–arsenic–antimony–mercury geochemical signature. The bulk of the gold occurs in the lattice structure of arsenopyrite. Stibnite, realgar and native arsenic are commonly observed associated with zones of higher-grade gold mineralization but do not appear to host any significant gold mineralization compared to arsenopyrite. Disseminated gold-bearing arsenopyrite can also be found typically adjacent to veins and vein zones. Mineralization is best developed in all intrusive rocks and, to a much lesser extent, sedimentary rocks (mainly greywacke). Sedimentary units in areas of ACMA–Lewis mineralization typically show no contact metasomatic effects.

9.0 MINERALIZATION

North–northeast-oriented fracture zones that dip to the southeast are the primary control on gold-bearing vein distribution within the north-northeast mineralized corridors. Composite vein zones or mineralized corridors range up to 30 m in width and extend for hundreds of metres along strike. Intrusive rocks and to a lesser extent competent massive greywacke are the most favoured host rocks, and act as a secondary control on the mineralization. Gold distribution in the deposit closely mimics the intrusive rocks, which contain about 74% of the Mineral Resource. Structural zones in competent sedimentary units account for the remaining 26%.

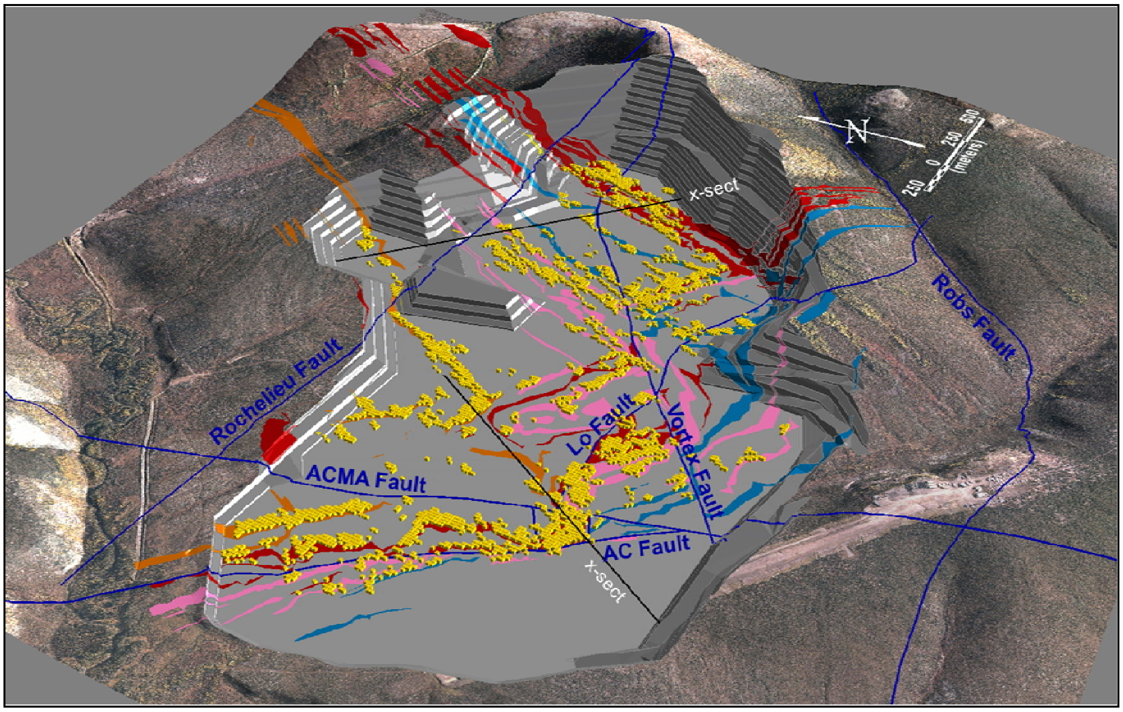
Mineralized material in the ACMA or sill-dominant part of the deposit tends to be higher grade and more continuous compared to Lewis and other dyke dominant areas of the deposit. The most extensive and highest grade mineralized zones in ACMA are located where “feeder” dykes intersect the sill sequence. Mineralized zones follow steeply-dipping dykes and sills beyond the depth limits of current drilling, or over a vertical range of at least 945 m. Figure 9-1 illustrates the general gold distribution relative to intrusive rocks in plan, projected to the 100 m elevation level. Figures 9-2 and 9-3 are sections through Lewis and ACMA respectively, displaying the gold distribution.

9.1 Veins

Vein mineral assemblages show a continuum from pyrite through arsenopyrite, native arsenic, and realgar, rather than discrete paragenetic stages.

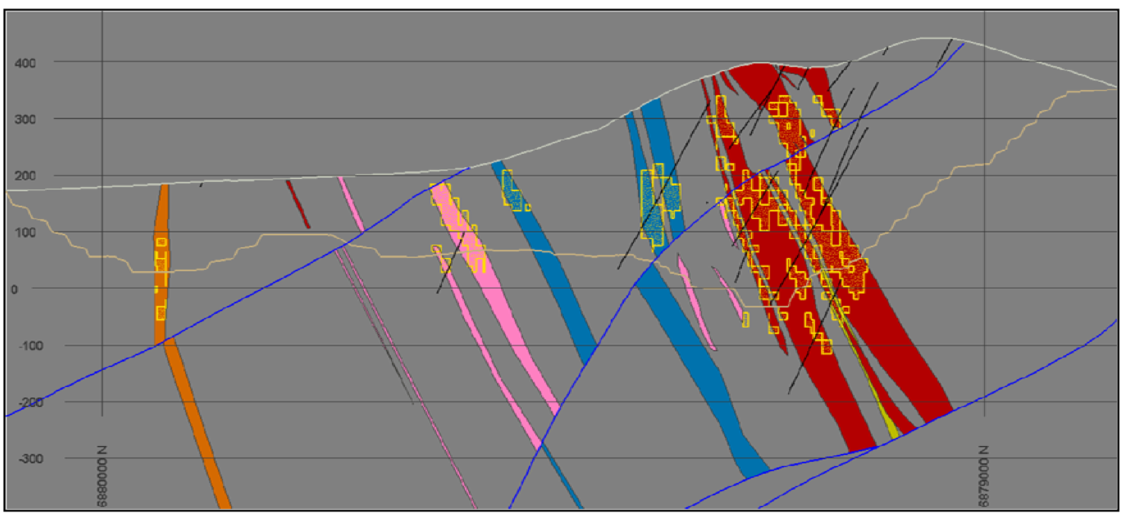
Stibnite is ubiquitous in all vein types, but seems to increase in later vein stages. Gold grade and vein quartz generally increase from vein types V1 through V3 and then markedly decrease in V4, a carbonate-dominant vein type. Reflectance spectroscopy was routinely used to quantitatively define specific clay, illite, and carbonate alteration zones in intrusive rocks. Proximal to distal silicate alteration zones and carbonate and graphite alteration products are associated with the ACMA–Lewis hydrothermal system; silica is largely restricted to veins and is not an important wall rock alteration product.

Figure 9-1: 100 m Bench Level Gold Distribution (>1 g/t Au Grade Blocks)



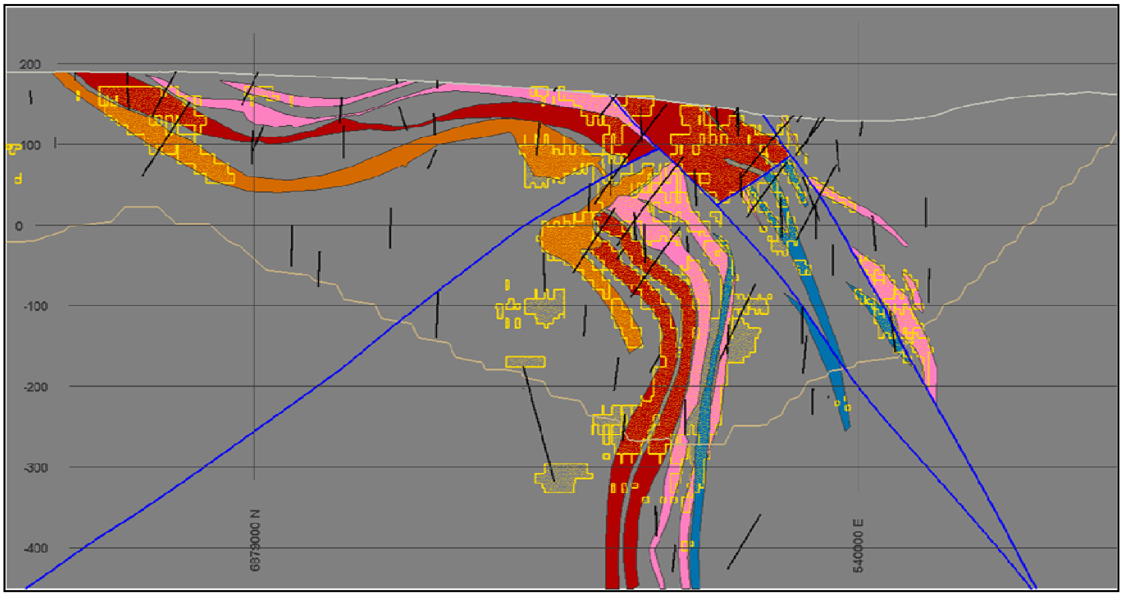
Note: Figure courtesy Barrick. Pit outline displayed is the DC8 pit

Figure 9-2: Lewis Area Gold Distribution (>1 g/t Au Grade Blocks)



Note: Figure courtesy Barrick. Pit outline displayed is the DC8 pit. Key for rock type legend is included in Figure 7-5

Figure 9-3: ACMA Area Gold Distribution (>1 g/t Au Grade Blocks)



Note: Figure courtesy Barrick. Pit outline displayed is the DC8 pit. Key for rock type legend is included in Figure 7-6

9.2 Mineralization

Gold-bearing sulphides occur in both veins and disseminated zones in mafic igneous bodies, rhyodacite dykes and sills, and sedimentary rocks. Quartz-carbonate-sulphide (pyrite, stibnite, and arsenopyrite) veins are the primary mineralized features, but gold also occurs in thin, discontinuous sulphide fracture fillings.

Vein widths seldom exceed 1 cm; vein densities range up to 5 to 10 per metre; and the widths of the vein zones vary from 2 m to 35 m wide. Individual vein zones generally display limited lateral and vertical continuity. However, swarms of many anastomozing vein zones form larger mineralized corridors characterized by extensive lateral and depth continuity.

The ACMA–Lewis style of mineralization is consistent with a low-temperature, low-sulphidation, genetic gold model. The deposit(s) is characterized by an Au–As–Sb–Hg geochemical signature, quartz ± carbonate and sulphide veins, and disseminated sulphide minerals. Common minerals observed in the mineralized zones include pyrite, marcasite, arsenopyrite, stibnite, realgar, and native arsenic.

Pyrite is the most common mineral and appears to be the earliest sulphide phase. It is ubiquitous in the rhyodacite, and occurs as disseminated grains or micro-fracture fillings. Disseminated pyrite in the sedimentary rocks occurs as fine to coarse grains (up to 5 mm across) preferentially concentrated near dyke/sill contacts or as syngenetic pyrite along sedimentary laminations. Relative abundance of pyrite is not an indicator of gold grade.

Broad selvages of disseminated gold-bearing arsenopyrite and pyrite are found adjacent to veins and vein zones. Arsenopyrite commonly replaces pyrite, and typically occurs as fine to very fine grains disseminated in intrusive rocks, and as coarser aggregates in fractures and quartz–carbonate veins. In practice, fine-grained arsenopyrite can be difficult to distinguish from ubiquitous disseminated graphite. Disseminated sulphides typically replace biotite or other mafic mineral sites and rim or replace illite–clay–carbonate-altered feldspar phenocrysts.

Native arsenic occurs as dark grey, granular, massive to botryoidal grains that often fill vugs in quartz–carbonate ± sulphide veins, and open spaces in breccias or fractures. Realgar occurs in late, quartz–sulphide veins. Stibnite commonly occurs as disseminated grains and masses within carbonate veins and occasionally as interlocking needles in open spaces within quartz–carbonate veins and on fracture surfaces. Other accessory sulphides and sulphosalts observed in the deposit include marcasite, pyrrhotite, chalcopyrite, chalcocite, covellite, bornite, tennantite, tetrahedrite, galena, sphalerite, and boulangerite. Pyrrhotite, stibnite, and boulangerite are paragenetically late and appear to post-date most deformation while chalcopyrite, tennantite–tetrahedrite, pyrite, and arsenopyrite are both pre- and post-deformation.

Very rare native gold particles (1 µm to 20 µm in size) have been observed in process mineralogy studies of the sulphide grains from the ACMA–Lewis area, but most of the gold is in “solid solution” in the crystal structure of arsenopyrite, and to a lesser extent, in pyrite. Typical native gold seen in polished sections occurs as 1 µm to 3 µm blebs with no clear paragenetic relationship to other minerals. Gold-bearing arsenopyrite in ACMA is associated with gold-bearing marcasite and rarely with pyrite. Lewis pyrite is generally not gold bearing. Fine-grained arsenopyrite (<20 µm diameter) contains five to ten times more gold than coarse-grained arsenopyrite. Stibnite, realgar, and native arsenic are often associated with higher gold grades, but contribute very minor gold compared to associated arsenopyrite.

9.3 Minor Elements and Deleterious Materials

The most abundant minor elements associated with gold-bearing material are iron, arsenic, antimony, and sulphur. They are contained primarily in the mineral suite associated with hydrothermal deposition of gold, including pyrite, arsenopyrite, realgar, native arsenic, and stibnite. Minor hydrothermal pyrrhotite, marcasite and syngenetic or sedimentary pyrite, also account for some of the iron and sulphur.

Much less abundant elements such as copper, lead, and zinc are contained in the relatively rare or accessory hydrothermal mineral species observed in the deposit, including chalcopyrite, chalcocite, covellite, tennantite, tetrahedrite, bornite, native copper, galena, sphalerite, and boulangerite. Small amounts of silver in the deposit are most likely accommodated within the crystal structures of tetrahedrite and galena, and to a lesser extent in some of the other sulphides. Molybdenum occurs as rare molybdenite. Very minor nickel in the secondary sulphide mineral millerite and minor cobalt in various secondary minerals have been observed in the sedimentary rocks. The Ni and Co probably have a sedimentary origin.

Three elements that have particular processing significance are mercury, chlorine, and fluorine. Graphitic carbon and carbonate minerals may also negatively affect the metallurgical process.

Most of the Hg occurs as colloidal or microscopic cinnabar inclusions in finer-grained sulphides. Pyrite accounts for about 66% of the Hg in sulphide concentrates, followed by marcasite (18%) and arsenopyrite (3.6%). Elevated Hg is also associated with realgar. Macroscopic cinnabar (HgS) is generally absent or exceedingly rare in the area of the Mineral Resource estimate.

Chlorine in chloride ions can dissolve gold during pressure oxidation (POX) as AuCl3. This gold compound is “preg-robbed,” or adsorbed, by carbonaceous matter and may become incorporated in iron precipitates, resulting in gold losses up to 10%. Fluorine is very corrosive in the POX process. Process mineralogy studies show that muscovite and apatite are the principal sources of chlorine and fluorine in sulphide concentrates and that the relatively more abundant muscovite accounts for most of the chlorine (59%) and fluorine (93%). Muscovite is normally a rock-forming mineral, but it can also form during hydrothermal alteration along with structurally similar alteration products (illite) associated with gold-bearing rocks. Apatite is commonly found as an accessory mineral in igneous and sedimentary rocks and as a hydrothermal alteration or vein mineral.

Graphitic carbon is preg-robbing and relatively abundant in the sedimentary rocks and variably disseminated in the intrusive rocks as a possible alteration product. Sulphide-carbonate binary particles tend not to float well. Carbonate minerals occur as both pervasive, fine-grained hydrothermal alteration products that are often intergrown with fine disseminated sulphide, and also in carbonate and quartz–carbonate ± sulphide veins. Common carbonate minerals include calcite, ankerite, dolomite, and very minor siderite.

9.4 Comment on Section 9

The mineralization style and setting is sufficiently well understood to support Mineral Resource and Mineral Reserve estimation. The deposit contains elements that may be deleterious in the proposed processing facility.

10.0 EXPLORATION

10.1 Grids and Surveys

Drill hole collar and trench locations were tied to a surveyed ground control net using conventional theodolite survey methods from 1988 through 1993. Drill hole collars were surveyed with Brunton compass and hip chain in 1995. A Motorola global positioning system (GPS) was used in early 1996 to establish survey control monuments and to survey some drill collars.

Traditional survey methods were subsequently used to locate all 1995–1999 and 2001 drill collars and trenches. An Ashtech Promark2 GPS post-processed system that consists of a base unit and up to two roving units was introduced in 2002.

10.2 Geological Mapping

A number of geological mapping programs were completed, using air photograph bases, at 1:20,000 and 1:10,000 regional scales (Table 10-1). Mapping was primarily completed during the mid to late 1990s.

10.3 Geophysical Surveys

An airborne magnetic survey was flown on behalf of WestGold in 1988–1989. Subsequently, ground electromagnetic, resistivity and induced polarization (IP) surveys were completed at regional and prospect scale to aid in drill targeting.

10.4 Geochemical Sampling

Geochemical sampling to support exploration-stage work programs was undertaken as summarized in Table 10-1. This work was superseded by the drill programs completed on the property.

10.5 Trenching

Trenching programs were completed as part of exploration-stage activities, and are summarized in Table 10-1. Trench data are used in the resource model, and to aid in constructing the geological model.

Table 10-1: Exploration Summary Table, Exclusive of Drilling

Year	Company	Work Performed	Results
1909 to 1956	Various prospectors and placer miners	Gold discovered on Donlin Creek in 1909. Placer mining by hand, underground, and hydraulic methods.	Total placer gold production of approximately 30,000 oz.
1970s to present	Robert Lyman and heirs	Resumed sluice mining in Donlin area and placer mined Snow Gulch.	First year of mining Snow Gulch produced best results, with 800 oz Au recovered.
1974, 1975	Resource Associates of Alaska (RAA)	Regional mineral potential evaluation for Calista Corporation. Soil grid and three bulldozer trenches dug in Snow Gulch area.	Soil, rock, and vein samples have anomalous gold values. Trench rock sample results range from 2 g/t Au to 20 g/t Au.
1984 to 1987	Calista Corporation	Minor work. Various mining company geologists including Cominco and Kennecott visit property.	
1986	Lyman Resources	Auger drilling for placer evaluation finds abundant gray, sulphide-rich clay near Quartz Gulch.	Assays of cuttings average over 7 g/t Au. Initial discovery of Far Side (Carolyn) prospect.
1987	Calista Corporation	Rock sampling of ridge tops and auger drill sampling of Far Side prospect.	Anomalous gold values from auger holes: best result = 9.7 g/t Au.
1988 to 1989	Western Gold Exploration and Mining Co. (WestGold)	Airborne geophysics, geological mapping, and soil sampling over most of Project area. Total of 13,525 m of D9 Cat trenching at all prospects. Over 15,000 soil, rock chip, and auger samples collected. Drilling included 3,106 ft of AX core drilling, 404 m in 239 auger holes, and 10,423 m of RC drilling (125 holes). First metallurgical tests and petrographic work.	Initial work identified eight prospects with encouraging geology ± Au values (Snow, Dome, Quartz, Carolyn, Queen, Upper Lewis, Lower Lewis, and Rochelieu). Drilling at most of these prospects led to identification of the Lewis areas as having the best bulk-mineable potential. Mineral Resource estimate completed.
1993	Teck Exploration Ltd.	D-9 Cat trenching (1,400 m) and two 500 m soil lines in Lewis area. Petrographic, fluid inclusion, and metallurgical work.	Identified new mineralized areas, updated Mineral Resource estimate.
1995 to 2000	Barrick (formerly Placer Dome)	87,383 m of core, 11,909 m of RC drilling and 8,493 m of trenching. Environmental monitoring and assessment.	Drilled the American Creek magnetic anomaly (ACMA), discovered the ACMA deposit. Numerous Mineral Resource estimation iterations.

Year	Company	Work Performed	Results
2001 to 2002	Donlin Creek Joint Venture	46,495 m of core, 38,022 ft of RC drilling, 89.5 m of geotechnical drilling, and 268 m of water monitoring holes.	Filed a preliminary assessment report on the Project. Updated resource estimate.
2003 to 2005	Donlin Creek Joint Venture	25,448 m of core and 5,979 m of RC drilling. Calcium carbonate exploration drilling; IP lines for facility condemnation studies.	Infill drilled throughout the resource area. Discovered a calcium carbonate resource. Poor quality IP data.
2006	Donlin Creek Joint Venture	92,804 m of core drilling to support Mineral Resource classification conversion, slope stability, metallurgy, waste rock, carbonate exploration, facilities, and port road studies.	Geological model and Mineral Resource update.
2007	Donlin Creek Joint Venture	Core drilling totalled 75,257 m and included resource delineation, geotechnical and engineering, and carbonate exploration. 13 RC holes for monitor wells and pit pump tests totalled 1,043 m.	Improved pit slope parameters, positive hydrogeological results. Carbonate exploration was negative. Updated Mineral Resource estimate. Completed feasibility study with positive results.
2008	DCLLC	108 core holes totalling 33,425 m for exploration and facility related geotechnical and condemnation studies. Updated resource models. Metallurgical test work: flotation variability and CN leach. 54 test pits and 37 auger holes were also completed for overburden characterization.	Resource expansion indicated for East ACMA. CN leach resource potential indicated for the main resource area, Snow, and Dome prospects. Facility sites condemnation drilling completed. Update of feasibility study, and updated geological models.

10.6 Drilling

Drill programs are discussed in Section 11 of this Report.

10.7 Geotechnical and Hydrology

A number of geotechnical and hydrological studies were completed in support of feasibility and environmental reports for Donlin Creek.

Rowland Engineering Consultants performed the geotechnical assessments for the geotechnical engineering for the access and interconnecting roads between the BTC port site, Crooked Creek, wind farm, airstrip, and proposed mine site.

The hydrological model is based on drill data. Lorax Environmental (Lorax) performed water quality modelling for the planned mine pit lake. CEMI provided design criteria and associated testwork for the water treatment plant requirements during construction, operations, and closure.

10.8 Petrographic and Other Studies

A number of specialist studies were performed on the Donlin Creek mineralization, including fluid inclusion studies, radiometric age dating (40Ar/39Ar), petrographic descriptions of rock types based on thin sections and electron microprobe data, whole rock analyses, trace element analyses, and sulphur, carbon, oxygen, and hydrogen stable isotope studies.

Technical papers on the geology of the region, the deposit, and on the mineralization were presented in peer-reviewed journals, and at conferences by Project personnel and personnel from the United States Geological Survey.

10.9 Exploration Potential

The Mineral Resource defined in this Report is confined to a small section of the Property. NovaGold believes there is considerable potential for additions to the Mineral Resources at Donlin Creek. Numerous other targets were identified along the five mile-long mineralized gold trend, and are defined by surface sampling and various historical drill holes containing significant gold values.

10.10 Comment on Section 10

The exploration programs completed to date are appropriate to the style of the Donlin Creek deposits. The research work supports the genetic and affinity interpretations for the deposits. Mineralization continues below the proposed ACMA pit, but expansion is limited due the proximity of Crooked Creek on the west and south, and by the process facilities to the west. Exploration potential is still open to the north.

11.0 DRILLING

Drilling on the property has been undertaken in a number of core and RC campaigns from 1988 to 2008, as summarized in Table 11-1. Approximately 1,676 exploration and development core (88%) and RC (12%) drill holes totalling 392,937 m were completed from 1988 through 2007. All but 8% (district exploration, carbonate resource, geotechnical, waste rock, condemnation, and hydrology) of this drilling was utilized for the feasibility study resource model. Approximately 50% of the core and 40% of all of the holes were drilled during the 2006-2007 period.

Of the drill total, 1,396 core (89%) and RC (11%) holes totalling 339,733 m, as well as 282 trenches totalling 21,441 m, were used to construct the Donlin Creek ACMA and Lewis area feasibility update Mineral Resource model. An additional 108 core holes totalling 33,425 m were added in 2008 to explore near-pit expansions and satellite deposits, and for facility-related condemnation and geotechnical studies. A drill location plan for the drilling to 2007 on the Project is included as Figure 11-2, and drilling to 2008 in Figure 11-3.

Drill programs have been completed primarily by contract drill crew, supervised by geological staff of the Project operator at the time. Where programs are referred to by company name, that company was the Project manager at the time of drilling, and responsible for data collection. Drill holes have typically been drilled at a inclination of -50°, to -70°, to provide optimal intercepts of the mineralized zones (Figures 11-4 and 11-5). As a result of the inclination, drilled thicknesses are typically greater than true thicknesses, but the relationship between the two is shown in the typical section.

11.1 Drilling Equipment

RC drilling was used by WestGold in 1989 for their initial exploration, by Barrick (formerly Placer Dome) in 1997 to reduce impact on wetlands areas, and by NovaGold in 2002 to conduct extensive early stage resource delineation in several areas of the deposit. Since 2002, core drills were exclusively used for all delineation drilling; RC drilling was used to inform condemnation and hydrology studies

Boart Longyear was the coring contractor from 1995 through 2008. RC down-hole hammer drilling was provided by Tester Drilling in 1989, Dateline Drilling in 1996 and 1997, and TJ Enterprises in 2002, 2004, 2005, and 2007.

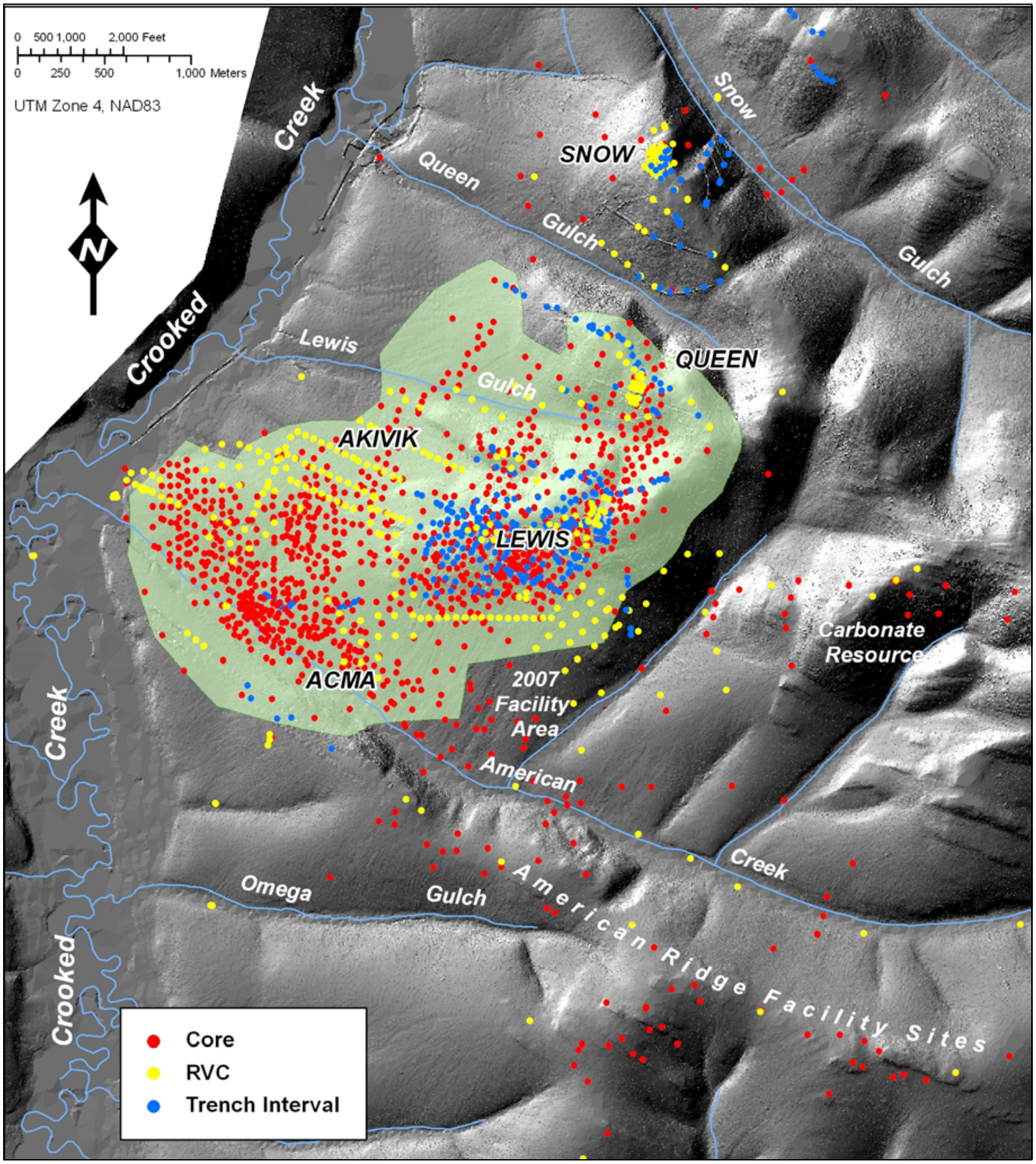
RC drill rigs were mounted on low ground pressure, self-propelled tracked carriers, and equipped with high volume air compressors, standard 10 cm dual walled pipe in 6 m lengths, and down hole pneumatic hammers with 13 cm carbide button bits.

Table 11-1: Drill Summary Table

Year	Core (m)	RC (m)	Total Holes (m)	No. of Holes	Company/Joint Venture	% RC	% Core	Total (m)	% Total (m)	Total Holes	% Total Holes
1988	947	-	947	33	West Gold	-	-	-	-	-	-
1989	-	10,423	10,423	125		92	8	11,370	3	158	9
1990	-	-	-	-		-	-	-	-	-	-
1991	-	-	-	-		-	-	-	-	-	-
1992	-	-	-	-		-	-	-	-	-	-
1993	-	-	-	-	Teck	-	-	-	-	-	-
1994	-	-	-	-		-	-	-	-	-	-
1995	6,117	-	6,117	33	Barrick (formerly Placer Dome)	-	-	-	-	-	-
1996	30,918	4,077	34,995	144		-	-	-	-	-	-
1997	15,744	8,126	23,870	118		-	-	-	-	-	-
1998	24,131	-	24,131	96		-	-	-	-	-	-
1999	9,189	-	9,189	33		-	-	-	-	-	-
2000	1,403	-	1,403	7		12	88	99,705	23	431	24
2001	24,288	-	24,288	42	Donlin Creek Joint Venture – NovaGold	-	-	-	-	-	-
2002	39,181	11,857	51,038	348		19	81	75,326	18	390	22
2003	-	-	-	-	Donlin Creek Joint Venture – Barrick (formerly Placer Dome)	-	-	-	-	-	-
2004	2,795	7,661	10,456	19		-	-	-	-	-	-
2005	24,596	-	24,596	90		22	78	35,052	8	109	6
2006	92,804	-	92,804	327	Donlin Creek Joint Venture – Barrick	-	-	-	-	-	-
2007	75,257	3,423	78,680	261		2	98	171,484	40	588	33
2008	33,425	-	33,425	108	DCLLC – Barrick/NovaGold	-	100	33,425	8	108	6
Totals	**380,795**	**45,567**	**426,362**	**1,784**							
%Total (m)	**89**	**11**	**100**								

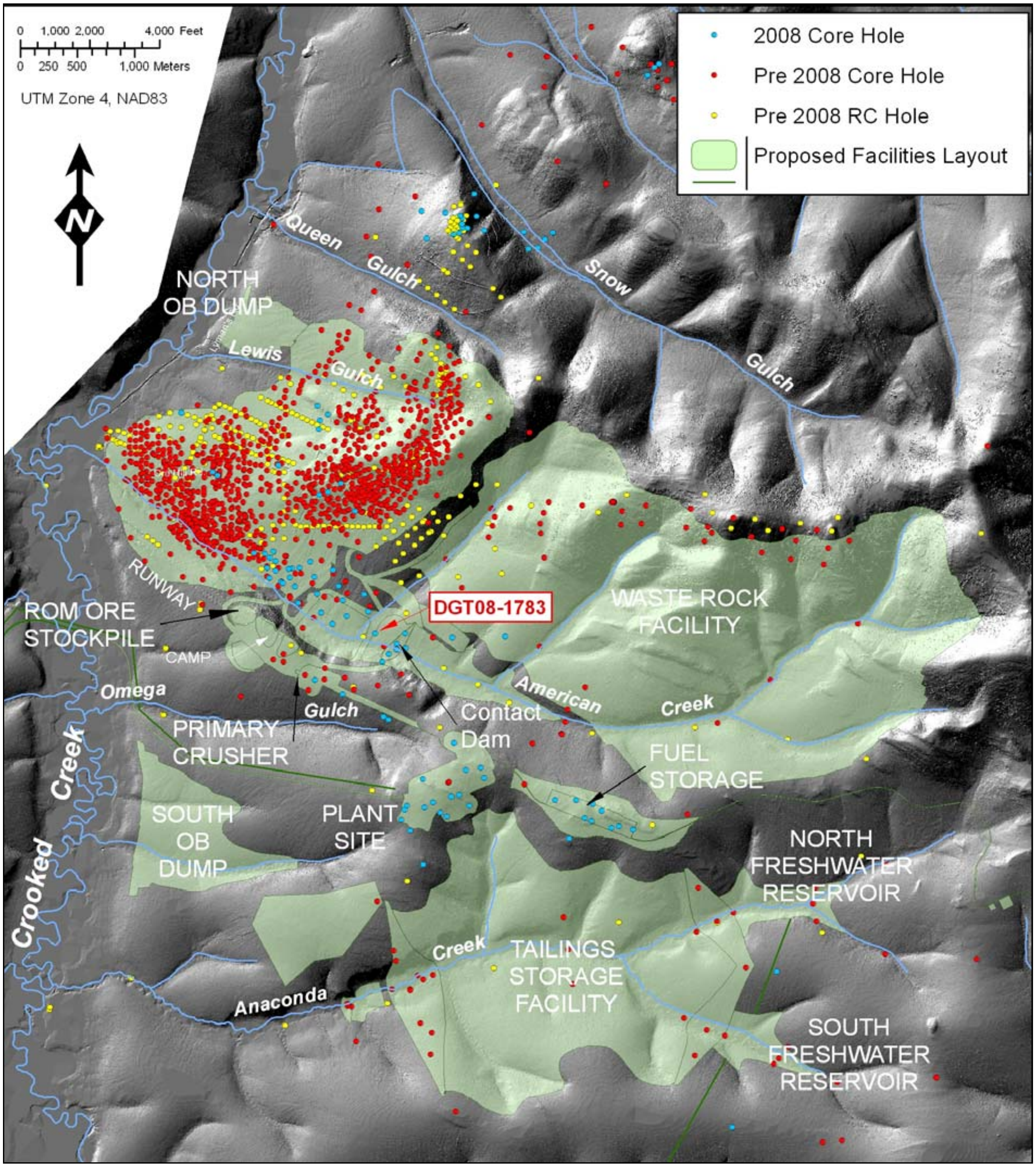
Note: * = Meterage not recorded; # = combined RC and diamond totals

Figure 11-1: Location Map, Mineral Resource Area Drill Holes 2007



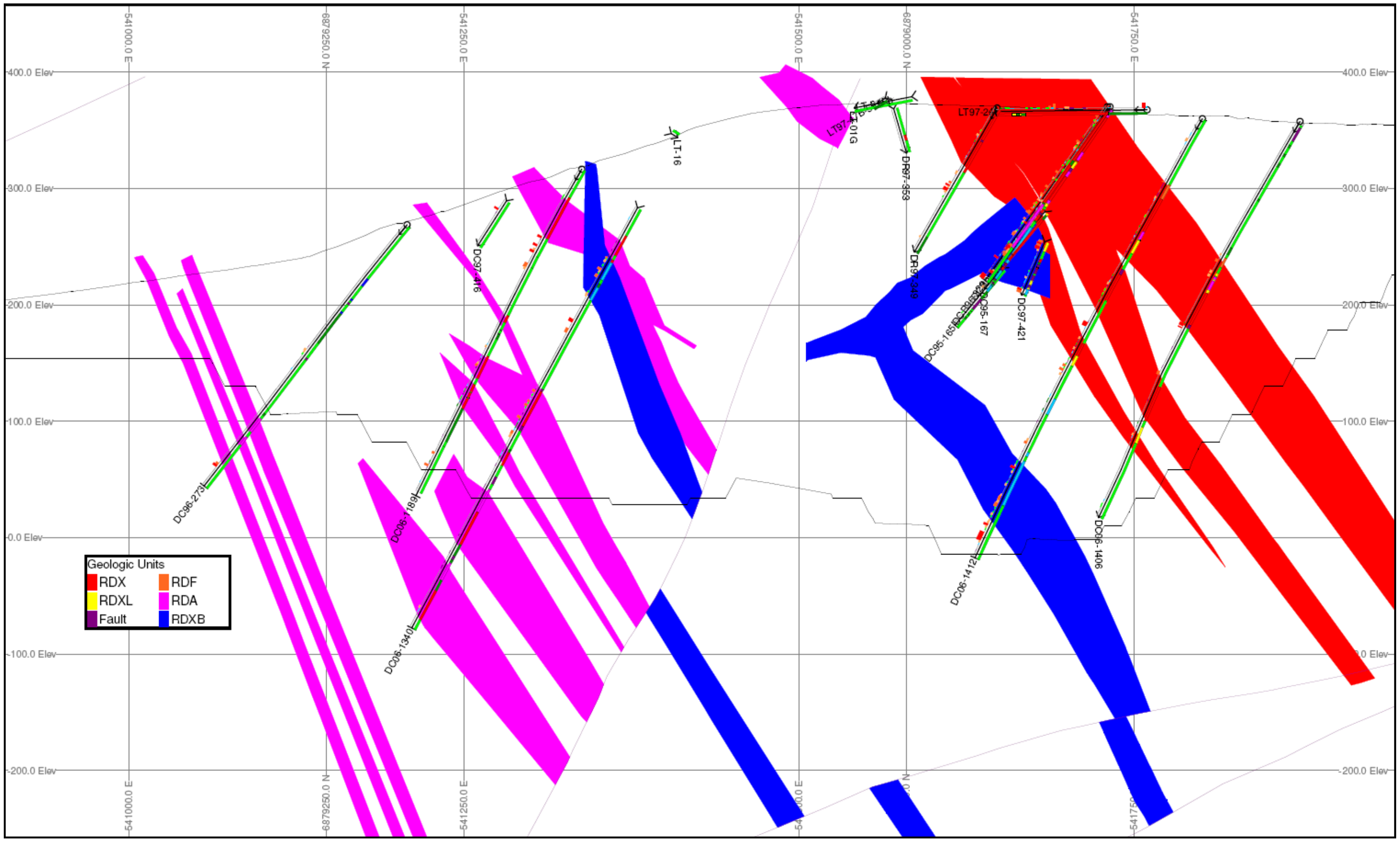
Note: Figure courtesy DCLLC. RVC = reverse circulation or RC drilling

Figure 11-2: Mineral Resource Area and Condemnation Drill Holes



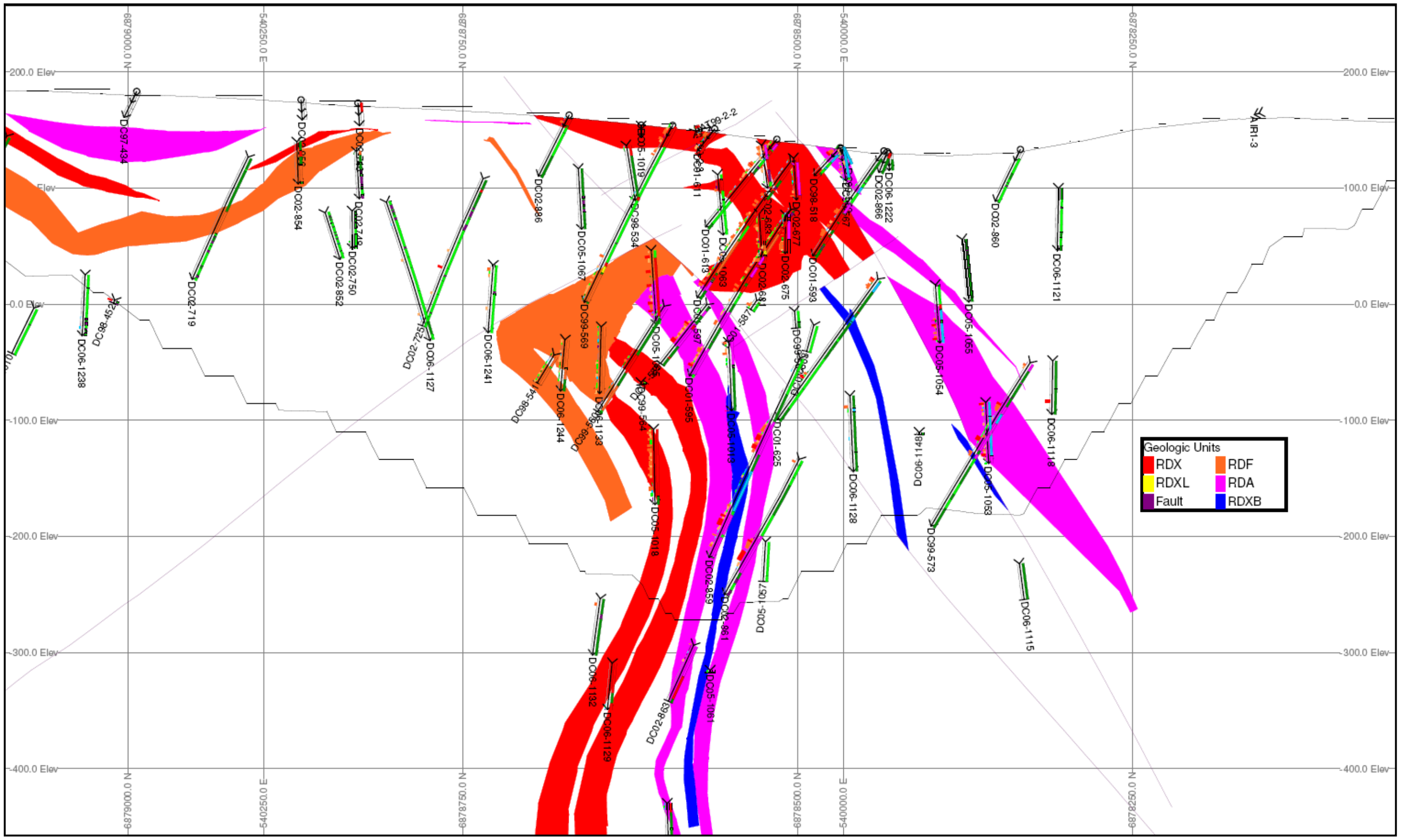
Note: Figure courtesy DCLLC.

Figure 11-3: Typical Drill Section, Lewis Area



Note: Figure courtesy Barrick. Section looks to the northeast. Grades are colour-coded down the drill hole trace such that higher-grade zones, typically co-incident with the intrusive units, are red, and areas that are lower grade or non-mineralized are green. Vertical grid scale is 100 m intervals; horizontal grid scale is 250 m intervals. Key: RDXB = Blue Porphyry; RDA = Aphanitic Porphyry; RDXL = Lath-Rich Porphyry; RDX = Crowded Porphyry; RDF = Fine-Grained Porphyry

Figure 11-4: Typical Drill Section, ACMA Area



Note: Figure courtesy Barrick. Section looks to the southeast. Grades are colour-coded down the drill hole trace such that higher-grade zones, typically co-incident with the intrusive units, are red, and areas that are lower grade or non-mineralized are green. Vertical grid scale is 100 m intervals; horizontal grid scale is 250 m intervals. Key: RDXB = Blue Porphyry; RDA = Aphanitic Porphyry; RDXL = Lath-Rich Porphyry; RDX = Crowded Porphyry; RDF = Fine-Grained Porphyry

Sample discharge and sample splitting equipment consisted of cyclone collectors mounted above Jones splitters for both wet and dry drilling in 1989 and three-tiered Jones splitters for dry samples and pneumatic rotating wet splitters for wet samples in 1996 and subsequent programs.

Core drilling was accomplished exclusively with LF-70 model drills which were set up in heli-portable configuration, mounted on skids, or mounted on self propelled tracked and low ground pressure Nodwell carriers. Standard wire-line core retrieval with 1.52 m or 3.05 m core barrels was used in all core drilling operations.

11.1.1 Drill Core Sizes

Systematic records of core size were not maintained in the database; therefore, an accurate account of HQ and NQ core cannot be easily determined. It can be stated that most of the core drilled since 1995 was HQ size, since all holes were started with HQ tools and reduced to the smaller diameter NQ size as necessary. The relative amount of NQ size core likely increased in recent campaigns as drilling probed deeper in the deposit.

Depth limits for HQ size holes were 475 m for dry conditions and 545 m for fluid-filled holes. HQ depth was generally limited to 426 m for holes with no planned reduction. Holes planned to depths greater than 475 m were reduced depending on bit changes or logistical scheduling at a depth range of 183 m to 244 m. Otherwise, depth capacity was dependent on in-hole tools, tool condition (including drill rod condition), ground conditions, drilling techniques, the variable operating capabilities of each individual drill, and crew safety.

Diamond coring bit sizes used over the duration of the drill programs at Donlin Creek are summarized in Table 11-2.

Table 11-2: Core Sizes

Bit Description	Core Diameter (mm)	Hole Diameter (mm)
NQ3	45.10	75.70
NQ	47.60	75.70
HQ3	61.20	96.00
HQ	63.50	96.00
PQ	85.00	122.60
HWT Casing Shoe	99.57	117.65

11.2 Drill Programs

11.2.1 Legacy Drilling (pre-1995)

In 1988, WestGold drilled 33 shallow (average 25 m), AX-diameter core holes with a Winkie drill and approximately 50 shallow (average 8 m) auger holes. These are not recorded in the Mineral Resource database. In 1989, WestGold drilled 125 RC holes. Identified in the database as RC-001 through RC-125, these consist of 31 holes in Far Side, 38 in Snow, 24 in Queen, eight in Rochelieu, and 24 in Lewis. WestGold terminated its lease after the 1989 field season.

11.2.2 Barrick (formerly Placer Dome) Drilling (1995–2000)

1995

Barrick (formerly Placer Dome) drilled 32 core holes, including 30 in Lewis, one at Rochelieu Ridge, and one near the mouth of Queen Gulch. DC95-162 is the first Barrick (formerly Placer Dome) drill hole. Most of the holes were surveyed down-hole using a Sperry Sun-type survey camera. These holes were all collared at HQ diameter and reduced to NQ at depth. Boart Longyear, which employed two LF-70 drill rigs, was retained as the diamond drilling contractor for all subsequent core drilling through 2008.

1996

Dateline Drilling completed 28 RC holes for Barrick (formerly Placer Dome) in 1996. Most of the holes were drilled on Lewis, and 17 of them twinned earlier core holes. Four water wells (three in camp, one in Lewis) were drilled with the RC drill, and five core holes in the 400 area were pre-collared through deep overburden. Barrick (formerly Placer Dome) drilled 116 core holes, using five LF-70 drill rigs. The database records 115 of these core holes; WW-1-96, drilled in camp, is excluded. As in 1995, most of the core was drilled at HQ diameter, and some holes were reduced to NQ at depth. All but eight of the core holes were drilled in Lewis or Queen. The others were distributed north of the current Mineral Resource estimate area in the Dome, Far Side, and Snow prospects.

1997

Dateline Drilling drilled 52 RC holes in 1997 at Lewis, Queen, Rochelieu, ACMA, 400 Area, Vortex, alongside the American Ridge runway, and Snow. Two of the RC holes in Snow are water wells. Two RC holes in ACMA tested coincident gold-in-soil and magnetic low anomalies, and led to further drilling in ACMA. Barrick (formerly Placer Dome) drilled 66 HQ core holes using five LF-70 drill rigs. Core holes were drilled at Lewis, Queen, 400 Area, ACMA, and north of the Mineral Resource estimate area at Quartz, Duqum, and Dome. The two core holes in ACMA followed up on the RC holes drilled earlier in the year. Orientation of core using the clay impression method was initiated with drill hole DC97-390. Thirty-nine core holes were oriented in 1997.

1998

Barrick (formerly Placer Dome) drilled 96 HQ core holes using three LF-70s, mainly in the Lewis, Queen, and ACMA areas. The drilling was done in two phases: four holes in the ACMA–400 area in March and April, and 41 closely-spaced holes in the Lewis area to test variography. Resource expansion drilling in the Lewis, Queen, and ACMA areas was also conducted. One hole that was collared toward ACMA from adjacent to the American Ridge airstrip confirmed that intrusive rocks do exist south of the postulated American Creek fault. Seventy holes were oriented using the clay impression method.

1999

Barrick (formerly Placer Dome) drilled 33 HQ core holes (some of which were reduced at depth to NQ), using three LF-70s. Twenty-six of these, totalling 6,690 m, were resource definition holes drilled in ACMA–400 between February and April. Later in the year, seven more holes were drilled:

- Two deep exploration holes, one in ACMA (711 m) and one in Lewis (588 m), neither of which reached the desired targets
- One hole to test a magnetic anomaly at Snow
- One hole to test a structural trend with a coincident magnetic low at Quartz
- Two holes in Nuno
- One hole in ACMA to test for an extension of a mineralized zone at a coincident east-west topographic linear and a geophysical anomaly.

2000

Barrick (formerly Placer Dome) drilled seven exploration HQ core holes, five at Dome and two at Quartz, for an evaluation of IP anomalies and testing of potential for the areas to host high-grade deposits.

11.2.3 NovaGold Drilling (2001–2002)

2001

NovaGold drilled 42 HQ core holes in 2001 to evaluate the potential for significant resource growth in the ACMA area.

2002

In 2002, T&J Enterprise drilled 146 RC holes for NovaGold, including 141 exploration and resource expansion holes in the ACMA, 400, Lewis, Akivik, Rochelieu, Vortex, and Far East prospects. Most holes were drilled north at 60° to 70° to optimize intersection of both sills and mineralized corridors. This effort resulted in discoveries in the Akivik and Aurora areas. Three water wells were drilled near the mouth of American Creek, and two were drilled in the Low Road on the south face of Lewis. NovaGold also completed 196 HQ core holes with four LF-70 diamond drill rigs. Some of these were reduced to NQ at depth, and several oriented holes were drilled using the clay impression method. Two of the core holes are geotechnical holes in the Anaconda Creek valley.

11.2.4 Donlin Creek Joint Venture Drilling (2003–present)

2003

Before the 2003 field season, Barrick (formerly Placer Dome) exercised its option to resume operation of the Project (as the Donlin Creek Joint Venture) and started a program to drill 16 RC water monitoring wells, using drill contractor T&J Enterprise. No core holes were drilled in 2003.

2004

Barrick (formerly Placer Dome) contracted T&J Enterprise (for the Donlin Creek Joint Venture) to complete the water monitoring well program started in 2003 and to drill 17 RC condemnation holes in the Anaconda Creek and upper American Creek valleys. Three HQ geotechnical core holes designed by Bruce Geotechnical Consultants were also completed.

2005

The Donlin Creek Joint Venture (Barrick formerly Placer Dome as manager) drilled 90 core holes totalling 24,596 m and 30 RC holes totalling 3,644 m. Three drill rigs were used for core drilling and focused primarily on infill in ACMA and Lewis. Most holes were drilled to the northwest at an inclination of -50° to -60°. Core holes ranged in length from 79 m to 544 m, and RC holes ranged in length from 102 m to 201 m.

2006

The Donlin Creek Joint Venture (Barrick as manager) drilled 92,804 m of core in 327 core holes with eight LF-70 drill rigs. The drilling was primarily carried out to convert mineralization classified as Inferred resources to Measured and Indicated resource classifications. However, significant drilling was also devoted to a broad range of pre-feasibility and feasibility objectives, including pit slope stability, metallurgy, waste rock studies, facilities condemnation and engineering, and calcium carbonate resource bulk sampling, delineation, and exploration.

2007

The 2007 Donlin Creek Joint Venture (Barrick as manager) drill program utilized six LF-70 rigs and included 75,257 m in 248 HQ/NQ core holes for pit resource infill, pit expansion, carbonate exploration, geotechnical, and engineering studies. Four core drills completed an aggressive geotechnical drilling program during the first quarter of the year. This work included 50 holes totalling 3,702 m to further evaluate pit slope stability parameters and engineering characteristics of facility sites. One limestone prospect near the BTC port site was tested with six HQ core holes totalling 720 m. Subsequently, two core holes totalling 122 m were drilled in the port site and five (155 m) were completed in the wind farm area for engineering purposes. Thirteen RC holes totalling 1,043 m were completed for monitor wells and pit pump tests. Numerous auger holes were also completed to evaluate overburden conditions.

2008

Four LF-70 rigs completed 108 holes totalling 33,425 m for exploration, resource infill, condemnation, and geotechnical studies. The three main objectives of the program were to investigate possible East ACMA pit expansion in adjacent 2007-proposed facility site areas, condemn the 2008 relocated facility sites, and conduct geotechnical investigations of the new and relocated facility sites. The condemnation drilling in the 2008 relocated facility sites did not identify near surface mineralized material or any favourable geologic environments within 500 m of the surface.

AMEC reviewed the results of the 2008 drilling and noted the following:

- Drill holes completed in 2008 within the current pit generally agreed with the existing model, and it is unlikely that the drill holes will have a significant impact of the Project
- Drill holes completed in 2008 support the previous favourable drill results located approximately 1,000 m northeast of the Lewis pit
- Four 2008 drill holes were drilled in the Dome area that showed anomalous gold mineralization
- Drill holes completed in 2008 for geotechnical and condemnation purposes intersected minor mineralization that had no economic potential.

Drill holes undertaken in the 2008 drill program that extended southeast from the southeast portion of the proposed ACMA open pit may contain significant mineralized intercepts. This mineralization, however, is deep, and appears to be localized near the edge of the ACMA pit. Work is currently in progress by the DCLLC to determine if firstly, this mineralization can support expansion of the planned open pits, and secondly, if the expansion is warranted when compared to the projected economics of moving the proposed facilities farther away from the open pits.

11.3 Geological Logging

11.3.1 Barrick (formerly Placer Dome) Drill Programs

Barrick (formerly Placer Dome) established standard logging and sampling conventions and codes for the Project. Drill core and RC chips were logged using paper forms, which captured lithological, alteration, mineralization, structural and geotechnical information.

Alteration data were collected during 2005–2006 using an analytical spectral device (ASD) shortwave infrared (SWIR) spectrometer (see discussion in Section 11.4.2). Virtually all core holes from the ACMA and Akivik areas, as well as a significant portion from south Lewis, were SWIR-analyzed during the 2002 field season.

11.3.2 NovaGold Drill Programs

Standard logging and sampling conventions were used to capture information from the drill core and, where applicable, RC chips. The core was logged in detail using paper forms during 2002 with the resulting data entered into the main database (Access® database) either by the logging geologist or a technician. Post 2002, data were directly entered into a laptop computer, and then downloaded on a daily basis into the Access® database.

Five types of data were captured in separate tables: lithology, mineralization, alteration (visual), structural, and geotechnical. Remarks were also captured. Lithology was recorded using a two to four letter alpha code. The mineral table captured visual percent veining (by type) and sulphide (pyrite, arsenopyrite, stibnite and realgar). Specific alteration features including FeOx and carbonate alteration were also captured using a qualitative scale. Structural data comprised type of structure, measurements relative to core axis and oriented core measurements, if applicable. The geotechnical table recorded percent recovery and rock quality designation (RQD) for the entire hole, and fracture intensity where warranted. The protocols and coding were similar to those used by Barrick.

During 2001, a preliminary study of alteration variability was undertaken on hand samples using a PIMA™ SWIR. Based on those results, a PIMA™ device was again utilized in early 2002 to ascertain alteration mineralogy in relation to detailed logging observations, as well as assay and geochemical results. That study successfully demonstrated that SWIR spectrometry was efficient in defining alteration assemblages controlling the distribution of gold grade.

A more serviceable, high throughput ASD SWIR spectrometer was subsequently used in 2002 in order to collect alteration data for the entire Donlin Creek area. Virtually all core holes within the ACMA, Aurora, 400 and Akivik areas, including core from previous drill campaigns, were analyzed using the spectrometer during the 2002 field season. A significant portion of drill core from the South Lewis and Vortex areas was also completed.

11.3.3 Donlin Creek Joint Venture Drill Programs

Standard logging conventions adopted by Barrick (formerly Placer Dome) were used to capture geologic data from both core and RC chips. The chips were logged on paper forms and the data entered into an electronic database. Core logging data were captured in five tables: lithology, mineralization, alteration (visual), structural, and geotechnical (percent recovery and RQD. Specific gravity measurements for representative samples were systematically collected at regular intervals through each core hole.

11.4 Collar Surveys

Drill hole collar and trench locations were tied to a surveyed ground control net using conventional theodolite survey methods from 1988–1993. Drill hole collars were surveyed with Brunton compass and hip chain in 1995. A Motorola® GPS system was used in early 1996 to establish survey control monuments and to survey some drill collars.

Traditional survey methods were subsequently used to locate all 1995–1999 and 2001 drill collars and trenches. An Ashtech Promark2® GPS post-processed system that consists of a base unit and up to two roving units was introduced in 2002. The roving Promark2® was used in the field to collect stationary readings over the drill collars. Data collected by the roving unit and base units were downloaded and post-processed through Ashtech Solutions Software®. The resulting processed drill collar survey data and vector information was checked for accuracy and quality control then copied to an Excel® survey data file. This in turn was copied to the acQuire® data base for archival. Based on Ashtech® surveys of control points, the approximate maximum horizontal and vertical variance of drill hole collar surveys under optimal conditions was 0.2 m and 0.6 m, respectively.

11.5 Down-hole Surveys

Directional surveys to determine down-hole deviation utilized the Sperry Sun® single shot camera method prior to 2000. Reflex EZ Shot® instrumentation was introduced in 2001. Six parameters: azimuth, inclination, magnetic tool face angle, gravity roll angle, magnetic field strength, and temperature were measured. Measurements were generally collected at 50 m intervals from 6 m off bottom of the hole to within 30 m of the surface. An integrated key pad and LCD display provide for manual operation and data retrieval. Handwritten data was delivered to the geologists with the shift reports for quality control and manual entry into the database.

11.6 Oriented Drill Core

Extensive oriented core was drilled to collect structural information for geotechnical and geological studies. Core orientation methods included clay impression, EZ Mark®, and Reflex ACT® instrumentation. The clay impression method was used through 2005, but proved problematic as average hole depth increased. Clay impression was replaced with EZ Mark® and Reflex ACT® tools in 2006. The Reflex ACT® tool was used almost exclusively for exploration and resource delineation holes while the EZ Mark® method was used as a backup and for some geotechnical holes. Oriented core required use of HQ3 and NQ3 bits to accommodate thin-walled inner tubes that reduced core rotation and fragmentation. These bits also produced a smaller diameter core.

11.7 Core Handling

Core was drilled using split double and triple tubes. Wherever possible, the inner tube was pushed out by water pressure, with minimal shaking and banging of the barrel. Core trays were placed near the core barrel so that the core was put in the tray in the same orientation (top–bottom) as it came out of the barrel. Rubble was piled to about the length of the whole core that its volume would represent.

Any break in the core made during removal from the barrel was marked with an "X". When breakage of the core was required to fill the box, edged tools and accurate measure of pieces to complete the channels was the common practice to minimize core destruction. The end of every run is marked with a wooden tick and the final depth of the run.

Core was transferred to wax-impregnated corrugated core boxes, marked with "up" and "down" signs on the edges of the boxes using indelible pen. The drill hole number, box number and starting depth for the box was written before its use, whilst end depth were recorded upon completion. All information was marked with indelible pen on the front side of the box and also on the cover.

Transport of core boxes to the core shed was done by personnel from the company that was managing the drill program, or the drilling supervisor. Core handling logs were completed that included details for all persons involved in any step during the logging and sampling procedures.

11.8 Recovery

Core recovery in the intrusive units, both where mineralized and unmineralized, was excellent, usually ranging from percentages in the mid 90s to 100%. Recovery in the shale dominant sediments was more variable, ranging from percentages in the 80s to percentages in the high 90s.

11.9 Comment on Section 11

The quantity and quality of the lithological, geotechnical, collar and down-hole survey data collected in the exploration and delineation RC and core drill programs are sufficient to support Mineral Resource and Mineral Reserve estimation:

- Approximately 68% of the core metreage and 58% of the core holes were drilled during 2006–2008
- Drill intersections, due to the orientation of the drill holes, are typically greater than the true width of the mineralization. An illustration of the relationship between drilled thickness and thickness of mineralization is shown in Figures 11-3 and 11-4.
- Orientation of the mineralization is outlined in the figures included in Section 7 and Section 9
- Core logging meets industry standards for gold exploration
- Collar surveys were performed using industry-standard instrumentation
- Down-hole surveys accurately represent the trajectories of the holes.

12.0 SAMPLING METHOD AND APPROACH

As the geochemical and trench analyses were superseded by the amount of drill data available, exploration-stage analytical data are not discussed further in this Report. Core and RC sampling were performed by company personnel from the company that managing the Project at the time of the drill programs.

12.1 RC Sampling

RC samples were continuously collected during drilling at 1.5 m intervals using a standard rotary sample splitter. An approximate 25% split of the total drilled material was collected for analysis.

12.2 Core Sampling

Holes are sampled from the top of bedrock to the end of the hole. Overburden, excluding the organic layer, may also be sampled if abnormally thick and composed of abundant rock clasts.

Core sample intervals are based on rock type, rock type breaks, and presence of visible sulphide/arsenic minerals. The maximum sample length in zones consisting of intrusive rocks, or that contain appreciable sulphide/arsenic minerals, is 2 m. Sedimentary rock zones that lack appreciable sulphide/arsenic minerals may have sample lengths of 3 m. A minimum of three additional 2 m samples are collected before and after each intrusive rock or mineralized zone.

An aluminium tag inscribed with the sample number is stapled to the core box with a same-numbered paper tag at each sample break. A sampling cutting list is generated that also specifies the insertion points for control samples.

The core is then digitally photographed and split in half with an electric rock saw that uses water-cooled diamond saw blades. Core cutters orient the core in the saw to ensure a representative split. One-half of the core is returned to the core box for storage at site, and the other half is bagged for sample processing.

12.3 Comment on Section 12

A description of the geology and mineralization of the deposits, which includes rock types, geological controls and widths of mineralized zones is given in Section 7 and Section 9.

A description of the sampling methods, location, type, nature, and spacing of samples is included in Section 10 and Section 12.

A description of the drilling programs, including sampling and recovery factors, are included in Section 11 and Section 12. No significant sample bias factors were identified with the programs that could affect Mineral Resource or Mineral Reserve estimation.

Figure 7-2 shows the drill hole collar locations for drilling that supports Mineral Resource and Mineral Reserve estimation, and indicate that the size of the sampled area is representative of the distribution and orientation of the mineralization. Sampling density is appropriate for the planned large-scale open pit mine.

Examples of relevant sample composites with sample values and estimated drill intercept widths were included in typical sections for deposits in Section 11 (Figures 11-2 and 11-3). These sections display typical drill hole orientations for the deposits, show composite values using colour ranges for composite intervals that include areas of non-mineralized and mineralized material. The sections confirm that sampling is representative of the gold grades in the deposits, reflecting areas of higher and lower grades.

Data validation of the drilling and sampling program is discussed in Section 14, and includes review of database audit results.

Sampling methods are acceptable, meet industry-standard practice, and are adequate for Mineral Resource and Mineral Reserve estimation and mine planning purposes, based on the following:

- Data are collected following mine site-approved sampling protocols;
- Sampling was performed in accordance with industry standard practices;
- Sample intervals of 1.5 m for RC drilling, and maximum lengths of 2–3 m for core drilling, broken at lithological and mineralization changes in the core, are typical of sample intervals used in the industry, are typical for the mineralization styles, and are considered to be adequately representative of the true thicknesses of mineralization. Not all drill material is sampled depending on location and alteration.

13.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

13.1 Sample Preparation

The camp sample preparation procedure consists of the following steps:

- The entire bagged sample is dried in an oven heated to between 85°C and 95°C for 12 hours.
- The sample is put into trays for processing through a jaw crusher. The sample tag stays with the sample.
- Blank samples are inserted into the sample stream.
- The sample is crushed until the end product passes 70% minus 2 mm (10 mesh). Sieve analyses are performed periodically to check crush quality, and the crusher jaws are adjusted as necessary. Blank material is periodically used to clean the crushers, and operators are alerted by the geologists to increase the cleaning frequency when unusually sulphide-rich material is processed.
- Crushed sample is then passed through a riffle splitter four to six times to obtain a nominal 250 g split. This sub-sample is put into a numbered pulp bag, and the remainder, or coarse reject, is put back into the original sample bag. The splitter and sample pans are cleaned with compressed air.
- Two additional control samples—standard reference material (SRM) and a duplicate split of crushed sample—are inserted as specified on the cutting list prepared by the geologist. Two of each control sample type including SRM, duplicates, and blanks are included in every batch of 78. The blank is prepared by processing a sample from a bin of gravel-size crushed rock by passing it through the jaw crusher and riffle-splitting it to 200 g. When a duplicate is required, the crushed core sample is passed through the riffle splitter once, and each half is split repeatedly to obtain a 200 g sample.

In December 2006 and January 2007, about 12,000 m of whole core was shipped to an off-site logging and core splitting facility in Anchorage, Alaska. This facility was managed by Alaska Earth Science (AES) and staffed with both AES and Barrick personnel to ensure that logging, sampling, core splitting and sample shipment procedures were identical to those used at the Donlin site facility.

The majority of core samples taken between 2005 and 2008 were crushed at the Donlin camp sample preparation facility and pulverized at the ALS Chemex Vancouver laboratory facility. Samples of 2006 core split in Anchorage were shipped to an ALS Chemex preparation laboratory for crushing and pulverizing.

13.2 Sample Transport

13.2.1 Donlin Creek Facility to Barrick (formerly Placer Dome) Laboratory

Core transportation procedures used to submit samples to the Barrick (formerly Placer Dome) internal laboratory are similar to those described in the next subsection for ALS Chemex.

13.2.2 Donlin Creek Facility to ALS Chemex Laboratory

Core samples are transported from the field and are brought to the yard adjacent to the geology office and logging tents at the end of each drill shift. Core storage is secure because Donlin is a remote camp and access is strictly controlled.

Unauthorized camp personnel have generally been excluded from the core cutting and sample preparation building, but strict access procedures were initiated following an audit in mid-2006.

Assay splits of prepared core, along with the control samples, are packed in a shipping bag, secured with a numbered security seal, and sealed in boxes for shipment. The coarse rejects and remaining split core are returned to a storage yard south of the airstrip for long-term storage.

The sample shipment procedure is as follows:

- Boxed assay splits are flown from the Donlin camp to Aniak airport via Vanderpool Flying Service.
- Samples are shipped from Aniak via Frontier Flying Service to the ALS Chemex laboratory facility in Fairbanks, Alaska. All sample shipments are accompanied by a Frontier Flying Service waybill. This allows each sample to be tracked from camp to ALS Chemex.
- The samples are logged into the ALS Chemex data system in Fairbanks before shipment to the ALS Chemex Vancouver (or other ALS Chemex facility), where they are pulverized and assayed. The Fairbanks laboratory returns a custody form that reports on the condition of security seals.

13.2.3 Anchorage Facility to ALS Laboratory

The Anchorage logging and splitting facility was housed in a secure, dedicated, warehouse/office facility. Visitor access to the facility was strictly controlled by AES, the facility manager. Outside visitation for tours or purposes other than daily delivery or pick-up required advance approval by the Donlin Project Manager.

Whole core shipped from camp to the facility was transported on Lynden Air Cargo. Lynden waybills and Barrick custody forms were used to track samples from camp to Lynden's Anchorage airport facility and from there by Lynden trucks to the Anchorage logging facility.

Split core samples shipped from camp to the ALS Chemex Fairbanks laboratory followed similar protocol. Bagged split core samples were tied into shipping bags and loaded into palletized supersacks. Supersacks were closed with numbered security seals and shipped on Lynden trucks to ALS Chemex in Fairbanks. Waybills aided tracking within the Lynden transport system, and ALS Chemex reported on the condition of security seals in the same manner as shipments from the Donlin Creek facility.

13.3 Sample Analysis

13.3.1 Barrick (formerly Placer Dome)

Most of the samples from Barrick (formerly Placer Dome)'s work were processed in Barrick (formerly Placer Dome)'s own laboratory.

Samples were pulverized into a pulp (to better than 90% minus 150 mesh, or 100 µm), a 25 g sub-sample was taken from the pulp, fire assayed and analyzed using an atomic absorption spectrometry (AAS) finish. Samples that assayed 3 g/t Au or more were re-assayed by fire assay pre-concentration with a gravimetric finish. During the 2005 drill campaign, samples that assayed 10 g/t Au or more were re-assayed by an "ore grade" AAS technique.

Barrick (formerly Placer Dome) created four in-house control standard reference materials or standards. Two were used consistently throughout Barrick (formerly Placer Dome) and NovaGold's work: Geological Gold Standard C and Geological Gold Standard D. These standards were made according to an accepted methodology of homogenization and round-robin assaying. The certification process was supervised by Barrick (formerly Placer Dome)'s assay team. One or both standards were inserted in all batches, depending on the range of expected values. During the 2005 drill program, the two Barrick (formerly Placer Dome) standards were exhausted, necessitating the purchase of additional standard material. Three standards were acquired: two from Analytical Solutions Ltd (Oreas 6Pb and Oreas 7Pb) and one from CDN Resource Laboratories Ltd (CDN-GS-3). These standards were also made according to an accepted methodology of homogenization and round-robin assaying.

Blanks were used to check for the presence of contamination in both sample preparation and assaying. Barrick (formerly Placer Dome) collected a large container of crushed, unmineralized, diorite-like material for use as the blank material.

13.3.2 NovaGold

Samples from the NovaGold programs were initially submitted to Bondar Clegg laboratory in Vancouver (now ALS Chemex). The samples for the NovaGold 2001 work were pulverized into a pulp (to better than 90% minus 150 mesh, or 100 µm) and analyzed by a 1-assay ton method, wherein a 29 g sub-sample was taken from the pulp sample, fire assayed and analyzed using an AAS finish. Samples that assayed 10 g/t Au or more were re-assayed by fire assay with a gravimetric finish.

Standard reference materials used during NovaGold programs were those established by Barrick (formerly Placer Dome). NovaGold also utilized the diorite-like material for blank samples.

13.3.3 Donlin Creek Joint Venture

Final sample preparation and chemical analysis for gold, sulphur and trace element suites on the drill samples from the Donlin Creek Joint Venture drill programs that were managed by Barrick were completed at ALS Chemex in Vancouver. The preparation consists of the following:

- The splits of crushed core were reduced to rock flour or “pulp” in a ring-and-puck grinding mill (to better than 85% passing minus 75 µm or 200 mesh).
- A 30 g sub-sample of the pulp was fire assayed primarily using a fire assay-atomic absorption spectroscopy (AAS) method. The primary gold-assay method used prior to 2007 was Au-AA23. This method had an analytical range of 0.005 g/t Au to 10 g/t Au. The Au-AA25 gold-assay method was used in 2007, and it had an analytical range of 0.01 g/t Au to 100 g/t Au. This switch was made to reduce the cost and time delay associated with re-assaying samples with values above the 10 g/t Au analytical limit.
- Samples that assayed >10 g/t Au were re-assayed from 2005 through 2006 by an “ore grade” fire assay-AAS method (Au-AA25) with a detection range of 0.01 g/t Au to 100 g/t Au. Sulphur in each sample was determined at ALS Chemex by the Leco method. Samples flagged for acid base accounting (ABA) also received carbon analyses by the Leco method as well as determination of neutralization potential (NP) and acid potential (AP) according to the industry standard Chemex ABA procedure.

- In 2007, one sample exceeding 100 g/t Au was assayed via gravimetric method Au-GRA21 with a detection range of 0.05 g/t Au to 1,000 g/t Au.

Nine new “matrix-matched” SRMs of varying gold grades were added in early 2007 and the older standards were eventually phased out. The new SRMs were created from coarse reject samples located throughout the deposit. Composites of this material were pulverized and homogenized at CDN Laboratory in Vancouver, BC.

Washed river gravel produced by Anchorage Sand and Gravel was used for blanks through early 2006 and then replaced by granite landscape chips purchased from Lowe’s in Anchorage.

The major proportion of trace and major element data for drill holes located within the resource model boundary was acquired prior to the 2005 program by various laboratories using industry standard acid digestions followed by atomic absorption (AA) or inductively coupled plasma (ICP) instrumental determinations.

13.4 Density/Specific Gravity

Earlier NI 43-101 Technical Reports have used two specific gravity (SG) values, 2.65 for the mineralized intrusive units, and 2.71 for the mineralized sedimentary rocks, based on 1,190 measurements and 700 measurements respectively. The density determination method for the density values collected by Barrick (formerly Placer Dome) is not known.

During 2006, additional specific gravity data were collected to provide better coverage of deposit rock units and geographic sub-regions, using the following methodology:

- Samples of whole core approximately 5 cm to 10 cm in length are first weighed dry and then weighed in water. The dry weighing tray assembly is replaced with a wire basket and the sample is submerged in a five-gallon bucket of water. A small tare weight (to compensate for the removed weighing tray) is attached midway up the wire assembly to facilitate alternating wet and dry measurements
- The formula for SG calculation is: Weight in Air/(Weight in Air – Weight in Water). The specific gravities are automatically computed in acQuire® when the weights are entered into the database.
- Measurements are collected for all rock types at a minimum frequency of one sample from all logged rock type intervals and one sample every 15 m to 20 m in the longer rock unit intervals. Mineralized rock takes precedence over unmineralized rock in a given rock type interval, but sufficient measurements of unmineralized material are also collected to document potential variability.

Statistical evaluations of the 2006 data showed that the SG values were similar to the historical intrusive rock and sedimentary rock SG values (Table 13-1). Therefore, the historic values for sediment-hosted and intrusive-hosted mineralization were used to support Mineral Resource estimation.

Table 13-1: Average Specific Gravity Values by Rock Type

Rock Types and Domain	No. of Samples (#)	Specific Gravity
Argillite	272	2.67
Conglomerate	9	2.71
Fault Zone	25	2.75
Greywacke	2,368	2.71
Mafic Dyke	473	2.73
Monzodiorite	2	2.70
Rhyodacite Aphanitic Porphyry	499	2.64
Rhyodacite Fine Grained Porphyry	315	2.67
Rhyodacite Coarse Grained Porphyry	1,339	2.66
Rhyodacite Coarse Grained Blue Porphyry	520	2.63
Rhyodacite Lath Rich Porphyry	216	2.64
Siltstone	838	2.72
Shale	387	2.70
Average All Rock Types	**7,370**	**2.69**

13.5 Quality Assurance and Quality Control

13.5.1 Pre-2005

Barrick (formerly Placer Dome) commenced quality assurance and quality control (QA/QC) programs during the 1995 drilling campaign. Coarse reject duplicate splits from 10% of the drill hole samples were submitted to Bondar Clegg. Standard reference material assay standards and blanks were added in 1996 and an ALS Chemex performed check assays, presumably of coarse reject duplicates. Check assays by a secondary assay laboratory were apparently discontinued after 1996.

A more structured assay QA/QC program, consisting of SRMs, blanks, and duplicates inserted in rotation every 15 m downhole, was initiated in 1997. This protocol evolved to random and blind insertion of SRMs, blanks, and coarse reject duplicates through the 2002 NovaGold program.

From 1996 to 2002, SRMs and coarse-reject duplicates were inserted at an average rate of one per 24 samples and blanks were inserted at an average rate of one per 25 samples. Almost all samples associated with SRM and blank control-samples that returned values beyond acceptable tolerance limits were re-assayed until the control sample results were either acceptable or validated by duplication.

13.5.2 2005–2006

Barrick (formerly Placer Dome) implemented a slightly modified QA/QC protocol in 2005, which was continued by the Donlin Creek Joint Venture (Barrick as manager) in 2006. Three QA/QC samples, one blank, one coarse reject duplicate, and one SRM, were randomly inserted into every block of 20 sample numbers.

Thus, in every block of 20 sample numbers there were 17 drill hole samples and three QA/QC control samples.

13.5.3 2007 to Current

The sample batch-size submitted to the ALS Chemex was increased from 20 to 78 samples in 2007. ALS Chemex had a fusion-batch size of 84 samples. The laboratory adds six internal control samples, leaving space for 78 client samples in a batch. This batch size avoids possible mixing during the fusion process of samples from the Project with samples from other ALS Chemex clients.

Each batch of 78 samples shipped to ALS Chemex for sample preparation and analysis contained nine control samples (12%) consisting of three each of standards, blanks, and crushed duplicates. As much as 5% field duplicates (remaining half-split of core) were added to the sample batch at the discretion of the geologists.

13.6 Databases

Data collected by Barrick (formerly Placer Dome) prior to 2001 was compiled into a MS Access® database. This database was subsequently updated by NovaGold into a later version of Access®.

ioDigital converted the NovaGold Access® database to an MS SQL Server® database in early 2005 using an acQuire Technology Solutions data model (acQuire®). Data obtained after the conversion were imported directly into the acQuire® database.

During 2005–2006 drill data was captured using acQuire® software and stored in MS SQL Server®. Geological logs, collar, and down-hole survey data were entered at the Donlin camp using acQuire® data entry objects. Assay data were imported directly from electronic files provided by the laboratories.

The master Donlin database was moved from the Donlin camp to the Anchorage office mid-year 2006. Assay data were imported directly into the master database in Anchorage for the remainder of 2006 and all of 2007. The acQuire® database was converted from the standard acQuire® data model to the more robust acQuire® "Corp" data model in early 2007.

Geological and sample data were entered into the Donlin Camp database, and merged into the master database several times per week.

13.7 Sample Security

Sample security was not generally practiced at Donlin Creek during the drilling programs, due to the remote nature of the site. Sample security relied upon the fact that the samples were always attended or locked in the on-site sample preparation facility. Sample collection, preparation, and transportation have always been undertaken by company personnel using company vehicles. Chain of custody procedures consisted of filling out sample submittal forms that were sent to the laboratory with sample shipments to make certain that all samples were received by the laboratory.

Transport and security procedures from the sample preparation facilities to the laboratory are discussed in Section 13.2.

Paper records are kept for all assay and QA/QC data, geological logging and bulk density information, down-hole and collar coordinate surveys. All paper records are filed by drill hole for quick location and retrieval of any information desired. Assays, down-hole surveys, and collar surveys are stored in the same file as the geological logging information. In addition, sample preparation and laboratory assay protocols from the laboratories are monitored and kept on file.

Digital data are regularly backed up in compliance with internal company control procedures.

13.8 Comment on Section 13

Sample preparation for core and RC samples has followed similar procedures throughout the Project exploration history.

Preparation and analytical procedures are in line with industry-standard methods, and suitable for the deposit styles.

A QA/QC program comprising blank, standard and duplicate samples was used on the Project since the mid-1990s. QA/QC submission rates meet industry-accepted standards of insertion rates.

The specific gravity determination procedure is consistent with industry-standard procedures. There are sufficient specific gravity determinations to support the specific gravity values utilized in waste and mineralization tonnage interpolations.

Data that were collected prior to the introduction of digital logging were subject to validation.

Sample security has relied upon the fact that the samples were always attended or locked in the on-site sample preparation facility. Chain-of-custody procedures consist of filling out sample submittal forms that are sent to the laboratory with sample shipments to make certain that all samples are received by the laboratory.

Current sample storage procedures and storage areas are consistent with industry standards.

14.0 DATA VERIFICATION

14.1 Database Verification

14.1.1 2002–2003

The work completed by Barrick (formerly Placer Dome) prior to 2001 was collected and compiled into a main MS Access® database. Upon finalization of a joint venture agreement with Barrick (formerly Placer Dome), NovaGold became operator of the property. NovaGold compiled the Barrick (formerly Placer Dome) Access® database into an updated Access® database and added the data from work completed in 2001 and 2002.

AMEC (2002a, 2002b, 2003) verified the Access® database to test data integrity. AMEC initially conducted a 5% check of randomly chosen drill holes in each of the ACMA and Lewis regions and checked gold values against the original electronic assay certificates. No errors were uncovered.

AMEC checked the down-hole survey data, by comparing camera shots from the check drill holes to those stored in the resource database. A significant transcription error rate was found in all regions. NovaGold, therefore, instituted a 100% check of the camera shot readings. AMEC re-checked the survey data after this work was completed and found no errors.

Collar co-ordinates were checked against the database entries. AMEC checked three randomly chosen drill collars with a GPS unit. Readings obtained matched those entered in the database. AMEC concluded that the assay and survey database was sufficiently free of error to be adequate for Mineral Resource estimation.

14.1.2 2003–2005

ioDigital converted the Access database to an MS SQL Server® database in early 2005 using an acQuire® Technology Solutions data model (acQuire®). Data obtained after the conversion were imported directly using the acQuire® software interface.

The 2005 geologic data and down-hole survey information were cross-checked with the paper logs by a Barrick (formerly Placer Dome) geologist at the end of the season, and errors were corrected when found. NovaGold conducted a 100% check of 2005 drill hole gold assays within the resource area against electronic assay certificates. An error rate of less than 1.5% was found. NovaGold also checked all 2005 collar and down-hole survey data. Electronic down-hole survey files were read for the drill holes and compared to those stored in the resource database. NovaGold determined that the integrity of the 2005 data was sufficiently free of error for resource estimation (NovaGold, 2006).

14.1.3 2006–2007

Drill data were captured using acQuire® software and stored in MS SQL Server®. Geologic logs, collar, and down-hole survey data were entered at the Donlin camp using acQuire® data entry objects. Assay data were imported directly from electronic files provided by the laboratories.

The master Donlin database was moved from the Donlin camp to the Anchorage office mid-year 2006. Assay data were imported directly into the master database in Anchorage for the remainder of 2006 and all of 2007. Geologic and sample data were entered into the Donlin Camp acQuire® database and merged into the master database several times per week.

The acQuire® database was converted from the standard acQuire® data model to the more robust acQuire "Corp"® data model in early 2007. Legacy data were verified further when they were migrated to the new data model.

Geology, collar, and down-hole survey data used in the 2006 and 2007 resource models were visually verified by comparing hard-copy logs to the acQuire® database. Approximately 90% of post-2005 drill hole data were cross-checked and verified. Verification of remaining geological and assay data is ongoing.

An independent audit by Resource Modelling Incorporated checked 5% of 2006 drill hole data and verified that the data were suitable for use in Mineral Resource estimation.

NovaGold verified the 2007 drill data and confirmed that these data were sufficiently free or error to be used for resource estimation. Five percent of the geologic logs and collar surveys, 10% of the down-hole surveys, and 100% of the assay data were checked against the original records. The 2007 assay data had a 1% error rate. The errors detected for all data types were corrected in the master acQuire® database.

14.1.4 2008

AMEC primarily reviewed drilling completed between 2005 and 2007. Checks included:

- Accuracy of the topographic wireframe and the drill hole coordinates. Drill collars were noted to be not on the topographic wireframe; variances were skewed with more collars above topography than below with an average difference of 1.36 m. Overall the differences between topography and database collar elevations are considered to be within acceptable ranges and are therefore considered suitable for supporting the Mineral Resource estimate.

- The topographic wireframe was also used to calculate the percentage of a model block that lies above the topographic surface which was used to adjust block tonnages. AMEC checked the block tonnage corrections for topography in the model and found it to be correctly calculated.
- Differences between hardcopy and database entries for down-hole survey data are considered to be within acceptable ranges and considered suitable for use in the Mineral Resource estimate.
- Specific gravity data were imported and viewed in relation to the pit design. The geometry of the SG data covers the entire deposit, but is more concentrated in the Lewis than the ACMA area. SG values were very similar between the intrusive rocks, and between the sedimentary rocks. There is little variability within the SG data, and a single density of 2.65 for all the intrusive units, and a single SG of 2.71 for all the sedimentary units is appropriate.
- The differences between hardcopy and database entries for lithology data are considered to be within acceptable ranges and are therefore considered suitable for use in the Mineral Resource estimate.
- AMEC checked a total of 12,309 samples (17%) for Au and 11,916 samples (16%) of the S assays. The assay and geological databases are within acceptable error rates.
- AMEC's review of the coarse blanks used at Donlin Creek between 2005 and 2007 indicate evidence of a consistent but low-level contamination. The amount of contamination, however, is low enough that it does not have material impact on Mineral Resource estimation.
- Although results show that some SRMs showed better accuracy than others, the only significant biases observed was for Std-SH. Since Std-SH is the SRM for shale, and shale is a relative small portion of the overall deposit (approximately 4% of the gold ounces), AMEC concludes that the accuracy of the ALS Chemex gold assays is adequate and suitable for supporting Mineral Resource estimation.

14.2 Core versus RC Drilling Comparison

Core and RC holes were compared in 1996 when 17 core holes in the Lewis area were twinned with RC holes. This study found that in most instances, composite assay intervals from the RC holes were thinner, less continuous, and lower grade than in the twinned core holes (Szumigala, 1997).

14.3 Drill Hole Orientation

Drill hole orientation relative to the contrasting Lewis dyke and ACMA sill orientations, combined with the primary north–northeast structural control of gold distribution, was investigated by Barrick (formerly Placer Dome) in 1998 and Barrick in 2006.

Barrick (formerly Placer Dome) conducted variography testing in the North and South Lewis areas. Fourteen core holes oriented approximately normal to the dykes and the north–northeast mineralized zones (295° azimuth, -50° dip) and located on a grid spacing of approximately 35 m showed excellent correlation with both the geological and mineralization models. Twenty-six core holes were also drilled in South Lewis in an area of west–northwest-striking, southwest-dipping sills. Nineteen of the 26 variography holes were oriented to optimize drilling across the north–northeast mineralized veins (280° azimuth/-50° dip), and seven were oriented specifically to test sill contacts (50° azimuth /-50° dip). All holes were drilled at approximately 35 m spacing. Results of the variography testing in this area showed some variation with the models, although the overall correlation was good (Baker, 1999).

The Donlin Creek Joint Venture (Jutras, 2006) further investigated the possibility of a gold grade bias in the then resource model. Five major drill hole orientations totalling 1,298 holes were observed: north (340° to 20° azimuth, 220 holes), northeast (20° to 70° azimuth, 195 holes), southwest (200° to 260° azimuth, 176 holes), northwest (265° to 335° azimuth, 656 holes), and vertical (dip of -90°, 51 holes). No significant grade bias was identified for northeast (sub-parallel to north–northeast mineral zones) holes relative to the other orientations, particularly the predominant northwesterly orientation. A standard northwesterly drill hole orientation “normal” to the north–northeast structural control (300° azimuth, -60° northwesterly dip) was adopted for all resource delineation programs from 2005 through 2007.

14.4 Comment on Section 14

The data verification programs undertaken on the data collected from the Project support the geological interpretations, and the analytical and database quality, and therefore data can support Mineral Resource estimation.

15.0 ADJACENT PROPERTIES

There are no adjacent properties that are relevant to the Project that is the subject of this Report.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 Metallurgical Testwork

Metallurgical testwork was performed in two major campaigns, the first by Barrick (formerly Placer Dome) from 1995 to 2005, and the second by the Donlin Creek Joint Venture post-2006. Testwork was completed at a number of laboratories and research facilities, including the independent SGS-Lakefield Research (SGS Lakefield), SGS Minerals Research (SGS), Hazen Research (Hazen), G&T Metallurgical Services (G&T), Dynatec and Polysius Corporation facilities, and the Barrick-operated Placer Dome Technical Services laboratory and Barrick Technology Centre.

16.1.1 Comminution

During 2002–2003, Hazen performed grind tests on material from ACMA and Lewis, which indicated the material was moderately hard.

During 2004, Lakefield tested two large Donlin Creek samples with the objective of comparing the power efficiency of using high pressure grinding rolls (HPGR) as opposed to semi-autogenous grinding to prepare the mineralization prior to ball milling. Both samples were characterized as medium in terms of Bond rod mill (RWI) and ball mill work indices (BWI), and were only mildly abrasive. However, the sedimentary sample was described as moderately hard with respect to resistance to impact, and with respect to resistance to abrasion breakage. The intrusive sample measured hard in terms of low-energy impact and medium in the drop-weight test, and hard with respect to resistance to abrasion breakage. The two samples were also submitted to a series of bench-scale HPGR tests. A reduction in the BWI was attributable to the HPGR processing.

Mineralization samples were submitted to Polysius Corporation (an equipment supplier for high pressure grinding rolls) in May 2005 for pilot-scale testing. Results were used to help select and size process equipment, and to forecast operating costs.

During 2006, SGS Lakefield conducted an extensive test program on existing 1999 and 2002 drill campaign HQ-size drill core to determine grinding parameters for the Donlin Creek mineralization, and included determination of the Minnovex SAG power index (SPI), crusher index (Ci), and modified Bond ball mill work index (Modified Bond test), SMC drop-weight index test (DWI), Bond low-energy impact (CWI), RWI, BWI, abrasion index (Ai) and high-pressure grinding roll energy test.

Samples from 2006 drill campaign PQ core holes were processed to develop JK grinding parameters in addition to conventional Bond ball mill and rod mill work index numbers to check the data obtained on the HQ core samples. Freshly-drilled PQ core was found to be consistently harder than the HQ core samples drilled in 1999 and 2001, and as a result all additional testwork was completed on fresh (recently drilled) core.

Using 2006 drill campaign core, variability testing was completed in 2007 at SGS Lakefield. The test program was designed to maximise generation of hardness properties relating to the semi-autogenous milling with ball milling and pebble crushing (SABC) circuit required parameters, and evaluated Minnovex SAG power index, Minnovex crusher index (Ci), Minnovex modified Bond ball mill work index, RWI and BWI. The program indicated consistently harder comminution properties than the testwork on the 1999–2001 core. To preserve the 2006 test results to provide an increase in the quantity of test samples for variance analysis and subsequent population of the geological model, the 2006 variability results were adjusted by Minnovex to match the hardness distribution of the later (harder) test results. Geostatistical evaluation of the Ci, SPI, and BWI data provided a block-by-block mill feed hardness schedule for the parameters that then, using a comminution economic evaluation tool developed by Minnovex, could be used to predict the milling capacity and power requirements for each ore block in the designed Donlin Creek circuit. In parallel with this testwork, grinding circuit design trade-off studies were performed.

A number of additional BWI tests at different final product sizes were undertaken by SGS Lakefield. As the target product size increased in fineness, the measured BWI of the test sample increased. On the blended pilot-plant sample a significant increase in BWI occurred between a P_{80} of 54 µm and 42 µm.

Barrick contracted Orway Mineral Consultants (OMC) and SGS Lakefield (Minnovex) in 2006 to perform appraisals of four comminution options: autogenous milling with ball milling and crushing of pebble reject (ABC); SABC; coarse crushing followed by HPGR with ball milling; and fine crushing followed by ball milling, and provide capital and operating costs.

The SABC circuit was selected as the comminution circuit for Donlin Creek, for several reasons:

- Lowest capital cost
- Ability to cope with the clay fraction in the ore
- Ability to cope with the climatic conditions
- General ease of operation and maintenance
- Flexibility in throughput rates
- Widely applied technology in the milling industry
- Barrick's extensive experience in SABC circuit application.

A mill–chemical–float–mill–chemical–float (MCF2) flowsheet incorporating two separate stages of grinding and flotation was used for Project design, using SABC as the primary grinding step. The sizing of grinding equipment and definition of the circuit capacities were derived over a number of study steps and iterations of those steps.

16.1.2 Flotation

Bench-scale

Bench-scale flotation testwork during 1995–2007 indicated that producing a bulk concentrate was the optimum route to maximize gold recovery. A number of different reagent schemes were tested; however, a simple flotation utilizing plant acid, copper sulphate, and xanthate was found to be appropriate. Nitrogen-based flotation technology was tested extensively, but no benefit to using nitrogen that would justify the extra costs and risks associated with this technology was identified. To achieve maximum recovery from the arsenopyrite, some of which is tied up with larger gangue particles, a long flotation retention time, of approximately 114 minutes, was found to be necessary.

Particle grind sizes were based on two different mineralization sizes: mineralization hosted in intrusive rocks in the range of 75 µm to 110 µm, and mineralization hosted in sedimentary rocks in the range of 60 µm to 80 µm. Testwork consistently showed that a fairly high mass pull in the range of 20%+, depending on mineralization type, was necessary for high recovery. With the use of dispersants, lower flotation feed pulp densities (approximately 30% solids content or lower), and cleaning, the overall mass pull to final concentrate was found to be able to be decreased to approximately 15%. Froth recovery was identified as a critical factor, and should be enhanced by the use of crowding cones and through launder design in the full scale plant flotation machines.

Testwork was performed to assess the outcome of blending the two mineralization types. Given adequate reagent dosages and residence times, it proved possible to produce high flotation recoveries with the life-of-mine test blends provided for the testwork.

Pilot-scale

During 2004, pilot-plant flotation test work by G&T was focused on producing a concentrate for pressure oxidation and other downstream work. Results are not useful for evaluation of expected flotation performance.

In mid-2006, testing of conventional–air flotation techniques at G&T confirmed the performance of the flotation circuit and its operating characteristics under continuous test conditions. The gold recoveries achieved from the pilot test program, to a target 7% sulphur grade concentrate, were lower than expected. Testwork subsequently carried out to investigate the results, found that the problem was largely insufficient froth recovery.

A subsequent bench and pilot flotation program in late 2006, completed by Lakefield, showed that a recovery of 91% to 92% on a LOM lithology blend was possible.

In early 2007, a series of bench flotation tests were initiated to explore the potential benefit of the MCF2 circuit configuration. The first stage of grind/flotation would enable recovery of the fast-floating liberated sulphide particles at a coarser grind size to enhance their flotation kinetics. The second stage of grind/flotation would reduce the primary rougher flotation tails down to a fine grind size to liberate the composite sulphide particles and then float these particles to concentrate, with an overall high final gold recovery. An initial series of tests were conducted by Lakefield at a nominal grind of P_{80} 40 µm for the second stage product, and with varying primary stage grind sizes. The alternate MCF2 grind/flotation configuration realized a measurable improvement in gold recovery of approximately 2% at the same final concentrate grade, as compared to the conventional grind/float arrangement.

A second SGS Lakefield pilot-plant campaign was undertaken in 2007, with the primary aim to confirm pilot-plant recovery of the conventional flotation circuit on a LOM lithological blend of freshly-drilled core composite. It was planned to test the use of a mild steel primary mill instead of stainless steel, the scavenger concentrate regrind option with a convention flotation circuit configuration, and also the full MCF2 grind/flotation configuration, to confirm the overall flowsheet for feasibility study purposes. As a result of the work, the MCF2 option was further evaluated as a potential base case for the Donlin Creek feasibility study grinding/flotation circuit design.

A financial comparison of the two circuits, conventional and MCF2, was undertaken, which indicated the preferred option was MCF2. Subsequent variability testwork was initiated using MCF2 as the basis of the flotation test procedure. Two types of variability bench-scale flotation tests were carried out, the first being a modified Minnovex Flotation Test (MFT), including repeats using adjusted froth scraping rates and air addition rates. The second type of test was a conventional bench flotation test (CFT), where cell air rates, froth scraping rates, and froth depth were adjusted as required. Changes to scraping rate, air flow rate and change to a CFT approach, all provided means to reduce froth mass recovery rate, with the net result of improving gold recovery to a given sulphur grade concentrate, due to less capture of gangue or entrainment to the concentrate.

A flotation variability testwork program was completed at SGS in 2008 on drill core material collected from the exploration drilling program conducted following the feasibility study. The purpose of the testwork was to confirm the flotation response and recovery of materials contained within the pit shell that were not previously tested, as well as confirming flotation recovery variabilities within the various defined geological domains. The results of the testing program were inconclusive, with additional work planned for the detailed design phase.

Mineralogy

Mineralogical assessments were undertaken on flotation products from various testwork programs during 2006–2007. Gold was found to exclusively occur as sub-micron particles disseminated in the crystal structure of arsenopyrite and pyrite, with arsenopyrite the main gold host. However, pyrite hosted a significant portion of the gold (as solid-solution gold within the crystal matrix), and therefore must also be recovered to concentrate.

The upper portion (10% of the total mineralization tonnage) of the Donlin Creek deposit contains mineralization that has some sign of geological oxidation or weathering associated with it. To quantify the potential impact of this oxidation on flotation, MFT and CFT bench-scale tests were performed. The presence of some form of geological oxidation extent was found to significantly affect the flotation performance. The average gold recovery was 72%, with a relatively high standard deviation of 22% recovery.

A series of bench flotation tests were initiated mid-2007 by Lakefield to attempt to improve the floatability of the oxidized samples employing commonly-used reagents for this purpose. However no obvious improvement in result was identified. It is recommended that further work be continued into the next study phase.

Design Modelling

Two flotation simulators were tested for Donlin Creek, JKSimFloat, and FLEET (Flotation Economic Evaluation Tool). Two JKSimFloat campaigns were carried out during 2006. The first campaign, at G&T, reviewed fast, medium, slow and non-floating floatability components to model the different mineralization types. The second campaign, at Lakefield, was carried out on a more optimum reagent scheme and circuit configuration along with more detailed measurements such as bubble size and superficial gas velocity in every pilot cell.

Various simulations were carried out by Barrick using different throughputs, circuit configuration, bank residence times, cell numbers and sizes, cell operating and hydrodynamic parameters. Economic evaluation was carried out for these scenarios to identify the most cost-effective circuit design for Donlin Creek.

The final flotation simulation predictions were based on parameters developed for the LOM composite sample tested at Lakefield. Thus, the model parameters and predictions may not be applicable to all possible combinations of mineralization blends that may be fed to the proposed Donlin Creek processing plant.

16.1.3 Pressure Oxidation

Pressure oxidation (POX) in gold processing generally refers to the oxidation of gold bearing sulphide minerals to metal sulphates, using a combination of heat (typically 200°C to 230°C), acid and oxygen sparging, in a specifically-designed pressure vessel. The breakdown of the sulphide particles effectively releases the gold locked within the mineral matrix, rendering it amenable to cyanidation.

Dynatec

During 2004, Dynatec carried out bench autoclave testing of four composite samples, named ACMA Sediment (sedimentary rocks from the ACMA deposit), ACMA Intrusive (intrusive rocks from the ACMA deposit), Lewis Sediment (sedimentary rocks from the Lewis deposit) and Lewis Intrusive (intrusive rocks from the Lewis deposit). Tests included kinetic and locked-cycle mass balance pressure oxidation tests on the concentrates, followed by neutralization tests on the pressure oxidation discharge liquors, and carbon-in-leach (CIL) cyanidation tests:

- Concentrates were relatively fine, with P_{80s} of 33 to 41 µm (82% to 89% minus 44 µm), and were tested without further size reduction.
- Direct CIL cyanide leaching of the unoxidized feeds yielded gold extractions between 3% (ACMA Sediment) and 11% (Lewis Intrusive).
- Gold extractions were highest from the solids oxidized at 220°C, and as a result, all subsequent pressure oxidation testwork on the four concentrates was conducted at 220°C.
- Sulphide sulphur oxidation kinetics was rapid, with more than 98% oxidation achieved within 30 minutes. Gold extractions from the oxidized concentrates were

correspondingly high after 30 min pressure oxidation and improved marginally to their maximum values of between 95.1% (ACMA Sediment) and 98.5% (ACMA Intrusive) after 45 min oxidation. With extended pressure oxidation time, however, the gold extractions declined, most markedly for the sediment concentrates, which had relatively high organic carbon content.

- A retention time of 45 min was selected for the pressure oxidation in the subsequent material balance and locked-cycle testwork, where flotation tailings / lime-neutralized acid discharge solution of a previous oxidation cycle was used to re-pulp fresh concentrate feed to the pressure oxidation of the next cycle.
- In the locked-cycle testwork, the pressure oxidation tests were conducted at pulp densities approaching those anticipated of the discharge slurries in commercial autoclave operation. The Lewis Sediment concentrate because of its considerably higher alkaline metal content (4.8% CO_2), required some pre-acidification to enable the pressure oxidation to proceed. The extents of sulphide sulphur oxidation, with the 45 min pressure oxidation retention time at 220°C exceeded 98%, with more than half over 99%.
- Stirred-tank CIL cyanide leach gold extractions varied from 90.3% to 98.8% for the four oxidized concentrates, with median extractions of 93.5% to 97.4%.
- Subsequent testing has indicated that oxidation rates and performance of the batch tests are significantly affected by the decision to pre-acidify the concentrate sample charge, or not, both at pilot scale and bench scale.

Batch Autoclave Testing

A series of batch autoclave tests was conducted at the Barrick Technology Centre during 2006 to understand the potential impact of autoclave temperature, oxidation time, pre-acidification, oxygen concentration, and thiocyanate concentration on autoclave performance:

- Gold recoveries of ~89% were reached in the first 15 minutes compared to 70% to 90% recovery achieved in the first 15 minutes for the Dynatec batch tests. Changes in recovery were attributed to the pre-acidification step used in the 2006 testwork.
- Autoclave temperatures of more than 192°C were required to achieve >92% gold recovery within a nominal 1 hour autoclave residence time, and 220°C to 230°C provide maximum recovery values.
- Some preliminary tests were undertaken to evaluate the potential impact of thiocyanate (SCN) dosed into the feed slurry on the CIL gold recovery of the autoclave products. No detrimental impact on recovery was indicated.

- Experiments were undertaken to test the potential application of higher-temperature POX. Gold recovery improves with extended autoclave time and the improvement is accelerated at 240°C.
- The oxidation rate (as indicated by gold recovery) is strongly affected by gaseous oxygen concentration.

During 2007, the Barrick Technology Centre performed bench autoclave testwork on a sub-sample of the concentrate generated from the SGS Lakefield December 2006 pilot flotation test program. The test program aimed to investigate the pressure oxidation characteristics of the new composite concentrate sample against previous testwork, and to investigate the effect of autoclave on pressure oxidation performance. Results showed that as the autoclave temperature decreased, the gold recovery improvement rate also decreased. The best result was at 220°C with 45 min residence time.

Pilot-Plant Testwork

Four phases of autoclaving pilot tests were carried out during the latter half of 2006 at the Barrick Technology Centre. Following the pilot flotation campaigns in 2006 at both Hazen and G&T, the various concentrates generated were combined into a blend composite to provide sufficient concentrate sample for pilot-plant operation at a nominal sulphide sulphur grade of 7%.

- A series of 220°C and one 225°C pilot campaigns were carried out. This temperature range represents the base case for the feasibility design. The three tests undertaken that generated final CIL gold recoveries of less than 95% were due to oxidation rate performance issues. A final run was attempted at 225°C, incorporating pre-acidification of the concentrate feed, with an extended residence time. The purpose was to determine under pilot conditions what recovery could be achieved at more complete oxidation levels. This run successfully demonstrated that the CIL gold recoveries of the autoclave residue in excess of 96% were possible.
- A pilot run at 240°C resulted in high gold recoveries being quickly achieved, albeit with faster oxidation kinetics than at 220°C. There is an improvement in recovery evident, but with an insufficient improvement rate that would required to achieve high gold recoveries under practical design constraints.

During February 2007, an additional autoclave pilot campaign was undertaken at the Barrick Technology Centre. This pilot run attempted to explore the three different operating temperatures 200°C, 210°C and 220°C. Based upon previous successes with pre-acidification in 2006, it was decided to utilize pre-acidification of the concentrate for each of these planned runs. The selected residence times for the operation of the unit was, however, purposely set to be higher than design, due to the desire to achieve more fully oxidized conditions at the discharge (than achieved in previous campaigns), and to provide a set of data covering a large range of operating residence times. Results were:

- CIL gold recovery achieved from the pilot autoclave is sensitive to the autoclave residence time, where recovery is lower than optimum, when autoclave residence time is too short, and also recovery is lower than optimum, if autoclave residence time is too long.
- Autoclave operating temperatures of 220°C and 225°C provided the highest gold recoveries with the lowest autoclave residence times, with optimum residence time of around 40 min to 55 min.

Because the target gold recoveries were demonstrated only on autoclave compartmental samples, not actual pilot autoclave discharge samples, it was decided to continue the pilot autoclave testing program. The program was completed June 2007. The aim of the 2007 Phase 2 testwork was to operate the pilot unit more closely to the design optimum autoclave residence time to demonstrate that target CIL gold recoveries could be achieved on final products from the autoclave, confirm potential CIL gold recoveries, and confirm the selected design criteria for the pressure oxidation circuit for the feasibility study. In addition, the test program provided additional autoclave profile data for oxidation rates and gold recoveries:

- Product CIL gold recoveries of 96.6% can be readily achieved, and at optimum operating conditions, recoveries of 97% were possible (as indicated by the tests undertaken on the discharge samples – not simply autoclave profile samples).
- CIL gold recovery achieved from the pilot autoclave is sensitive to the autoclave residence time. Gold recovery is slightly lower than optimum, when autoclave residence time is too short, due to incomplete sulphide sulphur oxidation. Also recovery can be lower than optimum, if autoclave residence time is too long, i.e., oxidation extent too excessive.
- Autoclave operating temperatures of 220°C and 225°C provided good results with optimum (for CIL gold recovery) residence times of 45 min to 49 min, based upon autoclave discharge samples.
- Measurable sulphide sulphur oxidation is essentially completed by 37 min to 42 min residence time, as indicated by analysis of the autoclave profile samples.
- The selected hot curing time of 6 hours (as per the feasibility design) is shown to provide good lime consumption results and CIL gold recovery performance, but

dissolution of arsenic is evident, subsequently requiring precipitation in the following neutralization stage.

Mercury Gas Emission Testing

Mercury gas emission testing was performed in 2006 and 2007. Very little mercury emissions were present in the combined gas streams. However, based upon the difficulty in obtaining a consistent gas mass flow measurement from the pilot autoclave unit and challenges associated with analysis of mercury at such low levels, it is recommended that consideration be given to the incorporation of a mercury abatement system for treatment of the autoclave off-gases as a back-up.

16.1.4 Neutralization

During 2004 Dynatec tested the neutralization properties of the acidic liquors generated from bench autoclave tests, using flotation tails, limestone and lime. Results were not favourable from a lime consumption perspective, with insufficient iron precipitation.

The Donlin Creek Joint Venture investigated the neutralization capacities of calcareous sandstone and flotation tails in 2006. The carbonate utilization of flotation tails and the lower grade calcareous sandstone material was found to be low. However, it was subsequently identified that the reaction kinetics for neutralization with flotation tails and calcareous sandstone material was unusually slow, and that simply increasing neutralization residence time improved carbonate utilization.

A batch neutralization testwork program was initiated in mid-2006. Results indicated that a flotation tails/lime neutralization option, with extended neutralization residence time was the most economic (lowest total cost).

A pilot neutralization testwork program was subsequently undertaken, and provided significant reductions to the lime consumption for neutralization through increasing flotation tails neutralization residence time.

A bench testing program was initiated during early 2007 to investigate the potential benefit of increasing the slurry temperature of the flotation tails stream, and to improve understanding of how lime consumption varied as the carbonate content of the flotation tails changes. A pilot-scale program followed, and results were used to determine the optimum residence time for the industrial scale neutralization circuit for Donlin Creek feasibility design. A total of five hours residence time was marginally the lowest total net present value (NPV) cost design. However given the insignificant difference between the cases considered, the selection of the largest tank was recommended, to provide additional residence time to improve the ability of the circuit to handle variation in feed rates and flotation tails carbonate contents, and to provide additional time for the operations team to respond to an unplanned shutdown of the flotation circuit.

A second pilot test phase was undertaken on the neutralization performance of the MCF2 pilot flotation tails. In addition to a six hour residence time run, a second three hour residence time campaign was also undertaken, to investigate the potential to reduce the size of the circuit for the detailed design phase. The results of the three hour test campaign were encouraging, suggesting the potential to further decrease the size of the neutralization circuit. However, the larger six hour residence time circuit was retained as the recommended design.

During July to September 2007, a variability neutralization testwork program was initiated at SGS Lakefield. The program had the dual aims to confirm the potential differences in neutralization performance of the varying lithologies, and to develop a confident relationship between lime consumption for final pH trim to 7, and the carbonate grade of the feed samples.

Prediction of lime consumption for acidic liquor neutralization for the feasibility study operating cost estimate was based upon the relationship between lime demand and flotation feed carbonate grade developed from this test data. A minimum addition rate of 0.1 kg/t lime (as CaO) was recommended for establishment of the plant operating costs, regardless of the anticipated mill feed carbonate grade, and despite test results indicating the potential for lower additions, to ensure adequate lime availability onsite to account for process variability.

16.1.5 Carbon-in-Leach (CIL)

Extensive cyanidation testing was undertaken on various samples of Donlin Creek, at various points in the flowsheet, since 1995.

Cyanidation (with or without the presence of activated carbon) on unoxidized Donlin Creek ores consistently yields very low gold recoveries (5% to 30%) either as flotation feed, flotation tails or concentrate. This is characteristic of mineralization where gold is predominantly associated with arsenopyrite or pyrite in solid solution form, such as at Donlin Creek.

The bulk of the cyanidation tests carried out to date, were largely undertaken on autoclave compartmental and discharge samples, where large numbers of relatively small samples are leached with high concentrations of carbon and cyanide, as a diagnostic tool, to enable establishment of the performance of various autoclave tests, without the added complication of the addition constraints that may be imposed by attempting to optimise leaching kinetics.

CIL gold recovery has shown in general to be more sensitive to the operating conditions of the autoclave, as opposed to the operation conditions and methods applied in the CIL circuit. The target of the metallurgical design of the CIL circuit is instead to ensure that good CIL recovery performance is achieved on the material presented to it from the autoclave, with optimum reagent (lime and cyanide) usage. Key aspects from the testwork include:

- Leach recovery is improved with the undertaking of cyanidation in the presence of activated carbon, due to the content of natural organic carbon in the ore, which has the capability to preg-rob gold that was leached from solution.
- Through MetSim modeling and metallurgical testing, it was shown to be favourable to operate the Donlin Creek CIL circuit at a relatively low pH of 9. To achieve the traditional CIL circuit pH levels of 10–11 would require complete precipitation of the magnesium in the feed solution.
- Assuming a CIL pH of ~9, lime addition is estimated to be in the order of 5 kg/t to 7 kg/t of concentrate, and cyanide addition in the order of 0.7 kg/t to 0.9 kg/t. The key component affecting both lime and cyanide consumption is the washing efficiency achieved through the autoclave product counter-current decant (CCD) wash circuit.
- Air/SO_2 cyanide detoxification is assumed for pre-treatment of the CIL tailings, prior to being transferred to the neutralization circuit for disposal.

16.1.6 Thickening and Counter-Current Decantation

Thickening test programs on Donlin Creek materials were conducted by representatives of Outotec (formerly Outokumpu Technology) and Dorr-Oliver Eimco. Five separate stages of thickening and counter-current decantation wash are envisaged in the proposed process flowsheet:

- Concentrate thickening after flotation
- CCD washing of pre-acidified concentrate with fresh water to provide optimal oxidation conditions
- CCD washing of hot cured autoclave product slurry with process water to reduce lime consumption ahead of CIL cyanide leaching
- Clarification of the portion of hot cure CCD overflow not reporting to pre-acidification to recover entrained gold values

- Thickening of flotation tailing prior to neutralization, to minimize dilution during neutralization and reclaim of process water.

Outotec representatives carried out thickening tests on samples of concentrate, pre-acidified concentrate, flotation tailing, and neutralization residue using a 94 mm diameter bench-scale high-rate thickener. Dorr-Oliver Eimco representatives performed thickening tests on Donlin Creek samples using a "Continuous Deep Fill Tube" test apparatus and Bohlin Visco 88 viscometer.

Current design criteria and reagent usage are derived from Outotec data due to more comprehensive testing and reporting available at the time of selection. However, results from Dorr-Oliver Eimco testwork indicate some potential for improved performance/reduced costs. Results of recent tests with MCF2 products and equilibrium process water indicate that compression type ("high compression" or "paste") rather than high rate thickeners will be required for acidified concentrate (counter-current decantation CCD wash), autoclave residue CCD wash and flotation tailing thickening. Dorr-Oliver Eimco test data address issues and requirements for compression thickening to meet key design criteria.

The Outotec thickening tests completed on flotation concentrates (prior to acidification) indicated satisfactory performance using flocculant MF 351. Thickening tests conducted on acidified flotation concentrates showed these apparent results:

- Under dynamic conditions, anionic (MF 455) flocculant performs significantly better than non-ionic MF 351.
- Under non-optimized (one dynamic test) conditions, flocculant dosage requirement is significantly higher (250 g/t) than indicated by earlier testwork.

Thickening tests undertaken on a limited quantity of autoclave products indicated in 2006 some difficulty in satisfactory flocculation of autoclave discharge material. Outotec noted a favourable response to two-stage flocculant addition. Additional testing of autoclave product was completed in 2007 by Outotec and Dorr-Oliver Eimco, test data for thickening hot-cured autoclave product supports the use of compression-type thickeners to achieve the design underflow density.

Scoping tests (cylinder settling) were performed to investigate clarification of gold-bearing overflow solids by addition of MF 10 flocculant. As a result, a clarifier was used in the feasibility study flowsheet pending further testwork.

Thickening tests were undertaken on flotation tailing for Donlin Creek, based on a finer grind from the use of an MCF2 flowsheet in the concentrator. Dorr-Oliver Eimco test data indicate a requirement for approximately two hour of compressive settling to achieve the design value of 55% solids using 40 g/t of flocculant.

A review of the value of a final tailings thickener to the Project was undertaken. The final tailings thickener could serve the purpose of de-watering the combined tailings stream (CIL tails, neutralized flotation tails and diluted autoclave acidic liquor), prior to pumping to the tailings storage facility. With the final tails thickener in the flowsheet, there was insufficient heat loss from the plant, to remove heat generated by the autoclave. The final tails thickener was acting as a heat recycle system, effectively reducing heat loss from the plant to the tailings storage facility. With the tailings thickener in place, it would be required to install a large slurry or water cooling tower within the flowsheet. A heat balance undertaken with the final tailings thickener removed confirmed that there was no required cooling system for the plant. Following an economic evaluation, the final tailings thickener requirement was removed from the proposed flowsheet.

An optimization study was carried out during early 2007 to determine the optimum number of autoclave discharge wash thickeners. A circuit of four CCDs was selected as a reasonable compromise between increased capital (with increasing to five CCDs) and incurring high water recycle rates with low slurry densities in acidification and neutralization (by decreasing to three CCDs).

16.1.7 Environmental Testwork

Testwork was also completed on the final plant tailings. Analyses were undertaken by Lakefield, and included:

- Detailed species analysis of the liquor component
- Final tails solids phase analyses
- Toxicity characteristic leaching procedure
- Synthetic precipitation leaching procedure
- Meteoric water mobility procedure
- Kinetic tests.

The metallurgical process adopted for Donlin Creek is favourable for the establishment of tailings that are not acid producing, through the undertaking of near-complete sulphide sulphur oxidation.

Experimental arsenic speciation mineralogy was carried out by Canadian Light Source on the various key pilot test streams, including both the detoxified CIL tails, and the neutralized tails. This experimental mineralogy indicated that the significant proportion of arsenic in the tailings streams occur as scorodite-based compounds.

To confirm the applicability of the Cherokee Chemical UNR mercury precipitation reagents, as used at operating mine sites in the USA, a set of tests were undertaken at the Barrick Technology Centre. It is recommended that the process plant design include a dosage facility for Cherokee reagent UNR 829, to permit addition to a recirculating water stream for precipitation of mercury in solution into a stable HgS solid, and thus eliminate potential build-up of mercury in the process water circuit.

16.2 Process Recovery

Gold is recovered in two areas in the proposed plant:

- Gold recovered from the flotation circuit to the flotation concentrate.
- Gold recovered through leaching/adsorption (CIL) of the pressure oxidized (autoclaved) flotation concentrate.

Overall gold recovery is estimated to be 89.5% based on the combined LOM average recovery of 92.6% from flotation and 96.6% from POX and CIL treatment of the concentrate.

Flotation Recovery

Due to the refractory nature of the Donlin Creek mineralization, and the relatively low grade of the flotation tails stream, it is not economically viable to recover gold from the flotation tails stream. Therefore gold not recovered to the flotation concentrate is directed to plant tails and represents a final gold loss.

Pilot flotation testing was exclusively undertaken on non-oxidized mineralization, as this represents the majority type of the Donlin Creek deposit (90%) with all partially oxidized mineralization excluded from the pilot composite sample. The establishment of gold recovery from the MCF2 pilot program was achieved by means of fitting a linear regression line through all the MCF2 pilot survey results. Gold recovery survey calculations incorporate both the primary rougher concentrate, and the secondary rougher cleaner concentrate. Cleaner scavenger concentrate was recirculated to the feed of the secondary rougher.

At the design target of 7% (total) sulphur, based upon linear regression fit to the MCF2 pilot plant results, gold recovery of the blended composite sample tested was 94.64%. This recovery estimate formed the basis of the flotation gold recovery estimate, but required adjustment to account for effect of geological domain and alteration (oxidation extent) effects.

The MCF2 pilot plant sample was relatively high in mineralization from the ACMA, Aurora and SHL geological domains, and low in mineralization from the Lewis and Vortex geological domains.

Adjusting the MCF2 pilot plant recovery based upon geological domain composition using the geological domain variability flotation performance, results in the overall reduction of the MCF2 pilot plant recovery from 94.64% down to 94.4%. It is therefore recommended that 94.4% is used as the basis of recovery of non-oxidized mineralization at Donlin Creek.

There is a large variation in flotation test results (i.e., gold recoveries to target concentrate grades) of the oxidation-affected mineralization. A geological wireframe was developed that incorporates all logged oxidation extent categories, aggregated into a single oxidation rating, as either oxidized, or not oxidized. From this wireframe, blocks could be allocated into a conceptual mill feed schedule, and based on the schedule, a flotation recovery for this component of the mill feed was estimated on a scheduled basis. The oxidized-affected tonnage portion of the deposit was estimated, based on the wireframe modelling, at 7%.

To allow the different geological domains to be assigned a specific flotation recovery within a proposed mine plan, and to improve the estimation of time-based cashflow from the mine, results from variability testwork programs can be used, once adjusted to match the MCF2 pilot plant results.

Based on this work, a recovery figure for the LOM overall flotation circuit was determined as 92.6%.

CIL Recovery

Hatch reviewed the pilot autoclave testwork completed to date on the Project and has concluded that, providing the concentrate sample used for piloting during the 2007 Phase 2 test program is representative of the overall mineralization, and based upon the proposed plant design, an overall gold recovery of 96.6% can be achieved through the POX/CIL circuits on a continuous and long-term basis.

16.3 Conceptual Plant Design

The planned process route for Donlin Creek is based on conventional technology, with the concentrator, pressure oxidation, and cyanidation facilities at the forefront of technology for large, modern gold mines. A flowsheet showing the process route is included as Figure 16-1.

Mineralization will be dumped by mine haul trucks into dump hoppers ahead of a gyratory crusher. The feeder discharges to a coarse ore-feed conveyor to the coarse mineralization (“coarse ore” or “crushed ore”) stockpile.

The covered stockpile is planned to have a capacity of 38,000 t, representing 16 hours of process plant operation, and a total capacity of approximately 174,000 t, representing 3.2 days of process plant operation.

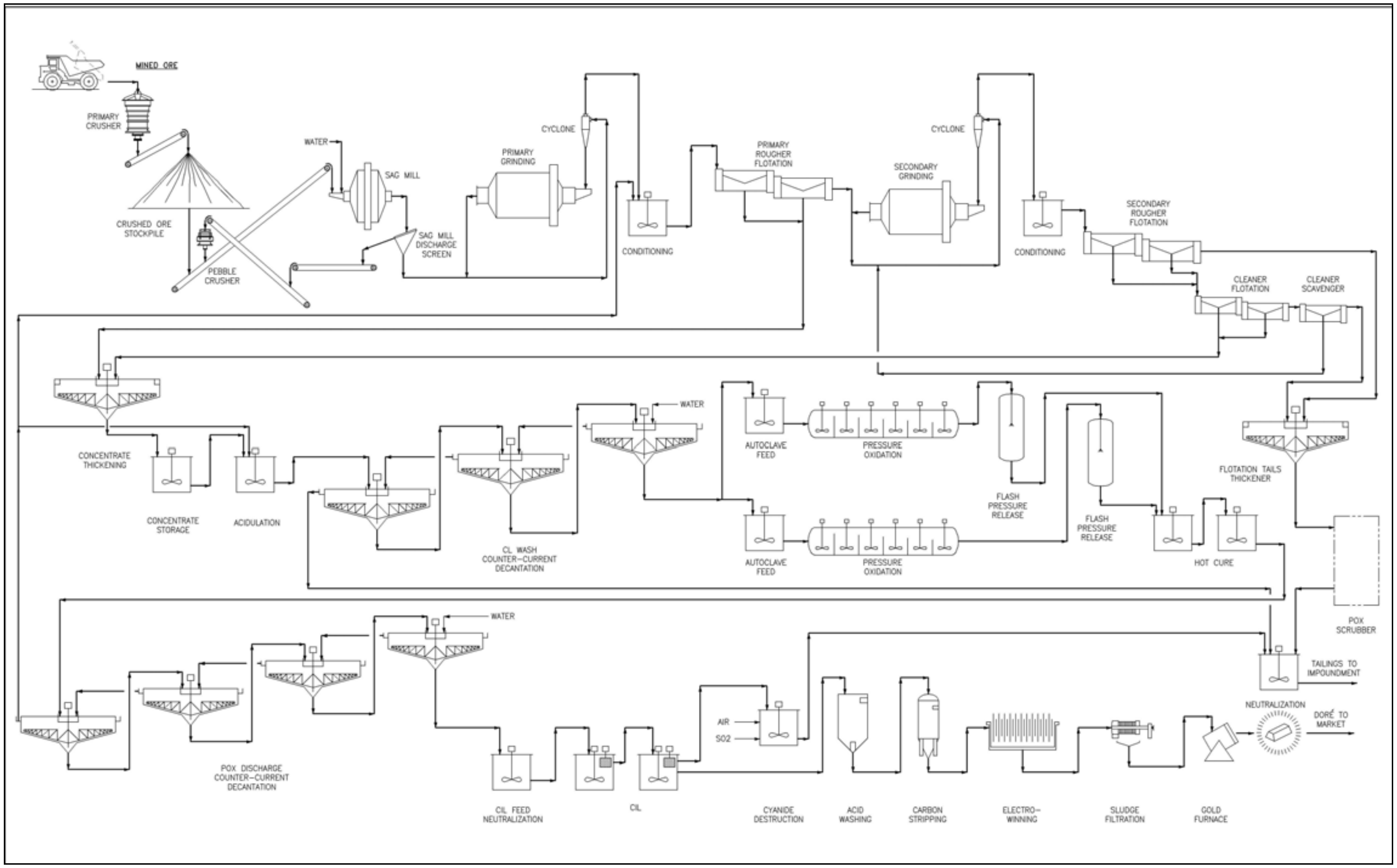
The reclaim tunnel and reclaim feeder chamber are to be sited underneath the coarse mineralization stockpile. Four apron feeders, are provided in the chamber. The nominal feed rate to the SAG mill will be achieved with three feeders operating. SAG mill critical size material reports to the pebble crusher. The discharge from the pebble crusher will join the new feed from the coarse mineralization stockpile.

The overall grinding configuration planned consists of an open-circuit SAG mill followed by a MCF2 circuit. The MCF2 circuit design entails a primary ball mill followed by a primary rougher flotation; the tailings produced from primary flotation are sent to a secondary ball mill, followed by a secondary rougher flotation. The two individual ball mills will operate in a closed circuit with their respective classification cyclones.

Under the design concept, SAG mill discharge is screened, and oversized pebbles are conveyed to two large cone crushers. Crushed pebbles will normally be returned to the SAG mill. The total system throughput is expected to average 53,500 t/d at 93% availability.

The SAG mill feed conveyor is planned to discharge into the SAG mill feed chute and then into the SAG mill. Process solution (primarily overflow from the concentrate and flotation tailings thickeners) will be added at this point to flush the mineralization into the mill and provide the correct dilution for grinding. Copper sulphate will then be added to the feed end of the SAG mill to activate sulphide mineralization.

Figure 16-1: Conceptual Process Block Flow Diagram



Note: Figure courtesy DCLLC

Top size SAG mill discharge will leave the mill through a trommel screen, where the bulk of the water and fine solids will be removed. The oversize will be washed over a vibrating screen. Undersize from the trommel screen and vibrating screen will collect in the SAG mill discharge launder. Vibrating screen oversize will discharge into the screen oversize chute, then flow by gravity to SAG mill discharge conveyor No. 1. Conveyor No. 1 is planned to discharge to SAG mill discharge conveyor No. 2, which in turn will discharge to the bin feeding the pebble crusher. Undersize material from the SAG mill discharge screen will drop into the primary grinding cyclone feed pumpbox, where it will join the discharge from the primary ball mill.

Discharge from the primary ball mill is designed to exit the discharge trunion into a trommel screen attached to the ball mill. Oversize material, consisting primarily of non-mineralization trash such as broken ball chips, is planned to drop from the end of the trommel screen into a rejects hopper. Undersize material will pass through the trommel screen into the primary cyclone feed pumpbox along with the SAG mill screen underflow.

The fresh feed for the secondary ball mill is expected to be a combination of the slurry from the primary rougher tailings pumpbox and the cleaner scavenger concentrate. These streams will flow into the secondary grinding cyclone feed pumpbox, where they will join the secondary ball mill discharge.

Discharge from the secondary ball mill is projected to exit in the same manner as for the primary mill. Oversize material will be dropped from the end of the trommel screen into a rejects hopper. Undersize material will pass through the trommel screen and into the secondary cyclone feed pumpbox, along with the rougher tailings and the cleaner scavenger concentrate. The secondary cyclone feed pump is expected to transport slurry to the secondary cyclone clusters. The secondary cyclone overflow stream will then be sent to the secondary rougher flotation feed distributor.

Primary rougher concentrate from the rougher portion is designed to be sent directly to the concentrate thickener. Primary rougher tailings will be sent to the secondary grinding cyclone feed pumpbox as part of the MCF2 circuit.

The secondary rougher concentrate will be sent to a cleaner flotation; the concentrate obtained from the cleaner flotation will be combined with the primary rougher concentrate. The tails obtained from the cleaner flotation will be sent to a cleaner scavenger flotation train. The cleaner scavenger concentrate is planned to be sent to the secondary grinding cyclone feed pumpbox, and the tails mixed with rougher tailings and sent to the flotation tailings thickener. The primary rougher concentrate and the concentrate produced from the cleaner flotation are designed to be sent to the concentrate thickener.

Concentrate from flotation passes to the concentrate thickener de-aeration tank and from there to the centre well of the concentrate thickener. Thickener overflow is returned to the grinding and flotation areas as process water while underflow is pumped to the concentrate storage tank circuit.

Acidic solution recovered from the POX counter-current decantation (CCD) wash circuit is mixed with the concentrate with the aim of consuming 85% to 100% of the carbonate gangue component of the concentrate. The acidulated material is washed in a three-thickener CCD circuit that displaces the solution with raw water to reduce the overall levels of soluble mineral ions reporting to the POX circuit in the slurry. Slurry is mixed with solution in the thickener feed tank at each stage before entering the thickener feed well. Washed slurry from the final thickener is pumped to the POX circuit. Overflow from the first thickener is sent to the flotation tailings neutralization circuit.

Autoclave feed slurry is transferred from the flotation concentrate storage tanks into two agitated autoclave feed storage tanks adjacent to the POX area. These tanks provide the autoclave plant with a continuous feed unaffected by upstream throughput variations. Slurry is transferred to the heater vessels that pre-heat the incoming slurry to varying temperatures, depending on the sulphide sulphur grade of the feed material, using flash steam from the autoclave discharge flash system. From each heater discharge, a single feed line feeds each autoclave. The autoclave is supplied with high-pressure oxygen gas, high-pressure cooling water, and high-pressure steam. Oxygen is produced at an on-site air separation plant.

Each autoclave discharges into a flash vessel. Autoclave discharge slurry is depressurized to atmospheric pressure, generating flash steam in the process, which is used as required to preheat autoclave feed or condensed in a quench system. Flash vessel underflow is directed by gravity to an oxidized slurry seal tank. Slurry from this tank is transferred by gravity to the downstream hot cure tanks.

Slurry flow from the POX circuit is washed in a four-thickener CCD circuit. Reclaim water is added to the last thickener in a flow direction counter to the solids in order to decrease the acidity of the pulp. Washed slurry in the underflow from the final thickener is pumped to the CIL solids neutralization circuit. Thickener overflow is clarified in a clarifier and used within the plant to provide acidification of the concentrate fed to the POX circuit and also to the flotation feed to assist in promotion of the sulphide mineral floatability, with the remainder reporting to neutralization. Clarifier sludge is intermittently returned to the first thickener in the circuit.

In the circuit, flotation tailing are pre-heated to 55°C through the autoclave quench vessel and then combined in a series of large aerated and agitated tanks, with the excess diluted acidic wash liquor from the chloride CCD wash circuit, where the flotation tailings act as neutralizing material (source of natural carbonates) for reaction with the acidic liquor.

Excess flotation tailings are collected, sampled, and passed to the flotation tailings thickener feed collection box, and from there to the centre well of the thickener. Thickener overflow is pumped to flotation process water tank. Thickener underflow is pumped to the POX circuit autoclave scrubber. Acidic solution from the POX CCD wash and spent acid from the elution circuit are combined with autoclave quench tails in the solution neutralization circuit.

Tailings from the cyanide destruction circuit are introduced into a lime neutralization tank where lime is added in the presence of air to bring the pH to 7. This material then flows by gravity to the final tailings pumpbox. A lime addition line is provided directly from the lime storage tanks to neutralization circuit in order to supply lime to neutralize the entire slurry when the flotation circuit is not in operation. Discharge from the neutralization circuit passes through a three-stage sampling system into the final tailings pumpbox. Provision is made for the addition of lime slurry, reclaim water, and flotation process water to adjust the pH of the final tailings, which are then pumped to the tailings storage facility (TSF).

Underflow from the final POX CCD wash circuit thickener is neutralized in the solids neutralization circuit. This material then passes through a sampling system to a pumpbox, from where it is pumped to the CIL circuit.

Discharge from the CIL feed neutralization circuit flows by gravity to a sampling system which produces a composite CIL feed sample for use in operational control and calculating metal balances. A nominal tonnage of 365 t/h at 35% solids will be pumped from the CIL feed pumpbox to the first of six CIL tanks by centrifugal slurry pumps. The slurry, with a retention time of four hours per tank, flows by gravity through each of the six tanks, ultimately reporting to the cyanide destruction reactor tank. This tank is covered, and agitated, and is where the residual WAD cyanide concentration is reduced from nominally 100 ppm to the cyanide levels required by permit. Sodium cyanide solution is pumped to the CIL circuit for cyanide leaching of gold. The system is capable of pumping 20% sodium cyanide solution to each of the first three tanks and adding the bulk of the cyanide to the first tank. A lime loop will allow for lime addition to each of the six CIL tanks. The pH will be monitored and lime added as needed to maintain a pH set point of approximately 9.0.

Loaded carbon, at a nominal gold loading of 4,800 g/t reports to one of two carbon acid wash vessels by gravity from the loaded carbon screen. After the acid wash and neutralization processes are complete, the carbon is pumped from the acid wash vessel to one of two strip vessels. A carbon strip begins as soon as the carbon is transferred to the strip vessel and the transport water has completely drained out of the vessel. Pregnant solution exits the strip vessel and flows through a heat exchanger before reporting to the pregnant tank.

Barren solution is pumped through the strip vessel for a nominal 8 hours to complete each strip. When the strip is complete one bed volume of raw water is pumped through the strip vessel. This solution rinses the residual solution from the carbon and cools the carbon in preparation for transfer. After the carbon is rinsed, it is pumped to the carbon dewatering screen before the kiln. The kiln is sized to process 100% of the carbon stripped to maintain high carbon activity levels throughout the carbon circuit. Carbon will be processed through the kiln at the rate of 1.5 t/h for reactivation. Kiln discharge reports to the carbon quench tank.

The pregnant solution is pumped through two parallel trains of two electrowinning cells. On exiting the cells, the solution reports to the barren solution discharge tank and is pumped to the barren tank.

The electrowinning cells will be taken out of service for cleaning three times each week. One cell will be shut down and cleaned at a time, allowing the electrowinning circuit to function normally while the cell is cleaned. The precious-metal-bearing sludge will be washed from the bottom of cell. The cathodes will be either washed in place or removed to a wash tank and be power-washed to release the sludge. The sludge from the electrowinning cell and the cathode wash tank will report to the electrowinning sludge tank by gravity and be pumped through one of two sludge filter presses. The solution discharged from the sludge press reports to the barren solution discharge tank to be returned to the barren tank.

The sludge filter presses will be taken down and cleaned after the electrowinning cells are cleaned. The sludge will be placed in pans, loaded into a mercury retort, and heated to remove mercury. Most of the mercury will report as elemental mercury and be collected in flasks and shipped off site. The remaining mercury collected in the retort will adsorb onto activated carbon within the retort. Periodically the activated carbon will become loaded with mercury and will be replaced with new carbon. The carbon loaded with mercury will be shipped off site.

Smelting fluxes are mixed with the sludge after the retort and the mixture is charged to the induction smelting furnace. Doré bars are poured from the smelting furnace and shipped off-site for further refining.

16.3.1 Process Water

Water for the plant distribution system comes from the following sources: contact water from the contact water pond, raw water from peripheral pit dewatering, reclaim water from the TSF, and fresh water from interception ponds (as required to make up shortfalls).

Contact water is water from the mine facilities and waste dump runoff that collects in the contact water pond. During periods of high runoff into the contact pond, when quality degrades and quantities are excessive, contact water substitutes for reclaim water in flotation and throughout the plant. In turn, raw water and fresh water can be substituted for normal contact water uses if the quality of the contact water suffers from high suspended solids.

The highest-quality water for use in the plant comes from the peripheral dewatering wells in the pit. As long as contact water is of sufficient quality and quantity, the raw water is treated in the water treatment plant (WTP) and discharged to the environment. When required to replace contact water, it is suitable for all contact water usages. Raw water is also important as the source of water for charging mill cooling and heat transfer systems.

When the quantity of pit dewatering water is insufficient, runoff water recovered from the diversion system around the TSF. This water will be pumped from the diversion dams to a fresh/firewater tank and from there to the raw water tank.

The reclaim water system supplies water to processes where high water quality is not required. Water is reclaimed from the TSF and pumped to a reclaim water head tank. Reclaim water is also supplied as the feed to the gland water system and flotation process water system.

16.4 Comment on Section 16

Metallurgical testwork completed on the Project was appropriate to establish the optimal processing route for a refractory gold deposit.

Metallurgical tests were performed on samples that were representative of the mineralization.

Recovery figures are based on pilot-plant testwork that is appropriate to the two areas of the plant where gold is proposed to be recovered.

The planned process route is based on conventional technology, with the concentrator, pressure oxidation, and cyanidation facilities at the forefront of technology for large, modern gold mines.

17.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

17.1 Mineral Resource Estimation

The geological and resource model for the Project that is the subject of this section is known by the descriptor "the DC8 model".

17.1.1 Database

The database used to support Mineral Resource estimation was restricted to holes and trenches through 2007 located within, and immediately adjacent to, the block model limits. This includes 1,059 core holes, 337 RC holes, and 282 trenches totalling 361,174 m.

17.1.2 Wireframes

Lithological codes in the drill hole database were used to construct polygons of the lithological groups in cross section and plans. Polygons were digitized on a computer screen by snapping to the drill holes in section. Wireframes were constructed from the polygon strings using Vulcan® software. Model blocks were first set to a default code (greywacke), and the solids were used to assign a corresponding geological code to the 3D block model. Different lithologies were prioritized when assigning lithology codes in the block model. The solids were validated and checked for crossing errors, consistency, and closure prior to use.

The 3D block model was created with a constant block size of 6 m x 6 m x 6 m. AMEC considers that the 6 m block size is a good balance between the blocks being small enough to reflect the geometry of the lithological domains, but large enough to make construction of the model manageable. The 6 m block size is also a subunit of the SMU size used in the mine plan.

17.1.3 Exploratory Data Analysis

AMEC constructed univariate statistics, histograms, probability graphs, box plots, contact profiles and bivariate cross plots to establish data distributions and relationships between variables. After reviewing the statistics, AMEC considers the results of the exploratory data analysis (EDA) analyses are consistent with the modeling methodology.

17.1.4 Grade Capping

Raw assays in the database were examined for the presence of local high-grade outliers, and overall grade distributions were used to establish capping values. The raw assay data were grouped by rock type, and capping values for gold were determined for each major rock type using cumulative frequency plots. Total sulphur, arsenic, mercury, and antimony assays were not capped. Capping grades and metal loss for each rock type are summarized in Table 17-1.

AMEC reviewed the capping thresholds and is of the opinion that the capping thresholds applied by the DCLLC are reasonable. AMEC also checked the implementation of the capping thresholds in the composites and found the capping thresholds to be correctly applied.

17.1.5 Compositing

Assay intervals were composited prior to grade estimation to place the assay data on a near-constant support. Composites were created down each hole at 6 m intervals. The composites were not broken at intrusive or sedimentary boundaries. Composites that are less than 6 m do occur, but are at the base of drill holes away from the main mineralization.

Three composite databases were generated: one for Au values where non-assayed (missing) intervals are set to zero, one for sulphur, and one for arsenic, antimony, and mercury (multi-elements). In the latter two databases, assay values were not capped and missing intervals were ignored. The length of the assay interval and majority lithological code was recorded.

AMEC notes that the compositing strategy incorporates an amount of contact dilution, and is reasonable given the size of the mineralized domains and the anticipated production rate. Compositing the data using lithological breaks was reported by the DCLLC to generate lots of high-grade assays with short lengths that caused many problems during estimation.

Biases due to compositing were checked by AMEC by comparing the statistics of the assays and composites weighted by length. The bias for all elements is acceptable; however, the biases in the mercury compositing are higher than the other elements and should be investigated.

17.1.6 Discriminator Model

Gold, arsenic, antimony, mercury and sulphur grades were estimated using a combination of lithological and discriminator or indicator domains. The lithological domains were first simplified by combining all the intrusive rock units into a single intrusive category resulting in three lithological domains, greywacke, shale, and intrusive.

Table 17-1: Summary of Capping Grades for Rock Types

Rock Type	Capping Grade in Current Model (g/t Au)	Percentile	No. of Samples Capped	Metal Loss (%)
GWK	25	98.84	103	3.60
SHL/ARG	30	99.96	15	6.46
SLT	20	98.07	15	12.47
MD	30	97.60	23	13.14
RDA	20	99.70	43	1.23
RDF	16	99.29	26	3.37
RDX	26	99.64	54	1.63
RDXB	28	99.84	17	0.84
RDXL	10	99.34	39	2.05

The intrusive rocks and the shale were constrained by lithology wireframes, and mineralization in the greywacke was only constrained by the estimation parameters.

Difference in the style of mineralization between the lithological units is attributed to differential structural preparation before mineralization, and is a very key factor in the design of the modeling methodology. The intrusive rocks are more brittle, fracture more easily, and create openings for the mineralization. The sedimentary units tend to be more ductile during structural deformation, and the mineralization tends to be more confined with the competent greywacke being a more favourable host than the shale.

Two discriminator models were constructed. The first model used a gold indicator, and the second indicator model was constructed using a sulphur indicator. The gold indicator model was used for separating mineralized material from non-mineralized material for Au, As, Sb, and Hg grade estimations. The sulphur indicator model was used for separating sulphide mineralization from non-sulphide mineralization for sulphur estimations.

The gold discriminator model was constructed by first creating an indicator field in the composite file consisting of zeros and ones. If the gold assay was less than 0.25 g/t, the indicator field was set to zero, and if the gold assay was greater than or equal to

0.25 g/t, the indicator field was set to 1. The indicators were used to estimate the blocks within each lithological domain using an inverse distance squared method and a two-pass search criteria elongated along the structural trend striking 024° and dipping 68° to the southeast. In the first pass, a relatively large number of samples and drill holes were used to estimate the block probabilities. At least three drill holes were required to create an indicator value for each block based on the following sample selection criteria: a minimum number of six composites per estimate, a maximum of 13 composites per estimate, and a maximum of two composites per drill hole.

After the first pass, some areas in the indicator model did not receive an indicator value. To assign an indicator value to these blocks, a second indicator pass was performed with search and selection criteria the same as the first pass except that the minimum number composites required was reduced to four. This change required two drill holes per estimate instead of three to allow estimation of blocks that did not receive an indicator value in the first pass. If the estimated indicator value in the model were greater than or equal to 0.5 (or 50%), the zone was categorized as mineralized. If the indicator value was less than 0.5, the zone was categorized as non-mineralized.

Sulphur was estimated similar to the gold estimations except that the discriminator model used to divide the lithological domains into mineralized and non-mineralized zones was based on sulphur assays instead of gold assays. In the sulphur discriminator model, the sulphur indicators were constructed using a sulphur discriminator of 0.5%. If the sulphur assays were less than 0.50%, the indicator field was set to zero, and if the sulphur assay was greater than or equal to 0.5%, the indicator field was set to 1. The indicators were used to estimate the blocks within each lithological domain using an inverse distance squared method. If the estimated indicator value in the model were greater than or equal to 0.5 (or 50%), the zone was categorized as mineralized. If the indicator value was less than 0.5, the zone was categorized as non-mineralized.

AMEC checked the tagging of the composites to each lithological/indicator domain and found the tags were applied correctly.

17.1.7 Grade Estimations

Gold was then estimated for each of the six domains (shale mineralized, shale unmineralized, greywacke mineralized, greywacke unmineralized, intrusive mineralized and intrusive unmineralized). The domains were designed to confine the assays inside the mineralized domains from smearing into unmineralized domains and restrict assays in the unmineralized domains from diluting the grades in the mineralized zones. Arsenic, antimony, and mercury were estimated using the same gold domains and estimation parameters except that the assays were not capped.

Gold grades were then estimated into the block model using an inverse distance to the third power methodology for the two populations: (1) internal to the mineralized envelope, defined as blocks with indicator values greater than or equal to 50% and (2) external to the mineralized envelope, defined as blocks with indicator values less than 50%. Composites in the gold composite database were flagged as being either inside the 0.25 g/t Au indicator threshold (i.e., passing through blocks with an estimated probability of at least 50%) or outside the 0.25 g/t Au indicator threshold.

Estimation of grade into the blocks was broken into five passes based upon increasing search distances. The initial grade estimation pass used a “box search” with a search range having the same dimensions as a single block. A successive estimation pass used increasingly longer ranges out to a maximum of 125 m. Search ellipses were elongated along the structural trend striking 024° and dipping 68° to the southeast, and sample weights were adjusted based on the anisotropic model. Once estimated, blocks could not be overwritten by subsequent estimation passes.

Although the intrusive rocks are the primary host for the mineralization, economic mineralization is also found in the sedimentary units, mainly the greywacke. Approximately 74%, 22% and 4% of the ore-grade gold ounces are contained in the intrusive mineralized, greywacke mineralized and shale mineralized domains respectively. The non-mineralized domains contain less than one percent of the ore-grade mineralization, but are important for estimating the low-grade material for dilution calculations during the reserve calculations.

Sulphur grades were estimated using the same methods and parameters as for the gold grade estimation where sulphur data was available. Sulphur data, however, are less extensive than gold data, and a number of blocks were not estimated during the inverse distance estimation runs. To estimate these blocks, regression formulae were derived from the correlations between gold and sulphur for each of the major rock types, and these regression formulae were then used to assign sulphur values to unestimated blocks based on the estimated gold grade. Where gold grade was not estimated, a value of 0.001 g/t Au was assumed for the calculation.

Arsenic, Hg, and Sb grades were estimated using the same methods and parameters as for the gold grade estimation. A series of five passes was used to estimate blocks inside and outside the 0.25 g/t gold grade indicator populations. Separate estimation runs were generated for intrusive rocks, shale, and greywacke. Composites 6 m long were flagged as being either inside the 0.25 g/t Au indicator threshold (i.e., blocks with an estimated probability of at least 50% for intrusive rocks and 50% for shale and greywacke) or outside the 0.25 g/t Au indicator threshold.

Data for As, Hg, and Sb are much less extensive than for Au and S. Similar to the sulphur estimation methodology, regression formulae were derived from the relationship between gold and each of these elements for each of the major rock types. The regression formulae were then used to assign As, Hg, and Sb values to unestimated blocks based on the estimated gold grade. Where gold grade was not estimated, a value of 0.001 g/t Au was assumed for the calculation. AMEC reviewed the correlation, regressions, and implementation of the values and found the procedure to be reasonable given the strong correlations between the elements.

17.1.8 Dilution

Grade dilution will be a serious operational consideration given the nature of the narrow, steeply dipping mineralized zones that characterize the Donlin Creek gold system. Because of these narrow zones, the deposit was initially modelled with relatively small blocks to ensure that sufficient resolution was available to better characterize the deposit. Dilution and selectivity were determined using a Barrick in-house program referred to as “SMUman”, discussed in Section 17.2.3.

17.1.9 Variography

The 6 m composites were used to develop relative pair-wise and indicator variograms. Relative pair-wise variograms were generated for all sample data and by domain using orientations along the average strike and dip of the mineralized zones. This orientation was identified both geologically and through stereo-net analysis of oriented vein data. The analysis defines a plane striking 024° and dipping 68° to the southeast and forms the basis for search orientation during block estimation.

Indicator variograms were generated at 0.25 g/t Au for the 6 m composites. The correlograms at 0.25 g/t Au were fitted with a spherical model. Ranges of 30 m and 45 m were observed at 80% and 90% of the total sill variance.

17.1.10 Validation

DCLLC

Block model grades were validated by the DCLLC visually against drill holes and composites in section and plan view. A nearest-neighbour block model was also generated using 6 m composites to compare estimated grades in the block model.

Grade profile plots were generated by the DCLLC for the Measured and Indicated resource model as a further validation check. There was reasonable agreement between the composited assays and the estimated Au values for blocks classified as Measured and Indicated.

AMEC

AMEC visually examined estimated block model gold grades in cross section and level plan by comparing them with the composites in the drill holes. In general, the mineralization is controlled by the lithology wireframes, with a distinct striping of grades following the plane striking 024° and dipping 68° to the southeast dipping trend of the search range.

AMEC also noted that the area under the prominent ridge in the pit design (54,1000E, 6,879,500N) lacks drilling. AMEC recommends that this area should be explored as if economic mineralization could be found, it could have a significant impact on the design and potential economics of the pit. Estimation parameter files were checked for errors by AMEC and found to be created as intended. One rock code was omitted from S, As, Hg and Sb regressions, but is not a significant mineralization host and will have little impact on the Mineral Resources.

The block model for global bias was checked by comparing the average grades (with no cut-off) from the model (ID grades) with means from nearest-neighbour estimates for Measured and Indicated Resource blocks that lie inside the planned pit. No significant biases were observed for Au, As, Hg, and S, where the relative percent difference in grade is within ±5%. A significant bias of -10.63% bias was noted for Sb and should be reviewed.

Checks for local biases were performed by analyzing local trends in the grade estimates using swath plots created by the DCLLC. AMEC found minimal local bias for Au, S, As, and Hg. Antimony, however is locally biased with the inverse distance model over-predicting at depth and under-predicting between grid co-ordinates 6,880,000 and 6,880,500 north. This should be reviewed.

Domain Construction

Since the economic gold mineralization is predominantly hosted in the intrusive rocks, interpretation of the geologic codes in the drill holes to construct the intrusive boundaries is critical to define economic limits. AMEC considers that the best method to define the geometry of the intrusive rocks is to intersect the intrusive rocks perpendicular to their strike and dip so that the location of the contacts and true thickness can be adequately defined. The geometry and location of the intrusive rocks is particularly important at the bottom of the pits were the location of the limits of the economic mineralization defines the location of the highwalls, and small changes in the location of the economic mineralization at the toe of the highwall may result in millions of waste tons removed unnecessarily.

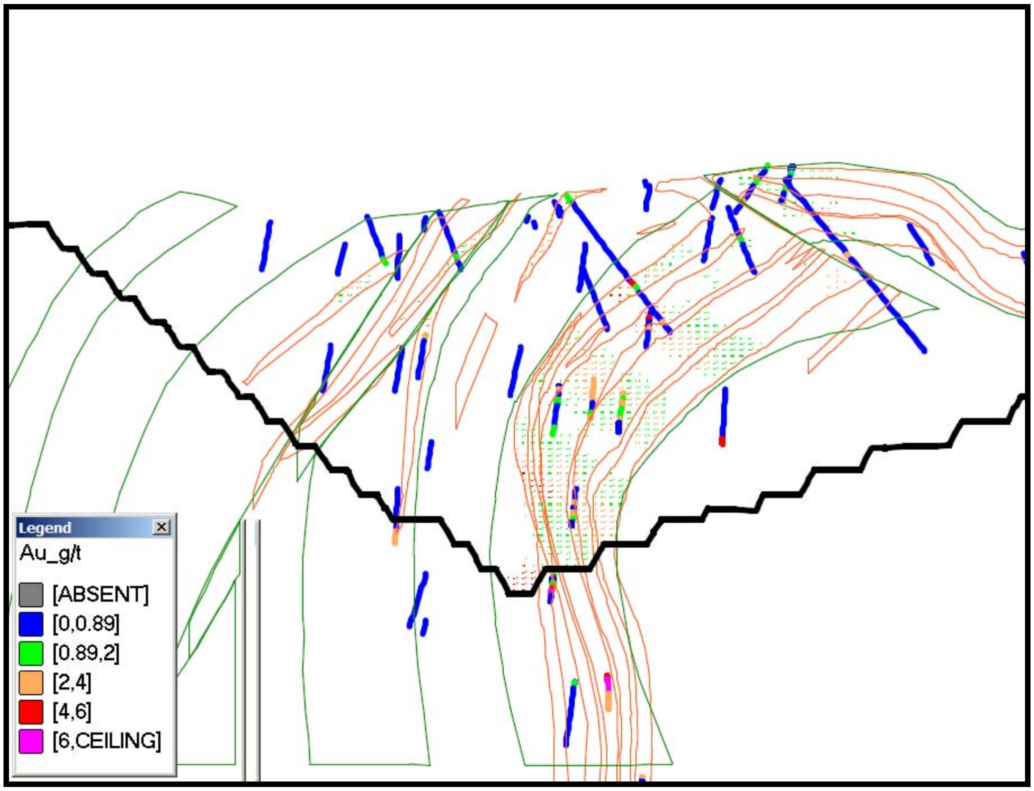
In the bottom of the ACMA pit, the majority of the mineralization is located within the intrusive rocks that were interpreted as trending northwest. The drill holes used to define the intrusive rocks and the economic mineralization, however, are predominantly in the same northwest direction since they were designed to intersect the northeast-trending mineralization in as perpendicular an orientation as possible. Defining northwest-trending intrusive rocks using northwest-trending drill holes makes establishing the limits of the economic mineralization subjective.

An example is shown in Figure 17-1 which is a close up view of the bottom of the ACMA pit looking northwest (315º). The figure illustrates that although the contacts of the intrusive rocks are well defined at higher elevations, the location of the intrusive contacts and hence the limits of the economic mineralization are subjective at the bottom of the pit. The figure is not demonstrating that the current location of the orebody is incorrect, it is only demonstrating that the location of the intrusive, and hence the economic mineralization, is subjective with the information available, which could lead to non-optimized location of the highwall with significant economic consequences. AMEC recommends that additional northeast-trending drill holes be drilled to confirm the location of the economic mineralization at the bottom of the ACMA pit before mining commits to the final highwall design.

ACMA Fault

Since the ACMA fault is reported to have up to 100 m of oblique left lateral reverse movement that post-dates mineralization, the fault should be included in the resource estimation domains. This is especially import in the greywacke where the geometry of the mineralization is not constrained by wireframes. AMEC recommends that any post-mineral faults be incorporated into the resource estimation model as the faults could have a significant change in the geometry of the mineralization and location of the pit. Similar studies as to those carried out on the ACMA fault should be performed on the other faults that cross the resource model.

Figure 17-1: Cross Section across Bottom of ACMA Pit Looking Northwest



Note: Figure courtesy DCLLC. The limits of the intrusive rocks are coloured orange, the limits of the shale are coloured green, the drill hole composites are color coded by gold values, the model ore blocks are color coded by gold values showing only those values greater than 0.889 g/t Au, and the horizontal distance across the figure is approximately 700 m.

17.1.11 Mineral Resource Classification

The resource model was classified using distance to nearest composite as stored in the model blocks during the nearest-neighbour grade estimate. Classification distances are based on the 80% and 90% of variance from the omni-directional indicator variogram model generated with 6 m composites using a 0.25 g/t Au discriminator, and fitted with the spherical model. The classification methodology is summarized in Table 17-2. Ranges of 30 m and 45 m correspond to 80% and 90% of the total sill variance respectively.

Table 17-2: Donlin Creek Mineral Resource Classification Methodology

Category	Minimum Distance to Nearest Drill Hole (m)	Maximum Distance to Nearest Drill Hole (m)	Minimum Number of Drill Holes	Intrusive Indicator Block Condition Criteria	Sediment & Greywacke Indicator Block Criteria
Measured	0	3	Block pierced by drill hole	≥0.0	≥0.0
Indicated	0	30	≥2	≥0.0	≥0.0
Indicated	30	45	≥2	≥0.5	≥0.7
Inferred	30	45	≥2	≥0.0 & <0.5	≥0.0 & <0.7
Inferred	45	60	≥2	≥0.5	≥0.7

To evaluate the resource classification, AMEC reviewed the continuity of the mineralization by performing a large block confidence limit study of gold using different drill hole spacing. AMEC considers that Indicated Resources should be known within ±15% with 90% confidence on an annual basis (production year), and Measured Resources should be known within ±15% with 90% confidence on a quarterly basis (production quarter). At this level, the drilling is usually close enough to permit the assumption of continuity between points of observation.

Based on the confidence limit study, a drill hole spacing of 50 m x 50 m would be required to classify material as Indicated which is generally in agreement with DCLLC criteria. Measured classification would require a 25 m x 25 m drill hole spacing indicating that the current criteria for classifying material as Measured is conservative.

Visual inspection of the blocks classified as Measured shows that the Measured material is a collection of isolated spots and does not convey any information about the continuity of the mineralized trends. AMEC recommends that the Measured classification method be revised so that the block-by-block resource classifications are geologically reasonable, with coherent zones that reflect a realistic level of geological and grade estimation confidence.

17.1.12 Assessment of "Reasonable Prospects of Economic Extraction"

Mineral Resources were confined within a Whittle® pit shell using economic parameters summarized in Table 17-3. Mill recoveries vary by rock type, domain, and degree of oxidation. Recoveries used for calculation of net smelter return (NSR) are summarized in Table 17-4.

Table 17-3: Economics used in Calculation of NSR for Mineral Resources

Item	Assumptions
Au selling price (Mineral Resources)	$US850/oz
Grams per troy ounce	31.10348
Process cost ($/t)	2.7273 * (sulphur grade) + 11.664
Administrative cost	$1.56/t
Refining, freight & marketing (selling costs)	$0.573/oz recovered
Royalty	3.75% – (Au price – Selling cost)

Table 17-4: Mill Recoveries used in Calculation of NSR for Mineral Resources

Rock Type and Domain	Recovery
Intrusive rocks – Akivik	94.17%
Intrusive rocks – 400	93.55%
Intrusive rocks – ACMA	93.05%
Intrusive rocks – Aurora	93.61%
Intrusive rocks – Vortex	91.82%
Intrusive rocks – Lewis	91.52%
Greywacke (all domains)	88.22%
Shale (all domains)	86.66%
Oxide / weathered rocks – S grade >1.8%	87.90%
Oxide / weathered rocks – S grade ≤1.8%	((8.7361*S3 - 49.806*S2 + 95.233*S + 30.004) *0.966)

The NSR cut-off for reporting was US$0.01/t. This figure represented the break-even cut-off grade for delineation between material designated as “waste” and material that would be designated “mineralization”. The average NSR value for blocks in the model that are classified as mineralization is higher than this mineralization/waste NSR cut-off value.

Undiluted Mineral Reserves that included 9.35 Mt of above resource cut-off-grade Inferred material that was reclassified to either Measured or Indicated was subtracted from the total resource reported from this pit optimization to determine the gold Mineral Resources.

17.1.13 Mineral Resource Statement

The gold Mineral Resources at Donlin Creek (Table 17-5) were classified in accordance with the 2005 CIM Definition Standards for Mineral Resources and Mineral Reserves, incorporated by reference into NI 43-101. Mineral Resources have an effective date of 31 December, 2008. Gordon Seibel, MAusIMM, an AMEC employee, is the Qualified Person for the estimate. AMEC cautions that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 17-5: Mineral Resource Statement, Effective Date 31 December, 2008, Gordon Seibel, MAusIMM.

Category	Tonnage (Mt)	Au (g/t)	Contained Au (Moz)
Measured	1.2	2.19	0.08
Indicated	93.4	1.97	5.92
Total Measured and Indicated	**94.6**	**1.98**	**6.01**
Inferred	54.5	2.29	4.02

Note:
1) Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
2) Mineral Resources are reported to an Au price of US$850/oz
3) Mineral Resources are exclusive of Mineral Reserves, and reported on a 100% basis
4) Sums may not agree due to rounding

17.2 Mineral Reserve Estimation

17.2.1 Throughput Rationalization Studies

A number of throughput rationalization studies were completed in 2007. Four block models were built for the throughput studies, based on the following:

- 6 m x 6 m x 6 m model = 6 m composites; 12 m x 12 m x 12 m model = 12 m composites; 15 m x 15 m x 15 m model = 15 m composites and 20 m x 20 m x 15 m model = 15 m composites.
- A potential overestimation of sedimentary blocks at larger block sizes was identified and was resolved by re-estimating grades in these blocks using composites that were constrained by the geologic wireframes. This prevented composites within the intrusive wireframes from being used to estimate grades in the shale or greywacke blocks. Similarly, only composites within the existing shale wireframes were used for estimation of shale blocks, and only composites outside all wireframes were used for estimation of greywacke blocks. Re-blocking was performed where applicable.

- Slope parameters were provided by BGC and were modeled in Whittle® using a zone code imported from MineSight®. The inter-ramp slope parameters (see Section 18.3) were flattened to accommodate assumed ramp requirements.
- Crooked Creek, which runs along the northwest region of the deposit, was a limiting factor in the optimization of pit shells. A minimum setback of 25 m from the creek boundary was used as a constraint in the ultimate pits.
- Different mining fleets were considered for the different scenarios, reflecting mine selectivity, bench height, peak mine production rate, and power supply source. Conceptual capital and operating costs were estimated for these fleets on an iterative basis to reflect preliminary mining schedules derived from initial optimization runs. Given the iterative form of the analysis, operating costs evolved from initial assumptions to final estimates; if significant variations resulted, then the analyses were repeated. The increase in mining costs resulted from the greater haul depth due to the larger size pits, relocation of the waste dump, and the additional ore tonnages.
- The metal price assumption for pit optimization was $750/oz gold. No other metals or metal prices were used. An assumed oil price of $85 per barrel was used.
- An average royalty cost was calculated for each scenario, based on a 1.5% royalty until Year 5 and 4.5% thereafter. Process recoveries varied by rock type.
- “Non operational” internal phases and ultimate pits were designed for each case. To speed the design process, details of benches and roads were not included; however, appropriate slopes were applied to accommodate anticipated ramp locations and minimum mining widths for all scenarios. Where appropriate, some design shapes were shared between cases. Each scenario had six phases in the ACMA mining area and three in the Lewis mining area, with the phases scaled to reflect the mining rate and width requirements of the particular scenario. While differing in size, phase design shapes were quite similar from case to case.
- Mine scheduling was carried out in a manner that attempted to minimize subjective differences from case to case. The Whittle® Milawa NPV scheduling algorithm was used to generate mine plans from the non-operational pit phase designs, resource models, and cost models. Buffer stockpiles were incorporated to ensure realistic usage of available mine capacity from year to year. The scheduling algorithm considered operating costs and revenues while attempting to maximize NPV.
- Elevated cut-off grades and use of stockpiling improved pre-capital NPV.
- As input to the initial limit optimization and subsequent mine scheduling, a resource model was used that attempted to reflect the selectivity of the proposed mining method. Subsequent review of mine schedules on a bench basis,

considering mining direction and potential blast dig-lines, indicated that the amount of dilution and loss implied by block regularization was inadequate and that an additional dilution factor should be applied.

The following conclusions were drawn from the throughput rationalization studies:

- Results confirm those of previous throughput rationalization studies.
- The preferred process throughput is 50 kt/d.
- Grid tie-in power is less attractive than site-generated power under the assumptions considered.
- An elevated cut-off and stockpiling policy in early years adds significant Project value on a discounted basis.

The studies resulted in the preferred development scenario being a 50 kt/d process facility with on-site power and a mine capacity of 440 kt/d with an elevated cut-off policy applied in the initial part of the mine life.

17.2.2 Pit Optimization

Two pit optimization exercises were undertaken:

- Initially, the raw resource model was optimized using a gold price of $750/oz. No dilution or ore loss was applied to the resource model prior to optimization. The objective was to generate shells to guide preliminary pit design work.
- A preliminary pit design, including internal phases, was used in the SMUman software to select bulk and selective mining areas. A second pit optimization exercise was then undertaken based on the resulting diluted model generated from SMUman. The study gold price assumption was revised to $725/oz following the first pit optimization exercise.

All figures were based on a $725/oz optimization of a diluted model.

Key features of the optimization runs were:

- Pit shell generation was not constrained by infrastructure because the only existing features are an aircraft landing strip, exploration camp, and drilling access roads. All the major infrastructure facilities planned for the Project will be external to the ultimate pit design and its area of influence. The pit shell generation, however, was constrained by Crooked Creek which is located to the west of the proposed pit area. Crooked Creek is a salmon-bearing stream.

- Mining will be performed on both 6 m and 12 m benches. Large, primarily waste regions and contiguous ore zones will be mined on 12 m high benches to increase mining productivity and reduce costs. Selected primary ore zones will be selectively mined on 6 m high benches.
- For the study, no sustaining capital allowances were added to either the processing or mining costs. As well, no discounting was applied, either in the form of direct bench discounting or net present value (NPV) analysis of scheduled nested shells. Revenue factor 1.0 shells were used for ultimate pit design guidance.
- The ore considered for processing in the optimization was based on a marginal cut-off grade that varied from block to block. Material was considered to be ore if the revenue of the block exceeded the processing and general and administrative (G&A) cost. The revenue was based on net gold price after refining charges and royalties had been deducted. The processing cost was a function of the sulphur content of the material being processed; therefore, the marginal cut-off grade varied block by block. Neither a minimum cut-off grade nor raised cut-off metal grade was applied.
- A net smelter return (NSR) value per tonne was then coded into each block of the resource model. This was done to provide a variable marginal cut-off grade based on processing costs, selling costs, and royalties, rather than gold grade alone. Measured and Indicated blocks were treated as potential mill feed, while Inferred and unclassified blocks were treated as waste. The processing cost is variable and is proportional to the sulphur content of the material being processed. To calculate an NSR value into each block of the resource model, a Vulcan script was written and used. NSR parameters are summarized in Table 17-6.

Table 17-6: Assumptions used for Calculation of NSR Values for Mineral Reserves

Economic Parameters	Assumptions
Au selling price (Mineral Reserves)	US$725/oz
Grams per troy ounce	31.10348
Process cost ($/t)	2.7948 * (sulphur grade) + 12.82
Administrative cost	$1.61/t
Refining, freight and marketing (selling costs)	$0.573/oz recovered
Royalty	3.75% − (Au price – selling cost)

- The mining cost was based on first principle calculations for a remote conventional open pit mine using a truck and shovel fleet. Costs include direct operations and maintenance for drilling, blasting, loading, and hauling. Other costs are general mine support for road, bench, and dump maintenance, dewatering, ore control, and re-handling a nominal 25% of run-of-mine (ROM) ore. The mining cost also includes \$0.08/t for G&A cost directly allocated to mining for aviation, catering, camps, clinic, health and safety, and logistics.

- Since G&A is typically included in the processing cost for L–G analysis in cases limited by processing, assigning a portion of the G&A cost to mining has the effect of lowering the marginal cut-off grade; only G&A costs directly attributable to mining were assigned in this way. The reference mining cost used in the pit shell generation was \$1.68/t of material mined. Ore and waste mining costs were assumed to be equal. An incremental increase in cost with depth of \$0.0025/t/m was applied to blocks below a reference elevation of 196 m to represent increased haulage cost with pit depth.
- The processing cost was based on first principle calculations for a 53.5 kt/d processing facility. It also included an allowance of \$0.51/t for G&A directly allocated to processing that covers aviation, catering, camps, clinic, health and safety, and logistics. Processing costs were variable based on sulphur grade according to the following: 2.7948 x (S%) + 12.82. Average processing costs were \$15.97/t. Costs were not included for any expansion of the tailings storage facility or other sustaining capital.
- The mining cost for delivered diesel of \$0.80/ℓ was slightly higher than the price of \$0.76/ℓ used for the process cost calculation. Delivered diesel price was refined during the course of the study, and pit optimization inputs were calculated using the parameters available at the time.
- Gold recovery values were based on work completed for the Project. Recoveries for non-oxide ores are quoted as a constant for each rock type, whereas recoveries for oxide ores vary with sulphur grade (see recovery data in Table 17-4).
- G&A cost was based on first principle calculations for a remote open pit mine supported by fly-in operation and a camp. G&A costs associated with aviation, catering, camps, clinic, health and safety, and logistics were built into the base mining and processing costs and were proportioned according to workforce. The remaining G&A of \$1.61/t of material processed was added to the processing cost in the L–G analysis as is typically assumed for cases limited by processing rate.
- Refining, freight, and royalties values were provided by Barrick. Based on actual costs at Barrick operations, the combined refining and freight cost was \$0.573/oz of gold. An average royalty charge of 3.75% of the net gold value was added after the refining and freight cost had been applied. The 3.75% royalty was based on an assumed 20-year mine life.

- A gold price of $725/oz was used for the pit optimization. No other metals or metal prices were used.
- Geotechnical domains, design sectors, slope angles, and associated assumptions were provided by BGC. BGC's inter-ramp slope angles were reduced for each design sector in each of the geotechnical domains to flatten the generated pit shell; this allowed for the haulage ramps that would be included in the mine design. Slope angle reductions were based on the haulage ramp width, the number of times a haulage ramp traversed a design sector, and the overall slope height of the sector. Certain slope angles were further adjusted to smooth the transition to an adjacent design sector. This enabled the L–G software to generate structural arcs in cases where the slope angles contrasted sharply in "narrow" design sectors. The slope angles were either increased or decreased to enable the generation of arcs while attempting to preserve slope steepness.

The ultimate pit shell includes 382,980 kt of ore containing 29,527 koz (918,386 kg) of in-situ gold and has a strip ratio of 5.09. The subsequent mine design, complete with haulage access, includes 383,791 kt of ore containing 29,269 koz (910,368 kg) of in-situ gold and has a strip ratio of 5.69. The pit shell considered Measured and Indicated Mineral Resources. An additional 3.6% of Inferred Mineral Resource tonnes (3.0% Au oz) was added to the optimization resource base due to the reclassification of Inferred to Measured or Indicated during the SMUman 12 m block category allocation (see Section 17.2.3).

The base mining cost (before incremental mining cost with depth) was $1.68/t, the average processing cost was $15.97/t, and the G&A cost was $1.61/t. The ultimate pit mine design, although slightly larger than the pit shell, fits reasonably well, with an increase of 10.2% in total rock mined, a decrease of 1.1% in gold grade, and a decrease of 0.9% in contained metal.

17.2.3 SMUman Dilution

During the feasibility study update, NCL Ingineria y Construccion S.A. (NCL) developed an in-house software application in consultation with Barrick that is referred to as "SMUman", which was used to assist in the identification of areas within the Donlin Creek Mineral Resource where selective mining could be economically beneficial. The economic analysis incorporates mining dilution and ore losses associated with the assumed level of mining selectivity. The software calculates the NSR of each block assuming it is mined on a 6 m bench and as part of a 12 m bench. Element grades for a 12 m bench block are derived by performing a tonnage based weighted average of the upper and lower 6 m blocks.

A mining penalty is applied to the selective mining scenario. This is a differential unit mining cost to reflect the additional costs associated with selective mining on a 6 m bench versus on a 12 m bench. This cost would include allowances for additional drilling and blasting, reduced loader productivity, and increased ancillary equipment requirements.

Using the NSR value of each block, the "SMUman" software identifies which blocks are "ore" for both the 6 m and 12 m mining scenarios. Ore loss and dilution is then simulated for each scenario based on the assumed degree of mining selectivity. The minimum selective mining unit (SMU) size for calculation of ore loss and dilution was assumed to be approximately 5,000 t when mining a full 12 m bench:

- On a 6 m mining bench, a minimum SMU size of four contiguous 6 m x 6 m x 6 m blocks (in plan view) was required to form an ore pod. This can include diluting waste as long as the overall pod grade is above cut-off, and equates to a minimum mining unit size of approximately 2,330 t.
- On a 12 m mining bench, a minimum SMU size of four contiguous 6 m x 6 m x 12 m blocks (in plan view) was required to form an ore pod. This can include diluting waste as long as the overall pod grade is above cut-off, and equates to a minimum mining unit size of approximately 4,660 t.
- Conversely, for both 6 m and 12 m scenarios, a minimum of four contiguous waste blocks (in plan view) was required to be successfully separated from adjacent ore. Any less than four blocks is assumed to be diluting waste.

Polygons representing potential selective mining areas were digitized into "SMUman" on a 12 m bench-by-bench basis. The software reports the overall NSR $/t (of rock) of the polygon assuming selective mining on two 6 m benches versus bulk mining on a single 12 m bench. The selective mining unit cost penalty was deducted from the NSR $/t for the selective mining scenario. A visual comparison of the results was used as a guide for identifying areas to be selectively mined.

Practical mining areas were designated for selective mining if they demonstrated a significant NSR $/t benefit over bulk mining. This significant benefit was chosen as being approximately 5%. In general, this benefit occurred in the ACMA deposit, which includes flatter-dipping areas and is less contiguous than the Lewis deposit.

Inferred Mineral Resources within Optimization Shell

The mine plan is based on Measured and Indicated Mineral Resources. All selective mining areas (on 6 m benches) based on the 6 m x 6 m x 6 m block treat Inferred material as waste. In bulk mining areas, however, "SMUman" provided options for the treatment of Inferred material. The grades of a 12 m block are calculated from the weighted average of the upper and lower 6 m blocks.

A question arises when assigning a confidence category to a 12 m block when one of the constituent 6 m blocks is Inferred (e.g., an Inferred 6 m block below an Indicated 6 m block). The conservative approach would be to assign the lowest confidence category. For instance, if one of the 6 m blocks is Inferred, then the entire 12 m block is treated as Inferred.

For this study, AMEC agreed with a pragmatic approach such that the 12 m block was assigned the higher confidence classification. For instance, if one of the 6 m blocks was classified as Inferred and one was Indicated, then the entire 12 m block was treated as an Indicated classification for planning purposes. As a result of the chosen method of treating Inferred material during the 12 m re-blocking process, 3.6% of the total ore tonnage (3.0% of the ounces) comprises Inferred blocks that were reclassified as either Measured or Indicated blocks. The material was included in the Proven and Probable Mineral Reserves, the mining plan, and the financial analysis, but subtracted out of the Mineral Resource tabulations.

17.2.4 Mineral Reserves Statement

Mineral Reserves were estimated using L–G pit optimizations. Mineral Reserves were optimized for all Measured and Indicated blocks assuming a gold selling price of $US725/oz. An in-house program (“SMUman”) was used to evaluate the block model to identify blocks amenable to bulk mining and selective mining and apply dilution. Mill recoveries vary by rock type, domain, and degree of oxidation. The NSR cut-off for reporting was US$0.01/t.

The gold Mineral Reserves at Donlin Creek were classified in accordance with the 2005 CIM Definition Standards for Mineral Resources and Mineral Reserves, incorporated by reference in NI 43-101.

Table 17-7 summarizes the total Mineral Reserves for the Project, reported on a 100% basis. Mineral Reserves have an effective date of 31 December, 2008. The QP for the estimate is Kirk Hanson, P.E.

Table 17-7: Proven and Probable Mineral Reserve Statement, Effective Date 31 December, 2008, Kirk Hanson P.E.

Category	Tonnes (Mt)	Au (g/t)	Contained Au (Moz)
Proven	8.4	2.59	0.70
Probable	375.4	2.37	28.57
Total	**383.8**	**2.37**	**29.27**

Note:

1) Mineral Reserves reported to an Au price of US$725/oz

2) Mineral Reserves are reported on a 100% basis

3) Sums may not agree due to rounding

18.0 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORT ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

18.1 Planned Mining Operations

Throughput studies were performed during 2007–2008. Mine design and production schedules were developed for a nominal mill throughput of 19.5 Mt/a, or 53,500 t/d. Open pit mining on both 6 m and 12 m high benches provided the best Project economics. About 40% of the ore and 19% of the waste, or 22% of the total tonnage, is planned to be selectively mined on 6 m benches.

Mining operations are envisaged as 355 d/a, with 10 days allowed for delays due to winter conditions; however, the plant is provisionally scheduled to operate 365 d/a. Maximum vertical advance per phase per year is sixteen 6 m benches. Where the vertical advance rate is more than ten 6 m benches per year, some or all benches will be 12 m high so that the combined vertical development rate does not exceed ten benches per year.

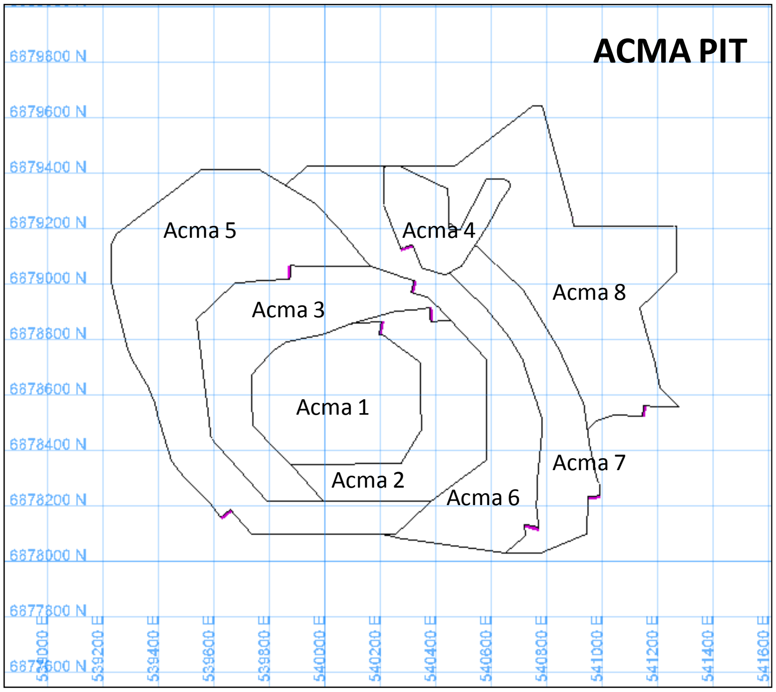
The ACMA pit has a top elevation of 268 m above sea level (asl), cuts across the American Creek drainage at 178 masl, and has a bottom elevation of 272 m below sea level (bsl). The grade of the gold mineralization in ACMA is higher than in the Lewis area. The Lewis pit is on a hill directly above and to the northeast of the ACMA pit, at elevations ranging from 436 masl to 56 m bsl.

A set of fourteen mining phases were designed, eight in the ACMA pit (Figure 18-1) and six in the Lewis pit (Figure 18-2). This sequence aims to deplete ACMA as early as possible to maximize use of the waste backfill dump designed inside the pit while minimizing deviation from the optimal economic mining sequence. The initial phases of the two pits are independent, but they partially merge later in the mine life.

Donlin Creek is envisaged to be mined by a conventional truck-and-shovel operation. Initial pioneering and pit development will be undertaken to remove overburden, develop mine access roads suitable for large mining equipment, and “face-up” the initial pit into productive set-ups for the large shovel and mining equipment.

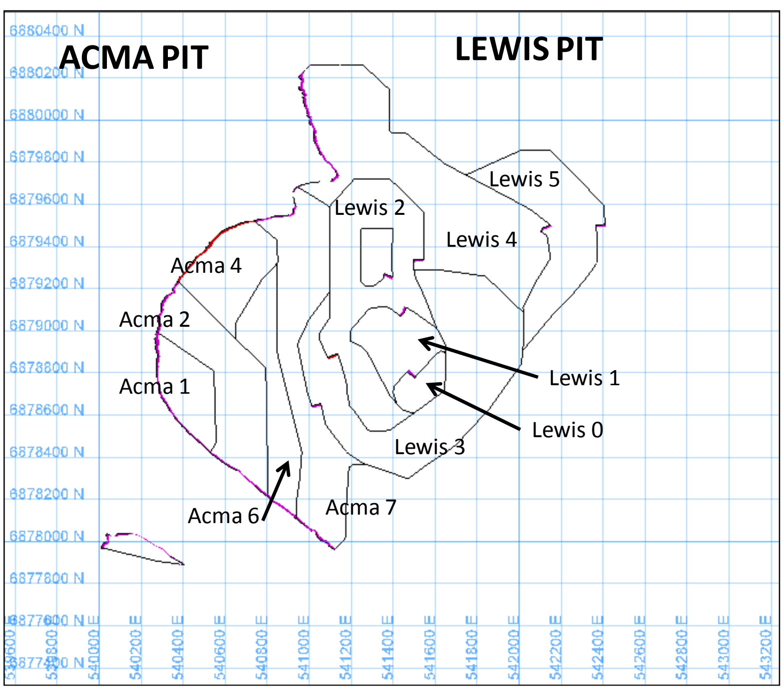
Large hydraulic shovels mining the full 12 m benches will be the primary loading equipment in zones of waste and steeply dipping ore. The same primary shovels will be used on the 6 m split benches, thereby avoiding the need for a mixed fleet of hydraulic shovels. Large 360 t capacity haul trucks will be used for transporting both ore and waste out of the pit.

Figure 18-1: ACMA Phases in Plan at 94 m Elevation



Note: Figure courtesy DCLLC

Figure 18-2: Lewis Phases in Plan at 178 m Elevation



Note: Figure courtesy DCLLC

Haul roads are designed at 10% maximum grade for uphill loaded haulage and at a maximum of 8% for downhill loaded haulage. The final road width design is 40 m.

Blasting will be required. Blast hole drilling in predominantly waste areas will be performed with nominal 251 mm diameter production drills. Ore zones will be drilled on a single 12 m bench with 200 mm diameter holes or a single 6 m bench with 140 mm diameter holes, depending on the size and continuity of the ore blocks outlined by grade control drilling. All blasting will be based on 70% emulsion/30% ANFO, which will be manufactured on site.

Support equipment will be used for road, bench, and dump maintenance and miscellaneous projects. Track dozers and rubber-tired dozers will spot loads and maintain the waste spoil dumps. A fleet of graders will maintain the roads. Crushed rock will be provided to help maintain good roads and improve truck tire life. Water trucks will spray roads and working areas during dry and dusty periods. Small backhoes will be used for ditch work and other dewatering projects. Dozers will be used on larger construction projects such as re-contouring waste dumps and spreading reclamation materials.

The projected total labour force complement for mine operations, maintenance, engineering, and contractors is 442 in 2015 (Year 1), peaks at 646 in 2025 (Year 11), and decreases to 83 in the final full year of pit operation.

18.2 Proposed Production Plan and Schedule

The operating mine life is estimated to be 20 years based on the nominal processing rate of 53,500 t/d. Mine start-up is proposed for 2014, ceasing in 2034. The processing rate is variable from period to period as a function of sulphur grade and ore hardness. To maximize plant utilization, long-term ore stockpiling is required to balance sulphur feed grades. Short-term stockpiling will also be required to handle crusher downtime and production fluctuations in the pit.

Preproduction covers the first 15 months of the mine plan, when mining activities will focus on providing sufficient ore exposure for plant start-up. Ore mined during preproduction will be stockpiled and rehandled to the mill during operations. Average production during the production stage will be 335 kt/d. The peak rate of 425 kt/d is reached in 2021 (Year 7). Mining is initially focused on the ACMA pit to access the highest-value ore. Planned production by pit is included as Table 18-1, and annual production forecasts are summarized in Table 18-2. Years shown in Table 18-2 are for illustrative purposes only, as a decision to proceed with mine construction still requires regulatory approval and approval of the DCLLC.

Table 18-1: Donlin Creek Planned Production by Area

Pit	Classification	Tonnage (kt)	Au (g/t)	Contained Au (koz)	Waste (kt)	Total Material (kt)	Strip Ratio
ACMA	Measured	3,593	2.85	330	-	-	-
ACMA	Indicated	185,928	2.52	15,061	-	-	-
ACMA	Total ACMA Measured & Indicated (MI)	189,521	2.53	15,391	1,137,102	1,326,623	6.00
Lewis	Measured	4,776	2.40	368	-	-	-
Lewis	Indicated	189,494	2.22	13,510	-	-	-
Lewis	Total Lewis Measured & Indicated (MI)	194,270	2.22	13,878	1,046,833	1,241,103	5.39
Combined Lewis & ACMA	**Measured & Indicated (MI)**	**383,791**	**2.37**	**29,269**	**2,183,935**	**2,567,726**	**5.69**

Note: Includes 10.4 Mt of Inferred resources in 6 m blocks reclassified to Measured or Indicated resources when combined as part of a 12 m block

Table 18-2: Summary of Proposed Annual Mine Production

		Total Ore									
Period	Year	Tonnes (kt)	NSR ($/t)	Au (g/t)	Su (%)	Waste NAG (kt)	Waste PAG 5 (kt)	Waste PAG 6 (kt)	Waste PAG 7 (kt)	Waste Overburden (kt)	Total Rock (kt)
2014	0	2,839	27.83	2.40	0.86	16,491	1,174	4,371	707	4,418	30,000
2015	1	21,844	29.98	2.41	1.13	76,417	6,456	12,782	844	1,687	120,031
2016	2	25,625	29.75	2.46	1.07	68,331	6,861	13,784	2,256	3,107	119,964
2017	3	24,579	29.45	2.38	1.10	74,895	7,135	11,026	970	5,402	124,008
2018	4	24,153	32.75	2.52	1.12	89,561	5,436	7,360	665	5,842	133,018
2019	5	25,144	30.14	2.39	1.09	102,014	4,573	5,649	613	7,251	145,243
2020	6	22,963	30.53	2.37	1.10	117,182	4,658	6,265	849	3,070	154,987
2021	7	19,902	30.58	2.37	1.13	117,267	6,195	9,733	916	987	155,000
2022	8	16,882	30.98	2.36	1.07	112,773	4,097	6,005	104	119	139,980
2023	9	22,658	34.89	2.58	1.13	116,328	4,582	5,564	467	392	149,990
2024	10	23,558	24.91	2.11	1.14	115,277	5,959	6,624	626	2,202	154,246
2025	11	19,427	24.81	2.08	1.09	115,788	6,004	6,479	85	1,387	149,170
2026	12	19,656	33.28	2.49	1.09	105,416	5,670	6,254	48	-	137,043
2027	13	19,755	36.60	2.65	1.13	98,796	4,603	5,513	7	-	128,674
2028	14	16,020	31.05	2.38	1.10	105,850	4,278	3,958	307	204	130,616
2029	15	11,998	25.85	2.17	1.10	110,142	4,216	4,822	205	498	131,881
2030	16	16,539	26.95	2.22	1.12	101,091	3,312	6,815	1,199	1,501	130,456
2031	17	12,316	25.33	2.13	1.16	112,199	2,749	3,571	248	917	132,000
2032	18	15,907	31.06	2.43	1.21	86,038	3,591	4,436	28	-	110,000
2033	19	15,983	28.62	2.34	1.34	43,965	3,362	3,579	111	-	67,000
2034	20	6,042	28.20	2.31	1.41	16,595	1,053	717	10	-	24,417
Total		**383,791**	**30.03**	**2.37**	**1.13**	**1,902,413**	**95,965**	**135,308**	**11,264**	**38,984**	**2,567,726**

Note: Years shown in Table 18-2 are for illustrative purposes only, as a decision to proceed with mine construction still requires regulatory approval and approval of the DCLLC.

18.3 Geotechnical

BGC provided feasibility-level slope design criteria for the Donlin Creek open pit. Slope design criteria for the bench scale (including bench face angle and berm widths), inter-ramp scale (inter-ramp angle), and overall slope scale (overall angle) were determined from geotechnical data collected and analyzed by BGC between 2004 and 2008.

Four geotechnical domains were identified:

- Domain I represents the moderately southwest dipping monocline that hosts the entire proposed Lewis pit. Major faults include the Rochelieu Ridge, Vortex, and Lo Faults. Seven minor fault sets were identified, as well as a fault set that parallels the Vortex fault. Bench face angle recommendations range from 43° to 65°, inter-ramp slope angles from 32° to 46.5°, and overall slope angles range between 32° and 46°.
- Domain II includes the west syncline limb between syncline axial trace and anticline axial trace. Folding has resulted in complex bedding sets. Faults include the Lo and Vortex Faults. Bench face angle recommendations are 65°, inter-ramp slope angles from 26° to 35.5°, and overall slope angles range between 26° and 35.5°
- Domain III comprises steeply-dipping sediments that have two bedding sets, and includes all of the sedimentary geotechnical units except the basal shale. The Lo and Vortex faults lie in the southern part of this domain, while the AC and ACMA faults divide Domain III from Domain IV. Bench face angle recommendations are 65°, inter-ramp slope angles from 28° to 47°, and overall slope angles range between 28° and 47°.
- Domain IV geotechnical units are the mid-shale, mid-greywacke, upper shale, and upper greywacke. The sediments occur as beds dipping moderately to the southwest. The mine-scale geological model interprets the bedding as dipping steeply at depth, similar to that observed in Domain III. The feasibility-level structural database, which is currently based on a limited number of exploration core holes, does not support this interpretation. Major faults identified in the areas of the two pits include the AC, ACMA, Vortex, Hello, Upper Lo, and Lo. Six minor fault sets were identified, as well as sets that parallel the AC Fault and sub-parallel the Lo Fault. Bench face angle recommendations are 65°, inter-ramp slope angles from 30.5° to 50°, and overall slope angles range between 30.5° and 45°.

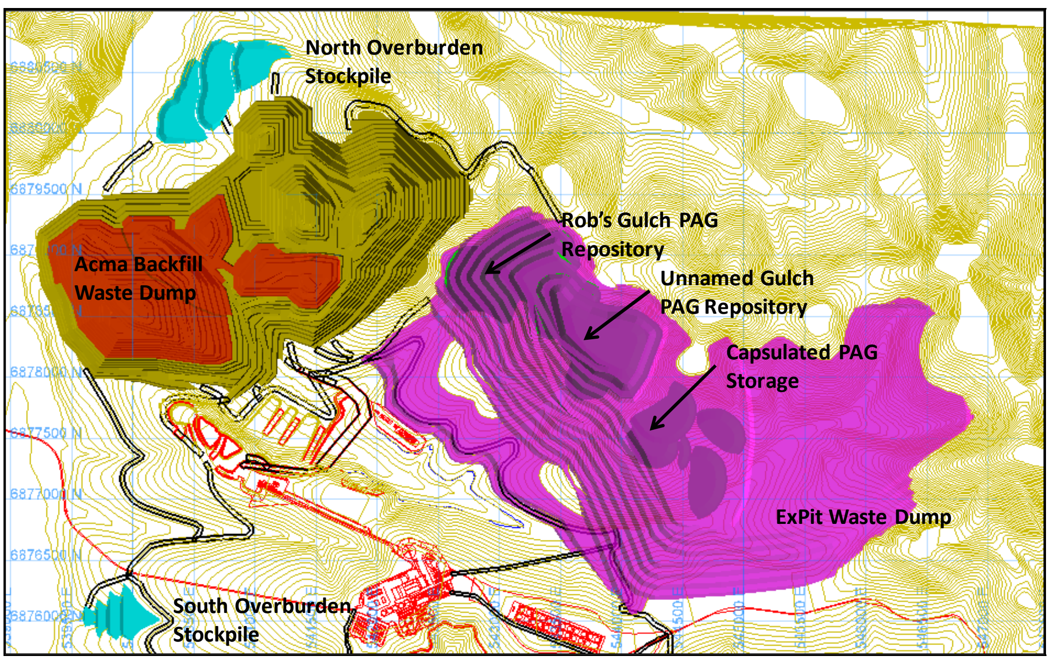
Two areas were noted that will require detailed geotechnical management, the northeast wall of the Lewis pit, and the south–southwest wall of the ACMA pit. All slopes require depressurization. Those that need complete depressurization on the overall slope scale to minimize the potential of rock mass failures include:

- South wall of the ACMA pit
- South wall of the Lewis pit
- Footwall slope of the Lewis pit.

18.4 Waste Dumps

Waste rock from open pit mining will be placed in an ex-pit waste rock facility (WRF), in the American Creek valley, east of the pit area, or in a backfill dump in ACMA (Figure 18-3). The ultimate footprint of the facility covers an area of approximately 9.6 km^2. With the elevation of the top lift of the dump at approximately 550 masl, the maximum dump height will be about 350 m, and the maximum thickness about 290 m. The WRF will be developed entirely from the bottom up. Construction of the first lift will begin at the start of the preproduction period. Most of the WRF will be constructed in 30 m lifts.

Figure 18-3: Proposed Donlin Creek Waste Dumps



Note: Figure courtesy DCLLC. Acma as shown on plan = ACMA deposit

The potential magnitude of flow in the American Creek drainage, as well as discharge from springs in the valley floors, warrants the construction of an engineered rock drain system below the WRF, including connecting secondary rock (finger) drains in the smaller contributing drainages. The rock drains were sized to contain the peak instantaneous flow associated with the 100-year return period, 24-hour duration rainfall event for American Creek.

Sufficient overburden will be stored separately for use in final site reclamation; the remainder will be dumped into the WRF or used for construction and concurrent reclamation. A total of 1.69 Gt of waste will be stored in the WRF and another 404 Mt in the ACMA backfill dump. Backfilling will commence in 2029 (Year 15) and continue until the end of mine life.

A total of 38 Mt of in-pit overburden will be mined at Donlin Creek, of which 7.7 Mt of peat and loess and 9.6 Mt of colluvium/terrace gravel will be stockpiled over the LOM to meet site reclamation requirements. The remainder will be stored within the WRF. Where overburden directly removed from the pit is unavailable, it will be reclaimed from the stockpiles. Some 17.3 Mt of overburden will stored in overburden stockpiles.

Waste rock was characterized by its potential for acid generation and was assigned reactivity categories. Categories 1 to 4 are non-acid-generating (NAG), and categories 5 to 7 are potentially acid-generating (PAG). Waste rock consists of NAG and PAG rock from the ACMA and Lewis pits. PAG-7 rock will potentially start producing acid in less than a few years, PAG-6 in less than a decade, and PAG-5 after several decades. PAG-5 rock will be blended with NAG rock when placed in the WRF; the NAG rock has enough neutralizing potential to prevent the PAG-5 waste from producing acid. PAG-6 waste will initially be placed in encapsulated cells in the WRF. Water infiltration into this cell will be minimized by a cover of compacted colluvium or terrace gravel.

The PAG-7 waste will ideally be used to construct the water reclaim structure in the tailings impoundment. This point will require addressing during detailed design and operational scheduling. Additional PAG-7 waste will be stockpiled in the long-term ore stockpile area. The stockpiled PAG-7 waste will then be rehandled into the ACMA pit below the final pit lake water level.

The WRF was designed to meet or exceed a factor of safety (FS) of 1.5 under static loading conditions and an FS of 1.1 under seismic (pseudo-static) loading. The stability of the WRF exceeds these design criteria.

Concurrent reclamation of the WRF will be undertaken during operations as area becomes available.

18.5 Hydrology

18.5.1 Proposed Water Management

The main objectives of the water management plan for the Project are to minimize or eliminate the need for treatment and discharge of contact water during mine construction, operations, and closure; to achieve the pit-slope depressurization requirements; and to provide adequate quantity and quality of water supply to the mill.

The Project is expected to operate with an overall water surplus, based on the large catchment areas of the American Creek and Anaconda Creek drainage basins, which will yield large volumes of water during the spring and summer (April to October) from rainfall, snowmelt run-off, and groundwater base flow.

ACMA pit will transect American Creek near its confluence with Crooked Creek in Year 1 of operations, and the WRF will ultimately occupy a significant proportion of the remaining American Creek basin upstream from the pit. Contact water will be stored behind a dam in American Creek, and tailings will be stored in the adjacent Anaconda Creek basin. Staged diversion structures will be required to divert fresh water out of the Project area during construction, operations, and closure.

18.5.2 Contact Water Dam and Diversion Dams

The contact water dam (CWD) will be constructed in the American Creek valley downstream of the waste rock dump to collect runoff and seepage water from the WRF. Water that collects in open sumps within the pit will be pumped into the contact pond. An ancient landslide mass covering an area of approximately 50,000 m^2 was interpreted adjacent to the CWD on the southwestern flank of the American Creek valley. The landslide will be stabilized with a berm constructed of good-quality waste rock.

The CWD is sized to contain the probable maximum precipitation 24 hour duration rainfall for the entire American Creek catchment, plus emergency freeboard. The stability of the CWD yields static and pseudostatic factors of safety of 1.56 and 1.18, respectively.

The CWD will be breached and excavated down to original grade upon closure at the end of operations. The exposed rockfill below grade will be covered with growth medium and revegetated.

Three sequenced freshwater diversion dams will be constructed in the American Creek basin to manage fresh water upstream of the WRF and reduce water handling. The diversion dams are sequenced throughout the mine life to correspond to the WRF construction. The freshwater diversion dams are sized to contain the 10-year return period snowmelt event and to pass the peak instantaneous flow associated with the 100-year return period, 24-hour duration rainfall.

18.5.3 Proposed Construction Water Management

Water management objectives during construction are to eliminate the need to treat and discharge contact water; to ensure an adequate supply of water for commissioning and start-up of the process plant; and to eliminate the need to store water in the TSF until immediately before mill start-up. To meet these objectives, the contact water dam (CWD), diversion dam, and Rob's Gulch diversion dam need be constructed by the beginning of 2014 (Year -1). Pit dewatering and pit dewatering treatment infrastructure will need to be installed approximately six months in advance of this timeframe. The CWD will intercept and store runoff from the Project areas disturbed by mine development and construction activities, including the pit and WRF.

18.5.4 Proposed Operations Water Management

The operations water management system addresses runoff from the plant site; runoff and pit dewatering flows from the Lewis and ACMA open pits; runoff and seepage from the WRF; required storage and pumping capacities for the CWD and associated freshwater diversion structures in American Creek; and required storage and pumping capacities for the lined TSF and associated upstream freshwater reservoirs and channel diversion in Anaconda Creek.

In American Creek, runoff from the WRF will be captured in the CWD approximately 500 m upstream of the pit. Surface runoff from the pit will be directed to the CWD. Pit dewatering groundwater from in-pit and perimeter dewatering wells will primarily be directed to the CWD for use in the mill process. In surplus water conditions, groundwater flows from the perimeter wells can be routed to the operations water treatment plant (WTP) for treatment if necessary and discharge to Omega Gulch. Two stages of non-contact water diversion dams in the mid and upper reaches of American Creek will collect runoff above the WRF. This water will be pumped around the WRF and discharged into Omega Gulch.

Run-off to the TSF in Anaconda Creek will be minimized by the construction of staged diversion channels and two upstream dams that form two freshwater reservoirs (north and south). A minimum pool of water will be maintained in these reservoirs to provide make-up water for the process plant on an as-needed basis; however, water collected in these reservoirs will generally be pumped through a lined diversion channel around the north side of the TSF and discharged into Anaconda Creek. A seepage collection and pump-back system was included downstream of the TSF for both operational and closure conditions as a safeguard in case seepage water quality was unacceptable for discharge without treatment.

During operations water unsuitable for discharge without treatment will be pumped back to the TSF; similarly, if seepage water quality is unacceptable during the closure and post-closure periods, a contingency allows for this water to be pumped to the pit lake.

18.5.5 Proposed Closure Water Management

Infrastructure considered for closure in American Creek includes the backfilled ACMA pit a covered WRF, and a pit lake. In the Anaconda drainage, a cover will be placed over the tailings, and runoff from the reclaimed TSF will drain into the upper north reservoir. Tunnels will be constructed between the north reservoir and the south reservoir, and from the south reservoir to Crevice Creek.

The TSF facility will be reclaimed over approximately 43 years. Supernatant water from the TSF will be pumped immediately to the pit lake at closure, and the TSF will be progressively covered over four years.

18.6 Proposed Tailings Storage

The TSF in the Anaconda Creek basin will be a fully lined impoundment with cross-valley dams at both the upstream (“upper dam,” comprising upper north and upper south) and downstream (“main” dam) ends.

All tailings dams will be constructed of compacted rock fill using the downstream method with a composite liner on the upstream face. The tailings impoundment footprint will be lined with a linear low density polyethylene liner over a layer of broadly graded silty sand and gravel acting as low permeability bedding material and providing secondary containment.

Material for construction will be sourced from the plant site and fuel farm during initial construction and from the open pit for the later raises during operations.

Based on the flood and tailings storage requirements, the starter dams are required to store one year of tailings, plus flood and freeboard, will be 52 m high for the main dam, while the upper north and upper south dams will be 16 m and 12 m, respectively. Ultimate heights will be 144 m for the main dam and 105 m for the upper dam, measured from the downstream toe to the crest. The TSF will have an ultimate capacity of 311.43 Mm^3, corresponding to an ultimate impoundment surface area of 549 ha. The total catchment area of the TSF will be 705 ha.

The TSF was designed to meet appropriate dam safety guidelines. The TSF inflow design flood was the 200-year return period snowmelt and 24-hour probable maximum precipitation. The stability of the tailings dams yielded static and pseudo-static factors of safety of 1.5 and 1.15, respectively. The TSF was designed to withstand the maximum credible earthquake.

Water dams are required during the construction period and initial years of operation to protect the lined upstream faces of the upper north and south tailings starter dams from a significant flood event, to provide a reliable source of fresh water during operation of the process plant, and to minimize runoff to the TSF. The water dams will be incorporated into the downstream toe of the upper dams and are planned to be constructed simultaneously with the starter dams before tailings placement. The north and south freshwater reservoirs will reach maximum depths of 19 m and 8.5 m, respectively. Based on storage requirements, the north water dam will be 42 m high and the south water dam 33 m high.

18.7 Infrastructure

The Project is a greenfields site. In addition to the proposed plant site at the mine the main proposed development sites are the wind farm, an airstrip, barge terminals at Bethel and BTC, and an access road connecting BTC to the mine site (see Figure 4-3).

18.7.1 Planned Off-site Infrastructure

The entire road will be new construction in an untracked region, with no passage through or near any settlements or communities, and no junctions with any existing road system. Forty-three stream crossings were identified along the BTC route. Of these, eight require bridges directly along the road, and one more crosses Getmuna Creek to access the major Getmuna Flats material site. Bridge lengths vary from 10 m to 35 m.

The primary purpose of the road is to transport freight by mostly conventional highway tractors and trailers. However, critical elements of the design will be dictated by specific oversize and overweight loads associated with mine facility construction. Only mine support traffic will use the road, and the design assumes that mine operations will control and manage traffic on the road.

The fuel pipeline from the BTC port site to the mine site was incorporated into the road alignment. The pipeline will be buried where it passes though areas of thaw-stable ground and supported above ground on piled foundations where the ground is susceptible to instability.

18.7.2 Planned Site Infrastructure

Planned site infrastructure (see Figure 5-2) comprises:

- Access roads
- Airstrip
- Accommodation camp
- Plant site and fuel storage
- Primary and pebble crushers
- Coarse ore conveyor and coarse ore stockpile
- Concentrator
- Water treatment plants
- Boiler house
- Utilidors and access walkways
- Waste and tailings storage facilities
- Truck shop, truck wash, workshops and vehicle repair facilities
- Assay laboratory
- Administration facilities
- Change-rooms

The plant site and fuel storage compound are located in the Anaconda valley, above the tailings storage area. This arrangement contains the process areas within the Anaconda and American Creek valleys, with essentially no impact on Crooked Creek.

The primary crusher is located on a ridge on the south side of American Creek. This location is compatible with the mining plan, haul road layouts, and ultimate pit limits as well as the location of the contact water dam and contact water pond. The crusher was orientated to make use of the southern slope of the ridge, minimize the length of the conveyor, and permit the design of the vertical and horizontal alignment to tie into the coarse ore stockpile at the plant site. The process plant was orientated on the plant site to take advantage of the natural topography, with the long axis of the plant following the slope of the rounded hill to the south.

Power and communication descriptions are included in Section 5 of this Report.

18.8 Environmental Considerations

The environmental work completed to date on the Project is discussed in Section 4.

18.8.1 Proposed Mine Closure Plan

In its ongoing efforts at Donlin Creek, the DCLLC recognizes that its responsibility to the communities of the Yukon–Kuskokwim Delta extends beyond exploration, development, and operations to the even more critical stage of mine closure. Since the very inception of the Donlin Creek exploration program, there was a conscious effort to design exploration, development, and operations for closure. By “designing for closure” at a very early stage in the life of a project, the potential cumulative impacts on the physical resources of the area and the post-closure impact on local communities can be addressed. Realizing that the Project clearly has a role to play in contributing to the long-term sustainability of the communities surrounding the Project, planning for closure in collaboration with state and local authorities is essential.

In addition to the basic goal to reclaim disturbances associated with mining, processing, and ancillary support facilities in a manner compatible with the designated post-mining land use, careful planning will minimize the area affected by the operations. During operations, whenever possible, concurrent reclamation will be performed in those areas that are no longer required for active mining.

The DCLLC will complete a Closure Social Impact Assessment (CSIA), targeted for three years prior to closure of any operation. While appropriate planning of sustainable community projects support the long-term sustainability of nearby communities, the CSIA will focus on the net positive benefits from the operation and identify alternative uses for the skills and infrastructure that were developed during operations.

Closure planning also includes assisting employees with identifying new career opportunities as appropriate. Where possible, the goal is to offer continuing employment opportunities or alternatively, offer out-placement services to employees who are not able to relocate.

Reclamation and closure of the Project falls under the jurisdiction of the Alaska Department of Natural Resources (ADNR) Division of Mining, Land, and Water Management; the Alaska Department of Environmental Conservation (ADEC); the U.S. Army Corps of Engineers (USACE); and the U.S. Environmental Protection Agency (USEPA). The Alaska Reclamation Act (Alaska Statute AS 27.19) is administered by the ADNR and applies to state, federal, municipal, and private land and water subject to mining operations. Except as provided in an exemption for small operations, a miner may not engage in a mining operation until the ADNR has approved a reclamation plan for the operation.

The ADNR may enter into a cooperative management agreement with the federal government or other state agencies to implement a requirement of the Reclamation Act or a regulation adopted under it. The Closure and Reclamation Plan for a mining project that involves both federal and state permits requires joint approval. Financial surety for mine closure and reclamation is a requirement of federal and state agencies. ADNR has historically been the agency that holds the surety for both. The approved plan and associated surety are reviewed and revised at five-year intervals. The landowner participates in the planning process with regard to determining and concurring with the designated post-mining land use.

A modified version of the Barrick Reclamation Cost Estimator (BRCE) was used to develop reclamation and closure cost estimates. Estimated costs are based on the Project as currently presented, with the realization that closure and reclamation plans and costs will be routinely updated throughout the detailed design phase and during operations.

The final reclamation cost estimate is $96.1 million. This amount is included in a “Reclamation, Closure and Post-Closure Maintenance Trust Fund” model prepared to determine the funding that is required to generate sufficient cash flow to cover costs for tunnel construction from Anaconda creek to Crevice Creek, capital to construct the WTP, perpetual water treatment, and associated facility and access maintenance. The total amount to cover reclamation and closure costs and post-reclamation and closure maintenance is estimated at $7.44 million, paid annually over the three year construction and 20 year LOM.

Various pit-lake filling options were modelled to assess filling rates, physics, and geochemistry, with the intent of ultimately predicting the quality of water that would eventually discharge from the ACMA pit lake into the receiving environment, approximately 45 years after cessation of mining operations.

The WTP will use chemical precipitation technology to target dissolved elements such as arsenic, antimony, and manganese. Since the water quality predictions also indicate elevated levels of selenium and sulphate, reverse osmosis technology will be used to decrease levels to below discharge limits. Reverse osmosis represents the best available technology for the removal of selenium. The sludge from the WTP will be a chemically stable material and will be sent to the bottom of the open pit for final storage. It is currently anticipated that the water stored in the pit after closure will not meet the water quality criteria for a few parameters and will require treatment before discharge into Crooked Creek.

18.9 Markets

The Limited Liability Company Agreement of the DCLLC dated 1 December 2007 provides in Section 11.1 that “The Company shall distribute to each Member in kind its share of all Processed Products in accordance with such Member’s Percentage Interest at the time of such distribution…” The marketing plan, therefore, is for the members of the DCLLC to take in kind their respective shares of the gold production, which they can then sell for their own benefit. Under the agreement, the Manager shall give the Members prompt Notice in advance of the delivery date upon which their respective shares of gold production will be available.

Section 11.3 also provides that neither Member shall have any obligation to account to the other Member for, nor have any interest or right of participation in, any profits or proceeds, nor have any obligation to share in any losses from futures contracts, forward sales, trading in-puts, calls, options or any similar hedging, price protection or marketing mechanism employed by the other Member with respect to its proportionate share of the production.

Under certain conditions provided for in the agreement, the Manager can sell the Members’ share of the production. If necessary in such special circumstances, gold can be readily sold on numerous markets throughout the world and it is not difficult to ascertain its market price at any particular time. Since there are a large number of available gold purchasers, the Members would not be dependent upon the sale of gold to any one customer. Gold can be sold to various gold bullion dealers or smelters on a competitive basis at spot prices.

Spot prices are determined by open markets. The “London Gold Fixing” is the procedure by which the price of gold is set on the London market by five members of the London Gold Pool (who are all members of the London Bullion Market Association). The London Gold Fixing is designed to fix a price for settling contracts between members of the London bullion market but is internationally recognized as a benchmark for gold prices and is used in the pricing of the majority of gold products throughout the world’s markets.

It is expected that selling contracts for NovaGold’s share of the gold production will be typical of, and consistent with, standard industry practice, and be similar to contracts for the supply of doré elsewhere in the world.

18.10 Taxation

Taxes that may be levied on the Project can be summarized as follows:

- Federal Income Tax – the greater of the U.S. Regular Tax of 35% or Alternative Minimum Tax of 20%
- Alaska State Income Tax – 9.4% of income over $90,000
- Alaska State Mining License Tax – 7% of taxable mining income, less depletion. There is a 3.5-year tax holiday on the mining license tax.

Income tax becomes payable after deductions for capital allowances. Taxation considerations are included in the cash flow analysis in Section 18.13.

18.11 Capital and Operating Cost Estimates

18.11.1 Capital Costs

The feasibility study capital cost estimate was developed in accordance with Association for the Advancement of Cost Engineering (AACE) Class 3 requirements, consisting of semi-detailed unit costs and assembly line items. The level of accuracy for the estimate is ±15% of estimated final costs, per AACE Class 3 definition.

Costs expressed in third quarter (Q3) U.S. dollars were subsequently de-escalated using a de-escalation model to adjust the estimate to fourth-quarter (Q4) 2008 U.S. dollars. No allowances are included for escalation through construction, interest during construction, taxes, or duties.

The de-escalation model determines potential savings to the Project due to the global recession and downturn of the world economies since the Q3 2008 pricing. There was a significant reduction in world commodity prices in Q4 2008, particularly in metal prices within the mining industry. Costs in the estimate that were priced in either Q4 2008 or January 2009 U.S. dollars were not included in the de-escalation model. The model provides a Monte Carlo-type simulation that also includes currency impacts. The model looks at the minimum line and the base line estimate (Q3 2008 U.S. dollars) as the maximum. The result, depending on which probability factor is used, will determine the outcome. A probability factor (P_{50}) was used for de-escalation in the estimate.

The total estimated cost to design and build the Project is $4,481 million, including an Owner-provided mining fleet and self-performed pre-production mine development.

Capital costs are summarized in Table 18-3. Sustaining capital requirements total $803 M and are presented in Table 18-4.

Table 18-3: Summary of Capital Costs by Major Discipline

Discipline	Cost ($000)
Direct Costs	
Civil	383,298
O/L Piping	124,804
Mining	431,636
Concrete	183,043
Structural	181,293
Architectural	105,990
Mechanical	1,104,979
Piping	190,137
Electrical	360,026
Instrumentation	60,641
Coatings	14,986
Total Direct Costs	3,140,833
Indirect Costs	
Owner's Costs	191,921
Project Indirect Costs	925,821
Total Indirect Costs	1,117,742
Subtotal	4,258,575
Contingency @ P50	394,625
Total Project Cost	**4,653,200**
De-escalation @P50	(172,600)
Net Project Cost	**4,480,600**

Table 18-4: Sustaining Capital Requirements ($)

Area	Area Name	2009	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024
1130	Mine Haul Road			1,167,110		256,923	933,664	-	-	-	859,607	-
1210	Mine Mobile Equipment		-	9,249,081	28,570,248	53,802,819	42,252,571	48,459,283	6,777,206	3,863,830	14,898,310	10,564,735
1310	Mine Dewatering and Drainage		1,921,296	-	2,603,397	-	1,761,268	3,780,281	1,218,871	963,639	1,445,441	870,376
2250	American Creek Waste Dump Preparation		-	-	-	-	-	1,019,876	-	1,667,516	-	-
2500	Airstrip		-	-	-	72,000	-	-	72,000	-	-	72,000
4110	Tailings Storage Facility		69,553,047	7,596,682	-	-	53,556,097	-	-	-	45,666,090	-
6110	Administration Building/Dry		-	200,000	200,000	200,000	200,000	200,000	200,000	200,000	200,000	200,000
6410	Accommodations Complex		-	-	-	-	-	-	-	-	-	-
6710	Truck Shop		-	-	-	-	-	14,922,071				
6810	Plant Mobile Equipment		-	4,860,381	1,538,216	1,076,892	3,329,604	4,105,066	4,265,679	6,698,283	3,692,535	793,202
7220	Mobile Equipment Bethel		-	-	-	-	-	110,376	-	3,565,375	-	-
7310	BTC		-	-	-	10,000	-	18,420	-	-	3,900	10,000
7320	Mobile Equipment BTC		-	-	-	-	-	149,089	-	5,001,524	-	-
7230	Kuskokwim River		-	-	-	-	25,070	-	-	-	21,170	-
9200	EPCM		839,690	83,969	83,969	83,969	839,690	83,969	83,969	-	839,690	
9320	Construction Catering		2,179,796	113,132	14,527	1,294	1,708,017	102,037	21,343	8,806	1,450,660	9,950
9500	Freight		198,591	799,811	1,522,339	2,772,641	2,726,426	3,236,060	597,122	1,075,448	1,307,196	595,184
	Total		74,692,421	24,070,166	34,532,696	58,276,538	107,332,406	76,186,528	13,236,190	23,044,421	70,384,599	13,115,447
	De-Escalation @ P 50	(47,000,000)										
	Net Total											

Note: Years shown in Table 18-4 are for illustrative purposes only, as a decision to proceed with mine construction still requires regulatory approval and approval of the DCLLC.

Table 18-4 cont: Sustaining Capital Requirements ($)

Area	Area Name	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	Total
1130	Mine Haul Road	-	-	65,440	2,780,805	-	-	-	-	-	-	6,063,549
1210	Mine Mobile Equipment	46,371,946	126,921,470	4,312,482	5,760,590	14,845,040	12,627,670	15,794,784	58,800	-	-	445,130,865
1310	Mine Dewatering and Drainage	1,425,482	1,154,435	1,463,777	1,698,755	1,067,869	1,018,356	1,094,035	1,386,595	1,062,942	570,115	26,506,932
2250	American Creek Waste Dump Preparation	-	-	-	-	-	-	-	-			2,687,392
2500	Airstrip	-	-	-	-	-	-	-	-			216,000
4110	Tailings Storage Facility	-	-	42,563,880	-	-	-	32,649,358	-			251,585,155
6110	Administration Building/Dry	200,000	200,000	200,000	200,000	200,000	-	-	-			2,800,000
6410	Accommodations Complex	-	-	-	-	-	-	-	-			-
6710	Truck Shop											14,922,071
6810	Plant Mobile Equipment	3,050,752	-	2,297,935	1,258,784	4,611,958	645,008	-	-			42,224,295
7220	Mobile Equipment Bethel	110,376	-	-	-	2,874,640	110,376	-	-			6,771,143
7310	BTC	18,420	-	10,000	-	-		10,000	-			80,740
7320	Mobile Equipment BTC	149,089	-	-	85,679	5,068,614	149,089	-	-			10,603,083
7230	Kuskokwim River	-	-	-	-	-	-	-	-			46,240
9200	EPCM			839,690				839,690				4,618,,295
9320	Construction Catering	27,810	18,955	1,349,220	21,194	20,646,	18,457	1,048,233	20,000	20,547	16,716	8,171,338
9500	Freight	2,519,050	6,369,406	689,163	547,994	1,393,653	690,135	898,966	34,395	13,362	11,086	27,998,028
	Total	53,872,925	134,664,266	53,791,588	12,353,801	30,082,421	15,259,091	52,355,066	1,499,790	1,096,851	597,917	850,425,127
	De-Escalation @ P 50											(47,000,000)
	Net Total											803,425,127

Note: Years shown in Table 18-4 are for illustrative purposes only, as a decision to proceed with mine construction still requires regulatory approval and approval of the DCLLC.

18.11.2 Operating Costs

The operating cost estimates were assembled by area and component, based on estimated staffing levels, consumables, and expenditures, according to the proposed mine plan and process design. LOM operating costs are shown in Table 18-5 and annual operating costs in Table 18-6. Operating costs were prepared in 4Q 2008 U.S. dollars with no allowances for escalation, sales tax, import duties, or contingency.

Table 18-5: LOM Operating Cost ($000)

Area	Total LOM	$/t Milled	$/t Mined	$/oz
Mine Operations	5,226,143	13.62	2.08	200
Processing Operations	5,664,194	14.76	2.26	216
Administration	589,596	1.54	0.24	23
Refining	43,858	0.11	0.02	2
Total	**11,523,790**	**30.03**	**4.60**	**440**

Table 18-6: Annual Operating Costs ($000)

Year	Mining	Processing	Admin	Refining	Total	$/t Milled	$/t Mined	$/oz
1	167,979	192,156	30,461	1,773	392,369	32.21	4.36	371
2	218,836	279,989	29,492	2,638	530,955	28.62	4.43	337
3	246,977	287,613	29,279	2,572	566,441	29.58	4.57	369
4	272,433	285,862	29,011	2,746	590,052	30.61	4.44	360
5	327,788	286,810	28,840	2,663	646,101	32.95	4.45	406
6	329,142	289,350	27,633	2,560	648,685	32.56	4.19	424
7	308,641	276,280	28,107	2,278	615,307	33.14	3.97	452
8	320,402	274,666	28,344	2,085	625,496	33.90	4.47	502
9	325,752	289,704	28,025	2,764	646,244	32.57	4.31	392
10	312,814	287,946	27,535	2,266	630,562	32.43	4.09	466
11	321,978	284,991	27,829	1,985	636,784	32.74	4.27	537
12	306,923	286,361	28,448	2,405	624,137	31.75	4.55	435
13	313,602	288,920	28,561	2,581	633,663	32.08	4.92	411
14	317,727	275,637	28,574	1,991	623,928	33.68	4.78	525
15	270,787	274,334	28,622	1,547	575,291	31.45	4.36	623
16	232,201	285,905	28,495	1,904	548,505	28.31	4.20	483
17	222,243	285,696	28,459	1,597	537,996	27.71	4.08	564
18	177,261	286,603	29,243	2,011	495,119	25.41	4.50	412
19	142,562	286,309	30,785	1,928	461,584	23.46	6.89	401
20	71,023	283,165	32,424	1,299	387,910	19.68	15.89	500
21	19,072	75,896	11,429	265	106,662	19.56	N/A	675
Total	**5,226,143**	**5,664,194**	**589,596**	**43,858**	**11,523,790**	**30.03**	**4.60**	**440**

The LOM direct production mining cost is shown in Table 18-7. Preproduction mining costs were capitalized and included in the capital cost estimate.

Table 18-7: LOM Mining Costs ($000)

Area	Total	$/t Mined	$/t Milled
General	395,471	0.16	1.03
Drilling	626,295	0.25	1.63
Blasting	547,248	0.22	1.43
Loading	749,465	0.30	1.95
Hauling	2,019,834	0.81	5.26
Support	567,921	0.23	1.48
G&A Reallocation	319,909	0.13	0.83
Total	**5,226,143**	**2.08**	**13.62**

The LOM direct processing costs are shown in Table 18-8 and the LOM general and administrative (G&A) costs in Table 18-9. Some costs included in G&A were allocated back to the mine and process departments to the extent that these costs can be reasonably related to the respective department (i.e., based on direct usage, percentage of total labour hours, or percentage of volumes shipped).

Table 18-8: LOM Processing Costs ($000)

Item	Total	$/t Milled
Labour	438,203	1.14
Reagents and Consumables	1,621,878	4.23
Power	3,069,843	8. 0
Maintenance Supplies	357,876	0.93
G&A Reallocation	176,394	0.46
Total	**5,664,194**	**14.76**

Table 18-9: LOM General and Administrative Costs ($000)

Cost Centre	LOM Total	Allocations	LOM Net	$/t milled
Logistics	258,816	217,166	41,650	0.11
Camp & Catering	223,926	163,344	60,582	0.16
Finance & Administration	197,387	22,254	175,133	0.46
Insurance	139,576	-	139,576	0.36
Site Maintenance & Mobile Equipment	117,867	-	117,867	0.31
Aviation	77,428	56,450	20,979	0.05
Power	45,725	37,089	8,635	0.02
Environmental	25,175	-	25,175	0.07
Total	**1,085,899**	**496,303**	**589,596**	**1.54**

18.12 Financial Analysis

The results of the economic analysis represent forward-looking information that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

The overall economic viability of the Project was evaluated by both discounted and undiscounted cash flow analyses, based on the engineering studies and cost estimates discussed in this study. Assumptions in the model comprised:

- For discounted cash flow (or NPV) purposes, the model is based from 1 January 2009. Estimates were prepared for all the individual elements of cash revenue and cash expenditures for ongoing operations.
- Estimated cash flows from revenue are based on a gold price of $725/oz as provided by the DCLLC, which is the price used for reporting the 2008 Mineral Reserves. The pit has also been optimized at the same gold price of $725/oz. At the effective date of this Report, gold was trading at around $950/oz.
- Recovery is estimated to average 89.5% over the LOM based on work and testing performed for feasibility study and feasibility study update purposes.
- Doré refining and shipping charges were estimated at $0.95/oz based on actual refining charges for Barrick’s Goldstrike operations and a quotation for transportation and insurance costs from the Donlin Creek mine site to a U.S.-based refinery. An additional 0.1% of gold produced from the mine is included in refining costs. This amount represents the refiner’s estimate of the loss of gold that will occur during the refining process.
- The current hydrometallurgical process selection renders any contained silver into a greater refractory state, which provides less than 10% silver recovery through standard metal leaching. As a consequence, no silver credit was applied to the Project.
- Assets will be sold over the course of the mine life, when they are no longer required for project-based work, as well as at the end of the mine life. Total recovered value from these sales is estimated at $33 million.
- Reclamation and closure costs were estimated at $96 million and are primarily incurred in the first five years after the mine closes (2035 to 2039), although some expenditures begin immediately after construction and during operations with concurrent reclamation. The funding amount that is required to generate sufficient cash flow to cover costs for tunnel construction from Anaconda Creek to Crevice Creek, employee severance payments, capital to construct the water treatment plant (WTP) for perpetual water treatment, and associated facility and access

maintenance, and closure costs is estimated at $7.44 million provided annually over the 3-year construction and 21-year LOM, for a total of $179 million.

- During the non-shipping season (October through May), the Project-owned barging fleet will be leased for other haulage uses. The total net revenue determined from this leasing arrangement is estimated at $166 M over the period 2012 to 2035. Of this amount, $10 million earned during preproduction was credited against initial capital costs. The remaining $156 million is credited against operating costs.
- Inventory, including 85% of consumables, is included in the financial model as cash outflows in the year before start-up of operations. Other warehouse inventory, excluding capital spares, is estimated at approximately $25.3 million by the DCLLC and was developed from first principles based on the value and quantity drivers of warehouse inventory held by Barrick's Goldstrike operation.

The Project is expected to generate net cash flows of $1.1 billion and yield an internal rate of return (IRR) of 2.3%, under a long-term gold price assumption of $725/oz. Table 18-10 summarizes the base case performance statistics and two alternate cases at gold prices of $900/oz and $1,000/oz.

The base case NPV (5%) of the Project is a negative $733 million. At recent gold prices of $1,000/oz (Alternative Case 2) the project has an NPV (5%), after tax, of $1,674 million and an after-tax IRR of 10.2% (Table 18-11). Table 18-12 lists the sensitivities of after-tax net cash flow (NCF), NPV (5%), and IRR to variations in the gold price.

From the base case of gold at $725/oz and oil at $75/barrel, each $1/barrel increase in the price of oil requires approximately a $1.50/oz increase in the price of gold to offset the impact. Project sensitivity to oil price is shown in Table 18-13.

Sensitivities to changes in the gold price, operating costs, and capital costs (-20% to +20%) are illustrated in Figure 18-4 for the net cash flow and Figure 18-5 for the IRR. The base case gold price assumed in the sensitivity analysis is $725/oz. For the purposes of the sensitivity analysis, the DCLLC assumed that the Project sensitivity to changes in gold grades was mirrored by the sensitivity of the Project to changes in the gold price.

Table 18-10: Donlin Creek Project Financial Summary (Base Case US$725/oz Au)

Donlin Creek[1]	Unit	LOM	$/oz	$/t milled	$/t mined
Total Mined	Mt	2,567.7	-	-	-
Ore Milled	Mt	383.8	-	-	-
Strip Ratio (waste:ore)	t:t	5.69	-	-	-
Gold Grade	g/t	2.37	-	-	-
Contained Gold	Moz	29.269	-	-	-
Average Gold Recovery	%	89.5	-	-	-
Recovered Gold	Moz	26.184	-	-	-
Mine Life	Years	21			
Oil Price	$/barrel	75			
Revenue	$M	18,983	725		
Mining Cost	$M	5,226	200	13.62	2.08
Process Cost	$M	5,664	216	14.76	2.26
G&A	$M	590	23	1.54	0.24
Refining	$M	44	2	0.11	0.02
Operating Costs	$M	11,524	440[2]	30.03	4.60
Royalties	$M	693	26	1.81	0.28
Total Cash Costs	$M	12,217	467	31.84	4.87
Other Revenue	$M	(156)	(6)	(0.41)	(0.06)
Depreciation (Excluding Sunk Costs)	$M	5,242	200	13.66	2.09
Trust Fund	$M	179	7	0.47	0.07
Total Production Costs	$M	17,481	668	45.55	6.97
Cash Taxes	$M	402	15	1.04	0.16
Working Capital, Net	$M	(2)	-	(0.01)	0.00
Total Costs, Including Taxes and Working Capital[3]	**$M**	**17,881**	**683**	**46.59**	**7.13**
	Unit	**Base Case**	**Alternative Case 1**	**Alternative Case 2**	
Gold Price	$/oz	725	900	1,000	
Oil Price	$/barrel	75	75	75	
Average Annual Cash Flow (EBITDA)					
First Full 5 years	$ (M)	521	790	944	
First Full 10 years	$ (M)	415	663	805	
Average Total Cash Costs					
First Full 5 years	$ per ounce Au	394	398	400	
First Full 10 years	$ per ounce Au	442	448	451	
Life of Mine	$ per ounce Au	467	473	477	
Financial Results					
Undiscounted Cumulative Net Cash Flow After-Tax (NCF)[4]	$ (M)	1,103	4,166	5,876	
IRR Pre-tax	%	3.0	9.4	12.3	
IRR After-tax	%	2.3	7.7	10.2	
Payback Year	years	15	7	5	

Notes: EBITDA = earnings before interest, taxes, depreciation and amortization; NPV = Net Present Value of Cumulative Cash Flow; IRR = Internal Rate of Return. NPV and IRR figures are discounted to January 1, 2009.

1) Numbers shown on 100% project basis. NovaGold and Barrick each own 50% of the Donlin Creek project subject to a 5% to 15% back-in right by Calista Corporation
2) Operating cost figure is rounded
3) Does not include sunk costs, closure costs or credit for salvage values
4) Net of initial and sustaining capital and operating costs

Table 18-11: Project Sensitivity to Gold Price

	Unit	Base Case	Alternative Case 1	Alternative Case 2
Gold Price	$/oz	725	900	1,000
Oil Price	$/barrel	75	75	75
Undiscounted Cumulative Net Cash Flow Pre-tax	$	1,504	5,915	8,435
Undiscounted Cumulative Net Cash Flow After-tax	$	1,103	4,166	5,876
NPV (5%) Pre-tax	$	(592)	1,525	2,735
NPV (5%) After-tax	$	(733)	829	1,674
IRR Pre-tax	%	3.0	9.4	12.3
IRR After-tax	%	2.3	7.7	10.2
Payback	Years	15	7	5

Table 18-12: Project Sensitivity Ranges in Gold Prices

Gold Price ($/oz)	NCF ($M)	NPV @ 5% ($M)	IRR (%)
550	(2,938)	(2,722)	(9.5)
600	(1,706)	(2,128)	(4.5)
650	(485)	(1,538)	(1.1)
700	631	(981)	1.4
725	1,103	(733)	2.3
750	1,554	(498)	3.2
800	2,430	(45)	4.8
850	3,300	395	6.3
900	4,164	828	7.7
950	5,024	1,255	9.0
1,000	5,875	1,674	10.2

Table 18-13: Project Sensitivity to Oil Price at $725/oz gold

Oil Price ($/barrel)	Net Cash Flow ($M)	NPV @ 5% ($M)	IRR (%)
35	2,106	(236)	4.2
50	1,744	(415)	3.5
75	1,103	(733)	2.3
100	430	(1,069)	0.9

Figure 18-4: Net Cash Flow Sensitivity Spider Graph

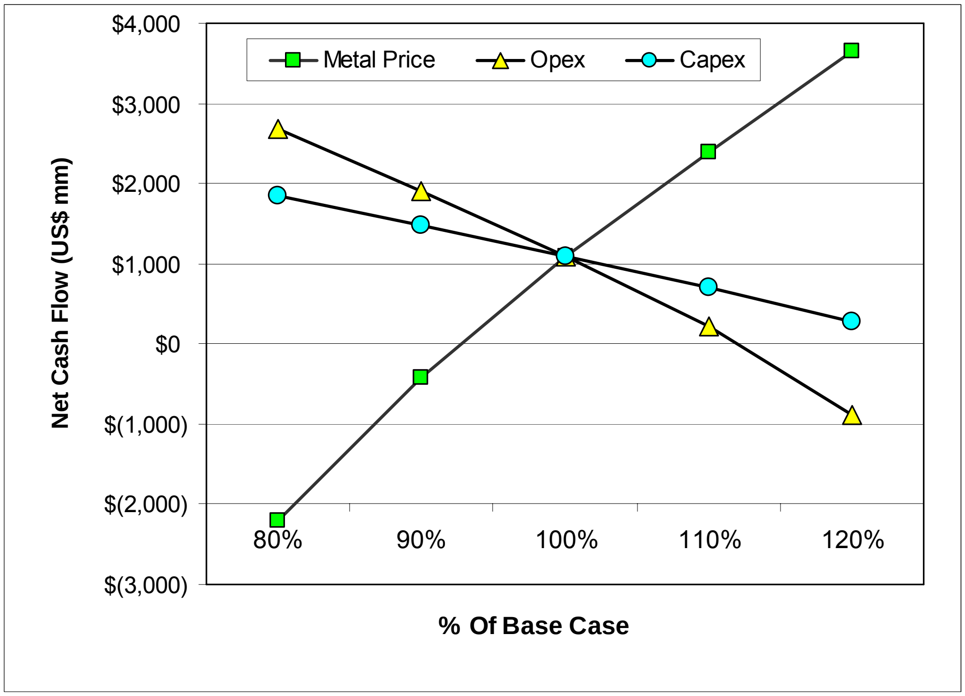
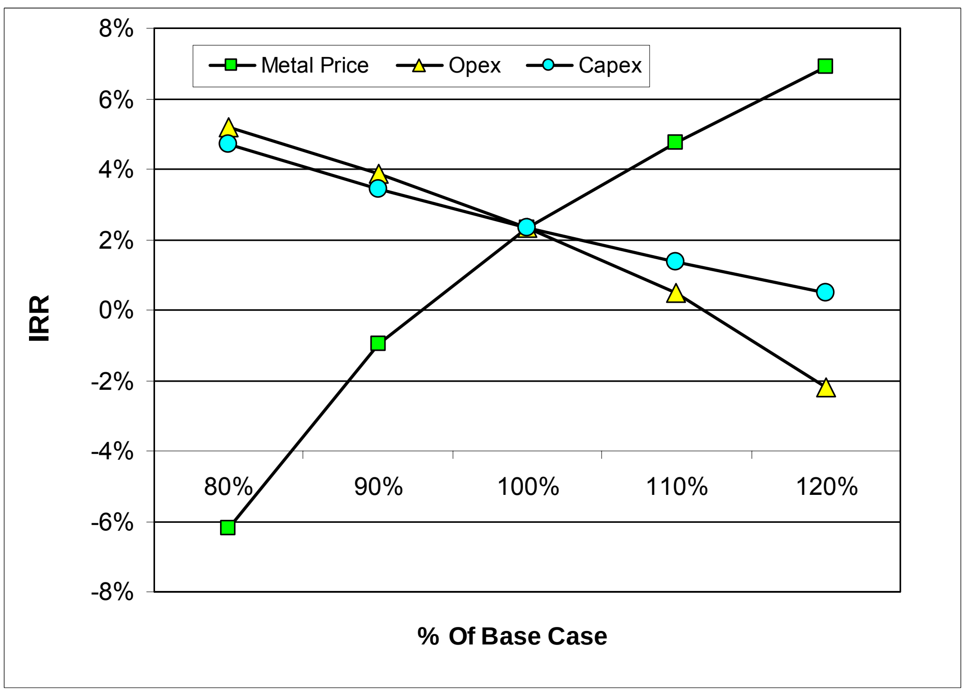


Figure 18-5: IRR Sensitivity Spider Graph



19.0 OTHER RELEVANT DATA AND INFORMATION

There are no other data that are relevant to the Report.

20.0 INTERPRETATION AND CONCLUSIONS

AMEC has reviewed the information incorporated in the earlier chapters of this Report, together with supporting data supplied by NovaGold, the DCLLC, and the Donlin Creek feasibility study update. As a result, AMEC has concluded:

- The tenure and surface rights are valid for the Donlin Creek area, and can support declaration of Mineral Reserves and Mineral Resources. Additional surface rights will need to be acquired to support planned infrastructure at Bethel and BTC and for a portion of the proposed tailings dam. A right-of-way will be required from the State of Alaska for the road alignment where it crosses State lands. Negotiations will also be required for lands needed for the wind farm. Negotiations regarding the additional Native lands are ongoing with both TKC and Calista
- Agreements exist between the DCLLC and Calista and TKC, and between NovaGold and Barrick, and are sufficient to support development of the Project. Two royalties will be in effect, to Calista and Lyman Resources
- All exploration activities on leased lands are covered under the terms of the lease agreement with Calista and the surface use agreement with TKC. Activities on Native-owned lands not currently within the agreement, or on state and federal lands, are permitted on an individual basis as required. Drilling operations on the Project are covered under the Alaska Placer Mining Application process and related permits
- The proposed Donlin Creek operation will require a considerable number of permits and authorizations from both federal and state agencies. The DCLLC is aware of the required permits, application procedures, and required time-frames for approvals
- The geology of the Donlin Creek deposit is well understood. Mineralization types and extents are well-defined and can support declaration of Mineral Resources and Mineral Reserves. Geological interpretations for the area are based on surface exposures, trenches and drill information. Mineralogical interpretations are based on data returned from a number of research studies and metallurgical testwork programs, and support the planned process route
- The exploration programs completed to date are appropriate to the style of the Donlin Creek deposits, and has identified numerous zones of anomalous gold and copper grades. As the geochemical and trench analyses were superseded by the amount of drill data available, exploration-stage analytical data were not reviewed. Research work supports genetic and affinity interpretations for the deposits

- The quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the exploration, drilling and infill delineation programs are sufficient to support Mineral Resource and Mineral Reserve estimation
- Sampling methods are considered to be acceptable, are consistent with industry-standard practices, and are adequate for supporting Mineral Resource and Mineral Reserve estimation and for mine planning purposes
- The quality of the gold analytical data is reliable and that sample preparation, analysis, and security are generally performed in accordance with exploration best practices and industry standards
- Data collected from the Project adequately support the geological interpretations and the database quality, and therefore support the use of the data in Mineral Resource and Mineral Reserve estimation
- Metallurgical testwork completed on the Project was appropriate to establish the optimal processing route, and was performed using samples that are typical of the mineralization within the Project. Recovery factors appear appropriate for the mineralization styles and planned process route. The process route is feasible and uses industry standard equipment and techniques
- Mineral Resources and Mineral Reserves were estimated in accordance with the CIM (2005) Definition Standards for Mineral Resources and Mineral Reserves
- The open pit mine plan is appropriate to the style of mineralization. Production forecasts are achievable with the equipment and plant planned. There is some upside for the Project if the Inferred Mineral Resources that are identified within the LOM production plan can be upgraded to higher confidence Mineral Resource categories. The predicted mine life of 20 years is achievable based on the projected annual production rate and the Mineral Reserves estimated
- The marketing plan assumes that each partner in the DCLLC is responsible for marketing its share of the gold production. NovaGold has reviewed the gold spot market. Sale of production is expected not to be an issue
- Doré refining contracts are expected to be typical of, and consistent with, standard industry practice, and be similar to contracts for the supply of doré elsewhere in the world
- The EMS and permit review process will determine the precise number of management plans required to address all aspects of the Project to ensure compliance with environmental design and permit criteria. The environmental impact of the operation, and subsequent closure and remediation requirements will be addressed in the proposed mine plan and EIS, following receipt of commentary

that may be associated with Project approvals. Management of the Crooked Creek waterway is noted as critical

- Taxation considerations are limited to a review of the major applicable taxes for incorporation in the financial analysis
- Capital and operating costs are based on 2008 estimates. Capital costs consist of semi-detailed unit costs and assembly line items to AACE Class 3 standards; operating costs were estimated by area and component, based on estimated staffing levels, consumables, and expenditures, according to the mine plan and process design. Costs are considered to be in line with third-quarter 2008 rates
- The financial analysis shows that the Project is positive using base case assumptions as detailed in this Report
- The Project economics are particularly sensitive to the gold price, and to a lesser extent to the oil price. For the purposes of the sensitivity analysis, DCLLC assumed that the Project sensitivity to changes in gold grades was mirrored by the sensitivity of the Project to changes in the gold price.

Mineralization continues below the proposed ACMA pit, but expansion is limited due the proximity of Crooked Creek on the west and south, and by the location of the planned process facilities to the west. Exploration potential is still open to the north. A small mineralized area approximately 1,000 m to the north of the Lewis pit was drilled on 40 m spacing, but was not included in the resource model. The area under the prominent ridge in the pit design (54,1000E, 6,879,500N) lacks drilling. AMEC recommends that this area should be explored, for if economic mineralization could be found, it could have a significant impact on the design and efficiency of the pit as well as the Project economics.

The Project remains open along the Donlin trend to the north. The discovery potential in the remaining 6 km geologic trend is high. An integrated exploration program, including mapping, geochemical characterization, geophysics, and drilling, would be required to test known targets and pit area extensions, and to identify new targets within the Donlin trend.

In the opinion of the QPs, the Project outlined in this Report has met its objectives. Mineral Resources and Mineral Reserves were estimated for the Project and a proposed plan for mining operations was outlined. This indicates data supporting the Mineral Resource and Mineral Reserve estimates were appropriately collected, evaluated and estimated, and the original Project objective of identifying mineralization that could support development of a mining operation was achieved.

21.0 RECOMMENDATIONS

As a result of the feasibility study update, a number of recommendations were made for Project advancement and risk mitigation. These were divided into a two-phase work program, with the second phase dependent on the results of the first. A total budget allocation for both phases of work is approximately $2.5 million. The programs are broken down by area.

21.1 Phase 1

The planned Phase 1 work program is estimated to cost about $2.33 million, and consists of the following:

- Exploration – Additional field mapping ($50,000)
- Structural model – update the structural model for the Project ($25,000)
- Update grade model to include 2008 drilling ($30,000)
- Additional mining studies such as drill and blast ($50,000)
- Ore control studies – optimize the proposed mine plan to reflect the mining sequence ($10,000)
- Contact water dam location and other hydrological studies ($50,000)
- Process and water treatment study ($250,000)
- Water management studies ($75,000)
- Waste rock facility additional studies and design ($50,000)
- Tailings storage facility additional studies and design ($200,000)
- Power additional studies and design ($15,000)
- River survey ($500,000)
- Review of logistics providers ($25,000)
- Baseline mercury studies ($60,000)
- Tailings permitting strategy ($60,000)
- Local and regional consultation ($1,000,000)

21.2 Phase 2

The planned Phase 2 work program is dependent upon results of the Phase 1 work. It is estimated to cost about $125,000, and consists of the following:

- Detailed design of pre-engineered buildings ($15,000)
- Dewatering studies ($50,000)
- Contact water dam design ($20,000)
- Workforce development ($20,000)
- Execution strategy and design ($20,000)

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22.2 Glossary

Abbreviation/acronym	Meaning
'	seconds (geographic)
'	foot/feet
"	minutes (geographic)
"	inches
#	number
%	percent
/	per
<	less than
>	greater than
®	registered name
µm	micrometer (micron)
a	annum/ year
Å	angstroms
AA	atomic absorption spectroscopy
AES	Alaska Earth Science
Ai	abrasion index
AMR	advance minimum royalty
ANC	acid-neutralizing capacity
ANCSA	Alaska Native Claims Settlement Act
ANP	acid-neutralizing potential
ARD	acid-rock drainage
ASD	analytical spectral device
asl	above sea level
AuAA	cyanide-soluble gold
AuEq	gold equivalent
AuFA	fire assay
AuPR	preg-rob gold
AuSF	screen fire assay
BFA	bench face angle
BLEG	bulk leach extractable gold
BLM	US Bureau of Land Management
BQ	36.5 mm size core
BTC	Birch Tree Crossing
BWI	Bond work index
c.	circa

Abbreviation/acronym	Meaning
C.P.G.	Certified Professional Geologist
Capex	capital expenditure
CCD	counter-current decant
CCGT	combined cycle gas turbine
CFT	conventional bench flotation test
Ci	crusher index
CIL	carbon-in-leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CRM	certified reference material
CTOT	carbon total
Cu Eq	copper equivalent
CuCN	cyanide-soluble copper
CWI	Bond low-energy impact index
d	day
d/wk	days per week
DCLLC	Donlin Creek LLC
dmt	dry metric tonne
DWI	drop weight test
E	east
EIS	Environmental Impact Statement
EIS	environmental impact statement
EMS	Environmental Management System
EOM	end of month
EOY	end of year
fineness	parts per thousand of gold in an alloy
FLEET	Flotation Economic Evaluation Tool
g	gram
G&A	general and administrative
g/cm^3	grams per cubic centimetre
g/dmt	grams per dry metric tonne
g/m^3	grams per cubic meter
Ga	billion years ago
GPS	global positioning system
H	horizontal
ha	hectares
HP	horsepower
HPGR	high pressure grinding rolls
HPGR	high pressure grinding rolls
HQ	63.5 mm size core
ICP	inductively-couple plasma
ICP-MS	inductively-coupled plasma mass spectrometry
ICP-OES	inductively-coupled plasma optical emission spectrometry
ID	inverse distance interpolation; number after indicates the power, eg ID6 indicates inverse distance to the 6th power.
JCR	joint condition rating
kg/m^3	kilograms per cubic meter
km	kilometre
km^2	square kilometres

Abbreviation/acronym	Meaning
koz	thousand ounces
kV	kilovolt
KV	kriging variance
kVA	kilovolt–ampere
kW	kilowatt
kWh	kilowatt hour
lb	pound
L–G	Lerchs–Grossmann
LOM	life-of-mine
M	million
m	metre
m^3	cubic metre
m^3/hr	cubic metres per hour
Ma	million years ago
MAusIMM	Member of the Australasian Institute of Mining and Metallurgy
MCF2	mill–chemical–float–mill–chemical–float
mesh	size based on the number of openings in one inch of screen
MFT	Minnovex flotation test
mi	mile/miles
MIK	multiple-indicator kriging
Mlb	million pounds
Mm	million meters
mm	millimetre/millimetres
Moz	million ounces
Mt	million tonnes
Mt/a	million tonnes per annum
MW	megawatts
MWMS	mine water management system
MWMT	meteoric water mobility testing
N	north
NAG	net acid generation/net acid generating
NAPP	net acid-producing potential
NEPA	National Environmental Policy Act
NI 43-101	Canadian National Instrument 43-101 “Standards of Disclosure for Mineral Companies”
NN	nearest-neighbour
NNP	net neutralizing potential
NPI	net profits interest
NQ	47.6 mm size core
NSR	net smelter return
NW	northwest
º	degrees
ºC	degrees Celsius
OK	ordinary kriging
Opex	operating expenditure
oz	ounce/ounces (troy ounce)
oz/t	ounces per tonne
p	passing
P.E.	Professional Engineer
P.Eng.	Professional Engineer
P.Geol	Professional Geologist
PAG	potentially acid-generating (

Abbreviation/acronym	Meaning
pH	measure of the acidity or alkalinity of a solution
PLI	point load index
PLO	Public land order
pop	population
POX	Pressure oxidation
ppb	parts per billion
ppm	parts per million
PQ	85 mm size core
PSI	yield strength
QA/QC	quality assurance and quality control
QLT	quick leach test
QP	Qualified Person
R	range
RAB	rotary air blast
RC	reverse circulation
RMR	rock mass rating
ROD	Record of Decision
ROM	run-of-mine
RPL	environmental monitoring plan
RQD	rock quality designation
RWI	Bond rod mill work index
S	south
SABC	semi-autogenous milling with ball milling and pebble crushing
SAG	semi-autogenous grind
SCN	thiocyanate
SE	southeast
SEIS	Supplemental Environmental Impact Statement
SG	specific gravity
SMU	selective mining unit
SPI	Minnovex SAG power index
SRM	standard reference material
SS	sulphide sulphur
SS	State-selected

Abbreviation/acronym	Meaning
ST	scavenger tailings
STOT	sulphur total
SWIR	shortwave infrared
t	metric tonne
T	Township
t/a	tonnes per annum (tonnes per year)
t/d	tonnes per day
t/h	tonnes per hour
t/m^3	tonnes per cubic meter
TA	tentative approval
TDS	total dissolved solids
TF	tonnage factor
TKC	The Kuskokwim Corporation
Topo	topography
TSF	tailings storage facility
TSS	total suspended solids
UHF	ultra-high frequency
USGS	United States Geological Survey
V	vertical
VHF	very high frequency
W	west
WRD	waste rock dump
wt%	weight percent
XRD	X-ray diffraction
XRF	X-ray fluorescence

23.0 DATE AND SIGNATURE PAGE

The effective date of this Technical Report, entitled “NovaGold Resources Inc., Donlin Creek Gold Project, Alaska, USA, NI 43-101 Technical Report” is 1 April 2009.

on behalf of AMEC Americas Limited.

“Signed”

Bob Stanlake
President, Mining and Metals,
for AMEC Americas Limited.

Dated 2 June 2009